Pretium Resources Inc.: Brucejack Project

Mineral Resources Update Technical Report

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Final

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1 Summary

In August 2013, Snowden Mining Industry Consultants Inc. ("Snowden") was engaged by Pretium Resources Inc. ("Pretivm"), to complete an update of the Mineral Resource estimate for the Valley of the Kings (VOK or VOK Zone) at the Brucejack Project in compliance with National Instrument ("NI") 43-101 and Form 43-101F1. In addition, the West Zone estimate created as part of the April 2012 Mineral Resource (Jones, 2012a), has been documented in this report for completeness. West Zone was not updated for this Mineral Resource as there has been very little additional drilling in this area. The zones are gold-silver zones of mineralization that are part of Pretivm's Brucejack Project that was the subject of a Feasibility Study in 2013 based on the previous November 2012 Mineral Resource. The Feasibility Study will be updated in 2014 to reflect the change in the Mineral Resource, as well as using revised economic parameters.

The purpose of this Technical Report is to support the news release of 19 December 2013 in which an updated Mineral Resource estimate was reported for VOK as well as to support the reported estimates as noted above.

The Brucejack Property (the "Property") is situated approximately at 56°28′20″N Latitude by 130°11′31″W Longitude, a position approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine in the Province of British Columbia. The Brucejack Property consists of six mineral claims totalling 3,199.28 ha in area and all claims are in good standing until 31 January 2024.

The Property and the surrounding region have a history rich in exploration for precious and base metals dating back to the late 1800s. More recently in 2009 Silver Standard Resources Inc. ("Silver Standard") began work on the Property. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core. In 2010, pursuant to a purchase and sale agreement between Silver Standard (as the seller) and Pretivm (as the buyer), Silver Standard sold to Pretivm all of the issued shares of 0890693 BC Ltd., the owner of the Brucejack Project and the adjacent Snowfield Project.

In 2010, Silver Standard's drill program was designed to further define bulk tonnage mineralization found the previous year, as well as to attempt to define a high grade resource for the VOK. In this year, 73 diamond drillholes were completed which totalled 33,480 m. Of this, 11 drillholes comprising 3,693 m targeted the VOK, and two drillholes totalling 1,119 m targeted the footwall of West Zone. In the VOK, wide spaced drilling intersected enough high grade mineralization to confirm the exploration potential of the zone. The exploration potential included the preliminary definition of some of the ore controls which put the intersections into a geologic context. The West Zone drilling intersected a broad zone of bulk tonnage mineralization within which were several high grade intersections.

Pretivm's 2011 diamond drill program was the first in almost 20 years that was focused specifically on defining high grade resources. In this year 178 drillholes were completed totalling 72,805 m in drillholes SU-110 to SU-288. Included in this were 97 drillholes (41,219 m) that targeted the VOK, 16 drillholes (7,471 m) that targeted West Zone, and 21 drillholes (7,220 m) that targeted the surrounding areas. The remaining drilling was focused on expansion of Shore Zone, testing for structurally controlled high grade mineralization in Galena Hill (now part of the VOK) and Bridge Zones, as well as testing new target areas.

The 2012 diamond drill program focused on defining the high grade resource for the VOK, specifically targeting geological and structural features believed to be associated with high grade gold mineralization. Diamond drilling also focused on expanding the known extents of the VOK Zone, both west of the Brucejack Fault and along trend to the east of the main mineralized zone. A total of 301 drillholes were completed, totalling 105,500 m of drilling during the 2012 drilling program.

The 2013 surface diamond drill program focused on further defining the high grade resource for the VOK as well as the geological and structural features believed to be associated with gold mineralization. A total of 24 surface diamond drillholes (5,200 m) of the 37 surface drillholes (5,770 m; drillholes SU-590 to SU-626) completed during the 2013 drilling program were focussed on the VOK.

In 2013 Pretivm also excavated a bulk sample from within the VOK to further evaluate the geological interpretation and Mineral Resource estimate. The location of the proposed bulk sample was selected to be representative of the grade and character of the overall mineralization of the VOK.

Underground development reached the bulk sample area in May 2013 and underground drilling to support the bulk sample program began in mid-May. A total of 409 drillholes (38,840 m) were completed with 200 of these drillholes (16,640 m) testing the bulk sample area. The remainder (209 drillholes totalling 22,200 m) were testing targets outside of the bulk sample area.

The design of the bulk sample was limited by provincial legislation to a maximum allowable bulk sample size of 10,000 tonnes. The bulk sample was collected as a series of nominal 100 tonne rounds in underground development. Pretivm elected to process the bulk sample both through a sample tower on site and at a custom mill (Contact Mill) in Montana, USA. The results of assaying of the samples from the sample tower provided, in Snowden's opinion, an unacceptable degree of variation in the results.

The mill results from the underground bulk sample processing were used to validate the local accuracy of the November 2012 Mineral Resource in the VOK, and to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource. The result of the testwork is an improved confidence in both the geological model and the grade estimate, with the definition of Measured Resources as part of the December 2013 Mineral Resource.

1.1 Geology and mineralization

The Brucejack property is largely underlain by volcano-sedimentary rocks of the Lower Jurassic Hazelton Group. These rocks unconformably overlie volcanic arc sedimentary rocks of the Upper Triassic Stuhini Group along the westernmost part of the property. Hazelton Group rocks on the property include hornblende and/or feldspar-phyric volcanic (latite) flows, and pyroclastic fragmental rocks, locally derived heterolithic volcanic pebble to boulder conglomerate, volcanic sandstone, siltstone and mudstone. The Hazelton Group volcano-sedimentary rocks are interpreted as having been deposited in an extensional structural regime, proximal to a basin margin growth fault (or series of growth faults), owing to the presence of complicated lateral facies variations within the volcanic pile and the diachronous and immature nature of the rock units. The Brucejack Fault, which forms a distinct topographical feature across the property, is currently interpreted as being the reactivated expression of these basin margin structures.

Gold (± silver) mineralization is hosted in predominantly sub-vertical vein, vein stockwork, and subordinate vein breccia systems of variable intensity, throughout the alteration band. The stockwork systems display both parallel and discordant relationships to stratigraphy. These systems are relatively continuous along strike (several tens of metres to several hundreds of metres) and are characterized by the presence of millimetre- to decimetre-scale transitional to epithermal veins of pyrite, quartz, quartz-carbonate, and, less commonly quartz-adularia (e.g., in the West zone) that form cross-cutting networks of variable intensity within stockwork zones.

Several mineralization zones have been explored to varying degrees, including (from south to north): Bridge Zone, VOK, West Zone, Gossan Hill, Shore Zone, and SG Zone. There are numerous relatively unexplored mineralization showings within the alteration band across the property that are between the main mineralization zones, highlighting the exceptional exploration potential of the Brucejack property.

High grade gold mineralization at the VOK, the current focus of the Brucejack Project, occurs in a series of west-northwest (and subordinate west-southwest) trending sub-vertical corridors of structurally reoriented vein stockworks and vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate, and latite fragmental rocks. Relatively massive latite flows are present to the immediate north and south of the mineralization host rock sequence. Gold is typically present as gold-rich electrum within deformed quartz-carbonate (±adularia?) vein stockworks, veins, and subordinate vein breccias, with grades ranging up to 41,582 g/t Au and 27,725 g/t Ag over 0.5 m.

Recent underground exploration carried out as part of the bulk sample in the VOK has resulted in the recognition of sub-vertical north-northeasterly trending deformed, curviplanar, and sheared quartz-carbonate veins containing abundant visible electrum. These relatively rare structures are interpreted as structurally-controlled fluid conduits that were active during development of the porphyry system and associated volcanic pile in the early Jurassic, and which were reactivated during Cretaceous deformation.

The VOK deposit is currently defined over 1,200 m in east-west extent, 600 m in north-south extent, and 650 m in depth. The West Zone appears to form the northern limb of an anticline that links up with the VOK in the south, and the southern limb of a syncline that extends further to the north. This zone, which is currently defined over 590 m along its northwest strike, 560 m across strike, and down to 650 m in depth, is open to the northwest, southeast, and at depth to the northeast.

The Brucejack deposit is considered to be a transitional to intermediate sulphidation epithermal stockwork vein system-hosted gold-silver deposit that was developed in a dynamic extensional basin. It is likely associated with a deeper porphyry system that developed within an active island arc tectonic setting.

1.2 Mineral processing and metallurgical testing

1.2.1 Metallurgical testwork

Information in this section has been excerpted, condensed and updated from the Feasibility Study reported in 2013 (Ireland et al., 2013) and from Ghaffari et al., (2012).

In general, the VOK and West Zone mineralization is moderately hard. The mineral samples tested responded well to the conventional combined gravity and flotation flowsheet. The gold in the mineralization was amenable to centrifugal gravity concentration. On average, between 40% and 50% of the gold in the samples was recovered by gravity concentration. The flotation test results indicated that bulk flotation can effectively recover the gold in the gravity concentration tailings using potassium amyl xanthate (PAX) as a collector at the natural pH. It was also concluded that two stages of cleaner flotation would significantly upgrade rougher flotation concentrate.

With respect to metallurgical recovery, the gold in the mineralization showed better metallurgical performance when compared to silver. On average, 96% to 97% of the gold and 91% to 92% of the silver were recovered in the gravity concentrate and bulk flotation concentrate at the grind size of 80% passing approximately 70 µm to 80 µm. There was a significant variation in metallurgical performances among the samples tested. This may be a result of the variably nuggetty mineralization.

1.2.2 Bulk sample program

The 10,000 tonne bulk sample was mined during the third quarter of 2013 and processed in the fourth quarter of 2014. As a part of its engagement with Pretivm, Snowden reviewed the handling of the bulk sample from the mine to the production of final concentrates and tails at the Contact Mill. Snowden concluded that the metal accounting standards were maintained to high standards and that the treatment of the ore was conducted in line with reasonable expectations with regards to precious metal recovery and monitoring of process parameters. The metallurgical monitoring consisted of:

- A brief review of the sample tower operations to understand how the nominal 100 tonne samples were managed and processed on site.
- Management of and the chain of custody during shipping of the bulk sample.
- Processing of the bulk sample at the Contact Mill.
- Metal accounting procedures at the Contact Mill.
- Evaluation of the bulk sample processing results.

Sample shipping

After crushing and sampling through the sampling tower, the ore was loaded into bulk bags (approximately one tonne each), which were shipped via five transfer points from the Brucejack project site to the mill in Philipsburg, Montana.

Snowden considered the overall controls and the management of the shipping of the bulk bags, which was logistically a very complex operation, to have been handled extremely well and can confirm that more than 99% of the bags arrived at the mill intact and could be clearly identified for processing in the mill.

Contact Mill processing

The Contact Mill is an older, manually controlled mill, requiring experienced operators to ensure that the operational targets are achieved.

A relatively simple process flowsheet was used for the recovery of gold from the Brucejack bulk sample.

In spite of the age of the mill, it performed well mechanically, with only a limited number of breakdowns. Snowden found the staff to be competent and diligent in the execution of their duties.

Metal accounting

All metal accounting samples were manual samples, which were generated as composite samples from individual hourly samples per six hour period of operation. The mill feed rate was measured manually by stopping the feed conveyor every hour to take a belt cut, which was weighed accurately to determine the mill mass feed flow rate.

A spear sample (one spear) of the final flotation concentrate was taken by the mill operators and analysed on site. A second spear sample (12 spears) was taken independently by a Pretivm staff member and submitted separately to an external laboratory for analysis. The comparison of the site flotation concentrate analyses against the commercial laboratory shows an overall average difference for gold where the mill results are slightly higher than 5%. An analysis of the difference at different flotation concentrate grades shows that the difference varies depending on concentrate grade, but is similar for higher and lower grade concentrates. If the commercial laboratory is correct the average feed grade of the ore defined by processing will be around 5% lower than that reported in this document. The final smelter settlements, however, will dictate the finally agreed feed grades and recoveries. These data are not yet available.

Table gravity concentrate was collected and the full production dried and subsampled via a riffle splitter. A total of 3,645 ounces of gold were recovered via the gravity circuit.

Final flotation tails were sampled every hour by a mill operator, as the tails were transferred to the tailings storage facility (TSF). These samples were composited over four hours to generate six samples during a standard operating day.

Final results of the metal accounting balance across the mill are shown in Table 1.1.

Table 1.1 Summary mill mass balance results of the Brucejack bulk sample processing

Cross-cut	Tonnes treated (t)	Float con. Au (oz)	Table (gravity) con. Au (oz)	Tails Au (oz)	Total Au (oz)	Au recovery	Average grade per round Au (g/t)
426585 E	2,169	173.9	93.5	12.9	280.4	95.4%	4.02
426555 E	1,416	92.9	102.2	5.8	200.9	97.1%	4.41
426645 E	1,875	68.6	62.3	6.4	137.3	95.4%	2.28
426615 E	2,878	1,289.3	2,290.8	61.4	3,641.5	98.3%	39.35
615 L (incl. mixed bags)	1,964	477.1	1,096.4	37.5	1,611.0	97.7%	25.52
Final clean-out		52.0			52.0		
Total	**10,302**	**2,101.9**	**3,645.3**	**124.0**	**5,923.2**	**97.9%**	**17.88**

Overall recoveries achieved during the treatment of the bulk sample were excellent at 97.9%.

It is Snowden's opinion that the handling and treatment of the 10,000 tonne bulk sample was executed and managed well. Results are considered to be as accurate as can be expected for a bulk sample treatment campaign of this nature, providing Pretivm with a robust average gold grade for each crosscut as well as for the full 10,000 tonne bulk sample.

1.3 Drilling, sampling, assaying, and data verification

The input data for the VOK Mineral Resource estimate comprised 922 drillholes totalling 218,127 m. These included:

- Nine historic drillholes (579 m).
- 490 surface drillholes drilled between 2009 and 2012 (173,619 m).
- 24 surface drillholes drilled in 2013 (5,200 m).
- 409 underground drillholes drilled in 2013 (38,840 m).

The input data for the West Zone estimate comprised 756 drillholes (63,208 m) including 439 underground drillholes (24,688 m), 269 historical surface drillholes (21,321 m) and 48 surface drillholes (17,199 m) completed since 2009.

Historical drill core sizes for surface drillholes were generally NQ (47.6 mm diameter) or BQ (36.5 mm diameter). However, core size for drillholes collared from an underground exploration ramp at West Zone was AQ (27 mm diameter).

Core sizes for Pretivm's surface collared drillholes were PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ (47.6 mm diameter). Approximately 50% to 60% of the Pretivm core was HQ size. For drillholes less than 600 m length, core size was commenced at HQ and reduced to NQ when required. For drillholes greater than 600 m length the commencing core size was PQ which was run down to between 200 m and 300 m in order to minimise drill path deviation. All drillcore collected from the underground drilling in 2013 was HQ size. No significant bias was noted between the PQ and HQ drill core samples that intersected the VOK mineralization. No testing was required on the NQ drill tails as these were almost without exception at depths below the main mineralization zones.

The drill collars were surveyed by McElhanney Surveying from Terrace, BC. McElhanney Surveying used a total station instrument and permanent ground control stations for reference and have completed all the surveying on the project since 2009. All underground drill collars were surveyed by Procon.

Drillhole paths were surveyed at a nominal 50 m interval using a Reflex EZ single shot instrument. All drillhole paths were checked in a mining software package for deviation errors, which, if present, were corrected on a realtime basis. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results.

Split PQ samples weigh approximately 10 kg. HQ samples were around 6 kg, and NQ 3 kg to 4 kg. These weights assume a nominal 1.5 m sample length. In general, the average sample size submitted to the analytical laboratory, ALS Chemex ("ALS") was 6.5 kg. Samples at ALS were crushed to 70% passing 2 mm, (-10 mesh). Samples were riffle split and 500 g were pulverized to 85% passing 75 µm (-200 mesh). The remaining coarse reject material was returned to Pretivm for storage in their Stewart, BC warehouse for possible future use.

Gold was determined using fire assay on a 30 g aliquot with an atomic absorption (AA) finish. In addition, a 33 element package was completed using a four acid digest and ICP-AES analysis, which included silver. Density determinations were done by ALS using the pycnometer method on pulps from the drilling program.

Procedures undertaken by Pretivm have been under the supervision and security of Pretivm's staff, as far as drill core sampling prior to dispatch. Laboratory sample reduction and analytical procedures have been conducted by independent accredited companies with acceptable practices.

Pretivm ensured quality control was monitored through the insertion of blanks, certified reference materials and duplicates. All data was stored and managed by independent database managing company, Geospark Consulting who carried out real time QAQC analysis and sent batches for reassaying as required.

Snowden carried out several site inspections and reviewed Pretivm's procedures including:

- Independent sampling to verify the grade tenor.
- Inspection of the underground workings to confirm the mineralization style.
- Review of diamond core.
- Review of site procedures.
- Independent QAQC analysis.
- Independent data validation.

It is the author's opinion that the sample preparation, security, and analytical procedures are satisfactory and that the data is suitable for use in Mineral Resource estimates.

1.4 Mineral Resource estimate

In December 2013, Snowden completed a Mineral Resource estimate for the VOK Zone of the Brucejack Project. The West Zone estimate remains unchanged from the April 2012 Mineral Resource estimate (Jones, 2012a).

1.4.1 Estimation testwork in the bulk sample area

Prior to estimation of the December 2013 Mineral Resource, the underground bulk sample results were used to investigate the local accuracy of the November 2012 Mineral Resource estimate within the VOK, and to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource.

A series of statistical tests were run to determine whether any bias exists between the surface diamond drilling, underground diamond drilling, underground channel samples, and chip samples. No statistical bias, based on these statistics, was evident between the different sample types.

Additional testwork in the estimation did, however, display some bias caused by directional drilling in the area of the bulk sample. The underground drilling had been aligned in a north-south orientation which is consistent with the orientation of some high grade mineralization identified in the bulk sample. Removal of the underground drillholes resulted in an increase in the grade of the local estimate. This was particularly evident in those crosscuts dominated by north-south mineralization (e.g., 426615E), and resulted in a significantly better correlation with the results from processing in the mill.

Associated with the bulk sample, Pretivm also completed a substantial amount of underground drilling. This drilling was closely spaced, but based mostly on a north-south grid and appears to have created a directional bias in the drilling information because of a north-south aligned mineralization along the Cleopatra structure. However, this drilling, along with the results of processing of the bulk sample, was used to assist in the improvement of grade estimation parameters. It was noted as a part of this testwork, however, that the result of including the new drilling information in the resource estimation further under-estimated the grade in the bulk sample because of this directional bias.

While there is no bias evident between the channel samples and the drilling, the location of numerous channel samples in the centre of some of the higher grade mineralization does result in a local overestimation around the bulk sample crosscuts. Consequently the decision was made not to use the channel samples for the final Mineral Resource estimate.

The final metal and tonnes from the mill accounting were compared to those predicted by the November 2012 Mineral Resource estimate for each to assess the appropriateness of the modelling process. This test work has in part relied on comparisons between the test estimates and results from the bulk sample processing. However, the reader should be warned that there is a significant difference in the sample support for the resource estimate (each block in the resource estimate represents 2,700 tonnes whereas the bulk sample packages are around 100 tonnes), and the grade is not homogenous throughout any block. In other words, the grade can vary from a high grade side of the block to a low grade side of the block, whereas the block grade represents an average of the whole block. If the bulk sample happens to take a high grade part of the block, then the comparison will look like the resource estimate under-estimated the grade, and conversely if the bulk sample takes a low grade part of the block, then the comparison will look like the resource estimate over-estimated the grade in the block. Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally, it does provide the best opportunity to fine-tune the estimate to some hard data. The reader should be warned that the results are only used to give some local perspective to the grade estimates.

The results indicated that the November 2012 Mineral Resource underestimated the total metal content in the bulk sample by about 10%. In more detail, the November 2012 Mineral Resource estimated high grade into lower grade areas, and low grade into the high grade areas, a result of extrapolating the high grade values around the high grade core. This extrapolation of high-grade values was based on the nature of the mineralization and the interpreted continuity of the high grades.

Based on the bulk sample comparisons, Snowden concludes that the November 2012 Mineral Resource was a good indicator of the contained metal within the VOK deposit and satisfactory for bulk underground mining, but that it was not locally accurate at the 10 m block scale. As a result further testwork was undertaken to adjust the estimation methodology for the December 2013 Mineral Resource, to produce an estimate that is more responsive to the local high grades.

In order to produce an estimate that is more responsive to the local high grades, a series of test estimates were created inside the local test area surrounding the bulk sample crosscuts. The estimates were compared to the bulk sample mill results on a round-by-round basis, as well as on a more global basis within the local test area.

Several options were assessed as part of the testwork including:

- Looking at the use of channel samples to assist in defining the local grade more accurately around the bulk sample crosscut.

- These tests indicated that the use of channel samples resulted in a local overestimation of the grade in the bulk sample crosscuts. As a result channel samples were not included in the December 2013 Mineral Resource estimate.
- Assessing the impact of constraining the north-south mineralization and estimating it separately to the dominantly east-west mineralized corridors.
- The testwork indicated that the estimate without any north-south constraint is a better local predictor of metal, with underestimation of the north-south mineralization. Based on the outcomes of the testwork and bulk sample analyses, Snowden and Pretivm agreed that the more conservative approach, not using the north-south constraints, should be applied for the December 2013 Mineral Resource. As a result, the December 2013 Mineral Resource estimate is considered to be a conservative estimate of the contained metal in the VOK deposit.
- Adjusting the estimation parameters and methodology to reduce smoothing, including the method for reblocking the high grade Multiple Indicator Kriging (MIK) estimates, parent cell size, and search neighbourhood parameters.
- Review of the estimation parameters resulted in slight adjustments to the search neighbourhood to produce a more selective estimate, and changing the reblocking of the high grade estimates to use the parent cell size. A parent cell size of 10 mE by 10 mN by 10 mRL was retained for the estimation of the VOK and resulted in less smoothing of the estimate.
- Comparing ordinary kriging and inverse distance weighted estimation methods.
- Testwork using ordinary kriging and inverse distance weighted estimation methods showed that these methods, when run using a ‘typical’ top cut of the 99.9th percentile or lower, significantly underestimate the metal in the bulk sample crosscuts. Given the style of mineralization, the level of selectivity required and the estimation of unrealistic high grade areas defined by these methods, the methods were not considered appropriate for grade estimation at Brucejack.

1.4.2 Mineral Resource estimation

The mineralization in the VOK exists as steeply dipping semi-concordant (to stratigraphy) and discordant pod-like zones hosted in stockwork vein systems within the volcanic and volcaniclastic sequence. High grade mineralization zones appear to be spatially associated, at least in part, with intensely silicified zones resulting from local silica flooding and over-pressure caprock formation. High grade mineralization occurs both in the main east-west trending vein stockwork system, as well as in the rarer north-south trending part of the system. Snowden notes that Pretivm has taken these various observations into consideration in its interpretation of the mineralization domains for the VOK.

A threshold grade of 0.3 g/t Au was found to generally identify the limits to the broad zones of mineralization as represented in the drill cores at West Zone and the VOK. In the VOK, a 1 g/t Au to 3 g/t Au threshold grade was used together with Pretivm’s interpretation of the lithological domains, to interpret high grade corridors within the broader mineralized zones, and define a series of mineralized domains for estimation.

All data was composited to the dominant sample length of 1.5 m prior to analysis and estimation. Statistical analysis of the gold and silver data was carried out by lithological domain (at the VOK) and mineralized domain. Review of the statistics indicated that the grade distributions for the mineralization within the different lithologies are very similar and as a result these were combined for analysis. This is in agreement with field observations which indicate that the stockwork mineralization is superimposed on the stratigraphic sequence. The summary statistics of composite samples from all domains exhibit a strong positive skewness with high coefficient of variation and some extreme grades.

Because of the extreme positive skew in the histograms of the gold and silver grades within the high grade domains, Snowden elected to use a non-linear approach to estimation, employing the use of indicator and truncated distribution kriging. In this approach the proportion of high grade in a block was modelled, as was the grade of the high grade portion, and the grade of the low grade portion.

The high grade population, which contains a significant number of samples with extreme grades, required indicator kriging methods for grade estimation. The low grade population was estimated using ordinary kriging on the truncated (low grade; <5 g/t Au and <50 g/t Ag) part of the grade distribution.

Density was estimated using simple kriging of specific gravity measurements determined on sample pulps by ALS Chemex. As part of the 2012 drilling program, Pretivm selected a portion of the samples (207 samples) to undergo core density measurements in addition to the usual pulp specific gravity measurements to assess the impact of porosity on the density. A further 204 samples were collected for specific gravity and density measurements as part of the 2013 underground drilling program to increase the comparative dataset. The results of the comparison indicate that the core density is on average the same as the pulp specific gravity within the siliceous zone and 3% lower on average for all other rock types. Bulk density estimates in the final model were determined by simply factoring down pulp specific gravity estimates by 3% in all lithologies except in the intensely silicified conglomerate.

Grade estimates and models were validated by: undertaking global grade comparisons with the input drillhole composites; visual validation of block model cross sections; grade trend plots; and comparing the results of the model to the bulk sample cross cuts.

The resource classification definitions (Measured, Indicated, Inferred) used for this estimate are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document “CIM Definition Standards”.

In order to identify those blocks in the block model that could reasonably be considered as a Mineral Resource, the block model was filtered by a cut-off grade of 5 g/t AuEq. The gold-equivalent calculation used is: AuEq = Au + (Ag/53). These blocks were then used as a guide to develop a set of wireframes defining coherent zones of mineralization which were classified as Measured, Indicated or Inferred and reported (Table 1.2 and Table 1.3).

Classification was applied based on geological confidence, data quality and grade variability. Areas classified as Measured Resources at West Zone are within the well-informed portion where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Measured Resources within VOK are informed by 5 m by 10 m to 10 m by 10 m underground fan drilling and restricted to the vicinity of the underground bulk sample.

Areas classified as Indicated Resources are informed by drilling on a 20 m by 20 m to 20 m by 40 m grid within West Zone and VOK. In addition, some blocks at the edge of the areas with 20 m by 20 m to 20 m by 40 m drilling, were downgraded to Inferred where the high grades appear to have too much influence. The remainder of the Mineral Resource is classified as Inferred Resources where there is some drilling information (and within around 100 m of drilling) and the blocks occur within the mineralized interpretation.

Areas where there is no informing data and/or the lower grade material is outside of the mineralized interpretation are not classified as a part of the Mineral Resource.

The Mineral Resource was reported above a 5 g/t AuEq cut-off grade for the VOK and West Zone (Table 1.2 and Table 1.3).

Table 1.2 VOK Mineral Resource estimate based on a cut-off grade of 5 g/t AuEq – December 2013(1)(4)(5)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.0	19.3	14.4	1.2	0.9
Indicated	13.4	17.4	14.3	7.5	6.1
M + I	15.3	17.6	14.3	8.7	7.0
Inferred(2)	5.9	25.6	20.6	4.9	3.9

(1) Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues. The Mineral Resources in this Technical Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

(3) Contained metal and tonnes figures in totals may differ due to rounding.

(4) The Mineral Resource estimate stated in Table 1.2 and Table 1.3 is defined using 5 m by 5 by 5 m blocks in the well drilled portion of West Zone (5 m by 10 m drilling or better) and 10 m by 10 m by 10 m blocks in the remainder of West Zone and in VOK.

(5) The gold equivalent value is defined as AuEq = Au + Ag/53.

Table 1.3 West Zone Mineral Resource estimate based on a cut-off grade of 5 g/t AuEq – April 2012(1)(4)(5)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.4	5.85	347	0.5	26.8
Indicated	2.5	5.86	190	0.5	15.1
M+I	4.9	5.85	267	0.9	41.9
Inferred(2)	4.0	6.44	82	0.8	10.6

(1), (2), (3), (4) and (5) - See footnotes to Table 1.2.

1.5 Mineral Reserve

Information in this section has been excerpted from the Feasibility Study reported in 2013 (Ireland et al., 2013). This Mineral Reserve is based on the November 2012 Mineral Resource. A revised Mineral Reserve will be completed in 2014 based on the December 2013 Mineral Resource and using revised economic parameters.

A net smelter return (NSR) cut-off grade of $180/t of ore was used to define the Mineral Reserves (as used in previous studies).

The NSR for each block in the resource model was calculated as the payable revenue for gold and silver less the costs of refining, concentrate treatment, transportation and insurance. The metal price assumptions used were US$1,350/oz gold and US$22/oz silver.

Table 1.4 shows the dilution and recovery factors used in the Mineral Reserve estimation.

Table 1.4 Dilution factors and recovery factors by type of excavation

Type Of Excavation	Dilution Factor* (%)	Recovery Factor* (%)
Primary Stopes	6.8	97.5
Secondary Stopes	15.2	92.5
Sill Pillar Stopes**	15.2	75.0
Ore Cross-Cuts	4.0	100.0
Production Slashing	7.5	100.0

*Notes: *Expressed on a weight basis.*
***Includes stope ore to 30 m beneath the surface crown pillar.*

The Mineral Reserves were delineated in an orebody consisting of numerous independent lenses in the VOK Zone and two distinct lenses in the West Zone (Table 1.5).

Table 1.5 Brucejack Mineral Reserves (2013)*, by Zone and by Reserve Category

Zone		Ore tonnes (Mt)	Grade		Metal	
			Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
VOK Zone	Proven	-	-	-	-	-
	Probable	15.1	13.6	11	6.6	5.3
	Total	15.1	13.6	11	6.6	5.3
West Zone	Proven	2.0	5.7	309	0.4	19.9
	Probable	1.8	5.8	172	0.3	10.1
	Total	3.8	5.8	243	0.7	30.0
Total Mine	Proven	2.0	5.7	309	0.4	19.9
	Probable	17.0	12.8	28	7.0	15.4
	Total	19.0	12.0	58	7.3	35.3

*Notes: *Rounding of some figures may lead to minor discrepancies in totals.*
**Based on Cdn$180/t cut-off grade, US$1,350/oz gold price, US$22/oz silver price, Cdn$/US$ exchange rate = 1.0*
**Based on the November 2012 Mineral Resource estimate.*

1.6 Interpretation and conclusions

An updated Mineral Resource estimate has been prepared for VOK Zone at the Brucejack Property of Pretivm located in northwest British Columbia. The Measured, Indicated and Inferred Mineral Resource estimates, effective December 2013 are intended for use in a Feasibility Study for a high grade underground mining scenario.

In the current study, Olssen & Jones have estimated and reported Mineral Resource estimates for the high-grade portions of West Zone and VOK that are considered to be potentially minable by underground methods, regardless of any open pit potential.

In 2013, a Feasibility Study was reported (Ireland et al., 2013) based on the November 2012 Mineral Resource (Jones, 2012c). This study considered the potential for underground mining of the high-grade portions of the deposits.

Subsequent to the completion of the feasibility study, a bulk sample was completed from the VOK. This sample was processed in nominal 100 tonne parcels through a sample tower, but the variability in grades from the sample tower on each 100 tonne round was considered too high to give an accurate representation of the grade of each round. No further use of the sample tower results was considered appropriate at Brucejack.

The 100 tonne parcels from the bulk sample were shipped to Montana and processed to give the ultimate grade of the bulk sample. The results of the processing were compared with the results of the sample tower and this confirmed the poor accuracy of the sample tower results. There was no further use of the sample tower results.

Processing of the bulk sample showed that the bulk sample responded well to the conventional combined gravity and flotation flowsheet. On average, between 96% and 97% of the gold and 91% and 92% of the silver were recovered to the gravity concentrate and bulk flotation concentrate at the grind size of 80% passing approximately 70 to 80 µm. There was, however, a significant variation in metallurgical performances among the samples tested, with a greater percentage of gold reporting to the flotation concentrate for the lower grade mineralization and a greater percentage of the gold reporting to the gravity concentrate for the higher grade mineralization. This is interpreted to be a result of the variably nuggetty mineralization.

The results of processing of the bulk sample indicated that, whilst there was apparent over-smoothing of the grade estimate locally, the resource estimate under-estimated the grade within the bulk sample. Over-smoothing was manifested in under-estimation of high grade areas and over-estimation of adjacent low grade areas. The number of ounces within the bulk sample, as estimated by the November 2012 Mineral Resource, was more than 10% lower than the number of ounces from processing (although this difference may change depending on the final smelter settlements. These data are not yet available.).

Associated with the bulk sample, Pretivm also completed a substantial amount of underground drilling. This drilling was closely spaced, but based mostly on a north-south grid and appears to have created a directional bias in the drilling information because of a north-south aligned mineralization along the Cleopatra structure. This drilling along with the results of processing of the bulk sample, was used to assist in the improvement of grade estimation parameters. It was noted as a part of this testwork, however, that the result of including the new drilling information in the resource estimation further under-estimated the grade in the bulk sample because of this directional bias.

The December 2013 Mineral Resource was estimated using all drilling information (including the underground drilling in the bulk sample area), even though it was fully understood that the resource estimate was biased low in the vicinity of the bulk sample.

The December 2013 Mineral Resource also confirms the contained metal represented by the November 2012 Mineral Resource (within adequate limits) and extends the Mineral Resource based on some new information. The principal difference is slightly less tonnes and higher grade, whilst retaining the contained metal locally (a response to the reduction in smoothing during grade estimation). In addition to the improvements in the model and the comparison with hard data (attained from the processing of the bulk sample), the current study has increased the confidence in the Mineral Resources.

As a result of the increased understanding of the mineralization and increased confidence in the resource estimate, a summary of the feasibility findings has been reproduced in this report. Principally, the findings of the study remain valid as the December 2013 Mineral Resource confirms the contained metal represented by the November 2012 Mineral Resource, with the tonnes decreasing by approximately 5% and the grade increasing by approximately 7%. An amended Feasibility Study for the VOK based on the updated Mineral Resource is expected in the first half of 2014.

Pretivm will continue to advance engineering at the Project in support of the ongoing permitting process, and anticipates filing its application for an Environmental Assessment Certificate in the first quarter of 2014. After obtaining permits, and subject to a production decision, Pretivm anticipates commencing construction of the mine in late 2014 or first quarter of 2015.

1.7 Recommendations

The author makes the following recommendations:

- Update the Feasibility Study to reflect the December 2013 Mineral Resource.
- Complete analysis to determine the optimum drill density for stope definition. (e.g. 7.5 m by 7.5 m or 10 m by 10 m).
- Extend a ramp down to the 1270 m level and open up that level to provide access to complete high density definition drilling down dip of the current underground drilling and along trend to the east.
- Extend a ramp up to the 1390 m level and open that level to provide access to complete high density definition drilling up dip of the current underground drilling and along strike to the west.
- Extend the 1270 m level approximately 400 m to the east and complete resource definition drilling of the far eastern Inferred Resources.
- When planning further drilling programs take into account orientation bias associated with variable vein directions in the mineralized stockwork system.

2 Introduction

2.1 Terms of reference

This NI 43-101 Technical Report has been prepared by Snowden Mining Industry Consultants ("Snowden") for Pretium Resources Inc. ("Pretivm"). The report was prepared to provide an update of the Mineral Resource Estimate of the high grade portion of the gold and silver mineralization for the Valley of the Kings (VOK or VOK Zone) on the Brucejack Property, Skeena Mining Division, BC (the "Brucejack Project", the "Project" or the "Property"). Pretivm has a 100% outright interest in the Property.

In addition, the West Zone estimate created as part of the, April 2012 Mineral Resource (Jones, 2012a), has been documented in this report for completeness. The West Zone estimate has not been updated from the April 2012 Mineral Resource as there has been very little additional drilling in this area.

In 2012, Pretivm commissioned a team of consultants to complete a Feasibility Study in accordance with National Instrument 43-101 (NI 43-101) for the Project. The following consultants were commissioned to complete the component studies for the purpose of the Feasibility Study:

- Tetra Tech: overall project management; mineral processing and metallurgical testing; recovery methods; access infrastructure; internal site roads and pad areas; grading and drainage; ancillary facilities; water supply and distribution; water treatment plant; communications; power supply and distribution; fuel supply and distribution; off-site infrastructure; market studies and contracts; capital cost estimate; processing operating cost estimate; financial analysis; and project execution plan.
- Snowden Mining Industry Consultants Inc. (Snowden): property description and location, accessibility, climate, and physiology, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, adjacent properties, and mineral resource estimates.
- AMC Mining Consultants (Canada) Ltd. (AMC): mining including mine capital and operating cost estimates, mineral reserve estimates.
- Rescan Environmental Services Ltd. (Rescan): environmental studies, permits, and social or community impacts; and tailings delivery system.
- BGC Engineering Inc. (BGC): geotechnical design, mine hydrogeological/groundwater; waste disposal; Brucejack outlet control; environmental water management and water quality, acid rock drainage (ARD) and metal leaching (ML).
- Alpine Solutions Avalanche Services (Alpine Solutions): avalanche hazard assessment.
- Valard Construction (Valard): transmission line.
- Paterson & Cooke Canada Inc. (P&C): paste fill distribution.

Sections 15 to 23 of the current Technical Report are summarised from the Feasibility Study results. The reader is referred to Ireland et al. (2013) for detailed information. The Feasibility Study was based on the previous November 2012 Mineral Resource and will be updated in 2014 to be based on the December 2013 Mineral Resource.

2.2 Sources of information and data used

Pretivm has provided to Snowden the data used as the basis of this report, including geological mapping, sampling, and assay data from various drilling campaigns.

This report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, and public information as listed in the “References” section at the conclusion of this report. Several sections from reports authored by other consultants have been directly quoted in this report, and are so indicated in the appropriate sections. Snowden has not conducted detailed land status evaluations, and has relied upon previous qualified reports, public documents, information available publicly on Mineral Titles Online (an Internet-based mineral titles administration system for British Columbia) and statements by Pretivm regarding property status and legal title to the Property.

2.3 Personal inspections

Ivor Jones, FAusIMM (CP), Executive Consultant, Snowden, Brisbane visited the project site in February 15 to 16 2012, June 3 to 6 2013 and August 16 to 21 2013 and takes overall responsibility for this report.

Harald Muller, MAusIMM (CP), Principal Consultant, Snowden, Brisbane, and Ivor Jones visited the contact mill in Philipsburg, Montana on 22 August 2013. Mr Muller subsequently visited the mill again between 10 November 2012 and 21 November 2012.

Lynn Olssen, MAusIMM (CP), Principal Consultant, Snowden, Perth and Harald Muller visited the project site in August 16 to 21 2013.

In June 2012, Adrian Martínez Vargas (Consultant, Snowden, Vancouver) under the supervision of Mr Jones completed a separate site visit and sample validation.

3 Reliance on other experts

The author has relied upon documentation provided by Pretivm in respect of the status of the Mineral Claims that cover the Brucejack Project. This is described in Section 4.3.

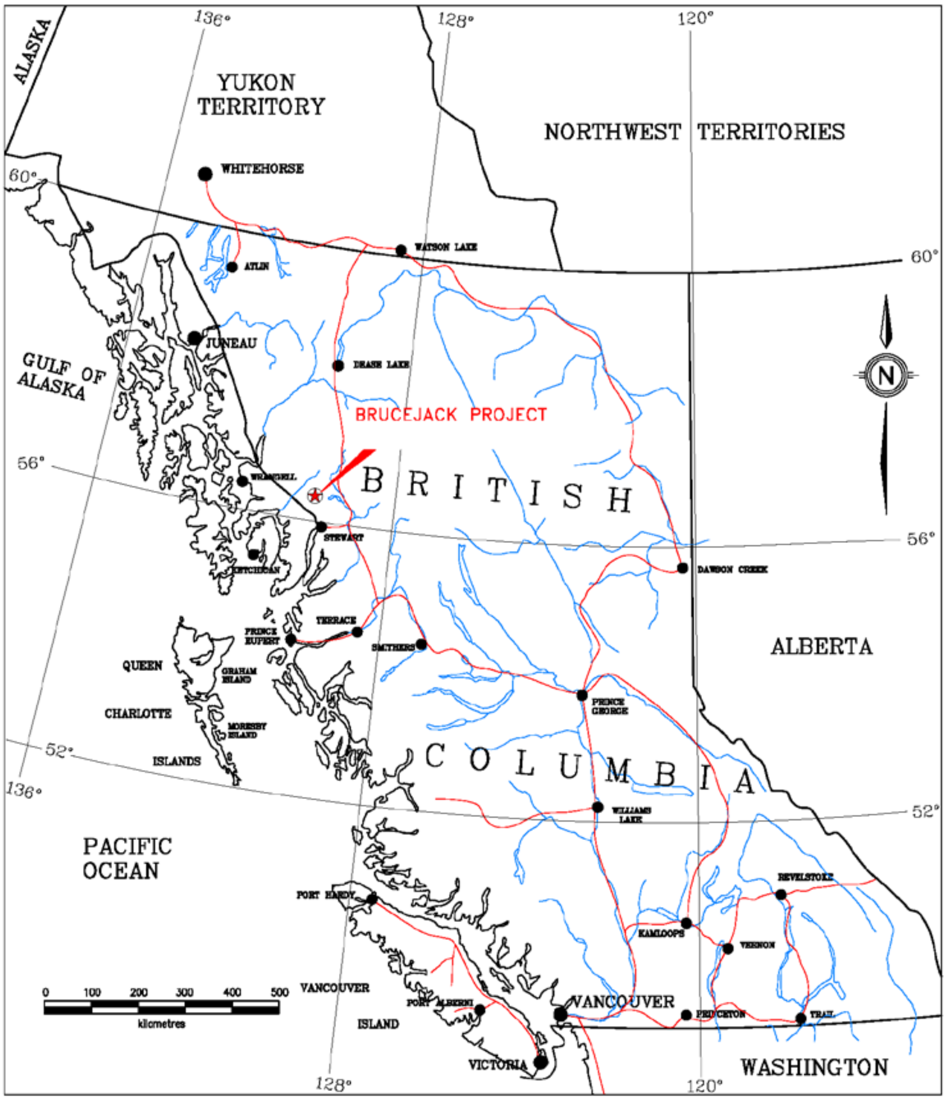
4 Property description and location

Information in this section has been excerpted from Jones (2012c) and Ireland et al. (2013).

4.1 Location

The Property is situated approximately at 56°28′20″N Latitude by 130°11′31″W Longitude (Universal Transverse Mercator (UTM) 426,967E 6,258,719N North American Datum (NAD) 83 Zone 9), a position approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine (Figure 4.1). The Property coordinates used in this report are located relative to the NAD83 UTM coordinate system.

Figure 4.1 Property location map



(Source: Pretivm)

4.2 Tenure

In 2010, pursuant to a purchase and sale agreement between Silver Standard (as the seller) and Pretivm (as the buyer), Silver Standard sold to Pretivm all of the issued shares of 0890693 BC Ltd., the owner of the Brucejack Project and the Snowfield Project. Subsequently, the name of 0890693 BC Ltd. changed to Pretium Exploration Inc.

4.3 Status of mining titles

According to the information available to Snowden, the Brucejack Property consists of six mineral claims totalling 3,199.28 ha in area (Table 4.1, Figure 4.2 and Figure 4.3) and all claims are in good standing until 31 January, 2024.

Table 4.1 List of mineral claims

Tenure No.	Tenure Type	Map No.	Owner	Pretivm Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	428.62
509397	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	375.15
509400	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	178.63
509463	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	482.57
509464	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	1,144.53
509506	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31, 2024	589.78
Total							**3,199.28**

Snowden has relied upon public information, as well as information from Pretivm, and has not undertaken an independent verification of title and ownership of the Property claims. However, information relating to tenure was verified by Snowden to the extent possible by means of the public information available through the Mineral Titles Branch of the BC MEMNG MTO land tenure database. In 2005, the six mineral claims that comprise the Property were converted from 28 older legacy claims to BC's new MTO system.

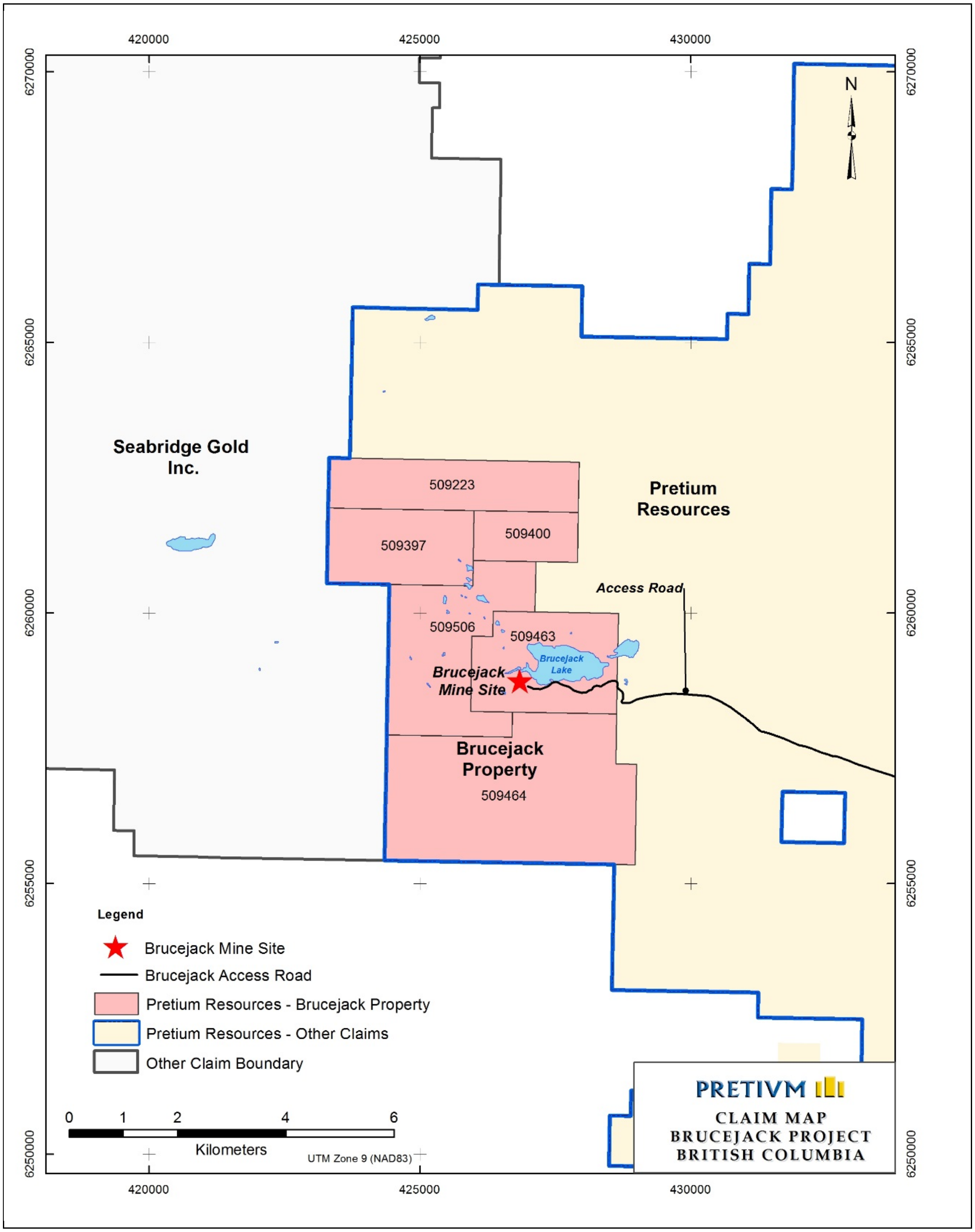
A legal land survey of the claims has not been undertaken.

There are no annual holding costs for any of the six mineral claims at this time, as the claims are paid up until January 31, 2024.

The majority of the Property falls within the boundaries of the Cassiar-Iskut-Stikine Land and Resource Management Plan (LRMP) area, with only a minor south-eastern segment of Mineral Claim No. 509506 falling outside this area. All claims located within the boundaries of the LRMP are considered areas of General Management Direction, with none of the claims falling inside any Protected or Special Management Areas.

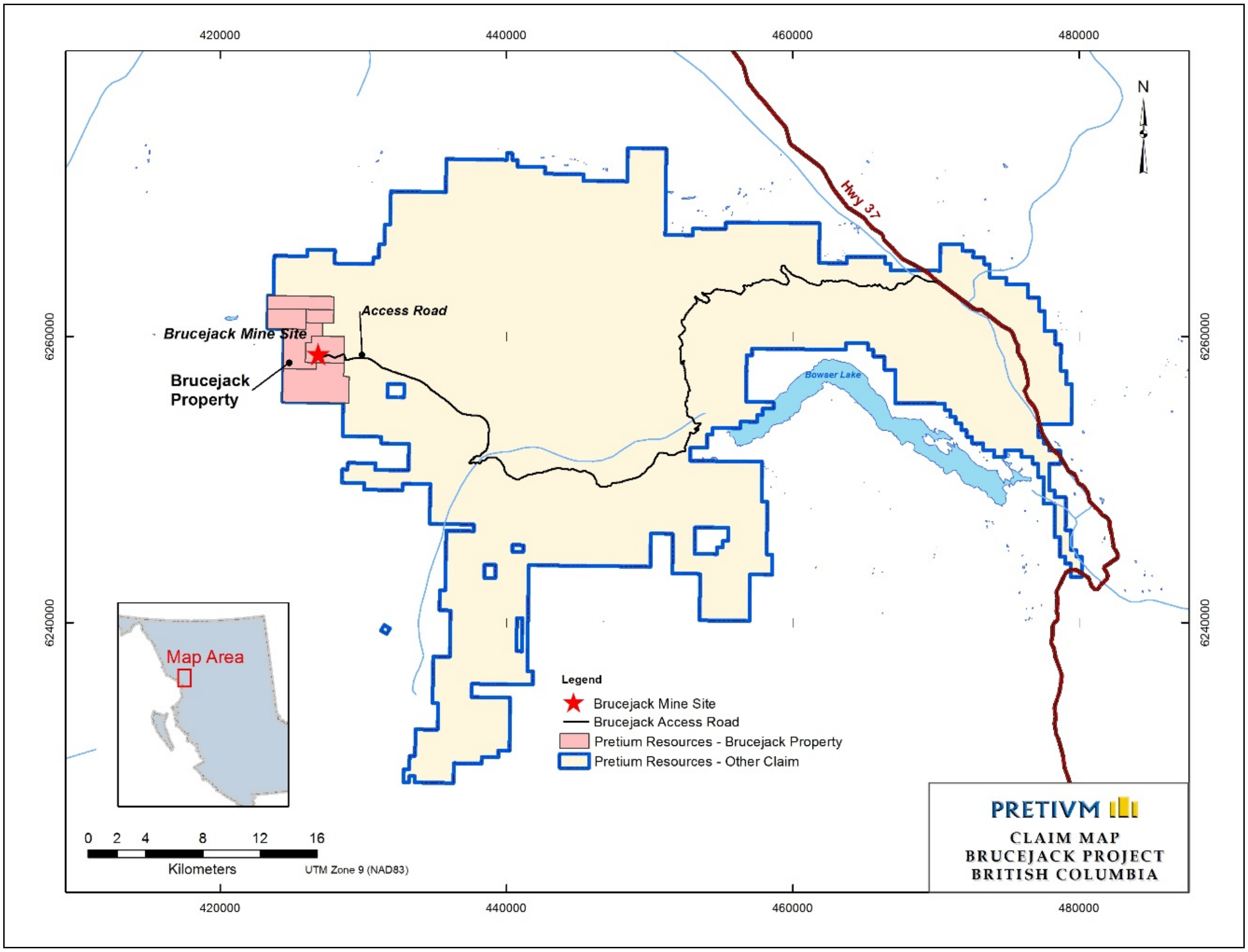
At the time of this report, the land claims in the area are in review and subject to ongoing discussions between various First Nations and the Government of BC.

Figure 4.2 Brucejack property mineral claim



(Source: Pretivm)

Figure 4.3 Pretivm mineral claims



(Source: Pretivm)

4.3.1 Confirmation of tenure

Snowden is not qualified to provide legal comment on the mineral title to the reported properties, and has relied on the provided information. No warranty or guarantee, be it expressed or implied, is made by Snowden with respect to the completeness or accuracy of the tenement description referred to in this document.

4.4 Royalties, fees and taxes

The royalties applicable to the Project are as follows:

- "Royalty" means the amount payable by the Owner, calculated as 1.2% of the NSR, with the following exemptions:
  - gold: the first 503,386 oz produced from the Project
  - silver: the first 17,907,080 oz produced from the Project.

Snowden understands that the 1.2% NSR royalty is, at the time of this report, in favour of Franco-Nevada Corporation.

5 Accessibility, climate, local resources, infrastructure and physiography

Information in this section has been excerpted from Jones (2012c) and Ireland et al. (2013).

5.1 Climate and physiography

The climate is typical of northwestern BC with cool, wet summers, and relatively moderate but wet winters. Annual temperatures range from +20°C to -20°C. Precipitation is high with heavy snowfall accumulations ranging from 10 to 15 m at higher elevations and 2 to 3 m along the lower river valleys. Snow packs cover the higher elevations from October to May. The optimum field season is from late June to mid-October.

5.1.1 Vegetation

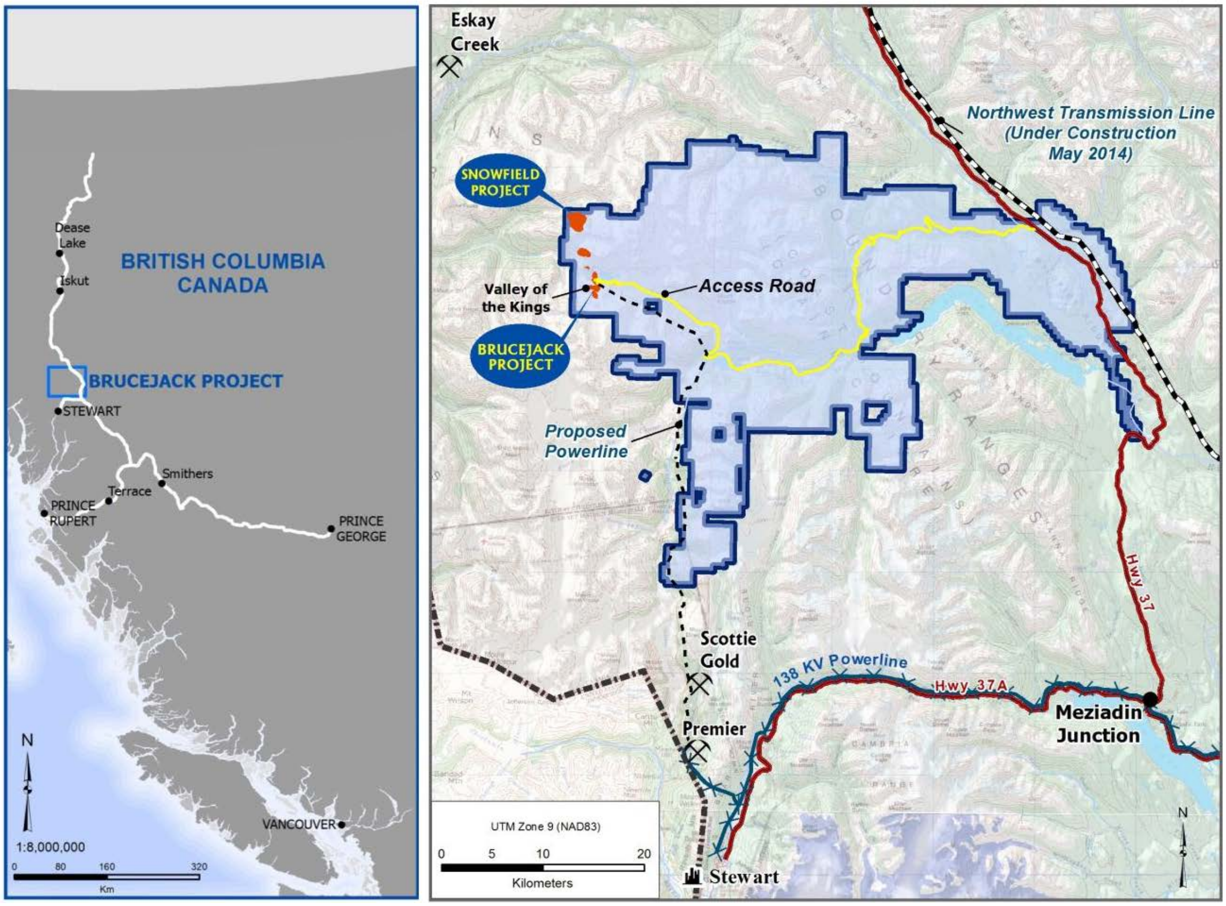
The tree line is at an elevation of approximately 1,200 m. Sparse fir, spruce, and alder grow along the valley bottoms with only scrub alpine spruce, juniper, alpine grass, moss, and heather covering the steep valley walls. The Property, at an elevation above 1,300 m, has only sparse mosses along drainages. Rocky glacial moraine and polished glacial-striated outcrops dominate the terrain above the tree line.

5.2 Accessibility

The Property is located in the Boundary Range of the Coast Mountain Physiographic Belt, along the western margin of the Intermontane Tectonic Belt. The terrain is generally steep with local reliefs of 1,000 m from valleys occupied by receding glaciers, to ridges at elevations of 1,200 masl. Elevations within the Property range from 1,366 masl along Brucejack Lake to 1,650 masl at the Bridge Zone. However, within several areas of the Property, the relief is relatively low to moderate.

Pretivm has completed construction of its 74 km access road that links the Brucejack Camp to Highway 37 via the Knipple Glacier, Bowser Camp and Wildfire Camp (Figure 5.1). Personnel, equipment, fuel, and camp provisions are driven to a staging area on the Knipple Glacier (at km 60), before being taken over the glacier to the Brucejack camp. This has significantly reduced transportation costs. The Property is also easily accessible with the use of a chartered helicopter from the town of Stewart, or seasonally from the settlement of Bell II. The flight time from Stewart is approximately 30 minutes and slightly less from Bell II; however, Stewart has the advantage of having an established year-round helicopter base.

Figure 5.1 Project access



(Source: Pretivm*)*

5.3 Infrastructure

The exploration access road from Highway 37 is complete and in use (Figure 5.1).

There are no local resources other than abundant water for any drilling work. The nearest infrastructure is the town of Stewart, approximately 65 km to the south, which has a minimum of supplies and personnel. The towns of Terrace and Smithers are also located in the same general region as the Property. Both are directly accessible by daily air service from Vancouver.

The nearest railway is the Canadian National Railway Yellowhead route, which is located approximately 220 km to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of BC.

Stewart, BC, the most northerly ice-free shipping port in North America, is accessible to store and ship concentrates. At the time of this report, the Wolverine and Huckleberry mines were shipping material via this terminal.

A high-voltage, 138 kV transmission line currently services Stewart, BC, and has sufficient capacity to provide power to the Project. BC Hydro is completing a facilities study in respect of the interconnection of a transmission line servicing the Project with the 138kv transmission line servicing the town of Stewart, BC (Figure 5.1). The study is expected to be completed in mid-2014.

6 History

Information in this section has been excerpted from Board and McNaughton (2013) and updated.

6.1 Early Exploration

Copper-molybdenum mineralization was discovered on the Sulphurets property by prospectors in 1935 in the vicinity of the Main Copper Zone, approximately six kilometres northwest of Brucejack Lake; however, these claims were not staked until 1960. From 1935 to 1959, the area was relatively inactive with respect to prospecting; however, it was intermittently evaluated by a number of different parties, and resulted in the discovery of several small copper and gold-silver occurrences Sulphurets-Mitchell Creek area. In 1960, Granduc Mines Ltd. (Granduc) and Alaskan prospectors staked the main claim group, covering the known copper and gold-silver occurrences, which collectively became known as the Sulphurets property. This was the start of what could be termed the era of modern exploration (Table 6.1).

Table 6.1 Exploration history of the Sulphurets property between 1960 and 2008

Date	Exploration
1960 to 1979	Granduc continued exploration, conducting further geological mapping, lithogeochemical sampling, trenching, and diamond drilling on known base and precious metal targets north and northwest of Brucejack Lake. This resulted in the discovery of gold-silver mineralization in the Hanging Glacier area and molybdenum on the south side of the Mitchell zone.
1980	Esso Minerals Canada Ltd. (Esso) optioned the Property from Granduc and subsequently completed an extensive program consisting of mapping, trenching, and geochemical sampling that resulted in the discovery of several showings including the Snowfield, Shore, West, and Galena zones. Gold was discovered on the peninsula at Brucejack Lake near the Shore Zone.
1982 and 1983	Exploration was confined to gold- and silver-bearing vein systems in the Brucejack Lake area at the southern end of the property from 1982 to 1983. Drilling was concentrated in 12 silver- and gold-bearing structures, including the Near Shore and West zones, located 800 m apart near Brucejack Lake. Drilling commenced on the Shore Zone.
1983 and 1984	Esso continued work on the property and (in 1984) outlined a deposit on the west Brucejack Zone.
1985	Esso dropped the option on the Sulphurets property.
1985	The property was optioned by Newhawk Gold Mines Ltd. (Newhawk) and Lacana Mining Corp. (Lacana) from Granduc under a three-way joint venture (JV) (the Newcana JV). The Newcana JV completed work on the Snowfield, Mitchell, Golden Marmot, Sulphurets Gold, and Main Copper zones, along with lesser known targets.
1986 to 1991	Between 1986 and 1991, the Newcana JV spent approximately $21 million developing the West Zone and other smaller precious metal veins, on what would later become the Bruceside property. In addition to surface work, a total of 5,276 m of exploratory underground drifting and 33,750 m of underground drilling in 442 drillholes was completed on the West Zone between 1987 and 1990.
1991 and 1992	Newhawk officially subdivided the Sulphurets claim group into the Sulphside and Bruceside properties and optioned the Sulphside property (including the Sulphurets and Mitchell zones) to Placer Dome Inc. (Placer Dome). From 1991 to 1994, joint venture exploration continued on the Bruceside property, including property-wide trenching; mapping; airborne surveys; and surface drilling, evaluating various surface targets including the Shore; Gossan Hill; Galena Hill; Maddux; and SG zones. Newhawk purchased Granduc's interest in the Snowfield Property in early 1992.

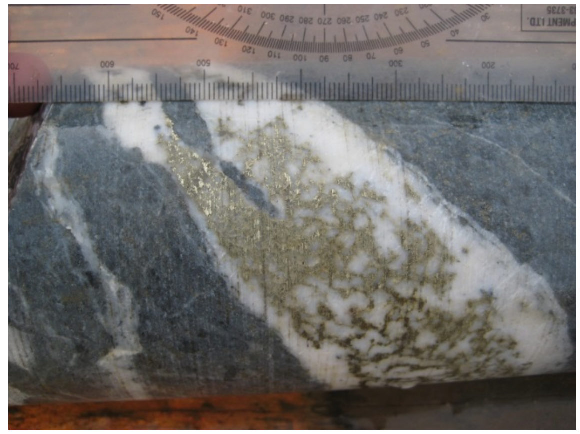
Date	Exploration
1991	Six drillholes were drilled at the Shore Zone, totalling 1,200 m, to test its continuity and to determine its relationship to the West and R-8 zones. Results varied from 37 g/t Au over 1.5 m, to 13 g/t Au over 4.9 m (www.infomine.com).
1994	Exploration in the Bruceside property consisted of detailed mapping and sampling in the vicinity of the Gossan Hill Zone, and 7,352 m of diamond drilling (in 20 drillholes) primarily on the West, R-8, Shore, and Gossan Hill zones. Mapping, trenching, and drilling of the highest priority targets were conducted on 10 of the best deposits (including the West Zone).
1996	Granduc merged with Black Hawk to form Black Hawk Mining Inc. (Black Hawk).
1997 and 1998	No exploration or development work was carried out on the Snowfield and Bruceside properties (Budinski et al. 2001).
1999	Silver Standard Resources Inc. (Silver Standard) acquired Newhawk and with it, Newhawk's 100% interest in the Snowfield property and 60% interest in the Bruceside property (www.infomine.com).
2001	Silver Standard entered into an agreement with Black Hawk whereby Silver Standard acquired Black Hawk's 40% direct interest in the Bruceside property, giving Silver Standard a 100% interest in the Bruceside property, which it subsequently renamed the Brucejack Project.
1999 to 2008	No exploration or development work was carried out on the Snowfield and Brucejack properties during the period from 1999 to 2008.

6.2 Exploration by Silver Standard Resources Inc. (2001-2010)

In 2009, Silver Standard began their first work on the Property following its acquisition. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core. Based on its successful bulk tonnage drilling on the Snowfield Property, Silver Standard designed the 2009 Brucejack drill program test for additional bulk tonnage resources on the Brucejack Property.

The 2009 program tested five zones with 37 drillholes totalling 18,000 m. A total of 12 drillholes were targeted at what would become the VOK. Drillhole SU-012 (Figure 6.1a) is credited as being the discovery drillhole for the VOK intersecting 16,948.5 g/t Au over 1.5 m.

Figure 6.1 Examples of high grade gold intersections in the VOK: dendritic latticework electrum in quartz-carbonate vein in a) discovery drillhole SU-012; b) drillhole SU-084; c) drillhole SU-115; and d) drillhole SU-452; core in photographs is HQ diameter (numbered clockwise starting in upper left)



a. b.

d. c.

(Source: Pretivm)

The 2010 drill program, which totalled 33,480 m in 73 drillholes, was designed to continue definition of the bulk tonnage mineralization as well as to determine the nature and continuity of the high grade mineralization observed at VOK. Approximately one third of the 2010 drilling targeted the VOK and included gold intersections of up to 5,840 g/t Au. The bulk tonnage drilling achieved its intended goal through the definition of more than 20 Moz at Brucejack (8 Moz in Measured and Indicated and 12.5 Moz gold in Inferred, at a 0.3 g/t AuEq cut-off; Ghaffari et al., 2011). The relatively dense drilling from the bulk tonnage drilling program, with drill spacings of 100 m by 100 m to 50 m by 50 m, formed the basis upon which the bulk tonnage resource model was built. Numerous high grade intersections were defined as part of this drilling, allowing for the initial delineation of high grade mineralization trends.

In 2010 Silver Standard proceeded with the sale of the Snowfield and Brucejack projects to a company formed by the former president specifically to acquire the projects (Pretium Resources Inc.).

6.3 Previous Feasibility Studies on the Property (1990)

Corona completed a Feasibility Study on a proposed underground mine with decline access for the Sulphurets Project (West and R-8 Zones only) in 1990. Total operating costs of $145 per tonne were estimated based on a 350 tonne-per-day mill facility for processing, a capital cost of $42.7 million and a 6.7% pre-tax return at a price of US $400/oz gold and $5/oz silver. The study concluded that higher metal prices must be realized before a production decision could be taken.

The reader is cautioned that the above mentioned 1990 Corona Sulphurets Project Feasibility Study is no longer relevant, is not NI 43-101 compliant and should not be relied upon.

6.4 Prior mineral production

In the 1980's, in excess of 5 km of underground ramps, level development and raises were completed on the West Zone down to the 1100 level. In 1993, a Project Approval Certificate was issued in respect of the Project by the Minister of Sustainable Resource Management and Minister of Energy and Mines for the Province of British Columbia. The Mine was not developed and the certificate as amended expired in 2006. No ore had, prior to 2013, been mined or processed from the Property, including the West Zone.

6.5 Preliminary Economic Assessment (2010)

Silver Standard commissioned Wardrop to complete a Preliminary Economic Assessment ("PEA") on the combined bulk-tonnage resources of the Brucejack Project and Snowfield Project in 2010 (Wardrop Engineering Inc., 2010).

The following consultants were commissioned to complete the component studies for the NI 43-101 Technical Report:

- Wardrop: processing, infrastructure, capital and operating cost estimates, and financial analysis
- AMC Mining Consultants (Canada) Ltd. (AMC): mining
- P&E Mining Consultants Inc. (P&E): Mineral Resource estimate
- Rescan Environmental Services Ltd. (Rescan): environmental aspects, waste and water treatment
- BGC Engineering Inc. (BGC): tailings impoundment facility, waste rock and water management, and geotechnical design for the open pit slopes.

Based on the results of the PEA, it was recommended that Silver Standard continue with the next phase - a Pre-Feasibility Study, in order to identify opportunities and further assess bulk-tonnage viability of the Property. This report was reissued for Pretivm in October 2010. The report, however, is no longer current.

7 Geological setting and mineralization

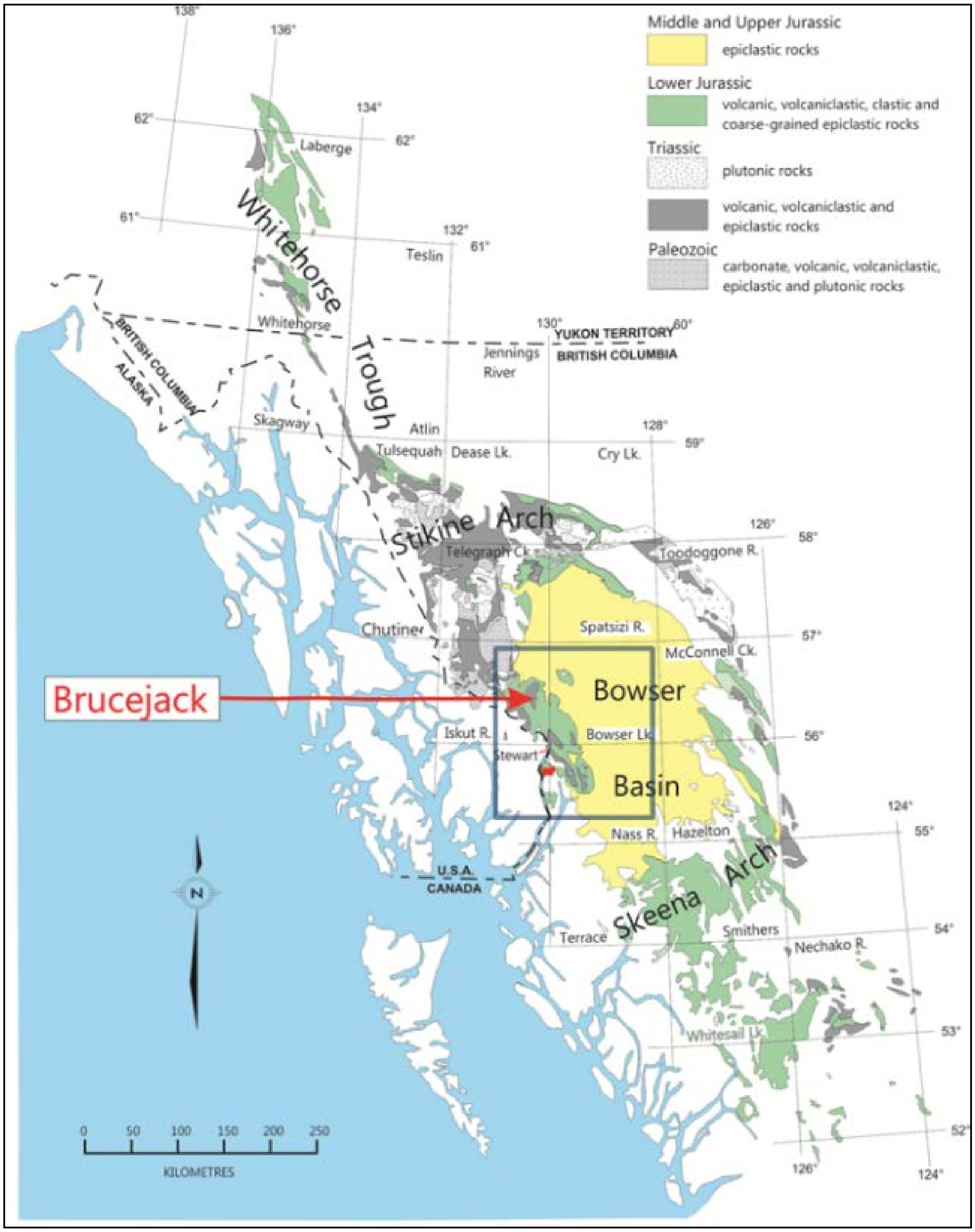
Information in this section has been excerpted and adjusted from Board and McNaughton (2013).

7.1 Regional geological setting

The Brucejack property is located in the western Stikine terrane (Stikinia), the largest and westernmost of several exotic terranes in the Intermontane Belt of the Canadian Cordillera (Figure 7.1). Stikinia is interpreted as an intraoceanic island arc terrane, formed between mid-Palaeozoic to Middle Jurassic time, when it was accreted to the North American continental margin (about 173 Ma; e.g., Nelson and Colpron, 2007; Evenchick et al, 2007; Gagnon et al., 2012). Western Stikina was subsequently strongly deformed during the Cretaceous accretion of the outboard Insular Belt terranes (about 110 Ma; Kirkham and Margolis, 1995). Volcano-sedimentary rocks and related Early Jurassic plutons in the north-west part of Stikina represent an exceptionally metals-rich tectonic assemblage in British Columbia (e.g., Nelson et al., 2013). This area includes volcanogenic massive sulphide deposits (e.g., Granduc, Dolly Varden-Torbrit, Anyox, and Eskay Creek), alkaline porphyry copper-gold deposits (e.g., Kerr, Sulphurets, Mitchell, Snowfield), and transitional epithermal intrusion-related precious metal deposits (e.g., Brucejack, Silbak-Premier, Big Missouri, Red Mountain, and Homestake Ridge) (Figure 7.2).

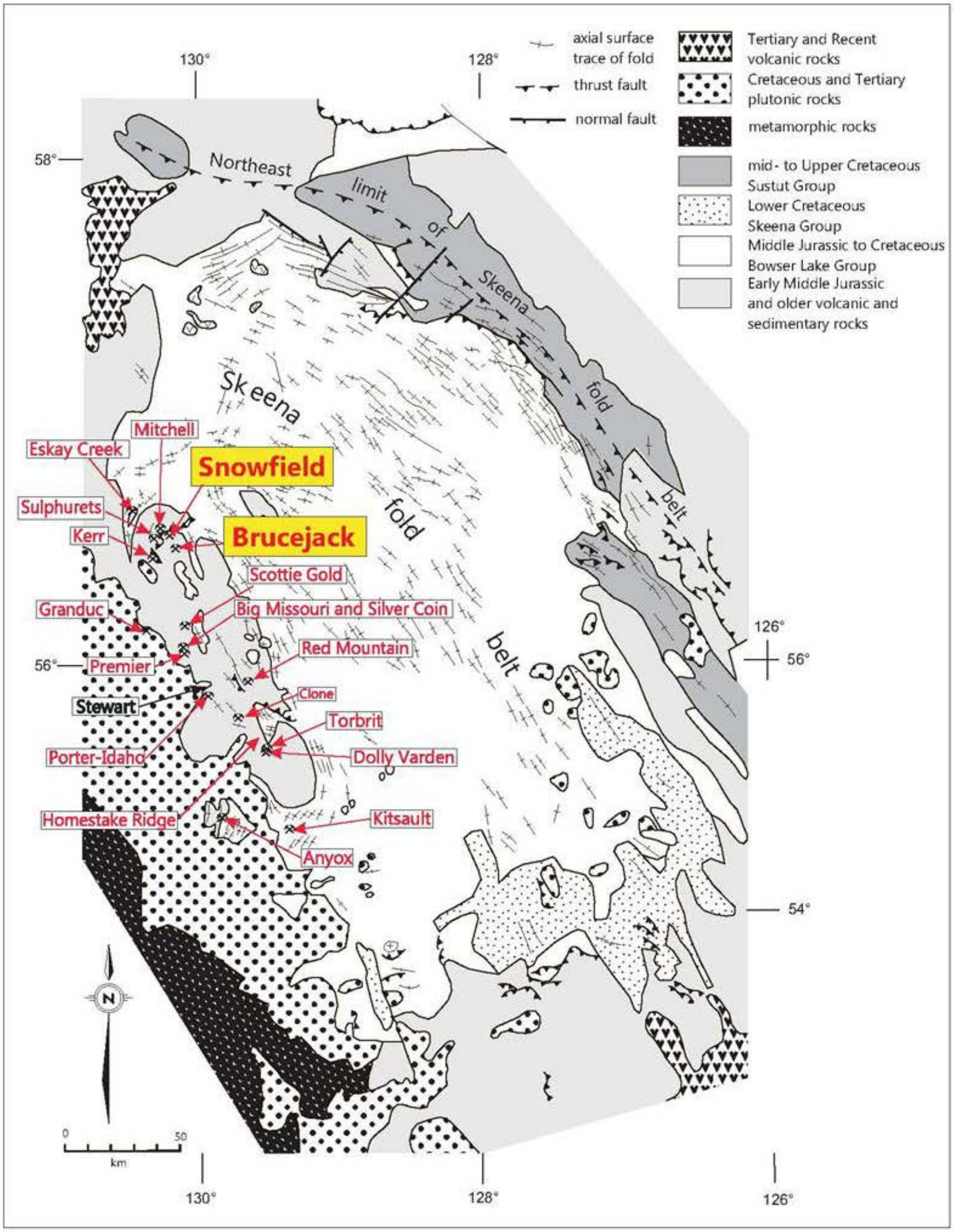
Cretaceous fold-and-thrust belt deformation resulted in, amongst other features, the formation of a major north-northwest trending structural culmination (elongated dome) in the western part of Stikinia (the ‘Stewart-Iskut’ culmination), thereby helping bring the older, mineralized volcano-sedimentary rocks close to surface in this region. The Brucejack property is located on the eastern limb of the McTagg Anticlinorium, the northern closure of the Stewart-Iskut culmination (Figure 7.3). As a result, rocks on the Brucejack property are tilted, as well as folded, and generally display a progressive younging towards the east. Volcanic arc-related rocks of the Triassic Stuhini Group form the core of the anticlinorium, and are successively replaced outwards by volcanic arc-related rocks of the Lower Jurassic Hazelton Group and clastic basin-fill sedimentary rocks of the Middle to Upper Jurassic Bowser Lake Group. The distinctive marine sedimentary and rift-related volcanic rocks of the Upper Hazelton Group (including the Spatisizi and Iskut River Formations; Gagnon et al., 2012) clearly delineate the outline of the anticlinorium on a regional-scale. A major angular unconformity characterizes the contact between the Stuhini and Hazelton Group rocks. Bowser Lake Group rocks exhibit a conformable to disconformable relationship to the underlying Hazelton Group rocks. The McTagg Anticlinorium is cut by a series of thrusts (e.g., south-east directed Mitchell Thrust) of mid-Cretaceous age, and late-stage brittle faults of probable Tertiary age, including the northerly trending Brucejack Fault.

Figure 7.1 Location of Brucejack and Snowfield deposits in the northwest-trending structural culmination of Lower Jurassic Rocks of the Stikine Terrane on the western side of the Bowser Basin



(Source: Pretivm)

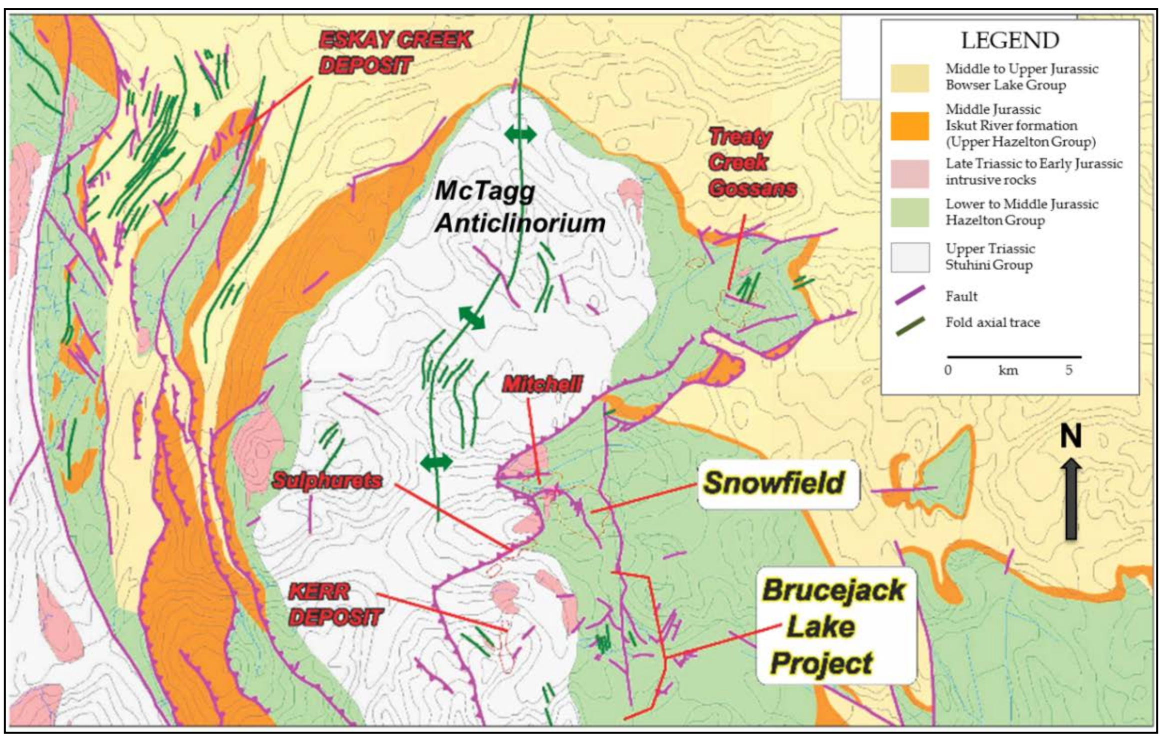
Figure 7.2 Location of Brucejack and Snowfield deposits relative to other metallogenic deposits of the region. Location of inset map shown as rectangle on regional map



Note: Shows significant past-producing mines as well as selected advanced exploration projects. (Source: Ghaffari et al., 2012)

Plutonic rocks of the Mitchell Intrusions (Kerr, Sulphurets, Mitchell, Snowfield), which display a close spatial and temporal relationship to porphyry-style copper-gold-molybdenum mineralization, are located to the west and north of the Brucejack property. U-Pb zircon age dates from various pre-, inter-, and post-mineral intrusive phases of these intrusions suggest an age of between 191-195 Ma for the porphyry style mineralization in these rocks (Kirkham and Margolis, 1995). Large hydrothermal alteration haloes are developed around the intrusive complexes. Potassic alteration is closely associated with copper and gold mineralization in the Mitchell Intrusions and adjacent Stuhini Group Rocks, and is variably overprinted by propylitic and chlorite-sericite alteration. Quartz-sericite-pyrite (sericitic) alteration is widely developed in the rocks of the Stuhini and Hazelton Groups to the east of the Mitchell Intrusions, as well as occurring as a pervasive overprint to earlier alteration in and around these bodies. The multiple stages of alteration indicate telescoping of the porphyry system. Quartz-sericite-pyrite alteration is the predominant style of alteration on the Brucejack property

Figure 7.3 Local area geology showing location of the Brucejack project on the eastern limb of the McTagg Anticlinorium, in close proximity to the Kerr-Sulphurets-Mitchell intrusions, the unconformity between the Triassic Stuhini and overlying Lower Jurassic Hazelton Group rocks, and the Brucejack Fault



(Source: Pretivm)

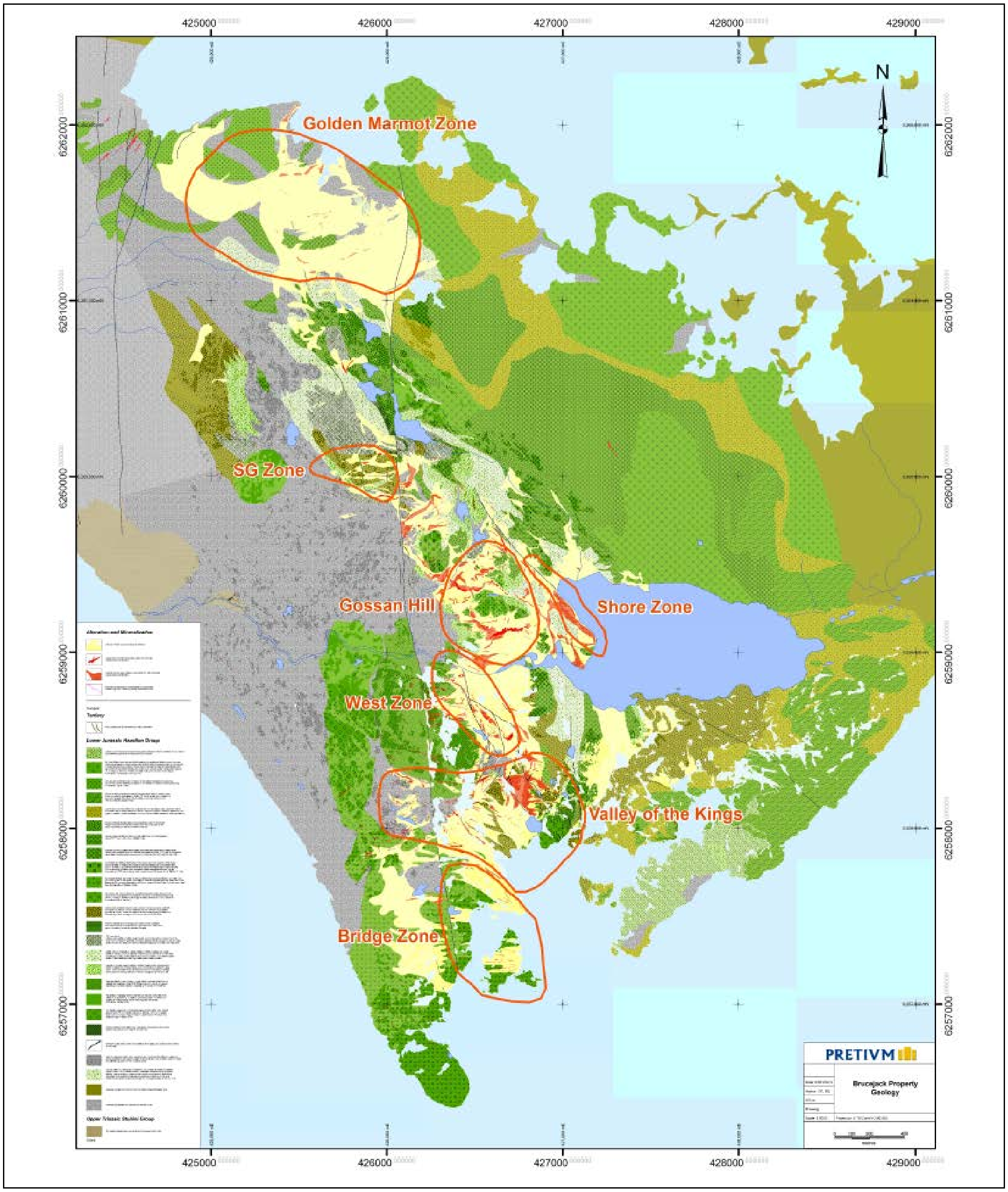
7.2 Brucejack property geology

The Brucejack property is largely underlain by volcano-sedimentary rocks of the Lower Jurassic Hazelton Group. These rocks unconformably overlie volcanic arc sedimentary rocks of the Upper Triassic Stuhini Group along the westernmost part of the property. Hazelton Group rocks on the property include hornblende and/or feldspar-phyric volcanic (latite) flows, and pyroclastic fragmental rocks, locally derived heterolithic volcanic pebble to boulder conglomerate, volcanic sandstone, siltstone and mudstone. The Hazelton Group volcano-sedimentary rocks are interpreted as having been deposited in an extensional structural regime, proximal to a basin margin growth fault (or series of growth faults), owing to the presence of complicated lateral facies variations within the volcanic pile and the diachronous and immature nature of the rock units. The Brucejack Fault, which forms a distinct topographical feature across the property, is currently interpreted as being the reactivated expression of these basin margin structures.

Alteration on the Brucejack property is characterized by variable, but generally intensely quartz-sericite-pyrite altered rocks that define a distinctive and continuous north-south arcuate (west-concave) band of gossanous rocks that is up to 0.5 km wide, and approximately 5 km in strike (north-south) extent (Figure 7.4). Quartz-sericite-pyrite alteration is broadly spatially associated with the Brucejack Fault and the unconformity between the Stuhini and Hazelton Group rocks, which suggests that these structures (the palaeo-growth fault zone in the case of the Brucejack Fault) may have acted as important fluid conduits during hydrothermal alteration and mineralization.

Gold (± silver) mineralization is hosted in predominantly sub-vertical vein, vein stockwork, and subordinate vein breccia systems of variable intensity, throughout the alteration band. The stockwork systems display both parallel and discordant relationships to stratigraphy, The stockwork systems are relatively continuous along strike (several tens of metres to several hundreds of metres) and are characterized by the presence of millimetre- to decimetre-scale transitional to epithermal veins of pyrite, quartz, quartz-carbonate, and, less commonly quartz-adularia (e.g., in the West Zone) that form cross-cutting networks of variable intensity within stockwork zones. Mineralization recognized in different stockwork vein generations across the property includes: pyrite, tetrahedrite, tennantite, chalcopyrite, galena, sphalerite, molybdenite, arsenopyrite, pyrrhotite, pyrargyrite, polybasite, acanthite, native silver, native gold, and electrum. Alteration, mineralization and vein texture variations across the property suggest spatial and/or temporal down-temperature thermal gradients towards the east (i.e., up stratigraphy) and north. Several mineralization zones have been explored to varying degrees, including (from south to north): Bridge Zone, VOK, West Zone, Gossan Hill, Shore Zone, and SG Zone (Ireland et al., 2013). There are numerous relatively unexplored mineralization showings within the alteration band across the property that are between the main mineralization zones, highlighting the exceptional exploration potential of the Brucejack property.

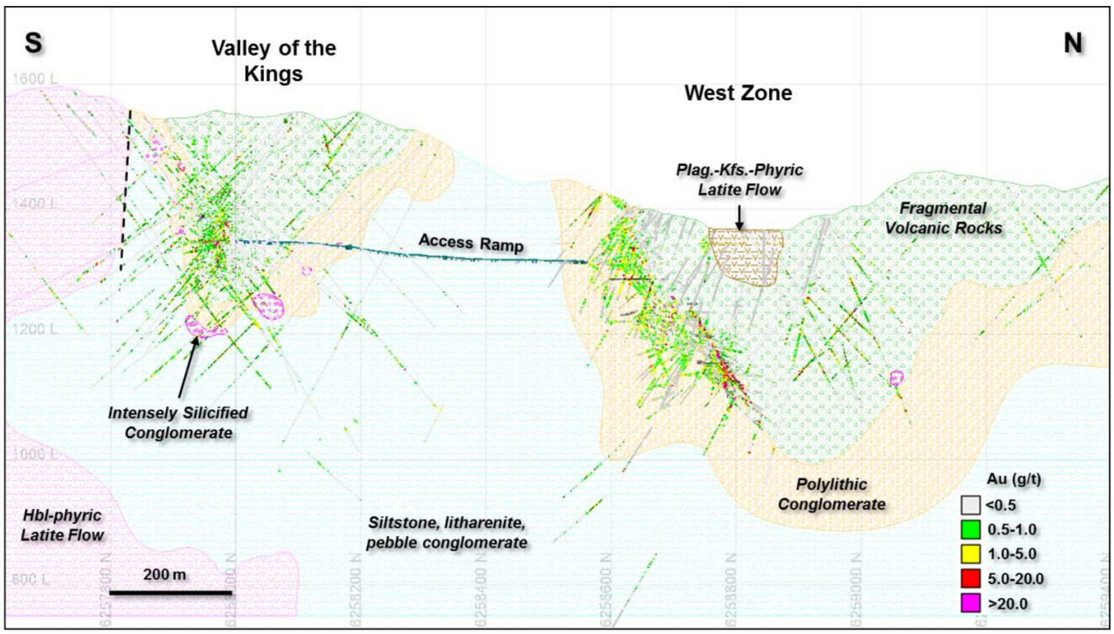
Figure 7.4 Geological map of the Brucejack property showing location of defined mineralized zones and their association with the arcuate band of quartz-sericite-pyrite alteration (shown in yellow)



(Source: Pretivm)

Hazelton Group rocks and mineralized vein stockworks on the Brucejack property display significant multi-phase post-mineralization deformation. A series of variably and locally doubly-plunging tight to open map-scale folds, host tight south-vergent meso- and parasitic scale folds coincident with a regional penetrative east-west trending foliation (e.g. Figure 7.5). Fabric development and folding was most intense in the quartz-sericite-pyrite alteration band, indicating preferential strain partitioning in these less competent rocks during deformation. The plunge of the minor folds varies and several lines of evidence suggest that they may reflect refolding of northerly trending tight, upright "early" mid-Cretaceous Skeena Fold Belt-age folds across roughly east-west axes. These later folds and the east-west foliation are spatially associated with the area of the footwall (and immediate hanging wall) of the regional-scale Mitchell thrust system, suggesting that they may have developed in the latter stages of Skeena Fold Belt development. Late stage north-south trending brittle faults are indicated by pronounced surface lineations.

Figure 7.5 General cross-section (south-north) from the VOK to West Zone



High grade gold mineralization in the VOK, the current focus of the Brucejack project, occurs in a series of west-northwest (and subordinate west-southwest) trending sub-vertical corridors of structurally reoriented vein stockworks and vein breccias, hosted in a folded sequence of fine grained volcanic siltstone and sandstone, variably silicified polylithic volcanic conglomerate, and latite fragmental rocks. Stockwork mineralization displays both discordant and concordant relationships to the volcanic pile stratigraphy. Relatively massive latite flows are present to the immediate north and south of the mineralization host rock sequence. Gold is typically present as gold-rich electrum within deformed quartz-carbonate (±adularia?) vein stockworks, veins, and subordinate vein breccias, with grades ranging up to 41,582 g/t Au and 27,725 g/t Ag over 0.5 m. Example plan and cross section views of the VOK are presented in Figure 7.6 and Figure 7.7. Readers interested in the mineralization details of the other zones on the Brucejack Property are referred to Ireland et al. (2013).

Figure 7.6 Plan view of the 1450 ±100 m level, VOK

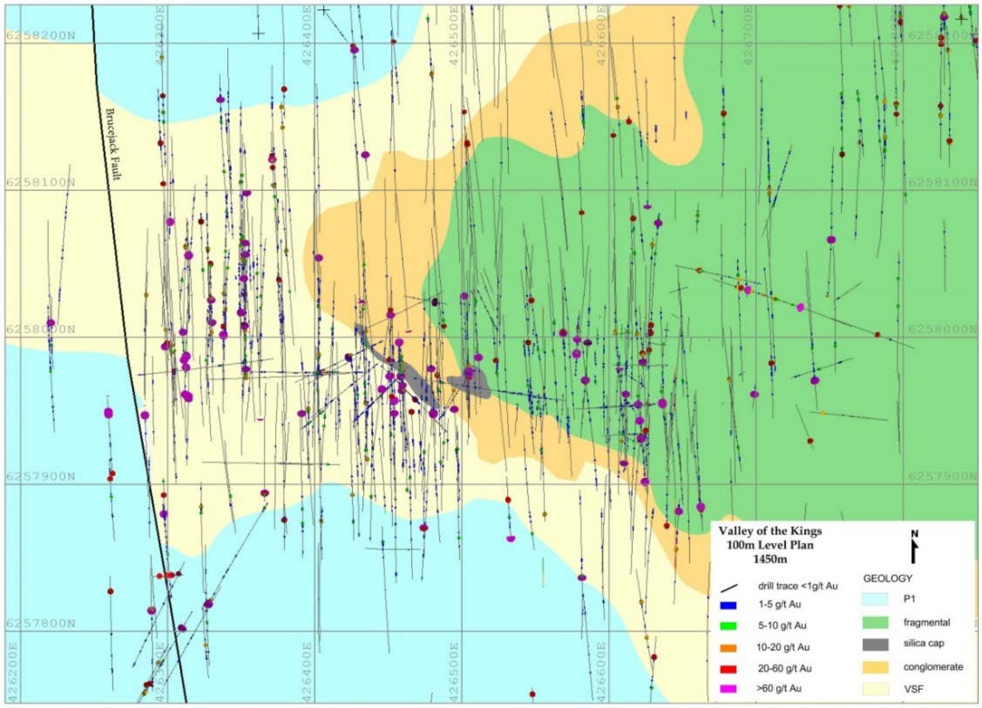
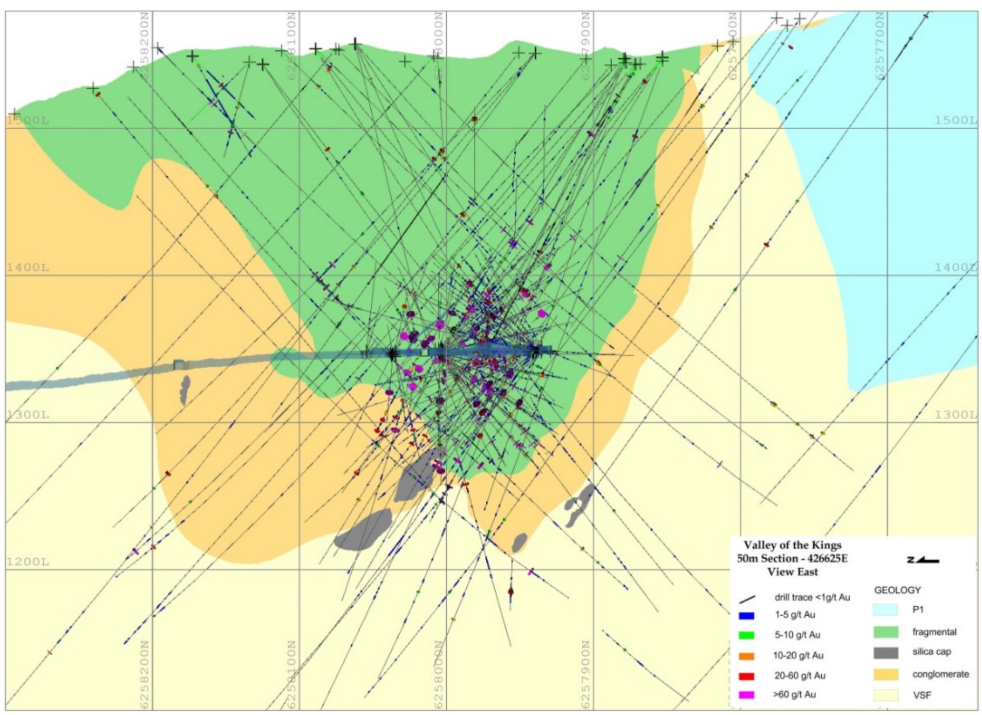


Figure 7.7 North-south cross section along 426625 ±50 mE, VOK



In addition to confirming the presence and dominant west-northwest trend of the deformed electrum-bearing stockwork vein system, recent underground exploration carried out as a part of the bulk sample (Section 9.4) in the VOK also resulted in the recognition of sub-vertical north-northeasterly trending deformed, curviplanar, and sheared quartz-carbonate veins containing abundant visible electrum. These relatively rare structures are interpreted as structurally-controlled fluid conduits that were active during development of the porphyry system and associated volcanic pile in the early Jurassic, and which were reactivated during Cretaceous deformation.

During the recent underground exploration, detailed geological mapping was conducted on all new underground workings from the start of the VOK access ramp to the bulk sample area (Section 9.4). Key outcomes of the underground mapping included:

- Confirmation and slight modification of the location and contacts of lithological units and corridors of stockwork-style mineralization in this part of the deposit as modelled from the drilling data alone (contacts were often within a metre or two of the modelled positions; Figure 9.4).
- Confirmation of the modelled stratigraphy in the bulk sample area. Steeply dipping latite fragmental rocks (lapilli tuff, tuff breccia, and ash (crystal) tuff) are the dominant lithologies in the bulk sample area. A thin reworked volcanic sandstone (epiclastic) unit forms the base of the fragmental rocks in the south of the bulk sample area. Partially silicified polylithic conglomerate underlies the sandy unit in the southernmost limits of the bulk sample area. Pod-like zones of intensely silicified conglomerate are associated with intense asymmetrical sericite and pyrite alteration.
- Confirmation of the repetition through folding of the volcanic and volcaniclastic mineralization host sequence and mineralized stockwork systems seen on the deposit to property scale.
- Confirmation of the presence of a pre-mineralization generation of widespread but discontinuous and narrow pyrite-chlorite-(quartz) veins.
- Confirmation of the presence and association of electrum with deformed quartz-carbonate veins, sheeted veins, vein breccias, and vein stockworks within intensely quartz-sericite-pyrite altered volcanic and volcaniclastic rocks. Confirmation of the vein paragenesis modelled for the deposit as generated from drillhole core logging.
- Good correlation between mapped occurrences of visible electrum and the relative grade distribution of the bulk sample area as modelled in the November 2012 mineral resource estimate (i.e., higher grade in the eastern parts and lower grade in the western parts of the area).
- Visible electrum was noted in five styles of deformed quartz-carbonate veins, sheeted veins, vein breccias, and stockworks. Significant visible electrum was noted in both dominant west-northwest trending veins and in subordinate west-southwest, east-west and north-south (including north-northwest to north-northeast) trending veins.
- One of the subordinate north-south components of the electrum-bearing vein system, known as the “Cleopatra structure” (north-south to north-northeast-south-southwest), is considered as being a part of the broader stockwork mineralizing system, rather than being a discrete and isolated feature for the following reasons:
  - There is a lack of obvious cross-cutting relationships between the uncommon north-south and the dominant east-west (*sensu lato*) trending components of the vein system, both in the bulk sample area and in surface outcrop. North-south and east-west veins appear to merge.

- There are similar styles and abundances of electrum mineralization, gangue mineralogy, and associated alteration between the east-west and north-south trending components of the vein stockwork system. Abundant dendritic electrum hosted in pink manganoan calcite veins (e.g., the Cleopatra structure) is also present in the broader east-west (both west-northwest and west-southwest) veins.
- The north-south and east-west veins are hosted in rocks of Lower Jurassic age and are both cut by deformed post-mineral dykes, also of Lower Jurassic age.
- The north-south and east-west veins in the bulk sample area as well as in surface outcrop exhibit a similar style and degree of deformation.

- Demonstrated along-strike, across-strike, and vertical continuity on the scale of tens of metres of mineralized stockwork vein systems and of the mineralization within them. This was accomplished for both the dominant west-northwest and the subordinate north-south components of the system, through crosscut, drift, bench, breastwork, and raise mining.
- Confirmed the broad stratigraphic control on mineralization and alteration. Silicification tends to follow stratigraphic contacts. Vein stockwork development is best developed in the vicinity of stratigraphic contacts, particularly those associated with intensely silicified host rocks (e.g., the contact between the overlying volcanic fragmental rocks and the underlying polymictic conglomerate in the southern parts of the bulk sample area).
- Confirmed the post-mineralization and pre-deformation relationship of volumetrically minor dykes of intermediate composition, which had been used to bracket the age for mineralization.
- Provided additional information on post-mineralization compressional deformation, which included folding, both left- and right-lateral shearing, and top-to-the-north thrusting in the bulk sample area. Reactivation of older (syn-mineralization) structures is clear on steep north-south structures. The Cleopatra part of the vein system is sheared, buckled, and disaggregated in places, where a later steeply dipping left-lateral north-south trending shear propagated along a pre-existing north-south trending plane of weakness. Displacement of mineralized stockwork along steep shears, moderately dipping thrust faults, and less common normal faults in the bulk sample area appears to be on the order of only several metres.
- Confirmation of the presence of two generations of post-mineralization veining, including: relatively continuous gentle to moderately dipping quartz-carbonate-chlorite shear-veins and associated (albeit less continuous) generally flat-lying tension gashes; and sub-horizontal quartz-(carbonate) extensional veins.

The VOK deposit is currently defined over 1,200 m in east-west extent, 600 m in north-south extent, and 650 m in depth. The West Zone appears to form the northern limb of an anticline that links up with the VOK in the south, and the southern limb of a syncline that extends further to the north (Figure 7.5). This zone, which is currently defined over 590 m along its northwest strike, 560 m across strike, and down to 650 m in depth, is open to the northwest, southeast, and at depth to the northeast.

8 Deposit types

Host rock and mineralization ages determined from the Brucejack property overlap with dates determined for known porphyry copper-gold deposits in the Intermontane Belt, particularly those of the nearby Kerr-Sulphurets-Mitchell (KSM) deposits. The spatial, stratigraphic, and geochronological association between the Brucejack deposits and the KSM porphyry intrusive rocks suggest a genetic link between the high grade gold mineralization at Brucejack and the KSM deposits. However, age constraints on the Brucejack hydrothermal system indicate it may have been driven, in part, by a somewhat younger and relatively long-lived porphyritic stock, or a series of successive porphyritic stocks, emanating from the same deep-seated arc-related magmatism that led to the formation of the slightly older KSM deposits.

Mineralized zones on the Brucejack property are considered to represent a deformed porphyry-related transitional to intermediate sulphidation epithermal high grade gold-silver vein, vein stockwork and vein breccia system that formed between approximately 192-190 Ma and 184 Ma (Figure 8.1 and Figure 8.2).

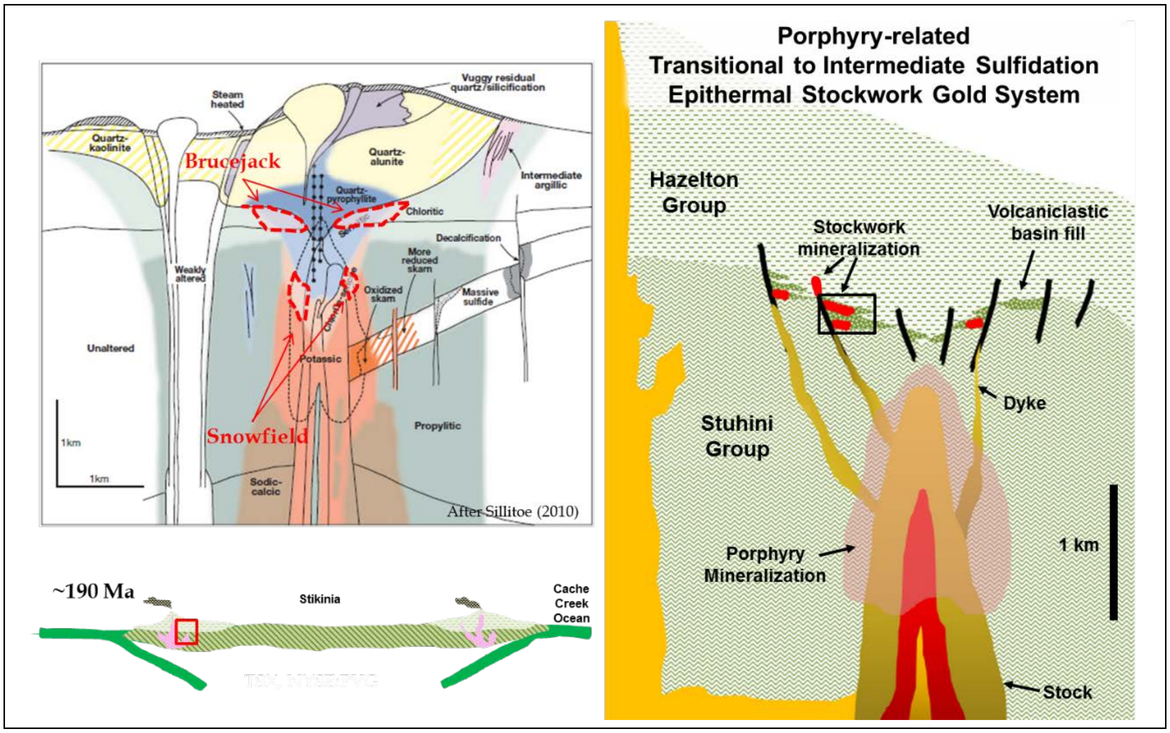
Initial disseminated mineralization and sulphidation of the host rocks occurred within the evolving intra-arc basin, with fluid flow along basin normal faults and along permeable stratigraphic boundaries (Figure 8.2, top left). Progressive development and telescoping of the porphyry system in the volcanic pile resulted in a widespread zonation of porphyry-style alteration and mineralization, and multiple stages of vein and alteration overprinting (Figure 8.2, top right).

Gold concentration and subsequent deposition probably occurred as a result of complex interactions between various physicochemical parameters (e.g., pressure, temperature, pH, activities of oxygen, sulphur, and other volatiles, concentration of dissolved salts, differential permeability of the volcanic pile) in the magmatic-heated seawater hydrothermal system developed above the pulsing porphyry system. Metal deposition was likely triggered by a combination of structural preparation (extensional deformation), depressurisation, cooling, phase separation, solution mixing, and fluid-host rock interactions.

Later intrusion of mafic dykes occurred during the waning of the porphyry system and possible further arc-related deformation (Figure 8.2, bottom left).

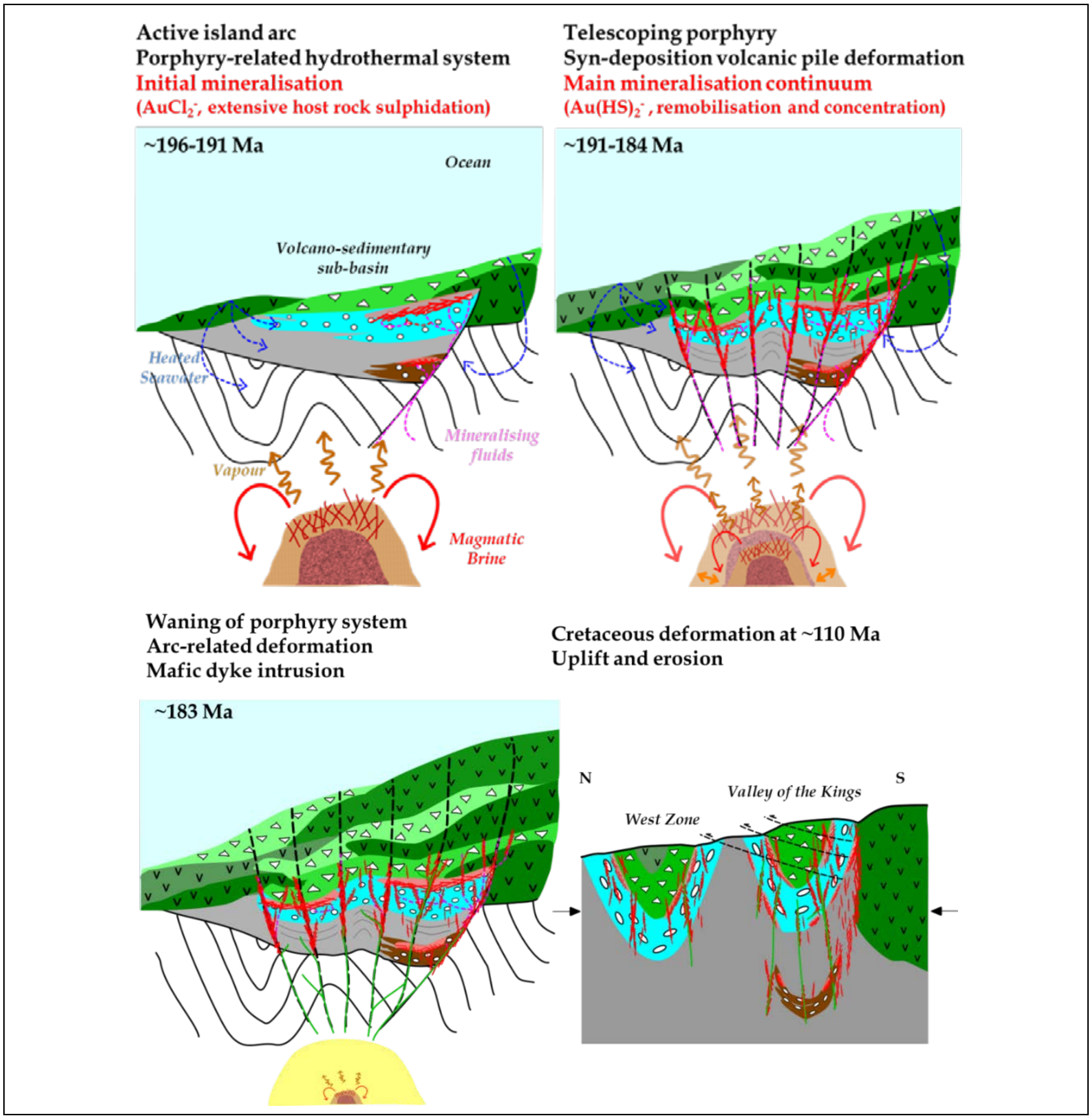
Figure 8.2 (bottom right) shows a schematic of the current deposit looking east. Significant deformation of strata arises from the Cretaceous accretion of the outboard Insular terrane, resulting in transposition/steepening up of competent silicified zones and stockworks due to marked competency differences compared to the intensely altered volcano-sedimentary host rock. Limited local physical remobilization and slickenslide development in gold patinas occurred during the Cretaceous deformation.

Figure 8.1 Schematic showing relative position of the Brucejack deposit to a porphyry copper-gold system



(Source: Pretivm)

Figure 8.2 Schematic showing geological evolution of the Brucejack property



Note: Red highlights zones of high grade mineralization.
(Source: Pretivm)

9 Exploration

9.1 2011

Surface diamond core drilling was the main form of exploration conducted on the Brucejack Property by Pretivm in 2011. Additional Brownfields exploration included detailed surface geological mapping, limited surface sampling, and limited geophysics (Spartan magnetotelluric survey; see Ireland et al., 2013).

In 2011 following the acquisition of the Brucejack Project in late 2010, Pretivm management decided to shift the exploration focus from the open pit bulk-tonnage approach in favour of an underground high-grade mining approach. A bulk-tonnage resource update was released in February 2011 with a high-grade sensitivity for the VOK.

The 2011 drill program consisted of 178 drillholes totalling 72,805 m. The program targeted previously defined high grade intersections primarily in the VOK (60% of the total), but also in the Gossan Hill, Shore, West, and Bridge zones. This drilling led to improved definition of the mineral corridor domains at the VOK. The geological model was tested and iteratively refined through drillhole core logging, detailed geological re-mapping of the property incorporating drillhole data and geological interpretations, and the completion of several geochemical (immobile trace elements) and spectral (clay alteration) analytical studies.

A total of 21 intersections that assayed greater than 1,000 g/t Au were defined during the 2011 drilling program, including the then record for the property of 18,754.5 g/t Au, which was intersected in drillhole SU-115 (Figure 6.1c). The average rate of intersecting a multi-kilogram gold assay during the three drilling programs of relatively widely spaced drilling conducted between 2009 and 2011 worked out to be one >1,000 g/t Au assay per 3,500 m of drilling.

Later in the year Pretivm decided to dewater the historical West Zone underground development to assess the condition of the workings and determine if they could be used as a launching point for a development drive to the VOK. This work was completed through the winter of 2011/12 and marked the beginning of year-round operations at Brucejack. During the autumn of 2011, work was initiated on the development of a 75 km access road to the project from Highway 37. The access road was subsequently completed in early 2013.

9.2 2012

Surface diamond core drilling was again the main exploration tool in 2012. Detailed Brownfields surface geological mapping and associated supplementary surface geochemical sampling was continued on the Brucejack Property.

The 2012 drilling program was primarily focused on upgrading the level of confidence in the existing Mineral Resource estimate for the VOK through closer-spaced infill drilling. Other potential high grade structures, including the down dip extension of the West Zone, were also tested as part of this program.

In 2012, drilling resumed in early March 2012 and continued through the end of September with up to nine drills working. A total of 301 drillholes were completed, totalling 105,500 m of drilling during the 2012 drilling program. Zones within 150 m of surface were drilled at 12.5 m centres, with the deeper parts (down to about 350 m below surface) being drilled at approximately 25 m centres. Drilling at greater depths was generally only able to reliably achieve 50 m centres. The 2012 drilling program resulted in an increase in Indicated Mineral Resources (Jones, 2012b). The drilling program intersected a total of 49 samples which assayed greater than 1,000 g/t Au. The highest value intersected was a 0.5 m interval in drillhole SU-452 which assayed 41,582 g/t Au (Figure 6.1d). The infill drilling program intersected multi-kilogram gold mineralization at a higher frequency (one >1,000 g/t Au assay every 2,150 m of drilling) than the 2009-2011 drilling programs (see above).

The results of the 2012 drilling were incorporated into a revised Mineral Resource estimate (Jones, 2012b). The November 2012 Mineral Resource for the West Zone and the VOK Zone (Jones, 2012b) was estimated to contain over 9.4 Moz of gold and 49 Moz of silver in the Measured and Indicated category, and 3.7 Moz of gold and 13 Moz of silver in the Inferred category, using a 5 g/t Au equivalent cut-off. This resource estimate formed the basis for a Feasibility Study on the Brucejack property, which was completed in June 2013 (Ireland et al., 2013). The Feasibility Study reported Probable Mineral Reserves of 6.6 Moz of gold (15.1 Mt grading 13.6 g/t Au) for the VOK (Section 15).

9.3 2013

The 2013 surface diamond drill program focused on further defining the high grade resource at the VOK and further targeting of the geological and structural features believed to be associated with gold mineralization. A total of 24 surface diamond drillholes (5,200m) were completed in drillholes SU-590 to SU-626. Surface geological mapping and supplementary surface geochemical sampling was continued albeit with a more Greenfields exploration goal (i.e., focussing on the broader area within Pretivm's claims) than in previous years.

9.4 2013 bulk sample

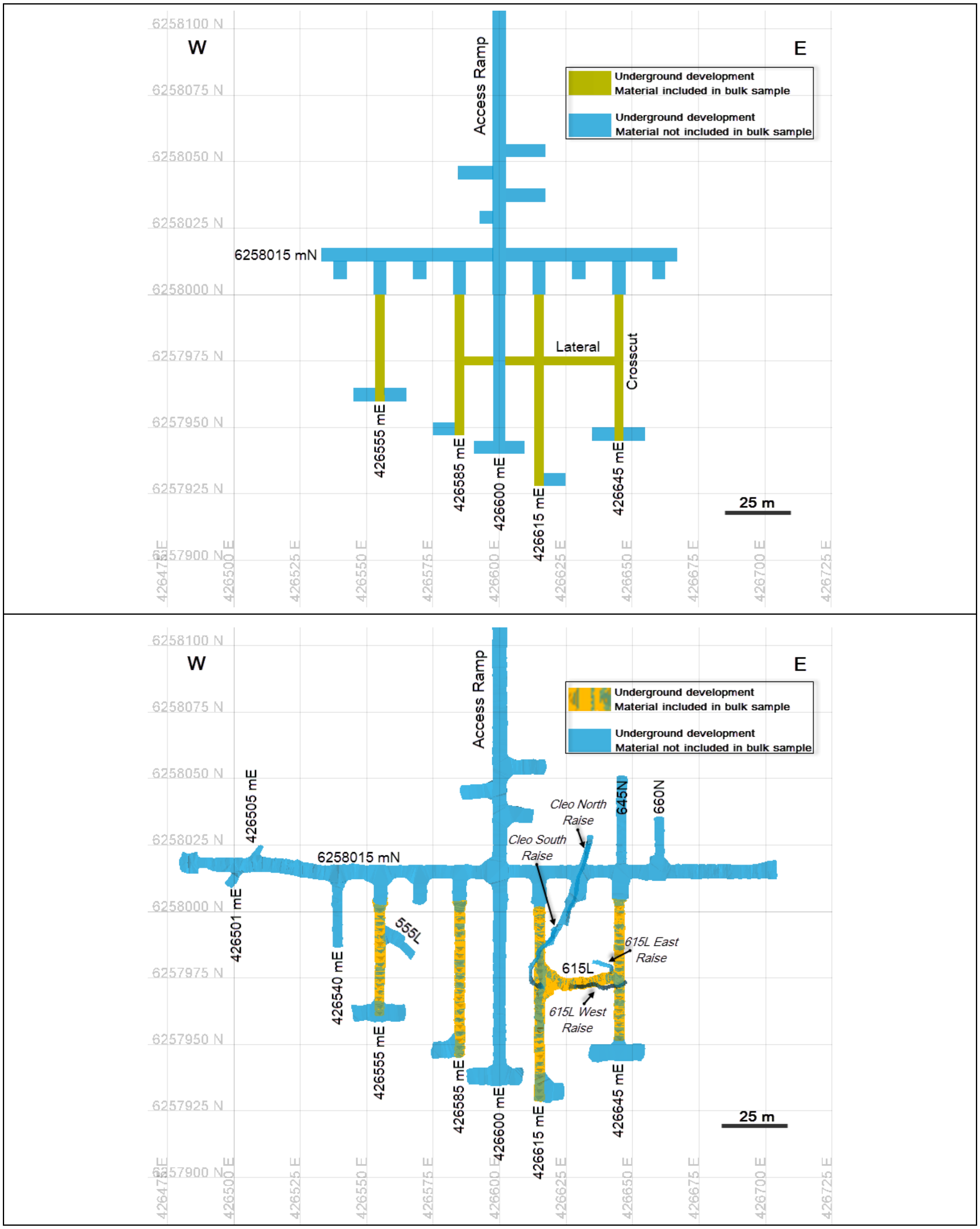
Pretivm elected to extract a bulk sample to further evaluate the geological interpretation and Mineral Resource estimate for the VOK deposit. Selection of a suitable location of the bulk sample required a detailed review of the November 2012 Mineral Resource estimate, consideration of potential future mine plans, and compliance with Section 17(3) of the Mineral Tenure Act Regulation of the Province of British Columbia. Section 17(3) of the Act states that "…a bulk sample of up to 10 000 tonnes of ore may be extracted from a mineral claim not more than once every 5 years". Pretivm decided upon the current location of the bulk sample after considering several possible locations. The original bulk sample location and layout was discussed and prepared in agreement with the input of independent consultants. The bulk sample location was selected such that:

- Mineralization in the bulk sample area was representative of the style of stockwork mineralization in the VOK deposit.
- Drilling density in the bulk sample area was representative of the average drilling density informing "Indicated" Mineral Resources in the November 2012 Mineral Resource estimate.
- The grade distribution of the drilling composites informing the Mineral Resource estimate in the vicinity of the bulk sample area was representative of the grade distribution of all drilling composites used for the global November 2012 Mineral Resource estimate.

- The November 2012 Mineral Resource estimate in the bulk sample area was representative of the average grade of the global VOK deposit above a 5 g/t AuEq cut-off.
- The material mined from the bulk sample area was representative of standard run-of-mine material for metallurgical testing purposes.
- The bulk sample area should sample waste material, as well as low to high grade mineralization, so as to test the predictability of the Mineral Resource estimate across the grade spectrum.

The original bulk sample layout is presented in Figure 9.1. The original plan was to mine four crosscuts (426555 mE, 426585 mE, 426615 mE, and 426645 mE) and three lateral drifts (between 426585 mE and 426600 mE, between 426600 mE and 426615 mE, and between 426615 mE and 426645 mE), which are shown in yellow on Figure 9.1. The two small lateral drifts between 426585 mE and 426615 mE were planned as a contingency in the event that additional tonnage was available after mining the four crosscuts and the 615L lateral drift. Crosscut and lateral development dimensions were planned at 3.5 m wide by 4 m in height. Each round was planned to be 2.7 m long, yielding approximately 100 tonnes per round. The crosscuts were designed to test the width of the mineralized corridors (i.e., cutting almost perpendicular to strike), whereas the lateral drifts were designed to test the along strike continuity of the mineralized system within a given corridor. Supporting development, shown in blue on the figure, included the 426600 mE crosscut (developed to provide material for bulk sample tower calibration purposes), drill stubs off the 6258015 mN drift (for drilling of the southward-directed underground drill fans), and drill bays off the ends of the crosscuts (for drilling of the northward-directed underground drill fans).

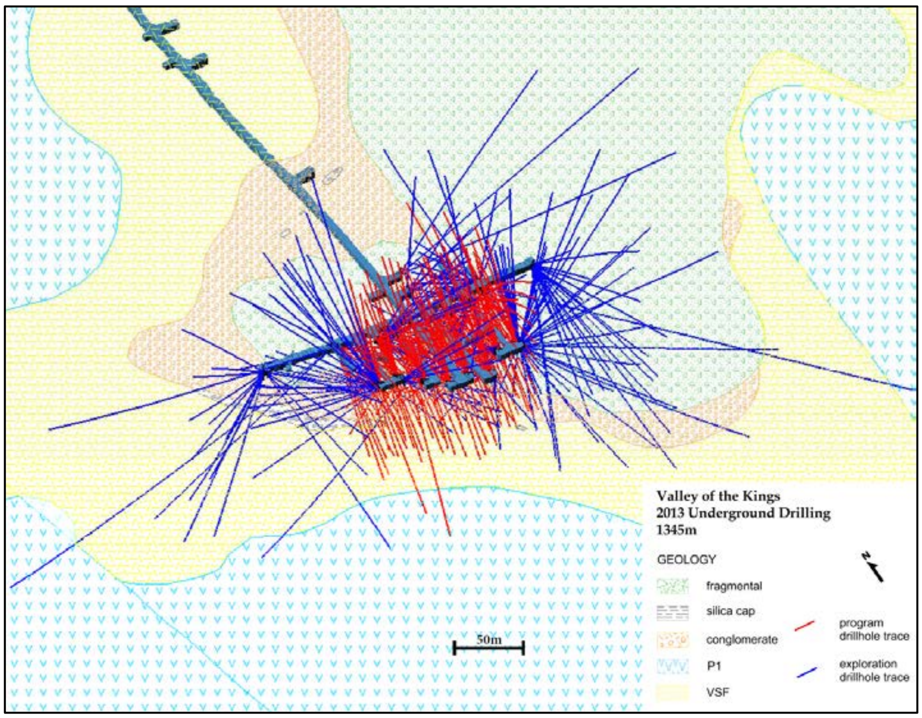
Figure 9.1 Planned (top) versus actual completed (bottom) bulk sample area layout on the 1345 m level, VOK deposit



(Source: Pretivm)

A contract miner (Procon) began underground work on the project in July 2012 by slashing the existing West Zone underground ramp to a 5 m by 5 m wide drift. The existing West Zone underground workings were followed for approximately 500 m before a new heading was opened, oriented due south along the 426600 m line of Easting. The access ramp, also with dimensions of 5 m by 5 m, was driven along 426600 mE from the 6258552 line of Northing on the 1315 m level. The bulk sample area was reached in early May 2013 and underground drilling to support the bulk sample program began in mid-May 2013. Owing to the legislative restrictions on the maximum extractable tonnage as part of the bulk sample program (10,000 tonnes), Pretivm decided to test a larger area around the bulk sample workings through underground drilling. A total of 16,500 m of underground drilling at 7.5 m centres was planned to drill off an area measuring 120 m along strike, 60 to 90 m across strike, and 60 m above and below the 1345 m level. An additional 16,500 m of underground exploration drilling was designed to test targets outside of the bulk sample area (Section 0). All underground drilling completed during 2013 is presented in Figure 9.2.

Figure 9.2 Underground drilling conducted during 2013 (bulk sample program drilling in red, underground exploration drilling in blue)



(Source: Pretivm)

Geological mapping (face, back, and ribs), channel, and chip sampling was conducted on a round-by-round basis. Channel samples were cut at the same height (waist level) along both walls of the access ramp, as well as along both walls of each crosscut. Channel samples were limited to faces in the lateral drifts. Additional channel samples were cut on faces where north-south trending components of the vein stockwork system were intersected (e.g., parts of the 426615 mE crosscut). Chip samples were collected using an artisanal hand chiselling tool, and were collected from the faces and/or ribs above the channel samples. Chip sample collection followed the same location protocol as the channel samples (i.e., generally collected along the ribs of the crosscuts, and faces of the lateral drifts, with face chip sampling conducted where orthogonal components of the vein stockwork system were present). Chip and channel samples were collected on 0.5 m sample intervals. The sample size for both chip and channel samples was designed such that the weight of each chip and channel sample would approximate the weight of half HQ core (roughly 4 kg per linear metre). This was done in an attempt to minimise bias associated with differences in sample support between the three sample types (HQ drillcore, chip, and channel). Channel samples were limited to a single 5 cm to 6 cm wide by 3 cm deep cut along the relevant mine feature, whereas chip samples covered a larger surface area of the feature (a series of shallow chiselled lines per half metre results, by virtue of the technique, in a panel of the rib or face being sampled to get sufficient material for the given chip sample). The start and end of each channel sample and the midpoint of each chip sample was surveyed by Procon.

Mapping was undertaken on a round-by-round basis in parallel with underground development, as well as of the underground workings as a whole. Lithology, structure, and alteration of the host rocks were recorded, as was information regarding vein type or style, mineralogy, and vein generation. The results of the mapping confirmed the style of mineralization and the geological model for the VOK, as discussed in Section 7.2. Figure 9.4 and Figure 9.5 illustrate the bulk sample crosscuts with the mapped stockwork veining shown against the lithological domains and mineralization domains respectively.

The material that made up the 10,000 tonnes of the bulk sample was collected from four of the crosscuts (426555 mE, 426585 mE, 426615 mE, and 426645 mE) spaced 30 m apart, as well as from the one lateral drift (615L, between 426615 mE and 426645 mE). The 10,000 tonne limit on the bulk sample was reached mining the four crosscuts and the 615L lateral. In order to mine the entire 615L lateral, four rounds were removed from the outer two crosscuts: one round from the southern end of the 426555 mE crosscut; three rounds from the southern end of the 426645 mE crosscut. Additional underground development completed in the bulk sample area included east and west extensions to the 6258015 mN drift, the development of the 426540 mE, 426600 mE, 426645 mE (645N; to the north), and 426660 mE (660N; to the north) crosscuts, two oblique crosscuts at 426501 mE and 426505 mE, a southeast-trending lateral drift from 426555 mE (the 555L lateral drift), as well as four raises (two each on the north-south Cleopatra and 615L east-west vein systems), and bench and breastwork on the 426615 mE crosscut (Figure 9.1). This development was conducted to support the underground exploration drilling (see above), as well as to further test both the geological and November 2012 Mineral Resource models for the VOK deposit.

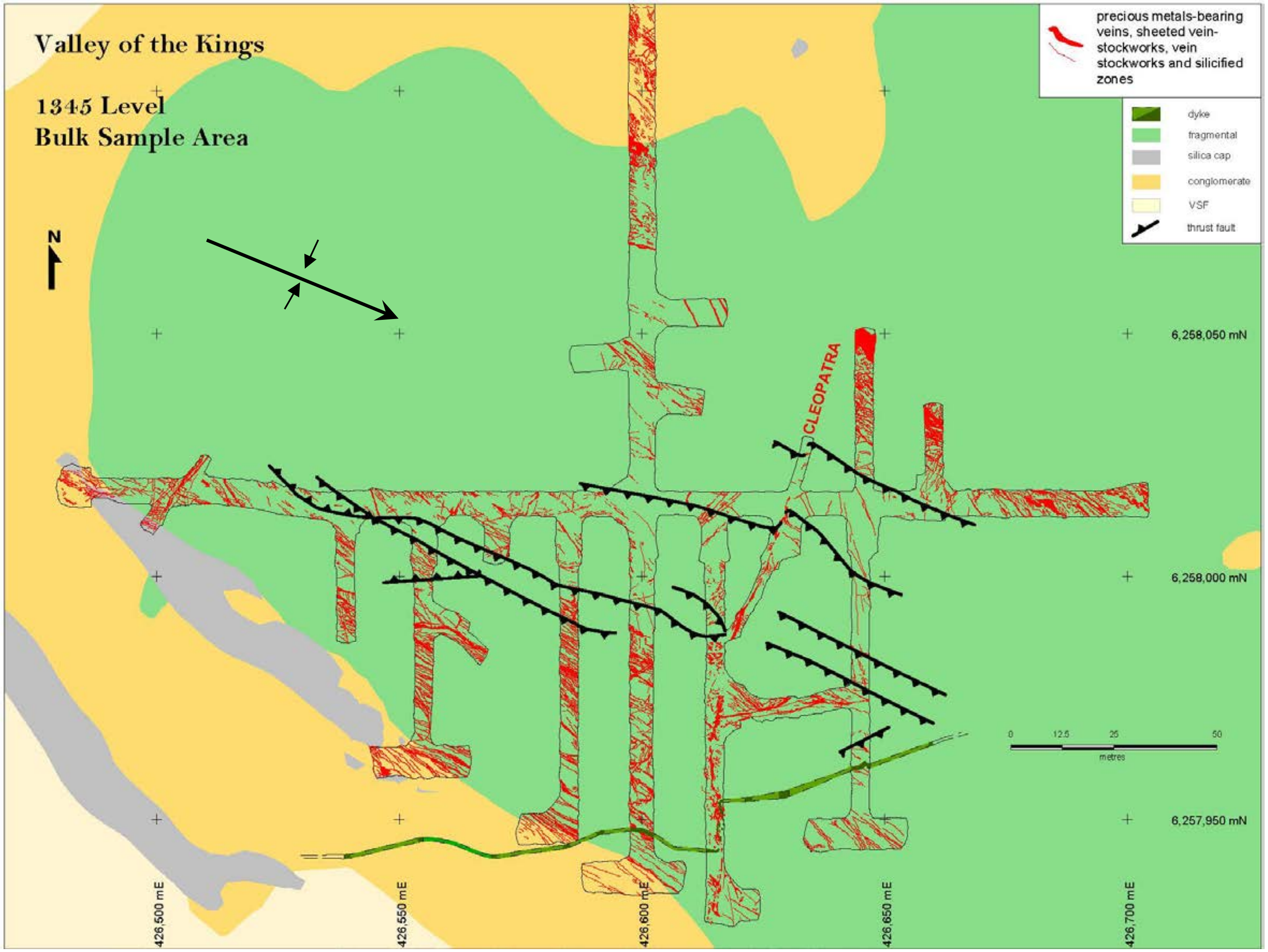
The bulk sample was collected in a series of nominal 100 tonne rounds in underground development, and processed through a sample tower on site (Figure 9.3). Each nominal 100 tonne round was split down to two 30 kg samples after processing through the sample tower. Normal practice is for one of the two samples to be analysed and the other to be retained in case additional testwork is necessary. Considering the higher than acceptable variance in results from the preliminary bulk sample rounds extracted from the 426600 mE crosscut due to the coarse nature of the electrum in the VOK, Pretivm elected to do four runs on the bulk sample material for a given round through the sample tower, yielding eight tower samples per round. From four up to all eight of the tower samples were sent for assay, the final number of samples for a given round submitted to the laboratory dependent upon the predicted grade for that round (Section 9.4.1).

The bulk sample material from each round (minus the sample splits collected at the sample tower) was sent as defined parcels to the Contact Mill in Philipsburg, Montana, USA, for processing. This was done to provide a comprehensive and irrefutable dataset for reconciliation purposes.

Figure 9.3 Pretivm bulk sample: sample tower layout, viewed to the west-northwest

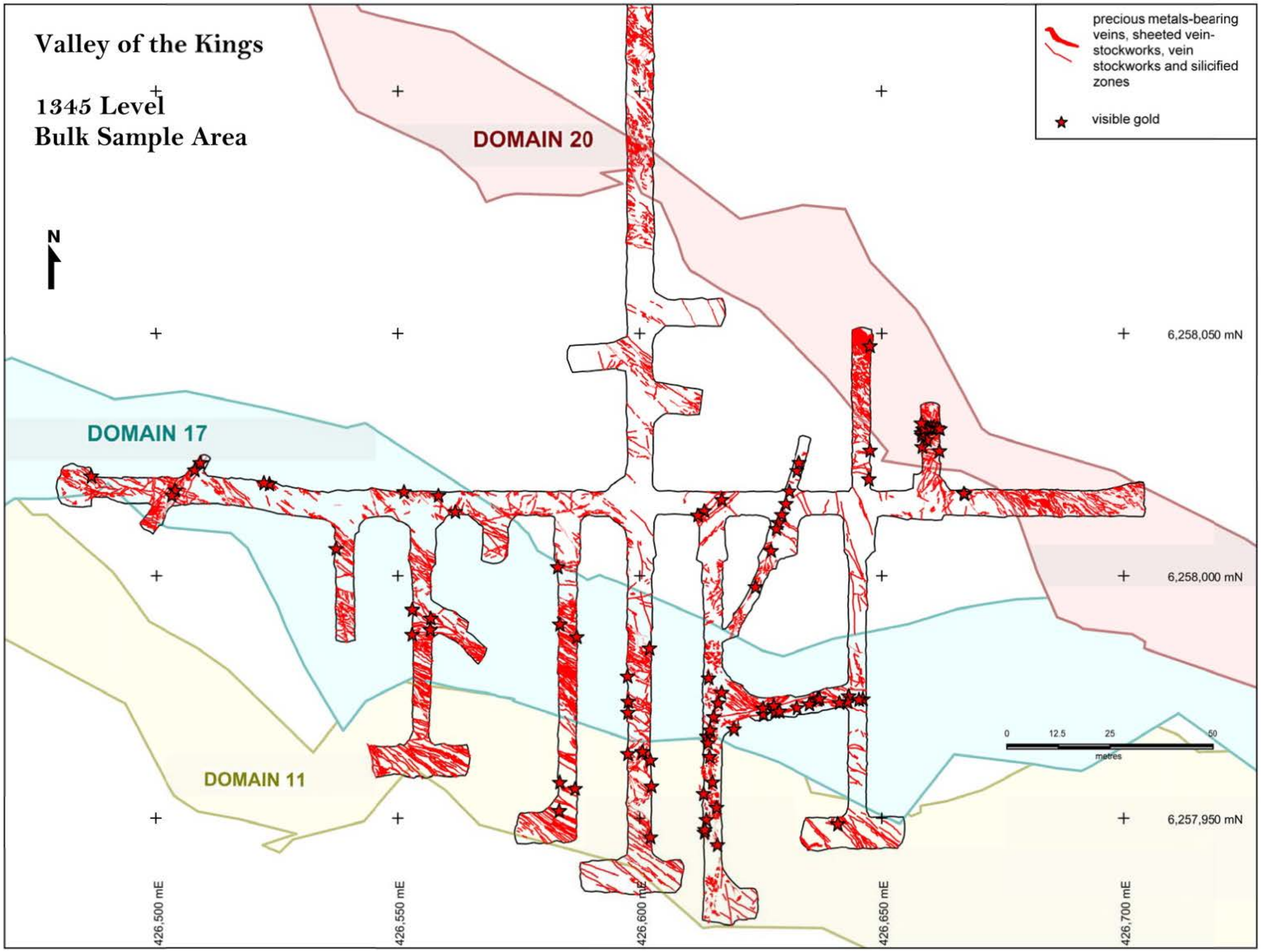
(Source: Pretivm)

Figure 9.4 Plan view (1345 m level) of bulk sample area geology and veining



(Source: Pretivm)

Figure 9.5 Plan view (1345 m level) of bulk sample area veining and mineralization domains



(Source: Pretivm)

9.4.1 Sample tower results

Snowden cautions that the inherently variable nature of high and extreme grade mineralization within the broader lower grade (0.5-2.5 g/t Au) mineralized stockwork vein system in the VOK makes the collection of representative small-scale samples extremely challenging. Elevated grade in the VOK occurs as agglomerations of electrum of up to several centimetres in size. The sample results will therefore be dependent on whether or not a fragment of electrum will make it into one of the 30 kg buckets out of the nominal 100,000 kg being sampled, and therefore these small-scale samples are heavily impacted by the probability of not getting the right proportion of electrum in the sample. This was clearly demonstrated in the early stage of processing of the bulk sample when the grade of crosscut 426585E was nearly double that predicted from the sample tower results.

The sample tower process was designed such that each individual 30 kg tower sample was to be representative of the round being sampled. The high degree of imprecision in the sample tower data for each round (as seen in the large variation between the minimum and maximum values for that round; Figure 9.6 to Figure 9.10) highlights the difficulty in generating a representative small-scale sample for that round (i.e., which one to choose as the representative one).

Consequently it is Snowden's opinion that the results of the assaying of the various samples per round, as generated by the sample tower, lacks sufficient precision to predict the grade of each nominal 100 tonne sample adequately. This was confirmed by the results of the processing which demonstrated some significantly different results from those of the sample tower. This is shown by the wide range of results from each round through the sample tower as opposed to the mill results (Figure 9.6 to Figure 9.10). Tabulations of the sample tower results for the various crosscuts are presented in Appendix A.

Figure 9.6 Crosscut 426555 E – maximum and minimum sample tower grades vs. mill feed grade per round

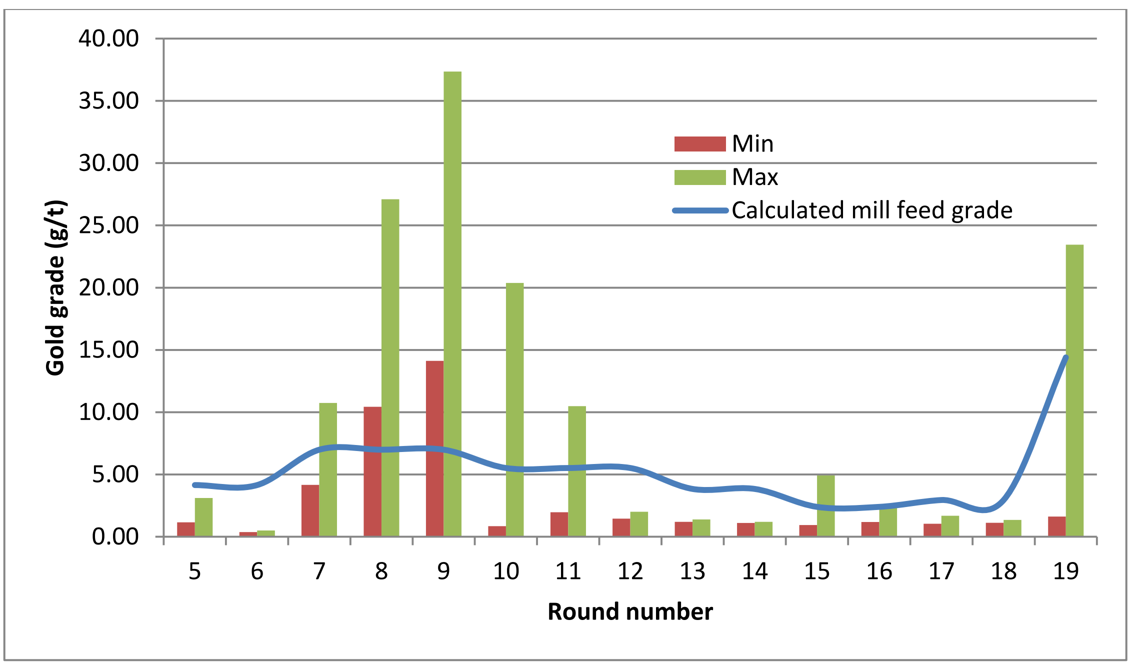


Figure 9.7 Crosscut 426585 E - maximum and minimum sample tower grades vs. mill feed grade per round

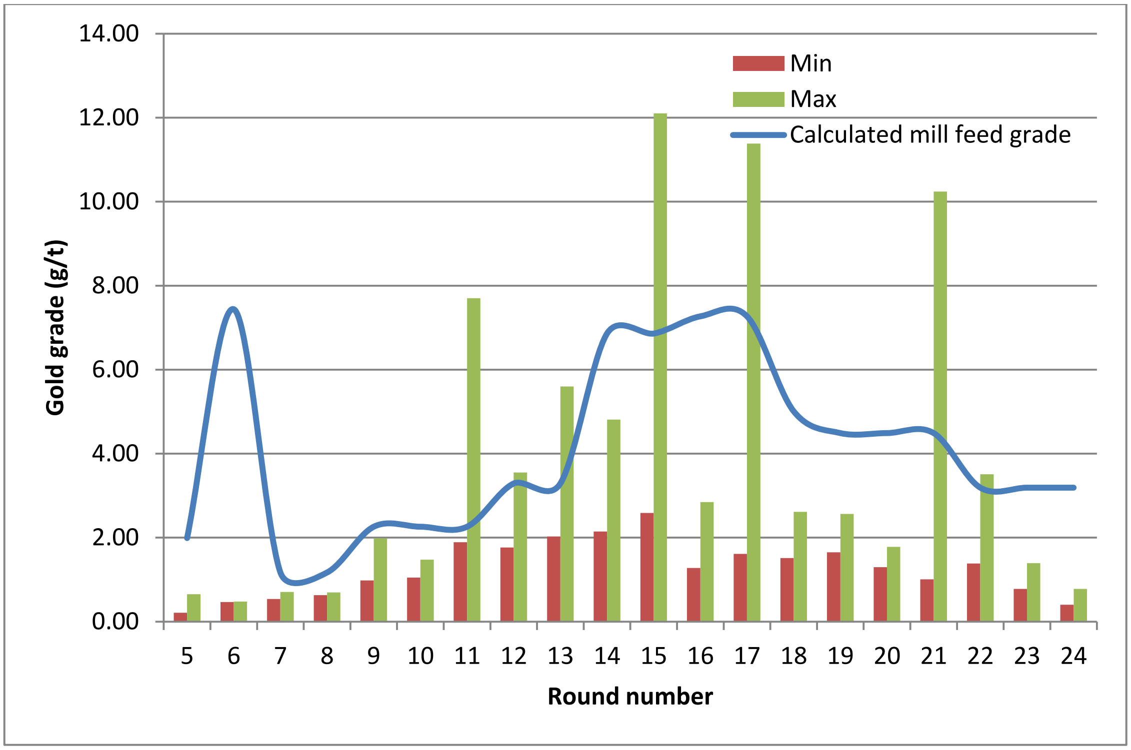


Figure 9.8 Crosscut 426615 E - maximum and minimum sample tower grades vs. mill feed grade per round

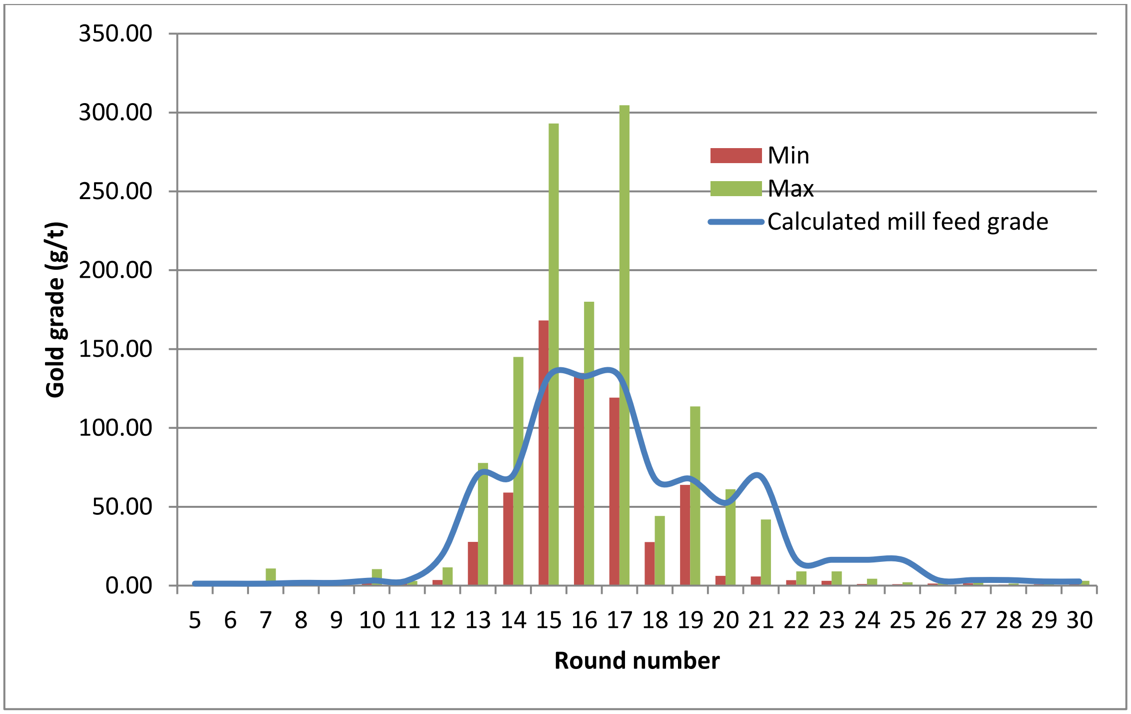


Figure 9.9 Crosscut 615 L - maximum and minimum sample tower grades vs. mill feed grade per round

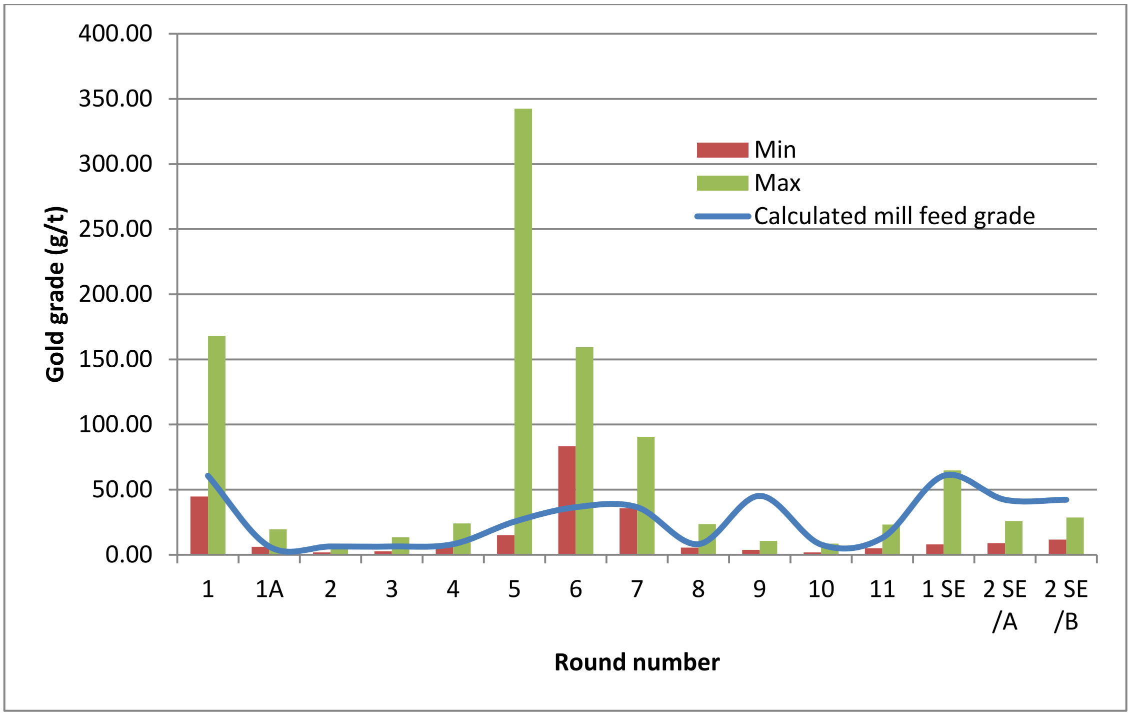
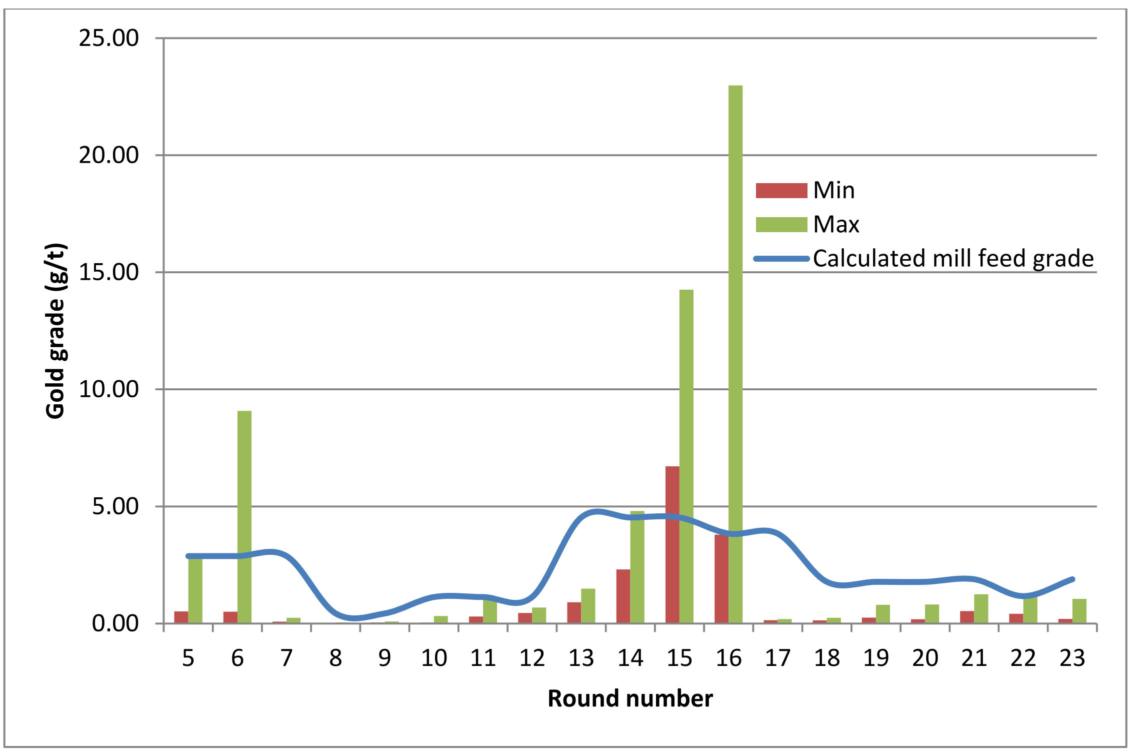


Figure 9.10 Crosscut 426645 E - maximum and minimum sample tower grades vs. mill feed grade per round



10 Drilling

Historical information in this section has been excerpted from P&E sections within Ghaffari et al., (2012). Snowden updated and verified this information.

10.1 Historical drilling

Drilling on the Brucejack Property dates back to the 1960's, although most of the historical drilling was completed in the late 1980's and early 1990's. Up to this time, 452 surface diamond drillholes were completed which totalled 60,854 m. These drillholes were relatively short, averaging 135 m per drillhole, and were mostly concentrated on West Zone, followed by Shore Zone, Galena Hill and Gossan Hill. As part of this exploration program, an exploration ramp was driven on the West Zone from which an additional 442 underground diamond drillholes were completed which totalled 33,750 m. This drilling was focused exclusively on West Zone and increased the drill density to approximately 5 m centres between 5 m and 10 m sections. Historical drill core sizes for surface drillholes were NQ (47.6 mm diameter) and BQ (36.5 mm diameter). Core size for drillholes collared from the underground exploration ramp was AQ (27 mm diameter).

10.2 Silver Standard drilling

Using the historical drill and trench data as a baseline, and following on the success of the Snowfield bulk tonnage drilling to the north, the 2009 Brucejack drill program was designed to test for additional bulk tonnage resources on the Property. This program successfully discovered several areas with bulk tonnage mineralization. Within the broader bulk tonnage mineralization were locally discreet high grade intersections. These included drillholes SU-005 and SU-012, which were drilled to test for the western extension of the previously defined Galena Hill. These two drillholes intersected 1.5 m of 215 g/t Au and 1.5 m of 16,949 g/t Au respectively in what would eventually be called the VOK. Drilling in 2009 totalled 17,846 m in 37 drillholes, of which 2,913 m in 6 drillholes was targeted at the VOK.

In 2010, the drill program was designed to further define the bulk tonnage mineralization found the previous year, as well as attempt to define a high grade resource at the VOK. In this year, a total of 73 diamond drillholes was completed which totalled 33,480 m. Of this, 11 drillholes comprising 3,693 m were targeted at the VOK, and two drillholes, totalling 1,119 m at the footwall of West Zone. In the VOK, wide spaced drilling intersected enough high grade mineralization to confirm the exploration potential of the zone. The exploration potential included the preliminary definition of some of the ore controls which put the intersections into a geologic context. The West Zone drilling intersected a broad zone of bulk tonnage mineralization within which were several high grade intersections.

10.3 Pretium Resources Inc. surface drilling

The 2011 diamond drill program was the first in almost 20 years that was focused specifically on defining high grade resources. In 2011, a total of 178 drillholes were completed totalling 72,805 m in drillholes SU-110 to SU-288. Included in this were 97 drillholes (41,219 m) targeted at the VOK, 16 drillholes (7,471 m) at West Zone, and 21 drillholes (7,220 m) targeting the surrounding areas. The remaining drilling was focused on expansion of Shore Zone, testing for structurally controlled high grade mineralization in the Galena Hill and Bridge zones, and testing new target areas. Drill collar coordinates, and results of the drilling in 2011 were described by P&E in Ghaffari et al., (2012).

The 2012 diamond drill program was focused on defining the high grade resource in the VOK, specifically targeting geological and structural features believed to be associated with gold mineralization. Diamond drilling was also focused on expanding the VOK Zone, both west of the Brucejack Fault and along trend to the east of the main mineralized zone. A total of 301 drillholes were completed, totalling 105,500 m of drilling during the 2012 drilling program. Zones within 150 m of surface were drilled at 12.5 m centres, with the deeper parts (down to about 350 m below surface) being drilled at approximately 25 m centres. Drilling at greater depths was generally only able to reliably achieve 50 m centres.

The 2013 surface diamond drill program focused on further defining the high grade resource at the VOK and further targeting of the geological and structural features believed to be associated with gold mineralization. A total of 24 surface diamond drillholes (5,200 m) of the 37 surface diamond drillholes completed on the Brucejack Property in 2013 (5,770 m in drillholes SU-590 to SU-626) were focused on the VOK.

Figure 9.13 illustrates all diamond drilling carried out across the Brucejack Property.

For the 2013 surface drilling program, the drilling contractors were Hy-Tech Drilling Limited from Smithers BC. The drill collars were surveyed by McElhanney Surveying from Terrace, BC. McElhanney Surveying used a total station instrument and permanent ground control stations for reference and had completed all the surveying on the project since 2009. Drillhole paths were surveyed at a nominal 50 m interval using a Reflex EZ single shot instrument. All drillhole paths were checked in a mining software package for deviation errors, which, if present, were corrected on a realtime basis. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results.

Core sizes for Pretivm's surface collared drillholes were PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ (47.6 mm diameter). Approximately 50 to 60% of core was HQ size. For drillholes less than 600 m in length, core size was commenced at HQ and reduced to NQ when conditions required a change. For drillholes greater than 600 m length, the commencing core size was PQ which was run down to between 200 m and 300 m in order to minimise drill path deviation. All drillcore collected from the underground drilling in 2013 was HQ size. No significant bias was noted between the PQ and HQ drill core samples that intersected the VOK mineralization. No testing was required on the NQ drill tails as these were almost without exception at depths below the main mineralization zones.

Drill core logging and handling procedures were the same for all four surface drilling programs. At the end of each drill shift all core was placed in wooden boxes (Figure 9.11), labelled by drillhole and interval and transported to the core logging and core splitting facility on site by snowcoach (Figure 9.12) or helicopter. The sample boxes were covered to avoid sample loss or contamination during transportation (Figure 9.12).

Figure 9.11 Core in wooden core boxes ready for transport.

Figure 9.12 Sample transportation by snowcoach

Prior to any geotechnical and geological logging, the entire drill core was photographed in detail using a digital camera. These images were stored in individual files per drillhole.

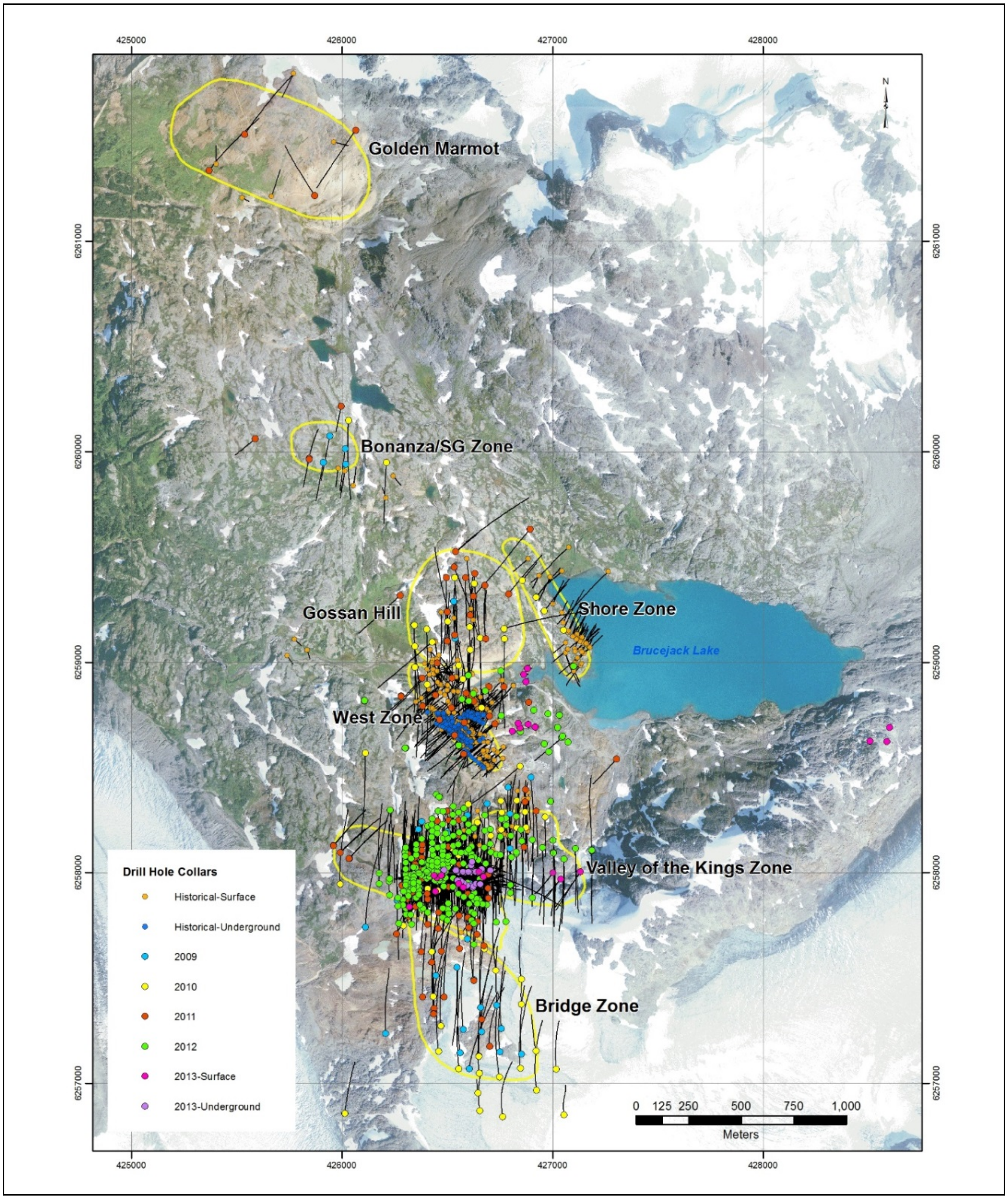
A trained geotechnician recorded the core recovery and rock quality data for each measured drill run. All lithological, structural, alteration and mineralogical features of the drill core were observed and recorded during the geological logging.

The geologist responsible for logging a drillhole assigned sample intervals with the criteria that the intervals did not cross geologic contacts and the maximum sample length was 2 m. Within any geologic unit, sample intervals of 1.5 m long could be extended or reduced to coincide with any geologic contact. Sample lengths were rarely greater than 2 m or less than 0.5 m, and generally averaged 1.5 m. Every drillhole was sampled in its entirety from top to bottom.

All data were directly logged into a centralized database by trained geotechnicians and geologists. The DHLogger software was used in 2011 and 2012, with Pretivm switching to the Geospark software for direct data entry in 2013.

It is the author's opinion that the core logging procedures employed were thorough and provided the appropriate level and quality of information required to model the geological and geotechnical aspects of the deposit. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results. The author believes that drilling has been conducted using industry standard guidelines and is appropriate for use in grade estimation.

Figure 9.13 Brucejack Property diamond drillhole plan



(Source: Pretivm)

10.4 Underground drilling associated with the bulk sample

For the underground bulk sample, a total of 16,500 m of underground diamond drilling at 7.5 m centres was planned to drill an area measuring 120 m along strike, 60 m to 90 m across strike and 60 m above and below the 1345 m level. An additional 16,500 m of drilling was designed to test targets outside of the bulk sample area.

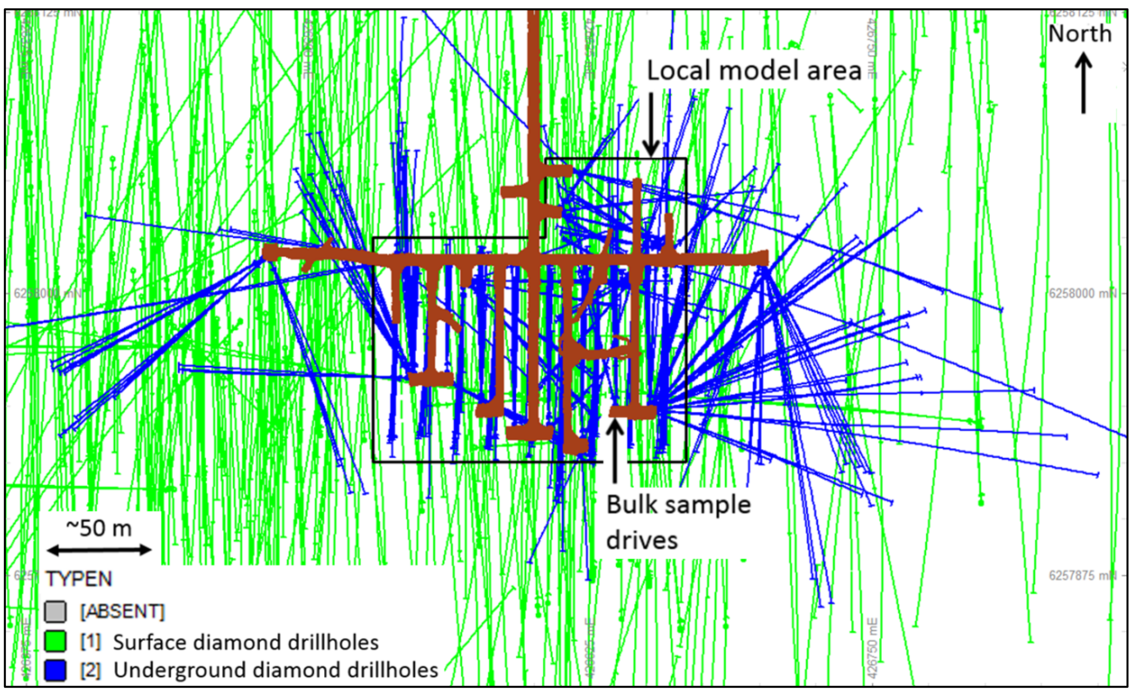
The drilling within the bulk sample area was designed as a series of fans trending north-south, between each north-south bulk sample crosscut. As a result, the orientation of the sampling is dominantly north-south. This may result in a bias, whereby the areas dominated by north-south trending mineralization were not fully sampled.

As part of the testwork carried out in the local area surrounding the bulk sample, estimates were generated with and without the underground drillholes to test for this directional bias. The results of this testwork indicate that there is some directional bias, as removing the underground drillholes increases the local grade of the estimate within the bulk sample crosscuts, particularly those crosscuts dominated by north-south mineralization (Section 14.5.1).

A total of 409 drillholes (38,840 m) were completed with 200 of these drillholes (16,640 m) being in the bulk sample area, and the remainder (209 drillholes totalling 22,200 m) testing targets outside of the bulk sample area. Drilling procedures were the same as for the surface diamond drilling (Section 10.3) except that the drillhole collars were surveyed by Procon for the underground drilling rather than McElhanney Surveying.

Drillholes range from 12 m to 450 m in length, with most drillholes being between 50 m and 150 m in length.

Figure 9.14 Plan view of underground bulk sample area showing surface and underground drilling



11 Sample preparation, analyses and security

Information in this section has been excerpted from P&E sections within Ghaffari et al., (2012) and Graindorge (2013), edited in part and modified.

11.1 Sample preparation before dispatch of samples

Upon completion of the geological logging, the core was moved to the splitting area where the core was either split or sawn in half lengthwise using a wet diamond saw. All PQ core was sawn as the core was too big to fit into the splitters (Figure 11.1). Likewise, any sample intervals which contained visible gold or interesting mineralization were also sawn. All other core was cracked in half using a standard hammer/blade core splitter. One-half of the drill core was placed in a plastic sample bag with the appropriate sample tag and the other half was returned to its original position in the core box. The sample bags were placed in four or five rice sacks and flown to the staging area by helicopter. Individual work orders were generally between 80 and 120 samples, including standards, blanks and field duplicates. At the staging area, a local expediter brought the samples to Stewart where they were loaded onto a five tonne truck and locked for the night in the company's warehouse. The next morning they were driven to the ALS Minerals sample preparation facility in Terrace, BC.

Figure 11.1 Cutting PQ core at the Brucejack Property

The cut PQ samples weigh approximately 10 kg. HQ samples are around 6 kg, and NQ samples between 3 kg and 4 kg. These weights assume a nominal 1.5 m sample length. In general, the average sample size submitted to the analytical laboratory was 6.5 kg.

Pretivm's Qualified Person for field activities is Mr Kenneth McNaughton, P.Eng., Chief Exploration Officer for Pretivm.

11.2 Analytical laboratory

Samples were shipped to, and then prepared for analysis at the ALS facility in Terrace, British Columbia. Two analytical laboratories were used by Pretivm to analyse samples from the Brucejack project – the ALS Global ("ALS") laboratory and the SGS Canada ("SGS") laboratory, both of which are located in Vancouver, British Columbia. The ALS laboratory was the primary laboratory, while the SGS laboratory acted as an umpire or check laboratory.

ALS Global is an internationally recognised company with minerals testing laboratories operating in 55 countries and has ISO 9001:2000 certification at most laboratories. The ALS analytical laboratory in Vancouver has also been accredited to ISO 17025 standards for general testing laboratory procedures by the Standards Council of Canada (SCC).

11.2.1 Method

Samples at ALS were crushed to 70% passing 2 mm, (-10 mesh). Samples were riffle split and 500 g were pulverized to 85% passing 75 µm (-200 mesh). The remaining coarse reject material was returned to Pretivm for storage in their Stewart, BC warehouse.

The gold grade was determined using fire assay on a 30 g aliquot with an atomic absorption (AA) finish. In addition, a 33 element package was completed using a four acid digest and ICP-AES analysis, which included the silver. The upper limit of acceptable accuracy for gold and silver was 10 ppm Au and 100 ppm Ag respectively using these methods. For samples above these levels, the gold and silver content was determined by gravimetric analysis.

11.2.2 Density determinations

Density determinations were done by ALS using the pycnometer method on pulps from the drilling program. Pretivm's Qualified Person (Mr Kenneth McNaughton, P.Eng., Chief Exploration Officer) selected the samples as the programs were progressing to maintain good coverage over a wide range of locations, rock types, styles of alteration and mineralization. A total of 2,621 pulp specific gravity determinations have been completed, including 213 determinations since the November 2012 estimate.

As part of the 2012 drilling program, Pretivm selected a portion of the samples (207 samples) for core density measurements as well as the pulp specific gravity measurements in order to determine whether there is any impact on the density as a result of porosity. A further 204 samples were collected for specific gravity and density measurements as part of the 2013 underground drilling program to increase the comparative dataset.

Core density measurements were carried out by ALS using a standard water displacement method calculation (weight in water, weight in air) on wax coated core samples.

Results of the comparison between the pulp specific gravity and core density measurements indicate that the core density is on average the same as the pulp specific gravity within the siliceous zone and approximately 3% lower, on average, for all other rock types. Consequently all specific gravity estimates in the Mineral Resource model (which are based on the pulp specific gravity measurements), with the exception of the siliceous zone, were factored down by 3% to give the bulk density.

11.3 Quality assurance and quality control

Snowden analysed the quality assurance and quality control ("QAQC") for the Brucejack Project. The Brucejack drillhole and QAQC database is managed by GeoSpark Consulting Inc. ("GeoSpark"), who also manage the routine analysis of the QAQC results for Pretivm. GeoSpark supplied Snowden with a QAQC database, in Microsoft Access format, containing the QAQC results for all drilling up to 5 December 2013.

The QAQC protocols included the use of field duplicates, standards and blanks. The quality control samples were included at a nominal rate of one field duplicate, one standard and one blank for every 20 samples. Check assays, in the form of pulp duplicates, were also completed by a different laboratory (SGS) and compared with the primary laboratory.

11.4 Precision

Field duplicates were included in the sample batches to assess the precision of the assay data. For the diamond core drilling, the field duplicates comprise either the remaining half core (2013 onwards) or a quarter core sample (prior to 2013).

Based on the HARD (half absolute relative difference) statistic, 90% of the diamond core duplicate pairs have a difference of less than 30% for gold and 25% for silver. Snowden considers the precision related to the results from field duplicates to be a reasonable outcome given the nuggetty nature of the mineralization at Brucejack.

11.5 Analytical accuracy

Analytical accuracy was assessed using standard samples of known grade and variability, with the assays compared to the expected grade of the standard sample. Pretivm inserted standard samples into the sample batches at a rate of one standard to every 20 samples.

Five different standards were used ranging from 0.87 ppm Au and 12.0 ppm Ag, up to 10.4 ppm Au and 276 ppm Ag. Two of the standards were commercial standards, while three of the standards were generated from material sourced from metallurgical samples from the VOK. All standards were certified standards.

Overall, it is Snowden's opinion that, whilst some standard assay results fall outside the control limits, the standard results show good analytical accuracy is being achieved at the ALS laboratory (the primary laboratory), with no evidence for analytical bias. No standards were submitted to the check laboratory (SGS) and as such Snowden is unable to comment on the analytical accuracy of the check laboratory, however the SGS check assays compare well with the ALS results, suggesting that analytical accuracy is unlikely to be an issue at the SGS laboratory.

11.6 Contamination

Blank samples were inserted into the sample batches to assess contamination of samples during the sample preparation and assaying. The blanks were inserted into the sample batches at a nominal rate of one blank for every 20 samples. Additional blanks were also included by Pretivm after visibly high grade mineralization was noted (e.g. visible gold observed in the diamond drill core).

The results indicate that, while some blanks samples show elevated gold and silver contents, overall, contamination during the sample preparation and assaying is considered reasonable and within acceptable tolerance intervals.

11.7 Check assays

Pulp check samples were assayed at the SGS laboratory. The check samples were selected upon finalisation of the primary sample assay results by GeoSpark, with samples selected randomly within seven grade bins, defined by percentiles, to ensure that all grade ranges were covered adequately.

The pulp check assays for gold show a good comparison between assays at ALS and SGS and that a good level of precision is being achieved for the pulp duplicates. Based on the HARD statistic, 90% of the duplicate pairs have a difference of less than approximately 13% (as measured by the HARD statistic). Snowden considers this to be a good level of precision for pulp duplicates, especially given the nuggetty nature of the gold mineralization in the VOK.

However, a complete comparison of the silver check assays was not possible owing to different levels of analytical precision along with different upper detection limits for the two laboratories. Above the upper detection limits, the assays were typically completed using gravimetric techniques to achieve a more reliable assay result; however this was not always done for the check assays. Where reliable assays have been completed at both ALS and SGS, the results show that reasonable precision is being achieved for silver.

11.8 Author's opinion on date sample preparation, security and analytical procedures

Procedures undertaken to date by Pretivm have been under the supervision and security of Pretivm's staff, as far as drill core sampling prior to dispatch. Laboratory sample reduction and analytical procedures have been conducted by independent accredited companies with acceptable practices.

Pretivm ensured quality control was monitored through the insertion of blanks, certified reference materials and duplicates.

It is Snowden's opinion that the sample preparation, security, and analytical procedures were satisfactory and appropriate for resource evaluation of Brucejack.

12 Data verification

Independent sampling and site verification visits were undertaken by Snowden in 2012 and 2013.

12.1 Data verification by Snowden

Snowden's QP Mr Ivor Jones visited the Brucejack site on 15 and 16 February 2012, June 3 to 6 2013 and August 16 to 21 2013, and takes the overall responsibility for data verification for this report.

In addition, Ms Lynn Olssen, MAusIMM (CP), Principal Consultant, Snowden, Perth and Mr Harald Muller, FAusIMM, Senior Principal Consultant, Snowden, Brisbane, visited the project site in August 16 to 21 2013.

The following items were verified:

- Cross-check of Pretivm drill logs with drill core. Example core was reviewed with Mr Kenneth McNaughton, P.Eng. and Dr Warwick Board, P.Geo, Pretivm's Chief Geologist.
- Core handling, storage and security at Pretivm's core storage facility in Stewart.
- Core logging process, alignment, recovery, mark-up and core sawing, sampling.
- Insertion of blanks, certified reference material.
- Core shack at the Brucejack Camp.
- Review of drill logs, assay records and interpretations at the Pretivm Vancouver office.
- Underground inspection of the bulk sample crosscuts including:
  - Review of visible electrum in various forms including aggregates and veins of various generations, confirming the validity of the presence of coarse gold.
  - Review of the nature of the mineralization confirming the appropriateness of the definition of domains, and the estimation method.
- The Geospark data validation work completed in 2011.
- Review of the location of the drillhole traces, confirming the appropriateness of the downhole surveying traces. Note, all surface drillhole intersections in the underground were found to be within a few metres of where the downhole survey traces indicated that they would be; thereby providing an additional level of validation as to the spatial location of the data.

Snowden carried out a basic statistical and visual validation of the data prior to estimation and found no significant issues.

In June 08 to 10, 2012 Adrian Martínez Vargas completed the sample validation under the supervision of Ms Lynn Olssen and Mr Ivor Jones (QP for this report). The following items were verified:

- Cross-check of collar coordinates.
- Core logging process, alignment, recovery, mark-up and core sawing, sampling.
- Insertion of blanks, certified reference material.
- Core shack at the Brucejack Camp.

- Sample transportation and delivery in ALS facilities in Terrace, BC.

In June 09, 2012 Mr Martínez Vargas collected twelve samples from six drillholes for assay. The samples were selected randomly as two contiguous intervals per drillhole. Each sample was taken as the halve interval of HQ cores, then documented, bagged and sealed. All the samples were selected from cores with good recovery with a weight averaging 6.74 kg. The samples were transported by Mr. Martínez by car to Terrace and by airplane to Vancouver, BC. The samples were under the direct supervision of Mr. Martínez for the duration of the transportation.

The samples were sent for assaying to the ALS laboratories in North Vancouver, BC. The sample preparation and assaying protocol requested to ALS was the same as that used by Pretivm in the ALS facilities in Terrace. Additionally two different standards provided by Pretivm were assayed in order to test the accuracy of the laboratories. The results for the standards assays are shown in the Table 12.1.

The comparison between the grade in the samples collected by Snowden and Pretivm are shown in Figure 12.1 and Figure 12.2. It should be noted that these samples are only validating the lower grade parts of the mineral system. This was intentional as the high grade samples have been verified by independent laboratories. From the work that has been completed, Snowden notes that the repeatability of the high grades is relatively good, whilst the precision in the low grade samples is very good in all duplicate sample work completed (refer to sections 11.4 and 11.7 of this report).

Figure 12.1 Sample verification results for Au grades

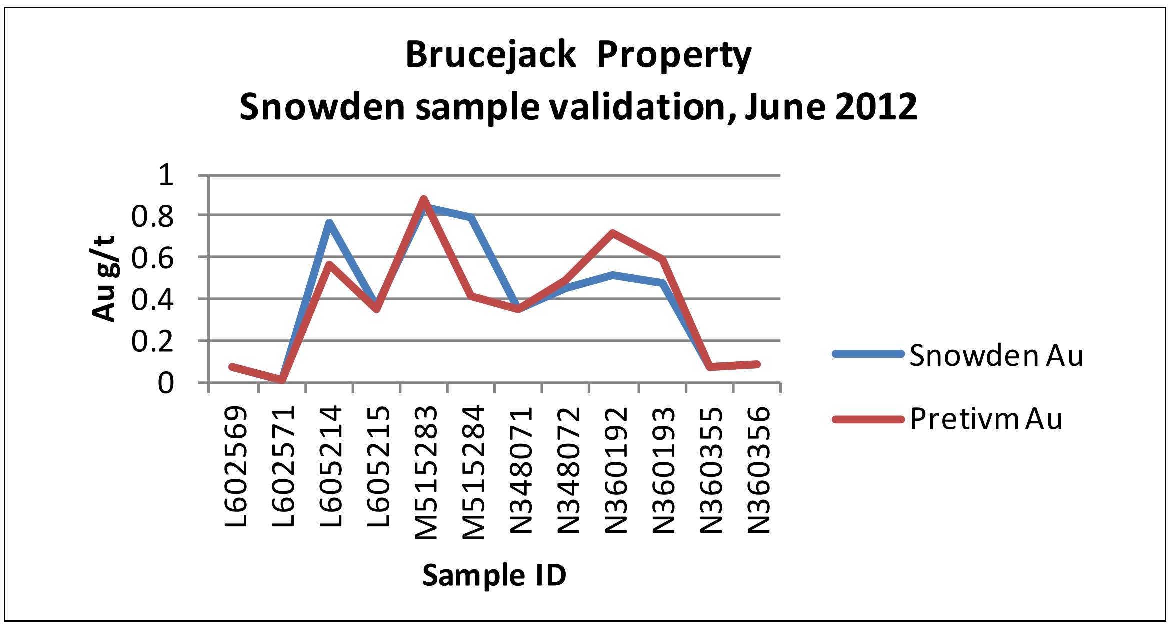


Figure 12.2 Sample verification results for Ag grades

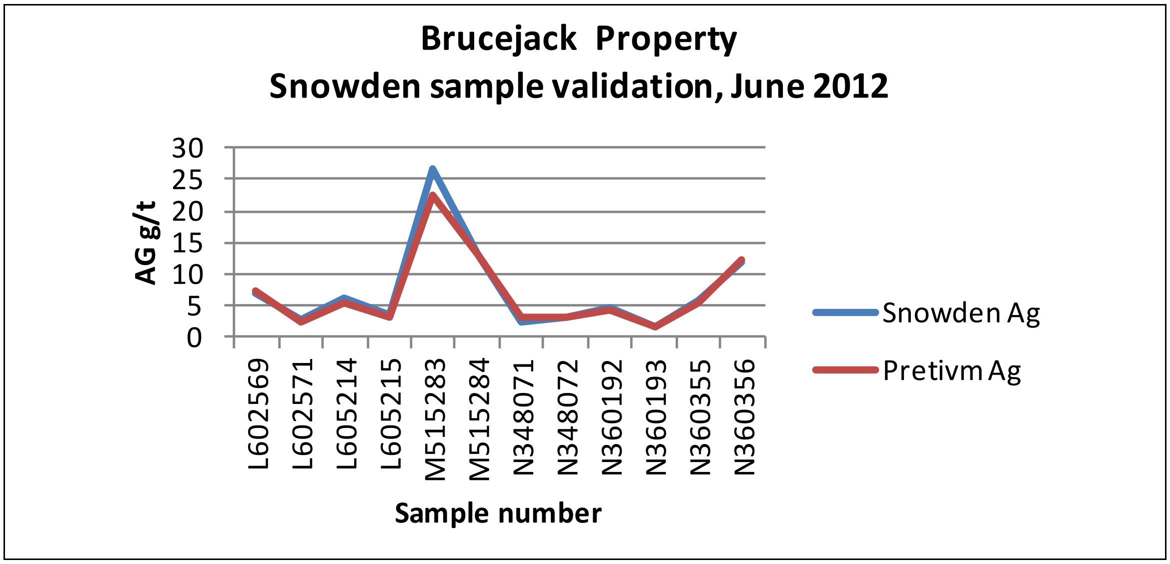


Table 12.1 Standard verification

ID	Snowden Au	Snowden Ag	Nominal Au	Nominal Ag
A	10.65	283	9.97±0.58	276.0 ± 17.1
B	0.86	40.3	0.87±0.09	39.3 ± 4.6

The author has not undertaken a complete data verification study, however sufficient checks have been completed to satisfy the author that the Brucejack drilling and sampling data is suitable to use in estimating a Mineral Resource.

13 Mineral processing and metallurgical testing

13.1 Metallurgical testwork

Information in this section has been excerpted from the Feasibility Study reported in 2013 (Ireland et al., 2013) and Ghaffari et al., (2012) and condensed and updated. The reader is referred to Ireland et al. (2013) and Ghaffari et al., (2012) for detailed information.

Several metallurgical test programs have been completed to investigate the metallurgical performance of the mineralization. The main test work was completed from 2009 to early 2013. The samples tested were generated from various drilling programs. The metallurgical test programs conducted on the Brucejack mineralization included head sample characteristics, gravity concentration, gold/silver bulk flotation, cyanidation and the determination of various process related parameters. The early test work focused on developing the flowsheet for gravity concentration, bulk flotation, and flotation concentrate cyanidation. The test work also studied the metallurgical responses of the samples to the gravity concentration flowsheet for gravity concentration followed by whole ore leaching. The later test work concentrated on the gravity-flotation concentration flowsheet.

In general, the VOK Zone and West Zone mineralization is moderately hard. The mineral samples tested responded well to the conventional combined gravity and flotation flowsheet. The gold in the mineralization was amenable to centrifugal gravity concentration. On average, 40 to 50% of the gold in the samples were recovered by gravity concentration. The flotation tests results indicated that bulk flotation can effectively recover the gold in the gravity concentration tailings using potassium amyl xanthate (PAX) as a collector at the natural pH. Two stages of cleaner flotation would significantly upgrade rougher flotation concentrate. However, the gold in the mineralization showed better metallurgical performance, compared to silver. On average, between 96% and 97% of the gold and 91% and 92% of the silver were recovered to the gravity concentrate and bulk flotation concentrate at the grind size of 80% passing approximately 70 to 80 µm. There was a significant variation in metallurgical performances among the samples tested. This may be a result of the variably nuggetty mineralization.

Cyanide leach tests were also conducted to investigate the gold and silver extractions from various samples, including head samples, flotation concentrates, flotation tailings and gravity concentrates. In general, most of the sample responded reasonably well to direct cyanidation, excluding a few of samples containing higher contents of graphite (carbon), arsenic, or electrum. Cyanide leach process has not been recommended for the study.

The test results suggest that the gold and silver recovery flowsheet for the mineralization should include gravity concentration, bulk rougher and scavenger flotation, rougher and scavenger concentrate regrinding, followed by cleaner flotation.

13.2 Bulk sample program

The 10,000 tonne bulk sample was mined during the third quarter of 2013 and processed in the fourth quarter of 2014 at the Contact Mill in Philipsburg, Montana. As a part of its engagement with Pretivm, Snowden reviewed the handling of the bulk sample from the mine to the production of final concentrates and tails at the Contact Mill. Snowden concluded that the metal accounting was at a high standard and that the treatment of the ore was conducted in line with reasonable expectations with regards to precious metal recovery and monitoring of process parameters. The metallurgical monitoring consisted of:

- A brief review of the sample tower operations to understand how the nominal 100 tonne samples were managed and processed on site.
- Management of and the chain of custody during shipping of the bulk sample.
- Processing of the bulk sample at the Contact Mill.
- Metal accounting procedures at the Contact Mill.
- Evaluation of the bulk sample processing results.

13.2.1 Sample shipping

After crushing and sampling through the sampling tower, the ore was loaded into bulk bags (approximately one tonne each), which were shipped via five transfer points from the Brucejack project site to the mill in Philipsburg, Montana. The trip included:

- Transport by truck from Brucejack to Bowser staging point.
- Transport by truck from Bowser to Wildfire staging point.
- Transport by truck from Wildfire to Terrace railway station.
- Transport by railcar from Terrace to Port of Montana.
- Transport by truck from Port of Montana to Contact Mill.

At each of the transfer points the bag numbers were logged and reported, allowing for accurate tracking of the bags from the project site up to delivery to the Contact Mill. Similarly the bag numbers were recorded by a security staff member as each bag was loaded into the feed chute for processing at the mill.

Each bag had been allocated a bar code, which allowed for monitoring of individual bags, but initially some of the labels were printed on poor quality paper and as a result these labels faded to the point that they could not be read. This did result in less than 1% of the bags not being identifiable, either due to damage to the bags or due to the markings not being legible. All the bags, however, were shipped to the mill and loaded, with the final batch of bags being those that could not be identified. This implies that the gold content of these bags could not be associated with a specific crosscut or round, but as the bags were part of the bulk sample they were included in the overall metal balance across the mill.

Snowden considered the overall controls and the management of the shipping of the bulk bags, which was logistically a very complex operation, to have been handled extremely well and can confirm that more than 99% of the bags arrived at the mill intact and could be clearly identified for processing in the mill.

13.2.2 Contact Mill processing

The Contact Mill is an older, manually controlled mill, requiring experienced operators to ensure that the operational targets are achieved. The only automated control was the discharge of gravity concentrate from the Knelson concentrator, which was set in accordance to the gold loading found in the feed ore.

The mill flowsheet selected for the processing of the bulk sample was simple and achieved very good recoveries during the campaign. The flowsheet consisted of the following processing stages:

- Crushing and milling of the feed ore.
- Recovery of gravity concentrate in a Knelson concentrator and subsequent treatment of the concentrate on a Gemini table to produce a final gravity concentrate.
- Flotation of the cyclone overflow in a flotation circuit consisting of:
    - rougher flotation
    - scavenger flotation
    - cleaner flotation
    - spiral recovery of a heavy fraction from the final flotation tails
    - filtration (dewatering) of the final flotation concentrate.

The mill was operated mostly from Monday morning until Friday morning, with weekends used as downtime. At the end of each week the sumps were cleaned out, however, the circuit, including the ball mill, was not emptied to allow a full mass balance on a weekly basis.

In spite of the age of the mill, it performed well mechanically, with only a limited number of breakdowns. Snowden found the staff to be competent and diligent in the execution of their duties.

13.2.3 Metal accounting

All metal accounting samples were taken manually every hour, with composite samples made up for every six hours of operation. The key metal accounting data points were:

- Mill feed (mass flow measurement only).
- Final flotation concentrate (total mass and chemical analysis).
- Final flotation tails (chemical analysis) – Tails mass flow is calculated from the mill mass balance.
- Final gravity concentrate (total mass and chemical analysis).
- Final gravity middlings concentrate (total mass and chemical analysis).
- Final gravity tails (initially gravity tails were returned to process, but during the high grade crosscuts it was decided to retain these separately) (mass measurement and chemical analysis).

The mill feed rate was measured manually by stopping the feed conveyor every hour to take a belt cut, which was weighed to determine the mill mass feed flow rate. Of all the measurements taken, this was the one with the highest inherent error. However, comparison of the total mill feed based on this measurement compared favourably with the total sample weight as shown in Table 13.1.

It should be noted that there were several measurements made to understand the tonnage mined as a part of the bulk sample. Each underground round mined averaged around 100 tonnes, and this was the primary estimation method. A second measurement was made at the sample tower after crushing of the sample whereby the sample was weighed prior to shipping to Montana. This is likely to be the most accurate measure of the tonnes processed, and was weighed wet. The third measurement was the mill feed belt weight which is believed to be a measurement with a lower level of accuracy. The difference in measurements is around 3% and can be accounted for in differences in moisture content of the sample, calibration of the scales and the accuracy of the overall measurement.

Table 13.1 Bulk sample tonnage comparisons

Cross-cut	Sample tower tonnes	Shipment tonnes	Tonnes treated
426555 E	1,440.20	1,504.18	1,415.98
426585 E	2,249.40	2,223.27	2,168.57
426615 E	2,946.30	3,031.33	3,039.29
426645 E	1,934.20	1,965.52	1,874.80
615 L	1,477.50	1,523.45	1,802.93
Mixed bags		96.72	
Total	**10,047.60**	**10,344.47**	**10,301.57**

A spear sample of the final flotation concentrate was taken by the mill operators and analysed on site. A second spear sample was taken independently by a Pretivm staff member, using a 12 sample pattern and submitted separately to an external laboratory (Inspectorate America Corporation ("Inspectorate")) for analysis. A comparison of the various assays for the flotation concentrate samples indicates a difference in the Contact Mill assays against the Inspectorate assays for gold and is shown in Table 13.2. An analysis of the difference at different flotation concentrate grades shows that the difference varies depending on concentrate grade, but is similar for higher and lower grade concentrates. The overall average difference for gold is slightly higher than 5%, however, the final smelter settlements will dictate the finally agreed feed grades and recoveries. These data are not yet available.

Table 13.2 Comparison of weighted average of Inspectorate assays vs. the Contact Mill assays

Cross-cut	Inspectorate average Au (g/t)	Contact Mill average Au (g/t)	Difference for Au
426585 E	36.03	39.35	9.20%
426645 E	14.67	15.23	3.86%
615 L	107.10	115.04	7.41%
426555 E	26.93	27.54	2.25%
426615 E	147.14	159.46	8.38%

Table gravity concentrate was collected and the full production dried and subsampled via a riffle splitter. Due to the high grades achieved in the gravity concentrates, the gravity table was located in a separate security area with access only allowed in the presence of security personnel. Daily gravity concentrate production was stored in a lockable safe in the secure area, ensuring that no concentrate would be inadvertently lost. Similarly, all subsamples were only handled in the presence of security personnel. As shown in Table 13.3, a total of 3,645 ounces of gold were recovered via the gravity circuit.

A similar comparison between internal and external assays for the gravity concentrate will be undertaken as soon as the external assays are available.

Final flotation tails were sampled every hour by a mill operator, as the tails were transferred to the tailings storage facility (TSF). These samples were composited over four hours to generate six samples during a standard operating day. Although Snowden considers an automated sampling system a better solution, the manual samples did provide good representation of the flotation tails and when compared with results of an independent second 15 minute sample also composited over four hours, results were shown to be similar with no definite bias in either direction.

Final results of the metal accounting balance across the mill are shown in Table 13.3.

Table 13.3 Summary mill mass balance results of the Brucejack bulk sample processing

Cross-cut	Tonnes treated (t)	Float con. Au (oz)	Table (gravity) con. Au (oz)	Tails Au (oz)	Total Au (oz)	Au recovery	Average grade per round Au (g/t)
426585 E	2,169	173.9	93.5	12.9	280.4	95.4%	4.02
426555 E	1,416	92.9	102.2	5.8	200.9	97.1%	4.41
426645 E	1,875	68.6	62.3	6.4	137.3	95.4%	2.28
426615 E	2,878	1,289.3	2,290.8	61.4	3,641.5	98.3%	39.35
615 L (incl. mixed bags)	1,964	477.1	1,096.4	37.5	1,611.0	97.7%	25.52
Final clean-out		52.0			52.0		
Total	**10,302**	**2,101.9**	**3,645.3**	**124.0**	**5,923.2**	**97.9%**	**17.88**

Overall recoveries achieved during the treatment of the bulk sample were excellent at 97.9%

13.2.4 Summary of primary conclusions with respect to the bulk sample

Snowden considered the overall controls and the management of the shipping of the bulk bags, which was logistically a very complex operation, to have been handled extremely well and can confirm that more than 99% of the bags arrived at the mill intact and could be clearly identified for processing in the mill.

A relatively simple process flowsheet was used for the recovery of gold from the Brucejack bulk sample, achieving good overall recoveries of more than 97%.

Due to the age of the mill, control of the operation is largely manual, relying on the experience of operating staff. Snowden found the staff to be competent and diligent in the execution of their duties.

The comparison of the site concentrate analyses against the commercial laboratory shows an overall average difference for gold where the mill results are slightly higher than 5%. An analysis of the difference at different flotation concentrate grades shows that the difference varies depending on concentrate grade, but is similar for higher and lower grade concentrates. If the commercial laboratory is correct the average feed grade of the ore defined by processing will be around 5% lower than that reported in this document. The final smelter settlements, however, will dictate the finally agreed feed grades and recoveries. These data are not yet available.

According to the mill, 10,340 tonnes of ore were treated (10,048 tonnes from the weights from site), recovering 5,747 ounces of gold in concentrate and 176 ounces in tails and 52 ounces in plant clean-up material, providing a total gold in feed of 5,923 ounces at an average gold feed grade of 17.88 g/t.

Snowden considers the handling and treatment of the 10,000 tonne bulk sample to have been executed and managed well. Results are considered to be as accurate as can be expected for a bulk sample treatment campaign of this nature, providing Pretivm with a robust average gold grade for each crosscut as well as for the full 10,000 tonne bulk sample.

14 Mineral Resource estimates

Snowden has carried out an updated Mineral Resource estimate for the VOK mineralized area within the Brucejack Project. This estimate is an update of the previous November 2012 Mineral Resource (Jones, 2012c). As part of the update, a series of test estimates were run in the local test area around the bulk sample program. The results of this testwork are documented in Section 14.5.

The resource estimate for West Zone was not updated for this Mineral Resource as there has been very little additional drilling in this area. The West Zone estimate created as part of the April 2012 Mineral Resource (Jones, 2012a), has been documented in this report for completeness. The estimates are reported at a high grade cut-off for potential underground extraction.

As part of September 2012 Mineral Resource (Jones, 2012b), Snowden updated the adjacent, lower grade mineralized areas including Bridge Zone, Gossan Hill and Shore Zone. These estimates have not been updated as the additional drilling has focused on the higher grade VOK area. These estimates are not included in this report as they do not form part of the high grade resources.

14.1 Disclosure

Mineral Resources were prepared by Snowden under the supervision of the author, Ivor Jones. Mr Jones is an employee of Snowden Mining Industry Consultants.

The author, by way of experience and qualifications, is a Qualified Person as defined by NI 43-101 and is independent of Pretivm.

14.2 Known issues that materially affect mineral resources

The author is not aware of any permitting, legal, title, taxation, socio-economic, and marketing or political issues that could materially affect the Mineral Resource estimates.

14.3 Assumptions, methods and parameters

The estimates were prepared in the following steps:

- Data validation.
- Data preparation – this and subsequent steps are summarised below.
- Exploratory data analysis of data.
- Geological interpretation and modelling.
- Establishment of block models.
- Compositing of assay intervals.
- Consideration of grade outliers.
- Variogram analysis.
- Derivation of kriging plan.
- Grade value estimation.
- Deduction for prior mined volume.
- Classification of estimates with respect to CIM Definition Standards.
- Resource tabulation and resource reporting.

14.4 Data provided

The drillhole database used by Snowden for the resource estimate was provided by Caroline Vallat, from GeoSpark Consulting Inc, ("GeoSpark"). Caroline Vallat is independent of Pretivm. This database is in Microsoft Access format and contains collar, survey, assay and specific gravity data.

Digital terrain models (DTMs) for the topographic elevation and the base of the ice cap were provided by Pretivm, together with solids for the lithological domains and solids for mining depletion.

For the resource estimate, Snowden used all drillholes with collars lying inside of the area covered by the topography; which comprises a rectangular area with coordinates 425450 mE to 427550 mE and 6256450 mN to 6260550 mN. The data used in the estimates exclude intervals with no gold (Au) and silver (Ag) values.

The input data for the VOK Mineral Resource estimate comprises 922 drillholes totalling 218,127 m. These include:

- Nine historic drillholes (579 m).
- 490 surface drillholes drilled between 2009 and 2012 (173,619 m).
- 24 surface drillholes drilled in 2013 (5,200 m).
- 409 underground drillholes drilled in 2013 (38,840 m).

The input data for the West Zone estimate comprises 756 drillholes (63,208 m) including 439 underground drillholes (24,688 m), 269 historical surface drillholes (21,321 m) and 48 surface drillholes (17,199 m) completed since 2009.

Figure 9.13 illustrates the plan location of the diamond drilling.

The sample database and the topographic surface were reviewed and validated by Pretivm prior to being supplied to Snowden. Snowden carried out basic validation checks and found no material issues with the database supplied.

14.5 Bulk sample program testwork

The underground bulk sample results were used to investigate the local accuracy of the November 2012 Mineral Resource within the VOK, and to determine whether the estimation methodology could be improved for the December 2013 Mineral Resource.

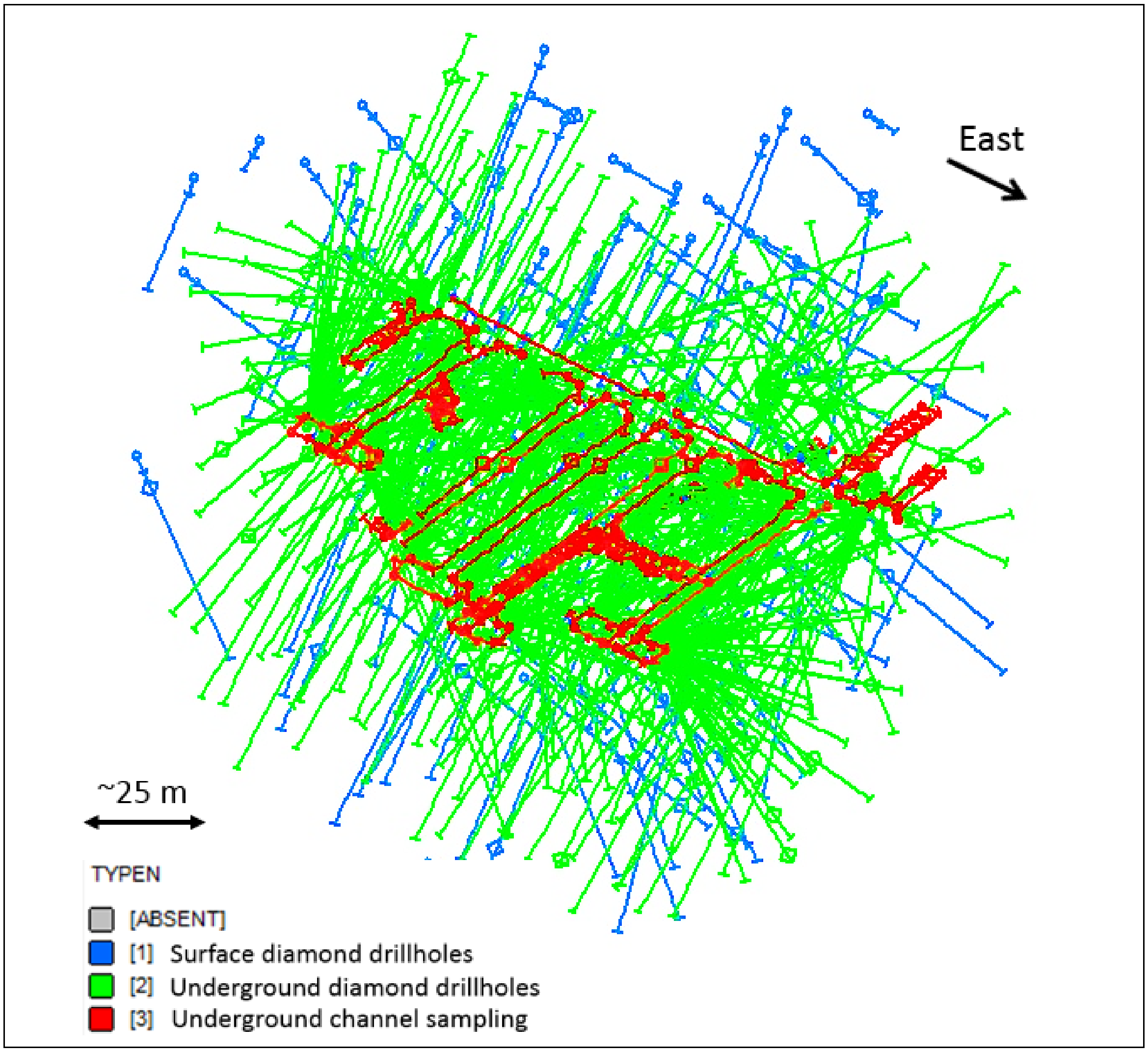
This test work has in part relied on comparisons between the test estimates and results from the bulk sample processing. However, the reader should be aware that there is a significant difference in the sample support for the resource estimate (each block in the resource estimate represents 2,700 tonnes whereas the bulk sample packages are around 100 tonnes), and the grade is not homogenous throughout any block. In other words, the grade can vary from a high grade side of the block to a low grade side of the block, whereas the block grade represents an average of the whole block. If the bulk sample happens to take a high grade part of the block, then the comparison will look like the resource estimate under-estimated the grade, and conversely if the bulk sample takes a low grade part of the block, then the comparison will look like the resource estimate over-estimated the grade in the block.

Therefore, whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally, it does however provide the best opportunity to fine-tune the estimate to some hard data.

14.5.1 Bias testwork

As part of the testwork, a series of test estimations were run to determine whether any bias exists between the surface diamond drilling, underground diamond drilling, underground channel samples, and chip samples. Only data within the local test area around the underground bulk sample area was used in this testwork (Figure 14.1).

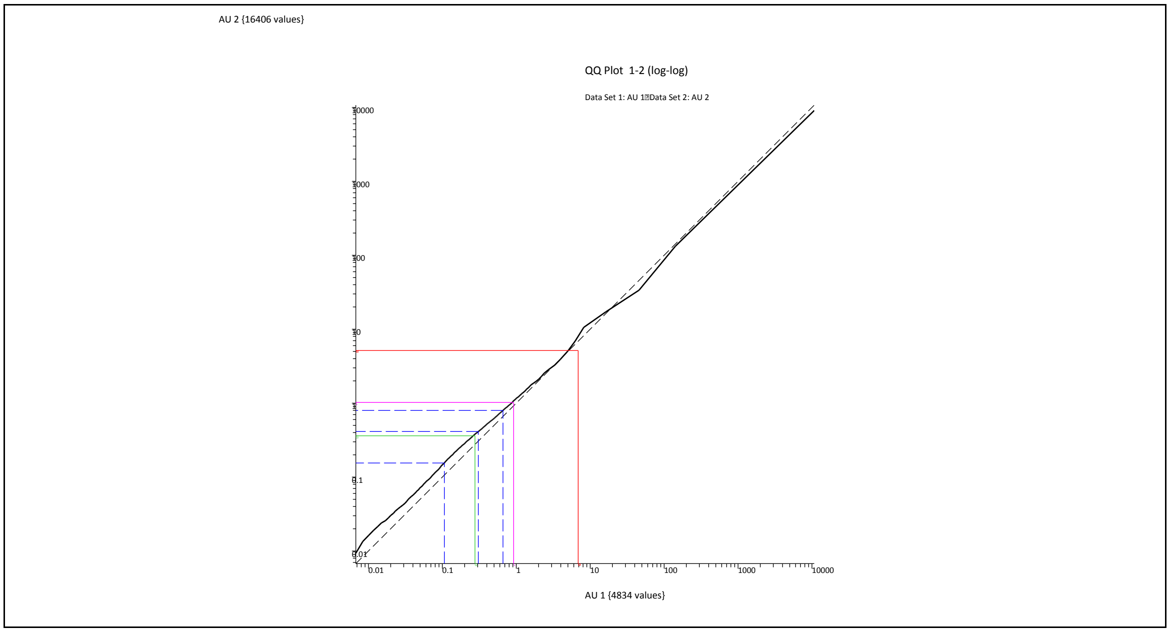
Figure 14.1 Oblique view of drillhole and channel data used for bias testwork



Surface diamond drilling versus underground diamond drilling

Initial analysis compared the assays of the surface diamond drilling to those of the underground diamond drilling. A quantile-quantile (Q-Q) plot was generated comparing the distributions of these two datasets within the test area (Figure 14.2).

Figure 14.2 Q-Q plot comparing surface (AU 1) and underground (AU 2) diamond drilling



The Q-Q plot shows that there is no obvious statistical bias between these two datasets. However, given the dominantly north-south orientation of the underground drilling, there is the potential for a directional bias in the sampling.

To test for a directional bias, estimates were generated with and without the underground drillholes using the updated 2013 estimation parameters. The estimates were compared to the final mill accounting results for each bulk sample crosscut (Table 14.1). The results of the testwork showed some directional bias, as removing the underground drillholes increased the local grade of the estimate within the bulk sample crosscuts particularly those crosscuts dominated by north-south mineralization (426615E), resulting in a better comparison to the mill results.

Table 14.1 Directional bias test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			No Underground Drilling			All Drilling		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555E	1,416	4.41	201	1,521	2.79	137	1,481	3.97	189
426645E	1,875	2.28	137	1,996	4.35	279	1,987	3.26	208
426585E	2,169	4.02	280	2,294	5.45	402	2,269	2.35	171
426615E	2,878	39.35	3,642	3,153	19.66	1,993	3,127	15.74	1,582
615L	1,964	25.52	1,611	1,578	33.35	1,692	1,574	38.42	1,945
Final clean out			52.0						
Total	**10,302**	17.88	5,923	10,541	13.29	4,503	10,438	12.20	4,096

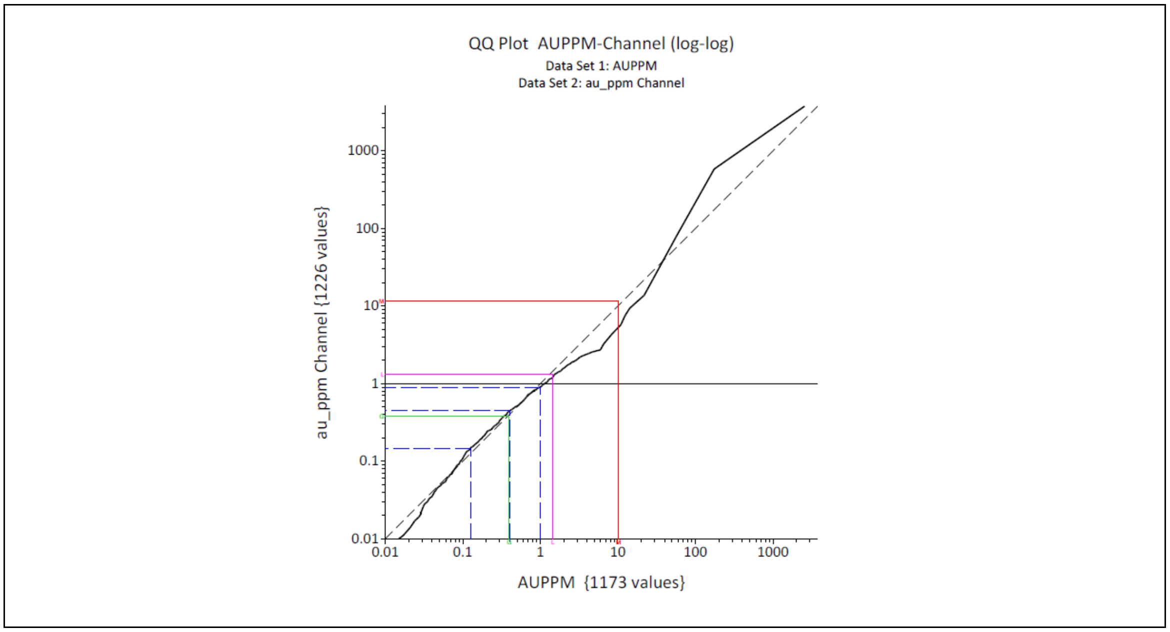
In the absence of a significant statistical bias between the drill all of the diamond drilling was combined for Mineral Resource estimation . Snowden recommends, based on the lower grade of the estimate that incorporates the underground drilling (Table 14.1), that future grade control drilling take into account all orientations of mineralization so as to not introduce a directional bias in the sampling and subsequently the estimate.

Underground channel samples versus underground chip samples

The channel samples and chip samples are mostly collocated in the bulk sample crosscuts. A nearest neighbour approach was used to determine the closest chip sample to each channel sample. This paired data was then used to generate a Q-Q plot to compare the distributions (Figure 14.3). The Q-Q plot shows that there is no bias between the channel and chip samples.

Given that the majority of the channel and chip samples are collocated, which can cause anomalies during estimation, and that the channel data can be more easily incorporated into the drillhole database, Snowden decided to use the channel samples for all further testwork.

Figure 14.3 Q-Q plot comparing underground chip (AUPPM) and underground channel (au_ppm Channel) samples



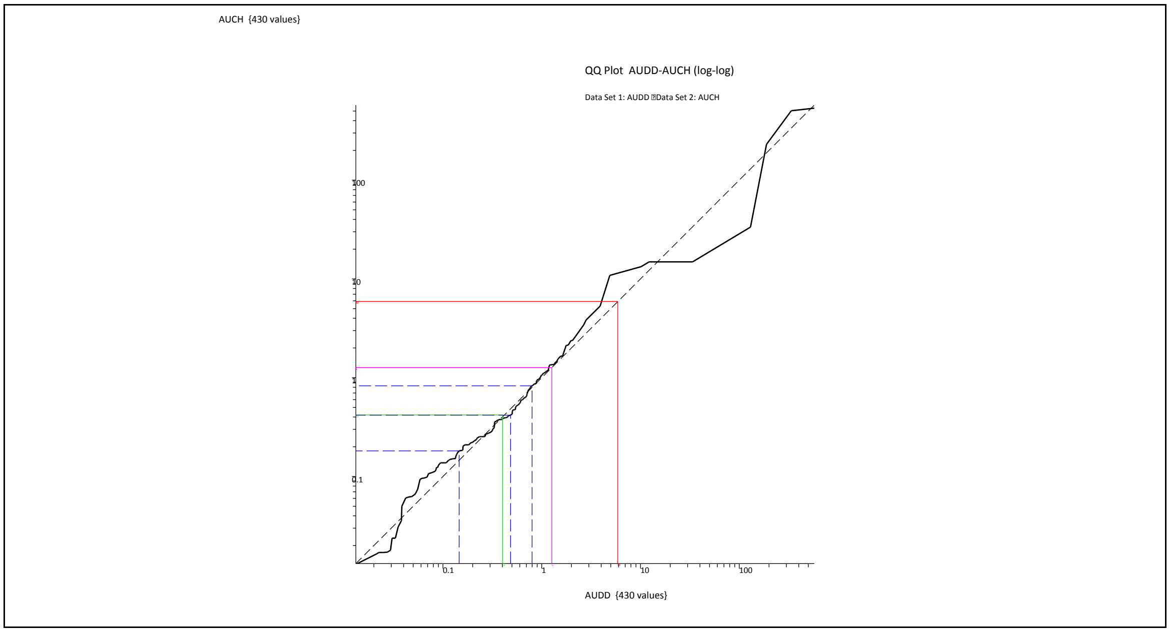
Diamond drilling versus underground channel samples

The final bias analysis compared the diamond drilling to the underground channel samples. A nearest neighbour approach was used to determine the closest diamond sample to each channel sample. This paired data was then used to generate a Q-Q plot to compare the distributions.

The Q-Q plot, which was generated from paired data collected using a search of 2.5 mE by 2.5 mN by 1 mRL, shows that there is no significant bias between the channel and diamond drillhole data (Figure 14.4).

While there is no significant bias evident between the channel samples and the drilling, the location of many of the channel samples in the centre of some of the higher grade mineralization does result in a local overestimation around the bulk sample crosscuts when the channel samples are used. Snowden therefore decided to remove the channel samples from the input data used in generating the final estimate. This is discussed in more detail in Section 14.5.3.

Figure 14.4 Q-Q plot comparing diamond drillholes (AUDD) and underground channel samples (AUCH) with restricted search



14.5.2 Validation of the estimation process – the November 2012 model

The final metal and tonnes from the mill accounting were initially compared to the November 2012 Mineral Resource estimate in order to assess the appropriateness of the modelling process (Table 14.2).

Table 14.2 2012 Mineral Resource versus Mill results

Mill			November 2012 Mineral Resource		
Tonnes	**Au (g/t)**	**Au (oz)**	**Tonnes**	**Au (g/t)**	**Au (oz)**
10,302	17.88	5,923	10,305	15.84	5,248

This is only to be expected from a deposit with such extreme grades. However, the knowledge gained from mapping and sampling in the bulk sample have shown that in some of the extreme grade drilling intersections, the continuity of mineralization is far greater than originally interpreted. Usually, an extreme grade would generally have continuity of at best a metre or two, whereas extreme grades in both the east-west (e.g., 615L and associated raises) and north-south (e.g., 426615 E crosscut and associated raises) components of the stockwork system at Brucejack are continuous up to the tens of metres scale. The outcome of this at Brucejack is that whilst the November 2012 estimate honours the input data, the high grade mineralization was under-estimated due to the lack of samples in the high grade, and the low grade was over-estimated – a function of the interpretation as to how to deal with the extreme grades.

Based on the bulk sample comparisons discussed above, Snowden concludes that the November 2012 Mineral Resource is a good indicator of contained metal content in the VOK deposit and was satisfactory for use in bulk underground mine planning, but that it is only locally accurate at the multiple block scale of the bulk sample area. It is however considered reasonably accurate on the scale of the bulk sample areas as a whole. As a result further testwork was undertaken to adjust the estimation methodology for the December 2013 Mineral Resource, to produce an estimate that is more responsive to the local high grades. However given the coarse gold and stockwork nature of the mineralization and the interpretation that continuous structures containing extreme grades are not likely to be prevalent, it is appropriate to incorporate a degree of smoothing into the estimate to provide a realistic representation of the mineralization.

14.5.3 December 2013 Mineral Resource testwork

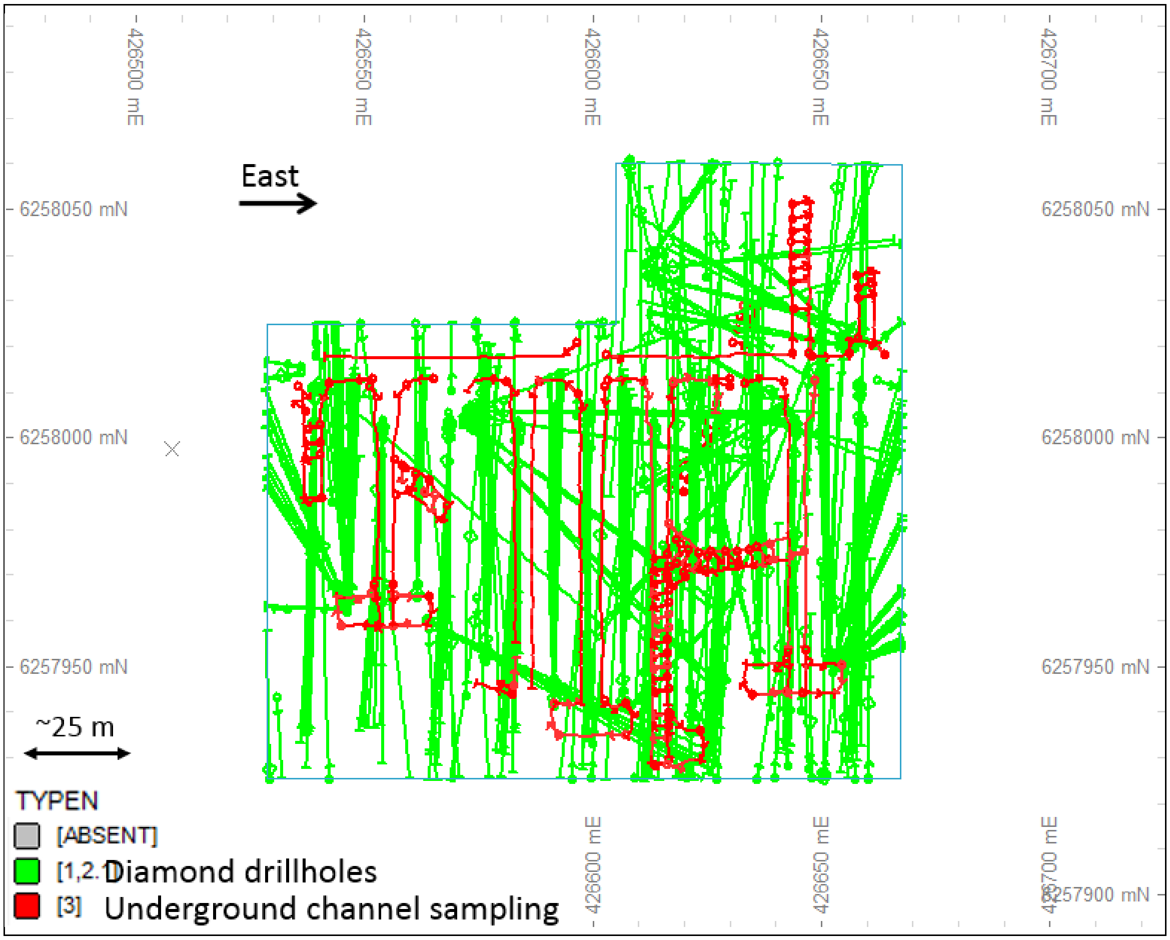
In order to produce an estimate that is more responsive to the local high grades, a series of test estimates were created inside the local test area surrounding the bulk sample crosscuts (Figure 14.5). The estimates were compared to the bulk sample mill results on a round-by-round basis, as well as more globally within the local test area. The analysis on a round-by-round basis is not entirely valid as the blocks are significantly larger than the size of the rounds, but it gave an indication as to which estimates were over-smoothing locally, and which were under-estimating the total contained metal. As such, the results are only used to give some local perspective and guidance to the grade estimates.

The updated database was used for the testwork, including the underground diamond drilling completed as part of the bulk sample program. Although some of the initial testwork used a preliminary database, it was effectively complete in the local test area. The updated mineralized corridor domains (see Section 14.6) were also used for the testwork.

The estimation testwork included:

- Looking at the use of channel samples to assist in defining the local grade more accurately around the bulk sample crosscut.
- Assessing the impact of constraining the north-south mineralization and estimating it separately to the dominantly east-west mineralized corridors.
- Adjusting the estimation parameters and methodology to reduce smoothing; including the method for reblocking the high grade MIK estimates, parent cell size, and search neighbourhood parameters.
- Comparing ordinary kriging and inverse distance weighted estimation methods.

Figure 14.5 Local test area used for estimation testwork



Channel sampling

Test estimates were run within the local test area to assess the impact of using the underground channel samples in the estimation process. As discussed previously (Section 14.5.1), chip samples were not used for any of the testwork or final estimation.

The test estimates were initially run using a parent block size of 5 mE by 5 mN by 5 mRL, with the updated search parameters and estimation methodology derived for the December 2013 Mineral Resource and no separate domain for the north-south mineralization. Two estimates were run, one using channel samples and the second without channel samples.

The resulting estimates were compared locally to the bulk sample mill results (Table 14.3). These results show that the use of the channel samples causes a local overestimation compared to the mill. The estimate using channel samples is overestimating in both the 426615E crosscut, which is dominated by north-south mineralization, and the crosscuts dominated by east-west mineralization (615L).

Excluding the channel samples results in an underestimation of the north-south mineralization with an improvement in the local estimation of the grade in the crosscuts and drift dominated by the east-west mineralization.

Table 14.3 Channel sample test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			Test Estimate No Channels			Test Estimate With Channels		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555	1,416	4.41	201	1,483	1.83	87	1,483	1.60	76
426645	1,875	2.28	137	1,986	2.77	177	1,986	2.73	174
426585E	2,169	4.02	280	2,264	2.60	189	2,264	1.94	141
426615E	2,878	39.35	3,642	3,125	7.80	783	3,125	64.47	6,476
615L	1,964	25.52	1,611	1,572	36.95	1,868	1,572	87.80	4,438
Final clean out			52.0						
Total	**10,302**	17.88	5,923	10,430	9.26	3,104	10,430	33.71	11,305

An additional test estimate was run using the channel samples; but treating the channels as a single drillhole and heavily restricting the number of composites allowed per drillhole (maximum of four composites per drillhole). This process improved the local response for the north-south mineralization compared to the mill results, however, it still resulted in an overestimation of the grade in the bulk sample development dominated by the higher grade east-west mineralization.

Table 14.4 Channel sample test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			Test Estimate With Channels – Restricted Search		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555	1,416	4.41	201	1,527	1.90	93
426645	1,875	2.28	137	2,012	4.39	284
426585E	2,169	4.02	280	2,324	2.15	161
426615E	2,878	39.35	3,642	3,189	29.66	3,041
615L	1,964	25.52	1,611	1,621	55.36	2,884
Final clean out			52.0			
Total	**10,302**	17.88	5,923	10,673	18.84	6,463

As a result of this testwork Snowden decided not to use the channel samples for the 2013 Mineral Resource estimate as they cause local overestimation in the bulk sample area. A further consideration from this testwork was that the block size of 5 m by 5 m by 5 m provided an unrealistic expectation of the accuracy of the block estimate, and provided a indication that the block size should not be adjusted from the 10 m by 10 m by 10 m block size of the November 2012 estimate.

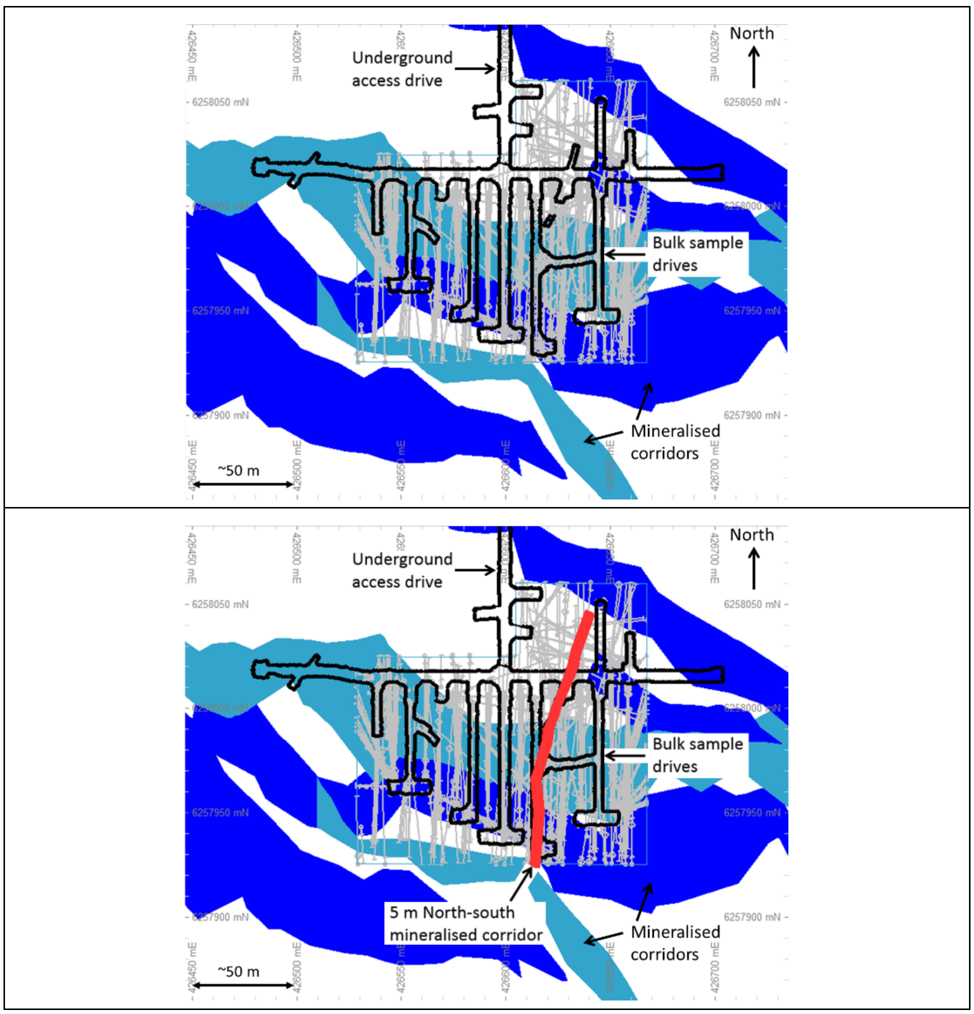
Constraint of north-south mineralization

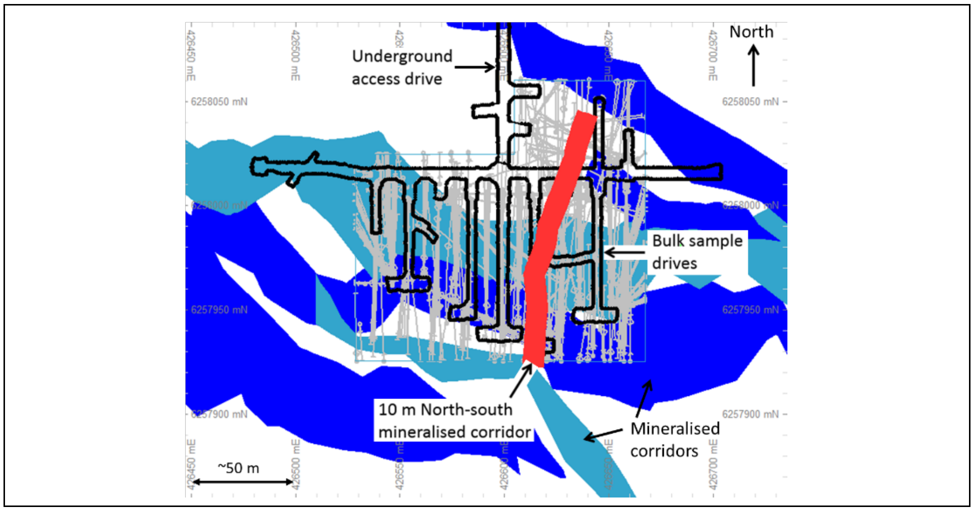
Test estimates were run within the local test area to assess the impact of constraining the north-south mineralization and estimating it separately to the dominantly east-west mineralized corridors.

The initial test estimates were run using a parent block size of 5 mE by 5 mN by 5 mRL with the November 2012 search parameters and estimation methodology, with reblocking of the high grade and probability estimates to 10 mE by 10 mN by 10 mRL. Channel samples were not used for the test estimates.

Domains were defined around the main north-south mineralization by Pretivm. Given the nature of the mineralization being a stockwork system, it was not possible to define a distinct edge to the high-grade mineralization. As a result, Pretivm defined a 'package' around the area containing dominantly north-south veining. It should be noted that north-south veining is present throughout the mineralized corridors and that, although the area defined in the 'package' around the 426615E bulk sample drive contains more intensive mineralization in this orientation, it also contains numerous veins of other orientations (including east-west). Pretivm generated two wireframes to constrain the north-south 'package' of mineralization: a 10 m wide envelope; and a 5 m wide envelope.

Figure 14.6 North-south mineralized domains without north-south constraints (top), 5 m north-south constraint (middle) and 10 m north-south constraint (bottom)





A base case test estimate was run with no separate constraint or estimation of the north-south mineralization apart from that incorporated into the main mineralized corridors. This was then compared to a second estimate which was run using the 10 m north-south constraint with a hard boundary and appropriately oriented estimation parameters.

The use of the 10 m north-south constraint with hard boundaries resulted in a local overestimation of the metal in the bulk sample crosscuts compared to the metal produced by the mill (Table 14.5). While this method reproduced the metal in the 426615E crosscut which is dominated by north-south mineralization, the estimates were significantly over-estimated in crosscuts dominated by east-west mineralization when compared to that produced by the mill.

The base case without the north-south constraint, resulted in an underestimation of the north-south mineralization but a slightly more realistic estimate of the grade of the crosscuts dominated by the east-west mineralization (Table 14.5). This is likely a result of a directional bias in the close spaced fan drilling informing this area, as almost all of this drilling is orientated north-south and may not have sufficiently sampled the north-south trending structure. Based on this observation Snowden considers that future grade control drilling should be designed to ensure both orientations of mineralization are properly sampled.

Table 14.5 North-south 10 m domain test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			Test estimate No North-South Constraint			Test estimate 10 m North-South Constraint		
	Tonnes	**Au (g/t)**	**Au (oz)**	**Tonnes**	**Au (g/t)**	**Au (oz)**	**Tonnes**	**Au (g/t)**	**Au (oz)**
426555	1,416	4.41	201	1,511	1.74	85	1,511	1.75	85
426645	1,875	2.28	137	2,011	4.53	293	2,011	4.15	268
426585E	2,169	4.02	280	2,333	2.26	170	2,333	2.72	204
426615E	2,878	39.35	3,642	3,165	11.75	1,195	3,168	34.90	3,555
615L	1,964	25.52	1,611	1,617	44.56	2,316	1,617	64.36	3,345
Final clean out			52.0						
Total	**10,302**	17.88	5,923	10,636	11.87	4,059	10,639	21.80	7,458

A third estimate was run using the 5 m north-south constraint with a soft boundary for estimation. A tighter search was imposed on this test estimate with the estimation parameters orientated north-south within the north-south domain to try and reproduce the north-south trend. The use of a soft boundary is more in line with the style of mineralization; which is a stockwork system with the veining in multiple orientations.

The 5 m constraint resulted in a similar pattern to that seen in the estimate which did not use a north-south constraint, although less metal was produced (Table 14.6).

Table 14.6 North-south 5 m domain test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			Test Estimate 5 m North-South Constraint		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555	1,416	4.41	201	1,483	1.75	84
426645	1,875	2.28	137	1,986	2.56	164
426585E	2,169	4.02	280	2,264	2.59	188
426615E	2,878	39.35	3,642	3,125	8.22	826
615L	1,964	25.52	1,611	1,572	36.44	1,842
Final clean out			52.0			
Total	**10,302**	17.88	5,923	10,430	9.25	3,103

Constraint of the north-south mineralization may be possible during grade control, however these features are currently only defined in a single ‘package’ within the local test area and cannot be extrapolated into the rest of the resource. Snowden reviewed the orientation of all high grade veins in the vicinity of the test area (based on geological logging) to try and define areas with similar north-south mineralization, however no definite north-south ‘packages’ could be defined. This is expected given the stockwork nature of the mineralization.

The testwork indicates that the estimate without any north-south constraint is a better local predictor of the contained metal in the dominant east-west stockwork mineralization system, albeit at the cost of underestimating the contained metal in the north-south components of the stockwork mineralization.

Based on the outcomes of the testwork and bulk sample analyses, Snowden and Pretivm agreed to adopt the more conservative approach of not constraining the north-south mineralization in the generation of the December 2013 Mineral Resource estimate. As a consequence the model would be reasonably locally correct for the dominant east-west component of the mineralization system, but would underestimate (both locally and globally) the contained metal in those parts of the deposit with significant north-south mineralization.

Estimation parameters and methodology

High grade reblocking

For the November 2012 Mineral Resource, the high grade MIK estimates were reblocked to 20 mE by 20 mN by 20 mRL which was twice the parent cell size, to ensure adequate smoothing of the extreme high grades and reduce the impact of extreme grades. Given that the testwork shows that the November 2012 Mineral Resource is over-smoothed (Section 14.5.2), the issue of reblocking was reviewed for the December 2013 Mineral Resource.

To test the impact of changing the reblocking of the high grade, the November 2012 Mineral Resource was regenerated using the 10 mE by 10 mN by 10 mRL parent cell size. The two estimates were then compared with the processing results from the bulk sample crosscuts.

The change in the reblocking of the high grade gave results that were more consistent with the trends observed from the mill results (less grade smearing) than the original November 2012 Mineral Resource. Whilst it was not appropriate to compare against the resource model for the reasons discussed at the start to Section 4.5, the observed trends indicated the reblocking to the parent cell size produced an estimate that was a better local representation of the high grade mineralization in the relevant crosscut/drift.

As a result of this testwork Snowden decided to use the high grade reblocking to the parent cell size (10 mE by 10 mN by 10 mRL) for the December 2013 mineral resource estimate.

Parent cell size

With the increased density of drilling data in the local test area, the use of a smaller 5 mE by 5 mN by 5 mRL parent cell in this area was reviewed. The initial testwork was carried out using the smaller parent cell, however, review of the estimates against the bulk sample mill results indicated that using the original 10 mE by 10 mN by 10 mRL parent cell provides a better reproduction of the metal locally within the bulk sample crosscuts (Table 14.7). Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally (section 4.5), it does provide the best opportunity to fine-tune the estimate to some hard data.

Kriging neighbourhood analysis also indicates that the larger parent cell produces a higher kriging efficiency and slope of regression. This reflects less estimation error in the estimate (Olssen and Jones, 2013).

Table 14.7 Parent cell size test estimates versus Mill results by bulk sample crosscut

Crosscut	Mill			Test Estimate 5 mE by 5 mN by 5 mRL			Test Estimate 10 mE by 10 mN by 10 mRL		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555E	1,416	4.41	201	1,483	1.83	87	1,481	3.97	189
426585E	2,169	4.02	280	2,264	2.60	189	2,269	2.35	171
426615E	2,878	39.35	3,642	3,125	7.80	783	3,127	15.74	1,582
426645E	1,875	2.28	137	1,986	2.77	177	1,987	3.26	208
615L	1,964	25.52	1,611	1,572	36.95	1,868	1,574	38.42	1,945
Final clean out			52.0						
Total	**10,302**	17.88	5,923	10,430	9.26	3,104	10,438	12.20	4,096

Globally within the local test area there is little difference in total metal between the two methods (Table 14.8: 838 koz versus 833 koz), however the 10 mE by 10 mN by 10 mRL parent cell is less selective (it better represents the nature of the mineralization) than the smaller parent cell. This 10 mE by 10 mN by 10 mRL estimate is still more selective than the previous November 2012 Mineral Resource however, and given the stockwork nature of the mineralization, some smoothing is required to provide an estimate with minimal estimation error.

Table 14.8 Parent cell size test estimates within local test area above a 5 g/t AuEq cut-off

Model	Tonnes	Au g/t	Au (koz)
5 mE by 5 mN by 5 mRL parent cell	1,089	23.93	838
10 mE by 10 mN by 10 mRL parent cell	1,326	19.54	833

As a result of this testwork Snowden decided to retain the 10 mE by 10 mN by 10 mRL parent cell size for the entire resource.

<u>Search parameters</u>

The testwork carried out has shown that the lower grades are typically locally overestimated, while the higher grades are underestimated locally in the November 2012 Mineral Resource estimate. As a result, the search parameters were reviewed and adjusted as part of the local area testwork to produce an estimate that is more responsive to the local high grades. The aim of the adjustments was:

- High grade domains - low grade population
  - Reduce the minimum number of samples to reduce smoothing in the low grade estimate and overall decrease the average of the lower grades.
- High grade domains - high grade population
  - Maintain smoothing in the high grade estimates to ensure that the extreme grades are not given too much influence, while reducing the spread of the high grades by tightening the search.
- High grade domains - probability
  - Reduce smoothing in the probability estimate to ensure that high grades are not pushed too far into the lower grade areas and to increase the probability in the higher grade areas, resulting in an increase in the average of the higher grades in such areas.

A comparison between the November 2012 and December 2013 estimation and search parameters for the VOK is provided in Table 14.9.

Table 14.9 November 2012 versus December 2013 estimation and search parameters within high grade mineralized domains for the VOK

Estimate	Parameter	November 2012	December 2013
High grade domains - low grade population	Estimation method	OK	OK
	Parent cell size	10 mE by 10 mN by 10 mRL	10 mE by 10 mN by 10 mRL
	Reblocking cell size	10 mE by 10 mN by 10 mRL	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	60 m by 100 m by 20 m	60 m by 100 m by 20 m
	Minimum samples – pass 1	20	12
	Maximum samples – pass 1	26	26
	Search ellipse – pass 2	120 m by 200 m by 40 m	120 m by 200 m by 40 m
	Minimum samples – pass 2	8	8
	Maximum samples – pass 2	26	26
	Maximum composites per drillhole	8	8
High grade domains - high grade population	Estimation method	MIK	MIK
	Parent cell size	2.5 mE by 2.5 mN by 2.5 mRL	2.5 mE by 2.5 mN by 2.5 mRL
	Reblocking cell size	20 mE by 20 mN by 20 mRL	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	50 m by 50 m by 20 m	35 m by 35 m by 20 m
	Minimum samples – pass 1	8	12
	Maximum samples – pass 1	20	16
	Search ellipse – pass 2	150 m by 150 m by 60 m	105 m by 105 m by 60 m
	Minimum samples – pass 2	2	2
	Maximum samples – pass 2	6	6
	Maximum composites per drillhole	8	8
High grade domains - probability	Estimation method	MIK	MIK
	Parent cell size	2.5 mE by 2.5 mN by 2.5 mRL	2.5 mE by 2.5 mN by 2.5 mRL
	Reblocking cell size	10 mE by 10 mN by 10 mRL	10 mE by 10 mN by 10 mRL
	Search ellipse – pass 1	35 m by 35 m by 10 m	35 m by 35 m by 15 m
	Minimum samples – pass 1	5	12
	Maximum samples – pass 1	20	16
	Search ellipse – pass 2	70 m by 70 m by 20 m	70 m by 70 m by 30 m
	Minimum samples – pass 2	5	2
	Maximum samples – pass 2	50	6
	Maximum composites per drillhole	6	8

December 2013 Mineral Resource

Based on the testwork described above, the updated 2013 December Mineral Resource was run using:

- No channel samples.
- No separate domain or estimation of the north-south mineralization apart from that incorporated into the main mineralized corridors.
- 10 mE by 10 mN by 10 mRL parent cells for the entire VOK.
- High grade MIK estimates reblocked to the parent cell size.
- Updated search parameters (Table 14.9).

Comparison of the updated Mineral Resource to the previous November 2012 Mineral Resource and the bulk sample results shows that the updated estimate is more locally accurate than the previous Mineral Resource (Table 14.10). The updated Mineral Resource underestimates the north-south mineralization which may result in additional ounces if more of these features are discovered during mining. Snowden considers that the underestimation is a function of an orientation bias in the underground drilling which is aligned with the highest grade mineralization. Whilst it is not entirely valid to compare the results of the bulk sample with the resource estimate locally, it does provide the best opportunity to fine-tune the estimate to some hard data. The reader should be warned that the results are only used to give some local perspective to the grade estimates.

Table 14.10 December 2013 Mineral Resource versus November 2012 Mineral Resource and Mill results by bulk sample crosscut

Crosscut	Mill			November 2012 Mineral Resource			December 2013 Mineral Resource		
	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
426555	1,416	4.41	201	1,491	4.89	234	1,481	3.97	189
426645	1,875	2.28	137	1,959	5.64	355	1,987	3.26	208
426585E	2,169	4.02	280	2,231	12.12	869	2,269	2.35	171
426615E	2,878	39.35	3,642	3,089	18.88	1,876	3,127	15.74	1,582
615L	1,964	25.52	1,611	1,535	38.78	1,914	1,574	38.42	1,945
Final clean out			52.0						
Total	**10,302**	17.88	5,923	10,305	15.84	5,248	10,438	12.20	4,096

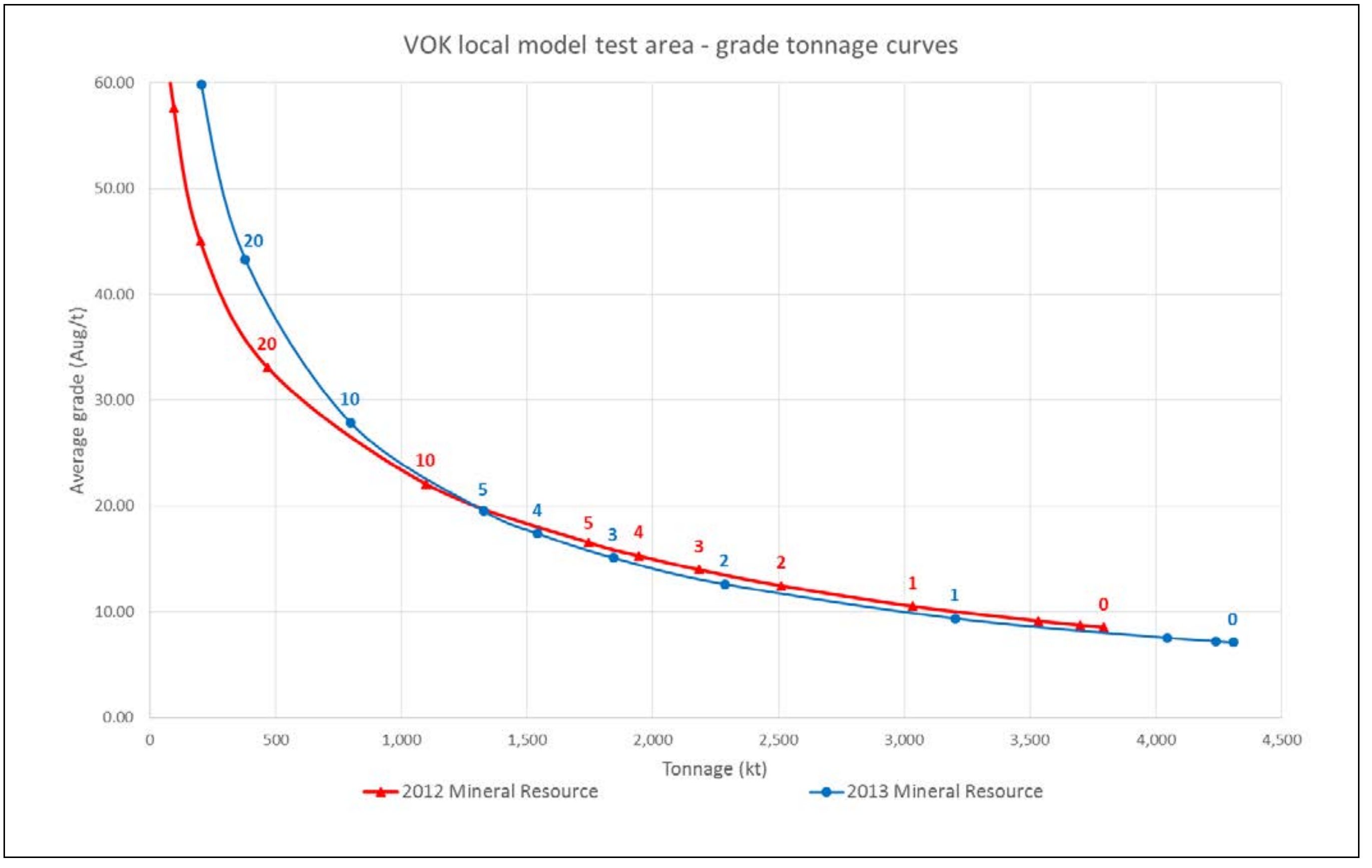
A more global comparison of the November 2012 Mineral Resource and the December 2013 Mineral Resource within the local test area highlights the additional selectivity in the updated 2013 Mineral Resource. There are less tonnes at a higher grade in the December 2013 Mineral Resource above a 5 g/t AuEq cut-off (Table 14.11 and Figure 14.7) than in the November 2012 estimate.

Note the change in total tonnes is a result of minor changes in the mineralized corridor domains and the change in the bulk density as discussed in Sections 11.2.2 and 14.12.

Table 14.11 November 2012 versus December 2013 Mineral Resource estimates within local test area above a 5 g/t AuEq cut-off

Model	Tonnes	Au g/t	Au (koz)
November 2012 Mineral Resource	1,745	16.54	928
December 2013 Mineral Resource	1,326	19.54	833

Figure 14.7 November 2012 versus December 2013 Mineral Resource estimates within local test area – grade tonnage curves with AuEq cut-off grades annotated



Ordinary kriging and inverse distance weighted check modelling

Inverse distance weighted (ID) and ordinary kriged (OK) estimates were also run using the 2013 dataset, for comparison to the December 2013 Mineral Resource. Inverse distance weighting to the power of 1, 2 and 3 were tested. The 2013 Mineral Resource estimation parameters were used with variable top cuts of 2,000 g/t Au, 1,500 g/t Au, 700 g/t Au (99.9th percentile) and 85 g/t Au (99.5th percentile) as well as estimates without a top cut.

Each estimation method was compared to the underground bulk sample results and the December 2013 Mineral Resource by reporting the total ounces within the bulk sample crosscuts (Table 14.12). Details of the grade for each estimation method are included in Table 14.13; all estimates comprised 8,972 tonnes of material within the bulk sample crosscuts.

Table 14.12 Ordinary kriging and inverse distance weighted estimated versus Mill results within bulk sample – total gold ounces

Model	Mill	Top cut (Au g/t)				
		No top cut	2000	1500	700	85
ID1	5,923	8,313	4,626	3,931	2,755	837
ID2	5,923	8,541	4,716	4,007	2,807	826
ID3	5,923	8,351	4,640	3,963	2,811	810
OK	5,923	6,932	3,925	3,440	2,582	790
December 2013 Mineral Resource		4,096				

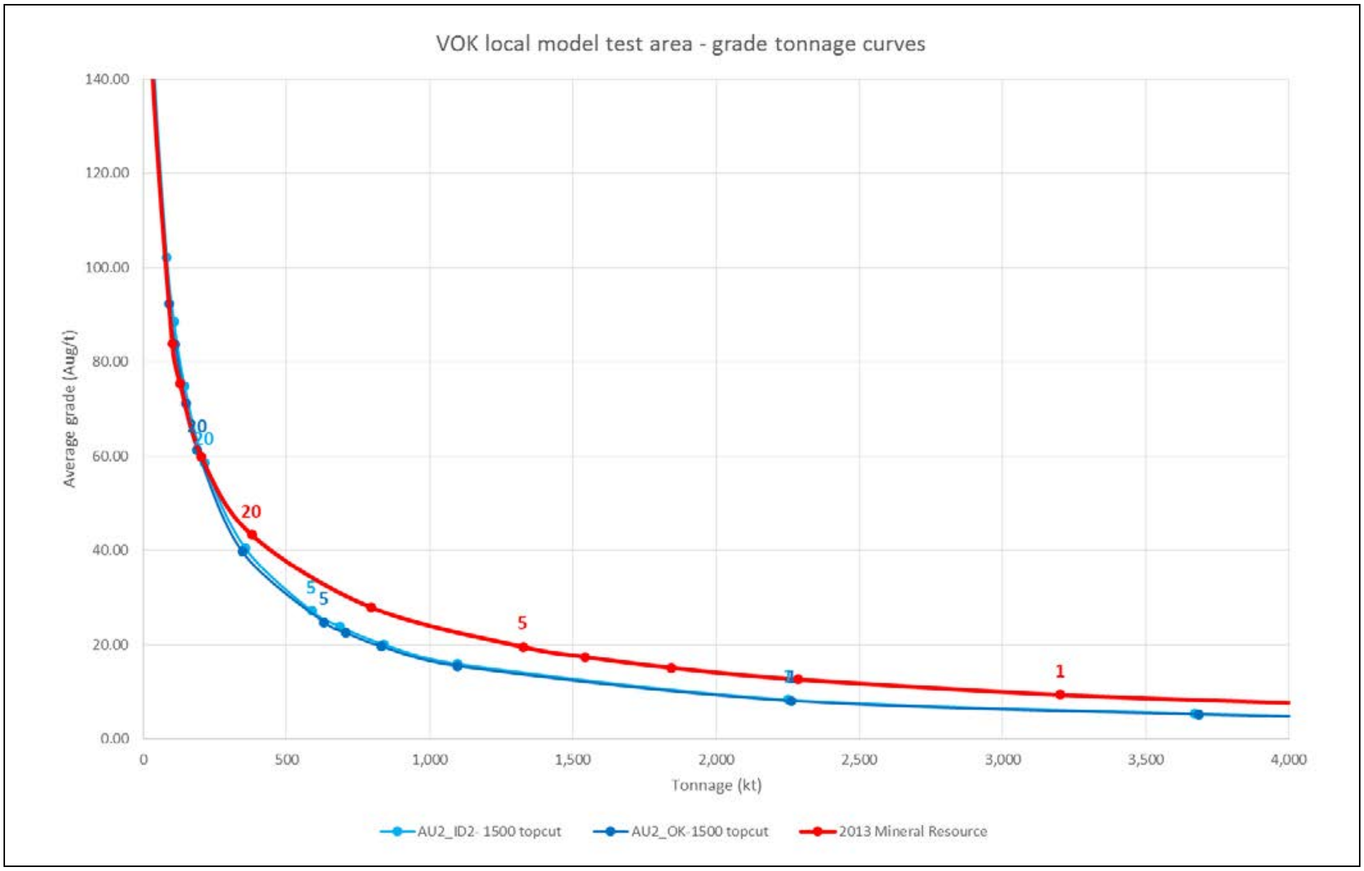
Table 14.13 Ordinary kriging and inverse distance weighted estimated within bulk sample – grade details

Model	Mill	Top cut (Au g/t)				
		No top cut	2000	1500	700	85
ID1	16.08	28.78	15.99	13.59	9.51	2.86
ID2	16.08	29.57	16.31	13.85	9.69	2.82
ID3	16.08	28.91	16.05	13.70	9.71	2.77
OK	16.08	24.00	13.58	11.90	8.92	2.71
December 2013 Mineral Resource		12.20				

These results show that using inverse distance weighting or ordinary kriging without applying a top cut results in severe overestimation of the total contained ounces within the study area. Using a 'typical' top cut based on the 99.9th percentile (700 g/t Au), or the more common 99.5th percentile (85 g/t Au), results in a significant underestimation of the total contained ounces. Inverse distance squared using a top cut of 1500 g/t Au gives the closest result to that achieved by the December 2013 Mineral Resource. However, it would be unusual to apply such a high topcut during grade estimation to such an extremely skewed distribution, and the choice of topcut and method using these techniques is entirely subjective. Grade trends of the input drillhole data were significantly better reproduced using the non-linear estimation methodology (December 2013 Mineral Resource estimate) than using the linear estimation techniques (OK, ID) with a topcut.

A more global comparison of the estimates was carried out by comparing the grade tonnage curves within the local test area. Figure 14.8 shows the grade tonnage curves for the December 2013 Mineral Resource as compared to the inverse distance squared estimate and ordinary kriged estimate using a 1500 g/t Au top cut.

Figure 14.8 December 2013 Mineral Resource estimate versus inverse distance squared and ordinary kriged estimates within local test area – grade tonnage curves with AuEq cut-off grades annotated



The grade tonnage curves show that the inverse distance weighted and ordinary kriged estimates give similar results, with both being more responsive to the drill grades than the Mineral Resource estimate is. Whilst these methods seem reasonable at first glance, they do not take into account the relatively random occurrence of high grade occurrences of mineralization scattered throughout the low grade mineralization which play an important part in the overall grade of the mineralization. This level of selectivity is not considered appropriate and is a function of the inverse distance weighted and ordinary kriged estimates not accounting for this style of mineralization.

14.6 Geological interpretation and modelling

14.6.1 VOK

The mineralization in the VOK exists as steeply dipping semi-concordant (to stratigraphy) and discordant pod-like zones hosted in stockwork vein systems within the volcanic and volcaniclastic sequence. High grade mineralization zones appear to be spatially associated, at least in part, with intensely silicified zones resulting from local silica flooding and over-pressure caprock formation. High grade mineralization occurs both in the main east-west trending vein stockwork system, as well as in the rarer north-south trending part of the system. Snowden notes that Pretivm has taken these various observations into consideration in the modelling of the mineralization domains for the VOK.

Within the bounding volcanic siltstone and sandstone ("VSF") and porphyry rocks, the mineralization forms a series of sub-vertical domains. While there is some indication of a hard boundary between the VSF and porphyry in places, the mineralization crosses this contact in other places and the two rock types appear to have similar statistical characteristics. As a result, the two lithological units were combined for estimation but the mineralized interpretation used the lithological boundaries as a guide where the contact appears to be hard.

The boundary between the VSF and polylithic conglomerate/porphyry mineralized domains is a hard contact in places due to the presence of a largely barren siliceous unit (or siliceous caprock) along most of the contact. There are places where this siliceous unit does not exist and the boundary is gradational, with the subvertical mineralized corridors crossing the contact.

Lithological interpretations were used together with a nominal 1 g/t Au to 3 g/t Au cut-off grade to define high grade corridors based on analysis of the statistical grade populations (Figure 14.9 and Figure 14.10).

14.6.2 West Zone

West Zone was interpreted for the April 2012 estimate using a nominal 0.3 g/t Au cut-off grade (Jones, 2012a). There are no high grade corridors defined at West Zone.

14.6.3 Domains used for modelling

Review of the grade distributions for the mineralized corridors and different rock types in the VOK indicates that they are statistically similar. As a result, in controlling the estimation each mineralized corridor was estimated using hard boundaries but where the corridors have been interpreted to cross the lithological boundaries, these contacts were treated as soft. The surrounding low grade (background) domain was estimated as a single domain using soft boundaries but excluding the high grade population from within the high grade domains.

The mineralized corridors within the VOK change orientation locally. This was addressed by using a series of 'search domains' with locally adjusted orientations applied for estimation. The boundaries between these search domains were treated as soft for estimation.

The West Zone was estimated using a single mineralized domain.

Figure 14.9 Cross section showing lithological interpretation and mineralized domain interpretation in the VOK

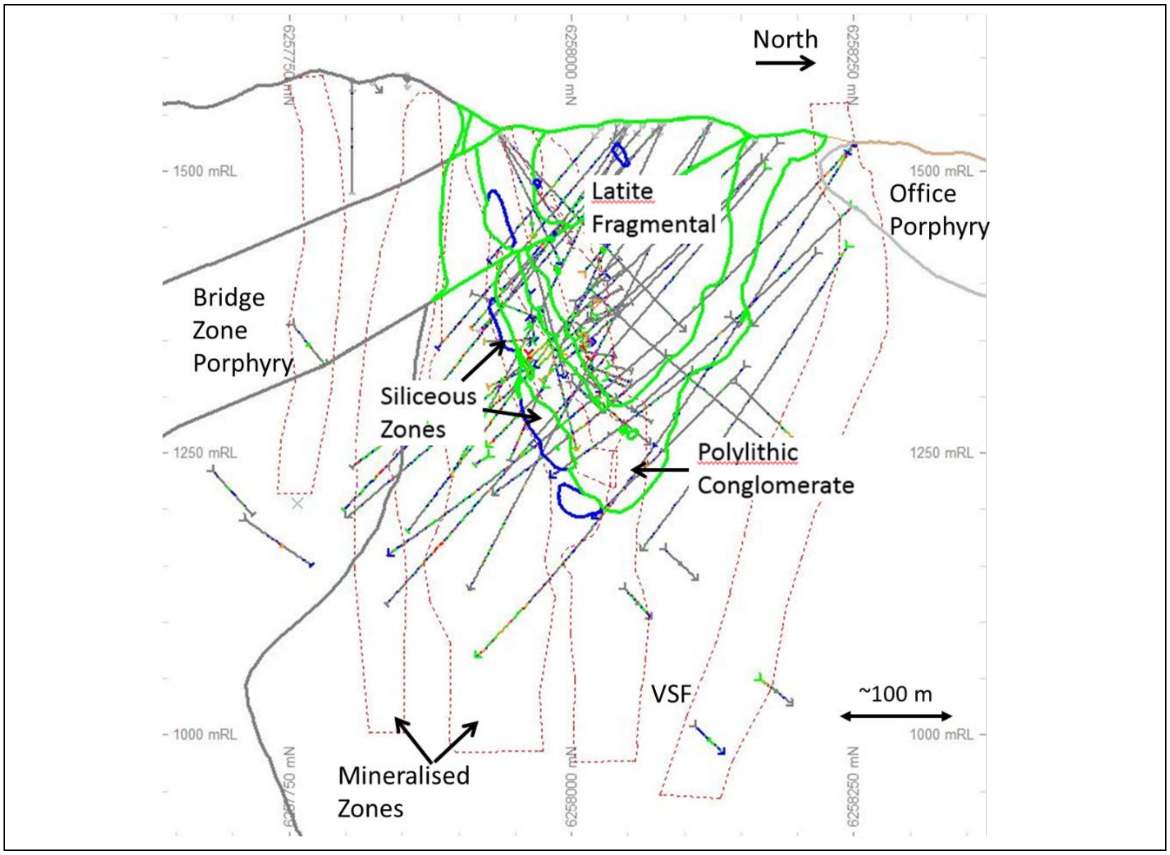
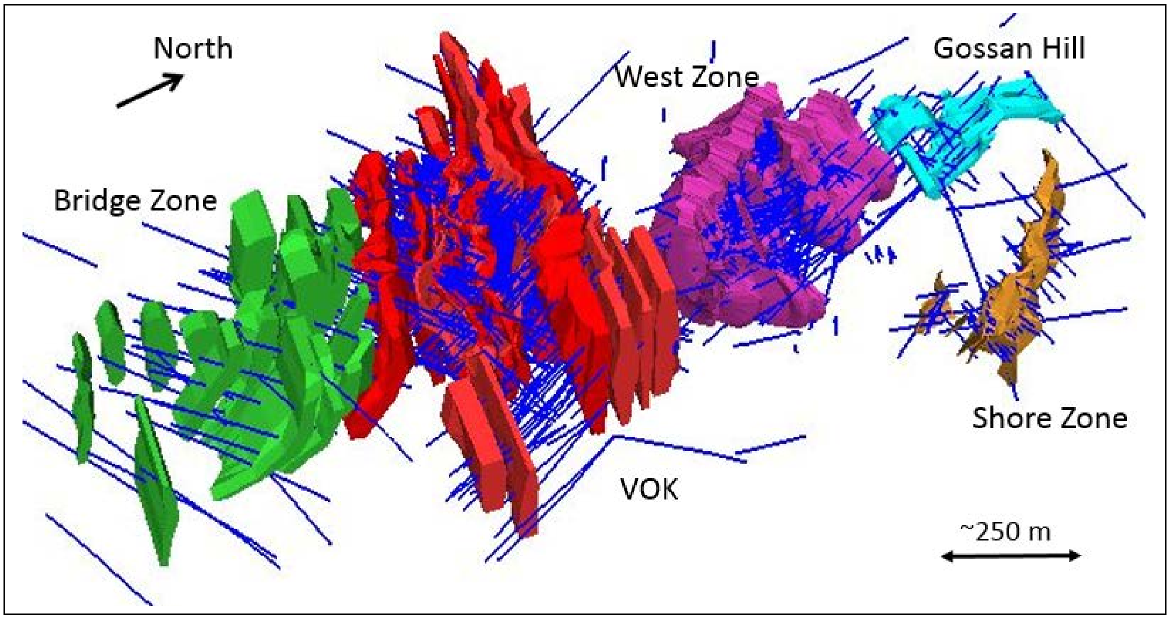


Figure 14.10 Orthogonal view of mineralized high grade domain interpretation



14.7 Compositing of assay intervals

All data was composited to the dominant sample length of 1.5 m prior to analysis and estimation.

14.7.1 Summary statistics

Statistical analysis of the gold and silver data was carried out by lithological domain (in the VOK) and mineralized domain (high grade and low grade).

Review of the statistics in the VOK indicated that the grade distributions for the mineralization within the different lithologies are very similar and as a result these were combined for analysis.

Initial review of the individual high grade corridors within each area showed that they have similar statistical distributions and hence the individual corridors within each area were combined for statistical analysis and variography. This fits with the interpretation of the stockwork mineralization system being superimposed on the underlying volcanic and volcaniclastic sequence.

All high grade domains, including the West Zone domain, exhibit a strong positive skewness with high coefficient of variation and extreme grades. The low grade domains also show positively skewed distributions but have lower coefficients of variation and few extreme grades.

Table 14.14 summarizes the statistics for gold and silver for the mineralized domains. Due to clustering of the drilling, the data for both West Zone and the VOK have been declustered for statistical analysis.

Table 14.14 Summary statistics of composited data for mineralized domains – VOK

Statistic	Gold g/t	Silver g/t
Samples	79,699	79,699
Minimum	0.00	0.25
Maximum	16,552	9,383
Mean	2.57	8.59
Standard deviation	69.54	41.94
CV	27.03	4.88
Variance	4,836	1,759
Skewness	109.40	70.11

Table 14.15 Summary statistics of composited data for mineralized domains – West Zone

Statistic	Gold g/t	Silver g/t
Samples	33,089	33,089
Minimum	0.00	0.25
Maximum	1,657	37,636
Mean	1.40	30.66
Standard deviation	17	219
CV	11.84	7.15
Variance	275	48,020
Skewness	66.98	62.65

14.7.2 Extreme values – gold and silver

Between 2% and 5% of the data within the mineralized high grade domains appears to form a separate higher grade population which contains a significant number of extreme grades. The treatment of these extreme grades is discussed in the following section.

14.8 Consideration of grade outliers and estimation method

Assay populations from gold deposits are generally skewed and contain high grade outliers that can introduce bias to mineral resource estimates.

Both the West Zone and the VOK exhibit extremely skewed grade populations where the high grades and the majority of the metal are located in less than 5% of the data, with individual raw gold grades of up to around 41,500 g/t Au. These grades have been shown from the mining to be a normal part of the mineralization, in some instances continuous, and definitely not anomalous. In addition, a review of the upper tail of the CDF shows that the extreme grade population is continuous and does not break down, supporting this observation. As a result of this population distribution, standard estimation techniques have been found to significantly over smooth the grades.

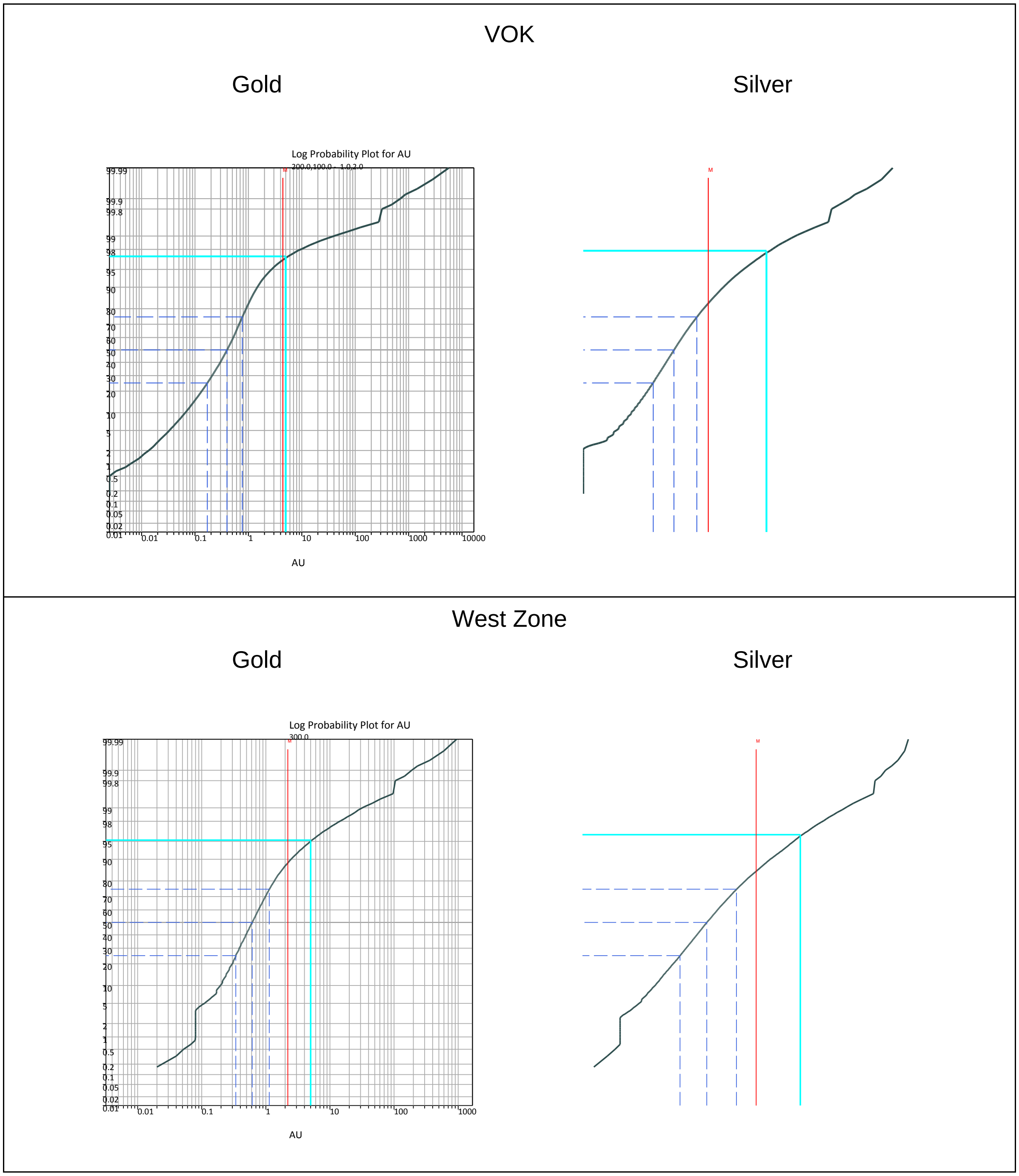
Discussions with Pretivm and analysis of the data indicated the mineralization can be split into a pervasive background mineralization and a separate high grade mineralization style. In order to model this style of mineralization without smearing grade, Snowden separated the lower grade 'background' population from the higher grade population and estimated them independently. For gold in West Zone and the VOK, a threshold of 5 g/t Au was selected to separate the two populations based on review of the population statistics and graphs. The silver data was treated in the same way using a threshold of 50 g/t Ag for the VOK and 300 g/t Ag for the West Zone (Figure 14.11).

The relatively low degree of skewness and the presence of only few samples with extreme grades in the low grade domains allowed for the estimation of grades using ordinary kriging with a top cut. A top cut of 4 g/t Au and 100 g/t Ag was selected for the VOK, based on the point of disintegration seen on the histogram and log probability plot. In the West Zone a top cut of 3 g/t Au and 100 g/t Ag was selected.

In the high grade domains (including the West Zone), the truncated lower grade population is amenable to the ordinary kriging method of grade interpolation.

Multiple indicator kriging was selected for estimation of the higher grade population within the high grade domains, to control the skewness of the data. Multiple indicator kriging involves modelling variograms at a series of grade thresholds which allows the range of continuity to be reduced at the higher grades. A mathematical model was then used to define the top end of the grade distribution. The result of this estimation method is that, while no top cut is used to limit the higher grades, the higher grades are limited in their influence using a mathematical model based on the higher grade data to estimate grades in the top class based on probability rather than using the individual extreme grades in the dataset for grade estimation. Furthermore this technique attempts to model and honour the actual grade distribution of the input data.

Figure 14.11 Log probability plot showing threshold between lower and higher grade populations



14.9 Variogram analysis

High grade domains - low grade population

Due to the positively skewed nature of the grade distributions, normal scores experimental variograms were modelled for gold and silver for the estimates of the lower grade population within the high grade domains.

For the low grade population in the VOK and West Zone, variograms were calculated and modelled using only data below the thresholds of 5 g/t Au and 50 g/t Ag. Due to the locally changing orientations in the VOK area, the orientations of the variogram models were locally adjusted to reflect the interpretation.

The normal scores models were back-transformed prior to estimation.

Table 14.16 Parameters to describe gold grade continuity for the low grade population estimates within the high grade domains

Area	Grade	Orientation	Nugget	Structure 1		Structure 2		Structure 3	
				Sill	Range	Sill	Range	Sill	Range
VOK	Gold	00→110 -90→000 00→020	0.17	0.36	6 6 5	021	20 35 12	0.26	127 169 50
VOK	Silver	00→110 -90→000 00→020	0.14	0.43	12 12 7	0.11	45 65 8	0.32	240 200 90
West Zone	Gold	00→120 -80→030 10→030	0.21	0.63	20 25 7	0.16	194 341 35		
West Zone	Silver	00→120 -80→030 10→030	0.15	0.46	17 21 6	0.39	248 294 59		

High grade domains - high grade population

Indicator variograms for gold were calculated and modelled for the high grade populations within the high grade domains for the VOK and West Zone.

In the VOK and West Zone, given the small proportion of data above the high grade population threshold grade, experimental variograms were poorly structured. As a result, experimental variograms were modelled for the 50^{th} percentile of the distribution and then adapted for the 10, 20, 30, 40, 60, 70, 80, 90, and 95^{th} percentiles of the distribution.

As with the low grade population, due to the locally changing orientations in the VOK area, the orientations of the variogram models were locally adjusted to reflect the interpretation.

It was not possible to model variograms for silver in West Zone. Given the high correlation between gold and silver (approximately 0.9 correlation coefficient) in this zone, the gold variograms were also used for the estimation of the silver high grade population for the West Zone.

The upper tail of the high grade population distributions, above the 95 percentile, was modelled using a hyperbolic or power mathematical model for each area for gold and silver.

Table 14.17 describes the indicator thresholds and associated cut-off grades for the VOK, while

Table 14.18 and Table 14.19 describe the variogram models for each indicator threshold for the VOK and the West Zone respectively.

Table 14.17 Indicator thresholds and associated grade cut-offs for the high grade population estimates within the high grade domains of the VOK

Percentile	Cut-off grade	
	Au (g/t)	Ag (g/t)
10	5.75	55.51
20	6.813	60.79
30	8.191	67.3
40	10.58	76.37
50	14.05	89.81
60	20.12	109.1
70	34.31	137.9
80	72.54	190
90	211.4	317
95	503	549.6

Table 14.18 Parameters to describe gold grade continuity for a range of indicators for the high grade population estimates within the high grade domains of the VOK

Grade	Cut-off (percentile)	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
Gold	10,20,30,40,50,60,70	00→110	0.53	0.23	5	0.24	35
		-90→000			5		35
		00→020			5		20
	80,90	00→110	0.53	0.23	4	0.24	28
		-90→000			4		28
		00→020			4		16
	95	00→110	0.53	0.23	3	0.24	21
		-90→000			3		21
		00→020			3		12
Silver	10,20,30,40,50,60,70	00→110	0.44	0.35	5	0.21	15
		-90→000			5		20
		00→020			5		15
	80,90	00→110	0.44	0.35	4	0.21	12
		-90→000			4		16
		00→020			4		12
	95	00→110	0.44	0.35	3	0.21	9
		-90→000			3		12
		00→020			3		9

Table 14.19 Parameters to describe gold grade continuity for a range of indicators for the high grade population estimates within the high grade domains of the West Zone

Grade	Cut-off (percentile)	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
Gold	10,20,30,40,50,60,70	-90→000 00→240 00→330	0.56	0.06	7 4 4	0.38	17 13 13
	80,90	-90→000 00→240 00→330	0.56	0.06	6 3 3	0.38	14 10 10
	95	-90→000 00→240 00→330	0.56	0.06	4 2 2	0.38	10 8 8

High grade domains - probability

In order to estimate the proportion of the high grade population within each block, an indicator variogram was calculated and modelled for the mineralized domains at the population threshold (5 g/t Au and 50 g/t Ag for the VOK, 5 g/t Au and 300 g/t Ag for the West Zone).

For the West Zone a lower cut-off was used to remove the background 'waste' and improve the quality of the indicator variograms. The lower cut applied was 1 g/t Au and 30 g/t Ag.

As with the low grade and high grade populations, due to the locally changing orientations in the VOK area, the orientations of the variogram models were locally adjusted to reflect the interpretation. Table 14.20 summarises the variogram models used for the probability estimate.

Table 14.20 Parameters to describe gold grade continuity at the low grade / high grade population threshold

Area	Cut-off threshold	Orientation	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
VOK	Gold 5 g/t	00→110 -90→000 00→020	0.50	0.39	4 4 3	0.38	15 15 12
	Silver 50 g/t	00→110 -90→000 00→020	0.38	0.33	4 4 2	0.29	16 25 11
West Zone	Gold 5 g/t	00→110 -90→000 00→020	0.37	0.45	4 4 2	0.18	10 10 4
	Silver 300 g/t	00→110 -90→000 00→020	0.41	0.21	5 2.5 2	0.38	9 4 3

Low grade domains

The low grade domains were estimated using the variograms defined for the low grade populations within the high grade domains.

Specific gravity

For the updated VOK estimate, given the amount of data, a single omnidirectional variogram was calculated and modelled for the total VOK dataset using specific gravity data.

For the West Zone estimate, a single omnidirectional normal scores variogram was calculated and modelled for the total West Zone dataset using specific gravity data. The normal scores models were back-transformed prior to estimation.

Table 14.21 summarizes the variogram model parameters for the specific gravity continuity.

Table 14.21 Parameters to describe specific gravity continuity

Area	Nugget	Structure 1		Structure 2		Structure 3	
		Sill	Range	Sill	Range	Sill	Range
VOK	0.06	0.45	15	0.14	60	0.35	200
West Zone	0.04	0.29	15	0.67	155		

14.10 Establishment of block models

A Datamine block model with cell dimensions of 10 mE by 10 mN by 10 mRL was coded to reflect the surface topography, base of overburden, lithological contacts, and the mineralization domains. This block model was used for estimation of the density, low grade domains and the low grade mineralized population within the high grade domains of the VOK and the majority of the West Zone.

Within the well-informed portion of the West Zone, with close spaced drilling of around 5 m by 5 m to 10 m by 10 m, the parent cell size was reduced to 5 mE by 5 mN by 5 mRL for estimation of the background grades and low grade mineralized population.

Two small scale discretized block models were created for the multiple indicator kriging estimates so that these point estimates could be subsequently reblocked to take into account the correct degree of smoothing for the final block size. The discretized block models have parent cells sizes of 2.5 mE by 2.5 mN by 2.5 mRL for the VOK and the majority of the West Zone and 1.25 mE and 1.25 mN by 1.25 mRL for the well-informed portions of the West Zone.

14.11 Grade interpolation parameters

Estimation and search parameters for the VOK were reviewed and adjusted based on the testwork carried out in the local test area around the underground bulk sample. A summary of the testwork and changes to the parameters is included in Section 0. The following sections describe the updated parameters in more detail for each estimate.

High grade domains - low grade population

The lower grade population within the high grade domains was estimated using ordinary kriging into 10 m by 10 m by 10 m parent blocks for the VOK and most of the West Zone. In the well-informed portion of West Zone a 5 m by 5 m by 5 m parent block used.

For the VOK the mineralized corridors were estimated with hard boundaries for estimation, except where the corridors merge together. A series of ‘search domains’ with locally adjusted orientations were applied for estimation to account for the local variability in the orientation of the corridors. The boundaries between these search domains were treated as soft for estimation.

For the West Zone the high grade domains were treated as a single domain for estimation for each area.

Gold and silver grades were interpolated using 1.5 m composites with all data above the population threshold removed (5 g/t Au and 50 g/t Ag for the VOK, and 5 g/t Au and 300 g/t Ag for West Zone). Estimation parameters were established from the variography analysis. Any target blocks that remained uninformed after the first pass search pass were subsequently estimated in a second search pass using a broader search ellipse and different restrictions.

The VOK interpolation was controlled by:

- Minimum / maximum numbers of composites: 12 / 26 per block (8 / 26 for pass 2).
- Discretization: 4 by 4 by 4.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 60 m by 100 m by 20 m (120 m by 200 m by 40 m for pass 2).

The West Zone interpolation was controlled by:

- Minimum / maximum numbers of composites: 20 / 26 per block (8 / 26 for pass 2).
- Discretization: 4 by 4 by 4.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 200 m by 300 m by 30 m (400 m by 600 m by 60 m for pass 2).

High grade domains - high grade population

The higher grade populations were estimated using multiple indicator kriging to control the skewness of the data. Indicator variograms were modelled up to the 95 percentile of the data with a mathematical model used to define the top end of the grade distribution. The threshold for the 95 percentile of the higher grade population is:

- For the VOK: 540.3 g/t Au and 549.6 g/t Ag.
- For the West Zone: 93 g/t Au and 3,479 g/t Ag.

The higher grade populations were estimated into small scale discretized blocks of 2.5 mE by 2.5 mN by 2.5 mRL for the VOK and the majority of West Zone and 1.25 mE and 1.25 mN by 1.25 mRL for the well-informed portions of West Zone.

For the previous estimates for the VOK and West Zone the estimates were reblocked into parent blocks twice the size of those used for the lower grade population estimates to further limit the influence of the highest grades in the highest grade areas. For the updated 2013 VOK model, the estimates were reblocked into the parent block size used for the lower grade population estimates as a result of testwork discussed in Section 0.

For the VOK, the mineralized corridors were estimated with hard boundaries for estimation, except where the corridors merge together. As with the low grade populations, a series of ‘search domains’ with locally adjusted orientations were applied for estimation to account for the local variability in the orientation of the corridors. The boundaries between these search domains were treated as soft for estimation.

For the West Zone, the high grade domains were treated as a single domain for estimation for each area.

Gold and silver grades were interpolated using 1.5 m composites with all data below the population threshold removed (5 g/t Au and 50 g/t Ag for the VOK, and 5 g/t Au and 300 g/t Ag for West Zone).

Estimation parameters were established from the variography analysis. Any target blocks that remained uninformed after the first pass search pass were subsequently estimated in a second search pass using a broader search ellipse and different restrictions. The maximum number of samples was kept small in the second search pass to prevent single extreme grades influencing the block estimates at a great distance.

The VOK interpolation was controlled by:

- Minimum / maximum numbers of composites: 12 / 16 per block (2 / 6 for pass 2).
- Discretization: 1 by 1 by 1 (indicator kriging).
- Maximum number of composites per drillhole: 8
- Search ellipse: 35 m by 35 m by 20 m (105 m by 105 m by 60 m for pass 2)

The West Zone interpolation was controlled by:

- Minimum / maximum numbers of composites: 8 / 20 per block (2 / 8 for pass 2).
- Discretization: 1 by 1 by 1 (indicator kriging).
- Maximum number of composites per drillhole: 10
- Search ellipse: 50 m by 50 m by 50 m (150 m by 150 m by 150 m for pass 2)

High grade domains - probability

The proportion of the higher grade mineralization was estimated into each block and used to combine the two estimates in the determination of the overall block grade. For example, if a block had a probability of 5% high grade then the final block grade would combine 95% of the low grade estimate with 5% of the high grade estimate. The influence of the high grade population above the 95 percentile is therefore greatly restricted.

An indicator estimate was run at the population threshold for gold and silver (5 g/t Au and 50 g/t Ag for the VOK, and 5 g/t Au and 300 g/t Ag for West Zone), using all of the data within the mineralized high grade domains.

Estimation of the VOK and West Zones was into the small scale discretized blocks used for the high grade population estimates. The resultant probabilities were reblocked into parent blocks the same size of those used for the lower grade population estimates.

The same domains and boundary conditions applied to the high grade and low grade populations were used for the probability estimate. Probability for gold and silver were interpolated using 1.5 m composites.

Estimation parameters were established from the variography analysis. Any target blocks that remained uninformed after the first pass search were subsequently estimated in a second search pass using a broader search ellipse and different restrictions. The maximum number of samples was kept small in the second search pass to prevent single extreme grades influencing the block estimates at a great distance.

The VOK interpolation was controlled by:

- Minimum / maximum numbers of composites: 12 / 16 per block (2 / 6 for pass 2).

- Discretization: 1 by 1 by 1 into small scale discretized blocks.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 35 m by 35 m by 15 m (70 m by 70 m by 30 m for pass 2).

The West Zone interpolation was controlled by:

- Minimum / maximum numbers of composites: 5 / 50 per block.
- Discretization: 1 by 1 by 1 into small scale discretized blocks.
- Maximum number of composites per drillhole: 10.
- Search ellipse: 75 m by 75 m by 30 m.

For the VOK, around 3% of the data is above 5 g/t Au and the average proportion of high grades within the blocks was estimated at 3% within the Measured and Indicated portions of the estimate. For the West Zone, around 5% of the data is above 5 g/t Au and the average proportion of high grades within the blocks was estimated at 5% within the Measured and Indicated portions of the estimate.

Low grade domains

The background low grade domain in the VOK was estimated with ordinary kriging using dynamic anisotropies to locally adjust the search and variogram orientations to reflect the main trends of the folding in this area. The variogram parameters used were the same as for the low grade population within the high grade domains.

The background low grade domain in West Zone was estimated with ordinary kriging into a single domain using the same variogram parameters as for the low grade population within the high grade domains.

The background low grade domains were estimated using 1.5 m top cut composites with soft boundaries between the low grade and high grade domains, but excluding the high grade population from within the high grade domains.

The VOK interpolation was controlled by:

- Minimum / maximum numbers of composites: 12 / 26 per block (8 / 26 for pass 2).
- Discretization: 5 by 5 by 5.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 200 m by 300 m by 30 m (400 m by 600 m by 60 m for pass 2).

The West Zone interpolation was controlled by:

- Minimum / maximum numbers of composites: 20 / 26 per block (8 / 26 for pass 2).
- Discretization: 4 by 4 by 4.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 100 m by 100 m by 100 m (200 m by 200 m by 200 m for pass 2).

14.12 Density estimation and assignment

The database used to estimate density is based on pulp specific gravity measurements. As part of the 2012 and 2013 drilling programs, Pretivm's QP selected a portion of the samples (207 samples) to undergo core density measurements as well as pulp specific gravity measurements to determine whether there is any impact on the density as a result of porosity. The results of the comparison indicate that the core density is similar to the pulp specific gravity within the siliceous zone and 3% lower on average in all other rock types (Section 11.2.2).

Tonnages were estimated on a dry basis for the VOK. Specific gravity values were estimated into the models using simple kriging where sufficient data was available. Specific gravity was estimated into 10 mE by 10 mN by 10 mRL parent blocks using 1.5 m composites with parameters established from the variography analysis.

The VOK interpolation was controlled by:

- Minimum / maximum numbers of composites: 12 / 26 per block (8 / 26 for pass 2).
- Discretization: 4 by 4 by 4.
- Maximum number of composites per drillhole: 8.
- Search ellipse: 200 m by 200 m by 200 m (400 m by 400 m by 400 m for pass 2).

Outside of these areas, the average specific gravity of 2.80 t/m^3 was applied. Block density values were then determined by factoring down the specific gravity estimates by 3% in all rock types other than in the siliceous zone. Snowden notes that there is little variation in density between the different rock types (Table 14.22).

A similar approach was taken in the determination of densities and tonnage for the West Zone. Specific gravity values were estimated into the models using simple kriging where sufficient data was available. Specific gravity was estimated into 10 mE by 10 mN by 10 mRL parent blocks using 1.5 m composites with parameters established from the variography analysis.

The West Zone interpolation was controlled by:

- Minimum / maximum numbers of composites: 8 / 30 per block (2 / 8 for pass 2).
- Discretization: 4 by 4 by 4.
- Maximum number of composites per drillhole: 9.
- Search ellipse: 300 m by 300 m by 300 m (900 m by 900 m by 900 m for pass 2).

Outside of these areas, the average density of 2.78 t/m^3 was applied. Based on visual observations at the time of estimation, Snowden notes that the West Zone core is very competent and as such there is not expected to be a material difference between the pulp specific gravity measurements and bulk density. Snowden therefore elected not to apply a factor to the specific gravity data for the West Zone. There is little variation in density between the different rock types in the West Zone (Table 14.22).

14.13 Prior mining

A 3D wireframe model of the underground development and stopes in the VOK and West zones was represented by wireframes and coded in the block model to ensure that the reported Mineral Resource estimates are depleted for prior mining. All underground surveying was completed by Procon who are the underground mining contractors. Check surveying of the underground workings was completed by Maptek using their I-site survey tool.

Table 14.22 Average bulk density values for each rock type and area

Description	Average bulk density
Bridge Zone Porphyry	2.68
Office Porphyry	2.69
VSF	2.71
Polylithic Conglomerate	2.74
Siliceous Zones	2.81
Latite Fragmental	2.74
West Zone	2.78
Non-mineralised	2.77

14.14 Model validation

In addition to conducting validation checks on all stages of the modelling and estimation process, final grade estimates and models were validated by: undertaking global grade comparisons with the input drillhole composites; visual validation of block model cross sections; and by grade trend plots.

14.14.1 Global comparisons

The final grade estimates were validated statistically against the declustered input drillhole composites. Table 14.23 and Table 14.9 provide a comparison of the estimated grades compared to the declustered input grades for the global estimates within the mineralized domains. This statistical comparison shows that the domains validate reasonably well globally.

Table 14.23 Comparison of the mean composite grade with the mean block model grade for the mineralized domains in the West Zone.

	Mineralized domain	
	Gold (g/t)	Silver (g/t)
Number of samples	33,089	33,089
Composite mean	1.40	30.66
Estimated mean	1.32	28.54

Table 14.24 Comparison of the mean composite grade with the mean block model grade for the mineralized domains in the VOK

	Mineralized domain	
	Gold (g/t)	Silver (g/t)
Number of samples	79,699	79,699
Composite mean	2.57	8.59
Estimated mean	2.36	8.23

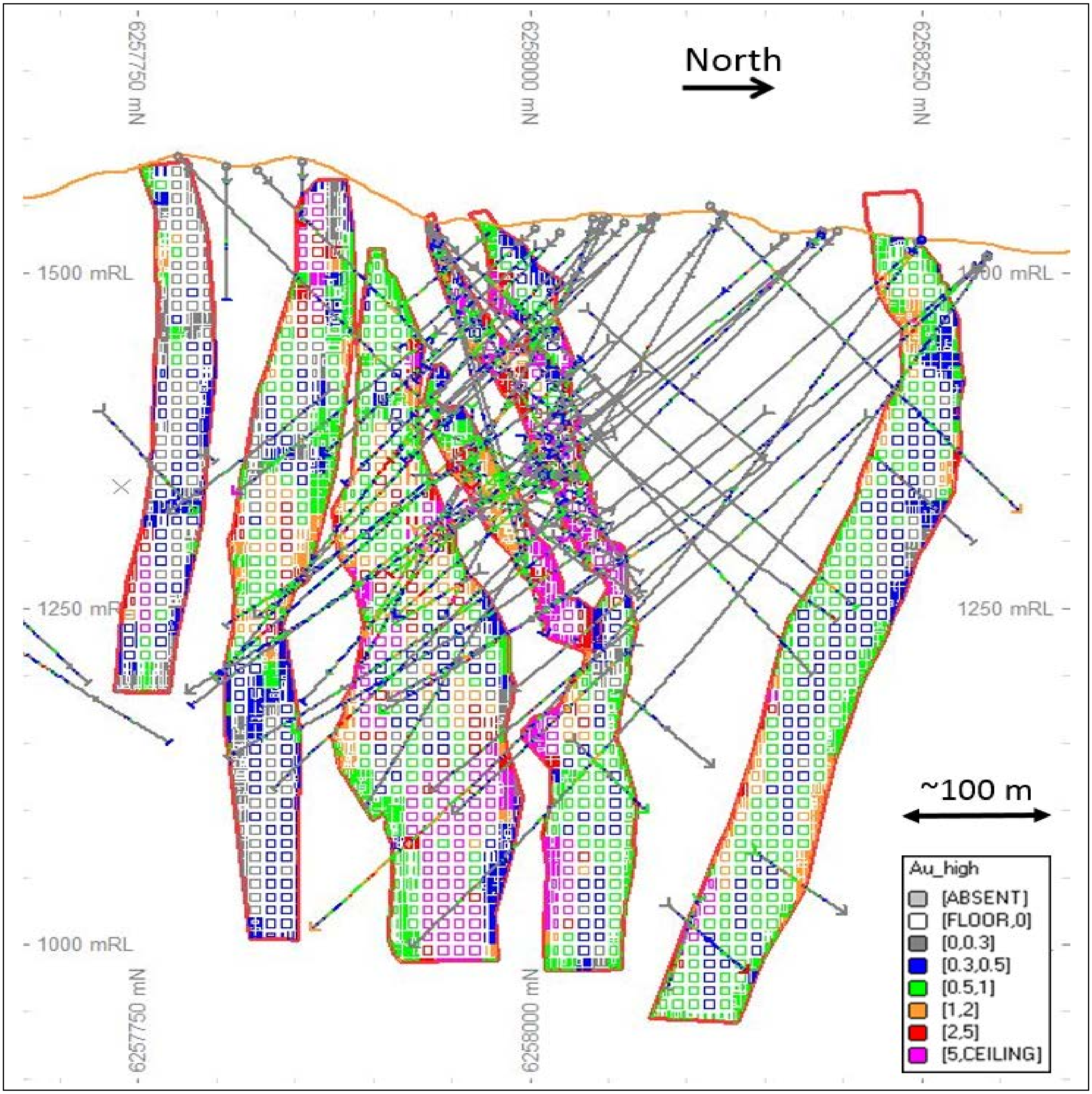
14.14.2 Visual validation

The gold and silver estimates show a good visual correspondence with the input composite grades. Example sections through the higher grade portions of the main mineralized areas are illustrated in Figure 14.12 and Figure 14.13 for the VOK and West Zone respectively.

Sections and plans showing some of the high grade blocks are presented from Figure 14.14 to Figure 14.19.

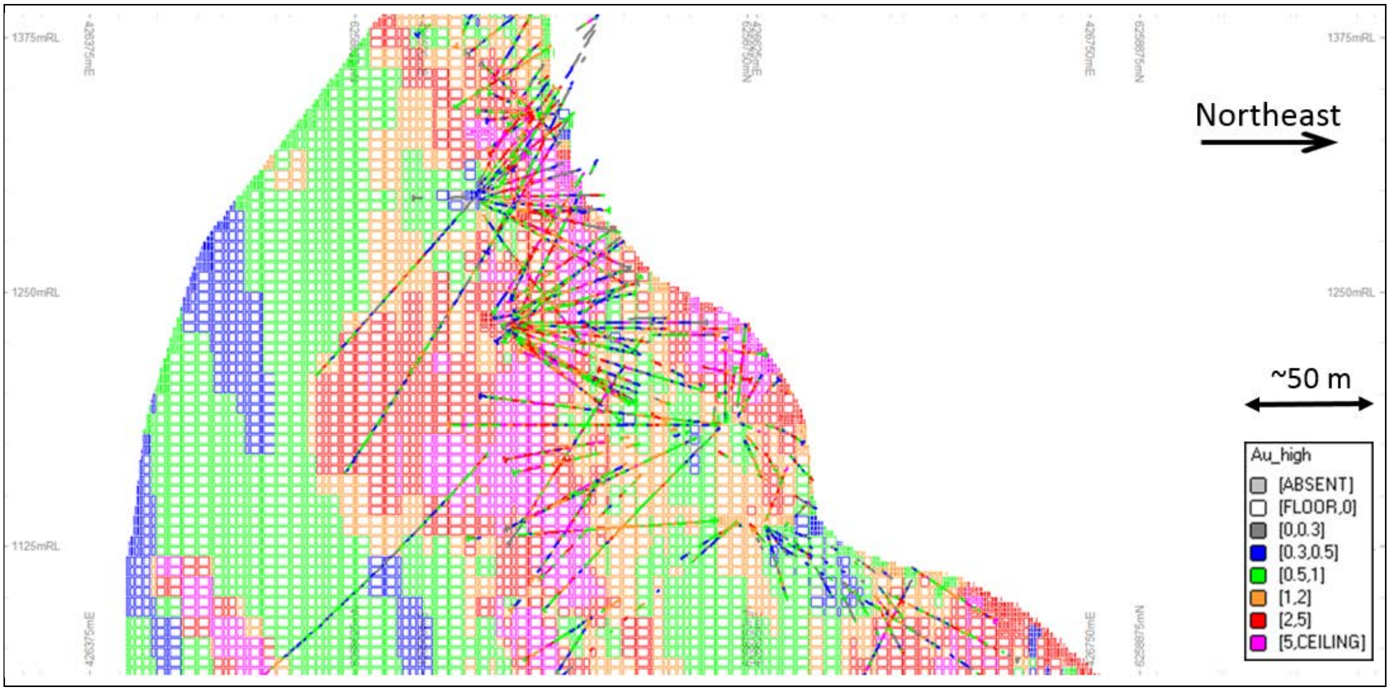
There were some small areas in the relatively poorly informed by drilling parts of the deposit, where grade appears visually to have been spread too far. In these areas the classification was set to Inferred to reflect the lower confidence in these grade estimates.

Figure 14.12 Example cross section showing estimated gold grades compared to input composites within the mineralized domains for the VOK



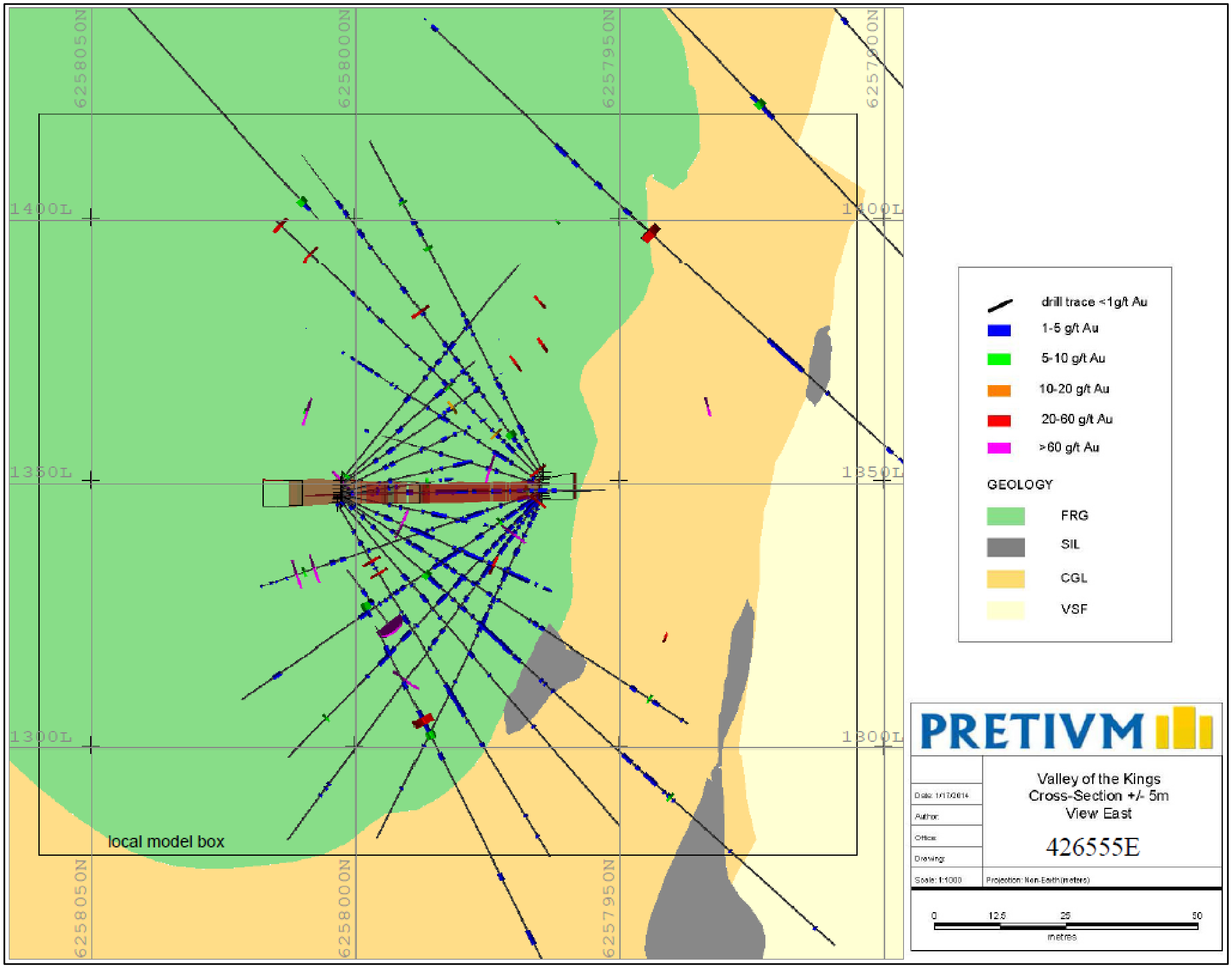
Note: Drillholes are shown with +/- 20 m clipping.

Figure 14.13 Example oblique section showing estimated gold grades compared to input composites within the mineralized domains for West Zone



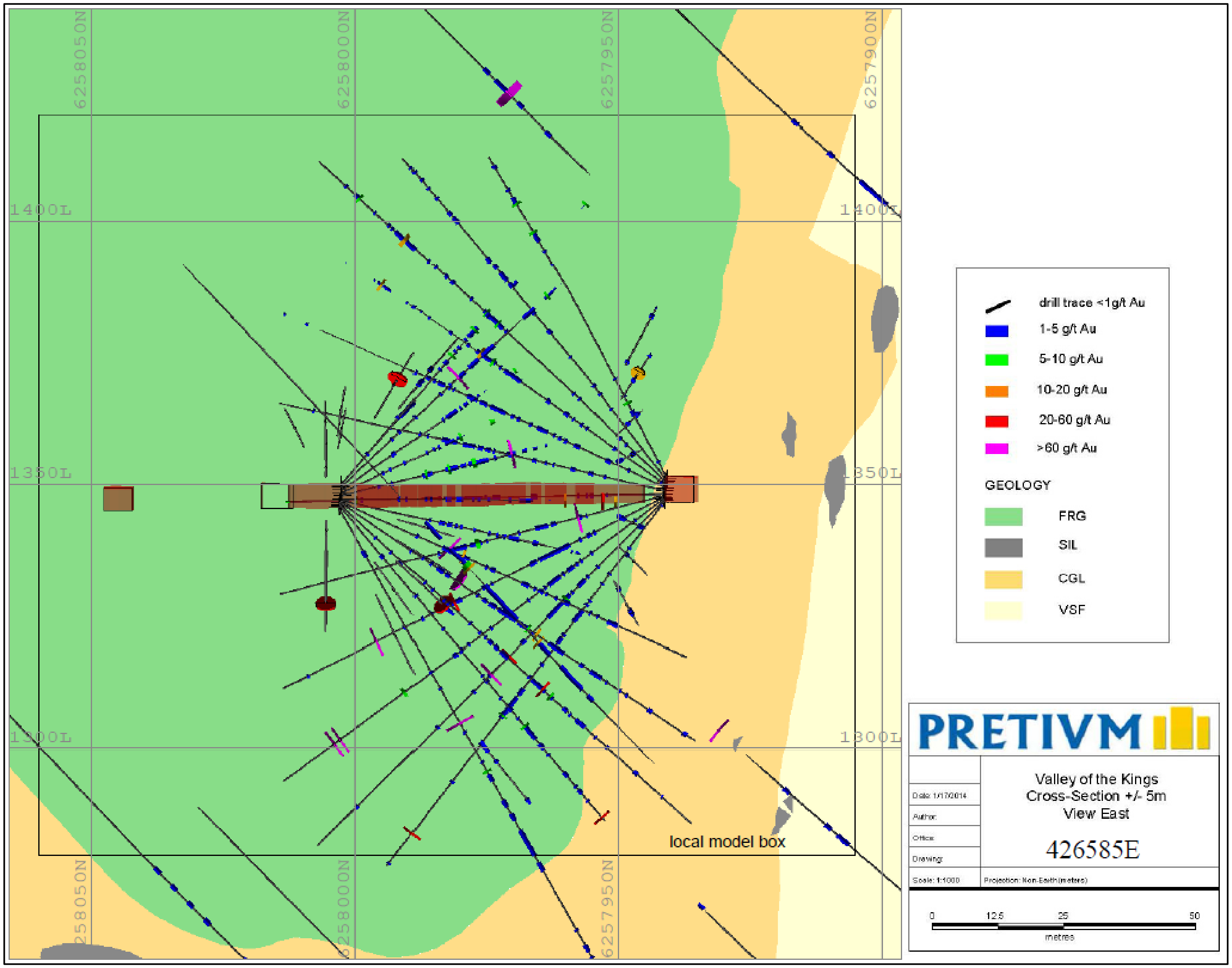
Note: Drillholes are shown with +/- 10 m clipping.

Figure 14.14 Cross-section along the 426555 mE crosscut showing geology and drilling



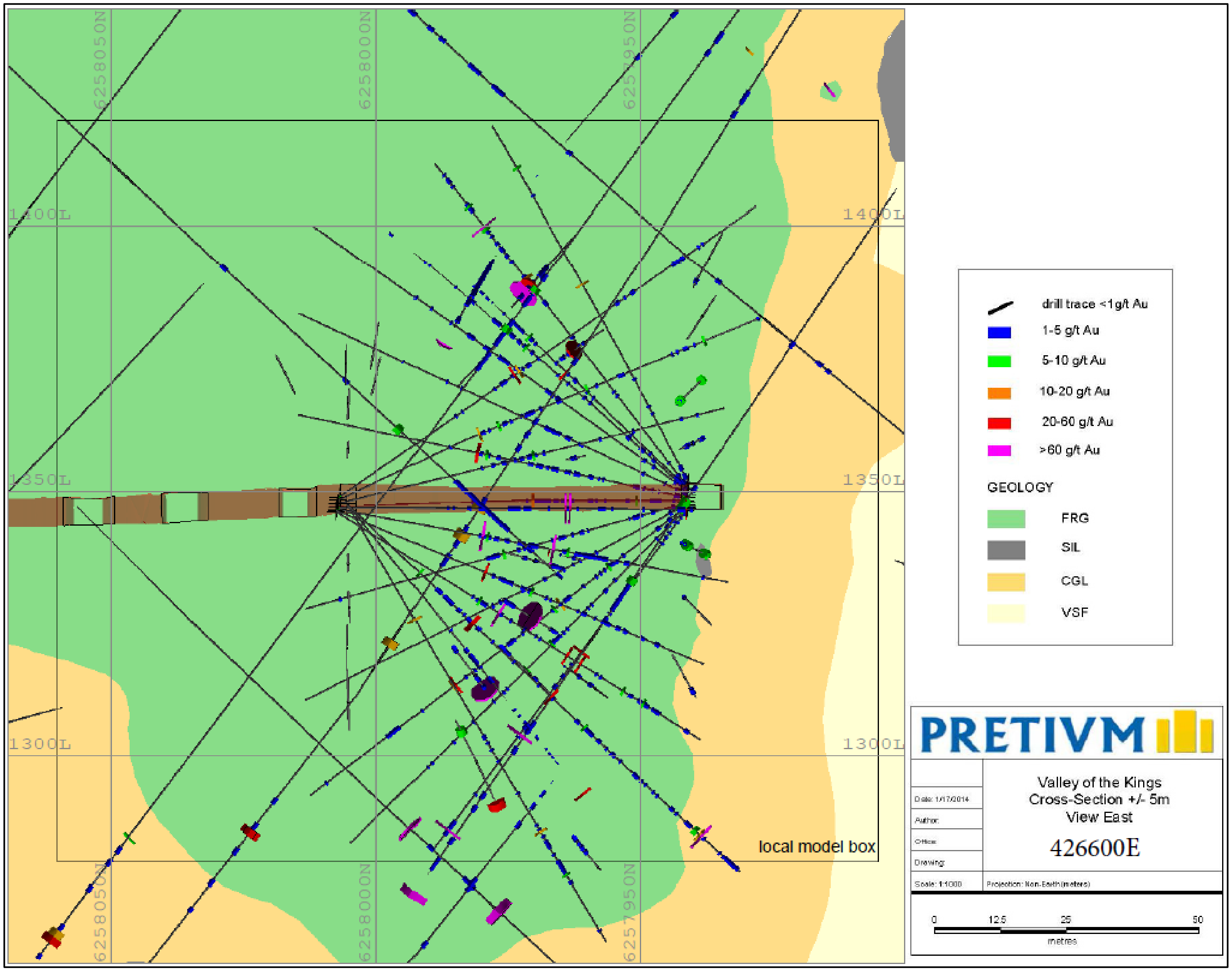
(Source: Pretivm)

Figure 14.15 Cross-section along the 426585 mE crosscut showing geology and drilling



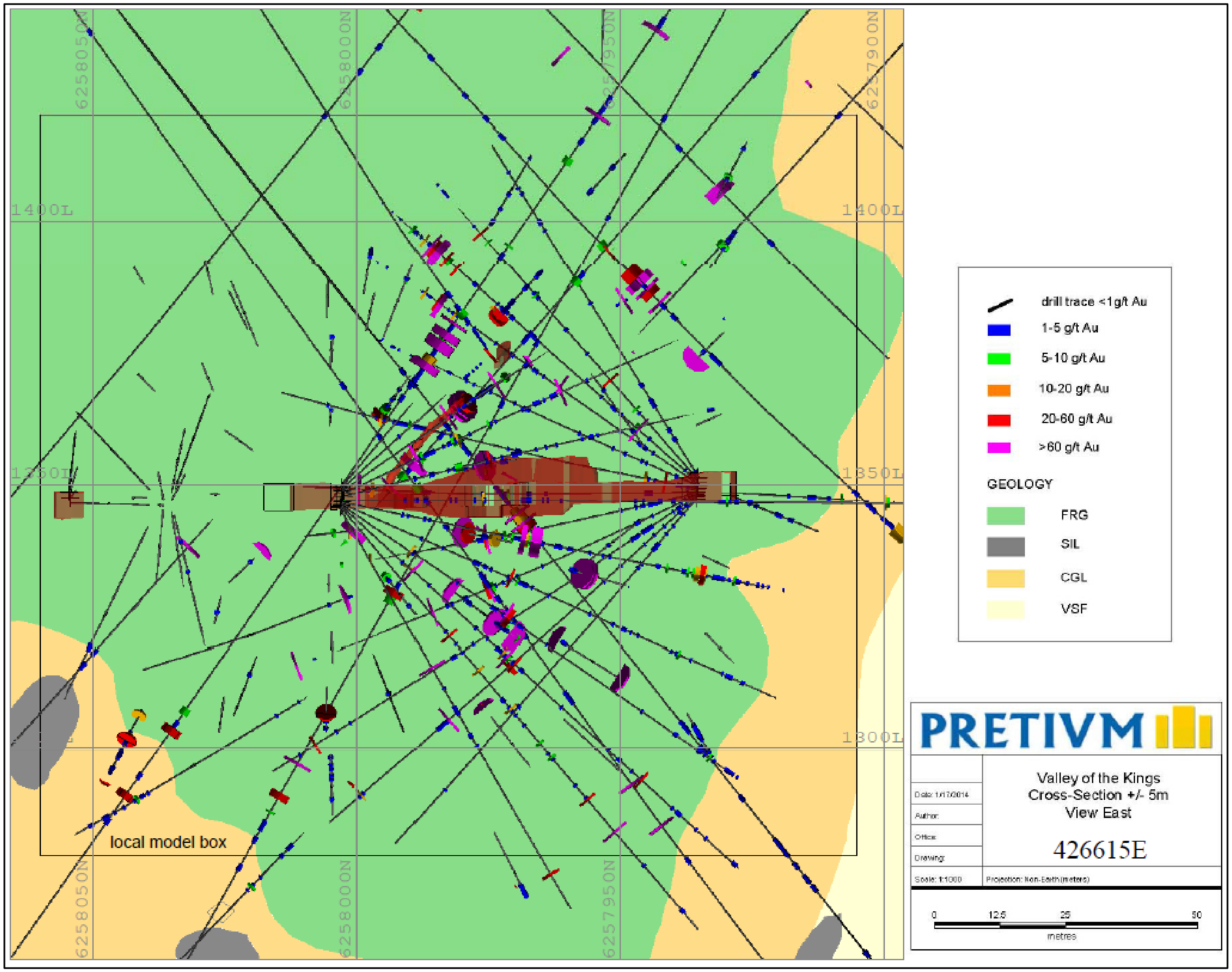
(Source: Pretivm)

Figure 14.16 Cross-section along the 426600 mE crosscut showing geology and drilling



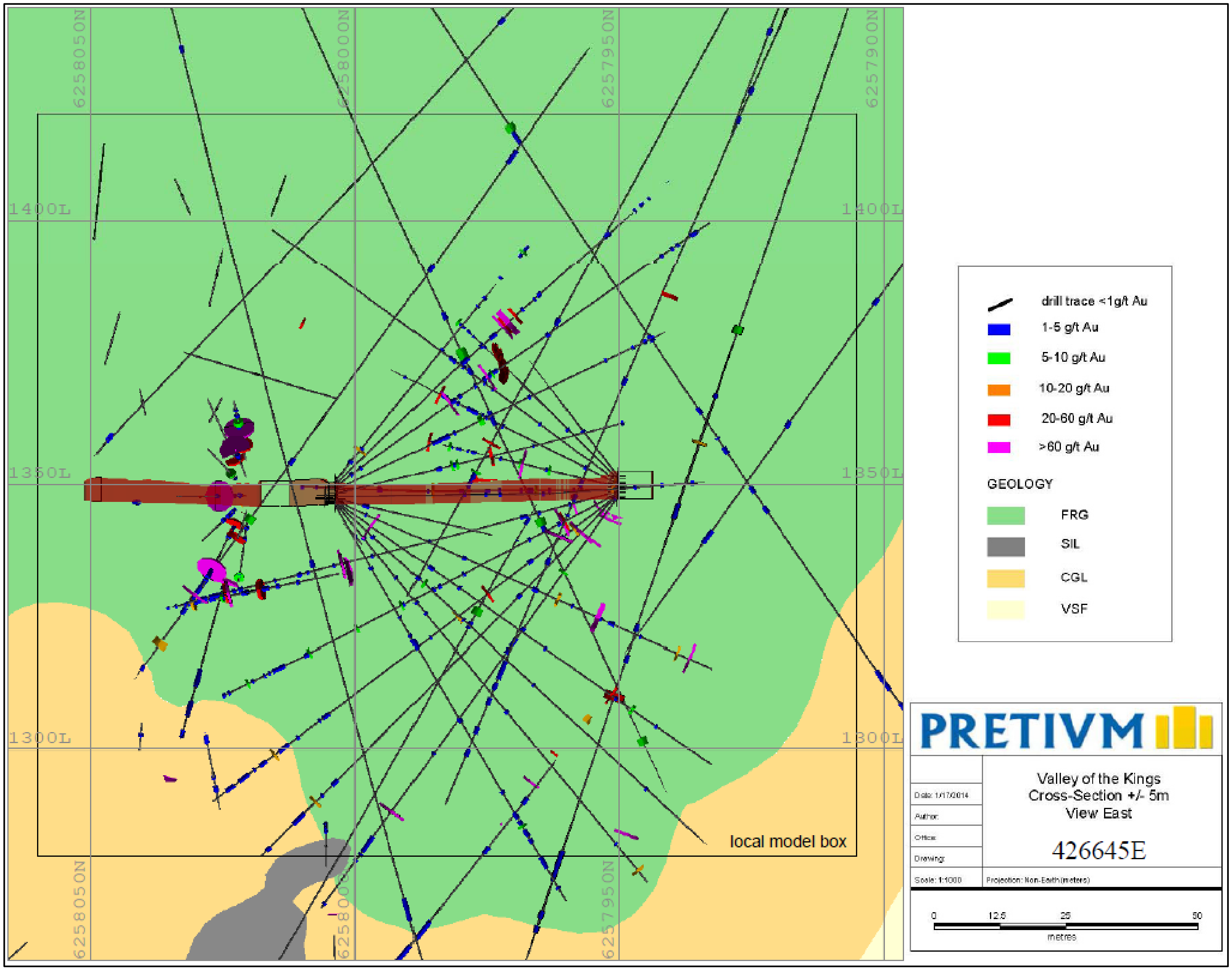
(Source: Pretivm)

Figure 14.17 Cross-section along the 426615 mE crosscut showing geology and drilling



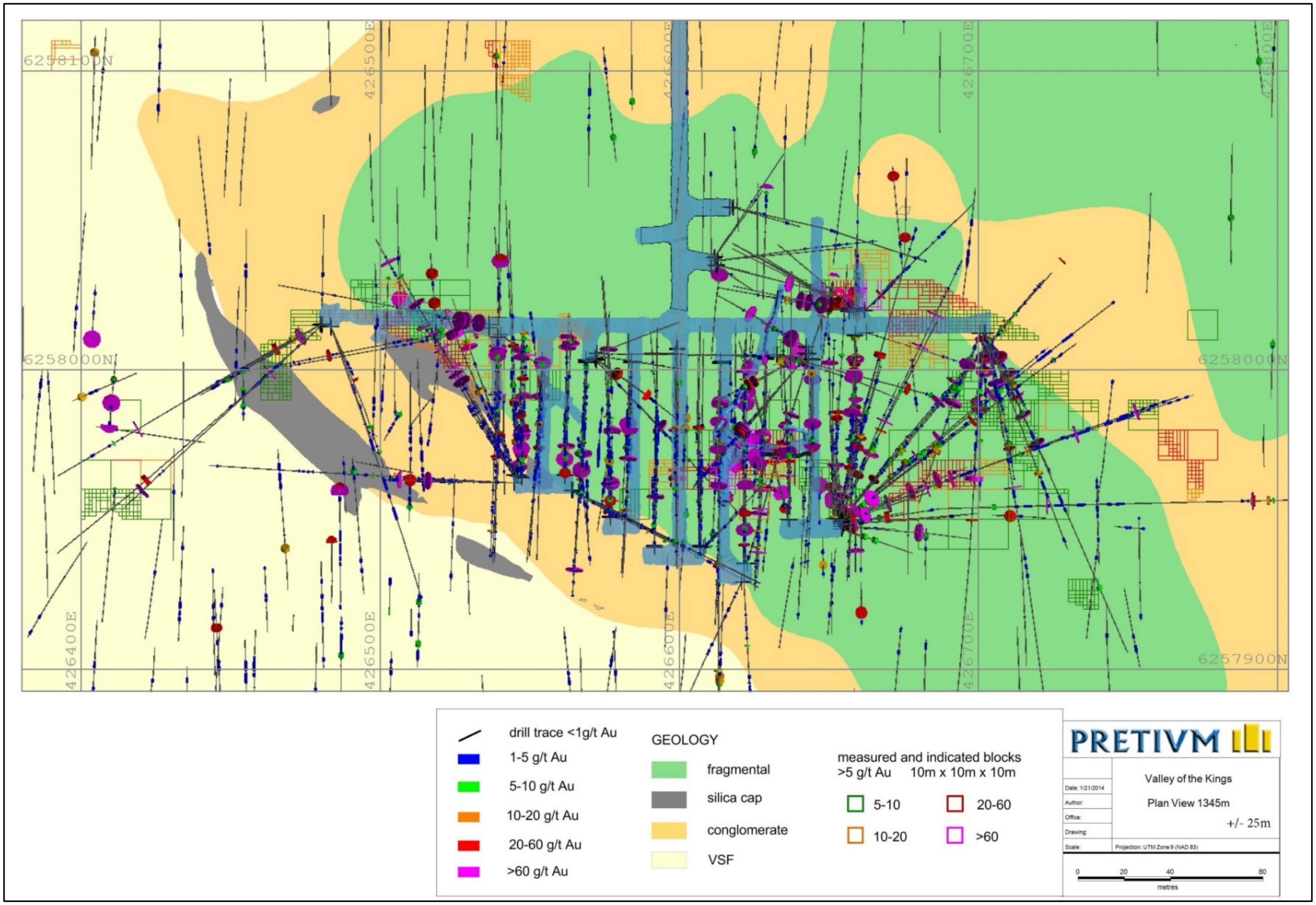
(Source: Pretivm)

Figure 14.18 Cross-section along the 426645 mE crosscut showing geology and drilling



(Source: Pretivm)

Figure 14.19 Plan view (1345 m level) of bulk sample area showing geology and drilling



(Source: Pretivm)

14.14.3 Grade trend plots

Sectional validation graphs were created to assess the reproduction of local means and to validate the grade trends in the model. These graphs compare the mean of the estimated grades to the mean of the input grades (declustered for West Zone, naïve and declustered for the VOK) within model slices (bins). The graphs also show the number of input samples on the right axis, to give an indication of the support for each bin.

Validation graphs were created for the low grade domains and high grade domains including the low grade population estimates, the high grade population estimates, the probability estimates and the final combined estimates for each area (Olssen and Jones, 2013). Within the VOK validation graphs were created for each mineralized corridor.

The graphs indicate that there is good local reproduction of the input grades in both the horizontal and vertical directions (Olssen and Jones, 2013). The high grade population estimate is quite smooth compared to the input data as expected. This smoothing was incorporated into the high grade population estimate to prevent the over influence of the individual high grade samples when the high grade and low grade estimates are recombined.

14.15 Resource classification

The resource classification definitions used for this estimate are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document “CIM Definition Standards”.

Measured Mineral Resource: *that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.*

Indicated Mineral Resource: *that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

Inferred Mineral Resource: *that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.*

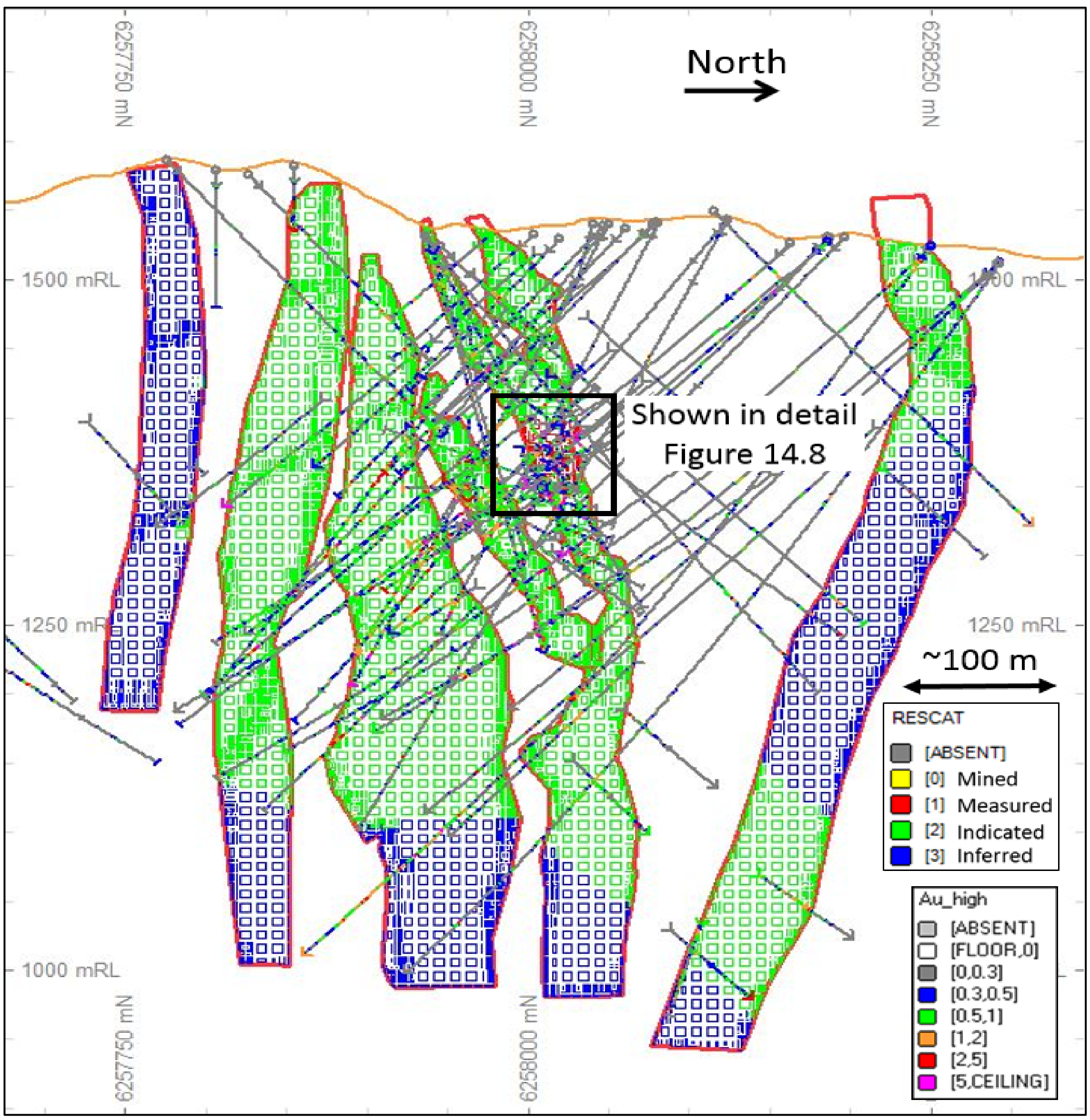
In order to identify those blocks in the block model that could reasonably be considered as a Mineral Resource, the block model was filtered by a cut-off grade of 5 g/t AuEq. The blocks occurring above 5 g/t AuEq were used as a guide to develop a set of wireframes defining coherent zones of mineralization which were classified as Measured, Indicated or Inferred. Classification was applied based on geological confidence, data quality and grade variability.

Areas classified as Measured Resources in the West Zone are within the well-informed portion where the resource is informed by 5 m by 5 m or 5 m by 10 m spaced drilling. Measured Resources within the VOK are informed by 5 m by 10 m to 10 m by 10 m underground fan drilling in the vicinity of the underground bulk sample program.

Areas classified as Indicated Resources are informed by 20 m by 20 m to 20 m by 40 m drilling within the West Zone and the VOK. The remainder of the Mineral Resource is classified as Inferred Resources where there is some drilling information and the blocks lie within the mineralized interpretation.

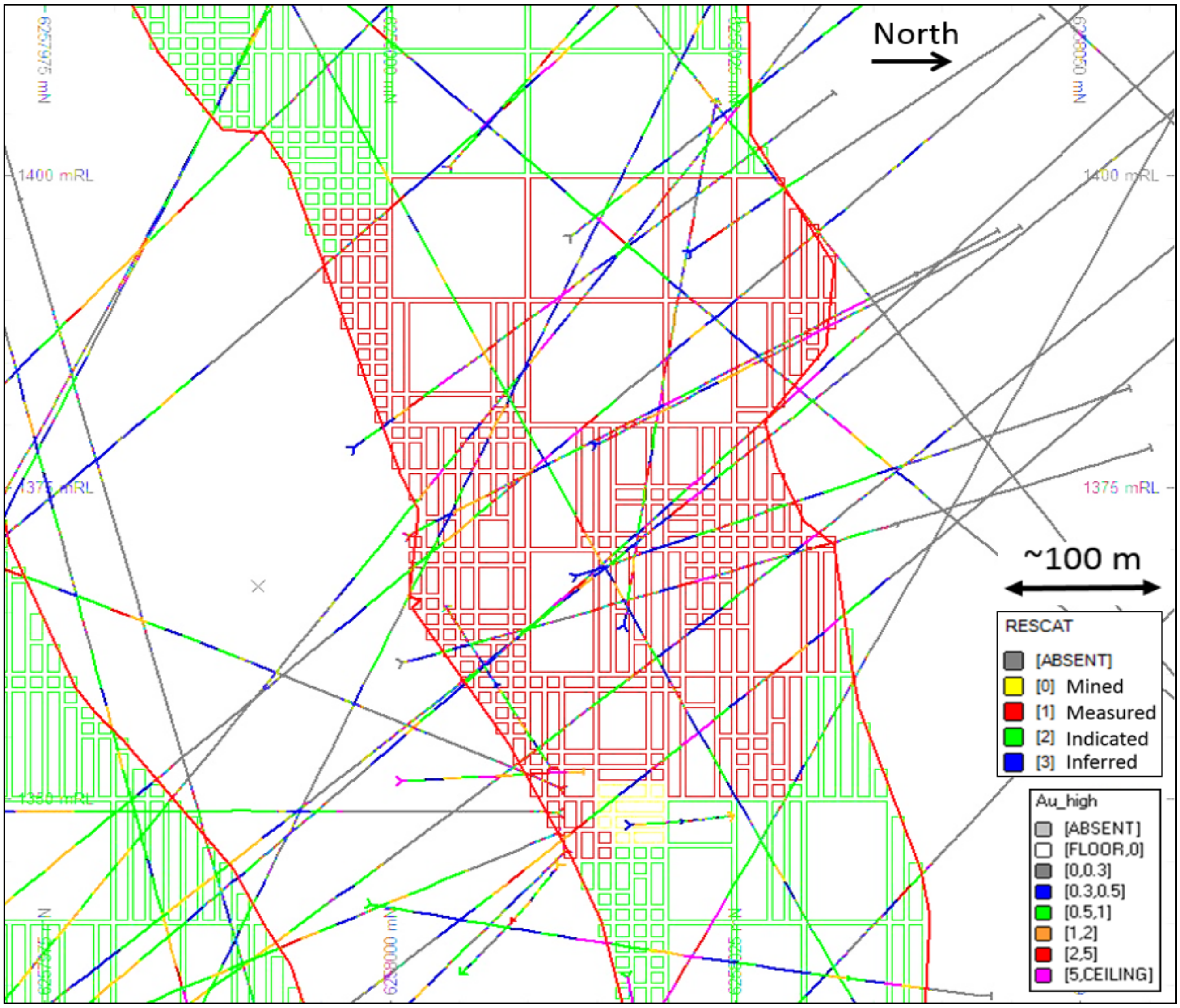
Figure 14.20 to Figure 14.22 illustrate example sections through the main areas of mineralization, coloured by resource classification for the VOK and the West Zone. Figure 14.23 shows an isometric view of the VOK with all estimated blocks above 5 g/t Au.

Figure 14.20 Example cross section showing classification of the Mineral Rresource estimate for the VOK (blocks coloured by classification category) with drilling coloured by gold grade



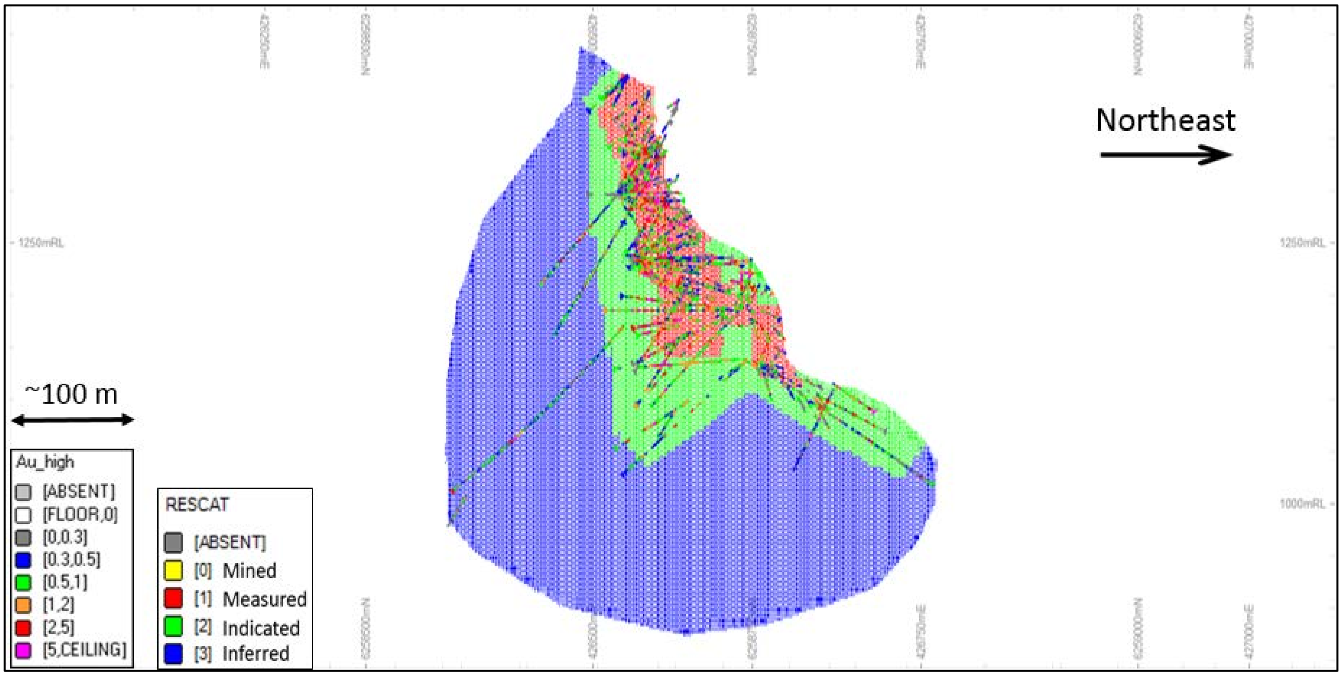
Note: Drillholes are shown with +/- 20 m clipping.

Figure 14.21 Example cross section showing classification of the Mineral Rresource estimate for the VOK (blocks coloured by classification category) with drilling coloured by gold grade in Measured area



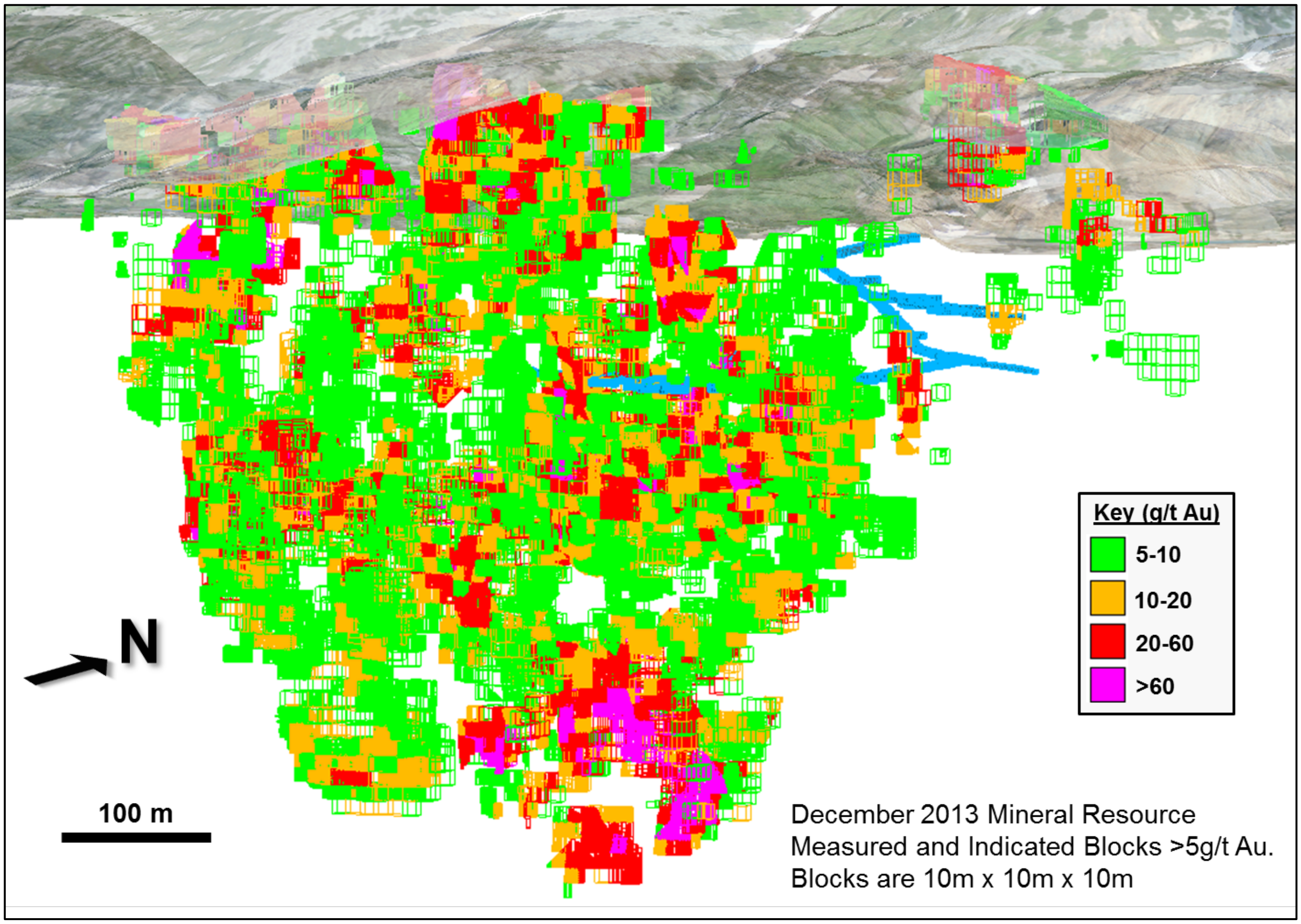
Note: Drillholes are shown with +/- 20 m clipping.

Figure 14.22 Example oblique section showing classification of the Mineral Resource estimate for the West Zone (blocks coloured by classification category) with drilling coloured by gold grade



Note: Drillholes are shown with +/- 20 m clipping.

Figure 14.23 Isometric view of the VOK showing Measured and Indicated blocks above 5 g/t Au



14.16 Resource reporting

The Mineral Resources are reported above a cut-off grade of 5 g/t gold equivalent (AuEq) which reflects the potential economics of a high grade underground mining scenario. The AuEq value for each block is consistent with the November 2012 Mineral Resource. In that evaluation, the AuEq value was calculated according to the formula (AuEq = Au + Ag/53) based upon prices of $US1,590/oz and $US30/oz for gold and silver respectively. Recoveries for gold and silver are assumed to be similar.

High grade Mineral Resources for the VOK and the West Zone are summarized in Table 14.25 and Table 14.26 respectively. Figure 14.24 and Figure 14.25 show grade tonnage curves for the Measured plus Indicated Mineral Resource and the total Mineral Resource respectively.

Table 14.25 VOK Mineral Resource estimate based on a cut-off grade of 5 g/t AuEq – December 2013(1)(4)(5)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.0	19.3	14.4	1.2	0.9
Indicated	13.4	17.4	14.3	7.5	6.1
M + I	15.3	17.6	14.3	8.7	7.0
Inferred(2)	5.9	25.6	20.6	4.9	3.9

(1) Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues. The Mineral Resources in this Technical Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

(3) Contained metal and tonnes figures in totals may differ due to rounding.

(4) The Mineral Resource estimate stated in Table 14.25 and Table 14.26 is defined using 5 m by 5 by 5 m blocks in the well drilled portion of West Zone (5 m by 10 m drilling or better) and 10 m by 10 m by 10 m blocks in the remainder of West Zone and in VOK.

(5) The gold equivalent value is defined as AuEq = Au + Ag/53.

Table 14.26 West Zone Mineral Resource estimate based on a cut-off grade of 5 g/t AuEq – April 2012(1)(4)(5)

Category	Tonnes (millions)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold (Moz)	Silver (Moz)
Measured	2.4	5.85	347	0.5	26.8
Indicated	2.5	5.86	190	0.5	15.1
M+I	4.9	5.85	267	0.9	41.9
Inferred(2)	4.0	6.44	82	0.8	10.6

(1), (2), (3), (4) and (5) - See footnotes to Table 14.25

Figure 14.24 Grade tonnage curve showing combined Measured plus Indicated Mineral Resources for the VOK

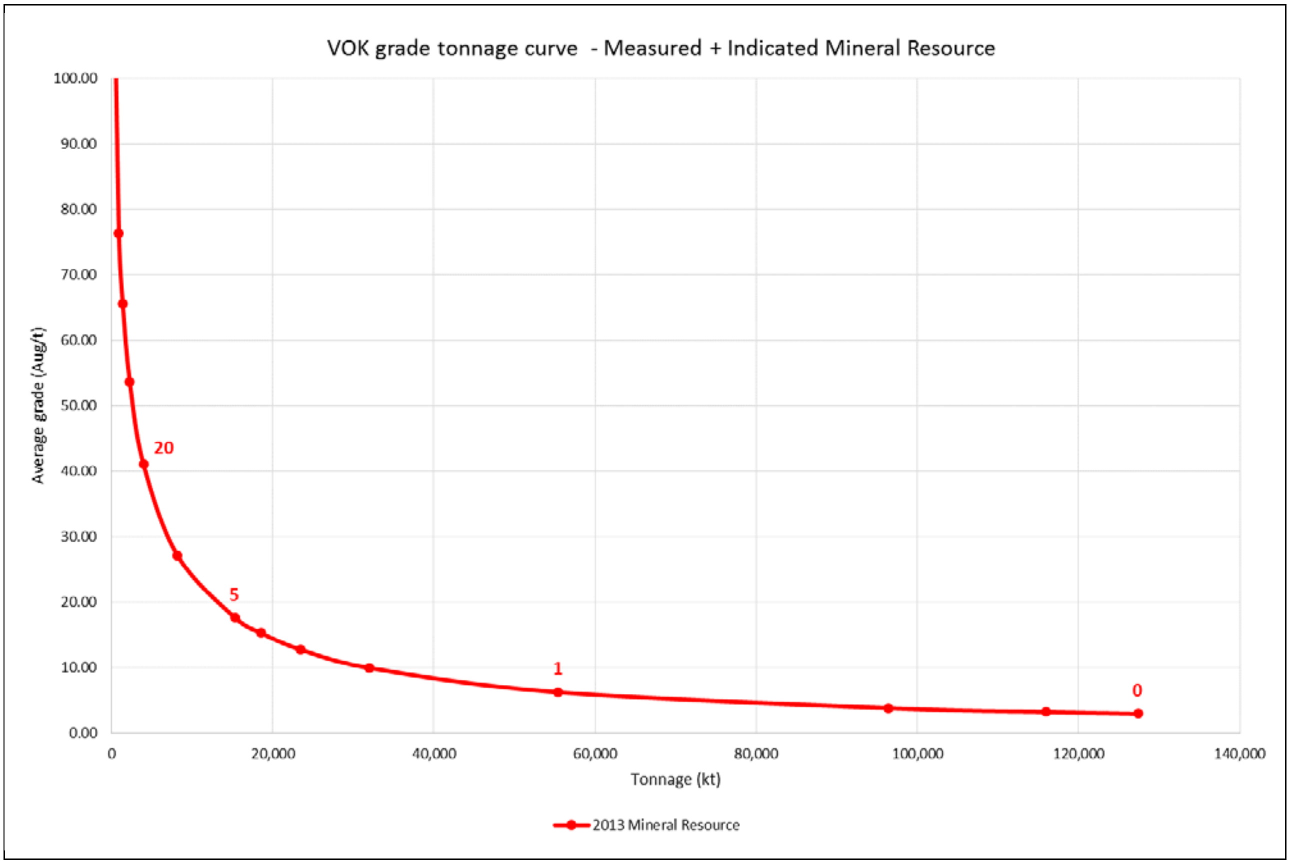
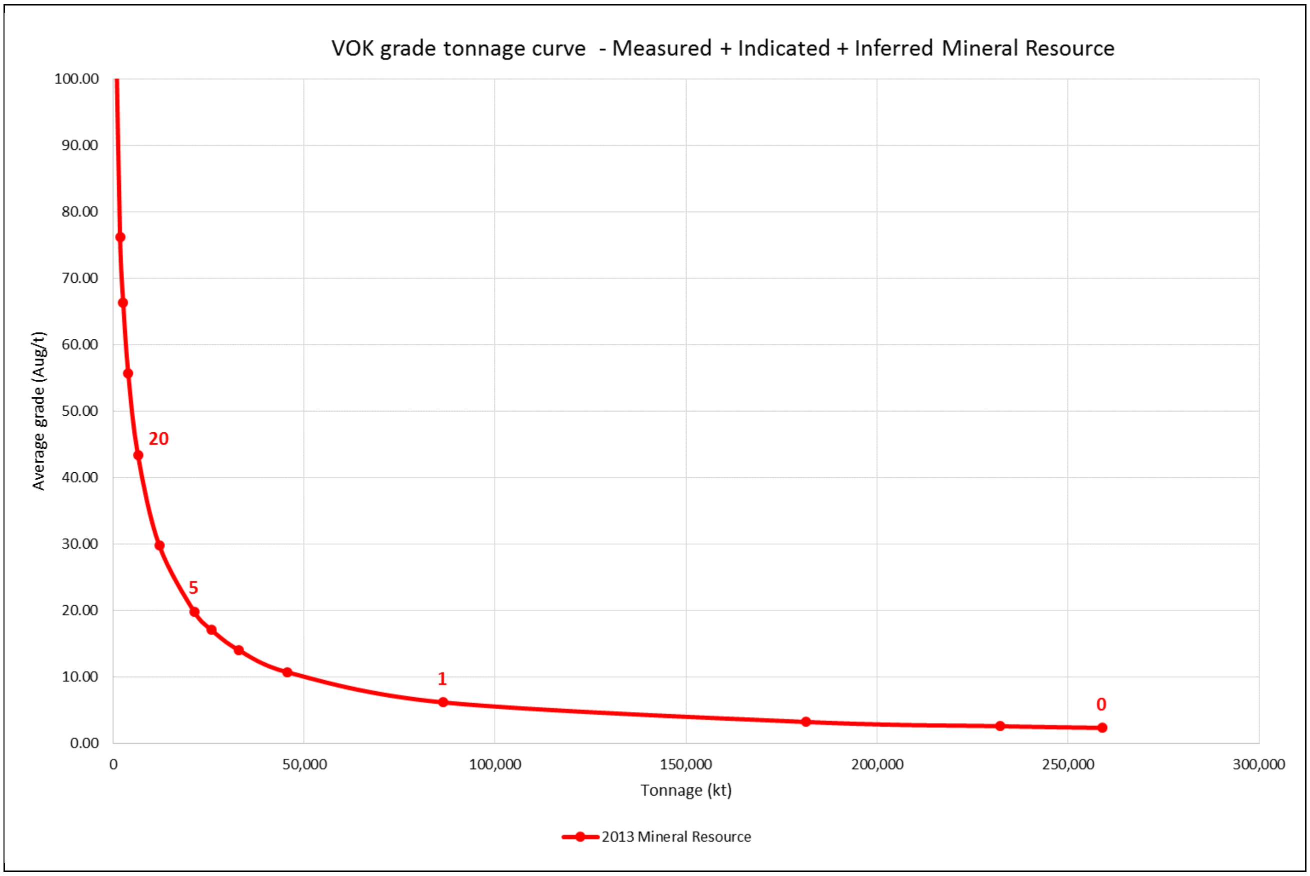


Figure 14.25 Grade tonnage curve showing combined Measured plus Indicated plus Inferred Mineral Resources for the VOK



14.17 Comparison with previous Mineral Resource estimate

In the November 2012 study, Jones (Jones, 2012c) reported Mineral Resource estimates for the high grade portion of the West Zone and the VOK assuming underground extraction of the mineralization.

The December 2013 estimate has been updated based on over 40,000 m of additional drilling including 24 surface drillholes (5,200 m) and 409 underground drillholes (38,840 m) drilled in support of the underground bulk sample. In addition to the drilling, a 10,000 tonne bulk sample has been processed through a mill and detailed testwork has been carried out to both validate the previous Mineral Resource and refine the estimation process for the updated Mineral Resource.

The result of the testwork is an improved confidence in both the geological model and the grade estimate, with the definition of Measured Resources as part of the December 2013 Mineral Resource.

There were minor changes to the mineralized interpretation and estimation parameters as a result of the additional information, but overall, the previous interpretation has been shown to be robust. Details of these changes and the testwork are provided in Section 14.5, together with a detailed comparison between the 2012 and 2013 Mineral Resources within the local test area surrounding the bulk sample crosscuts.

Whilst the November 2012 Mineral Resource estimate provided a reasonable prediction of the total contained metal ounces in the overall bulk sample area, improvements in the understanding of the mineralization as a result of the bulk sample exercise have demonstrated improvements were possible to reduce smoothing and improve local accuracy of the estimates. The December 2013 Mineral Resource estimate provides a reasonable prediction of the local scale (drifts and crosscuts) for the dominant east-west component of the stockwork mineralization, but underestimates the total contained metal ounces in the overall bulk sample area. The underestimation is considered to be a function of an orientation bias in the drilling, whereby the north-south drilling has not representatively sampled the high grade north-south trending gold mineralization in the 426615E crosscut. This has prevented the generation of a locally accurate estimate in this part of the bulk sample area. Snowden and Pretivm consider the improved local accuracy of the December 2013 Mineral Resource estimate with regards to the dominant east-west mineralization as taking precedence over the local underestimation of the rarer north-south component to the mineralization. As a result, the December 2013 Mineral Resource estimate is considered to be a conservative estimate of the contained metal in the VOK deposit. Improved local resolution on the north-south components to the mineralized stockwork will be realized through variably oriented grade control drilling.

15 Mineral Reserve estimates

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. This Mineral Reserve is based on the November 2012 Mineral Resource. A revised Mineral Reserve will be completed in 2014 based on the December 2013 Mineral Resource and using revised economic parameters.

A net smelter return (NSR) cut-off grade of $180/t of ore was used to define the Mineral Reserves (as used in previous studies).

The NSR for each block in the resource model was calculated as the payable revenue for gold and silver less the costs of refining, concentrate treatment, transportation and insurance. The metal price assumptions are US$1,350/oz gold and US$22/oz silver.

Table 15.1 shows the dilution and recovery factors used in the Mineral Reserve estimation.

Table 15.1 Dilution factors and recovery factors by type of excavation

Type of excavation	Dilution factor* (%)	Recovery factor* (%)
Primary stopes	6.8	97.5
Secondary stopes	15.2	92.5
Sill pillar stopes**	15.2	75.0
Ore cross-cuts	4.0	100.0
Production slashing	7.5	100.0

*Notes: *Expressed on a weight basis.*
***Includes stope ore to 30 m beneath the surface crown pillar.*

The Mineral Reserves were delineated in an orebody consisting of numerous independent lenses in the VOK Zone and two distinct lenses in the West Zone (Table 15.2). The mineral reserves were developed from the resource model, "bjbm_1211_v2_cut" provided by Snowden–on behalf of Pretivm–to AMC in November 2012.

Table 15.2 Brucejack Mineral Reserves*, by Zone and by Reserve Category

Zone		Ore tonnes (Mt)	Grade		Metal	
			Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
VOK Zone	Proven	-	-	-	-	-
	Probable	15.1	13.6	11	6.6	5.3
	Total	15.1	13.6	11	6.6	5.3
West Zone	Proven	2.0	5.7	309	0.4	19.9
	Probable	1.8	5.8	172	0.3	10.1
	Total	3.8	5.8	243	0.7	30.0
Total Mine	Proven	2.0	5.7	309	0.4	19.9
	Probable	17.0	12.8	28	7.0	15.4
	Total	19.0	12.0	58	7.3	35.3

*Notes: *Rounding of some figures may lead to minor discrepancies in totals.*
**Based on Cdn$180/t cut-off grade, US$1,350/oz gold price, US$22/oz silver price, Cdn$/US$ exchange rate = 1.0*
**Based on November 2012 Mineral Resource estimate.*

16 Mining methods

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. This mine plan is based on the November 2012 Mineral Resource. A revised mine plan will be completed in 2014 based on an updated Mineral Reserve which uses the December 2013 Mineral Resource and revised economic parameters.

The underground mine design supports the extraction of 2,700 t/d of ore via transverse longhole open stoping (LHOS) and longitudinal LHOS. Paste backfill and modern trackless mobile equipment will be used. Mine access will be by a main decline from a surface portal close to the concentrator. A parallel decline will be dedicated to conveying crushed ore directly to the concentrator via a 650 m long conveyor. There will be a two-year pre-production development period, with steady-state production being reached in Year 2 of a 22-year LOM. The highest value ore will be targeted in the early production years. Steady-state production from years 2 through 18 will average about 980,000 t/a.

Geotechnical designs and recommendations are based on the results of site investigations, and geotechnical assessments that include rock mass characterization, structural geology interpretations, excavation and pillar stability analyses, and ground support design.

The groundwater flow system was conceptualized to provide inflow estimates to mine workings. These estimates referenced results of site investigations and hydrogeologic testing and were used to size dewatering equipment and as input to the process water balance.

During the pre-production period, most of the mobile equipment for development and stoping work will be supplied by the Owner and operated by a contractor. Key equipment requirements will include jumbos, load-haul-dumps (LHDs), haulage trucks, bolters, shotcrete sprayers, a long hole drill and a cable bolter. Raise development will be contracted out.

Manpower will consist of technical staff, mining crews, mechanics, electricians, and other support personnel. Pre-production manpower will be supplied by contractor, except for technical support. Mining Manpower attains 349 personnel at full production, with up to 163 personnel on site at any given time.

Infrastructure design is for a mine life over 20 years. Electric power use will be maximized.

The ventilation system is designed to meet BC regulations. Permanent surface fans will be located at the portals of the twin, intake declines. All intake air entering the mine will be heated above freezing point.

Paste fill distribution design is based on a dual pumping system. A positive displacement pump in the paste fill plant will provide paste to all of the West Zone and the lower zones of the VOK. The paste plant pump will also feed a booster pump located near the crusher station at the bottom of the conveyor ramp and near to the main entrance to the VOK area on the 1,330 Level.

Ore will be trucked from working areas to an underground crusher and then transferred to surface via transfer belt to a 42 inch main conveyor. Waste will be trucked to surface waste piles.

The mine will be dewatered using a dirty water system of sumps and pumps. Submersible and centrifugal pumps will be used for development and permanent mine operations. For underground worker safety, both permanent and portable refuge stations are planned. The emergency warning system will include phones, cap lamp warning system, and stench gas.

The total project mining capital, including a 10% contingency, was estimated at $210 million. Sustaining mining capital of $265 million has been estimated for the production period. The total underground operating cost over the LOM was estimated to be $1,769 million, at an average LOM cost of $94.40/t

17 Recovery methods

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the recovery methods and an associated updated Mineral Reserve is scheduled to be completed in 2014. This updated Mineral Reserve will be based on the December 2013 Mineral Resource and revised economic parameters.

The process flowsheet developed for the Brucejack mineralization is a combination of conventional bulk sulphide flotation and gravity concentration to recover gold and silver. The processing plant will produce a gold-silver bearing flotation concentrate and gold-silver doré that will be produced by melting the gravity concentrate produced from the gravity concentration circuits. Based on the LOM average, the recovery process is estimated to produce approximately 4,300 kg of gold and 1,500 kg of silver as doré per year and 42,000 tonne of gold-silver bearing flotation concentrate per year from the mill feed, grading 12.0 g/t gold and 57.9 g/t silver. The estimated gold recoveries to the doré and flotation concentrate are 41.6% and 54.9%, respectively, totalling 96.5%. The estimated silver recoveries reporting to the doré and flotation concentrate are 3.0% and 86.6%, respectively, totalling 89.6%. The LOM average gold and silver contents of the flotation concentrate are anticipated to be approximately 130 g/t gold and 1,000 g/t silver. The flotation concentrate will be shipped off site to a smelter for further treatment to recover the gold and silver.

The process plant will consist of:

- one stage of crushing (located underground)
- a surge bin with a live capacity of 2,500 tonne on surface
- a semi-autogeneous grinding (SAG) mill and ball mill primary grinding circuit integrated with gravity concentration
- rougher flotation followed by rougher flotation concentrate regrinding
- cleaner flotation processes.

A gravity concentration circuit will also be incorporated in the bulk concentrate regrinding circuit. The final flotation concentrate will be dewatered, bagged, and trucked to the transload facility in Terrace, BC. It will be loaded in bulk form into rail cars for shipping to a smelter located in eastern Canada. The gravity concentrate will be refined in the gold room on site to produce gold-silver doré.

A portion of the flotation tailings will be used to make paste for backfilling the excavated stopes in the underground mine, and the balance will be stored in Brucejack Lake. The water from the thickener overflows will be recycled as process make-up water. Treated water from the water treatment plant will be used for mill cooling, gland seal service, reagent preparation, and make-up water.

The simplified flowsheet for the operation is shown in Figure 17.1.

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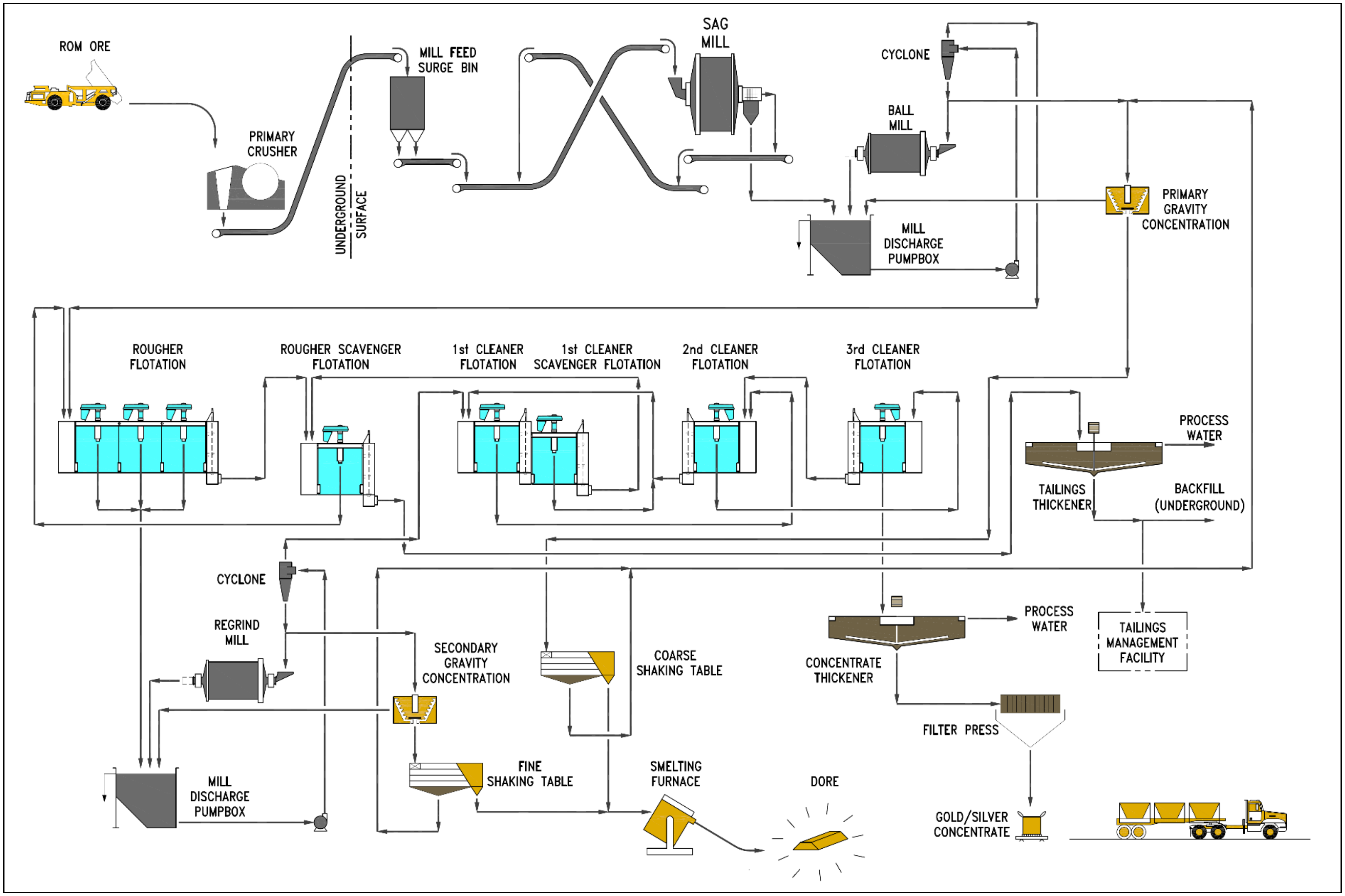
Figure 17.1 Simplified process flowsheet



18 Market studies and contracts

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the market studies and contracts and an associated updated Mineral Reserve is scheduled to be completed in 2014. This updated Mineral Reserve will be based on the December 2013 Mineral Resource and revised economic parameters.

The final products that will be produced at Brucejack will be gold and silver doré and a gold-silver flotation concentrate. The gold and silver doré will likely be transported to a North American-based precious metals refinery or sold to precious metals traders, most likely located in Asia, Europe, and North America. The flotation concentrate, will likely be sold to a base metal smelter or metal traders. Based on the LOM average, the gold-silver flotation concentrate is expected to contain approximately 130 g/t gold and 1,000 g/t silver.

Pretivm contacted the metal trader Transamine for information regarding concentrate sales, and subsequently received indicative smelting terms based on the assay data of the concentrate that was produced from the 2012 test work. According to the terms received from Transamine, it is anticipated that the concentrate will be trucked to Terrace, BC, and then transported by rail to a smelter in eastern Canada.

Tetra Tech recommended conducting further marketing studies for shipping concentrate to smelters located in Asia for a potential reduction in the shipping costs.

19 Project infrastructure

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the project infrastructure and an associated updated Mineral Reserve is scheduled to be completed in 2014. This updated Mineral Reserve will be based on the December 2013 Mineral Resource and revised economic parameters.

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet. Figure 19.1 illustrates the overall site layout for the Project. Figure 19.2 illustrates the mill site layout and Figure 19.3 illustrates the Knipple Transfer Station facility layout.

Project infrastructure will include:

- a 79 km access road at Highway 37 and travelling westward to Brucejack Lake with the last 12 km of access road to the mine site traversing the main arm of the Knipple Glacier
- internal site roads and pad areas
- grading and drainage
- avalanche hazard assessment
- transmission line
- ancillary facilities
- water supply and distribution
- water treatment plant
- waste disposal
- tailings delivery system
- Brucejack outlet control
- communications
- power supply and distribution
- fuel supply and distribution
- off-site infrastructure including the Bowser Airstrip and Camp and the Knipple Transfer Station facilities.

Figure 19.1 Overall site layout

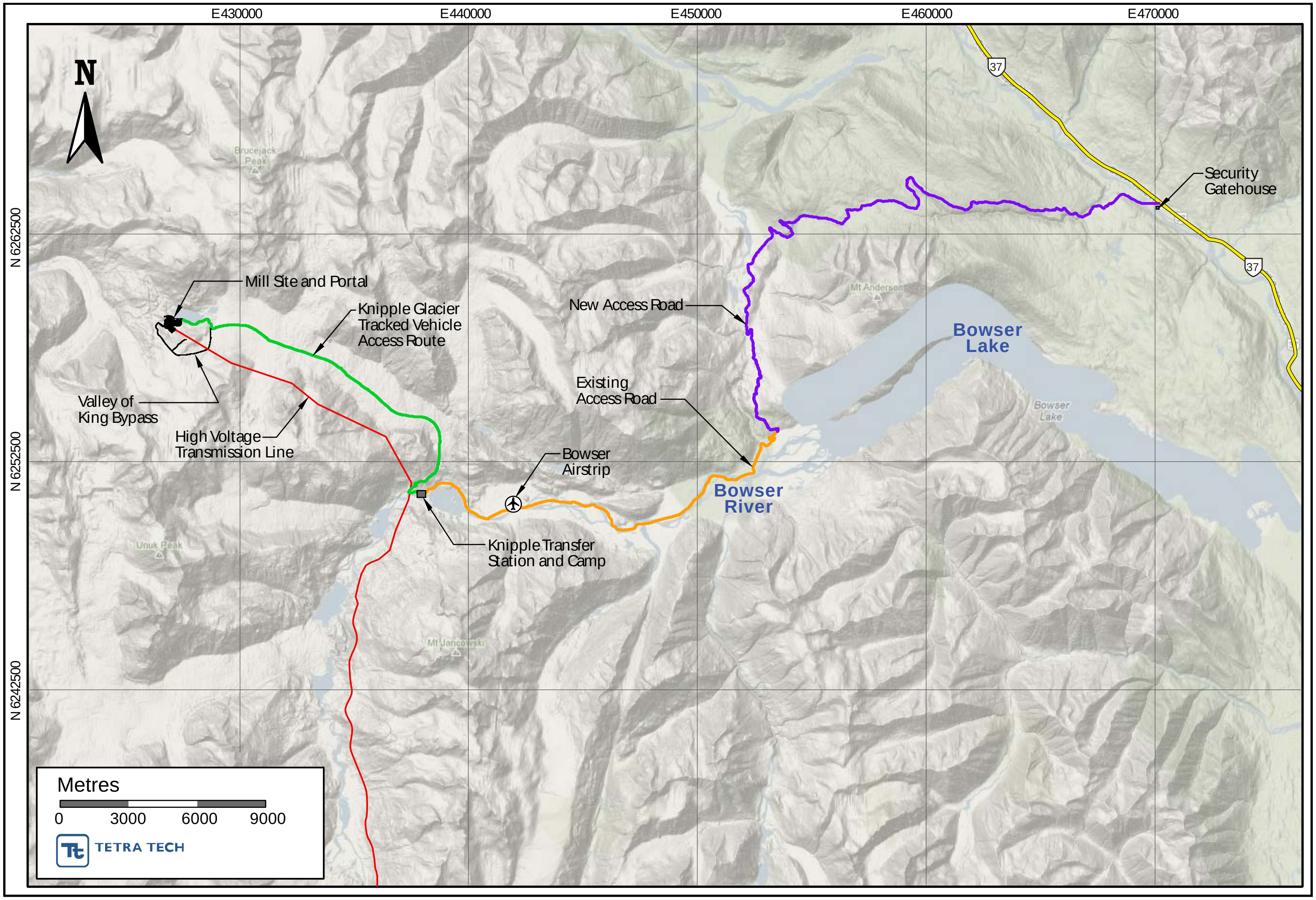


Figure 19.2 Mill site layout

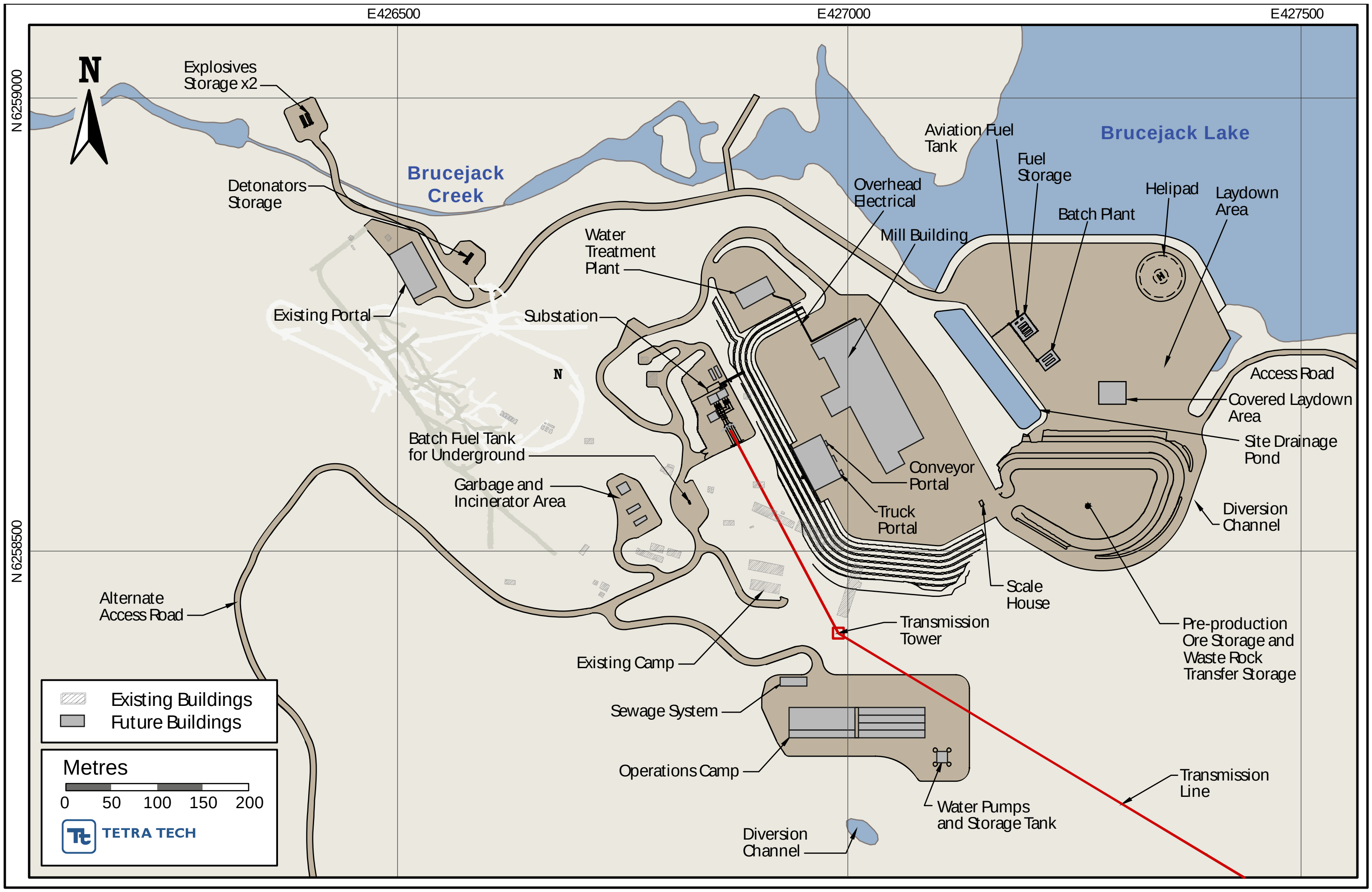
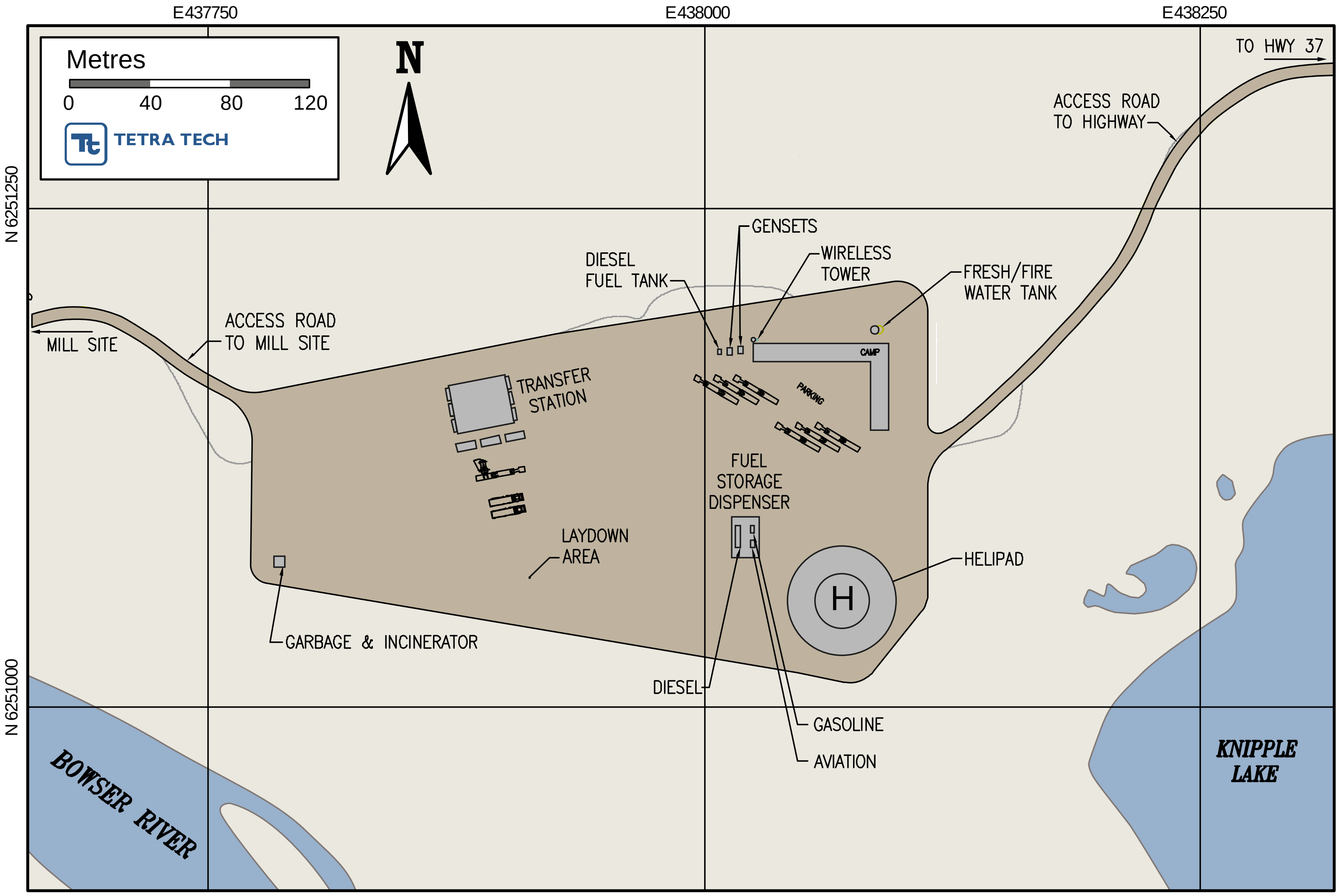


Figure 19.3 Knipple transfer station facility layout



19.1.1 Avalanche hazard assessment

An avalanche hazard assessment has been completed for the Project. Facilities and access routes are exposed to approximately 15 avalanche paths or areas. Avalanche magnitude varies between Size 2 and 4. Avalanche frequency varies between annual and 1:100 years. Potential consequences of avalanches reaching the Brucejack mine facilities, transmission line, worksites, and roads include damage to infrastructure, worker injury (or fatality), and project delays. Potential consequences of static snow loads on transmission towers include damage to towers and foundations, and potential loss of electrical service to the mine. Without mitigation to the effects of avalanches and static snow loading, there is a high likelihood of some of the above consequences affecting operations on an annual basis.

Avalanche mitigation for the Project includes location planning, in order to avoid placement of facilities in avalanche hazard areas. For areas where personnel and infrastructure may be exposed, an avalanche management program will be implemented for mine operations during avalanche season (October through June). The program will utilize an Avalanche Technician team to determine periods of elevated avalanche hazard and provide recommendations for closures of hazard areas. The options for reducing control include explosive control, or waiting for natural settlement. Areas that are expected to have increased frequency of hazard and consequences will be evaluated for the installation of the remote avalanche control system (RACS) in order to allow for avalanche explosive control during reduced visibility (darkness and during storms). An allowance has been made in the capital and operating cost estimates for six RACSs.

19.1.2 Transmission line

For the Brucejack transmission line, Pretivm retained Valard to review potential routes and develop an initial design for the transmission line to the Project site, based on Valard's current experience in the area. To this end, Valard reviewed potential routes and determined the preferred route to be an extension from an existing transmission line from a hydro generation facility to the south (near Stewart, BC) to the Project site. Based on the terrain and the expected construction conditions, single metal monopole towers are recommended for the design. Site review indicates that the hazards in the area can be avoided through diligent siting of the tower structures as well as through an active snow avalanche program.

19.1.3 Tailings delivery system

Approximately one half of the tailings produced by mine operations will be stored underground as paste backfill and approximately one half will be placed on the bottom of Brucejack Lake. Tailings will be pumped from the tailings thickener at the process plant by slurry pipeline to the deepest location in the lake (80 m depth). A deposit of solids will intentionally be allowed to build over the end of the outfall. The solids will act as a filter to minimize the transport of fine particulate solids to the surface layer of Brucejack Lake with the goal of minimizing total suspended solids (TSS) concentrations in the lake outflow. This approach–discharge through a deposit of tailings–has successfully reduced suspended solids concentrations at other projects.

Fine particulate solids may also be suspended in the lake surface layer if fine waste rock is placed in the lake. Investigations on minimizing or eliminating this source of suspended solids in the lake outflow are underway

20 Environmental studies and permitting

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information.

Major mining projects in BC are subject to environmental assessment and review prior to certification and issuance of permits to authorize construction and operations. Environmental assessment is a means of ensuring the potential for adverse environmental, social, economic, health, and heritage effects or the potential adverse effects on Aboriginal interests or rights are addressed prior to project approval. Depending on the scope of a project, assessment and permitting of major mines in BC will proceed through the BC Environmental Assessment (EA) process pursuant to the BC Environmental Assessment Act (BCEAA) and the Canadian Environmental Assessment Act 2012 (CEAA).

At a provincial level, proposed mining developments that exceed a threshold criterion of 75,000 t/a (or 205 t/d) as specified in the Reviewable Project Regulations, are required under the BCEAA to obtain an Environmental Assessment Certificate from the Ministry of Energy Mines and Natural Gas and Ministry of Environment before the issuance of any permits to construct or operate. The Project will thus require a provincial Environmental Assessment Certificate, because its proposed production rate exceeds the specified threshold.

At a federal level, proposed gold mine developments (other than placer mines) that exceed a threshold criterion of 600 t/d as specified under the Regulations Designating Physical Activities, are required to complete an Environmental Impact Study (EIS) pursuant to the CEAA. Thus completion of an EIS will be necessary for the Project since the proposed production rate exceeds the specified threshold.

Pretivm has formally entered both the provincial and federal EA processes. While the provincial and federal decisions are made independently, the two levels of government work together to allow for a coordinated effects assessment process. In relation to the provincial EA process, Pretivm has submitted a Project Description and has received Section 10 and 11 orders under the BCEAA. Federally, Pretivm has submitted a Project Description and has received final EIS guidelines from the Canadian Environmental Assessment Agency. Pretivm is targeting completion of a combined application for an Environmental Assessment Certificate (EAC) and EIS by the end of the first quarter of 2014. Provincial and federal decisions on the EA process are expected in Q4 2014 or Q1 2015. Provincial approval of the EAC application and federal approval of the EIS will then allow for the issuance of the necessary statutory permits and authorizations to commence construction of the Project.

21 Capital and operating costs

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the capital and operating costs and an associated updated Mineral Reserve is scheduled to be completed in 2014. This updated Mineral Reserve will be based on the December 2013 Mineral Resource and revised economic parameters.

21.1 Capital costs

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $663.5 million. A summary breakdown of the initial capital cost, including direct costs, indirect costs, Owner's costs, and contingency is provided in Table 21.1.

Table 21.1 Summary of initial capital cost

Major area	Area description	Capital cost ($ million)
Direct costs		
	Mine site	32.7
	Mine underground	174.5
	Mine site process	80.1
	Mine site utilities	23.7
	Mine site facilities	43.7
	Mine site tailings	3.5
	Mine site temporary facilities	10.2
	Mine site (surface) mobile equipment	14.3
	Off site infrastructure	69.1
Subtotal direct costs		**451.8**
	Indirect costs	125.0
	Owner's costs	22.3
	Contingencies	64.4
Total Initial capital cost		**663.5**

Note: numbers may not add up due to rounding

This estimate was a Class 4 feasibility cost estimate prepared in accordance with the standards of the Association for the Advancement of Cost Engineering International (AACE). There was no deviation from the AACE's recommended practices in the preparation of this estimate.

This feasibility estimate was prepared with a base date of Q2 2013 and does not include any escalation beyond this date. The quotations used for this Feasibility Study estimate were obtained in Q2 2013, and had a validity period of 90 days.

The capital cost estimate uses Canadian dollars as the base currency. Foreign exchange rates were applied as required. Duties and taxes and taxes are not included in the estimate. This estimate was developed based largely on first principles and was prepared from a design, planning, and cost basis.

21.2 Operating costs

The total LOM average operating cost for the Project is estimated at $156.46/t ore milled which includes for:

- mining
- process
- material re-handling in Year 1 for the stockpiled ore produced during pre-production
- general and administration (G&A)
- surface services
- backfill, including paste preparation
- water treatment.

The operating costs exclude sustaining capital costs, off-site costs (such as shipping and smelting costs), taxes, permitting costs, or other government imposed costs, unless otherwise noted.

A total of 542 personnel (underground and surface) are projected to be required for the Project. The unit cost estimates are based on the LOM ore production and a mine life of 22 years. The currency exchange rate used for the estimate is 1:1 (Cdn$:US$). The operating cost for the Project has been estimated in Canadian dollars within an accuracy range of ±15%. A summary of the overall operating cost is presented in Table 21.2. The cost distribution is illustrated in Figure 21.1.

Table 21.2 Overall operating cost

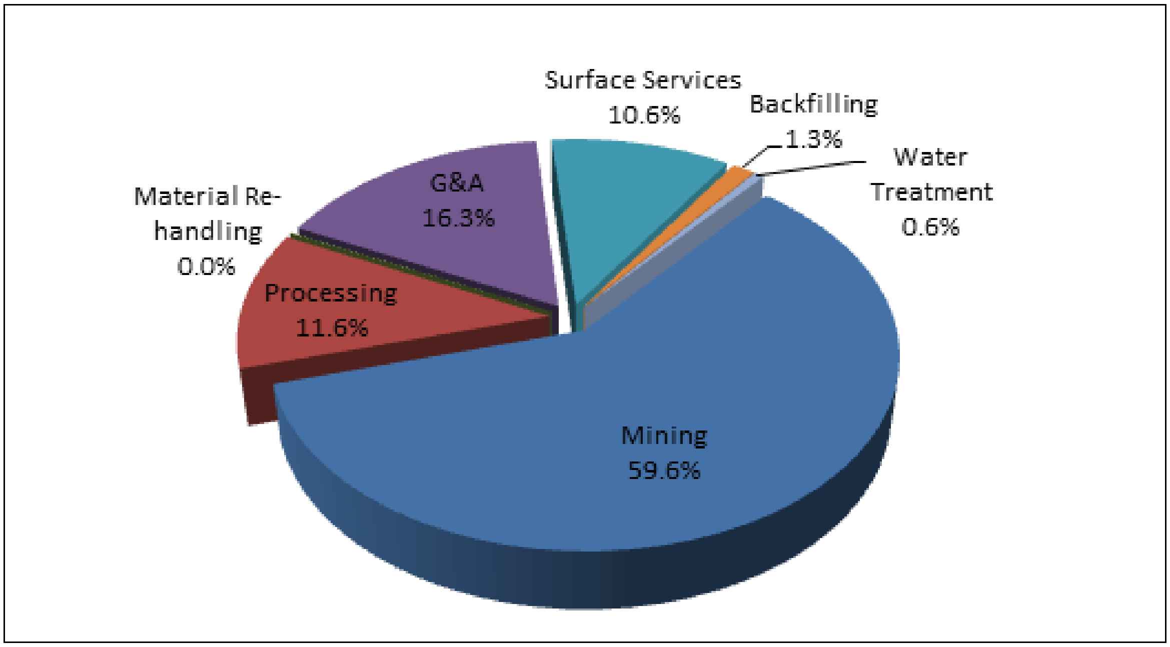
Area	Personnel	Unit operating cost ($/t milled)
Mining*	316**	93.18
Processing	95	18.16
Material re-handling***	Contract	0.07
G&A	43	25.47
Surface services	78	16.53
Backfilling	6	2.10
Water treatment	4	0.95
Total	**542**	**156.46**

Notes: *Average LOM mining cost including crushing cost and cement cost for backfill; if excluding the ore mined during preproduction, the estimated unit cost is $94.40/t.

**316 workers during Year 1 to 14 and then reduce to 167 workers at the end of the mine life.

***Material re-handling cost is the LOM average cost, which will occur in Year 1 only. The operation is assumed to be contracted with approximately eight workers required.

Figure 21.1 Overall operating cost distribution



22 Economic analysis

Information in this section has been excerpted and summarised from the Feasibility Study reported in 2013 (Ireland et al., 2013). *The reader is referred to Ireland et al. (2013) for detailed information.* Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the economic analysis and an associated updated Mineral Reserve is scheduled to be completed in 2014. This updated Mineral Reserve will be based on the December 2013 Mineral Resource and revised economic parameters.

Tetra Tech prepared an economic evaluation of the Project based on a pre-tax financial model. For the 22-year LOM and 18.99 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 42.9% internal rate of return (IRR)
- 2.1-year payback on the US$663.5 million initial capital
- US$2,687 million net present value (NPV) at 5% discount rate.

A post-tax economic evaluation of the Project was prepared with the inclusion of applicable taxes (Section 22.0).

The following post-tax financial parameters were calculated:

- 35.7% IRR
- 2.2-year payback on the US$663.5 million initial capital
- US$1,763 million NPV at 5% discount rate.

The base case metal prices used for this study are as follows:

- gold – US$1,350/oz
- silver – US$20.00/oz
- exchange rate – 1.00:1.00 (US$:Cdn$).

23 Adjacent properties

Snowden notes:

- This information was publicly disclosed by the Owner or Operator of the adjacent property and was sourced as per the notes in the relevant section below.
- The QP has been unable to verify the information provided here, except against what has been publicly reported.
- The information is not necessarily indicative of the mineralization at Brucejack.

23.1 Kerr-Sulphurets-Mitchell

Within the adjacent KSM Property there are four copper-gold mineral deposits, namely Kerr, Mitchell, Sulphurets, and Iron Cap. All of these occurrences are situated within the claim holdings that are, at the time of writing this report, owned and operated by Seabridge Gold.

Seabridge Gold acquired the KSM Property from Placer Dome in June 2000.

In May 2012, Seabridge published a revised preFeasibility Study, which resulted in a Mineral Reserve of 2.2 Bt (2.2 billion tons) of gold, copper, silver, and molybdenum ore (Table 23.1). Seabridge Gold reported that all ore will be mined using open pit methods for the first 25 years, and will switch to underground block caving in Year 26. Over the entire 55-year mine life, ore will be fed to a flotation mill, which will produce a combined gold/copper/silver concentrate. The concentrate will be transported by truck to the nearby deep-water sea port at Stewart, BC, for shipment to a Pacific Rim smelter. Extensive metallurgical testing confirmed that KSM could produce a clean concentrate with an average copper grade of 25%, making it readily saleable. Separate molybdenum concentrate and gold-silver doré will be produced at the KSM processing facility. All information for this section has been taken from the Seabridge Gold website (www.seabridgegold.net).

The QP for this section of the report has not verified the information concerning the KSM Deposits, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

Table 23.1 Mineral Reserve estimates for the KSM Property as at 31 December 2012

Zone	Mining method	Reserve category	Tonnes (Mt)	Average grades				Contained metal			
				Gold (g/t)	Copper (%)	Silver (g/t)	Moly (ppm)	Gold (Moz)	Copper (Mlb)	Silver (Moz)	Moly (Mlb)
Mitchell	Open Pit	Proven	476	0.67	0.17	3.05	60.9	10.3	1,798	47	64
		Probable	497	0.61	0.16	2.78	65.8	9.8	1,707	44	72
	Block Cave	Probable	438	0.53	0.17	3.48	33.6	7.4	1,589	49	32
Iron Cap	Block Cave	Probable	193	0.45	0.20	5.32	21.5	2.8	834	33	9
Sulphurets	Open Pit	Probable	318	0.59	0.22	0.79	50.6	6.0	1,535	8	35
Kerr	Open Pit	Probable	242	0.24	0.45	1.2	0.0	1.9	2,425	9	0
		Proven	476	0.67	0.17	3.05	60.9	10.3	1,798	47	64
		Probable	1,688	0.51	0.22	2.65	40.1	27.9	8,090	144	149
		Total	2,164	0.55	0.21	2.74	44.7	38.2	9,888	191	213

Note: Cut-off values were defined based on NSR values and type of mining. The reader should refer to the information provided by Seabridge Gold to get an accurate appreciation of the definition of the cut-off values for reporting.

23.2 High Property

The Teuton Resources Corporation (Teuton) High Property is located immediately to the south of the Brucejack Property. Teuton conducted limited preliminary exploration of the High Property in 2011, 2012, and 2013 including prospecting, collection of surface grab samples, and drilling. Results posted on Teuton's website (www.teuton.com) indicate the presence of porphyry-style gold and base metal sulphide mineralization on the High Property. A single drillhole through a hypabyssal porphyry body was reported as intersecting 222 m of 0.88 g/t Au (in the King Tut Zone). Several recent surface grab samples (UH-1 through UH-10) returned assays of 4.8-63 g/t Au, 18-86 g/t Ag, 0.31-2.42% Pb, and 1.4-6.5% Zn for mineralization hosted in quartz vein stockworks in pervasively altered chlorite-sericite andesitic volcanic and volcaniclastic rocks.

The QP for this section of the report has not verified the information concerning the High Property, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

23.3 Treaty Creek Property

The Treaty Creek Property, the title for which is currently under litigation (www.teuton.com and www.americancreek.com) adjoins directly northeast of the Seabridge Gold's KSM gold-copper property and is underlain by a similar geology. Exploration work uncovered several zones, the most promising of which are the Copper Belle (porphyry-style), GR2 (feeder zone to a volcanogenic massive sulphide (VMS)), Eureka (porphyry-style with a gold-silver epithermal overprint), and Treaty Ridge (VMS/Sedex?) zones.

The QP for this section of the report has not verified the information concerning the Treaty Creek Property, and the information is not necessarily indicative of the mineralization at the Brucejack Project.

24 Other relevant data and information

24.1 Preliminary economic assessment 2011

Pretivm commissioned Wardrop to complete a preliminary economic assessment ("PEA") on the high grade gold and silver resources at the Brucejack Project as a "stand-alone" project, and results were made public in June, 2011 (Wardrop Engineering Inc., 2011). The results of this report are no longer current.

The following consultants were commissioned to complete the component studies for the NI 43-101 Technical Report and Preliminary Economic Assessment:

- Wardrop: processing, infrastructure, capital cost estimate, processing, operating cost estimate, and financial analysis;
- AMC Mining Consultants (Canada) Ltd. (AMC): mining including mine capital and operating cost estimates;
- P&E Mining Consultants Inc. (P&E): Mineral Resource estimate;
- Rescan Environmental Services Ltd. (Rescan): environmental aspects, waste and water treatment;
- BGC Engineering Inc. (BGC): tailings impoundment facility, waste rock and water management, and geotechnical design.

24.2 Updated preliminary economic assessment 2012

Pretivm commissioned Wardrop to complete an updated PEA on the Brucejack Project as an underground mining project in late 2011, and results were reported in Ghaffari et al., (2012). The results of this report are no longer current.

The following consultants were commissioned to complete the component studies for the NI 43-101 Technical Report and Preliminary Economic Assessment:

- Wardrop: processing, infrastructure, capital cost estimate, processing, operating cost estimate, and financial analysis;
- AMC Mining Consultants (Canada) Ltd. (AMC): mining including mine capital and operating cost estimates;
- P&E Mining Consultants Inc. (P&E): Mineral Resource estimate;
- Rescan Environmental Services Ltd. (Rescan): environmental aspects, waste and water treatment;
- BGC Engineering Inc. (BGC): tailings impoundment facility, waste rock and water management, site wide groundwater studies, and geotechnical design for onsite facilities;
- GeoSpark Consulting Inc. (GeoSpark): quality assurance and quality control (QA/QC) and database management.

24.3 Project execution plan

Information in this section has been excerpted from the Feasibility Study reported in 2013 (Ireland et al., 2013). The reader is referred to Ireland et al. (2013) for detailed information. Note that the feasibility study and derived information was based on the November 2012 Mineral Resource.

An update to the Project execution plan is scheduled to be completed in 2014.

This information is based on the November 2012 Mineral Resource. An updated project execution plan will be completed in 2014 based on an updated Mineral Reserve which uses the December 2013 Mineral Resource and revised economic parameters

The Project will take approximately 37 months to complete from the start of basic engineering, through construction, to introduction of first material into the mill. A further three to four months is planned for commissioning and production ramp-up. The Project execution schedule was developed to a Level 2 detail of all activities required to complete the Project.

The Project will transition from the study phase to basic engineering in Q3 2014 and will move forward in the following phases:

- Stage I – early works including mine development, the environmental assessment certificate (EAC) application, permitting, access road upgrades, preliminary power transmission line right-of-way (ROW), basic engineering, and the procurement of long-lead equipment.
- Stage II – full project execution (following permit approval), including detailed engineering, procurement, construction team mobilization, construction, and commissioning.

The Project schedule identifies the following significant key milestone dates (Table 24.1) from feasibility completion to project handover.

Table 24.1 Key milestone dates

Year	Quarter	Activity
2013	2	Feasibility Study Completion
2013	3	Start of Basic Engineering
2014	1	EPCM Award
2014	3	Start of Stage I Early Infrastructure Construction Works
2015	1	Detailed Engineering Completion
2015	1	Start of Stage II Mine Site Surface Construction
2015	4	Mechanical Completion Stage I Works
2016	2	Mechanical Completion Stage II Works
2016	3	Underground Development Completion
2016	3	Mine Site Commissioning Completion
2016	3	Project Handover

Note: EPCM = engineering, procurement, construction management

25 Interpretation and conclusions

An updated Mineral Resource estimate has been prepared for the VOK Zone at the Brucejack Property of Pretivm located in northwest British Columbia. The Measured, Indicated and Inferred Mineral Resource estimates, effective December 2013 are intended for use in a Feasibility Study for a high grade underground mining scenario.

In the current study, Olssen & Jones have estimated and reported Mineral Resource estimates for the high-grade portions of West Zone and the VOK that are considered to be potentially minable by underground methods, regardless of any open pit potential.

In 2013, a Feasibility Study was reported (Ireland et al., 2013) based on the November 2012 Mineral Resource (Jones, 2012c). This study considered the potential for underground mining of the high-grade portions of the deposits.

Subsequent to the completion of the feasibility study, a bulk sample was completed from the VOK. This sample was processed in nominal 100 tonne parcels through a sample tower, but the variability in grades from the sample tower on each 100 tonne round was considered too high to give an accurate representation of the grade of each round. No further use of the sample tower results was considered appropriate at Brucejack.

The 100 tonne parcels from the bulk sample were shipped to Montana and processed to give the ultimate grade of the bulk sample. The results of the processing were compared with the results of the sample tower and this confirmed the poor accuracy of the sample tower results. There was no further use of the sample tower results.

Processing of the bulk sample showed that the bulk sample responded well to the conventional combined gravity and flotation flowsheet. On average, between 96% and 97% of the gold and 91% and 92% of the silver were recovered to the gravity concentrate and bulk flotation concentrate at the grind size of 80% passing approximately 70 to 80 µm. There was, however, a significant variation in metallurgical performances among the samples tested, with a greater percentage of gold reporting to the flotation concentrate for the lower grade mineralization and a greater percentage of the gold reporting to the gravity concentrate for the higher grade mineralization. This is interpreted to be a result of the variably nuggetty mineralization.

The results of processing of the bulk sample indicated that, whilst there was apparent over-smoothing of the grade estimate locally, the resource estimate under-estimated the grade within the bulk sample. Over-smoothing was manifested in under-estimation of high grade areas and over-estimation of adjacent low grade areas. The number of ounces within the bulk sample, as estimated by the November 2012 Mineral Resource, was more than 10% lower than the number of ounces from processing (although this difference may change depending on the final smelter settlements. These data are not yet available.).

Associated with the bulk sample, Pretivm also completed a substantial amount of underground drilling. This drilling was closely spaced, but based mostly on a north-south grid and appears to have created a directional bias in the drilling information because of a north-south aligned mineralization along the Cleopatra structure. However, this drilling, along with the results of processing of the bulk sample, was used to assist in the improvement of grade estimation parameters. It was noted as a part of this testwork, however, that the result of including the new drilling information in the resource estimation further under-estimated the grade in the bulk sample because of this directional bias.

The December 2013 Mineral Resource was estimated using all drilling information (including the underground drilling in the bulk sample area), even though it was fully understood that the resource estimate was biased low in the vicinity of the bulk sample.

The December 2013 Mineral Resource also confirms the contained metal represented by the November 2012 Mineral Resource (within adequate limits) and extends the Mineral Resource based on some new information. The principal difference is slightly less tonnes and higher grade, whilst retaining the contained metal locally (a response to the reduction in smoothing during grade estimation). In addition to the improvements in the model and the comparison with hard data (attained from the processing of the bulk sample), the current study has increased the confidence in the Mineral Resources.

As a result of the increased understanding of the mineralization and increased confidence in the resource estimate, a summary of the feasibility findings has been reproduced in this report. Principally, the findings of the study remain valid as the December 2013 Mineral Resource confirms the contained metal represented by the November 2012 Mineral Resource, with the tonnes decreasing by approximately 5% and the grade increasing by approximately 7%. An amended Feasibility Study for the VOK based on the updated Mineral Resource is expected in the first half of 2014.

Pretivm will continue to advance engineering at the Project in support of the ongoing permitting process, and anticipates filing its application for an Environmental Assessment Certificate in the first quarter of 2014. After obtaining permits, and subject to a production decision, Pretivm anticipates commencing construction of the mine in late 2014 or first quarter of 2015.

26 Recommendations

The author makes the following recommendations:

- Update the Feasibility Study to reflect the December 2013 Mineral Resource.
- Complete analysis to determine the optimum drill density for stope definition. (e.g. 7.5 m by 7.5 m or 10 m by 10 m).
- Extend a ramp down to the 1270 m level and open up that level to provide access to complete high density definition drilling down dip of the current underground drilling and along trend to the east.
- Extend a ramp up to the 1390 m level and open that level to provide access to complete high density definition drilling up dip of the current underground drilling and along strike to the west.
- Extend the 1270 m level approximately 400 m to the east and complete resource definition drilling of the far eastern Inferred Resources.
- When planning further drilling programs take into account orientation bias associated with variable vein directions in the mineralized stockwork system.

The budget for phase 1 of the program consists of:

- $6.5 million for development of 400 m of access ramp and lateral drift
- $3.5 million for drilling of 15,000 m of underground drilling off the access drift

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28 Certificate of author

I, Ivor W.O. Jones, Senior Principal Consultant of Snowden Mining Industry Consultants Inc., 87 Colin Street, West Perth, Western Australia; do hereby certify that:

(a) I am the author of the report titled, "Pretium Resources Inc.: Brucejack Project Mineral Resource Update Technical Report" (the "Technical Report") with an effective date of 19 December, 2013.

(b) I graduated with an Honours Degree in Bachelor of Science in Geology from Macquarie University in Sydney in 1986. In 2001 I graduated with a Master of Science degree in resource estimation from the University of Queensland. I am: a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy. I have worked as a geologist continuously for a total of 25 years since graduation. I have been involved in resource evaluation for 20 years and consulting for 15 years, including resource estimation of primary gold deposits for at least 5 years. I have been involved in gold exploration and mining operations for at least 5 years. I have read the definition of 'qualified person' set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument.

(c) I visited the Brucejack Property from 15 February to 16 February, 2012, from 3 June to 6 2013 and from 16 to 21 August 2013.

(d) I am responsible for the overall preparation of the Report.

(e) I am independent of the issuer as defined in section 1.4 of the Instrument.

(f) I have had prior involvement with the property that is the subject of the Report. I completed prior reports dated 30 April 2012, 18 September and 20 November 2012; and have reviewed a technical review prepared by Dr W. Board of Silver Standard in 2010.

(g) I have read the Instrument and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.

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

(i) I consent to the filing of the Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Report.

Dated at Brisbane this 1st Day of February, 2014.

Ivor W.O. Jones

Appendix A Sample tower results

Figure A1 Sample tower round details – 1345 m level plan - eastern side

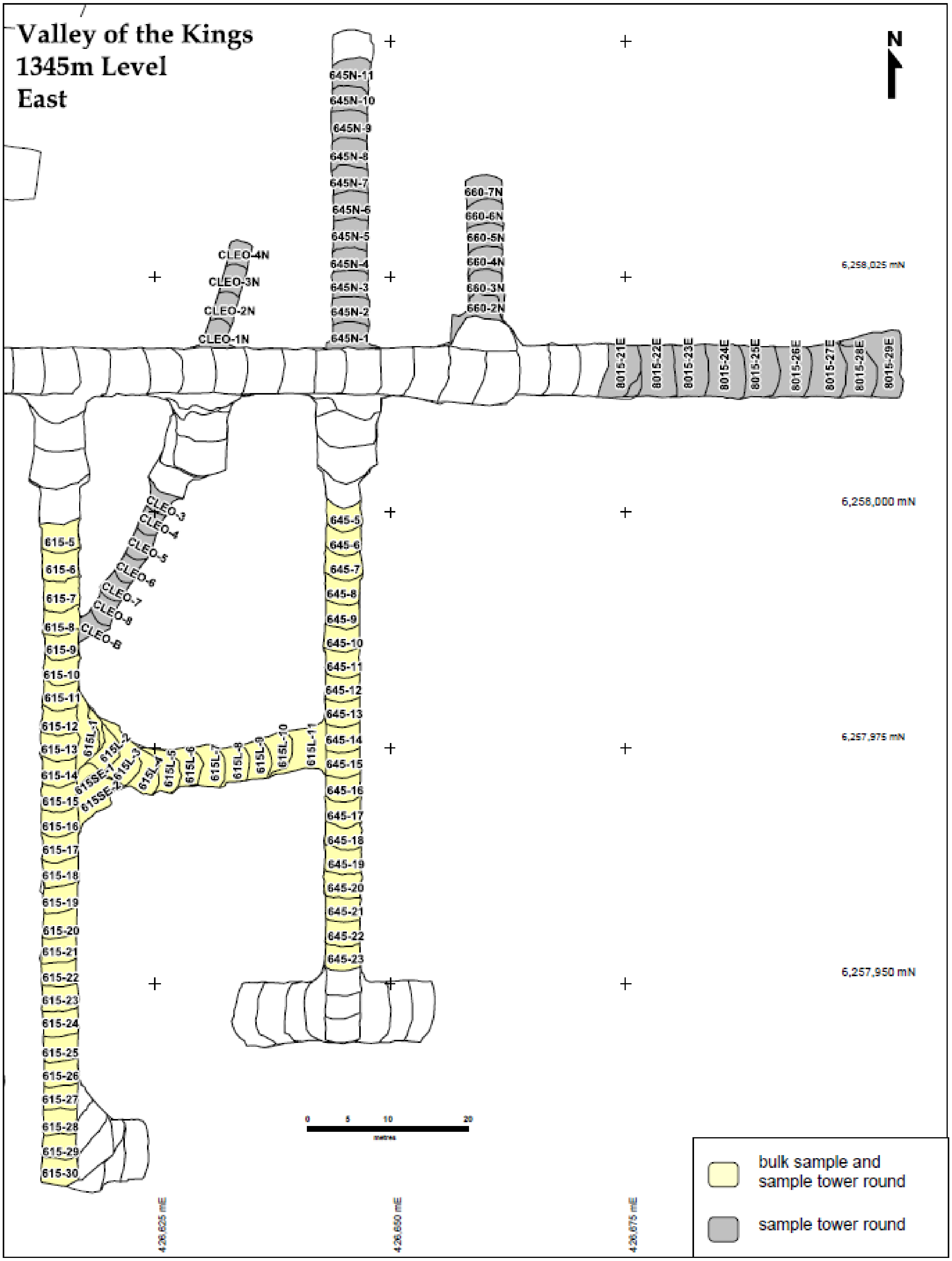
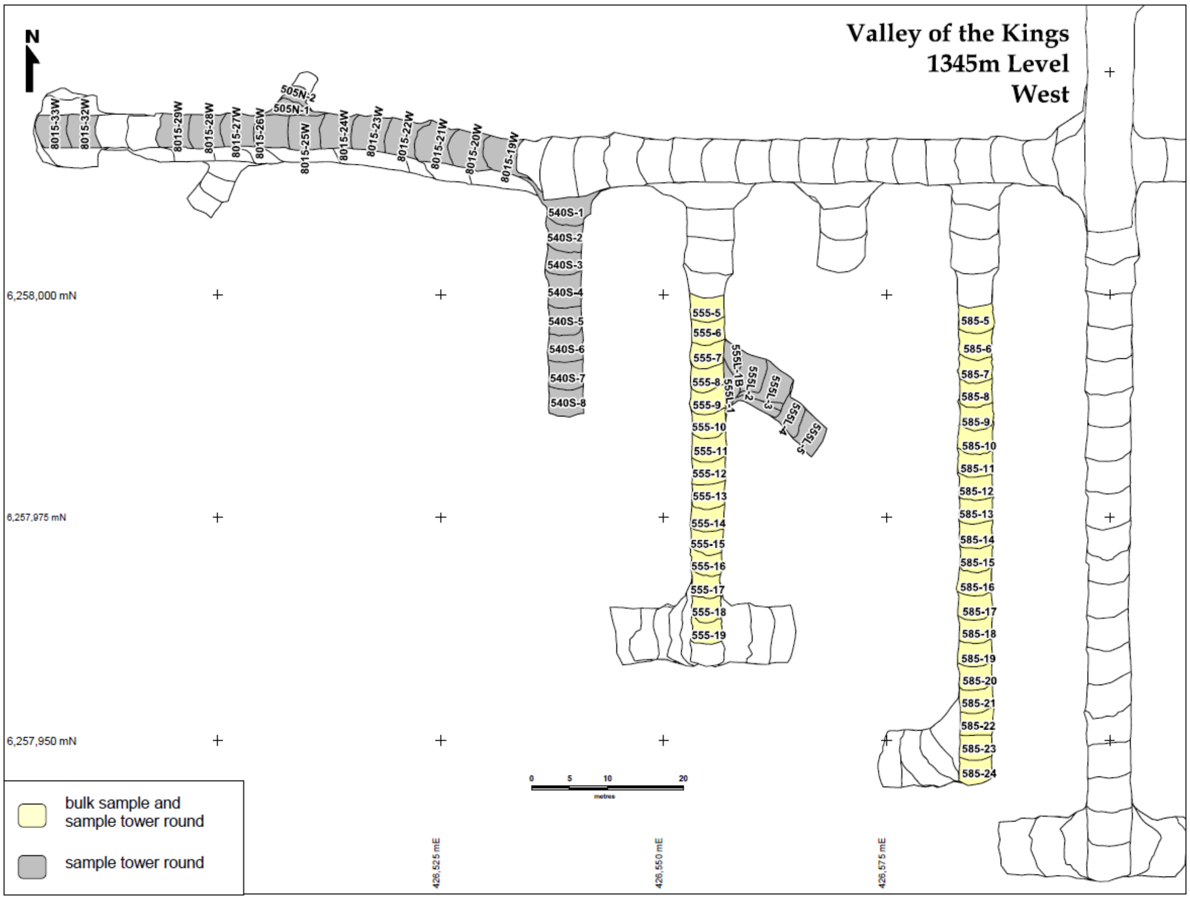


Figure A2 Sample tower round details – 1345 m level plan - western side



555 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
5-555E XC	1	A-1,A-2.A-3	L 366 551, 552, 553	79.7	0.059%	49.0	1	G & T	1,985	**3.11**
							2	G & T	1,995	**1.62**
		B-1, B-2. B-3,B-4	L 366 554, 555, 556, 557	79.8	0.059%	40.3	1	G & T	1,962	**1.47**
	2	C-1. C-2	Not assayed	59.0	0.044%					
		D-1, D-2	Not assayed	48.7	0.036%					
	3	E-1, E-2	L 366 563, 564	56.2	0.042%	53.9	1	G & T	1,994	**1.22**
							2	G & T	1,981	**1.16**
		F-1, F-2	L 366 565, 566	42.6	0.032%		1	G & T	1,974	**2.50**
	4	G-1, G-2	Not assayed	41.9	0.031%					
		H-1, H-2	Not assayed	36.5	0.027%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
6-555E XC	1	A	L 366 571	32.1	0.032%	31.5	1	G & T	2,010	**0.51**
							2	G & T	2,014	**0.38**
		B	L 366 573	30.2	0.030%	29.8	1	G & T	2,007	**0.42**
	2	C	Not assayed	31.0	0.031%					
		D	Not assayed	30.7	0.031%					
	3	E	L 366 576	30.9	0.031%	30.4	1	G & T	1,961	**0.39**
							2	G & T	1,966	**0.42**
		F	L 366 577	29.8	0.030%	29.4	1	G & T	2,011	**0.39**
	4	G	Not assayed	29.0	0.029%					
		H	Not assayed	30.9	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
7-555E XC	1	A-1, A-2	L 366 580, 581	36.9	0.038%	36.1	1	G & T	1,946	**7.7**
							2	G & T	1,962	**6.6**
		B-1, B-2	L 366 582, 583	50.9	0.052%	34.0	1	G & T	1,933	**10.7**
	2	C	Not assayed	35.4	0.036%					
		D	Not assayed	32.8	0.034%					
	3	E	L 366 588	33.7	0.035%	33.1	1	G & T	2,015	**4.6**
							2	G & T	1,911	**6.3**
		F-1, F-2	L 366 589, 590	34.3	0.035%	33.6	1	G & T	1,950	**4.2**
	4	G	Not assayed	31.9	0.033%					
		H-1, H-2	Not assayed	34.7	0.036%					

555 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
8-555E XC	1	A-1, A-2	L 366 595	32.6	0.034%	32.1	1	G & T	1,939	**27.1**
							2	G & T	1,994	**26.0**
		B-1, B-2	L 366 596	32.8	0.034%	32.2	1	G & T	1,959	**10.4**
	2	C	Not assayed	30.4	0.031%					
		D	Not assayed	31.6	0.033%					
	3	E	L 366 599	30.3	0.031%	29.8	1	G & T	1,934	**17.4**
							2	G & T	1,962	**10.6**
		F	L 366 600	29.9	0.031%	29.5	1	G & T	1,948	**11.1**
	4	G	Not assayed	28.7	0.030%					
		H	Not assayed	30.3	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
9-555E XC	1	A-1, A-2	L 366 604, 605	37.7	0.038%	37.1	1	G & T	1,982	**16.1**
							2	G & T	1,974	**15.0**
		B-1, B-2	L 366 606, 607	40.0	0.040%	39.4	1	G & T	1,993	**20.5**
	2	C-1, C-2	Not assayed	37.8	0.038%					
		D-1, D-2	Not assayed	38.4	0.039%					
	3	E	L 366 612	31.6	0.032%	31.1	1	G & T	1,965	**14.1**
							2	G & T	1,975	**15.2**
		F	L 366 613	33.2	0.034%	32.7	1	G & T	1,931	**37.4**
	4	G	Not assayed	29.2	0.029%					
		H	Not assayed	32.8	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
10-555E XC	1	A	L 366 617	32.9	0.032%	32.4	1	G & T	1,936	**1.24**
							2	G & T	1,920	**0.85**
		B-1, B-2	L 366 618, 619	35.1	0.034%	34.7	1	G & T	1,920	**20.38**
	2	C	Not assayed	32.1	0.031%					
		D	Not assayed	32.3	0.032%					
	3	E	L 366 622	31.1	0.030%	30.7	1	G & T	1,916	**3.64**
							2	G & T	2,006	**2.79**
		F	L 366 623	31.6	0.031%	31.2	1	G & T	1,970	**1.02**
	4	G	Not assayed	31.6	0.031%					
		H	Not assayed	30.4	0.030%					

555 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
11-555E XC	1	A	L 366 627	32.3	0.032%	32.1	1	G & T	1,952	**2.44**
							2	G & T	1,922	**1.97**
		B	L 366 628	32.1	0.032%	31.8	1	G & T	1,889	**2.33**
	2	C	Not assayed	32.2	0.032%					
		D	Not assayed	32.1	0.032%					
	3	E	L 366 631	31.7	0.031%	31.4	1	G & T	1,924	**5.17**
							2	G & T	1,910	**10.49**
		F	L 366 632	32.5	0.032%	33.2	1	G & T	1,976	**4.69**
	4	G	Not assayed	31.2	0.031%					
		H	Not assayed	30.3	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
12-555E XC	1	A	L 366 635	31.6	0.030%	31.4	1	G & T	1,903	**1.45**
							2	G & T	1,899	**1.58**
		B	L 366 636	31.6	0.030%	31.4	1	G & T	1,779	**1.67**
	2	C	Not assayed	31.8	0.030%					
		D	Not assayed	30.9	0.030%					
	3	E	B 00 314 769	32.0	0.031%	31.5	1	G & T	1,941	**1.99**
							2	G & T	1,887	**2.01**
		F	B 00 314 770	30.5	0.029%	30.0	1	G & T	1,977	**1.52**
	4	G	Not assayed	31.9	0.031%					
		H	Not assayed	30.6	0.029%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
13-555E XC	1	A-1, A-2	B 00 314 754, 755	45.5	0.048%	44.8	1	G & T	1,954	**1.32**
							2	G & T	1,951	**1.20**
		B-1, B-2	B 00 314 756, 757	48.6	0.051%	47.6	1	G & T	2,024	**1.39**
	2	C-1, C-2	Not assayed	42.2	0.044%					
		D-1, D-2	Not assayed	44.0	0.046%					
	3	E-1, E-2	B 00 314 762, 763	35.2	0.037%	34.3	1	G & T	1,985	**1.20**
							2	G & T	2,001	**1.31**
		F-1, F-2	B 00 314 764, 765	35.9	0.038%	34.9	1	G & T	1,946	**1.31**
	4	G	Not assayed	30.2	0.032%					
		H	Not assayed	29.9	0.031%					

555 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
14-555E XC	1	A-1, A-2	B 00 314 772, 773	48.2	0.053%	47.4	1	G & T	1,947	**1.11**
							2	G & T	1,923	**1.12**
		B-1, B-2	B 00 314 774, 775	50.9	0.056%	49.6	1	G & T	1,926	**1.12**
	2	C-1, C-2	Not assayed	32.5	0.036%					
		D-1, D-2	Not assayed	35.4	0.039%					
	3	E	B 00 314 780	31.4	0.034%	30.6	1	G & T	1,935	**1.19**
							2	G & T	1,964	**1.14**
		F	B 00 314 781, 782	33.8	0.037%	32.9	1	G & T	1,960	**1.20**
	4	G	Not assayed	29.1	0.032%					
		H	Not assayed	30.9	0.034%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
15-555E XC	1	A	B 00 314 785	29.7	0.034%	29.3	1	G & T	1,968	**0.97**
							2	G & T	1,953	**1.02**
		B	B 00 314 786	29.3	0.033%	28.8	1	G & T	1,923	**4.93**
	2	C	Not assayed	27.7	0.031%					
		D	Not assayed	28.2	0.032%					
	3	E	B 00 314 789	26.5	0.030%	26.2	1	G & T	1,950	**0.94**
							2	G & T	1,920	**0.99**
		F	B 00 314 790	26.8	0.030%	26.3	1	G & T	1,937	**1.00**
	4	G	Not assayed	25.9	0.029%					
		H	Not assayed	27.2	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
16-555E XC	1	A	B 00 314 794	30.3	0.039%	29.7	1	G & T	1,959	**2.55**
							2	G & T	2,036	**1.61**
		B	B 00 314 795	30.9	0.040%	30.3	1	G & T	1,968	**1.98**
	2	C	Not assayed	25.1	0.032%					
		D	Not assayed	25.0	0.032%					
	3	E	B 00 314 798	22.7	0.029%	22.3	1	G & T	2,008	**1.47**
							2	G & T	1,966	**1.79**
		F	B 00 314 799	26.9	0.035%	26.5	1	G & T	1,938	**1.19**
	4	G	Not assayed	23.3	0.030%					
		H	Not assayed	24.7	0.032%					

555 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
17-555E XC	1	A	B 00 314 903	28.4	0.035%	27.9	1	G & T	1,963	**1.38**
							2	G & T	1,931	**1.47**
		B	B 00 314 904	29.3	0.037%	28.8	1	G & T	2,008	**1.33**
	2	C	Not assayed	27.9	0.035%					
		D	Not assayed	28.0	0.035%					
	3	E	B 00 314 907	25.0	0.031%	25.5	1	G & T	1,949	**1.38**
							2	G & T	2,035	**1.05**
		F	B 00 314 908	25.4	0.032%	24.9	1	G & T	1,995	**1.68**
	4	G	Not assayed	24.4	0.030%					
		H	Not assayed	25.0	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
18-555E XC	1	A	B 00 314 912	30.2	0.036%	29.7	1	G & T	2,038	**1.12**
							2	G & T	2,037	**1.27**
		B	B 00 314 913	30.4	0.036%	29.9	1	G & T	2,018	**1.20**
	2	C	Not assayed	27.1	0.032%					
		D	Not assayed	27.2	0.032%					
	3	E	B 00 314 916	27.7	0.033%	27.3	1	G & T	1,955	**1.33**
							2	G & T	2,007	**1.36**
		F	B 00 314 917	27.7	0.033%	27.3	1	G & T	2,029	**1.21**
	4	G	Not assayed	26.9	0.032%					
		H	Not assayed	27.0	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
19-555E XC	1	A-1, A-2	B 00 314 921. 922	42.2	0.048%	41.2	1	G & T	1,958	**5.10**
							2	G & T	1,914	**4.01**
		B-1, B-2	B 00 314 923, 924	46.1	0.053%	45.0	1	G & T	2,021	**1.63**
	2	C-1, C-2	Not assayed	38.5	0.044%					
		D-1, D-2	Not assayed	38.2	0.044%					
	3	E	B 00 314 929	30.2	0.035%	29.5	1	G & T	1,937	**2.23**
							2	G & T	1,937	**2.16**
		F	B 00 314 930	29.8	0.034%	29.2	1	G & T	1,948	**23.45**
	4	G	Not assayed	29.3	0.034%					
		H	Not assayed	28.6	0.033%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
5-585E XC	1	A-1	B 00 314 091	29.4	0.062%	28.9	1	SGS	2,355	0.21
							2	SGS	2,151	0.25
		A-2	B 00 314 092	32.3		32.1	1	SGS	2,187	0.25
		A-3	B 00 314 093	33.1		32.4	1	SGS	2,000	0.23
		B-1	B 00 314 094	22.0	0.033%	21.6	1	SGS	1,873	0.23
		B-2	B 00 314 095	28.5		28.0	1	SGS	2,083	0.21
							2	SGS	2,098	0.23
		B-3	B 00 314 096	29.4		28.8	1	SGS	2,047	0.22
	2	C	Not assayed	60.7	0.039%					
		D	Not assayed	62.2	0.040%					
	3	E-1, E-2, E-3	B 00 314 129, 136, 142	55.3	0.036%	53.1	1	SGS	2,132	0.28
		F-1, F-2, F-3	B 00 314 130, 137, 143	56.0	0.036%	54.8	1	SGS	2,141	0.65
	4	G	Not assayed	52.2	0.034%					
		H	Not assayed	53.3	0.035%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
6-585E XC	1	A-1,A-2,A-3	B 00 314 153, 155, 173	57.6	0.048%	57.0	1	SGS	1,973	0.47
							2	SGS	1,934	0.47
		B-1, B-2, B-3	B 00 314 154, 156, 172	56.0	0.046%	55.4	1	SGS	2,022	0.47
	2	C-1. C-2	Not assayed	43.8	0.036%					
		D-1, D-2	Not assayed	43.0	0.036%					
	3	E-1, E-2	B 00 314 160, 162	40.6	0.034%	40.1	1	SGS	2,052	0.48
		F-1, F-2, F-3	B 00 314 161, 163	37.4	0.031%	36.9	1	SGS	2,002	0.48
	4	G-1, G-2	Not assayed	38.6	0.032%					
		H-1, H-2	Not assayed	40.0	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
7-585E XC	1	A-1, A-2	B 00 314 182, 183	40.4	0.037%	30.3	1	SGS	1,880	0.71
							2	SGS	1,892	0.58
		B	Not assayed	44.4	0.041%					
	2	C-1, C-2	B 00 314 184, 185	39.2	0.036%	29.4	1	SGS	2,051	0.57
		D	Not assayed	41.5	0.038%					
	3	E-1, E-2	B 00 314 186, 187	37.4	0.034%	27.6	1	SGS	1,940	0.56
		F	Not assayed	36.0	0.033%					
	4	G-1, G-2	B 00 314 188, 189	36.6	0.034%	27.0	1	SGS	1,974	0.54
		H	Not assayed	35.2	0.032%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
8-585E XC	1	A-1, A-2	B 00 314 201, 202	43.7	0.039%	26.0	1	SGS	1,992	**0.63**
		B	Not assayed	45.0	0.040%					
	2	C-1, C-2	Not assayed	40.8	0.036%					
		D	Not assayed	40.7	0.036%					
	3	E-1, E-2	B 00 314 205, 206	37.1	0.033%	27.0	1	SGS	2,023	**0.69**
		F	Not assayed	38.4	0.034%					
	4	G-1, G-2	Not assayed	36.7	0.033%					
		H	Not assayed	37.6	0.034%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
9-585E XC	1	A-1,A-2,A-3	B 00 314 218, 219, 220	57.0	0.059%	28.2	1	SGS	1,957	**1.33**
		B-1, B-2, B-3	B 00 314 221, 222, 223	57.4	0.059%	29.5	1	SGS	1,964	**0.98**
	2	C	Not assayed	37.1	0.038%					
		D	Not assayed	33.0	0.034%					
	3	E-1, E-2	B 00 314 225, 226	33.5	0.034%	28.0	1 2	SGS SGS	2,108 1,986	**1.34** **1.98**
		F-1, F-2	B 00 314 227, 228	32.4	0.033%	31.2	1	SGS	2,024	**0.98**
	4	G	Not assayed	32.4	0.033%					
		H	Not assayed	30.5	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
10-585E XC	1	A-1, A-2	B 00 314 229, 230	39.2	0.032%	29.5	1	SGS	1,867	**1.48**
		B-1, B-2	B 00 314 230, 231	37.0	0.031%	28.5	1	SGS	1,996	**1.26**
	2	C	Not assayed	41.4	0.034%					
		D	Not assayed	38.3	0.032%					
	3	E-1, E-2	B 00 314 233, 234	40.0	0.033%	27.0	1 2	SGS SGS	1,993 2,051	**1.06** **1.05**
		F-1, F-2	B 00 314 235, 236	38.7	0.032%	20.7	1	SGS	2,020	**1.17**
	4	G	Not assayed	37.2	0.031%					
		H	Not assayed	37.6	0.031%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
11-585E XC	1	A-1, A-2	B 00 314 246, 247	41.3	0.035%	30.3	1	SGS	1,891	1.89
		B-1, B-2	B 00 314 249, 250	34.2	0.029%	25.0	1	SGS	1,980	3.59
							2	SGS	1,922	2.38
	2	C	Not assayed	38.2	0.033%					
		D	Not assayed	32.0	0.027%					
	3	E-1, E-2	B 00 314 251, 252	37.8	0.032%	27.8	1	SGS	2,110	3.59
							2	SGS	2,039	2.74
		F-1, F-2	B 00 314 253, 254	34.1	0.029%	25.1	1	SGS	1,930	7.70
	4	G	Not assayed	36.5	0.031%					
		H	Not assayed	32.8	0.028%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
12-585E XC	1	A-1, A-2	B 00 314 268, 269	53.6	0.047%	52.4	1	G & T	2,000	1.92
							2	G & T	1,998	3.14
		B-1, B-2	B 00 314 270, 271	51.2	0.044%	50.1	1	G & T	2,000	1.80
	2	C	Not assayed	49.7	0.043%					
		D	Not assayed	45.46	0.039%					
	3	E-1, E-2	B 00 314 272, 273	40.4	0.035%	39.6	1	G & T	2,000	2.80
							2	G & T	2,000	3.55
		F-1, F-2	B 00 314 274, 275	37.6	0.033%	36.9	1	G & T	1,996	1.77
	4	G	Not assayed	38.3	0.033%					
		H	Not assayed	34.7	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
13-585E XC	1	A-1, A-2	B 00 314 290, 291	44.5	0.038%	44.5	1	G & T	2,000	4.09
							2	G & T	2,000	2.55
		B-1, B-2	B 00 314 292, 293	37.1	0.032%	36.3	1	G & T	1,997	5.60
							2	G & T	1,984	2.47
	2	C-1. C-2	Not assayed	41.0	0.035%					
		D	Not assayed	30.0	0.026%					
	3	E	B 00 314 297	34.1	0.029%	33.5	1	G & T	1,995	2.46
							2	G & T	1,995	2.03
		F	B 00 314 298	36.4	0.031%	35.8	1	G & T	1,999	2.92
							2	G & T	1,942	2.96
	4	G	Not assayed	34.5	0.030%					
		H	Not assayed	35.0	0.030%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
14-585E XC	1	A-1, A-2	B 00 314 313, 314	37.5	0.034%	36.8	1	G & T	1,998	**2.74**
							2	G & T	1,999	**2.15**
		B-1, B-2	B 00 314 315, 316	37.5	0.034%	36.7	1	G & T	2,000	**3.35**
							2	G & T	1,986	**4.81**
	2	C-1, C-2	Not assayed	36.1	0.032%					
		D-1, D-2	Not assayed	37.6	0.034%					
	3	E-1, E-2	B 00 314 321, 322	37.8	0.034%	36.9	1	G & T	2,000	**2.84**
							2	G & T	2,000	**2.21**
		F-1, F-2	B 00 314 323, 324	37.7	0.034%	36.8	1	G & T	2,000	**2.86**
							2	G & T	1,984	**2.87**
	4	G-1, G-2	Not assayed	37.0	0.033%					
		H-1, H-2	Not assayed	35.7	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
15-585E XC	1	A-1, A-2	B 00 314 373, 374	46.5	0.042%	45.3	1	G & T	2,000	**5.35**
							2	G & T	2,000	**12.10**
		B-1, B-2	B 00 314 375, 376	45.8	0.041%	44.6	1	G & T	2,000	**6.28**
							2	G & T	1,990	**2.59**
	2	C-1, C-2	Not assayed	38.8	0.035%					
		D-1, D-2	Not assayed	38.0	0.034%					
	3	E-1, E-2	B 00 314 381, 382	33.3	0.030%	32.6	1	G & T	1,996	**9.47**
							2	G & T	1,999	**3.97**
		F-1, F-2	B 00 314 383, 384	39.1	0.035%	38.3	1	G & T	1,999	**3.10**
							2	G & T	1,977	**7.38**
	4	G-1, G-2	Not assayed	34.5	0.031%					
		H-1, H-2	Not assayed	34.8	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
16-585E XC	1	A-1, A-2	B 00 314 390, 391	38.7	0.032%	38.0	1	G & T	2,000	**2.74**
							2	G & T	2,000	**2.12**
		B-1, B-2	B 00 314 392, 393	37.2	0.030%	36.4	1	G & T	2,000	**2.06**
							2	G & T	1,967	**2.14**
	2	C-1, C-2	Not assayed	37.8	0.031%					
		D-1, D-2	Not assayed	38.6	0.032%					
	3	E-1, E-2	B 00 314 398, 399	38.0	0.031%	37.3	1	G & T	1,995	**1.55**
							2	G & T	2,000	**2.85**
		F-1, F-2	B 00 314 400, 401	36.8	0.030%	36.2	1	G & T	2,000	**1.70**
							2	G & T	1,996	**1.28**
	4	G-1, G-2	Not assayed	35.3	0.029%					
		H-1, H-2	Not assayed	37.9	0.031%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
17-585E XC	1	A-1, A-2	B 00 314 407, 408	41.9	0.035%	41.0	1	G & T	2,000	**2.84**
							2	G & T	2,000	**2.56**
		B-1, B-2	B 00 314 409, 410	40.9	0.034%	42.0	1	G & T	2,000	**3.01**
							2	G & T	1,980	**2.56**
	2	C-1, C-2	Not assayed	56.9	0.047%					
		D-1, D-2	Not assayed	61.5	0.051%					
	3	E-1, E-2	B 00 314 450, 451	39.2	0.032%	38.3	1	G & T	2,000	**3.02**
							2	G & T	1,998	**11.38**
		F-1, F-2	B 00 314 452, 453	38.8	0.032%	37.9	1	G & T	2,000	**2.25**
							2	G & T	1,985	**1.61**
	4	G-1, G-2	Not assayed	35.3	0.029%					
		H-1, H-2	Not assayed	37.9	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
18-585E XC	1	A-1, A-2	B 00 314 459, 460	40.8	0.037%	40.0	1	G & T	1,995	**1.57**
							2	G & T	1,998	**1.89**
		B-1, B-2	B 00 314 461, 462	43.1	0.039%	42.0	1	G & T	2,000	**2.25**
	2	C-1, C-2	Not assayed	51.2	0.046%					
		D-1, D-2	Not assayed	54.0	0.049%					
	3	E-1, E-2	B 00 314 467, 468	34.9	0.032%	34.0	1	G & T	2,000	**1.56**
							2	G & T	1,996	**2.62**
		F-1, F-2	B 00 314 469, 470	36.4	0.033%	35.5	1	G & T	2,000	**1.51**
	4	G-1, G-2	Not assayed	35.9	0.032%					
		H-1, H-2	Not assayed	33.6	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
19-585E XC	1	A-1, A-2, A-3	B 00 314 513, 514, 515	56.6	0.051%	55.4	1	G & T	2,000	**2.56**
							2	G & T	2,000	**2.56**
		B-1, B-2, B-3	B 00 314 516, 517, 518	56.4	0.051%	55.3	1	G & T	2,000	**2.25**
	2	C-1, C-2	Not assayed	33.8	0.031%					
		D-1, D-2	Not assayed	33.3	0.030%					
	3	E	B 00 314 523	32.5	0.029%	31.9	1	G & T	2,000	**1.85**
							2	G & T	2,000	**1.94**
		F	B 00 314 524	34.3	0.031%	33.7	1	G & T	2,000	**1.65**
	4	G	Not assayed	31.8	0.029%					
		H	Not assayed	34.5	0.031%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
20-585E XC	1	A-1, A-2	B 00 314 493, 494	44.8	0.041%	43.9	1	G & T	2,000	**1.61**
							2	G & T	2,000	**1.38**
		B-1, B-2	B 00 314 495, 496	43.6	0.040%	42.8	1	G & T	2,000	**1.30**
	2	C-1, C-2	Not assayed	42.2	0.039%					
		D-1, D-2	Not assayed	48.1	0.044%					
	3	E-1, E-2	B 00 314 501, 502	36.9	0.034%	36.2	1	G & T	2,000	**1.78**
							2	G & T	2,000	**1.33**
		F-1, F-2	B 00 314 503, 504	34.4	0.032%	33.8	1	G & T	2,000	**1.51**
	4	G	Not assayed	34.6	0.032%					
		H	Not assayed	31.3	0.029%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
21-585E XC	1	A-1, A-2	B 00 314 551, 552	45.1	0.043%	44.2	1	G & T	2,000	**2.85**
							2	G & T	2,000	**1.38**
		B-1, B-2	B 00 314 553, 554	39.0	0.037%	38.3	1	G & T	2,000	**2.61**
							2	G & T	1,998	**1.01**
	2	C-1, C-2	Not assayed	31.6	0.030%					
		D-1, D-2	Not assayed	33.9	0.032%					
	3	E	B 00 314 559	30.5	0.029%	30.0	1	G & T	2,000	**7.23**
							2	G & T	2,000	**10.24**
		F	B 00 314 560	33.2	0.031%	32.6	1	G & T	2,000	**2.28**
							2	G & T	1,983	**2.83**
	4	G	Not assayed	31.4	0.030%					
		H	Not assayed	31.5	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
22-585E XC	1	A-1, A-2	B 00 314 595, 596	37.9	0.038%	37.1	1	G & T	2,000	**1.38**
							2	G & T	2,000	**2.01**
		B-1, B-2	B 00 314 597, 598	38.2	0.039%	37.4	1	G & T	2,000	**3.51**
	2	C-1, C-2	Not assayed	52.1	0.053%					
		D-1, D-2	Not assayed	54.9	0.056%					
	3	E-1, E-2	B 00 314 603, 604	38.9	0.040%	38.3	1	G & T	2,000	**1.44**
							2	G & T	2,000	**1.75**
		F-1, F-2	B 00 314 605, 606	43.7	0.044%	42.9	1	G & T	2,000	**1.46**
	4	G-1, G-2	Not assayed	30.1	0.031%					
		H-1, H-2	Not assayed	32.2	0.033%					

585 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
23-585E XC	1	A-1, A-2	B 00 314 612, 613	37.9	0.040%	37.2	1	G & T	2,000	**1.39**
							2	G & T	2,000	**0.78**
		B-1, B-2	B 00 314 614, 615	36.8	0.038%	36.1	1	G & T	2,000	**0.83**
	2	C-1, C-2	Not assayed	37.5	0.039%					
		D-1, D-2	Not assayed	38.7	0.040%					
	3	E	B 00 314 620	31.3	0.033%	30.8	1	G & T	2,000	**1.14**
							2	G & T	2,000	**1.15**
		F-1, F-2	B 00 314 621, 622	34.4	0.036%	33.8	1	G & T	2,000	**1.04**
	4	G	Not assayed	29.9	0.031%					
		H	Not assayed	33.0	0.034%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
24-585E XC	1	A-1, A-2	B 00 314 877, 878	42.1	0.046%	41.2	1	G & T	2,000	**0.40**
							2	G & T	2,000	**0.50**
		B-1, B-2	B 00 314 879, 880	39.1	0.043%	38.2	1	G & T	2,000	**0.43**
	2	C-1, C-2	Not assayed	33.7	0.037%					
		D-1, D-2	Not assayed	35.8	0.039%					
	3	E-1, E-2	B 00 314 885, 886	32.6	0.035%	31.9	1	G & T	2,000	**0.43**
							2	G & T	2,000	**0.55**
		F-1, F-2	B 00 314 887, 888	34.9	0.038%	34.1	1	G & T	2,000	**0.78**
	4	G	Not assayed	31.9	0.035%					
		H	Not assayed	33.8	0.037%					

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
5-615E XC	1	A-1,A-2,A-3	B 00 314 097, 106, 114	78.2	0.050%	75.7	1	SGS	2,059.5	**0.20**
		B-1, B-2, B-3	B 00 314 098, 107, 115	72.0	0.046%	70.7	1	SGS	1,997.2	**0.59**
	2	C	Not assayed	68.7	0.044%					
		D	Not assayed	63.6	0.041%					
	3	E-1, E-2	B 00 314 101, 110	55.8	0.036%	54.9	1	SGS	1,982.1	**0.15**
							2	SGS	2,005.9	**0.21**
		F-1, F-2	B 00 314 102, 111	52.0	0.033%	51.2	1	SGS	1,995.5	**0.16**
	4	G	Not assayed	54.3	0.035%					
		H	Not assayed	47.4	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
6-615E XC	1	A-1, A-2	B 00 314 119, 123	64.1	0.049%	63.0	1	SGS	2,020.3	**0.23**
		B-1, B-2	B 00 314 120, 124	62.7	0.048%	61.6	1	SGS	2,051.8	**0.20**
	2	C	Not assayed	46.5	0.035%					
		D	Not assayed	43.8	0.033%					
	3	E-1, E-2	B 00 314 145, 147	41.0	0.031%	40.3	1	SGS	1,981.8	**0.19**
		F-1, F-2	B 00 314 146, 148	41.6	0.032%	40.9	1	SGS	2,023.0	**0.24**
	4	G	Not assayed	39.4	0.030%					
		H	Not assayed	42.1	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
7-615E XC	1	A-1, A-2, A-3	B 00 314 174, 175, 176	66.4	0.049%	32.5	1	SGS	1,960.9	**2.77**
							2	SGS	1,981.1	**2.04**
		B	Not assayed	74.0	0.055%					
	2	C-1, C-2	B 00 314 177, 178	59.6	0.044%	28.4	1	SGS	1,900.1	**2.24**
							2	SGS	1,946.6	**1.87**
		D	Not assayed	59.0	0.043%					
	3	E-1, E-2	B 00 314 179, 180	57.8	0.043%	28.2	1	SGS	1,953.5	**3.88**
							2	SGS	1,935.3	**2.82**
		F	Not assayed	56.9	0.042%					
	4	G-1	B 00 314 181	23.7	0.033%	23.7	1	SGS	2,038.3	**10.43**
							2	SGS	2,019.5	**10.93**
		G-2	B 00 314 191	21.5		21.0	1	SGS	2,005.5	**1.49**
							2	SGS	2,002.8	**2.05**
		H	Not assayed	43.9	0.032%					

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
8-615E XC	1	A-1, A-2	B 00 314 192, 193	40.1	0.035%	28.8	1	SGS	1,960.1	**1.34**
		B	Not assayed	36.7	0.032%					
	2	C-1, C-2	Not assayed	38.3	0.034%					
		D	Not assayed	35.7	0.031%					
	3	E-1, E-2	B 00 314 196, 197	40.4	0.036%	21.0	1	SGS	2,098.7	**1.20**
		F	Not assayed	35.4	0.031%					
	4	G-1, G-2	Not assayed	39.6	0.035%					
		H	Not assayed	33.6	0.030%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
9-615E XC	1	A-1, A-2	B 00 314 209, 210	45.5	0.040%	30.0	1	SGS	1,957.5	**1.92**
		B	Not assayed	42.9	0.038%					
	2	C-1, C-2	Not assayed	40.4	0.036%					
		D	Not assayed	38.6	0.034%					
	3	E-1, E-2	B 00 314 214, 215	38.8	0.034%	28.2	1	SGS	1,962.1	**1.68**
							2	SGS	1,989.6	**1.88**
		F-1, F-2	B 00 314 216, 217	38.0	0.034%	27.6	1	SGS	1,910.0	**1.39**
	4	G	Not assayed	38.3	0.034%					
		H	Not assayed	37.3	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
10-615E XC	1	A-1, A-2	B 00 314 238, 239	44.5	0.042%	21.6	1	SGS	2,087.3	**10.54**
		B-1, B-2	B 00 314 240, 241	38.5	0.037%	26.9	1	SGS	2,041.3	**9.74**
	2	C	Not assayed	37.3	0.036%					
		D	Not assayed	31.8	0.030%					
	3	E-1, E-2	B 00 314 242, 243	35.5	0.034%	26.2	1	SGS	1,923.9	**2.04**
							2	SGS	1,664.2	**2.31**
		F-1, F-2	B 00 314 244, 245	32.1	0.031%	31.4	1	SGS	2,031.8	**2.91**
	4	G	Not assayed	32.6	0.031%					
		H	Not assayed	32.6	0.031%					

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
11-615E XC	1	A	B 00 314 256	35.6	0.034%	34.8	1	SGS	2,034.8	**3.01**
		B	B 00 314 257	29.7	0.028%	29.1	1	SGS	2,010.0	**1.78**
							2	SGS	1,960.5	**1.58**
	2	C	Not assayed	34.4	0.033%					
		D	Not assayed	30.0	0.028%			`		
	3	E	B 00 314 258	33.3	0.032%	32.7	1	SGS	1,958.4	**1.68**
							2	SGS	1,940.0	**1.50**
		F	B 00 314 259	30.0	0.029%	29.5	1	SGS	1,949.0	**1.79**
							2	SGS	1,981.0	**2.08**
	4	G	Not assayed	33.7	0.032%					
		H	Not assayed	28.6	0.027%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
12-615E XC	1	A-1, A-2	B 00 314 260, 261	40.2	0.036%	29.5	1	SGS	1,953.0	**5.74**
		B	B 00 314 262	33.0	0.030%	32.4	1	SGS	1,983.0	**3.98**
	2	C	Not assayed	38.9	0.035%					
		D	Not assayed	34.0	0.031%					
	3	E-1, E-2	B 00 314 264, 265	39.6	0.036%	29.0	1	SGS	2,002.1	**11.64**
							2	SGS	1,998.0	**7.15**
		F-1, F-2	B 00 314 266, 267	34.1	0.031%	24.8	1	SGS	1,971.0	**3.91**
							2	SGS	1,973.0	**3.61**
	4	G	Not assayed	36.5	0.033%					
		H	Not assayed	35.4	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
13-615E XC	1	A-1, A-2	B 00 314 277, 278	54.6	0.044%	26.8	1	SGS	1,902.0	**40.6**
							2	SGS	1,893.0	**38.3**
		B-1, B-2	B 00 314 279, 280	40.1	0.033%	29.0	1	SGS	1,993.0	**28.6**
							2	SGS	1,970.0	**27.8**
	2	C-1, C-2	B 00 314 286, 287	46.2	0.037%	28.4	1	SGS	2,020.0	**48.7**
							2	SGS	1,995.0	**40.6**
		D-1, D-2	B 00 314 288, 289	38.0	0.031%	27.6	1	SGS	1,933.4	**77.8**
							2	SGS	1,974.0	**60.7**
	3	E-1, E-2	B 00 314 281, 282	46.8	0.038%	27.9	1	SGS	2,032.0	**59.3**
							2	SGS	2,015.8	**62.7**
		F-1, F-2	B 00 314 283, 284	39.5	0.032%	28.8	1	SGS	1,969.0	**62.8**
							2	SGS	2,030.7	**60.7**
	4	G-1, G-2	B 00 314 302, 303	43.1	0.035%	26.1	1	SGS	1,664.3	**35.4**
							2	SGS	2,014.7	**36.1**
		H-1, H-2	B 00 314 304, 305	38.1	0.031%	26.4	1	SGS	1,865.0	**48.4**
							2	SGS	1,910.9	**43.0**

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
14-615E XC	1	A	B 00 314306	34.4	0.031%	33.5	1	SGS	2,009.5	**77.5**
							2	SGS	2,031.6	**80.8**
		B	B 00 314307	35.3	0.032%	34.4	1	SGS	2,061.7	**59.5**
							2	SGS	2,021.2	**64.6**
	2	C-1, C-2	B 00 314 308, 309	37.3	0.034%	36.4	1	SGS	2,032.8	**76.3**
							2	SGS	1,920.1	**77.6**
		D-1, D-2	B 00 314 310, 311	38.4	0.035%	37.5	1	SGS	1,928.9	**82.7**
							2	SGS	1,876.0	**79.3**
	3	E-1, E-2	B 00 314 330, 331	36.9	0.033%	36.1	1	SGS	1,972.3	**115.9**
							2	SGS	1,921.8	**116.6**
		F-1, F-2	B 00 314 332, 333	38.2	0.034%	37.2	1	SGS	2,010.9	**78.5**
							2	SGS	1,997.3	**68.2**
	4	G-1, G-2	B 00 314 334, 335	37.5	0.034%	34.7	1	SGS	2,005.4	**68.1**
							2	SGS	1,900.4	**59.1**
		H-1, H-2	B 00 314 336, 337	38.5	0.035%	37.5	1	SGS	1,920.1	**145.0**
							2	SGS	1,998.1	**116.4**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
15-615E XC	1	A-1, A-2	B 00 314 338, 339	40.7	0.033%	28.1	1	SGS	1,921.0	**183.8**
							2	SGS	2,005.1	**205.6**
		B-1, B-2	B 00 314 340, 341	38.7	0.032%	27.3	1	SGS	1,957.0	**288.1**
							2	SGS	1,978.1	**293.0**
	2	C-1, C-2	B 00 314 342, 343	33.8	0.028%	27.1	1	SGS	2,089.5	**230.1**
							2	SGS	2,058.0	**231.9**
		D-1, D-2	B 00 314 344, 345	39.4	0.032%	28.0	1	SGS	1,977.0	**247.0**
							2	SGS	1,971.4	**225.9**
	3	E-1, E-2	B 00 314 346, 347	38.3	0.031%	28.7	1	SGS	2,045.0	**246.4**
							2	SGS	2,036.0	**244.8**
		F-1, F-2	B 00 314 348, 349	39.4	0.032%	19.9	1	SGS	1,985.0	**168.1**
							2	SGS	2,002.8	**174.0**
	4	G-1, G-2	B 00 314 351, 352	38.2	0.031%	27.6	1	SGS	1,941.0	**190.8**
							2	SGS	1,950.7	**187.4**
		H-1, H-2	B 00 314 353, 354	38.1	0.031%	24.6	1	SGS	1,987.9	**182.4**
							2	SGS	1,983.1	**194.4**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
16-615E XC	1	A-1, A-2	B 00 314 356, 357	39.2	0.031%	27.6	1	SGS	1,945.2	**180.1**
							2	SGS	1,936.9	**152.2**
		B-1, B-2	B 00 314 358, 359	39.4	0.031%	28.8	1	SGS	1,936.9	**146.4**
							2	SGS	1,918.0	**171.7**
	2	C-1, C-2	B 00 314 360, 361	40.4	0.031%	27.1	1	SGS	1,986.4	**134.8**
							2	SGS	1,982.2	**145.2**
		D-1, D-2	B 00 314 362, 363	42.0	0.033%	26.6	1	SGS	1,993.6	**163.4**
							2	SGS	1,995.8	**142.0**
	3	E-1, E-2	B 00 314 364, 365	41.1	0.032%	27.2	1	SGS	2,013.4	**152.0**
							2	SGS	1,976.3	**153.8**
		F-1, F-2	B 00 314 366, 367	41.3	0.032%	28.6	1	SGS	1,962.7	**147.0**
							2	SGS	1,980.0	**144.5**
	4	G-1, G-2	B 00 314 368, 369	39.6	0.031%	20.4	1	SGS	1,903.7	**151.9**
							2	SGS	1,948.4	**149.7**
		H-1, H-2	B 00 314 370, 371	42.2	0.033%	24.3	1	SGS	2,261.1	**137.1**
							2	SGS	1,906.7	**156.9**

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
17-615E XC	1	A-1, A-2	B 00 314 416, 417	44.0	0.036%	27.6	1	SGS	1,880.4	**150.1**
							2	SGS	1,909.9	**133.6**
		B-1, B-2	B 00 314 418, 419	42.1	0.034%	27.1	1	SGS	1,952.4	**165.6**
							2	SGS	1,953.8	**161.6**
	2	C-1, C-2	B 00 314 420, 421	42.1	0.034%	27.1	1	SGS	2,004.4	**162.6**
							2	SGS	1,965.6	**160.2**
		D-1, D-2	B 00 314 422, 423	45.7	0.037%	28.2	1	SGS	1,940.1	**238.4**
							2	SGS	1,948.0	**304.6**
	3	E-1, E-2	B 00 314 424, 425	39.3	0.032%	27.4	1	SGS	2,044.0	**148.7**
							2	SGS	2,088.0	**138.4**
		F-1, F-2	B 00 314 426, 427	41.8	0.034%	27.5	1	SGS	1,917.0	**127.4**
							2	SGS	1,919.8	**119.3**
	4	G-1, G-2	B 00 314 428, 429	36.7	0.030%	28.4	1	SGS	1,987.1	**137.9**
							2	SGS	1,975.5	**152.2**
		H-1, H-2	B 00 314 430, 431	40.0	0.033%	28.0	1	SGS	1,916.8	**176.4**
							2	SGS	1,978.7	**169.4**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
18-615E XC	1	A-1, A-2	B 00 314 433, 434	47.0	0.040%	28.2	1	SGS	1,947.9	**34.8**
							2	SGS	1,946.1	**32.8**
		B-1, B-2	B 00 314 435, 436	47.0	0.040%	29.8	1	SGS	1,885.0	**32.1**
							2	SGS	1,930.0	**34.0**
	2	C-1, C-2	B 00 314 437, 438	50.6	0.043%	28.4	1	SGS	1,986.0	**43.2**
							2	SGS	2,008.0	**42.0**
		D-1, D-2	B 00 314 439, 440	54.4	0.046%	28.3	1	SGS	1,964.3	**27.7**
							2	SGS	1,997.9	**32.9**
	3	E-1, E-2	B 00 314 441, 442	39.7	0.034%	28.3	1	SGS	1,968.2	**30.2**
							2	SGS	1,960.1	**29.3**
		F-1, F-2	B 00 314 443, 444	44.2	0.038%	28.3	1	SGS	2,001.6	**44.0**
							2	SGS	2,023.0	**44.2**
	4	G-1, G-2	B 00 314 445, 446	36.4	0.031%	28.3	1	SGS	1,929.7	**30.6**
							2	SGS	1,994.2	**32.9**
		H-1, H-2	B 00 314 447, 448	37.1	0.032%	27.4	1	SGS	1,944.0	**33.9**
							2	SGS	1,945.0	**31.6**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
19-615E XC	1	A-1, A-2	B 00 314 476, 477	49.6	0.046%	28.4	1	SGS	1,927.3	**83.1**
							2	SGS	1,905.5	**81.2**
		B-1, B-2	B 00 314 478, 479	60.9	0.056%	29.7	1	SGS	1,952.0	**86.2**
							2	SGS	2,033.6	**105.9**
	2	C-1, C-2	B 00 314 480, 481	44.2	0.041%	29.4	1	SGS	1,990.2	**81.2**
							2	SGS	2,050.1	**78.0**
		D-1, D-2	B 00 314 482, 483	56.7	0.052%	28.7	1	SGS	1,996.7	**94.8**
							2	SGS	1,935.2	**98.4**
	3	E-1, E-2	B 00 314 484, 485	44.0	0.040%	29.3	1	SGS	2,002.6	**76.3**
							2	SGS	2,060.8	**63.9**
		F-1, F-2	B 00 314 486, 487	49.5	0.046%	27.2	1	SGS	1,948.3	**113.6**
							2	SGS	1,968.2	**110.3**
	4	G-1, G-2	B 00 314 488, 489	40.6	0.037%	26.6	1	SGS	1,945.8	**86.2**
							2	SGS	2,003.3	**73.6**
		H-1, H-2	B 00 314 490, 491	33.9	0.031%	29.2	1	SGS	2,001.0	**99.9**
							2	SGS	1,989.1	**76.8**

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
20-615E XC	1	A	B 00 314 528	33.4	0.032%	32.9	1	SGS	1,922.6	**16.9**
							2	SGS	1,974.6	**14.4**
		B-1, B-2	B 00 314 529, 530	37.5	0.036%	26.2	1	SGS	1,895.5	**38.8**
							2	SGS	1,932.1	**61.1**
	2	C-1, C-2	B 00 314 531, 532	34.1	0.033%	28.4	1	SGS	2,000.1	**19.9**
							2	SGS	2,008.9	**15.2**
		D-1, D-2	B 00 314 533, 534	35.3	0.034%	25.9	1	SGS	1,982.0	**48.2**
							2	SGS	1,952.7	**44.5**
	3	E-1, E-2	B 00 314 535, 536	32.5	0.031%	25.9	1	SGS	1,952.7	**13.0**
							2	SGS	1,998.2	**12.7**
		F-1, F-2	B 00 314 537, 538	34.7	0.033%	26.1	1	SGS	1,927.6	**11.9**
							2	SGS	1,916.1	**12.8**
	4	G-1, G-2	B 00 314 539, 540	31.2	0.030%	29.6	1	SGS	1,974.1	**24.3**
							2	SGS	1,934.3	**25.2**
		H-1, H-2	B 00 314 541, 542	33.9	0.033%	29.4	1	SGS	2,037.3	**6.2**
							2	SGS	2,029.5	**6.6**
							1	GT & T		**19.9**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
21-615E XC	1	A-1, A-2	B 00 314 508, 509	38.2	0.037%	29.9	1	SGS	1,915.6	**12.9**
							2	SGS	1,994.8	**16.4**
		B-1, B-2	B 00 314 510, 511	32.8	0.032%	29.5	1	SGS	1,975.7	**6.5**
							2	SGS	2,001.0	**8.9**
	2	C	B 00 314 544	30.2	0.030%	29.9	1	SGS	1,967.2	**7.5**
							2	SGS	1,945.2	**6.1**
		D	B 00 314 545	33.4	0.033%	33.0	1	SGS	1,957.2	**5.8**
							2	SGS	1,961.1	
	3	E	B 00 314 546	28.9	0.028%	28.6	1	SGS	2,053.9	**42.0**
							2	SGS	1,984.0	**28.1**
		F	B 00 314 547	31.9	0.031%	31.6	1	SGS	1,891.0	**13.5**
							2	SGS	1,887.8	
	4	G	B 00 314 548	29.1	0.028%	28.7	1	SGS	1,963.0	**10.0**
							2	SGS	1,942.8	
		H	B 00 314 549	31.8	0.031%	31.5	1	SGS	1,957.0	**8.6**
							2	SGS	1,902.6	**11.5**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
22-615E XC	1	A-1, A-2	B 00 314 564, 565	33.1	0.031%	27.6	1	SGS	2,003.0	**9.1**
							2	SGS	2,095.0	
		B-1, B-2	B 00 314 566, 567	37.4	0.035%	29.2	1	SGS	2,011.8	**5.2**
							2	SGS	2,018.5	
	2	C-1, C-2	B 00 314 568, 569	32.2	0.030%	26.8	1	SGS	1,997.3	**6.4**
							2	SGS	2,008.7	
		D-1, D-2	B 00 314 570, 571	36.5	0.034%	26.9	1	SGS	2,087.9	**3.5**
							2	SGS	1,963.0	
	3	E	B 00 314 572	31.4	0.029%	31.0	1	SGS	2,014.1	**7.0**
							2	SGS	2,067.0	**6.5**
		F	B 00 314 573	34.0	0.032%	33.6	1	SGS	1,920.0	**7.5**
							2	SGS	1,945.2	
	4	G-1, G-2	B 00 314 576, 577	30.4	0.028%	27.6	1	SGS	2,024.4	**4.7**
							2	SGS	1,989.0	
		H-1, H-2	B 00 314 578, 579	32.8	0.031%	29.0	1	SGS	2,029.2	**5.6**
							2	SGS	2,025.4	**4.2**

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
23-615E XC	1	A-1, A-2	B 00 314 580, 581	33.9	0.030%	29.0	1	SGS	1,992.2	**4.8**
							2	SGS	2,019.2	**9.0**
		B-1, B-2	B 00 314 582, 583	36.4	0.032%	28.4	1	SGS	2,003.8	**7.4**
							2	SGS	2,021.2	**6.8**
	2	C-1, C-2	B 00 314 584, 585	30.9	0.027%		1	SGS	1,966.4	
							2	SGS	1,952.1	
		D-1, D-2	B 00 314 586, 587	40.3	0.035%		1	SGS	1,982.7	
							2	SGS	2,049.1	
	3	E	B 00 314 588	30.7	0.027%	30.1	1	SGS	2,006.2	**3.2**
							2	SGS	2,003.5	**3.1**
		F-1, F-2	B 00 314 589, 590	38.4	0.034%	27.7	1	SGS	1,930.0	**3.4**
							2	SGS	1,952.7	**3.6**
	4	G	B 00 314 591	33.2	0.029%		1	SGS		
							2	SGS		
		H-1, H-2	B 00 314 592, 593	36.1	0.032%		1	SGS		
							2	SGS		

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
24-615E XC	1	A-1, A-2	B 00 314 625, 626	37.2	0.036%		1	SGS	1,979.4	**1.15**
							2	SGS	2,036.3	**1.04**
		B-1, B-2	B 00 314 627, 628	34.5	0.033%		1	SGS	2,033.4	**1.69**
							2	SGS	2,000.8	**1.98**
	2	C-1, C-2	Not assayed	32.1	0.031%					
		D-1, D-2	Not assayed	34.3	0.033%					
	3	E	B 00 314 633	32.0	0.031%		1	SGS	2,007.0	**2.94**
							2	SGS	2,015.0	**2.74**
		F	B 00 314 634	33.9	0.033%		1	SGS	1,969.0	**1.90**
							2	SGS	1,969.0	**4.36**
	4	G	Not assayed	31.7	0.031%					
		H	Not assayed	34.2	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
25-615E XC	1	A-1, A-2	B 00 314 638, 639	33.0	0.034%		1	SGS	2,009.7	**0.95**
							2	SGS	2,028.3	**1.02**
		B-1, B-2	B 00 314 640, 641	33.4	0.035%		1	SGS	1,971.5	**0.98**
							2	SGS	1,964.1	**1.25**
	2	C	Not assayed	28.5	0.030%					
		D	Not assayed	33.0	0.034%					
	3	E	B 00 314 644	28.8	0.030%		1	SGS	2,014.3	**2.18**
							2	SGS	1,996.1	**1.33**
		F	B 00 314 645	32.6	0.034%		1	SGS	2,040.7	**1.11**
							2	SGS	2,022.9	**0.99**
	4	G	Not assayed	28.8	0.030%					
		H	Not assayed	32.0	0.033%					

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
26-615E XC	1	A	B 00 314 649	33.6	0.033%		1	SGS	1,965.1	**1.34**
							2	SGS	1,991.3	**1.46**
		B	B 00 314 650	34.6	0.034%		1	SGS	2,046.3	**1.60**
							2	SGS	2,034.2	**1.59**
	2	C	Not assayed	30.2	0.030%					
		D	Not assayed	32.9	0.032%					
	3	E	B 00 314 801	28.7	0.028%		1	SGS	1,946.5	**2.05**
							2	SGS	1,975.7	**1.79**
		F	B 00 314 802	32.1	0.032%		1	SGS	1,968.2	**1.59**
							2	SGS	1,960.1	**3.24**
	4	G	Not assayed	29.3	0.029%					
		H	Not assayed	31.6	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
27-615E XC	1	A-1, A-2	B 00 314 806, 807	33.9	0.033%		1	SGS	2,002.7	**2.60**
							2	SGS	1,974.4	**1.89**
		B-1, B-2	B 00 314 808, 809	37.1	0.036%		1	SGS	1,992.4	**1.94**
							2	SGS	2,003.6	**1.74**
	2	C-1, C-2	Not assayed	34.4	0.033%					
		D-1, D-2	Not assayed	34.6	0.034%					
	3	E-1, E-2	B 00 314 814, 815	33.8	0.033%		1	SGS	1,989.3	**2.06**
							2	SGS	2,016.2	**2.15**
		F-1, F-2	B 00 314 816, 817	33.6	0.033%		1	SGS	1,996.1	**1.61**
							2	SGS	2,001.3	**1.85**
	4	G	Not assayed	32.5	0.031%					
		H	Not assayed	32.9	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
28-615E XC	1	A-1, A-2	B 00 314 820, 821	36.5	0.034%		1	SGS	1,994.9	**0.69**
							2	SGS	2,006.0	**0.72**
		B-1, B-2	B 00 314 822, 823	36.2	0.034%		1	SGS	1,997.0	**0.68**
							2	SGS	1,996.5	**0.79**
	2	C	Not assayed	32.4	0.030%					
		D-1, D-2	Not assayed	34.1	0.032%					
	3	E	B 00 314 827	33.0	0.031%		1	SGS	1,918.8	**0.93**
							2	SGS	1,992.2	**0.76**
		F	B 00 314 828	33.2	0.031%		1	SGS	2,089.5	**1.24**
							2	SGS	2,034.1	**1.38**
	4	G	Not assayed	32.5	0.030%					
		H	Not assayed	32.4	0.030%					

615 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
29-615E XC	1	A-1, A-2	B 00 314 832, 833	37.4	0.036%		1	SGS	1,960.6	**0.92**
							2	SGS	2,028.0	**1.13**
		B-1, B-2	B 00 314 834, 835	37.7	0.036%		1	SGS	2,064.5	**2.82**
							2	SGS	2,023.6	**2.61**
	2	C-1, C-2	Not assayed	34.6	0.033%					
		D-1, D-2	Not assayed	38.1	0.037%					
	3	E	B 00 314 840	33.4	0.032%		1	SGS	2,000.7	**1.41**
							2	SGS	1,974.3	**1.23**
		F-1, F-2	B 00 314 841, 842	35.7	0.034%		1	SGS	1,960.1	**1.05**
							2	SGS	2,007.4	**1.23**
	4	G	Not assayed	32.1	0.031%					
		H	Not assayed	33.7	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
30-615E XC	1	A-1, A-2	B 00 314 846, 847	50.7	0.051%		1	SGS		**1.93**
							2	SGS		**1.90**
		B-1, B-2	B 00 314 848, 849	48.8	0.049%		1	SGS		**1.76**
							2	SGS		**3.12**
	2	C-1, C-2	B 00 314 850, 851	37.1	0.038%		1	SGS		**1.84**
							2	SGS		**1.82**
		D	Not assayed	33.9	0.034%					
	3	E-1, E-2	B 00 314 853, 854	35.4	0.036%		1	SGS		**1.84**
							2	SGS		**1.89**
		F	B 00 314 855, 856	33.8	0.034%		1	SGS		**2.02**
							2	SGS		**2.00**
	4	G-1, G-2	Not assayed	34.3	0.035%					
		H	Not assayed	33.8	0.034%					

615 Lateral & Raise

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
1-615L	1	A-1,A-2	B 00 204 783, 784	48.6	0.043%		1	SGS	1,993.7	75.39
							2	SGS	2,006.1	81.43
		B-1, B-2	B 00 204 785, 786	49.9	0.044%		1	SGS	1,949.8	68.82
							2	SGS	1,965.6	62.74
	2	C-1, C-2	B 00 204 788, 789	37.3	0.033%		1	SGS	1,939.7	98.37
							2	SGS	2,000.1	131.37
		D-1, D-2	B 00 204 790, 791	42.1	0.037%		1	SGS	1,992.6	118.22
							2	SGS	1,956.4	97.88
	3	E-1, E-2	B 00 204 792, 793	38.4	0.034%		1	SGS	1,988.9	168.15
							2	SGS	1,988.9	166.83
		F-1, F-2	B 00 204 794, 795	38.8	0.034%		1	SGS	1,967.7	88.41
							2	SGS	1,945.5	123.95
	4	G-1, G-2	B 00 204 796, 797	36.6	0.032%		1	SGS	1,975.9	44.73
							2	SGS	1,922.5	62.16
		H-1, H-2	B 00 204 798, 799	39.3	0.035%		1	SGS	2,001.1	110.55
							2	SGS	2,004.7	106.39

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
1A-615L	1	A	B 00 204 801	15.1	0.046%		1	SGS	1,995.8	10.61
							2	SGS	2,011.9	6.48
		B	B 00 204 802	14.8	0.045%		1	SGS	1,989.6	9.38
							2	SGS	2,008.8	8.02
	2	C	B 00 204 803	15.0	0.046%		1	SGS	1,989.2	11.91
							2	SGS	1,990.4	10.85
		D	B 00 204 804	14.9	0.045%		1	SGS	2,003.5	15.91
							2	SGS	1,999.4	19.48
	3	E	B 00 204 805	13.7	0.042%		1	SGS	1,930.9	8.27
							2	SGS	2,001.1	7.11
		F	B 00 204 806	13.8	0.042%		1	SGS	1,930.9	13.54
							2	SGS	1,953.7	15.78
	4	G	B 00 204 807	12.8	0.039%		1	SGS	2,001.3	10.57
							2	SGS	2,002.7	9.99
		H	B 00 204 808	13.2	0.040%		1	SGS	2,113.4	6.79
							2	SGS	2,118.3	6.17

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
2-615L	1	A	B 00 204 810	33.9	0.043%		1	SGS	1,924.0	2.52
							2	SGS	1,982.7	1.81
		B-1, B-2	B 00 204 811, 812	35.7	0.045%		1	SGS	1,981.7	2.56
							2	SGS	1,996.8	1.98
	2	C	B 00 204 813	32.6	0.041%		1	SGS	1,914.8	2.38
							2	SGS	2,034.5	2.34
		D-1, D-2	B 00 204 814, 815	36.1	0.046%		1	SGS	2,037.6	8.34
							2	SGS	1,997.8	2.54
	3	E	B 00 204 816	31.3	0.040%		1	SGS	1,998.7	5.24
							2	SGS	1,999.6	3.47
		F-1, F-2	B 00 204 817, 818	36.4	0.046%		1	SGS	2,001.6	5.32
							2	SGS	1,998.4	3.43
	4	G	B 00 204 819	29.1	0.037%		1	SGS	1,997.6	2.09
							2	SGS	2,008.2	1.99
		H-1, H-2	B 00 204 820, 821	36.7	0.046%		1	SGS	2,005.0	2.49
							2	SGS	2,006.0	2.37

615 Lateral & Raise

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
3-615L	1	A-1,A-2	B 00 204 823, 824	57.9	0.043%		1	SGS	1,975.0	3.48
							2	SGS	1,949.0	3.25
		B-1, B-2	B 00 204 825, 826	61.8	0.046%		1	SGS	1,952.0	3.30
							2	SGS	1,992.1	5.79
	2	C-1, C-2	B 00 204 827, 828	43.5	0.032%		1	SGS	2,016.2	3.54
							2	SGS	1,998.8	2.68
		D-1, D-2	B 00 204 829, 830	51.3	0.038%		1	SGS	1,987.0	3.94
							2	SGS	1,984.3	3.86
	3	E-1, E-2	B 00 204 832, 833	44.5	0.033%		1	SGS	2,001.7	5.02
							2	SGS	2,033.2	12.46
		F-1, F-2	B 00 204 834, 835	43.4	0.032%		1	SGS	2,002.6	4.64
							2	SGS	1,990.4	3.81
	4	G-1, G-2	B 00 204 836, 837	41.1	0.030%		1	SGS	1,991.2	5.37
							2	SGS	2,029.5	3.68
		H-1, H-2	B 00 204 838, 839	46.5	0.034%		1	SGS	1,944.2	13.52
							2	SGS	2,045.0	10.38

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
4-615L	1	A-1,A-2	B 00 204 840, 841	34.8	0.039%		1	SGS	1,996.4	23.99
							2	SGS	1,959.2	24.04
		B-1, B-2	B 00 204 842, 843	36.2	0.040%		1	SGS	2,019.7	9.97
							2	SGS	2,059.3	8.39
	2	C	B 00 204 844	30.9	0.034%		1	SGS	2,012.0	16.41
							2	SGS	1,994.3	10.29
		D	B 00 204 845	28.8	0.032%		1	SGS	1,941.7	8.82
							2	SGS	2,022.5	8.16
	3	E	B 00 204 846	27.9	0.031%		1	SGS	1,933.9	16.78
							2	SGS	1,959.1	11.18
		F	B 00 204 847	29.4	0.033%		1	SGS	1,976.5	11.18
							2	SGS	1,999.7	9.46
	4	G	B 00 204 848	37.2	0.041%		1	SGS	1,914.1	14.89
							2	SGS	1,934.3	18.74
		H	B 00 204 849	29.7	0.033%		1	SGS	1,908.5	12.71
							2	SGS	1,979.9	12.18

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
5-615L	1	A	B 00 204 850	31.9	0.034%		1	SGS	2,083.2	22.02
							2	SGS	2,116.9	19.68
		B-1, B-2	B 00 204 851, 852	35.9	0.038%		1	SGS	1,999.6	38.28
							2	SGS	2,025.6	37.54
	2	C	B 00 204 853	28.1	0.030%		1	SGS	2,014.9	20.40
							2	SGS	2,041.1	18.77
		D	B 00 204 854	32.8	0.035%		1	SGS	2,028.9	15.10
							2	SGS	2,036.2	15.10
	3	E	B 00 204 856	27.2	0.029%		1	SGS	1,957.2	52.55
							2	SGS	2,008.5	47.59
		F	B 00 204 857	32.7	0.034%		1	SGS	1,980.3	22.16
							2	SGS	1,985.7	27.00
	4	G	B 00 204 858	27.9	0.029%		1	SGS	1,947.2	23.40
							2	SGS	1,935.1	342.53
		H	B 00 204 859	30.6	0.032%		1	SGS	1,931.7	17.09
							2	SGS	1,997.6	17.09

615 Lateral & Raise

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
6-615L	1	A-1,A-2	B 00 204 860, 861	40.0	0.044%		1	SGS	2,000.3	120.56
							2	SGS	1,996.9	100.38
		B-1, B-2	B 00 204 862, 863	42.4	0.047%		1	SGS	2,003.0	113.10
							2	SGS	2,001.4	147.66
	2	C	B 00 204 864	31.1	0.034%		1	SGS	1,998.0	83.36
							2	SGS	2,002.7	108.44
		D	B 00 204 865	33.6	0.037%		1	SGS	1,999.1	115.55
							2	SGS	1,996.5	85.28
	3	E	B 00 204 866	30.9	0.034%		1	SGS	2,006.1	159.45
							2	SGS	2,031.5	138.53
		F	B 00 204 867	32.4	0.036%		1	SGS	1,993.1	123.66
							2	SGS	2,008.2	125.26
	4	G	B 00 204 868	29.7	0.033%		1	SGS	2,022.8	112.83
							2	SGS	2,031.2	105.65
		H	B 00 204 869	30.6	0.034%		1	SGS	2,037.3	132.11
							2	SGS	1,991.8	135.49

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
7-615L	1	A-1,A-2	B 00 204 870, 871	37.2	0.036%		1	SGS	2,000.2	46.91
							2	SGS	2,007.4	43.56
		B-1, B-2	B 00 204 872, 873	39.5	0.038%		1	SGS	2,000.0	57.69
							2	SGS	1,997.2	54.39
	2	C	B 00 204 893	32.6	0.031%		1	SGS	2,037.7	75.39
							2	SGS	1,991.1	65.53
		D	B 00 204 875	33.2	0.032%		1	SGS	2,008.4	54.80
							2	SGS	2,012.1	58.27
	3	E	B 00 204 876	31.4	0.030%		1	SGS	2,019.7	35.75
							2	SGS	2,029.0	38.84
		F	B 00 204 877	34.2	0.033%		1	SGS	2,040.2	38.59
							2	SGS	1,962.1	49.38
	4	G	B 00 204 878	30.9	0.030%		1	SGS	2,055.4	78.31
							2	SGS	2,057.7	90.64
		H	B 00 204 879	34.4	0.033%		1	SGS	2,017.3	42.87
							2	SGS	2,017.0	38.88

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
8-615L	1	A-1,A-2	B 00 204 880, 881	39.3	0.041%		1	SGS	2,011.5	8.54
							2	SGS	1,964.4	7.37
		B-1, B-2	B 00 204 882, 883	41.1	0.043%		1	SGS	1,998.1	8.01
							2	SGS	2,000.1	7.36
	2	C-1, C-2	B 00 204 884, 885	35.2	0.037%		1	SGS	1,992.4	5.55
							2	SGS	1,989.1	5.51
		D-1, D-2	B 00 204 886, 887	33.8	0.035%		1	SGS	2,008.6	11.17
							2	SGS	2,006.1	9.52
	3	E	B 00 204 888	32.6	0.034%		1	SGS	2,004.9	8.18
							2	SGS	2,001.7	6.66
		F	B 00 204 889	33.0	0.034%		1	SGS	2,003.9	9.70
							2	SGS	2,046.0	9.91
	4	G	B 00 204 890	31.0	0.032%		1	SGS	2,009.4	23.59
							2	SGS	2,025.4	11.92
		H	B 00 204 891	30.5	0.032%		1	SGS	1,966.5	18.82
							2	SGS	1,979.7	14.84

615 Lateral & Raise

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
9-615L	1	A	B 00 314 937	32.7	0.032%		1	SGS	2,038.4	7.33
							2	SGS	2,027.3	6.41
		B-1, B-2	B 00 314 938, 939	35.1	0.034%		1	SGS	1,989.9	3.89
							2	SGS	1,974.9	4.85
	2	C	B 00 314 940	30.8	0.030%		1	SGS	2,002.6	5.56
							2	SGS	1,995.5	6.43
		D	B 00 314 941	34.6	0.034%		1	SGS	2,000.0	5.96
							2	SGS	2,000.4	7.88
	3	E	B 00 314 942	31.4	0.031%		1	SGS	1,995.9	8.86
							2	SGS	1,996.4	10.67
		F	B 00 314 943	33.7	0.033%		1	SGS	2,036.1	7.21
							2	SGS	1,986.0	7.44
	4	G	B 00 314 944	31.0	0.030%		1	SGS	1,991.0	8.63
							2	SGS	1,962.5	6.71
		H	B 00 314 945	32.9	0.032%		1	SGS	1,960.8	4.50
							2	SGS	1,963.2	3.82

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
10-615L	1	A-1, A-2	B 00 314 947, 948	42.8	0.043%		1	SGS	1,996.8	2.90
							2	SGS	1,999.7	2.78
		B-1, B-2	B 00 314 949, 950	40.7	0.041%		1	SGS	2,006.7	3.60
							2	SGS	1,999.2	2.99
	2	C-1, C-2	B 00 206 001, 002	36.9	0.037%		1	SGS	2,014.5	4.73
							2	SGS	1,981.0	8.56
		D-1, D-2	B 00 206 003, 004	36.4	0.036%		1	SGS	1,964.8	1.98
							2	SGS	1,985.6	1.87
	3	E-1, E-2	B 00 206 005, 006	34.2	0.034%		1	SGS	1,980.9	2.70
							2	SGS	1,977.4	2.96
		F-1, F-2	B 00 206 007, 008	35.3	0.035%		1	SGS	2,001.4	3.04
							2	SGS	1,998.4	2.45
	4	G	B 00 206 009	31.3	0.031%		1	SGS	2,018.9	3.09
							2	SGS	2,000.1	2.54
		H	B 00 206 010	34.3	0.034%		1	SGS	1,998.5	6.42
							2	SGS	1,997.2	5.65

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
11-615L	1	A-1, A-2	B 00 206 011, 012	38.1	0.036%		1	SGS	1,998.0	**13.10**
		A-3, A-4	B 00 206 016, 017	43.4			1	SGS	1,994.0	**21.30**
		B-1, B-2	B 00 206 013, 014	42.2	0.037%		1	SGS	1,968.4	**5.61**
		B-3, B-4	B 00 206 018, 019	41.5			1	SGS	2,005.1	**23.23**
	2	C-1, C-2	B 00 206 020, 021	46.7	0.036%		1	SGS	2,005.6	**10.61**
		C-3, C-4	B 00 206 022, 023	34.5			1	SGS	1,983.4	**14.00**
		D-1, D-2	B 00 206 024, 025	44.8	0.035%		1	SGS	2,055.0	**9.33**
		D-3, D-4	B 00 206 026, 027	34.2			1	SGS	2,044.0	**5.56**
	3	E-1	B 00 206 029	22.3	0.033%		1	SGS	2,007.4	**5.21**
		E-2	B 00 206 030	24.4			1	SGS	1,999.8	**16.16**
		E-3	B 00 206 031	26.3			1	SGS	2,002.3	**7.37**
		F-1	B 00 206 032	23.3	0.035%		1	SGS	2,019.7	**9.07**
		F-2	B 00 206 033	26.0			1	SGS	2,028.1	**5.44**
		F-3	B 00 206 034	29.9			1	SGS	2,051.5	**18.47**
	4	G-1	B 00 206 035	23.2	0.030%		1	SGS	1,978.3	**5.07**
		G-2	B 00 206 037	23.9			1	SGS	1,979.7	**11.07**
		G-3	B 00 206 038	20.0			1	SGS	1,995.1	**5.95**
		H-1	B 00 206 036	26.9	0.036%		1	SGS	1,989.7	**7.21**
		H-2	B 00 206 039	29.4			1	SGS	1,996.9	**7.46**
		H-3	B 00 206 040	23.9			1	SGS	2,003.7	**11.05**

615 Lateral & Raise

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
#1SE-615L Slash	1	A	B 00 204 894	15.9	0.042%		1	SGS	2,002.2	**7.95**
							2	SGS	2,009.1	**10.43**
		B	B 00 204 895	14.4	0.038%		1	SGS	2,004.6	**8.90**
							2	SGS	2,001.9	**9.70**
	2	C	B 00 204 896	13.5	0.036%		1	SGS	2,003.0	**11.00**
							2	SGS	1,993.8	**10.80**
		D	B 00 204 897	12.6	0.033%		1	SGS	2,000.0	**12.17**
							2	SGS	2,006.4	**12.58**
	3	E	B 00 204 898	14.3	0.038%		1	SGS	1,992.6	**9.75**
							2	SGS	1,999.6	**8.99**
		F	B 00 314 934	13.7	0.037%		1	SGS	2,006.2	**9.36**
							2	SGS	2,026.1	**8.31**
	4	G	B 00 314 935	13.8	0.037%		1	SGS	2,009.9	**16.25**
							2	SGS	2,008.5	**13.51**
		H	B 00 314 936	14.4	0.038%		1	SGS	1,980.6	**51.88**
							2	SGS	1,965.2	**64.75**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
#2SE-615L Slash-A	1	A-1,A-2	B 00 206 041, 042	41.4	0.038%		1	SGS	1,998.6	**20.84**
							2	SGS	1,997.8	**21.85**
		B-1, B-2	B 00 206 043, 044	41.7	0.039%		1	SGS	2,000.7	**25.94**
							2	SGS	2,005.6	**22.24**
	2	C-1, C-2	B 00 206 045, 046	34.7	0.032%		1	SGS	2,006.5	**22.17**
							2	SGS	1,994.3	**18.17**
		D-1, D-2	B 00 206 047, 048	38.6	0.036%		1	SGS	2,014.1	**20.90**
							2	SGS	2,007.8	**17.43**
	3	E-1, E-2	B 00 206 049, 050	35.1	0.033%		1	SGS	2,057.8	**19.44**
							2	SGS	2,030.4	**16.19**
		F-1, F-2	B 00 206 051, 052	36.2	0.034%		1	SGS	2,049.2	**8.91**
							2	SGS	1,953.0	**9.33**
	4	G	B 00 206 053	30.9	0.029%		1	SGS	2,015.6	**14.00**
							2	SGS	2,026.4	**22.78**
		H-1, H-2	B 00 206 054, 055	38.7	0.036%		1	SGS	2,024.8	**20.52**
							2	SGS	2,030.1	**11.83**

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Cut	Assay Lab	Starting Mass (g)	Calculated Head (Au - g/t)
#2SE-615L Slash-B	1	A	B 00 206 057	22.3	0.032%		1	SGS	1,966.0	**12.48**
							2	SGS	1,947.7	**11.62**
		B	B 00 206 058	25.9	0.038%		1	SGS	1,998.3	**15.49**
							2	SGS	2,014.6	**19.34**
	2	C	B 00 206 059	30.4	0.044%		1	SGS	2,003.7	**21.60**
							2	SGS	2,006.2	**21.40**
		D	B 00 206 060	31.2	0.045%		1	SGS	1,999.9	**19.41**
							2	SGS	2,004.6	**15.52**
	3	E	B 00 206 061	25.6	0.037%		1	SGS	1,993.1	**28.59**
							2	SGS	1,999.5	**22.97**
		F	B 00 206 062	28.0	0.041%		1	SGS	1,978.1	**13.35**
							2	SGS	2,004.8	**12.21**
	4	G	B 00 206 063	27.2	0.040%		1	SGS	1,995.3	**13.76**
							2	SGS	2,010.6	**13.00**
		H	B 00 206 064	27.4	0.040%		1	SGS	1,946.0	**18.70**
							2	SGS	2,002.3	**18.21**

645 E Cross-Cut South

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
5-645E XC	1	A	B 00 314 653	29.6	0.030%	29.1	1	G & T		**0.73**
							2	G & T		**0.64**
		B	B 00 314 654	33.7	0.034%	33.0	1	G & T		**0.52**
	2	C	Not assayed	31.0	0.031%					
		D	Not assayed	34.1	0.034%					
	3	E	B 00 314 657	31.6	0.032%	31.1	1	G & T		**2.05**
							2	G & T		**2.82**
		F	B 00 314 658	31.7	0.032%	31.3	1	G & T		**0.53**
	4	G	Not assayed	30.7	0.031%					
		H	Not assayed	31.7	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
6-645E XC	1	A-1, A-2	B 00 314 662, 663	40.3	0.044%	39.5	1	G & T		**2.10**
							2	G & T		**9.08**
		B-1, B-2	B 00 314 664, 665	44.8	0.049%	43.7	1	G & T		**1.55**
	2	C	Not assayed	32.8	0.036%					
		D	Not assayed	33.1	0.036%					
	3	E	B 00 314 668	30.5	0.033%	29.8	1	G & T		**0.69**
							2	G & T		**0.66**
		F	B 00 314 669	33.1	0.036%	32.3	1	G & T		**0.51**
	4	G	Not assayed	30.9	0.034%					
		H	Not assayed	30.7	0.034%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
7-645E XC	1	A-1, A-2, A-3	B 00 314 672, 673, 674	69.5	0.069%	39.5	1	G & T	2,000	**0.25**
							2	G & T	2,000	**0.11**
		B-1, B-2	B 00 314 675, 676	44.5	0.044%	43.7	1	G & T	2,000	**0.17**
	2	C-1, C-2	Not assayed	41.1	0.041%					
		D-1, D-2	Not assayed	40.6	0.040%					
	3	E	B 00 314 681	34.8	0.035%	34.2	1	G & T	2,000	**0.14**
							2	G & T	2,000	**0.08**
		F	B 00 314 682	35.3	0.035%	34.7	1	G & T	2,000	**0.16**
	4	G	Not assayed	33.2	0.033%					
		H	Not assayed	35.2	0.035%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
8-645E XC	1	A-1, A-2	B 00 314 685, 686	37.8	0.039%	37.0	1	G & T	2,000	**0.04**
							2	G & T	2,000	**0.04**
		B-1, B-2	B 00 314 687, 688	35.0	0.036%	34.3	1	G & T	2,000	**0.03**
	2	C	Not assayed	32.6	0.034%					
		D	Not assayed	31.2	0.032%					
	3	E	B 00 314 691	29.6	0.031%	29.1	1	G & T	2,000	**0.05**
							2	G & T	2,000	**0.05**
		F	B 00 314 692	30.0	0.031%	29.4	1	G & T	2,000	**0.02**
	4	G	Not assayed	28.3	0.029%					
		H	Not assayed	30.1	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
9-645E XC	1	A	B 00 314 696	33.7	0.035%	32.2	1	G & T	2,000	**0.07**
							2	G & T	2,000	**0.05**
		B-1, B-2	B 00 314 697, 698	34.8	0.036%	34.0	1	G & T	1,999	**0.05**
	2	C	Not assayed	32.8	0.034%					
		D-1, D-2	Not assayed	37.2	0.039%					
	3	E	B 00 314 702	32.0	0.033%	31.2	1	G & T	1,996	**0.07**
							2	G & T	2,000	**0.07**
		F	B 00 314 703	33.1	0.034%	32.3	1	G & T	2,000	**0.09**
	4	G	Not assayed	30.8	0.032%					
		H	Not assayed	31.8	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
10-645E XC	1	A-1, A-2	B 00 314 707, 708	52.4	0.052%	52.0	1	G & T	2,000	**0.04**
							2	G & T	1,997	**0.07**
		B-1, B-2	B 00 314 709, 710	52.8	0.052%	51.5	1	G & T	1,994	**0.32**
	2	C-1, C-2	Not assayed	44.6	0.044%					
		D-1, D-2	Not assayed	43.4	0.043%					
	3	E-1, E-2	B 00 314 715, 716	36.7	0.036%	35.8	1	G & T	2,000	**0.09**
							2	G & T	2,000	**0.06**
		F-1, F-2	B 00 314 717, 718	37.8	0.037%	36.9	1	G & T	2,000	**0.09**
	4	G	Not assayed	33.3	0.033%					
		H-1, H-2	Not assayed	36.8	0.036%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
11-645E XC	1	A	B 00 314 723	29.9	0.031%	30.0	1	G & T	2,000	**0.53**
							2	G & T	2,000	**0.30**
		B	B 00 314 724	32.3	0.033%	31.7	1	G & T	1,996	**1.08**
	2	C	Not assayed	27.6	0.028%					
		D	Not assayed	31.5	0.032%					
	3	E	B 00 314 727	27.4	0.028%	26.9	1	G & T	2,000	**0.30**
							2	G & T	1,996	**0.34**
		F	B 00 314 728	32.0	0.033%	31.4	1	G & T	1,999	**0.39**
	4	G	Not assayed	29.1	0.030%					
		H	Not assayed	29.9	0.031%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
12-645E XC	1	A-1, A-2	B 00 314 732, 733	33.1	0.035%	32.4	1	G & T	2,000	**0.68**
							2	G & T	2,000	**0.67**
		B-1, B-2	B 00 314 734, 735	33.8	0.036%	32.9	1	G & T	2,000	**0.51**
	2	C	Not assayed	25.6	0.027%					
		D	Not assayed	27.7	0.030%					
	3	E	B 00 314 736	21.2	0.023%	20.9	1	G & T	2,000	**0.45**
							2	G & T	1,999	**0.56**
		F	B 00 314 737	23.2	0.025%	22.9	1	G & T	2,000	**0.58**
	4	G	Not assayed	20.4	0.022%					
		H	Not assayed	22.4	0.024%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
13-645E XC	1	A-1, A-2	B 00 314 743, 744	37.7	0.036%	37.2	1	G & T	2,000	**0.91**
							2	G & T	1,946	**0.97**
		B-1, B-2	B 00 314 745, 746	36.5	0.035%	36.0	1	G & T	1,953	**1.49**
	2	C-1, C-2	Not assayed	35.2	0.034%					
		D-1, D-2	Not assayed	35.5	0.034%					
	3	E	B 00 314 951	34.3	0.033%	33.8	1	G & T	1,992	**1.11**
							2	G & T	2,000	**1.16**
		F	B 00 314 952	33.4	0.032%	33.0	1	G & T	2,000	**1.13**
	4	G	Not assayed	31.8	0.030%					
		H	Not assayed	32.4	0.031%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
14-645E XC	1	A-1, A-2	B 00 314 956, 957	33.6	0.034%	33.1	1	G & T	1,985	**2.31**
							2	G & T	2,000	**4.37**
		B-1, B-2	B 00 314 958, 959	34.4	0.035%	33.9	1	G & T	2,000	**4.69**
	2	C	Not assayed	33.4	0.034%					
		D	Not assayed	33.3	0.034%					
	3	E	B 00 314 962	33.0	0.033%	32.4	1	G & T	1,993	**4.80**
							2	G & T	1,988	**4.42**
		F	B 00 314 963	31.7	0.032%	31.2	1	G & T	1,978	**4.22**
	4	G	Not assayed	32.1	0.032%					
		H	Not assayed	31.3	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
15-645E XC	1	A-1, A-2	B 00 314 967, 968	38.5	0.037%	37.6	1	G & T	2,000	**6.72**
							2	G & T	2,000	**10.81**
		B-1, B-2	B 00 314 969, 970	39.8	0.039%	38.8	1	G & T	1,999	**14.25**
	2	C-1, C-2	Not assayed	39.9	0.039%					
		D-1, D-2	Not assayed	41.1	0.040%					
	3	E-1, E-2	B 00 314 975, 976	35.5	0.034%	34.7	1	G & T	1,994	**9.52**
							2	G & T	1,996	**11.99**
		F-1, F-2	B 00 314 977, 978	35.6	0.035%	34.8	1	G & T	1,996	**7.81**
	4	G	Not assayed	33.2	0.032%					
		H	Not assayed	34.0	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
16-645E XC	1	A-1, A-2	B 00 314 982, 983	35.3	0.033%	34.5	1	G & T	1,994	**6.7**
							2	G & T	1,966	**3.8**
		B-1, B-2	B 00 314 984, 985	35.1	0.033%	34.8	1	G & T	1,986	**23.0**
	2	C	Not assayed	34.1	0.032%					
		D-1, D-2	Not assayed	37.5	0.035%					
	3	E	B 00 314 989	35.8	0.034%	35.2	1	G & T	1,971	**8.9**
							2	G & T	1,981	**6.7**
		F	B 00 314 990	34.3	0.032%	33.7	1	G & T	1,998	**4.4**
	4	G	Not assayed	34.5	0.033%					
		H	Not assayed	34.5	0.033%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
17-645E XC	1	A-1, A-2	B 00 314 993, 994	48.5	0.047%	47.5	1	G & T		**0.17**
							2	G & T		**0.19**
		B-1, B-2	B 00 314 995, 996	50.0	0.048%	49.0	1	G & T		**0.15**
	2	C-1, C-2	Not assayed	36.8	0.036%					
		D-1, D-2	Not assayed	34.7	0.034%					
	3	E	B 00 204 501	33.1	0.032%	32.5	1	G & T		**0.16**
							2	G & T		**0.14**
		F	B 00 204 502	33.8	0.033%	33.1	1	G & T		**0.15**
	4	G	Not assayed	32.1	0.031%					
		H	Not assayed	33.1	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
18-645E XC	1	A-1, A-2	B 00 204 506, 507	39.7	0.037%	38.8	1	G & T		**0.24**
							2	G & T		**0.17**
		B-1, B-2	B 00 204 508, 509	42.9	0.040%	41.9	1	G & T		**0.14**
	2	C-1, C-2	Not assayed	36.5	0.034%					
		D-1, D-2	Not assayed	36.7	0.034%					
	3	E-1, E-2	B 00 204 514, 515	35.6	0.033%	32.8	1	G & T		**0.17**
							2	G & T		**0.19**
		F-1, F-2	B 00 204 516, 517	35.3	0.033%	32.6	1	G & T		**0.24**
	4	G-1, G-2	Not assayed	34.3	0.032%					
		H-1, H-2	Not assayed	39.0	0.036%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
19-645E XC	1	A-1, A-2	B 00 204 523, 524	51.5	0.047%	50.4	1	G & T		**0.71**
							2	G & T		**0.54**
		B-1, B-2	B 00 204 525, 526	49.1	0.045%	48.0	1	G & T		**0.48**
	2	C-1, C-2	Not assayed	42.6	0.039%					
		D-1, D-2	Not assayed	40.1	0.037%					
	3	E-1, E-2	B 00 204 531, 532	36.9	0.034%	36.1	1	G & T		**0.80**
							2	G & T		**0.50**
		F-1, F-2	B 00 204 533, 534	36.5	0.033%	35.7	1	G & T		**0.25**
	4	G-1, G-2	Not assayed	35.2	0.032%					
		H-1, H-2	Not assayed	34.4	0.031%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
20-645E XC	1	A-1, A-2	B 00 204 540, 544	43.3	0.039%	42.4	1	G & T		**0.81**
							2	G & T		**0.48**
		B-1, B-2	B 00 204 542, 543	44.6	0.041%	43.8	1	G & T		**0.19**
	2	C-1, C-2	Not assayed	37.0	0.034%					
		D-1, D-2	Not assayed	35.8	0.032%					
	3	E-1, E-2	B 00 204 549, 550	36.5	0.033%	35.9	1	G & T		**0.19**
							2	G & T		**0.20**
		F-1, F-2	B 00 204 551, 552	37.6	0.034%	37.0	1	G & T		**0.35**
	4	G-1, G-2	Not assayed	35.5	0.032%					
		H-1, H-2	Not assayed	35.7	0.032%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
21-645E XC	1	A-1, A-2	B 00 204 558, 559	41.5	0.039%	40.0	1	G & T		**1.11**
							2	G & T		**0.85**
		B-1, B-2	B 00 204 560, 561	39.7	0.037%	38.3	1	G & T		**0.53**
	2	C-1, C-2	Not assayed	39.6	0.037%					
		D-1, D-2	Not assayed	39.1	0.036%					
	3	E-1, E-2	B 00 204 566, 567	36.3	0.034%	35.5	1	G & T		**1.25**
							2	G & T		**1.00**
		F-1, F-2	B 00 204 568, 569	35.7	0.033%	35.0	1	G & T		**0.56**
	4	G-1, G-2	Not assayed	34.2	0.032%					
		H-1, H-2	Not assayed	35.8	0.033%					

Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
22-645E XC	1	A-1, A-2	B 00 204 574, 575	37.2	0.037%	36.3	1	G & T		**0.69**
							2	G & T		**1.16**
		B-1, B-2	B 00 204 576, 577	35.7	0.036%	34.9	1	G & T		**0.42**
	2	C	Not assayed	33.3	0.033%					
		D-1, D-2	Not assayed	34.7	0.035%					
	3	E-1, E-2	B 00 204 582, 583	34.7	0.034%	33.9	1	G & T		**1.15**
							2	G & T		**0.70**
		F	B 00 204 584	33.2	0.033%	32.5	1	G & T		**0.41**
	4	G	Not assayed	31.7	0.032%					
		H	Not assayed	32.9	0.033%					

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Round #	Tower Sample Run	Pail	Sample Designation	Mass Shipped (kg)	Sample Ratio	Mass Milled (dry - kg)	Sample Ratio	Mass Milled (dry - kg)	Starting Mass (g)	Calculated Head (Au - g/t)
23-645E XC	1	A-1, A-2	B 00 204 587, 588	35.0	0.033%	34.2	1	G & T		**0.21**
							2	G & T		**0.31**
		B-1, B-2	B 00 204 589, 590	36.2	0.034%	35.3	1	G & T		**0.20**
	2	C-1, C-2	Not assayed	34.2	0.032%					
		D-1, D-2	Not assayed	34.3	0.032%					
	3	E	B 00 204 595	33.6	0.032%	31.5	1	G & T		**1.05**
							2	G & T		**0.49**
		F-1, F-2	B 00 204 596	32.5	0.031%	31.7	1	G & T		**0.19**
	4	G-1, G-2	Not assayed	32.6	0.031%					
		H-1, H-2	Not assayed	32.1	0.030%					