RL 4/2005 - RIVER LEA

ANNUAL REPORT TO 7th AUGUST, 2010

Compiled by: Peter McNeil, M.Sc.

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Frontier Resources Ltd

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Summary

Exploration completed on the Retention License during the year consisted of:

- An independent consultant estimated an Inferred Resource for the Stormont Deposit using Surpac software.
- Subsequently, the Narrawa - Stormont Conceptual Mining Study was updated showing the potential for a positive cash flow.
- Limited petrological work.
- Metallurgical testwork.

Table 1. Inferred Mineral Resource Summary Showing Grade Estimates at Various Gold Cut-off Grades, using Ordinary Kriging

Cut-off Gold Grade (g/t)	Contained Gold (ounces)	Tonnes	Gold (g/t)	Bismuth (%)	Silver (g/t)
0.5	14,585	124,300	3.65	0.26	3.35
1	14,250	112,500	3.94	0.27	3.41
1.5	13,430	91,400	4.57	0.30	3.52
2	12,525	75,500	5.16	0.32	3.32
2.5	11,625	63,200	5.72	0.34	3.38
3	10,880	54,400	6.22	0.35	3.39
3.5	10,500	50,800	6.43	0.36	3.34

The Stormont Deposit is located 6.5km west of Frontier's Narrawa precious - base metal skarn-style stratiform deposit located in the core and on the limbs of a shallowly southeasterly plunging syncline (at its northwestern end). The deposit is located on or very near surface and ranges in stratigraphic thickness between 10m and 15m.

A consistently mineralised resource was modelled in a 150m long NW part of the central syncline, referred to as the high grade zone.

An Inferred Resource was estimated for the Stormont Deposit (table 1) , based on all drilling and trenching to date (in accordance with the 2004 JORC code).

Significant high grade gold+/-bismuth intersections have been demonstrated over the entire 300m known length of the central syncline, with drillholes SD8, SD10, SD33 and SD44, returning up to 4m of 12.7 g/t gold (see figure 11), NOT included in the resource.

There is good scope to increase the resource in the SE of the central syncline, the untested western sector of the western syncline and proximal to the eastern thrust The Conceptual Mining Study is now being updated, utilising long term projected metal prices and it will recommend possible development paths forward.

The Revised Conceptual Mining Study relating to possible mining and processing of the gold and base metal mineralisation at the Narrawa and Stormont Deposits has demonstrated the potential for a positive theoretical cash flow.

The Conceptual Mining Study shows a satisfactory theoretical cash flow from processing the mineralisation at the Narrawa and Stormont Deposits, based on a capital expenditure estimated at A$8 million (neglecting working capital and provision for contingencies).

The CMS is not a feasibility study, but a <u>detailed</u> evaluation designed to determine if there are economic reasons for pursuing and further advancing a project that is known to contain certain types and grades of mineralisation.

The Conceptual Mining Study demonstrates that the continuation of Feasibility Studies is strongly warranted to evaluate the ultimate economic potential of the Narrawa and Stormont Deposits and to move them toward future production and cash flow for Frontier.

Figure 1. Stormont gold + bismuth high grade resource and geology)

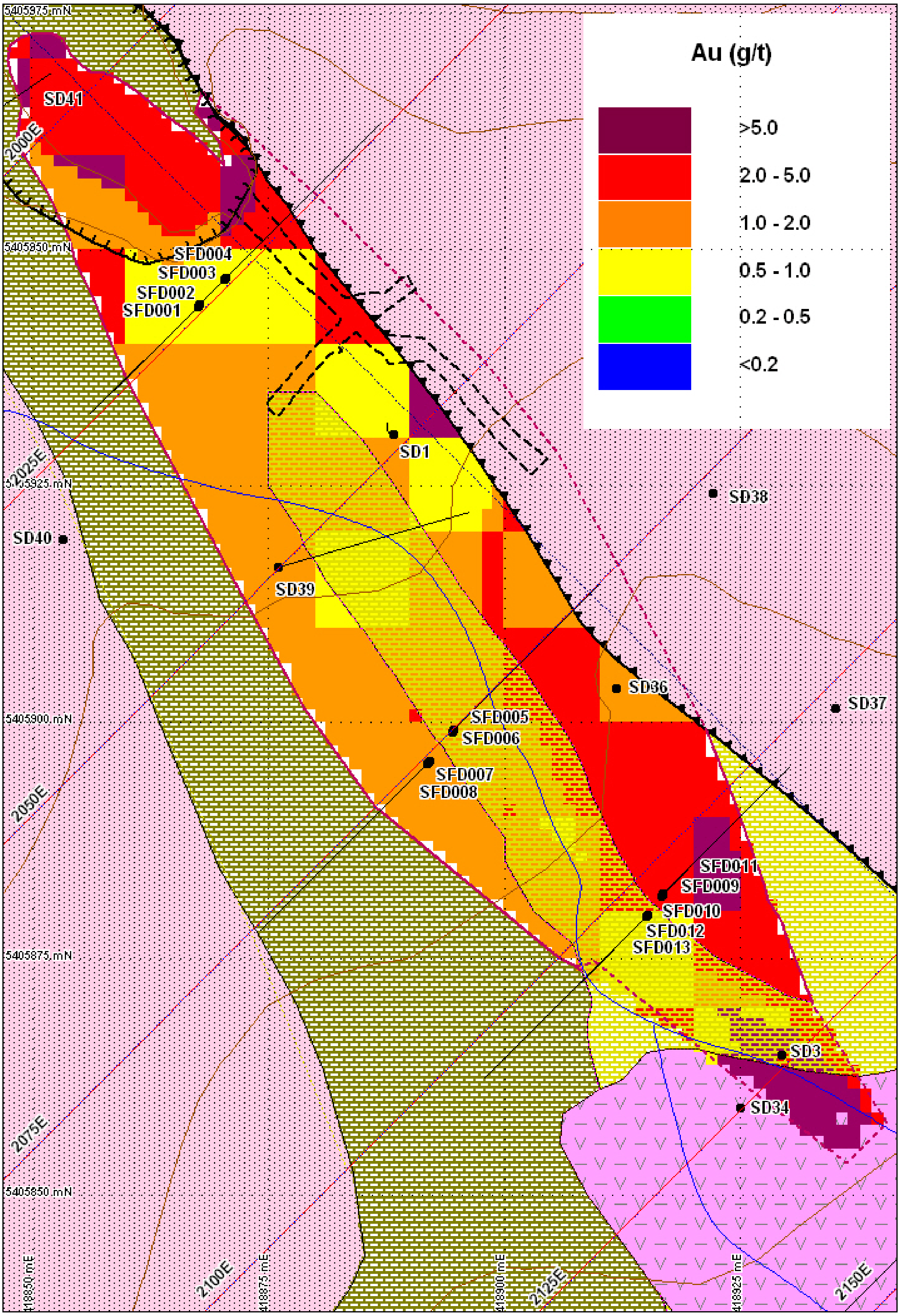


Figure 2. Section 2075E looking to 325 degrees AMG (legend as per figure 1)

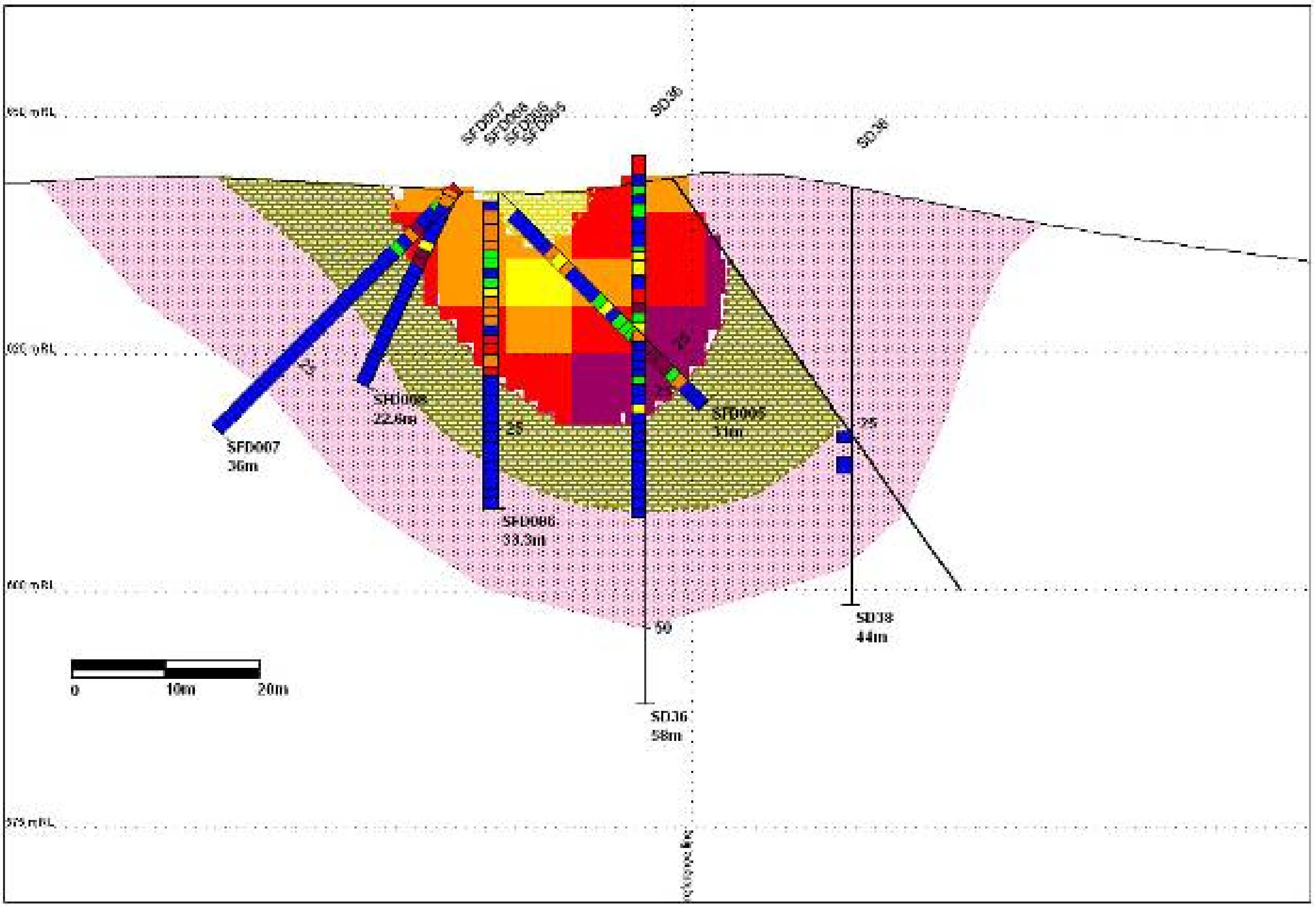
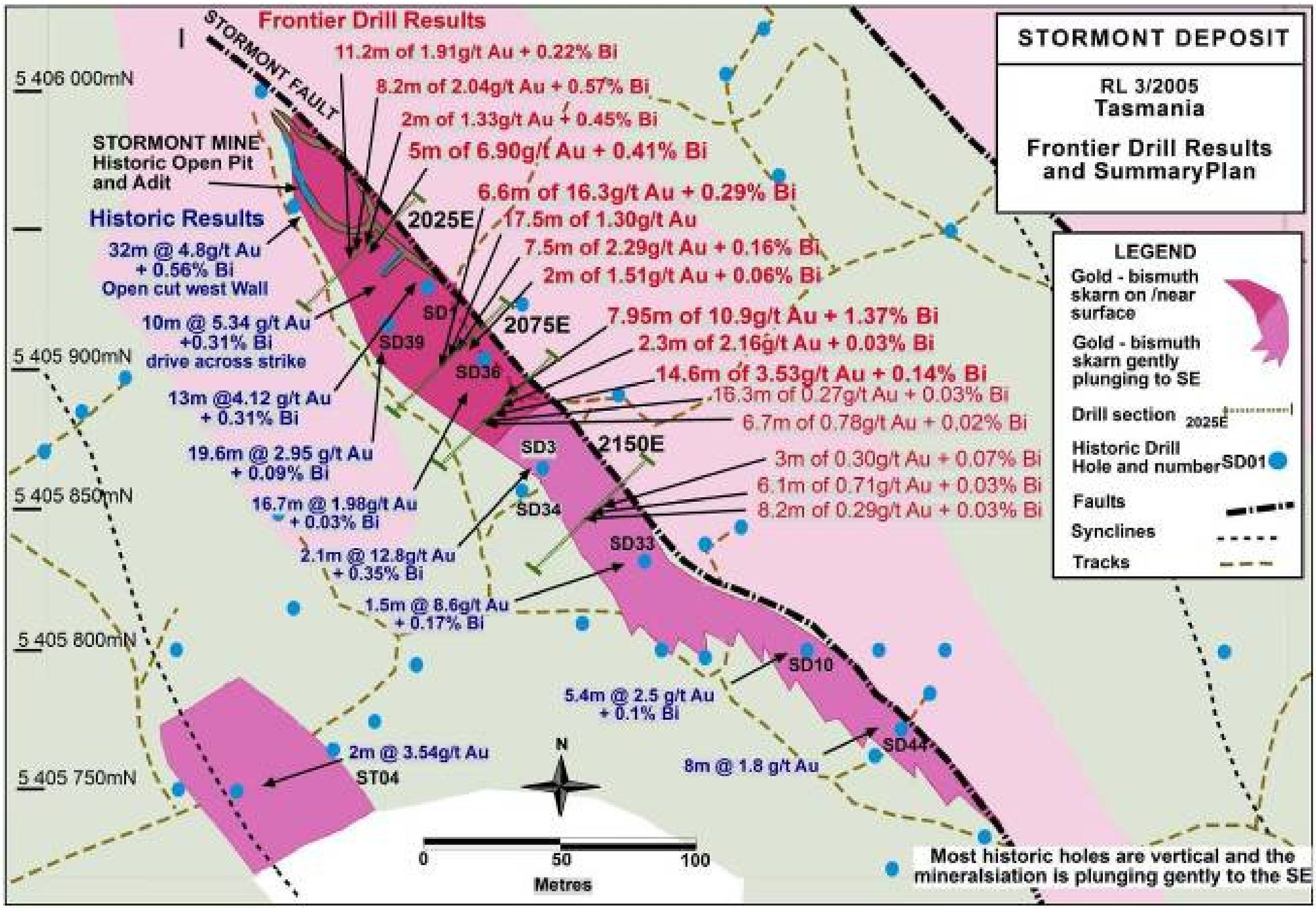


Figure 3. Drill holes locations and weighted average assay results



The CMS was based on the current Indicated and Inferred Resource for the Narrawa Deposit and the current Inferred Resource for the Stormont Deposit. The total resource at Narrawa contains 23,550 ounces of gold equivalent grading 3.5 g/t gold equivalent and consists of 14,125 ounces of gold, plus 131,300 ounces of silver, 2,765 tonnes of lead and 2,335 tonnes of zinc, at a 0.5g/t gold equivalent cut-off grade. The mineralisation is contained within 209,330 tonnes of rock grading 2.10 g/t gold, 19.5 g/t silver, 1.32% lead and 1.12% zinc. The Indicated Resource at Narrawa consists of 162,755 tonnes grading 3.61 g/t gold equivalent, consisting of 2.11 g/t gold, 20.5 g/t silver, 1.42% lead and 1.2% zinc.

The Inferred Resource for the 'high grade' zone at Stormont contains 13,430 ounces gold, 27.7 tonnes bismuth and 10,340 ounces silver, within 91,400 tonnes of mineralised rock grading 4.57g/t gold, 0.30% bismuth and 3.52g/t silver, at a 1.5g/t gold cut-off grade. Not all known mineralisation was included in the Stormont Inferred Resource due to the limited drill holes.

Theoretical cash flows for the project were estimated for Narrawa then Stormont, vice versa and then both combined for simultaneous processing utilising extractive technology which has become available in Australia. The evaluation included order of magnitude estimates of capital, operating costs, personnel costs and logistical requirements.

The CMS determined cash flows for only the Indicated portion of the Narrawa Resource, the results of which Frontier could then report to the ASX and shareholders. This is because those figures would not be very meaningful in the present context as Frontier are not able to report the possible cash flows from the Inferred Resource at Stormont. Cash flows for the combined operation were determined and are encouraging, however, under ASX guidelines, cash flow figures and financial evaluations can only be published in relation to Indicated or Measured Resources, not Inferred Resources. As such, Frontier note that the results from the Narrawa/Stormont CMS should be regarded with appropriate caution.

The Conceptual Mining Study was undertaken by Mr H.D.Swain, Mining Engineer and Director of Frontier Resources Ltd, with more than 40 years professional experience in many different types of deposits. Mr Swain noted in the conclusions to his report:

> *The philosophy of mining and processing at the Narrawa and Stormont Deposits is to adopt a simple approach utilising local workforce and contractors to mine and process the low tonnage Mineral Resources.*
>
> *The Conceptual Mining Study demonstrates the potential of a satisfactory investment, which will yield a future source of income to the shareholders of Frontier Resources Ltd.*

Extraction of gold from the Stormont mineralised material would potentially be by the Carbon in Pulp (CIP) process, utilising the 'future' Narrawa plant/infrastructure to minimise capital costs. Metallurgical testwork was received on the Stormont mineralisation in mid 2009 (Amdel Laboratories -Perth) and it returned high recoveries of 92% for gold from gravity separation (and leaching of the gravity separation products) with normal CIP processing. The overall CIP gold recoveries where shown to improve with the fineness of the grind, as did recovery of gold into the gravity concentrate (which ranged from 22-29%). Bond Work Index was 14kWhr/t.

Mineralised material from the Narrawa Deposit would potentially be mined and concentrated onsite, then smelted at Hobart's Risdon refinery. The metallurgical testwork concentrate for Narrawa showed high recoveries for each metal, low reagent consumption, low Bond Work Index and production of a high quality concentrate. As a result, low Operating Costs are anticipated. Metallurgical testwork was conducted in 2008 on the Narrawa mineralisation (Amdel Laboratories - Perth) and it returned very high recoveries of 96.7% for gold, 98.5% for zinc, 95.6% for lead and 92.4% for silver, indicating non-refractory gold plus zinc, lead, copper and silver mineralisation.

Bond Work Index was 14kWhr/t.

Simple mining practices and low waste to ore ratios are anticipated at both deposits, given the orientation of the mineralised zones relative to local topography. The Narrawa Deposit is anticipated to have a good stripping ratio of 1.0 tonne of waste to 1.0 tonne of ore and Stormont should have an excellent stripping ratio of 0.5 tonnes of waste to 1.0 tonne of ore.

Cash operating costs are estimated to be $46/tonne for Narrawa and $55/tonne for Stormont. Non-labour operating costs are estimated to be A$10.97/tonne for Narrawa and A$14.19/tonne for Stormont. These costs could be markedly reduced for a larger scale simultaneous operation and/or if alternate metallurgical processes live are successful.

Due to the complex polymetallic nature of the mineralised material at Narrawa Deposit, toll smelting and refining is likely to be relatively expensive but due to the unusually high quality of the concentrate, charges and losses are anticipated to be low at say about 5% of the value of contained metal (compared to the normal charges approaching 7% to 10% of the value of the metal). The charge would be deducted from gross revenue as a royalty by the smelter company.

The Conceptual Mining Study has indicated that there could be a theoretical pre-tax profit after the return of capital expenditure costs for an operation based solely on the Narrawa Deposit, but the possible profit improves significantly with an increase in total available resources from a combined operation and would improve further from a larger operation. This finding rationalises why Frontier will undertake additional resource expansion and infill definition drilling at the Narrawa and Stormont Deposits when possible.

The establishment of an 'extractive processing centre' in the region of Retention Licenses 3/2005 (Narrawa) and 4/2005 (Stormont) would allow other known gold and polymetallic mineralisation to be targeted for conversion to resources by additional drilling and for possible subsequent exploitation.

The Stormont resource is wholly classified as Inferred and was included in the CMS to evaluate the robustness of the overall project. The 2004 JORC Code states "Caution should be exercised if this category (Inferred) is considered in technical and economic studies". Metals prices utilised in the CMS were from 3/7/2009, being US$940/oz gold, US$0.71.44/lb zinc, US$0.7738/lb lead, US$13.70/oz silver.

LOCATION/ACCESS

Please refer to Section 1.4 of the Inferred Resource Report located in Appendix 1.

TENURE AND LAND USAGE

Please refer to Section 1.5 of the Inferred Resource Report located in Appendix 1.

EXPLORATION AND MINING HISTORY

Please refer to Section 2.1. of the Inferred Resource Report located in Appendix 1.

RECENT EXPLORATION

Please refer to Section 2. 1.2 of the Inferred Resource Report located in Appendix 1.

PREVIOUS RESOURCE ESTIMATES

Please refer to Section 1.4 of the Inferred Resource Report located in Appendix 1.

GEOLOGICAL SETTING

Please refer to Section 3.0 of the Inferred Resource Report located in Appendix 1.

PROJECT GEOLOGY

Please refer to Section 3.2 of the Inferred Resource Report located in Appendix 1.

RESOURCE GEOLOGY

Please refer to Section 3.3 of the Inferred Resource Report located in Appendix 1.

WORK CONDUCTED

Exploration completed on the Retention License during the year consisted of:

- An independent consultant estimated an Inferred Resource for the Stormont Deposit using Surpac software (See Appendix 1).
- Subsequently, the Narrawa - Stormont Conceptual Mining Study was updated showing the potential for a positive cash flow (See Appendix 2).
- Limited petrological work was completed (See Appendix 3).
- Metallurgical testwork was undertaken as documented in Appendix 4. The results of the gravity separation and leaching of the gravity separation products show overall gold recoveries ranging from 80-85% with the highest recovery at the finest grind.
    - Recovery of gold into the gravity concentrate ranged from 22-29% with recovery increasing with fineness of grind.
    - The leach residues of the gravity tailings leach tests were high ranging from 1.85 to 1.4g/t reducing with finer of grind.
    - The gravity tailings leaching curves indicate that the gold is present in two phases i.e. a non-refractory fast leaching phase with 80-90% of the leachable gold recovered in the first two hours of the test and a slow leaching component with the remaining gold recovered in the following 46 hours. The results indicate that this slow leaching component is still leaching at 48 hours.

RECOMMENDATIONS

Mining Lease applications covering the deposits, possible satellite mineralised areas and the required plant and tailings areas, should be initiated and submitted to Mineral Resources Tasmania as the next phase toward possibly developing either a self mined/treated, self mined/toll milled or other type of extractive development.

Feasibility studies are suggested to continue with:

- Expansion drilling to increase the total resources and thus further improve the overall project economics
- Infill drilling to improve the classification or confidence of the resources associated with each deposit, so reserves can be estimated, fiscal outcomes published and development capital ultimately be raised
- Additional metallurgical testwork to maximise metal recoveries and minimise operating costs and
- Environmental and other evaluations.

APPENDIX 1.

Frontier Resources Ltd
Stormont Gold + Bismuth Project
Resource Estimate
April 2009

Grant MacDonald
105 Rowella Road,
Sidmouth, 7270
Tasmania

Disclaimer

Executive Summary

Resource

An inferred resource (using a 1.5g/t Au cut-off grade) of 91,400t @ 4.57g/t Au, 0.30% Bi and 3.52g/t Au for 13,430 oz Au, 27.7t Bi and 10,340 oz Ag has been estimated for the high grade zone at Stormont.

Geological model

- The high grade Au + Bi resource at Stormont is a skarn-style stratiform deposit hosted near the base of the Ordovician Gordon Group Limestone. The deposit which ranges in stratigraphic thickness between 10m and 15m (lying 8m to 15m from the base of the limestone) is located in the core and on the limbs of a shallowly southeasterly plunging syncline at its northwestern most end. The high grade resource outcrops for the most part with only the central most part of the southern half and the southernmost end of the resource covered by unskarnified limestone and Tertiary cover. A section of the northeastern edge of the high grade zone has been faulted off by the Stormont Thrust Fault (with subsequent erosion removing mineralised skarn from the up faulted northeastern block). The southeastern end of the high grade zone is the only gradational boundary and is defined by high grade mineralisation becoming less continuous.
- The high grade zone is approximately 150m long and 30m wide on the surface as its widest point.
- Any further structural control on high grade mineralisation is unclear and the high grade zone has been modelled as a stratiform body.

Geostats

- There is a significant discrepancy between the Au assays of Frontier's supplied certified gold standard (Geostat G905-6). The certified value for the standard by fire assay is 5.96 g/t (standard deviation 0.26) yet 26 assays of the standard by Burnie Research Laboratories consistently assayed 10% below this averaging 5.34 g/t with a standard deviation of 0.02. This discrepancy raises into question all of Frontier's drill core and channel sample Au assays.
- There is only a moderate correlation between Au and Bi in the high grade zone with a correlation coefficient of 0.39. For this reason the resource should be seen as a primarily gold deposit with Bi (and Ag) credits.
- High grade outliers of Au, Bi and Ag have been dealt with by top cutting to 25g/t Au, 15,000 ppm Bi and 22g/t Ag.
- Variography, using a correlogram on 1m composites, showed similar ranges for each of Au, Bi and Ag. The variogram model shows a low nugget effect of 25%. The variogram shows a strike of 135º in the z plane, 0º in the x plane and 0º in the y plane with a short range structure with a range of 28m and an overall range of 55m.

Block model

- A block with parent cell size 10m (North) x 10m (East) x 5m (RL) with sub-blocks of 1.25m x 1.25m x 1.25m was constructed.
- Grades were estimated into blocks using ordinary kriging. 3 passes were used with a minimum of 12 samples and maximum of 30 samples in each pass. The first pass had a search ellipse of 30m x 8m x 8m, the second 60m x 16m x 16m and the third 120m x 32m x 32m.
- An overall average bulk density of 2.9 g/cm3 calculated from 84 ore samples measured by the water immersion method was used in determining tonnages.

Recommendations

- A prime recommendation is to address the apparently consistently inaccurate assay results for Frontier's supplied gold standard included in batches of drill core and channel samples assayed at Burnie Research Laboratories. There is a very strong possibility that all such drill core and channel sample assays may be undercalled by 10.5%

in which case the gold grade of the resource would be elevated by the order of 7% to 9%. This work should be done regardless of whether any other recommendations are accepted and actioned.

To increase the status of the resource estimate from inferred to indicated the following work is required.

- Drilling density needs to be increased to 12.5m sections using current fan geometry with additional angled holes collared northeast of the thrust and drilled southwest to intersect the high grade zone on its northeast limb.
- Surface trenching should be carried out in all areas of outcrop again ideally on 12.5m sections.
- Further channel sampling should be carried out in the old workings to achieve a greater density of sampling. Both walls of drive should be sampled and both horizontally and vertically, ideally achieving a 2m x 2m mesh across all exposures.
- Drillhole collars and channel sample locations have been surveyed by a range of relatively inaccurate means and require conventional surveying.
- The surface DTM has a large component which has been determined by GPS surveying and needs to be surveyed by conventional means.
- Field duplicates of existing half core from previous drilling programmes should be obtained from Mineral Resources Tasmania's core store, probably as ¼ core, and assayed.
- Drillhole and channel assays have seen very limited field duplicates and a low percentage of in-house laboratory duplicates.
- All core should be re-logged and surface exposures should be mapped with a foci on (1) mineral assemblage control and Au and Bi mineralisation and (2) geological structure in order to understand the structural control on mineralisation and for geotechnical purposes.

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1.0 Introduction

1.1 Scope of work

Grant MacDonald (consultant) was commissioned by Frontier Resources Ltd (Frontier) to 3D model and geostatistically calculate (using SURPAC software) a resource estimate of the Stormont gold + bismuth deposit in March/April 2009 using the results of Frontier's recently completed (2008) diamond drilling and channel sampling programme in addition to the existing drilling and channel sampling results from previous exploration.

1.2 Participants

The geostatistical modelling was carried out by Grant MacDonald using data supplied by Rob Reid of Frontier Resources Ltd as well as data obtained independently from Mineral Resources Tasmania's online digital report library. The 3D model of the deposit was constructed in part from 2D sectional interpretations supplied by Rob Reid, in part from 3D point data modelled directly off the drillhole database by Grant MacDonald.

1.3 Principal sources of information

The principal sources of information were twofold.

(1) Rob Reid of Frontier Resources Ltd supplied geological interpretation, coded drill logs, drillhole and channel sampling data (collars and surveys) and assays in comma delimited files.

(2) Hard copies of previous reporting including geological interpretations, descriptive drill logs, drillhole and channel sampling data and assays were downloaded from Mineral Resources Tasmania's website.

1.4 Project location and access

The Stormont gold + bismuth deposit is located in Tasmania's central north approximately 40km south-southwest of Devonport (which lies on the states north coast). Access to the deposit is via sealed road for the most part with approximately 3 kilometres of well formed gravel road (2WD) and 3 kilometres of all-weather 4WD gravel road making up the last section.

The deposit lies at an elevation of between 600m and 700m A.S.L on the northwestern flank of Stormont (1007m A.S.L) in the headwaters of the Lea River. The deposit area and surrounds are covered by rainforest. Rainfall ranges from 1500 to 2000 mm p.a. and light winter snow is not uncommon.

1.5 Tenure and land usage

The deposit lies within RL 4/2005 held by Frontier Resources Ltd.

The land on and around the deposit is classified as State Forest and whilst subject to forestry activities is available for mining. It is understood that an archaeological study has been conducted with no evidence of aboriginal habitation. Perhaps more significantly is the heritage value of the old workings themselves. The northern end of the high grade zone lies in and around the old open cut and underground workings. Any mining activity would necessitate the removal of these. The fact that these old workings date from the late 1920's is likely to downgrade any heritage concerns.

2.0 Project background

Frontier Resources Ltd originally held the Stormont prospect as part of its EL 29/03 "Gowrie Park" granted in 2003. Frontier relinquished all but the Narrawa Creek and Stormont in 2005 converting these to retention licences.

2.1 Exploration and mining history

2.1.1 Discovery and early production

The Stormont Au + Bi deposit was originally discovered in 1925 by the prospector Richard Mages (McKintosh-Reid 1927). (It is important here to distinguish the Stormont Au + Bi mine from the similarly named Stormont Au mine which lies ~700m to the south-southwest of the Stormont Au + Bi deposit. This geographically and geologically distinct gold mine has been inadvertently confused with the Stormont Au + Bi mine in a number of earlier reports on exploration at Stormont).

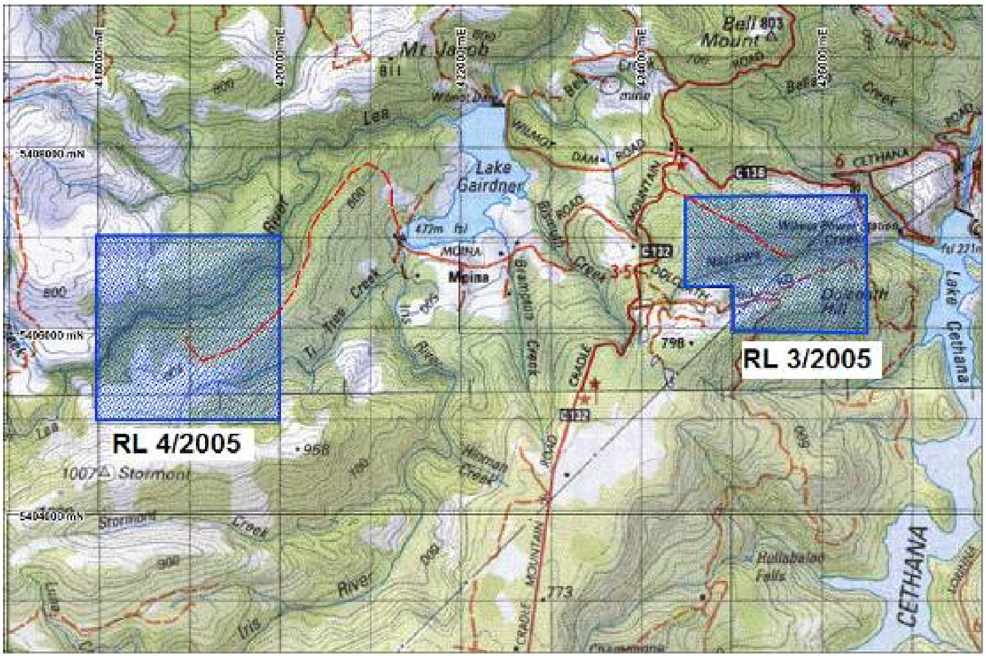

Figure 1.1: RL 4/2005, holding the Stormont resource and RL 3/2005, holding the Narrawa Creek resource (graticules are 1km)

At the Stormont Au + Bi deposit trenching and the mining of an adit commenced in 1927 (Scott, 1929) with the production of a bismuth + gold concentrate commencing in 1928 (Burns, 1959). Production records were supplied informally by P.Doyle to Burns (1959) and listed in the appendix to his report. They indicate the production of 6.33 tonnes of concentrate averaging 435 g/t Au for a total of 86 oz Au and an average of 62.4 % Bi for 3.96 tonnes Bi between October 1928 and April 1934. These figures differ somewhat from those detailed in Keid (1943) as 59 oz Au and 4.35 tonnes Bi but he states that production only occurred between 1930 and 1933 and so the figures of Burns are considered more reliable. Certainly at the time of Keid's visit in 1934 the mine lay dormant.

In early 1934 government geologist E. Broadhurst's visited the (then active) mine, however, his report (Broadhurst, 1934) focussed more on the nature and genesis of the Bi and Au mineralisation (with the significant conclusion that better Bi and Au grades are associated with fractures within the skarn) but provides no details regarding any recent or ongoing mining or production.

The principal (producing) old workings consist of an open cut, ~40m long ~25m wide and up to ~10m high, which extends into the northwest nose of the ridge. At the southeastern end (deepest) of the open cut an adit extends into the hill in a southeasterly direction for ~42m with short cross-cuts 16m in to the southwest (12m long) and northeast (6m long). The total volume of material removed from these workings combined is ~4000 m3 though clearly from the scale of the production figures not all of this material was treated as ore.

In 1929 government geologist J.B. Scott (Scott, 1929) visited and described the then current operation of the Stormont Au + Bi mine (~1 tonne/hour operation) and the problems of the gravity method underpinning the ore separation which Scott concluded was weakened by the partially oxidised nature of the ore. Scott (1929) describes gold as being associated with native bismuth as well as hosted within bismuthinite with the other ore minerals consisting of bismuthite (carbonate of bismuth) and bismite (oxide of bismuth). Scott concludes that whilst gravitational separation was not efficient in the partly oxidised ore it should improve in unoxidised material but further recommends that oil flotation be investigated.

Figure 2.1: Photograph looking southeasterly into the thickly overgrown old open cut (photograph supplied by Frontier)

Figure 2.2: Photograph inside adit at point where cross-cuts commence to right and left – inside high grade zone (photograph supplied by Frontier)

2.1.2 Recent exploration

Introduction

The following history of recent exploration around the Stormont Au + Bi prospect is focussed on those aspects which have direct relevance to the resource itself, i.e. drilling, channel sampling, surveying and petrology.

Comalco (1979) – channel sampling

Roberts (1987) says Comalco (Askins, 1979) collected 2 grab rock samples from open cut which assayed >2 g/t Au and also channelled (2m samples) a length of the open cut and 10m's into adit but only 3 of these channel samples were apparently assayed with the samples 3124W, 3144W and 3164W assaying 1.95, 2.1 and 5.1g/t Au respectively. These samples have not been included in the resource estimation though the tenor of their reported values does not contradict subsequent sampling results.

Goldfields (1987) – channel sampling

Gold Fields Exploration Pty. Limited (GFEL) held the ground containing the Stormont Au + Bi deposit from 1983 to 1991. In 1987 (Roberts, 1987) GFEL systematically channel sampled the southern wall of the open cut and most of the walls and some the backs of the underground workings. Samples were taken horizontally over nominally 2m contiguous intervals using a pneumatic chisel. 26 samples for 52 metres in total were taken from the open cut and 41 samples for 77.1 metres were taken from the underground workings.

53 of these channel samples (for 102.1m) have been included in the resource estimate. GFEL ascribed names describing their locations to contiguous channel samples. These have been coded as GFSTC01 to GFSTC15 in this resource estimate. Significant results are detailed in Table 2.1.

Two samples of mineralised garnet + actinolite skarn collected from the old mine workings were submitted to H.W. Fander at Central Mineralogical Services for polished section analysis, one of which contained gold (Fander in Roberts, 1987).

RGC (1988) – diamond drilling SD1 to SD6

In 1988 Renison Goldfields Consolidated (RGC), who had evolved corporately from GFEL, carried out a 6 hole diamond drilling programme around the Stormont Au + Bi mine (SD1 to SD6) for a total of 446.0m (Fleming, 1988). All holes were vertical with a triconed top, followed by HQ then NQ core.

Three of these holes (SD1, SD3 and SD5) were drilled to the southeast of the old open cut. All intersected mineralised skarn. Hole SD4 was drilled into an area of outcropping skarn to the southwest of these holes with holes SD2 and SD6 drilled into an area of outcropping skarn to the northeast. These three areas of skarn, which have been shown by mapping and later drilling to correspond to three distinct synclinal cores, are referred to herein as the central, western and eastern zones respectively. Significant results are detailed in Table 2.1.

The high grade resource estimated and described in this report occurs wholly within the central zone at its northwestern end and is referred to herein as "the high grade zone". Significant intersections in the central zone to the southeast of the estimated resource indicate the potential to extend this resource and are noted below.

Holes SD1 and SD3 made significant intersections used in this resource estimate. Hole SD5 made a significant intersection in the central zone which was not used in this estimate. Significant results are detailed in Table 2.1.

RGC (1990) – diamond drilling SD7 to 21

In early 1990 RGC carried out further drilling in the Stormont area, drilling holes SD7 to SD21 (for 571.2m) (Castro, 1990) with most holes drilled on gridline 5800N (nominally AMG east-west grid with line 5800N ~5405800mN) in areas of elevated magnetism considered due to skarn. Only holes SD7 to SD11 were drilled in the central zone with the other holes drilled into the western (SD20 & 21) and eastern (SD13 to 19) zones. All holes were vertical with a triconed top, followed by HQ then NQ core.

None of these holes are used in the resource. Holes SD8 and SD10 made significant intersections in the central zone not included in this resource estimate. Holes SD7, 9 and 11 intersected unmineralised skarn in the central zone. Significant results are detailed in Table 2.1.

RGC also carried out petrological studies of mineralisation with Dr Scott Halley (who had done his PhD on the contemporaneous Mt. Bischoff skarn) giving a detailed description of the paragenesis and location of gold within the skarn assemblage.

RGC relinquished the Stormont Au + Bi prospect at the end of 1990.

Goldstream/Titan (1996/97) – diamond drilling SD30 to SD61

In September 1992 Goldstream Mining N.L. (Goldstream) pegged the area including the Stormont Au + Bi prospect as EL 20/92 and entered a joint venture with Titan Resources N.L. (Titan). After focussing on skarns elsewhere in the licence the Goldstream/Titan J.V. carried out an initial drilling programme in 1995/96 drilling holes SD30 to 42 (for 711.5m) (Newnham, 1996). All core was HQ.

Holes SD36 and SD39 made significant intersections used in this resource estimate. Hole SD33 made a significant intersection in the central zone not included in this estimate. Holes SD30, 31, 32, 34, 35, 37, 38, 40 and 41 intersected unmineralised skarn in the central zone. Significant results are detailed in Table 2.1.

In 1996/97 Goldstream/Titan drilled a further 16 holes SD43 to SD61 (for 711.5m) (Newnham, 1997) in and around the Stormont prospect. All core was HQ.

Hole SD44 made a significant intersection in the central zone not included in this estimate. Hole SD43 intersected unmineralised skarn in the central zone. Significant results are detailed in Table 2.1.

Jervois (2000) – diamond drilling ST01 to ST04

In August 1999 EL 20/92 was transferred to Jervois Mining N.L. (Jervois). In early 2000 Jervois drilled 4 holes (NTW size = 64mm) in the Stormont prospect (Purvis, 2000). Holes ST01, ST02 and ST03 were drilled at the southeastern end of the central zone with ST04 drilled into the western zone. All holes were drilled vertically. Both ST01 and ST02 intersected unmineralised skarn. ST03 was drilled apparently east of the Stormont fault. The results of the three holes in the central zone were poor but appear to have closed off the main mineralised zone between SD44 and ST01. ST04 in the western zone intersected 2.0m at 3.5g/t Au. Significant results are detailed in Table 2.1.

Frontier (2008) – diamond drilling SFD001 to SFD016 and channel sampling

After initial work on the Higgs Skarn at Narrawa Creek Frontier commenced work on the Stormont prospect in early 2008, completing diamond drillholes SFD001 to SFD016 (for 543.9m). All holes were drilled HQ/NQ. Frontier also carried out further channel sampling in the open cut with 16 samples for 21.8m. Contiguous channel samples were named FRSTC01 to FRSTC04.

Holes SFD1 to 13 made significant intersections which have been included in this resource estimate. Holes SFD 14 to 16 made significant intersections in the central zone not included in this estimate. 14 of the channel samples for 17.8m have been included in this resource estimate. Significant results are detailed in Table 2.1.

Frontier submitted 227 samples of drill core for bulk density determination of which 84 were from the high grade zone.

Table 2.1 All high grade intersections

				Uncut grades		**Cut grades**	
Hole ID	**From**	**To**	**Downhole depth**	**Au g/t**	**Bi %**	**Au g/t**	**Bi %**
Central zone - High Grade Zone - used in resource estimate							
SD1	4.5	17.5	13.0	4.12	0.46		
SD3	16.9	19.0	2.1	12.80	0.35		
SD36	0.0	2.2	2.2	4.19	0.05		
SD36	14.2	19.7	5.5	4.34	0.11		
SD39	0.0	19.6	19.6	2.95	0.09		
SFD001	10.6	11.5	0.9	18.50	0.43		

SFD002	5.0	10.0	5.0	3.04	0.75		
SFD003	8.5	10.5	2.0	1.33	0.45		
SFD004	7.0	12.0	5.0	6.90	0.41		
SFD005	7.9	28.0	20.1	5.68	0.23	4.26	
SFD006	1.8	19.3	17.5	1.30	0.08		
SFD007	0.0	7.5	7.5	2.29	0.16		
SFD008	0.0	9.0	9.0	1.66	0.08		
SFD009	3.1	11.0	7.9	11.04	1.38		0.82
SFD010	2.7	5.0	2.3	2.16	0.03		
SFD011	2.4	17.0	14.6	3.53	0.14		
SFD013	7.2	9.2	2.0	1.53	0.04		
FRSTC01	4.0	10.0	6.0	1.27	0.37		
FRSTC02	0.0	2.5	2.5	0.97	0.33		
FRSTC03	0.0	8.0	8.0	3.13	0.20		
FRSTC04	0.0	1.3	1.3	26.70	0.55	25.00	
GFSTC01	0.0	19.0	19.0	10.00	0.77	8.62	0.70
GFSTC02	0.0	2.0	2.0	5.79	0.35		
GFSTC03	0.0	24.0	24.0	10.10	0.52		0.49
GFSTC04	0.0	10.5	10.5	5.41	0.30		
GFSTC05	0.0	7.0	7.0	26.50	0.53	17.37	
GFSTC06	0.0	1.2	1.2	36.53	1.10	25.00	
GFSTC07	0.0	1.2	1.2	36.47	0.53	25.00	
GFSTC08	0.0	1.2	1.2	12.46	0.50		
GFSTC09	0.0	1.2	1.2	6.29	0.24		
GFSTC10	0.0	1.2	1.2	11.20	0.47		
GFSTC11	0.0	1.2	1.2	8.48	0.25		
GFSTC12	0.0	1.2	1.2	3.78	0.13		
GFSTC13	0.0	1.2	1.2	3.44	0.07		
GFSTC15	4.0	34.0	30.0	5.08	0.57		
Other central zone intersections - not used in resource estimate							
SD8	28.1	29.4	1.3	2.99	0.02		
SD10	18.6	23.0	4.4	12.70	0.11		
SD33	27.5	29.0	1.5	9.00	0.17		
SD44	13.5	21.5	8.0	1.81	0.06		
Western zone intersection							
ST04	20.5	22.5	2.0	3.50	0.21		

2.2 Previous resource estimates

The Stormont central zone resource has been estimated twice previously, although by (admittedly in both instances) quite rough methodology.

In the first instance, in 1996 by Goldstream (Newnham, 1996) immediately post their 1995/96 drilling programme, the resource was estimated as a Au only resource. Their resource estimate extends further southeast than the high grade zone estimated herein.

In the second instance, in 2000 (Purvis, 2000) by Jervois immediately post their drilling of ST01 to ST03 the resource was roughly estimated as a Au + Bi resource. Their resource estimate correspond broadly with the high grade zone estimated herein.

Neither estimate includes the results of Frontier Resources Ltd's diamond drilling and channel sampling. This resource estimate is the first rigorous estimate using sufficient samples and geostatistically defensible methodology.

Goldstream's estimated is detailed herein

> "Whilst more data is required to elevate this deposit to the resource category, it is clear that a deposit of pre-resource mineralisation has been identified, and an approximate grade and tonnage can be estimated as follows:

A central section extending 90m, southeast of the open cut, embracing the intersections in SD1, SD36, SD39, and the underground sampling: SD1 13.0 v.m. @ 4.12g/t Au, SD36 9.5 v.m. @ 2.7g/t Au and SD39 18.4 v.m. @ 2.95 g/t Au and underground samples averaging 9.5g/t Au. Assuming an average width of 30m, a thickness of 13m and an S.G. of 2.5, this section would contain 88,000 tonnes. The weighted average of the drillholes is 3.2 g/t Au.

A northern section extending around the western side of the open cut and beneath the open cut. This section is about 30m along strike, 30m wide and may average about 4m thick (0-20m). Using a 2.5 S.G., this section may contain 9,000 tonnes. Channel sampling in the open cut averaged 4.8 g/t Au.

A southern section extending 80m southeast of the central section embracing intersections in SD3, SD33 and influenced by SD8, SD10, further to the south: SD3 2.1m @ 12.8 g/t Au, SD33 10.5m @ 1.4g/t Au (inc. 1.5m @ 9.0), SD8 74m @ 0.67 g/t Au (inc. 1.3m @ 2.99) and SD10 14.45m @ 0.95 g/t Au (inc 4.4m @ 2.9). Assuming a width of 15m, thickness 4m and an S.G. 2.5, this section would contain 12,000 tonnes. This may be conservative because of poor recoveries above the indicated interval in SD3. The average grade could be anywhere between say 2-5 g/t Au.

Combined these three sections indicate a mineralised body of approximately **100,000 – 150,000 tonnes with an average grade in the range 2-4g/t Au**. This estimate is arguably conservative because of the low SG used, and the interpretation placed on several drillholes." (Newnham, 1996)

Jervois's estimate is detailed herein:

"Three drillholes and channel samples of the old workings provide data for the resource calculation:

Open cut west wall:	32m @ 4.8 g/t Au, 0.56% Bi	along strike
No.2 Cross-cut	10m @ 5.34 g/t Au, 0.31% Bi	across strike
SD1: (vertical)	13m @ 4.12g/t Au, 0.46% Bi	(4.5m – 17.5m)
SD36: (vertical)	16.7m @ 1.98 g/t Au, 0.03% Bi	(0 – 16.7m)
SD39: (-70 to ENE)	19.6m @ 2.95 g/t Au, 0.09% Bi	(0 – 19.6m)"

"Channel sampling along-strike in the adit, although within the resource, has not been used as it was apparently driven on a relatively narrow unrepresentative high-grade zone:

Adit east wall:	42m @ 9.56 g/t Au, 0.50% Bi	along strike

The main body of the resource extends 90m SE from the open cut, incorporating the adit, SD1, SD36 and SD39. Forty-one metres SSW of SD36, SD3 intersected 2.1m @ 12.8 g/t Au & 0.35% Bi (16.9m – 19m). This was beneath an interval of clayey skarn that was triconed (unsampled) to 11m and had only 10% recovery from 11-14m.

The southern boundary of the resource has been drawn midway between SD36 and SD3, but there is a good chance there are additional resources in the vicinity of SD3.

The average width of the body outlined above is 30m (this is conservative to allow for the synclinal shape). The average thickness, from the true width of the three drill intersections, is 17m. Grade obtained from the weighted average of the cross-cut channel results and the three drillholes.

Results are as follows:

Length:	90m
Width:	30m
Thickness:	17m
SG:	2.75 (assumes 15% magnetite and 10% oxidation loss)
Tonnage and grade:	126,000 tonnes @ 3.34 g/t Au & 0.19% Bi

Remnant ore exists around and beneath the old open cut, in a wedge with a total length of 40m and a width at the SW end of 25m. This remnant zone is estimated at between 9,000t and 14,000t, depending on the thickness of skarn below the open cut. Grade, from the open cut channel samples, is 4.8 g/t Au & 0.56% Bi.

Overall Stormont Resource total: **135,000 tonnes @ 3.44g/t Au and 0.21% Bi**." (Purvis, 2000)

3.0 Geological setting

3.1 Introduction

The geology of the Stormont Au + Bi deposit is collated from descriptions given in Scott (1929), Broadhurst (1934), Roberts (1987), Fleming (1988), Castro (1990), Newnham (1996 & 1997) and Purvis (2000). The author has not visited the area nor inspected any drill core, however, for the most part these descriptions either concur or show development in the understanding of the areas geology, particularly associated with new drilling intersections.

There is still more geological work which should be carried out in order to enhance the understanding of the resources geology and optimise its extraction.

3.2 Project geology

The Stormont Au + Bi deposit is a body of skarn-type mineralisation which is apparently stratabound in a lower Ordovician sedimentary sequence with mineralisation introduced by a Devonian (Dolcoath Granite) granitic intrusion.

Ordovician sediments of the Denison and Gordon Groups underlie much of the licence area but are themselves obscured over a significant portion of the licence by a thin veneer (<25m based on drilling) of unconformably overlying Tertiary basalt and lesser sediment.

The lowermost unit of the Ordovician sequence exposed in outcrop and drilling within the licence area is of the quartzose Moina Sandstone which is between 80m and 100m thick. This unit is conformably overlain by a thin, approximately 20m thick sequence of interbedded calcareous siltstone with lesser calcareous sandstone and limestone known informally as "transition beds". These two units constitute the upper units of the Denison Group. The "transition beds" are conformably overlain by the Gordon Limestone which is approximately 400m thick regionally but within the licence only the basal 40m or so remains uneroded in the core of synclines.

Regionally this conformable sequence has been intruded by the Middle-Devonian Dolcoath I-type Granite with formation of a number of discrete skarn type orebodies. Within the Stormont RL the granite is not exposed in outcrop or drill core but is believed from gravity data to underlie the licence at a depth of <500m.

The Ordovician sequence is openly folded on northwest trending sub-horizontal fold axes with wavelengths around 150m. Skarnified limestone is preserved in the cores of these synclines. Within the licence area three such synclines have been recognised with the skarn bodies located in each of these synclines described as the central, western and eastern zones. The Stormont high grade Au + Bi resource estimated here lies at the northwestern end of the central zone. The folding is considered to have taken place during the Middle Devonian Tabberrabberran Orogeny.

3D modelling of stratigraphic contacts suggests that the Ordovician sequence is also gently folded on open northeast to easterly trending sub-horizontal fold axes resulting in the central northwest syncline plunging shallowly southwesterly north of approximately 5405850mN, and plunging shallowly northwesterly south of this point. No over printing relationships between the two fold orientations have been described and so the relative ages are unclear.

Northwest trending southwest verging thrust faults are mapped in the region and are also attributed to Middle Devonian Tabberrabberran Orogeny. The northeastern margin of the central zone skarn is apparently defined by a fault with this orientation over part of its length.

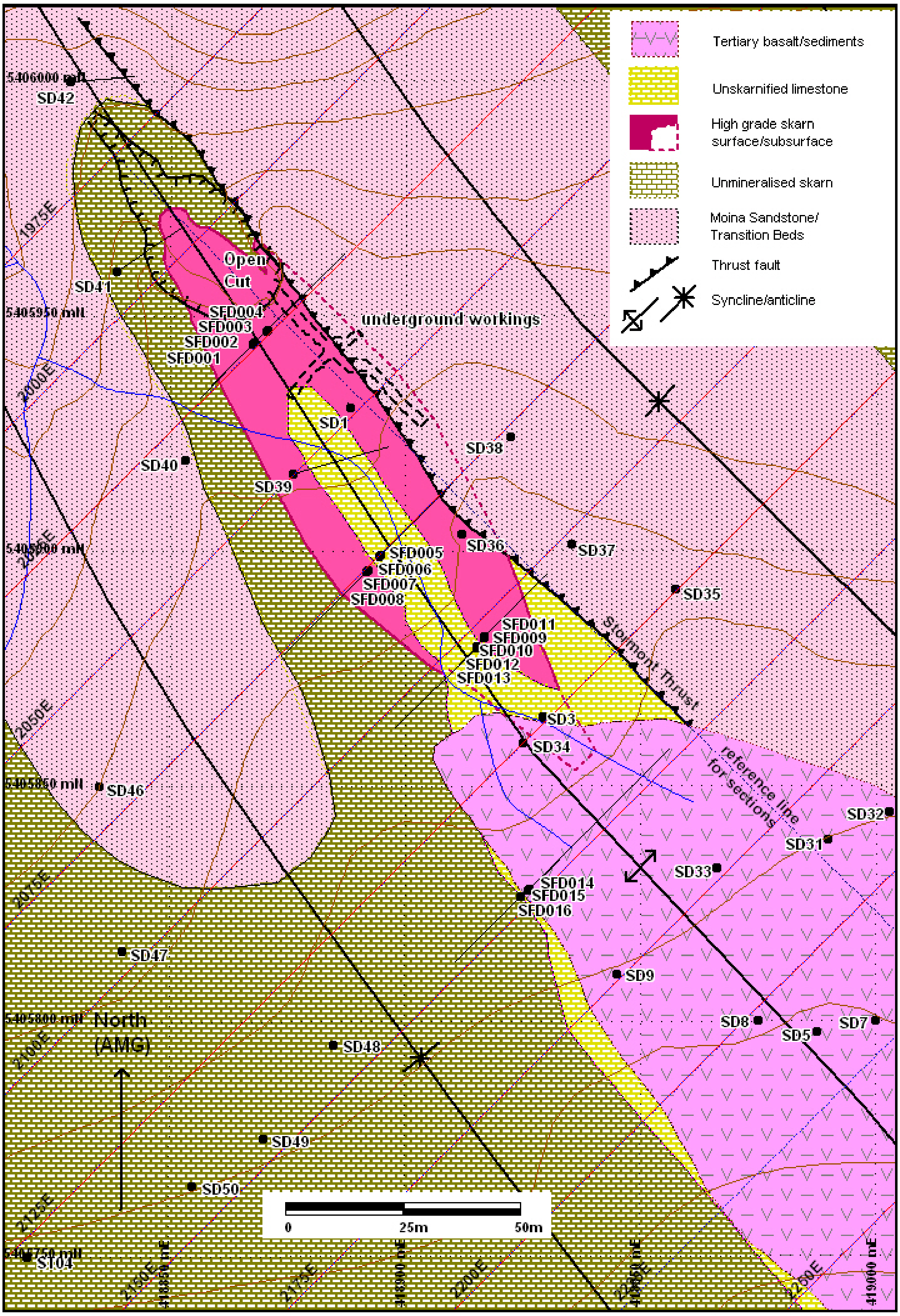

Figure 3.1: Geology of the Stormont Au + Bi deposit area

3.3 Resource geology

3.3.1 Introduction

The high grade Au + Bi resource estimated herein lies at the northwestern end of the central syncline within skarnified limestone near the base of the Gordon Limestone. Drill logs and mapping in the old workings show that massive skarn style mineralisation represented by a number of mixed assemblages extends essentially from the top of the Transition

Beds into the basal portion of the Gordon Limestone for a stratigraphic thickness of ~35m. Thinner discontinuous lenses of skarn mineralisation are also found in the Transition Beds with greisen style veining in the Transition Beds and extending into the upper part of the Moina Sandstone.

Whilst the basal 20m to 35m of the Gordon Limestone has been skarnified throughout (with unskarnified limestone overlying this commonly represented by dark grey or orange clay), the high grade Au + Bi resource estimated herein lies in the upper part of this skarnified zone and has a stratabound form. From previous workers descriptions of the geology drill logs the overall skarnified zone consists of a number of mineral assemblages, the spatial distribution of which appear unclear.

3.3.2 Relationship between Au+/-Bi and skarn mineral assemblages

Perhaps the most relevant description of the geology of Au+/-Bi mineralisation is that given in the petrological study by Dr Scott Halley (in Castro, 1990).

Dr Scott Halley carried out transmitted light petrology on 21 thin sections from skarn samples taken from SD1, 3, 5 & 6. He describes at least two phases of mineralisation with an earlier higher temperature primary assemblage of garnet+pyroxene+calcite overprinted by a later lower temperature retrogressive assemblage of actinolite+epidote+magnetite+/-fluorite at the expense of the earlier phase.

> “The Stormont deposit shows a typical skarn paragenesis. It has a primary skarn assemblage of garnet+pyroxene, with variable amounts of interstitial calcite. This has been overprinted by a retrograde assemblage of hydrous minerals, namely actinolite+epidote along with magnetite and minor fluorite. In turn, this stage has been overprinted by late stage veining which contains minor amounts of sulphide.
>
> The primary skarn contains clots of coarse-grained reddish-brown garnet within massive fine-grained olive-green diopside. Short intervals of skarn may be mono-mineralic. In places the skarn exhibits a distinct banding, with bands of garnet contained within massive diopside. This banding may reflect relict banding although it is not clear whether it is controlled by variations in original composition or permeability within the limestone. Primary skarns commonly show a zonation outwards from the granite contact or fluid feeder, from garnet to pyroxene to marble. However, no such zonation is evident at Stormont.
>
> During the retrograde overprint, diopside was the least stable and most easily altered mineral, being replaced by actinolite with minor calcite. During the early stages of retrograde alteration, garnet + actinolite appears to have been a stable assemblage. The retrograde skarn generally occurs as a massive replacement of the primary skarn by actinolite, epidote and calcite. The retrograde skarn is best developed, although not exclusively, at the base of the skarn unit. In many places, the entire thickness of primary skarn is overprinted.
>
> Magnetite shows a very patchy distribution through the retrograde skarn, with contents locally up to 50%. It occurs from spotty disseminations to thick bands. Much of the magnetite occurs in intricately banded wrigglite. Two types of wrigglite occur, a relatively early formed magnetite+garnet wrigglite, and a later magnetite+actinolite wrigglite. Occasionally, fluorite was observed in the magnetite+actinolite wrigglite, but generally it is too fine-grained to detect. Known examples of wrigglite skarns have more than 10% F e.g. Moina, Mt. Bischoff, Mt. Garnet (Qld.), and Lost River (Alaska), and the Stormont wrigglite most probably is also fluorine-rich. Without thin-section work, it is impossible to be sure of its mineralogy, but it may well also contain minerals such as Fe-Mg micas and vesuvianite. The actinolite+magnetite wrigglite is cut by later bands of actinolite or epidote.
>
> Magnetite rich rocks, particularly those with abundant veinlets, commonly show overprinting of actinolite by a later assemblage of quartz+magnetite with chlorite or with green mica. These rocks are difficult to distinguish macroscopically from the actinolite assemblage as both are dark green and fine-grained.
>
> Abundant, late-stage, thin “greisen” veins cross cut the skarn, particularly the retrograde skarn. These veins contain fluorite and quartz, with conspicuous selvedges of coarse-grained muscovite, and have haloes of magnetite in the adjacent wallrock. They also contain minor amounts of pyrite, pyrrhotite, chalcopyrite, sphalerite and bismuthinite. Late fractures, without veins, also contain alteration envelopes of magnetite. Epidote+calcite+fluorite veins also occur in places.

Assay results indicate a very good correlation between bismuth and gold. Bismuthinite has three main occurrences. Assays indicate Bi levels of 400 to 1000ppm Bi in the wrigglites, although it is usually too fine-grained to detect visually. Visible bismuthinite commonly occurs in the late-stage greisen veins. **The third and relatively limited occurrence, is in garnet+pyroxene skarn with incipient alteration of pyroxene to actinolite. This is the type of mineralisation that occurs in the Stormont Bismuth Mine, where relatively good gold grades are associated with coarse-grained bismuthinite**."

Fander in Roberts (1987) carried out polished section petrology on two samples of garnet-actinolite with close affinities of which one, sample T4888, contained gold and bismuth minerals which he described as follows:

"This rock may be classified as a <u>garnet-actinolite rock</u> or <u>skarn</u>. It consists essentially of medium-grained (mean 750µ) complexly growth-zoned and sector-twinned pale yellow grossular-andradite with semi-pervasive included fine-grained actinolite. Sporadic relatively massive lenses and crude bands of actinolite are present. Accessory poikilitic quartz occurs within the actinolite aggregates and intergranular to garnet. Minor discontinuous quartz veinlets exhibit selvedges of garnet. Isolated flakes of dark green biotite occur in the relatively massive actinolite aggregates. These are partly altered to microcrystalline cloudy calcite and chlorite and are partly weathered, with associated Fe-stainings.

One actinolite aggregate includes a 2x4mm ovoid aggregate, and subordinate actinolite-interstitial fine-grained disseminations of, bismuthinite. The coarse aggregate is granular-textured, with disseminated bismuthinite-intergranular patches of bismuth ranging to 160µ diameter. Both bismuthinite and bismuth exhibit very thinly dispersed included blebs of gold ranging to 25µ diameter (ovoid bleb in bismuth), but typically <10µ. Bismuth including gold is largely untwined and these composites <u>may</u> represent degraded maldonite"

Genetically the relationship between Bi and Au is reasonably well supported, however, statistically the correlation between Au and Bi in the high grade zone is only 0.39. Therefore the Stormont resource should be seen as a gold deposit with bismuth credits. The relationship between Au and actinolite and the other retrograde minerals does not appear to have been investigated in recent work. The distribution of Au and Bi with respect to the macroscopic skarn assemblages should be one of the foci in a thorough relogging of core and remapping programme.

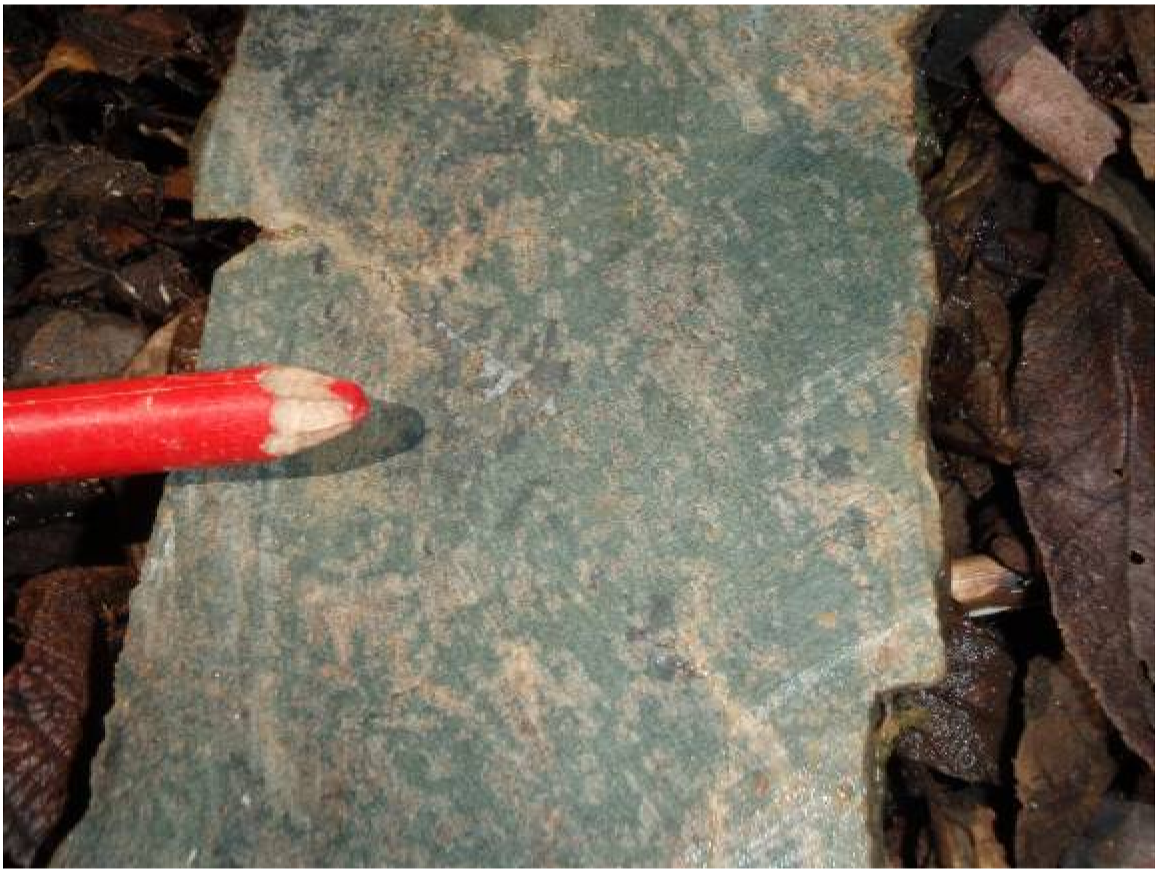

Figure 3.2: Mineralised skarn from Frontier channel FRSTC03 (Sample Number 434513) included in resource estimate – channel assayed 1.2m at 13.5g/t Au, 9 ppm Ag and 0.49% Bi

3.3.3 Spatial and structural controls on mineralisation

Geological cross-sections showing drillhole and channel sampling Au results superimposed upon the interpreted geology and colour coded block model slices are presented in figures 3.4 to 3.14.

The high grade zone is intersected on sections 2002E to 2125E. These sections show high grade intersections to be distributed fairly consistently around the syncline both in the axes to the fold and on both limbs, predominantly within a well defined stratigraphic interval which range in thickness from 10m to 15m thickness and generally between 8m and 15m (up to 20m) above the top of the Denison Group sediments (Moina Sandstone/Transition Beds). Consistently stratigraphically overlying this mineralised skarn zone is a unit of clay, orangey in colour near the surface, grey to dark grey (with thin shale lenses) in colour below this. This clay is interpreted by most previous workers as weathered unskarnified limestone. Thus high grade mineralisation appears to have a strong stratigraphic control. The reasons for this are unclear and require more detailed geological appraisal of this part of the stratigraphic sequence in stratigraphically corresponding yet unmineralised sections in the western or eastern synclines.

At first glance the relationship between high grades and the synclinal fold appear compelling, however, high grade intersections occur at shallow depths with the ground surface essentially defining the upper limits to high grade mineralisation on sections 2000E to 2100E. It is possible that prior to erosion this high grade zone continued in this part of the stratigraphic sequence away from the syncline. Conversely the fact that only occasional Au+/-Bi intersections have been made in the same stratigraphic position in the western and eastern synclines suggests that there did exist some along strike control on relatively consistent higher grade mineralisation prior to erosion. Further to this, the preserved sections of the synclinal fold limbs are relatively short and it does remain reasonable to interpret that may well be a relationship between high grade mineralisation in the favoured stratigraphic position and the synclinal fold axis.

The introduction of skarnifying fluids into reactive stratigraphic units usually requires a structure or structures. The location of higher grades towards the end of the old adit coincident with one or two northwest striking, steeply southwest dipping fault surfaces provided further incentive to investigate whether there is any clear structural control on the distribution of high grade mineralisation. Summary core logs from Frontiers drilling commonly refer to shearing in skarnified zones. These zones were plotted on 3D drillhole traces and assessed visually. Whilst a number do line up with a similar trend to the faults in the adit, the number of these and lack of fabric to core axis data in drill logs meant that any interpretation would be equivocal and so this approach was not pursued in modelling. The high grade channel assays in the adit were addressed by declustering and top cutting. It does not appear to be necessary to invoke a strong structural control on the distribution of high grade mineralisation.

The other structure of importance to the geometry of the resource is the Stormont Thrust. This structure is a little enigmatic in that it is apparently well expressed (according to the drill logs) in a number of drillholes (two of which lie sufficiently close for a dip of 52º to be modelled) and is necessary to explain the presence of Denison Group sediments at shallow depths to the immediate northeast of the skarn body, yet it is poorly expressed (again according to the drill logs) in other drillholes. It also apparently swings or is offset in a number of locations..

From the lack of any spatial correspondence between the Stormont Thrust and high grade mineralisation the fault does not appear to have played any role in introducing skarnifying fluids. Rather the fault acts to truncate the high grade zone by dislocating the northeast block upwards with subsequent erosion removing any mineralisation which may have existed in this northeast block. It makes structural sense that thrusting occurred late or post folding when mineralisation is more likely to have been introduced.

For these reasons the Stormont Au + Bi resource has been modelled as a stratiform deposit. The significance of the post-mineralisation Stormont Thrust is simply that in the main central part of the resource it has uplifted the skarn in the northeast block with its subsequent erosion.

The northern extent of high grade mineralisation is well defined by the intersection of the favoured stratigraphic section with the ground surface. The southern extent of high grade mineralisation is less clear. The high grade resource has been modelled to midway between SD3 and SD33. Discrete high grade intersections are made in drillholes SD8 and SD10 further southeast in the central zone, however, the presence of low grade intersections near to these holes does not allow the incorporation of these intersections into a coherent model with a sufficient level of geological integrity to be used in an inferred resource estimate. It is certainly the case that further drilling in this region is the best chance to volumetrically increase the resource.

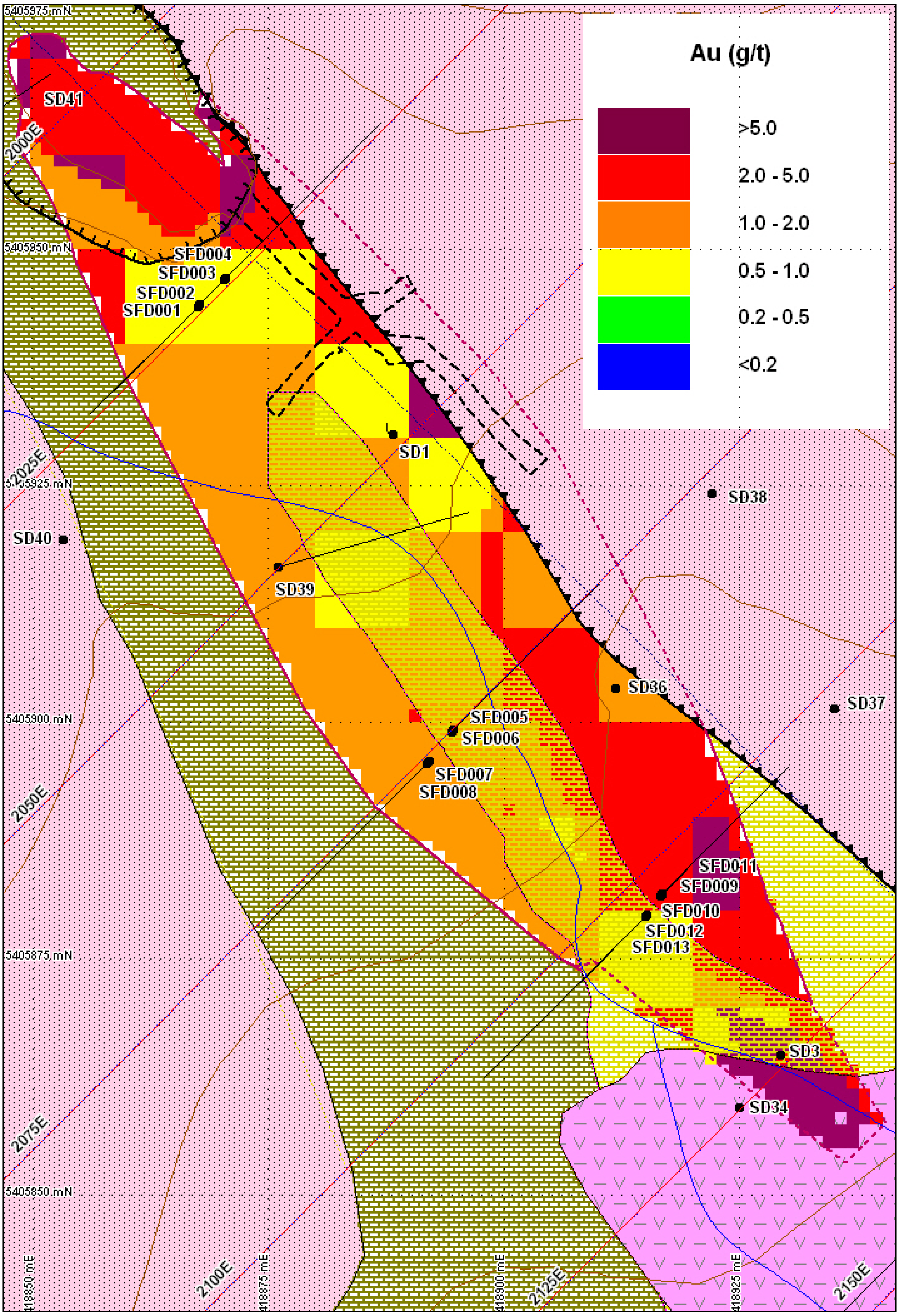

Figure 3.3: Geology of Stormont Au + Bi high grade resource (geology legend as for figure 3.1)

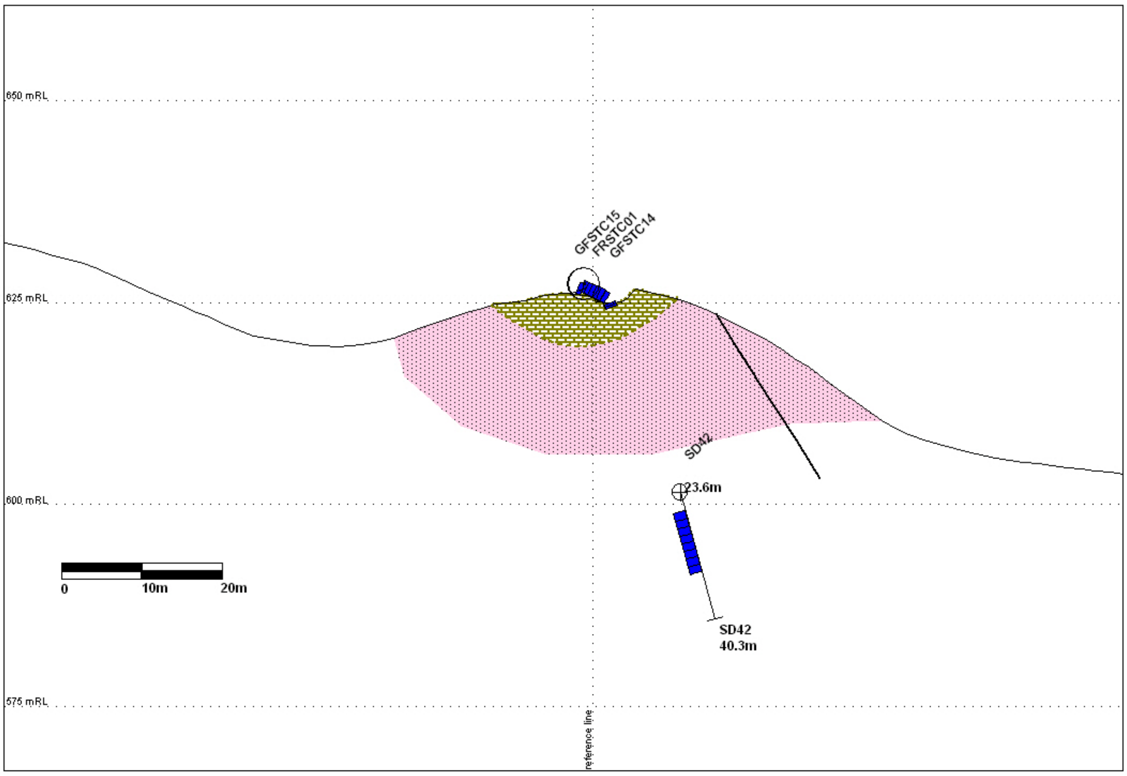

Figure 3.4: Section 1975E looking to 325 AMG (legend as per figures 3.1 and 3.3)

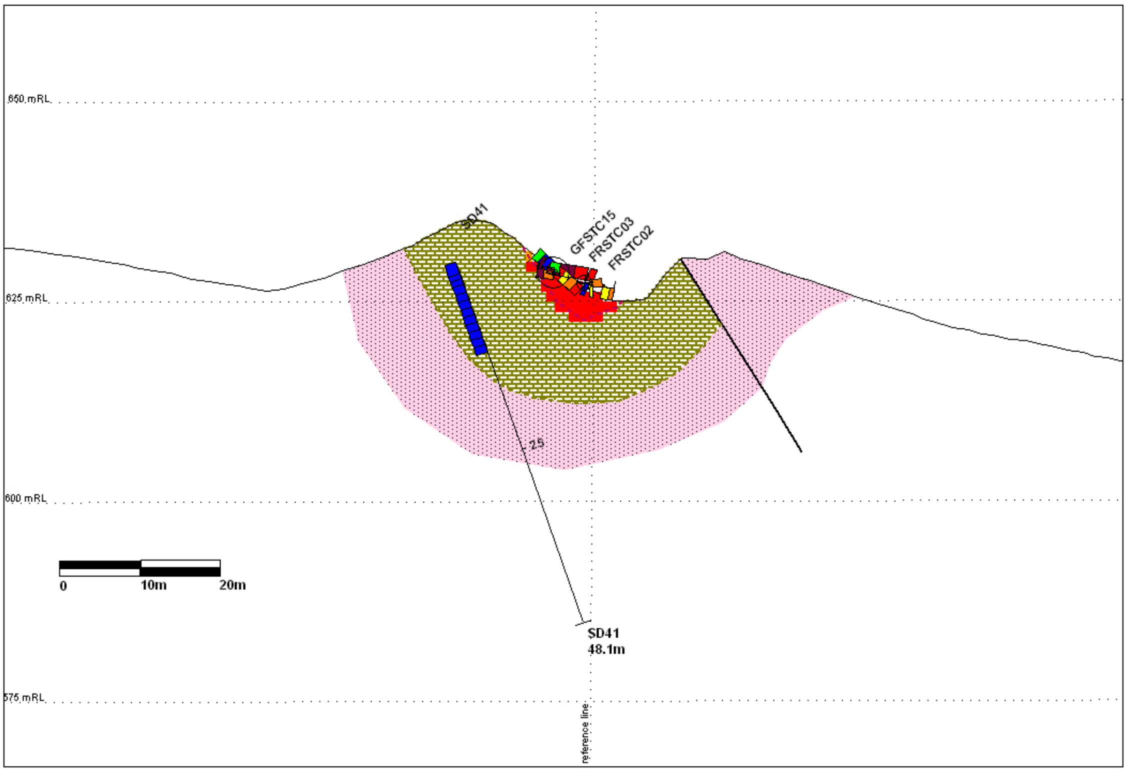

Figure 3.5: Section 2000E looking to 325 AMG (legend as per figures 3.1 and 3.3)

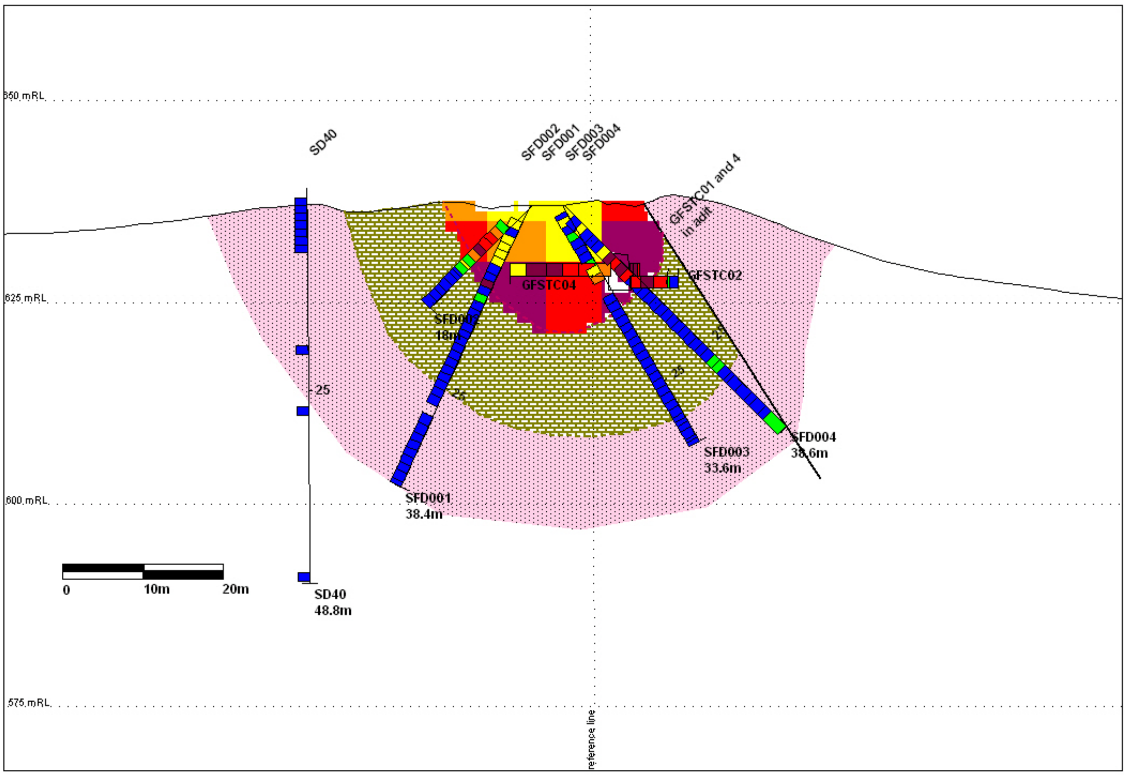

Figure 3.6: Section 2025E looking to 325 AMG (legend as per figures 3.1 and 3.3)

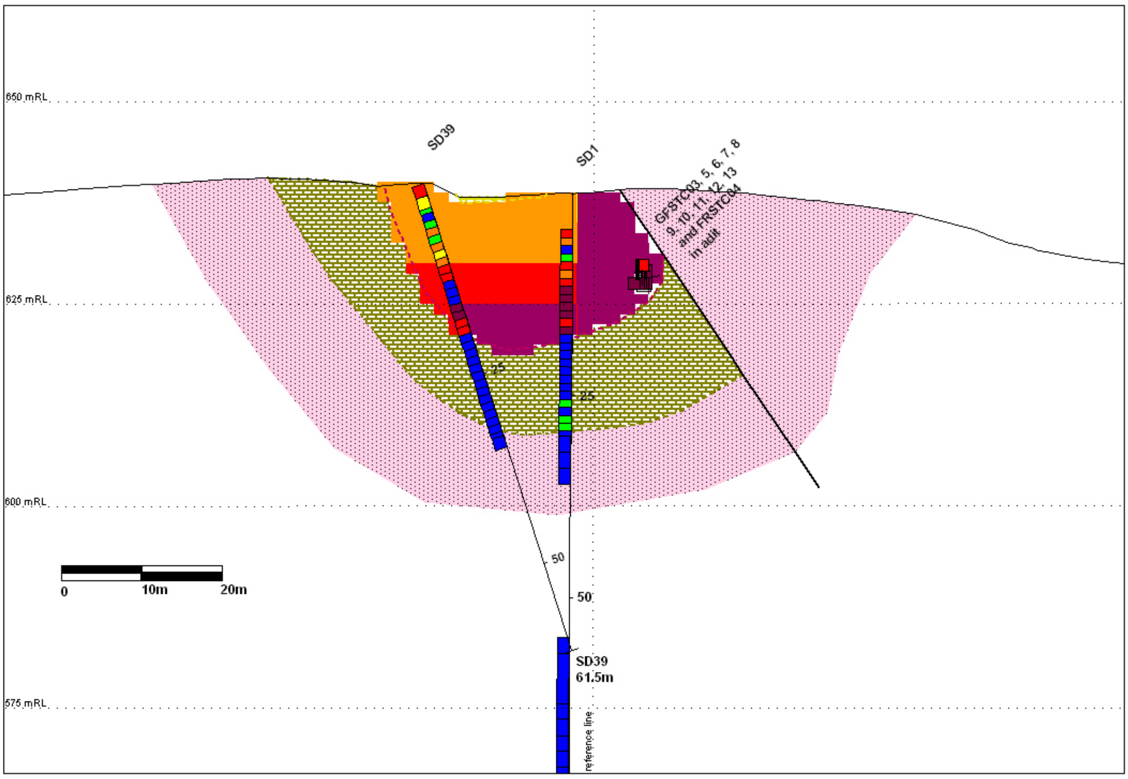

Figure 3.7: Section 2050E looking to 325 AMG (legend as per figures 3.1 and 3.3)

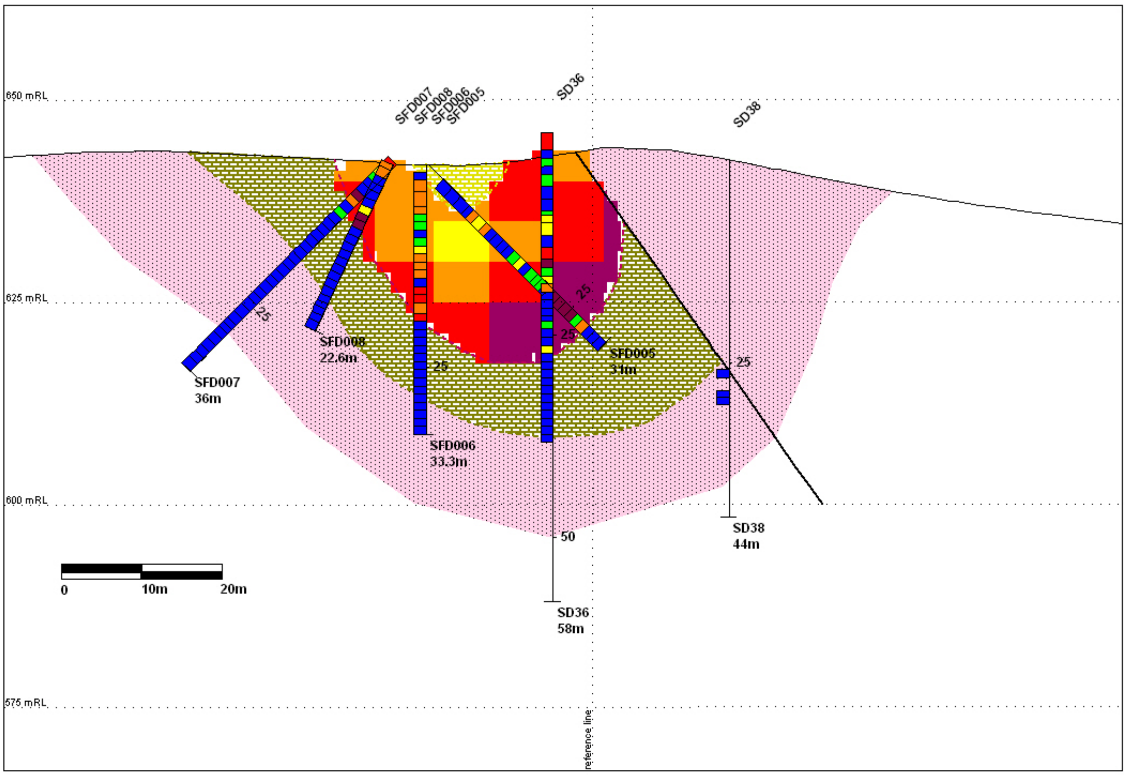

Figure 3.8: Section 2075E looking to 325 AMG (legend as per figures 3.1 and 3.3)

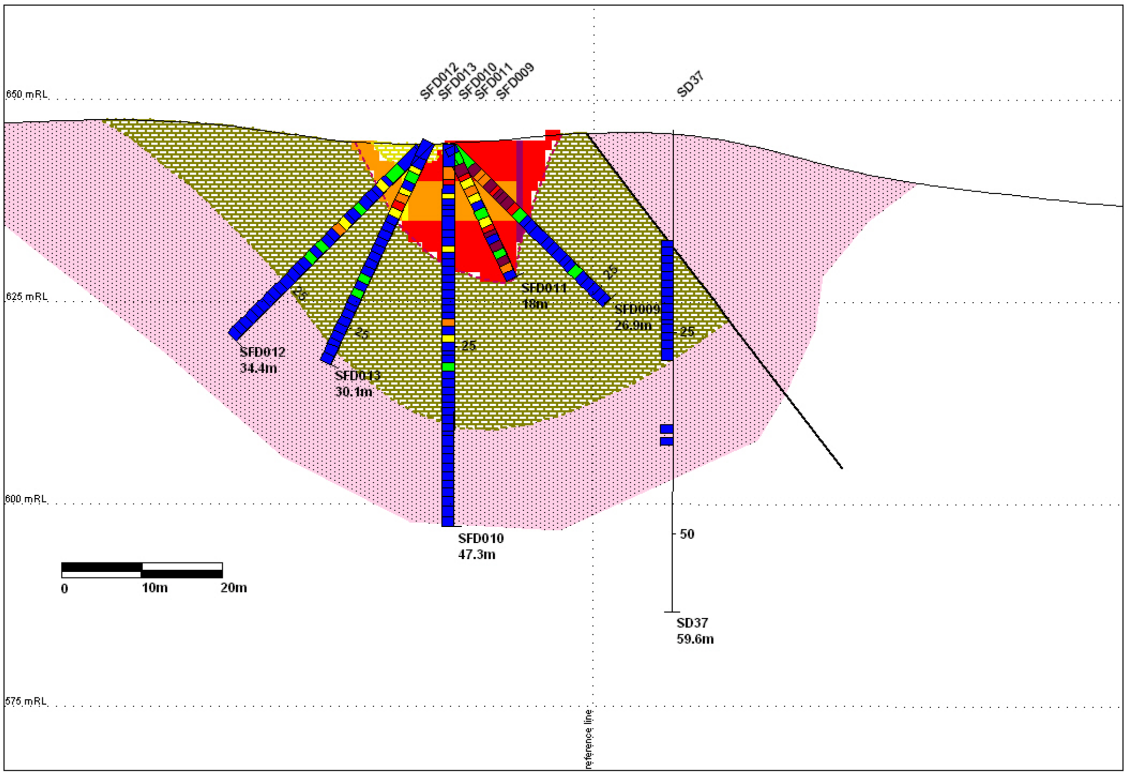

Figure 3.9: Section 2100E looking to 325 AMG (legend as per figures 3.1 and 3.3)

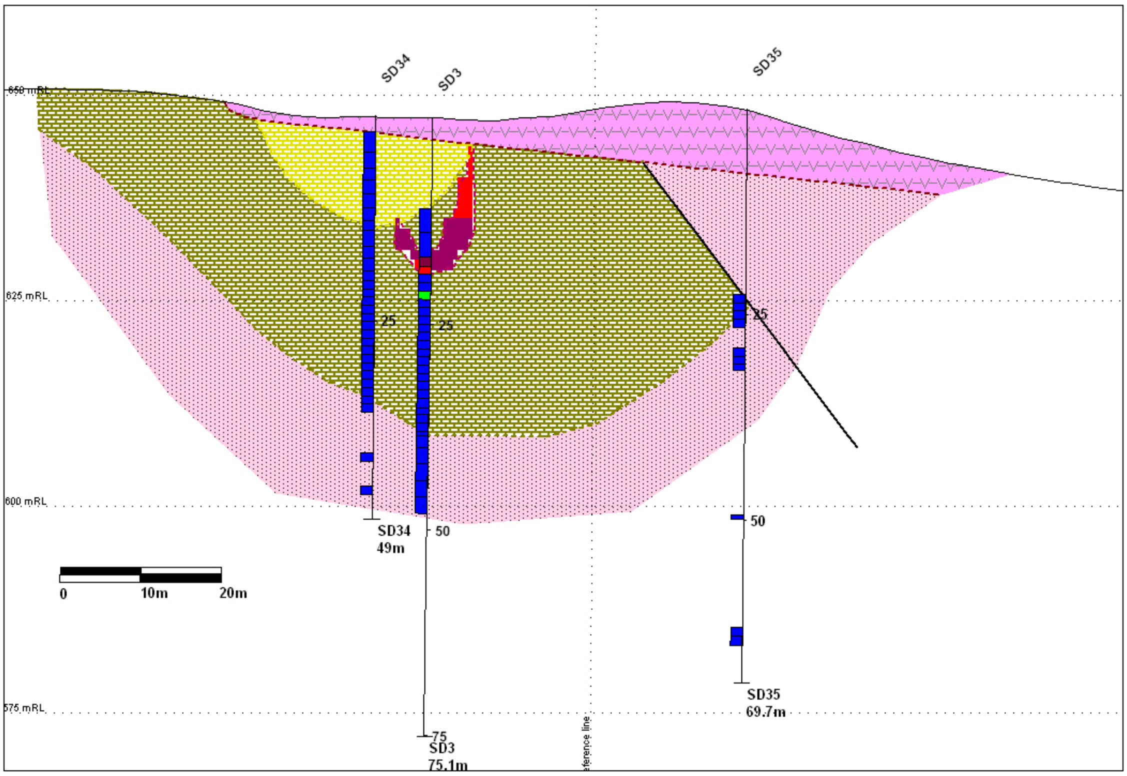

Figure 3.10: Section 2125E looking to 325 AMG (legend as per figures 3.1 and 3.3)

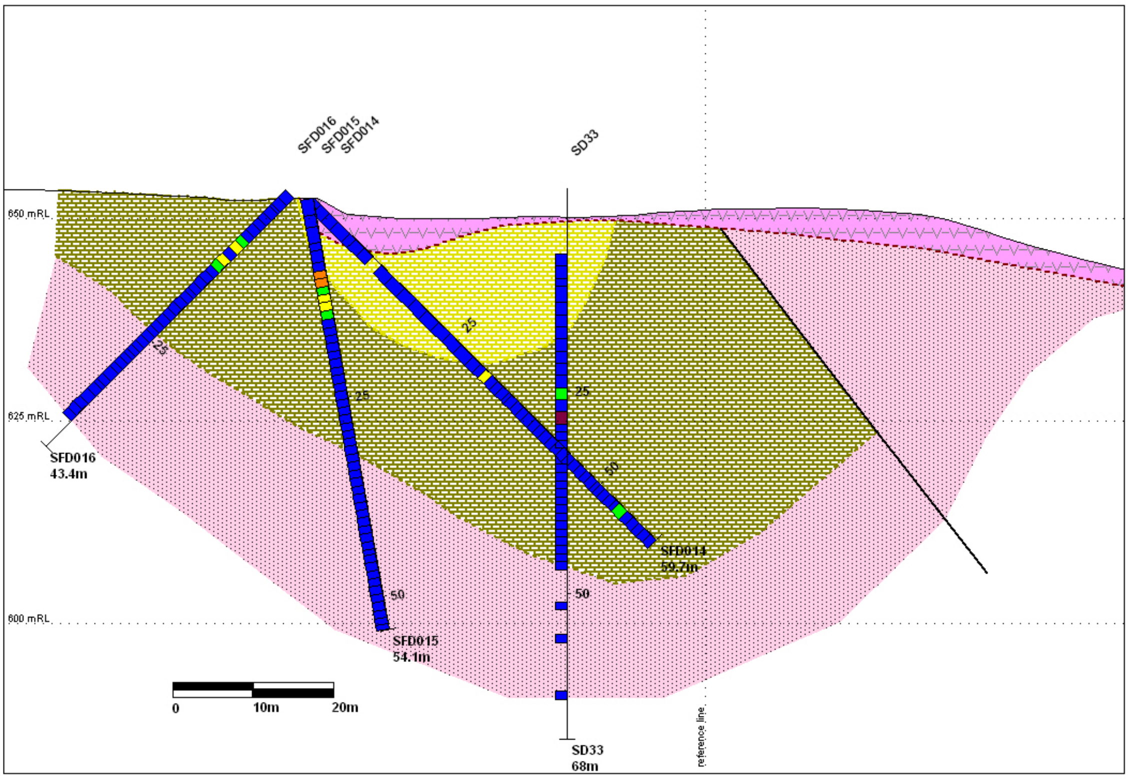

Figure 3.11: Section 2150E looking to 325 AMG (legend as per figures 3.1 and 3.3)

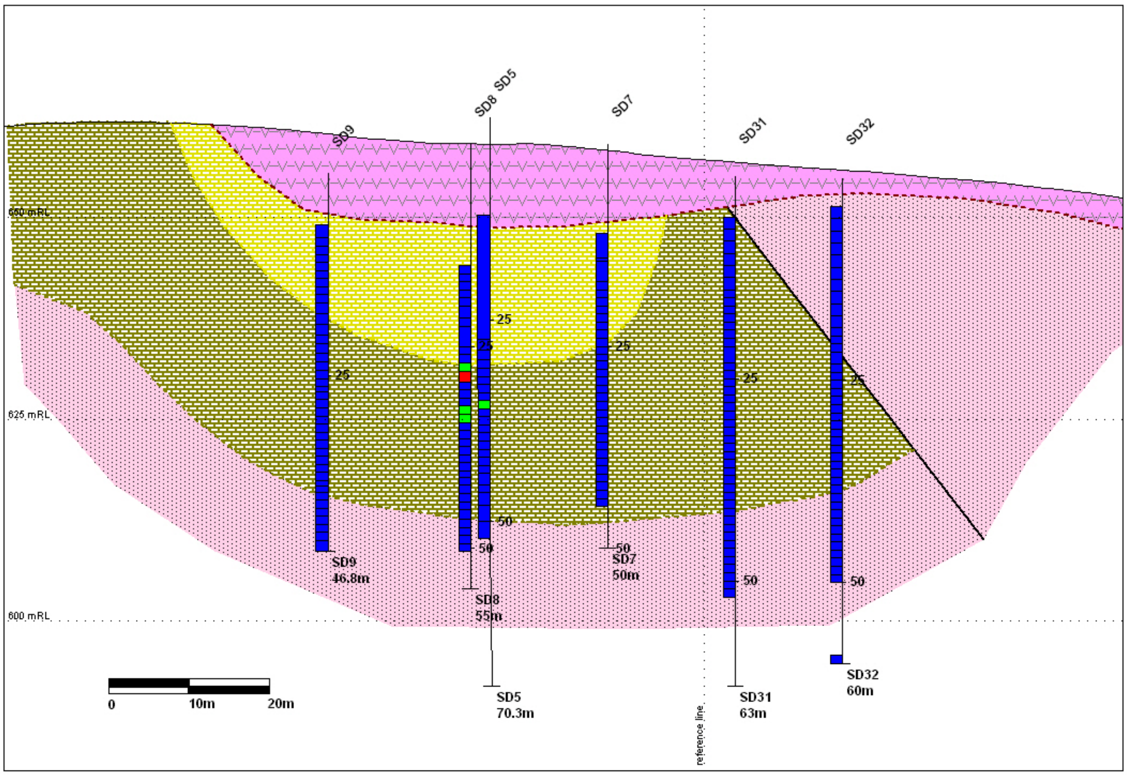

Figure 3.12: Section 2200E looking to 325 AMG (legend as per figures 3.1 and 3.3)

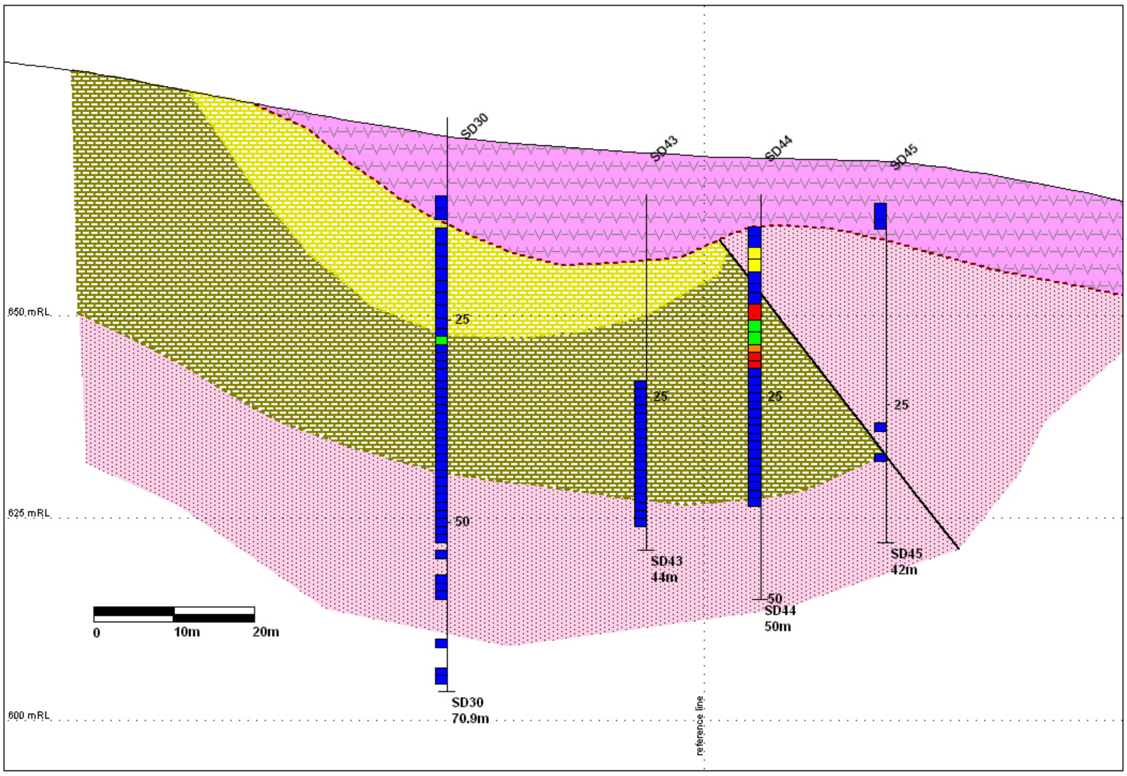

Figure 3.13: Section 2250E looking to 325 AMG (legend as per figures 3.1 and 3.3)

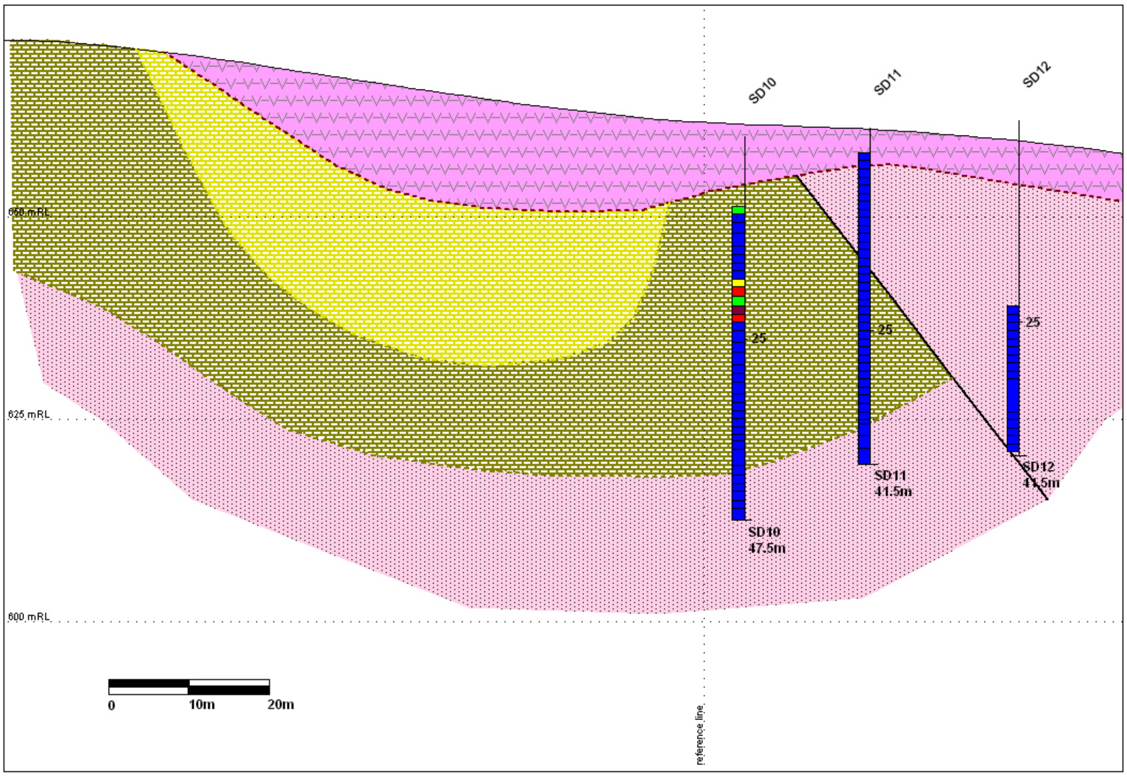

Figure 3.14: Section 2300E looking to 325 AMG (legend as per figures 3.1 and 3.3)

4.0 Data collection

This assay data used in this resource estimate comes from diamond core drilling on nominally 25m spaced sections and channel sampling along the walls of the open cut and underground workings.

Bulk density data utilised in this estimate has come solely from Frontier's recent drilling programme with a total of 70 ore grade samples submitted for bulk density analysis by the water immersion method.

4.1 Drilling

A summary of the drillholes in the central zone and those utilised in the resource estimate is given in table 4.1. Details of the relevant drilling programmes have been described in section 2.1.2. Locations of drillholes are shown in plan view in figure 3.3 and in section view in figures 3.4 to 3.14. SD1 is the deepest hole at 145.0m. All other holes are less than 75.1m deep.

Nearly all of the earlier drilling programmes (pre-Frontier) drilled vertical holes with the exceptions being SD39, SD41 and SD42 which were angled towards 74º, 56º and 86º (AMG) respectively at -70º. Frontier's drillholes were all angled holes oriented along their 045º-225º (AMG) grid as fans of angled holes commonly drilled from a central drill pad.

Table 4.1: Summary drillholes - Stormont Au+Bi resource

Total holes in central zone							
Company	**Year**	**No. holes**	**Total length**	**Ave. length**	**Core size**	**Hole ID's**	**No. samples**
RGC	1988	3	290.4	96.8	HQ/NQ	SD1, 3 & 6	88
RGC	1990	6	282.3	47.1	HQ/NQ	SD7 to 12	150
Goldstream/Titan	1995/96	13	740.9	57	HQ	SD30 to SD42	326
Goldstream/Titan	1996	3	136	45.3	HQ	SD43 to SD45	52
Jervois	2000	3	131.9	44	NTW	ST01 to ST03	74
Frontier	2008	16	543.9	34	HQ/NQ	SFD001 to SFD016	525
Total		**44**	**2125.4**	**48.3**			**1215**
Total holes used in resource estimate							
Company	**Year**	**No.**	**Total**	**Ave.**	**Core**	**Hole ID's**	**No.**

		holes	length	length	size		samples
RGC	1988	2	220.1	110.1	HQ/NQ	SD1 and SD3	
Goldstream/Titan	1995/96	2	119.5	59.8	HQ	SD33 and SD39	
Frontier	2008	13	402.1	30.9	HQ/NQ	SFD001 to SFD013	
Total		**17**	**741.7**	**43.6**			

Core loss both from surface as well as downhole may have affected the resource estimated. Most of the orebody essentially outcrops on the surface though most holes intersected 1m to 2m of (unsampled) soil/ colluvium at the collar. Hole SD1 which collared in outcropping skarn had complete core loss from 0.0m to 4.5m which potentially lies within the highest grade core of the resource. Sections with core loss in sections of skarn were modelled as such but with no sample data.

Hole SD3 has complete core loss from 0.0m to 11.0m at which point the hole passes into lower grade mineralised skarn. Due to the proximity of SD34 in which core loss was minimal it is most likely that this core loss took place in the dark grey clay interpreted as after limestone. SD36 and SD39 had generally good recoveries through the high grade section with 30% loss in SD35 between 0m to 17.2m.

Other holes have had partial core loss though mineralised sections. These have been considered on equal merit with fully recovered sections with no weighting.

4.2 Surveying

4.2.1 Collars

Drillhole collar survey accuracy is one of the most significant deficiencies of the Stormont database. Drillhole collars have been surveyed by a variety of means ranging from conventional, conventional from GPS control and direct GPS. Should this resource be upgraded to indicated status, and particularly if further drilling in the southern portion of the central zone is successful in extending this resource it is recommended that the all relevant drillhole collars be conventionally surveyed.

Some effort has been made in determining the most accurate collar position by both Frontier and by this author with reference made to original plans and reports.

Within the estimated resource Drillholes SD1 and SD3 have been conventionally surveyed (presumably from conventional control), drillholes SD31, 33, 34, 36, 37, 38, 39 and 41 (holes SD31, 33, 34, 37, 38 and 41 are not part of the resource but their position has helped define the bounds of the high grade zone) conventionally surveyed off GPS control (two stations) and holes SFD01 to SFD13 directly from GPS, all by the relevant explorers. The inaccuracies inherent in direct GPS and conventional off GPS control are estimated to generally be of the order of +/-3m in the x and y directions, however, GPS z coordinates can often be out by as much as 10 metres.

To add further to this problem the surface topographic 10m contours supplied by the Land Department's mapping branch clearly do not reflect the spur in the Stormont Au + Bi mine area. Frontier carried out a programme of GPS surveying including the GPS collar coordinates for theirs and a number of older drillholes (with some tape and compass traversing) to generate a more realistic surface contour DTM. It is this DTM which has been used in modelling the resource.

The z coordinates of SD36 and SD39 were not included in the creation of this DTM, however they match both the Lands Department topographic contours and Frontier surface DTM (which concur in this region). However, the z coordinates given for SD1 and SD3, whilst notionally conventionally surveyed are differ from the Frontier generated surface DTM by as much as 10m. In order to maintain a relative consistency between the positions of the drillholes used in the resource estimation the z coordinates of SD1 and SD3 were adjusted by 10.0m and 9.5m (up) respectively.

This lack of confidence in drill collar z coordinate and perhaps surface DTM contributes to the inferred status of the resource and needs to be addressed, particularly given the relatively small size of the resource.

Given the short nature of these holes (range from 41.5m to 55.0m) the lack of downhole surveys is not considered a significant problem with end of hole positions likely to be +/-0.5m at most (1 degree error in 50m is <0.1m).

4.2.2 Downhole surveying

Apart from holes SD1 to 5 there has been no downhole surveying of drillholes. Given the short nature of the holes this is reasonable for an inferred resource and is not of the same order as the potential error in collar position. Of the 5 holes surveyed (by Eastman single-shot) all holes remained essentially vertical with no holes deviating to a dip of less than -89.0°.

4.3 Channel sampling

A summary of the channel samples and those utilised in the resource estimate is given in table 4.2. Details of the relevant sampling programmes have been described in section 2.1.2. Locations of channels are shown in plan view in figure 4.1 and in section view (in part) in figures 3.4 to 3.7

Table 4.2: Summary channels - Stormont Au+Bi resource							
Total channel samples in central zone							
Company	**Year**	**Total length**	**Min. length**	**Max. length**	**Modal length**	**nominal ID's**	**No. samples**
GFEL	1987	129.1m	1.0m	3.0m	2.0m	GFSTC01 to 15	67
Frontier	2008	21.8m	0.5m	2.0m	variable	FRSTC01 to 04	16
Total		**150.9m**					**83**
Total channel samples used in resource estimate							
Company	**Year**	**Total length**	**Min. length**	**Max. length**	**Modal length**	**nominal ID's**	**No. samples**
GFEL	1987	102.1m	2.0m	3.0m	2.0m	GFSTC01 to 15	53
Frontier	2008	17.8m	0.5m	2.0m	variable	FRSTC01 to 04	14
Total		**119.9m**					**67**

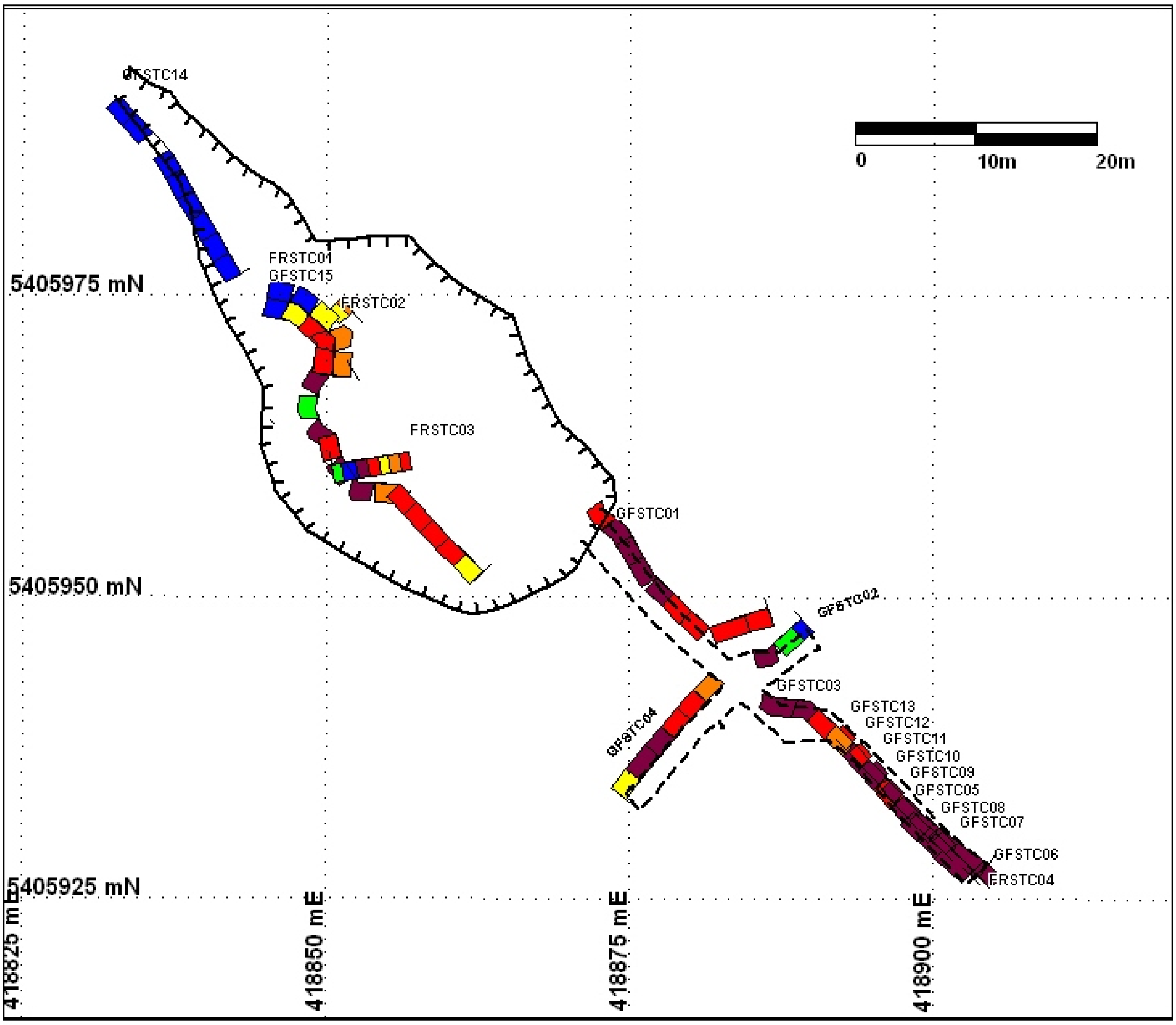

Figure 4.1: Channel samples (legend as per figure 3.3)

4.3.1 Surveying

Frontier channel sample positions in the old open cut were surveyed by GPS whilst those of GFEL were estimated from old plans. The exception to this is Frontier channel FRSTC01 which is a field duplicate of GFSTC15 in part and was consciously located coincident with the earlier channel..

Channel samples in the old workings for both GFEL and Frontier channels were measured by tape from known features within the workings (i.e. from corner of cross-cut with main drive) and have been positioned accordingly so as to be correct with respect to the old workings.

4.4 Logging

Core logging has been carried out by a number of geologists. These geologists are almost all known to the author and are technically sound. There is a degree of consistency between core logs in part because of the cross-referencing of geological interpretations with earlier logging/interpretation as well as the close spacing and thus largely contiguous geology intercepted in the drillholes.

As potentially economic mineralisation is found almost invariably within the massive skarn Au+/-Bi grades largely define the extent of mineralisation.

Perhaps the greatest room for error lies in the interpretation of highly weathered zones and/or clays which may be after skarn, unskarnified limestone or possibly after weathered Tertiary sediments. Should the resource be updated from inferred to indicated it is recommended that some relogging or check logging be carried out to validate some of this earlier work.

4.5 Sampling

4.5.1 Drill samples

In all drilling programmes diamond drill core was split and sampled half core. In most instance core was cut by diamond saw though in more highly weathered sections core was split by hand (by mortice or sampled from loose material). Sample intervals are demonstrably based on the logged geology in apparently all instances.

4.5.2 Channel samples

Channel sample length was chosen by nominal sample length modified by physical changes in the surface being sampled (i.e. corners of drives, changes in the wall of the open cut). There is no description of the volume of material sampled in individual samples. GFEL's channel sampling programme was carried out using a pneumatic chisel. Frontier's sampling was carried out using a diamond saw.

4.5.3 Sample preparation and analyses

The resource has been estimated from samples from five sampling programmes (3 drilling and 2 channel), however, results from the other three (all drilling programmes) have been used to constrain the resource.

Details of sample preparation are not given for the various programmes, however, since samples were all assayed by accredited laboratories with fire assay the method for gold analysis it is reasonable to assume that sample preparation was to industry standard.

Table 4.3 Assay method and detection limits

Sampling programme	Laboratory	Au		Bi		Ag	
		Method	Detection limit (ppm)	Method	Detection limit (ppm)	Method	Detection limit (ppm)
GFEL channels	Analabs, Burnie	Fire assay	0.005	AAS	10	AAS	1
RGC SD1 to SD6	Analabs, Burnie	Fire assay	0.008	AAS	10	AAS	0.5
RGC SD7 to SD12	Analabs, Burnie	Fire assay	0.008	AAS	10	AAS	0.5
Goldstream/Titan SD30 to SD42	Amdel, Adelaide	Fire assay	0.01	AAS	5	AAS	NA
Goldstream/Titan SD43 to SD45	Amdel, Adelaide	Fire assay	0.01	AAS	5	AAS	NA
Jervois ST01 to ST04	Analabs, Burnie	Fire assay	0.01	AAS	10	AAS	1

Frontier SFD01 to SFD16	Burnie Research Lab	Fire assay	0.01	AAS	10	AAS	1
Frontier channels	Burnie Research Lab	Fire assay	0.01	AAS	10	AAS	1

Over the five sampling programmes a range of elements have been assayed as well as Au, Bi and Ag. Due to the inconsistency of elements assayed only Au, Bi and Ag have been estimated in the block model though Ag was not assayed for in the Goldstream/Titan drillholes.

4.6 Quality control procedures

As is common with exploration drilling there has been only limited implementation of quality control procedures with most of the work being that have been so done by the assay laboratory as part of their own standard QA/QC procedures.

There has been only two sets of field duplicates collected and sampled with Frontiers channel samples FRSTC01 and FRSTC04 replicating GFEL channels GFSTC15 (in part) and GFSTC06 respectively. The latter was a successful attempt to repeat the high grade assay at the end of the adit with Frontier's sample assaying 26.7g/t Au as opposed to GFEL's assay of 36.0 g/t Au. Both samples have been topcut in the resource estimation.

Of greater significance is the attempt by Frontier to repeat high grade assays from GFEL's channel along part of the southwest wall of the open cut. Those results are presented in figure 4.2 and show that this was not achieved. Overall the high tenor of Frontier's other channel sampling around though not specifically corresponding to earlier GFEL channel locations in the open cut does support GFEL's results.

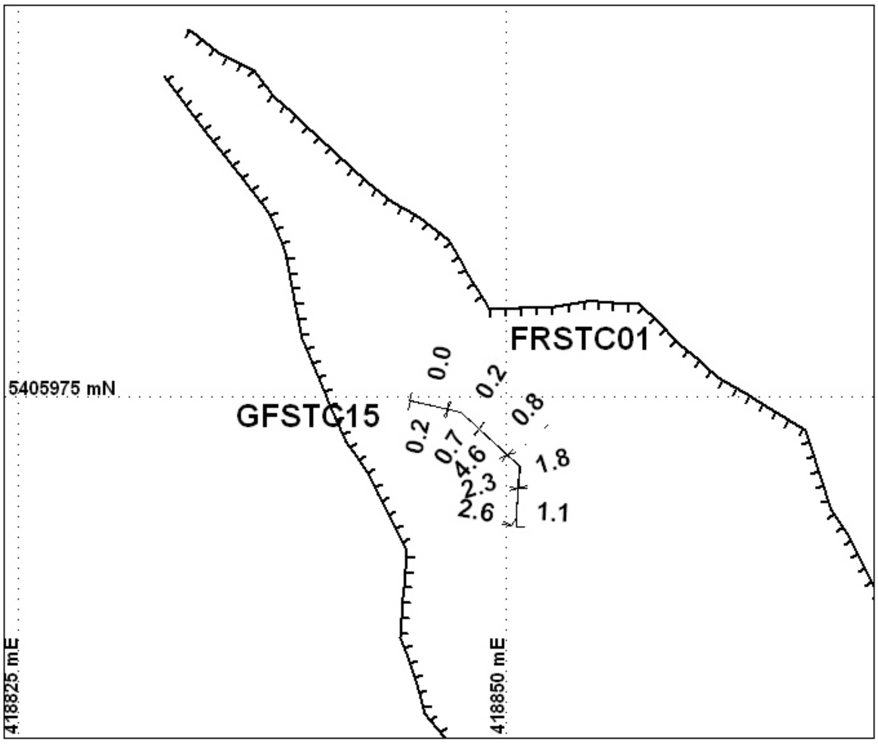

Figure 4.2: Location and assays of channel field duplicates

There has been no umpire re-assaying of residual samples by other laboratories in any of the sampling programmes.

Laboratory repeats have been at the lower end of standard practice with most batches re-assayed at ~1/20 and with no focus on high grade samples.

Table 4.4 Quality control repeats and standards

Sampling programme	Laboratory	Repeats	Standards reported
GFEL channels	Analabs, Burnie	No	no
RGC SD1 to SD6	Analabs, Burnie	No	no
RGC SD7 to SD12	Analabs, Burnie	No	no
Goldstream/Titan SD30 to SD42	Amdel, Adelaide	~1/20 and high grade zone	no

Goldstream/Titan SD43 to SD45	Amdel, Adelaide	~1/20	No
Jervois ST01 to ST04	Analabs, Burnie	~1/10	No
Frontier SFD01 to SFD16	Burnie Research Laboratories	~1/20	in-house and Frontier
Frontier channels	Burnie Research Laboratories	~1/20	in-house and Frontier

The exception to this is the re-assaying of almost all of the high grade samples in SD36 and SD39 by Amdel for Goldstream/Titan. The raw data from these re-assays is presented in table 4.5 and show good repeatability

Table 4.5 Au repeat assays SD36 and SD39

Hole ID	From	To	primary	repeat	2nd repeat
SD36	0	2.2	4.19	3.72	
SD36	5.2	6.7	0.45	0.05	
SD36	10.2	11.2	0.87	0.79	
SD36	11.2	12.7	0.59	0.58	0.64
SD36	12.7	14.2	0.12	0.1	
SD36	14.2	15.7	4.19	3.44	
SD36	15.7	16.7	15.3	14.5	
SD36	16.7	17.7	0.48	0.48	
SD36	17.7	18.7	0.72	0.76	
SD39	0	1.6	2.51	3.15	
SD39	1.6	3.1	0.77	0.76	
SD39	5.6	6.6	1.24	1.28	
SD39	7.6	8.6	1.57	1.75	
SD39	8.6	9.6	0.84	0.78	
SD39	9.6	10.6	1.27	1.17	
SD39	10.6	11.6	4.61	4.45	
SD39	11.6	12.6	3.11	2.84	
SD39	15.6	16.6	18.9	19.2	
SD39	16.6	17.6	14.6	13.9	13
SD39	17.6	18.6	3	2.67	
SD39	18.6	19.6	2.31	2.39	

Details regarding laboratory standards were made available by Burnie Research Laboratories. Frontier also submitted their own standard.

Table 4.6 Burnie Research Labs in-house and Frontier Au standards

Standard	Expected		Actual		
	assay	std. dvn.	number	ave. assay	std.dvn
BRL ST04	4.53	0.038	5	4.53	0.042
BRL ST05	2.33	0.017	5	2.33	0.019
BRL ST06	1.05	0.013	7	1.05	0.015
BRL ST10	3.86	0.015	5	3.86	0.016
BRL ST16	0.51	0.011	8	0.51	0.012
Frontier G905-6	5.96	0.26	26	5.34	0.020

The surprising result is that of Frontiers standard of which all assays were markedly less than the expected value (max. was 5.90 g/t) with 26 assays of the standard in 4 batches averaging 5.34 g/t Au with a standard deviation of 0.2g/t. This compares with Burnie Research Laboratories standards which perhaps unsurprisingly matched the expected values very closely (see table 4.6).

This discrepancy is a major concern as given the predominance of Frontier samples in the database used in the estimation, if the Burnie Research Laboratories consistently undercalled the Frontier assays by the 10.5% indicated from the difference in Frontier's standards results this would see the grade of the resource increase by something of the order (only estimated roughly) 7% to 9%.

It is recommended that umpire assays of Frontier residues be sent to another laboratory and it may well be worth re-assaying all samples which make up the high grade resource.

4.7 Bulk density

Frontier submitted 227 samples of drill core for bulk density determination by the water immersion method. Of the 84 are from the high grade zone.

5.0 Data verification

5.1 Assessment of quality control data

Due to the relatively low number of repeat assays in the Frontier database and the inferred status of the resource no in-depth statistical analysis has been performed on the data/

5.2 Assessment of project database

The project database supplied by Frontier was in a series of .csv files. A thorough assessment of this database was made with only a few relatively minor errors recognised. These .csv files were imported into an MS ACCESS database *Stormont4.mdb.* This database was mapped into SURPAC and checked for internal inconsistencies using SURPAC's database verification function.

5.3 Data quality summary

With the above caveat regarding the potential to increase the grade of the resource significantly by re-assaying the Burnie Research Laboratory samples the assay, sample recovery geological interpretation and drillhole/channel survey data is considered to be of sufficient quality to allow the estimation of an inferred resource. Perhaps the principal underpinning of this level of confidence is the disseminated nature of both Au and Bi mineralisation indicated by a nugget effect of 25% (see section 8.2). Upgrading this resource estimate indicated would require the following work

- **Resolution of the Frontier standard discrepancy by umpire re-assaying**
- More field duplicates particularly in the old workings where access is still available
- Field duplicates from Frontier and earlier drilled holes
- A greater percentage of in-house repeats

6.0 Geological interpretation and modelling

6.1 Methodology

3D modelling was carried out using both the 2D sectional approach and 3D on-the-fly approach using the same data set.

Interpretative linework was generated 2D Discover/MapInfo sections showing geology, Au and Bi grade and some limited structural information. These sections were generated at 25m spacings as per Frontier's 045º-225º (AMG) grid (lines 1975E to 3000E) with +/-12.5m search envelopes. This 2D linework was exported via .dxf files format into 3D SURPAC where the linework was rotated and transformed to its correct position in AMG AGD66 space. These 2D lines commonly carry an inherent inaccuracy due to the projection of drillholes up to +/-12.5 m away onto these idealised 25m sections.

The same geological and grade data as displayed on the 2D generated sections was displayed in 3D using SURPAC's Display Drillholes function with the same contacts able to be snapped onto directly on-the-fly in true 3D space.

The combination of sectional linework (now in 3D space) and point data snapped directly onto drillholes was converted into a 3D triangulated surface .dtm using SURPAC's CREATE DTM function. Visual discrepancies in the resultant .dtm shape between the sectionally generated linework and point data were corrected by moving individual points on sectional linework up or down (i.e. in z direction) or less commonly sideways in the x direction in order to generate a more geologically reasonable shape (all instances as a smoothed shape) but retaining the honouring of the point data snapped directly onto drillholes.

Geological shapes were completed by generating lines of intersection between cross-cutting geological features and incorporating these lines into the completed .dtm. The high grade mineralised 3DM used for resource estimation was created by bringing together the .dtm's of the relevant bounding surfaces (i.e. surface, base of high grade mineralisation, Stormont Thrust etc. as detailed in section 3.3)

6.2 Surface DTM

The surface .dtm was generated from a combination of 3D points supplied by Frontier from their GPS surveying of topography and drill collars particularly around the old open cut and spur immediately to its south, and the digital topographic contours obtained from Frontier but originating from the Tasmanian government's Lands Departments mapping branch.

6.3 Mineralisation domain modelling

Initially a 3D solid (3DM) was constructed containing all massive skarn mineralisation in the central zone from the base of the limestone to the top of the "transition beds" and extending from the northernmost end of the open cut to as far south as ST03 was generated. Initial geostatistics showed that using this all encompassing 3DM would result in significant downgrading of this high grade zone.

Visual inspection of 2D sectional data (see figures 3.4 to 3.14) and colour coded high grade sections in 3D shows that there is clearly a high grade Au zone in the northern part of the central zone lying in the stratigraphically upper part of the skarn immediately beneath the unskarnified limestone around the keel of the syncline though extending up both limbs in the middle part. Other higher grade Au intersections to the south of this northern high grade zone appear more discrete.

Visual inspection as well as only a moderate correlation coefficient between Au and Bi strongly argue for the resource to be seen as principally a Au deposit with Bi and some Ag credits. Modelling of the resource was based upon this assumption with the high grade zone defined by high gold grades.

A 3DM was then generated around the northern high grade zone again via the methodology detailed in section 6.1. This 3DM was used to generate the estimated resource.

This high grade zone "dugout canoe" shaped 3DM is constrained on its upper surface for the most part by the surface topography. Between 5405935mN and 5405864mN the middle of the DTM's upper margin is constrained by unskarnified limestone with the very southernmost part of the upper surface constrained by the base of the Tertiary sediments and basalt.

The bottom and sides of the 3DM are grade constrained for the most part with a nominal cut-off of 1g/t Au. Bi has not been used to constrain the high grade zone. The southeastern margin is constrained by low grade intersections in SFD014 on section 2150E. The exception to the grade constraint on the sides of the 3DM is the central portion of the northeastern side between 5405900mN and 5405965mN where the Stormont thrust fault acts as a hard boundary. Mineralisation northeast of this fault has apparently been displaced vertically and subsequently eroded.

6.4 Validation of geological interpretation and wireframe models

Validation of the geological interpretation and wireframe has been undertaken visually in 3D as well as on sections in figures 3.4 to 3.10 and plan 3.3.

7.0 Statistical analysis

7.1 Introduction

Statistical analysis was undertaken based on composited datasets of the gold, silver and bismuth assays. The activities completed in this phase of the study were as follows:-

- Compositing of the drillhole data to lengths within the coded mineralisation interval.
- Compilation of descriptive statistics and histogram plots of the composite gold, silver and bismuth datasets..
- Outlier grade analysis and determination of upper cuts.
- Assessment of data clustering and calculation of de-clustered grade statistics.

7.2 Data coding

The wireframe model of the mineralised high grade domain has been used to assign a code into the drillhole database to allow assessment of the variation in grade in the domain. The coding applied to the database is summarised in Table 7.1.

Table 7.1 Domain coding -					
Domain		**Wireframe**		**Variable**	
Type	**Description**	**Name**	**Type**	**Name**	**Code**
High Grade Mineralisation	Inside high grade domain	Stormont.dtm	solid	Ore	100

The domain coding assigned to the drillholes was visually compared with the corresponding wireframe boundaries in cross section and plan views to ensure all coding was robust.

7.3 Compositing

The drillhole database coded within each interpreted domain was composited as a means of achieving a uniform sample support. It should be noted, however, that equalising sample length is not the only criteria for standardising sample support. Factors such as angle of intersection of the sampling to mineralisation, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all affect the 'support' of a sample. Exploration/mining databases which contain multiple sample types and/or sources of data provide challenges in generating composite data with equalised sample support, and uniform support is frequently difficult to achieve.

A regular 1m run length (down hole) composite was chosen as the majority of samples (~65%) were collected over intervals between 0.5m and 1m (Figure 7.1). Any composites less than 0.5m in length were added to the previous composite to ensure the inclusion of the maximum amount of data possible. The distribution of composite lengths is displayed in Figure 7.2. The impact of 25% of sample lengths greater than 2m being split was considered to be minimal. Compositing to 2m intervals would dramatically decrease the number of composites, making it difficult to undertake statistical and spatial analyses.

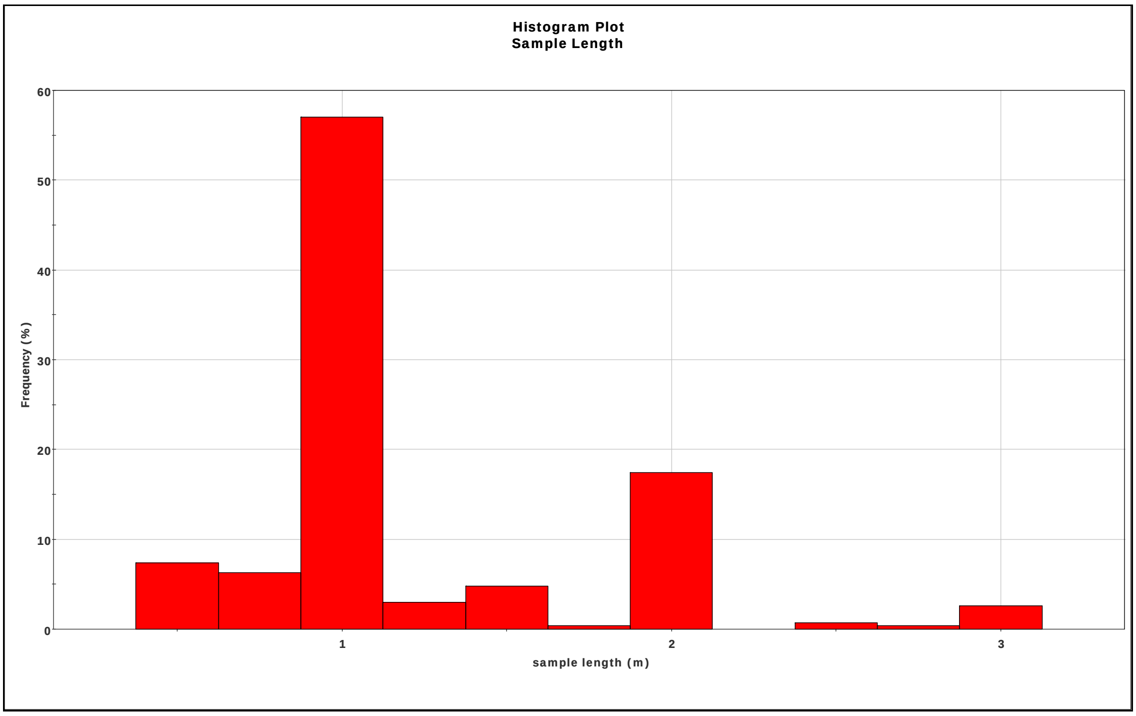

Figure 7.1: Histogram plot – Sample Length

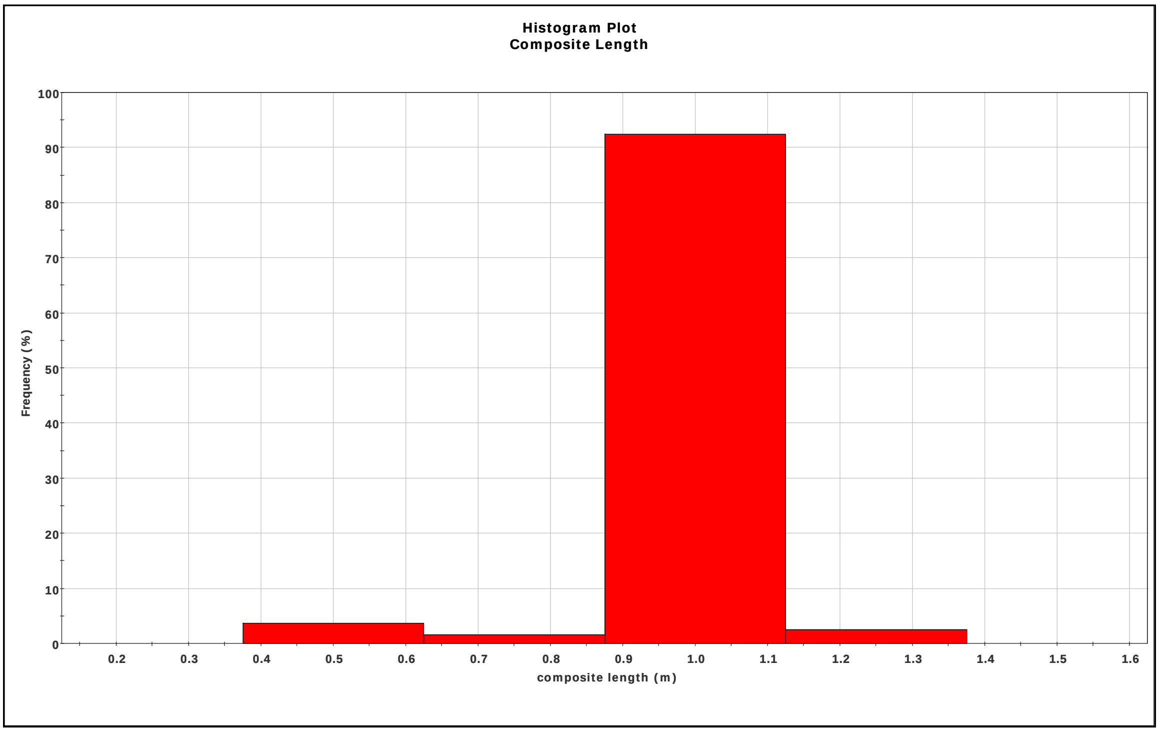

Figure 7.2: Histogram plot – Composite Length

7.4 Statistical analysis of composite data

Detailed statistical analysis of the gold, silver and bismuth composite data was conducted for all composites..

Descriptive statistics for the composites are presented in Table 7.2. The summary charts of these datasets indicate that they all form positively skewed distributions. Individual histograms, log histogram plots and probability plots for each element are presented in Appendix B. The log histograms and probability plots for each element indicate that there may be more than one population of grade in the mineralised domain. However, the relatively low number of composites and irregular data spacing preclude any further domain subdivisions.

Table 7.2 Summary Composite Statistics

Mineralised Domain	Au (ppm)	Bi (ppm)	Ag (ppm)
Count	327	324	275
Minimum	0.01	6.50	0.05
Maximum	48.47	32060	31.00
Mean	4.86	3192.22	4.95
Median	1.78	1501.65	3.50
Standard Deviation	7.48	4655.18	4.40
Coefficient of Variation	1.54	1.46	0.089

7.5 Assessment of upper cuts

Assessment of the composite outliers was completed to determine the requirement for high grade cutting (high grade cuts) for each of the input datasets to be used for resource estimation. The approach taken to the assessment of the high grade composites and potential outliers is summarised as follows:-

- Detailed review of histograms and probability plots of reef composites, with significant breaks in populations used to interpret possible outliers.
- Investigation of clustering of the higher-grade data. High grade data that are clustered were considered to be real while high grade composites not clustered with other high grade data were considered to be possible outliers, requiring further consideration via cutting.

- The ranking of the composite data and the investigation of the influence of individual composites on the mean and standard deviation (mean versus standard-deviation plots). Plots of all datasets accompany the report in Appendix B.

Following the compositing of the sample data a series of high grade cuts or caps were determined as presented in Table 7.3. The upper cuts as applied results in a reduction in mean grades of between 1% and 7% for the three elements.

Table 7.3 Composite Outlier Analysis

Element	No. of Data	Raw Data				Cut Data				No. of Data	Mean %
		Max.	Mean	Std. Dvn.	C.V.	Upper Cut	Mean	Std. Dvn.	C.V.	Cut	Decrease
Au	327	48.47	4.86	7.48	1.54	25	4.53	6.09	1.34	9	7%
Bi	324	32060	3192	4655	1.46	15000	2955	3666	1.24	12	7%
Ag	275	31	4.95	4.4	0.89	22	4.91	4.23	0.86	1	1%

7.6 De-clustered statistics

Cell de-clustering has been undertaken to assess the effects of the data clustering on the global mean grade. Clustering of high grade data is apparent from the sampling of the historical workings. Table 7.4 presents a comparison of the naïve and de-clustered mean grades for each element. It is evident that de-clustering results in considerably reduced mean grades for each element.

Table 7.4 Comparison of Naïve and De-clustered Composite Datasets

Element	Naïve Mean Grade	Cell size (m) Y x X x Z	De-clustered mean grade	% difference
Au	4.86	30m x 30m x 5m	3.41	30%
Bi	3192	30m x 30m x 5m	1936	39%
Ag	4.95	30m x 30m x 5m	3.78	24%

7.7 Correlation analysis

Bivariate analysis was completed on the uncut data between gold, silver and bismuth data located within the modelled mineralised domain. There is a moderate correlation between all three elements (Table 7.5).

Table 7.5 Correlation matrix correlation coefficient

	Au	Ag
Ag	0.34	
Bi	0.39	0.64

7.8 Bulk Density statistical analysis

Frontier 84 samples averaged 2.9 g/cm^3 with a maxima of 3.91, minima of 1.28 and standard deviation of 0.51. There is not a strong relationship between bulk density and depth. Bulk density has not been estimated into the block model but rather a single value of 2.9 g/cm^3 has been used.

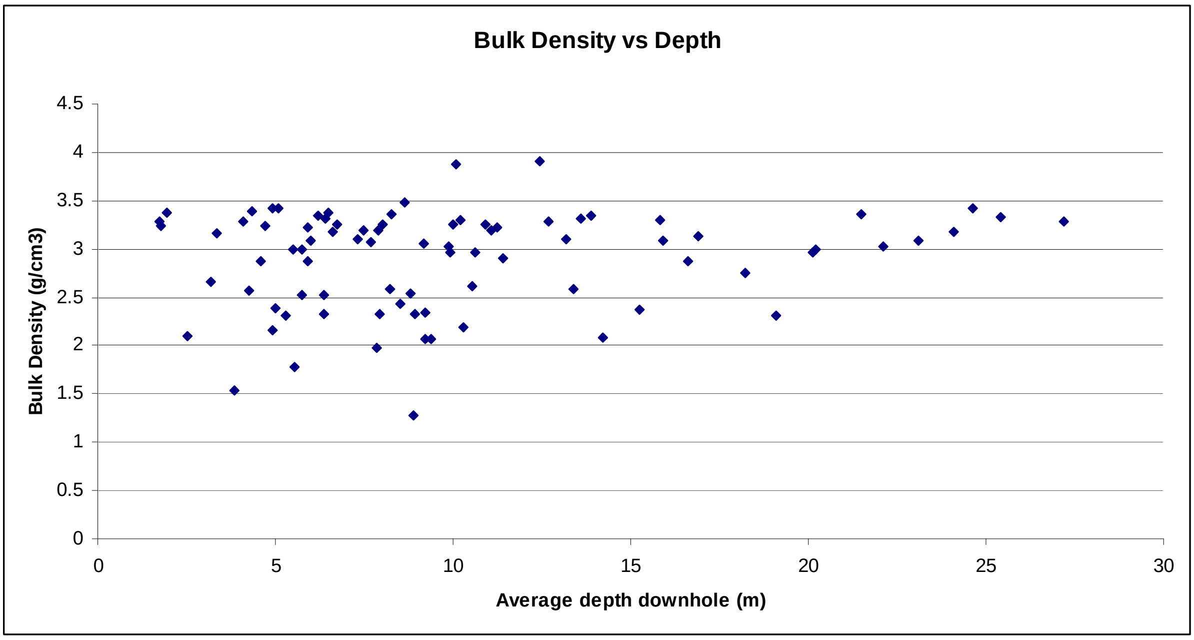

Figure 7.3: Relationship between bulk density and depth

8.0 Variography

8.1 Introduction

Detailed grade variography was generated and modelled for the Stormont Deposit in preparation for grade estimation. The variography was completed based on the uncut 1m downhole composites.

Variography is used to describe the spatial correlation (co-variance) between data points within an interpreted zone for a nominated distance or lag. All data points within the zone are compared at nominated lag distances with the average squared difference of the two sample points obtained. The averaged squared difference of the data point's gamma (h), for each lag distance, is plotted on an X-Y graph. The variogram displays the lag distance (h) on the X-axis and the average squared differences (gamma value) for the nominated lag distance as the Y-axis. This calculated graph is called an experimental semi-variogram. It should be noted that in the text semi-variograms will be referred to as variograms.

Variography can be undertaken in many forms depending on the aims of the study. The variography for the Stormont Deposit has been completed based directly on the composite data values using a normalised spatial measure known as the correlogram. However, in this document, the term “variogram” is used as a generic word to designate the function characterising the variability of variables versus the distance between two samples.

A series of mathematical models are fitted to the experimental variography which, when used in the Kriging algorithm, will recreate the spatial continuity observed in the modelled variography.

A standard approach was used to generate and model the variography for each envelope. The steps taken are summarised below:-

- Generate and interpret a 3D gridded variance map to aid in the determination of the major, semi-major and minor axes of continuity.
- Generate and model the downhole direction variogram, which allows the determination of the nugget effect (close spaced variability).
- Calculate and model the major, semi major and minor axes of continuity.

The variography was calculated and modelled using the Isatis geostatistical software package.

All of the captured variography was modelled with a nugget effect, and 2 spherical structures representing the larger scale spatial variability of the datasets.

8.2 Grade variography

Detailed variography was completed for the 1m composites for Au, Ag and Bi, coded within the interpreted mineralisation domain.

The direction of maximum continuity for Au, Ag and Bi is horizontal and approximates the strike of the mineralised domain.

A visual representation of the direction of maximum continuity and overall ranges is displayed in Figure 10.1.

The modelled variography for Au, Ag and Bi display very similar sills and ranges, therefore the variogram model for Au was adopted for all three elements. The variogram model displays a high level of short scale variability that is comprised of a moderate (25%) relative nugget. The variogram model is dominated by a short range structure that accounts for 72% of the total variance including nugget effect, with a range of 28m. The overall range is 55m. The semi-major and minor axes display equivalent ranges for both structures, with 5m for the short range and 14m for the overall range.

The fitted variogram model is presented in Table 8.1, while the variogram plot is included in Appendix C.

Table 8.1 Summary Au Variogram Model

Rotation (SURPAC)			Nugget	Sill	Range (m)			Sill	Range (m)		
Z	X	Y	(C0)	(C1)	major	semi-major	minor	(C2)	major	semi-major	minor
135	0	0	0.25	0.47	28	5	5	0.28	55	14	14

9.0 Block modelling

9.1 Introduction

A three dimensional block model was constructed using Surpac mining software. The block model contains sufficient variables to record the results of Ordinary Kriging (OK) grade estimates and other required parameters.

9.2 Block construction parameters

The block model was constructed using appropriate three dimensional extents encompassing the modelled mineralised domain. Parent block dimensions were selected based on both the data spacing and mine planning considerations, and sub-block dimensions were chosen to enable accurate reproduction of the wireframe volumes of the mineralisation domain. The coordinate extents of the block model and the dimensions are summarised in Table 9.1.

Table 9.1 Block model dimensions

	Model origin co-ordinates	Extent (m)	Number of blocks	Block size	
				Parent	Sub-block
East	418840	110	11	10	1.25
North	5405850	130	13	10	1.25
Elevation	610	40	8	5	1.25

The wireframed topographic surface and mineralisation domain have been coded to the block model. Table 9.2 displays a listing of the variables in the Stormont block model. Wireframe coding incorporated into the model is summarised in Table 9.3.

Table 9.2 Block Model Variables

Variable	Description
au	Estimated Au ppm
ag	Estimated Ag ppm
bi	Estimated Bi ppm
au_equiv	Calculated Au equivalent (Au + 0.01627Ag + 0.0002Bi)
dis	Distance to nearest sample used in Au estimation

avdis	Average distance to samples used in Au estimation
kv	Kriging variance
matl	Material (1=mineralised, 2=non-mineralised, 3=air)
numsamp	Number of samples used in estimation
pass	Estimation pass number (Au)

Table 9.3 Block Model Coding				
Variable	**Code**	**Constraint**	**Wireframes**	**Description**
matl	1	Inside	Stormont.dtm	High grade mineralisation
matl	2	Outside	Stormont.dtm	Non-mineralised
matl	3	Above	new_stromont_topo_clean_3.dtm	Air

9.3 Validation

The block model has been extensively validated against the mineralisation wireframe. The model has been validated by viewing in multiple orientations using the 3-D viewing tools in Surpac. Based on the visual review the block model was considered a robust representation of the interpreted mineralisation.

10.0 Grade estimation

10.1 Introduction

Resource estimation for the Stormont deposit was undertaken using Ordinary Kriging (OK) as the principal estimation methodology for gold, silver and bismuth.

10.2 Ordinary kriging

The grade interpolation for this exercise is based on Ordinary Kriging (OK), one of the more common geostatistical methods for estimating the block grade. In this interpolation technique, contributing composite samples are identified using a search volume applied from the centre of each block. Weights are determined so as to minimise the error variance considering both the spatial location of the selected composites and the modelled variogram. Variography describes the correlation between composite samples as a function of distance and direction. The weighted composite sample grades are then combined to generate a block estimate and variance.

10.3 Search neighbourhood

Search ellipse orientation and radii, as well as minimum and optimum number of samples were determined based on variogram orientation, variogram model anisotropy and ranges, horizon geometry and data distribution. Figure 10.1 displays the search ellipse in relation to the mineralised domain.

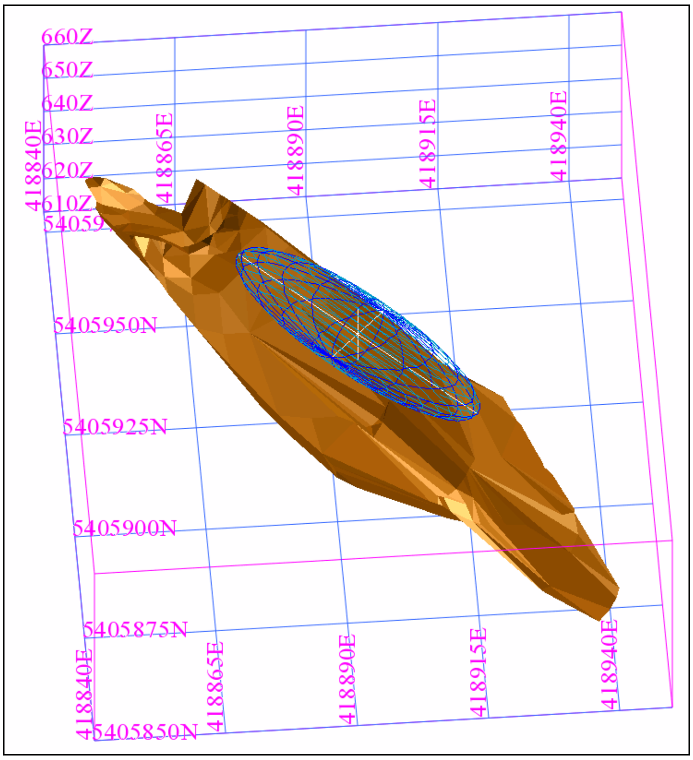

Figure 10.1: Search ellipse orientation

10.4 Grade estimation

Estimation used the variogram model parameters determined from grade variography, as discussed in Section 8.0.

OK estimates were completed using an optimised whole block discretisation of 4 points in the east-west dimension, 4 points in the north-south dimension, and 1 points in the vertical dimension for a total of 16 discretisation points per whole block estimate. Any sub-blocks within the 3-D limit of each whole block were assigned the whole block OK estimates.

A multiple search strategy was applied in obtaining the estimates. Table 10.1 provides the sample search parameters applied for each pass.

Domain control was used for both the input composite data and block selections.

The OK estimates were completed using Surpac mining software. In estimating the grade, the standard fields relating to the search neighbourhood used, number of composites selected, the distance to the nearest composite, the average distance of composites and the kriging variance were recorded. No change of support has been applied.

Table 10.1 Summary of Sample Search Parameters

Estimation Pass	Sample Search Orientation			Sample Search Distance (m)			Samples		
	Major	Semi-major	Minor	Major	Semi-major	Minor	Min.	Max.	Max per drillhole
1	135	0	0	30	8	8	12	30	5
2	135	0	0	60	16	16	12	30	5
3	135	0	0	120	32	32	12	39	5

10.5 Block model files

The resultant grade estimates are held in the model file *Stormont.mdl.*

10.6 Validation

The resulting estimates were extensively validated by visual and statistical comparison of block estimates against the source composite date for each estimated domain. Detailed visual estimation was also undertaken in multiple section views (cross section, long section and plan). A representative cross section through the block model is displayed in Figure 10.2.

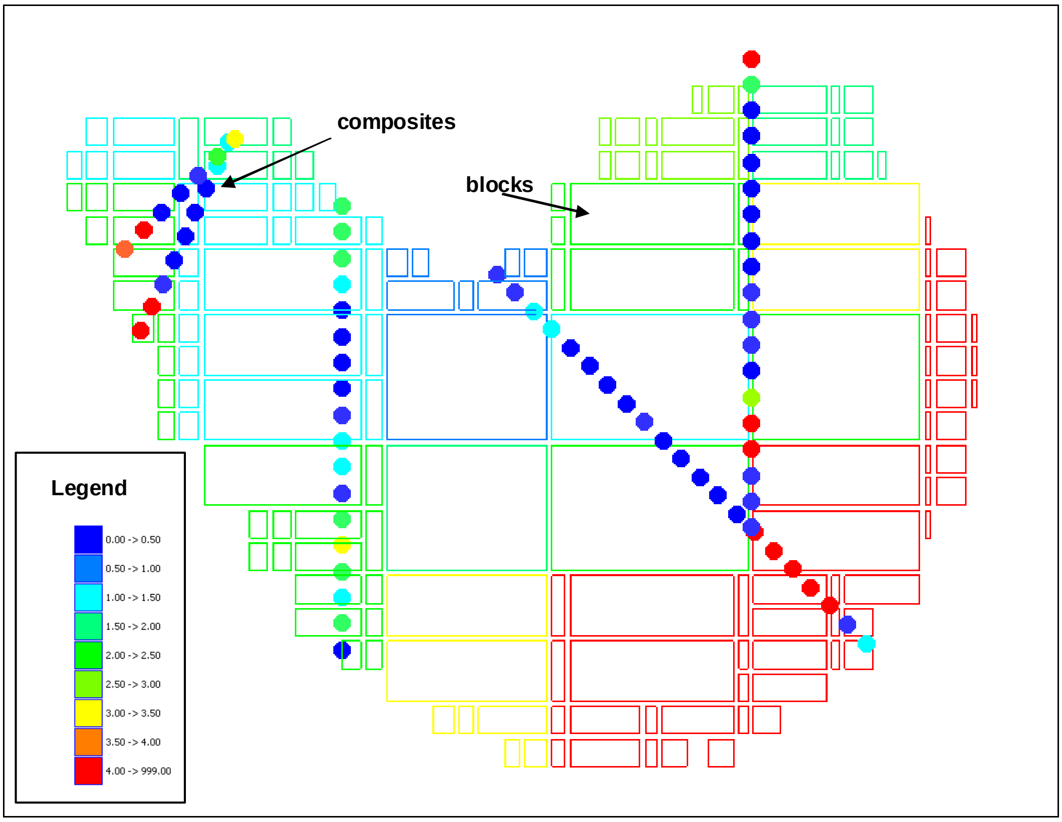

Figure 10.2: Cross-section Block Model and Composite Grade Comparison

Further validation of the estimate was made by comparison with the mean grades over 10m thick northing slices and 5m thick elevation slices (Figures 10.3 and 10.4).

The analysis clearly demonstrates that the grade variability in composites is greater than that of grade estimates. The directional trends observed in composites are more or less reproduced within the block estimates. Acceptable levels of reproducibility are noted between the input composites data and the block estimates on the basis of visual review. On this basis and the other validation checks, it is considered that the OK whole block estimates are appropriate and robust.

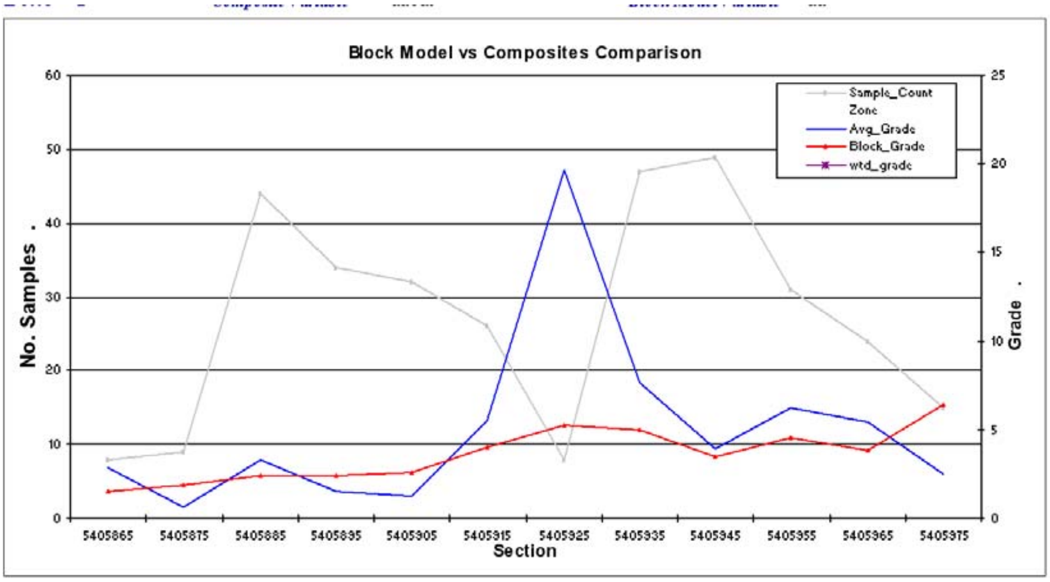

Figure 10.3: Stacked Northing Transects – Block Model and Composite Grade Comparison

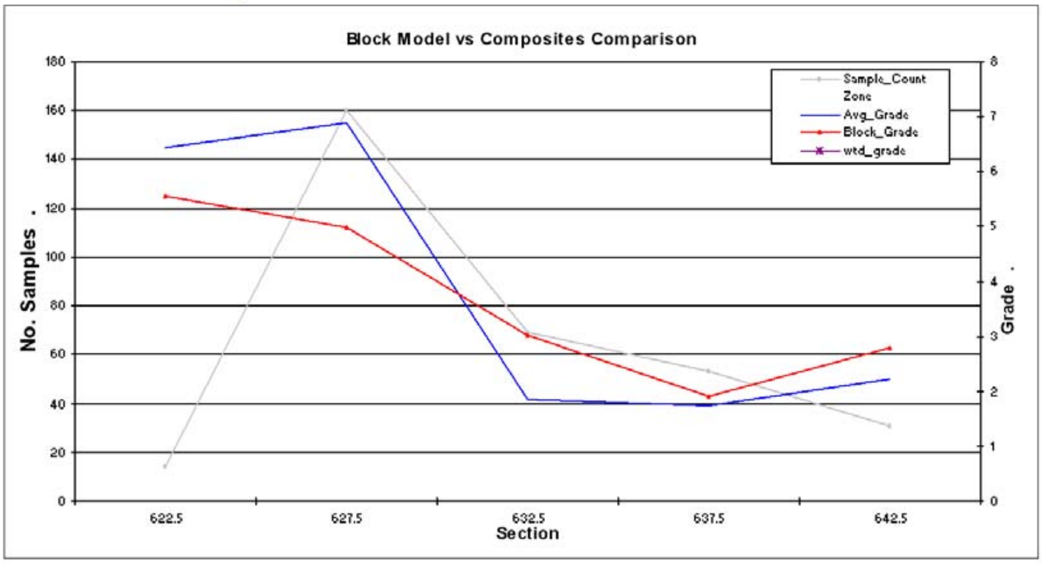

Figure 10.4: Stacked Elevation Transects – Block Model and Composite Grade Comparison

10.7 Resource reporting

The resource estimate for the Stormont Deposit has been classified as Inferred Mineral Resources in accordance with guidelines as set out in the Joint Ore Reserves Committee (JORC) Code (2004). The Resource category has been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the JORC Code categorisation guidelines.

The resource categorisation has been based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation.
- Variogram models and the ranges of the first structure in multi-structure models.
- Drilling density.
- Estimation statistics

The confidence levels of the key criteria that were considered during resource classification are presented in Table 10.2.

Table 10.2 Key Criteria Confidence Levels

Item	Discussion	Confidence
Drilling/channelling techniques	Industry standard diamond drilling and channels	high
Logging	Capable geologists with generally consistent approach.	moderate/high
	Limited structure detail and no magnetic susceptibilies	
Drill sample recovery	Generally good with exception of top 4.5m of SD1	moderate/high
Sub-sampling techniques and sample prep.	Industry standard	high
Quality of assay data	Major concerns regarding Frontiers assays potentially undercalled by 10.5%	low
Verification of sampling and assaying	Limited duplicates	moderate/low
Location of sampling points	Problem with collar positions particularly RLs	moderate/low
Data density and distribution	Reasonable for resource category	moderate
Database integrity	Quite thorough audit in this work	high
Geological interpretation	Geology sufficiently well understood for inferred resource at this scale	moderate/high
Estimation and modelling techniques	OK with well defined variography	high
Mining factors or assumptions	not applicable	N/A
Tonnage factors	Numerous measurements used as overall average	moderate/high

10.8 Grade tonnage report

The Mineral Resource, reported at various lower cutoff grades, as of April 2009 is presented in Table 10.3. The grade tonnage curve is presented in Figure 10.5.

Table 12 Mineral Resource Summary - April 2009 Ordinary Kriging Grade Estimates Subdivided by Lower Cut-off Grade					
Lower Cut-off Grade Au (g/t)	**Resource Category**	**Tonnes (t)**	**Au (g/t)**	**Bi (ppm)**	**Ag (g/t)**
0.5	Inferred	124300	3.65	2588	3.35
1	Inferred	112500	3.94	2718	3.41
1.5	Inferred	91400	4.57	3037	3.52
2	Inferred	75500	5.16	3175	3.32
2.5	Inferred	63200	5.72	3414	3.38
3	Inferred	54400	6.22	3531	3.39
3.5	Inferred	50800	6.43	3609	3.34

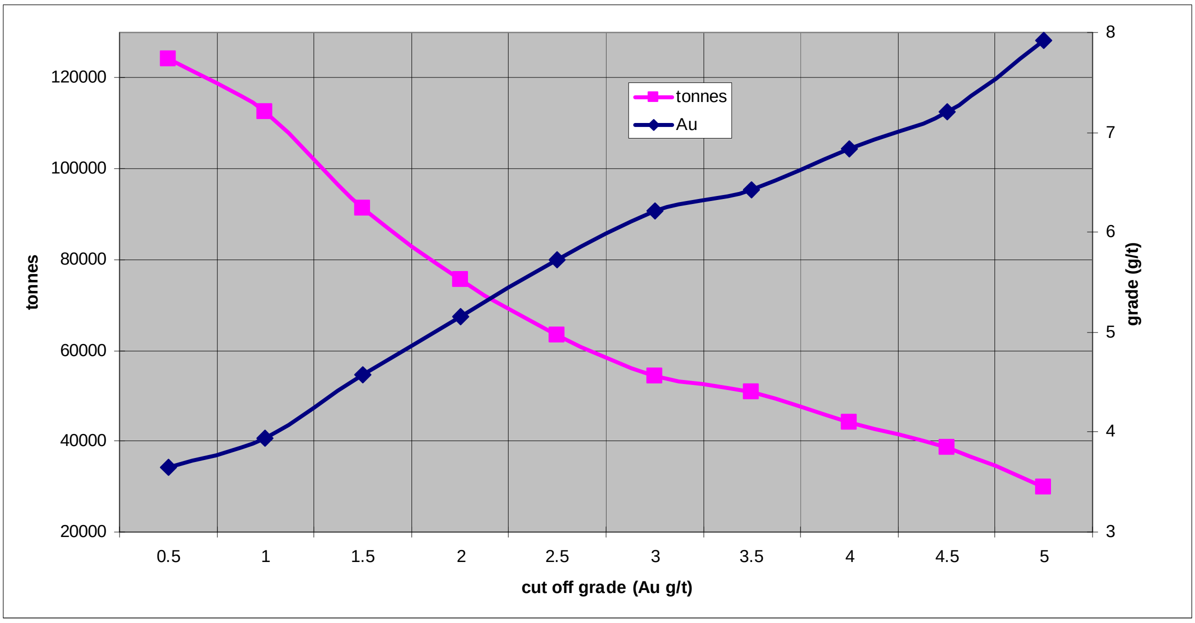

Figure 10.5: Grade Tonnage Curves

11.0 Conclusions and Recommendations

Whilst significant high grade (gold+/-bismuth intersections have been made over the entire length of the central zone (corresponding to the central syncline), a consistently mineralised resource can only be modelled with sufficient geological rigour in the area modelled as the high grade zone. Potential exists to extend this resource to the southeast as evidenced by high grade intersections in Frontier drillholes SFD008 and SFD010 but this would require further drilling.

An **inferred resource** can be reasonably confidently estimated for this Stormont high grade zone. It is conceivable that the resource is understated by the under calling of Au assay results by potentially 10% by Burnie Research Laboratories. Umpire re-assaying with another laboratory could see the resource increased significantly.

Upgrading this high grade zone resource to indicated/measured status would require the following:

- Infill drilling on 12.5m sections with holes using the current fan geometry.
- Drilling angled holes towards the southwest on 12.5m spaced sections designed to intersect the Stormont Thrust before passing into the high grade zone on the eastern limb of the syncline.
- Further more thorough channel sampling of the old workings with both walls of the underground workings channel sampled and another tier of channel sampling along the walls of the open cut.
- Surface trenching again on 12.5m sections over the area of outcropping mineralisation.
- Umpire re-assaying of existing and new sampling.
- Conventional surveying of existing drillhole collar and channel sample locations including the old open cut and underground workings.

- Creation of a more accurate surface DTM
- More thorough relogging of existing drill core and mapping of exposures with foci on (1) mineral assemblage control an Au and Bi mineralisation, and (2) geological structure, both in order to understand any structural control on mineralisation and for geotechnical purposes. Given the possible relationship between retrograde magnetite and Au + Bi mineralisation magnetic susceptibilities should be measured for all drill core. The use of PIMA which may be able to map out retrograde actinolite, probably associated with Au + Bi mineralisation.

12.0 References

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Appendix A

List of abbreviations used in text

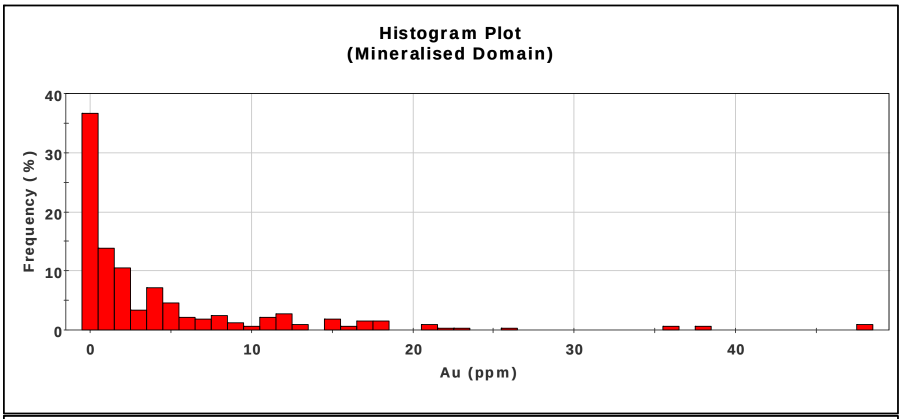
Abbreviation	Descriptions
2D	two dimensional
3D	three dimensional
3DM	SURPAC term to describe three dimensional solid
Ag	silver
AMG	Australian Map Grid
Au	gold
Bi	bismuth
Comalco	Comalco Limited
.dtm	SURPAC file type/term to describe three dimensional shape which is not a solid
DTM	Digital terrane model
Frontier	Frontier Resources Ltd
g/t	grams per tonne (is equivalent to ppm)
GFEL	Gold Fields Exploration Pty. Limited
Goldstream	Goldstream Mining N.L.
GPS	Global positioning system
HQ	diamond drill core size = 63.5mm diameter
Jervois	Jervois Mining N.L.
m	metre
m.a.s.l.	meters above sea level
NQ	diamond drill core size = 47.6mm diameter
NTW	diamond drill core size = 56.0mm diameter
OK	ordinary kriging
p.a.	per annum
ppm	parts per million (is equivalent to g/t)
RGC	Renison Goldfields Consolidated
RL	Relative level
Titan	Titan Resources N.L.
v.m.	vertical metre

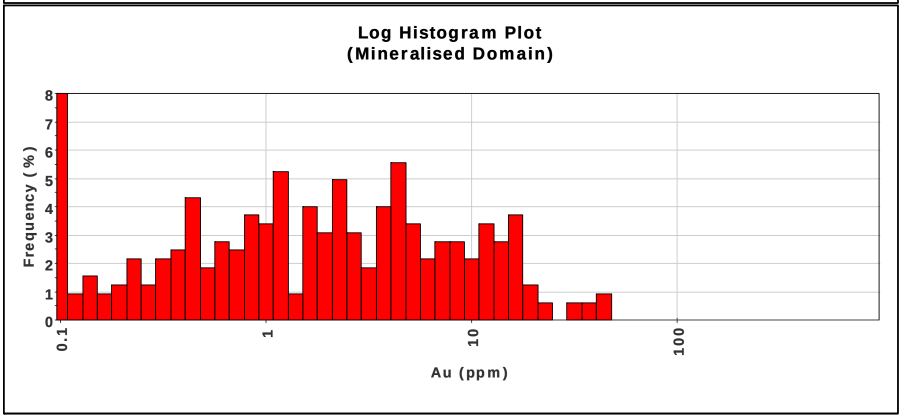
Appendix B

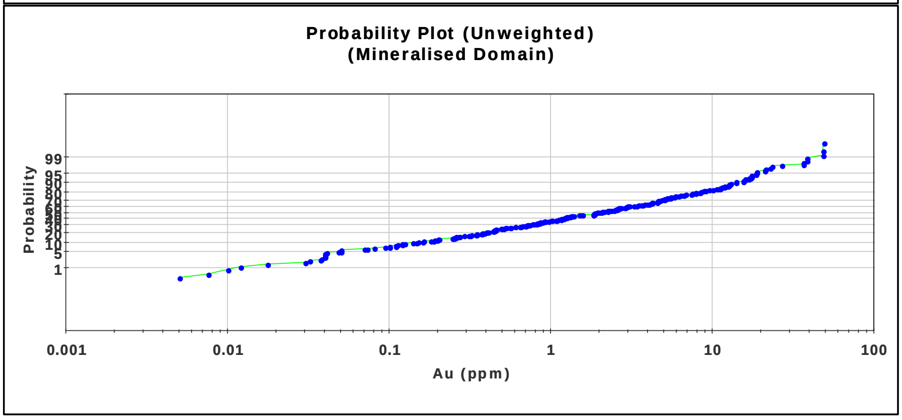
1m composite statistics

Frontier Resources - Stormont Deposit
Summary 1m composite statistics
(Mineralised Domain)

	Unweighted	Weighted	Units
Samples:	324	N/A	
Minimum:	0.01	N/A	ppm
Maximum:	48.79	N/A	ppm
Mean:	4.97	N/A	ppm
Median	1.90	N/A	ppm
Std. Deviation:	7.68	N/A	ppm
Coefficient of Variation:	1.55	N/A	







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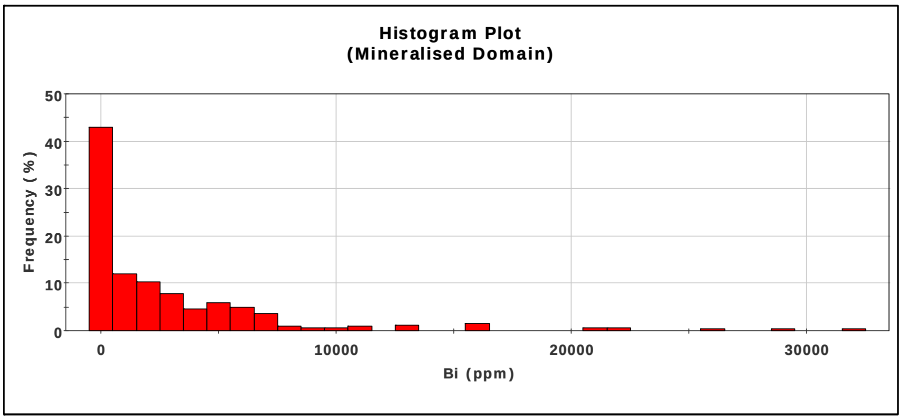
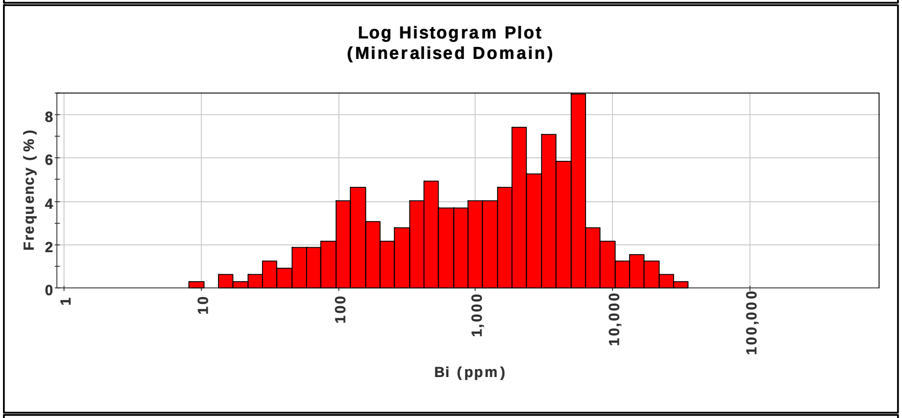
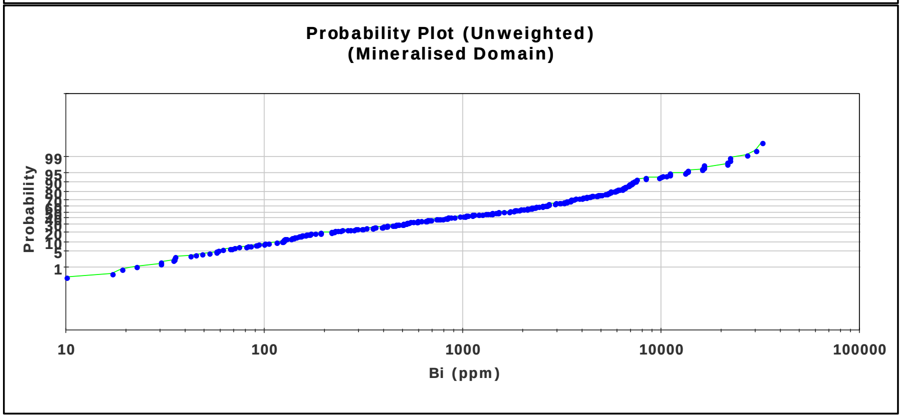
Frontier Resources - Stormont Deposit
Summary 1m composite statistics
(Mineralised Domain)

	Unweighted	Weighted	Units
Samples:	324	N/A	
Minimum:	10.00	N/A	ppm
Maximum:	32,060.00	N/A	ppm
Mean:	3,209.88	N/A	ppm
Median	1,501.65	N/A	ppm
Std. Deviation:	4,644.61	N/A	ppm
Coefficient of Variation:	1.45	N/A	







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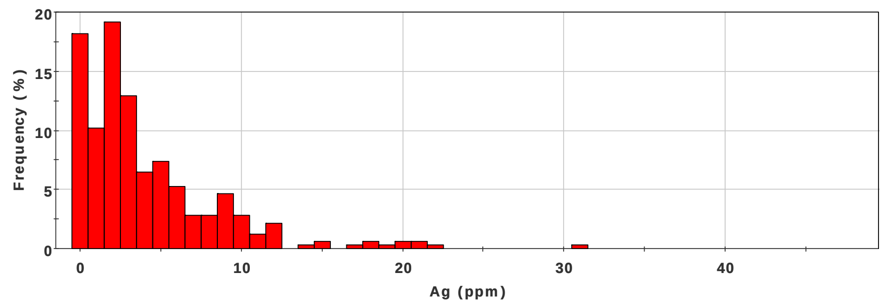
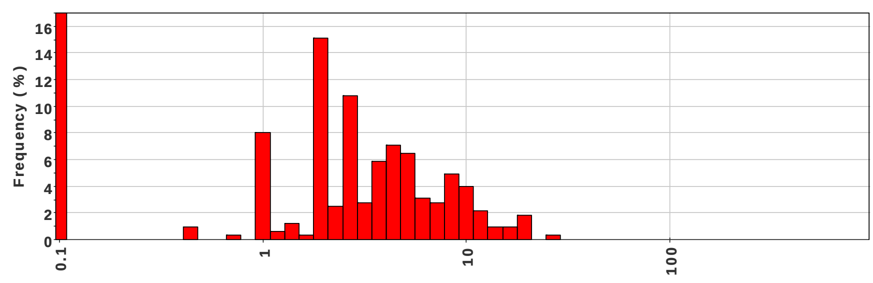
Frontier Resources - Stormont Deposit
Summary 1m composite statistics
(Mineralised Domain)

	Unweighted	Weighted	Units
Samples:	324	N/A	
Minimum:	0.00	N/A	ppm
Maximum:	31.00	N/A	ppm
Mean:	4.19	N/A	ppm
Median	3.00	N/A	ppm
Std. Deviation:	4.46	N/A	ppm
Coefficient of Variation:	1.06	N/A	

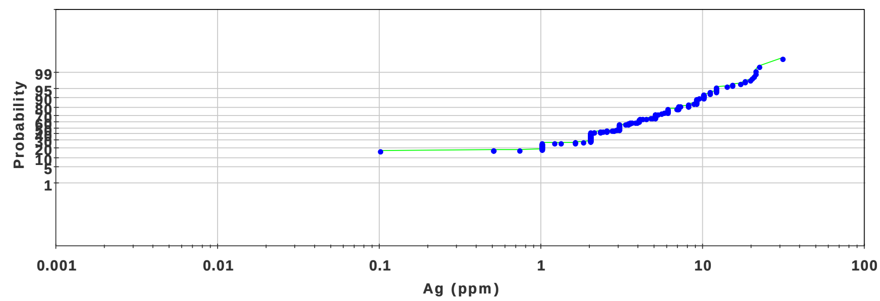
Histogram Plot
(Mineralised Domain)



Log Histogram Plot
(Mineralised Domain)



Probability Plot (Unweighted)
(Mineralised Domain)



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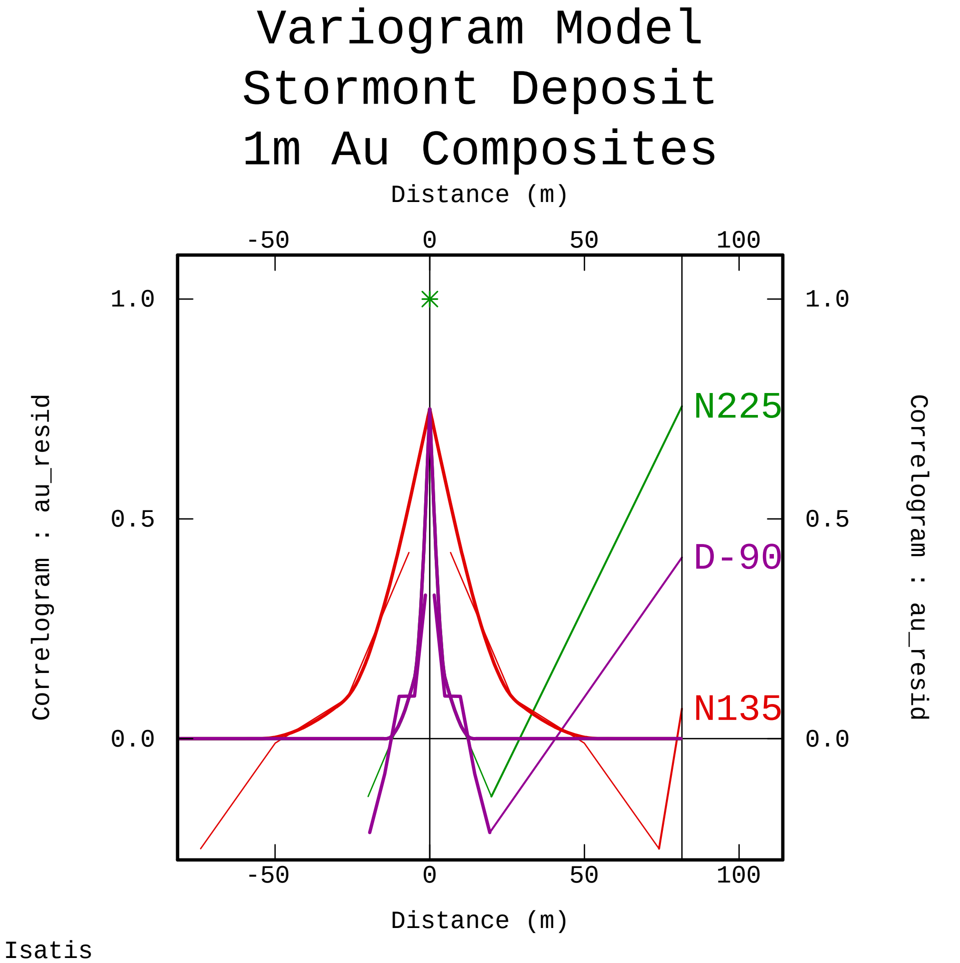
Appendix C (of FNT Annual Report Appendix 1)

Variogram model




```
points/1m comps au(au gt 0.5)
- Variable #1 : au_resid
Experimental Variogram : in 3 direction(s)
D1   : N135
       Angular tolerance =  15.00
       Lag =  25.00m, Count = 9 lags, Tolerance = 50.00%
       Horizontal Slicing =  10.00m
       Vertical Slicing =  10.00m
D2   : N225
       Angular tolerance =  15.00
       Lag =  25.00m, Count = 9 lags, Tolerance = 50.00%
       Horizontal Slicing =  10.00m
       Vertical Slicing =  10.00m
D3   : D-90
       Angular tolerance =  45.00
       Lag =  5.00m, Count = 11 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N135.00 (Geologist)
S1   - Nugget effect, Sill =       0.25
S2   - Spherical - Range = 5.00m, Sill =       0.47
       Directional Scales = (      28.00m,       5.00m,       5.00m)
S3   - Spherical - Range = 14.00m, Sill =       0.28
       Directional Scales = (      55.00m,      14.00m,      14.00m)
```

APPENDIX 2

SWAIN ENGINEERS

Consulting Mining Engineers

CONCEPTUAL MINING STUDY FOR THE NARRAWA AND STORMONT DEPOSITS
RETENTION LICENCES 3/2005 AND 4/2005 ; NORTH - CENTRAL TASMANIA

EXECUTIVE SUMMARY

Frontier have rights to explore and potentially develop Narrawa and Stormont Deposits located in RL3/2005 and RL4/2005 near Gowrie Park in North Central Tasmania. The RL's are valid on a renewable basis until 12 May 2009 and 6th August 2010. Exploration was completed at Stormont during 2008 and the following statement of an Inferred Resource has been issued shown in Table 1. (Grant MacDonald & Associates)

Table 1 : Stormont : Tonnage and Grade and possible Mineralised Resource

Classification*	COG	Tonnes	AuEq g/t	Au (g/t)	Bi (ppm)	Ag (g/t)
Inferred	0.5g/t	124300	3.88	3.65	2588	3.35
Inferred	1.0g/t	112500	4.18	3.94	2718	3.41
Inferred	1.5g/t	91400	4.83	4.57	3037	3.52
Inferred	2.0g/t	75500	5.43	5.16	3175	3.32
Inferred	2.5g/t	63200	6.01	5.72	3414	3.38
Inferred	3.0g/t	54400	6.52	6.22	3531	3.39
Inferred	3.5g/t	50800	6.73	6.43	3609	3.34

NB AuEq (g/t) = Au(g/t) + 0.00009 x ppm Bi + 0.01462 x g/t Ag

The Narrawa area shows many interesting sample results. Drill samples taken from the vicinity of the old open cut in early 2008 have enabled Frontier to state the Indicated and Inferred Resource (*Classifications per the JORC Code) shown in Table 2 (Geostat Pty. Ltd. Perth).

Table 2 : Narrawa Deposit : April 2009

Classification*	COG	Tonnes	AuEq g/t	Au g/t	Pb %	Zn %	Ag g/t
Indicated	0.0g/t	163,170	3.77	2.06	1.40	1.19	20.3
Inferred	0.0g/t	66,473	2.26	1.31	0.78	0.63	12.6
Total @	0.0g/t	229,643	3.33	1.84	1.22	1.03	18.1
Indicated	1.0g/t	159,690	3.83	2.09	1.43	1.21	20.6
Inferred	1.0g/t	39,220	3.23	2.07	0.93	0.78	15.5
Total @	1.0g/t	198,910	3.71	2.09	1.33	1.13	19.6

NB AuEq (g/t) = Au g/t + (Pb% x 0.56447) + (Zn% x 0.52113) + (Ag g/t x 0.01462)

Table 3 : Snapshot Metal Prices USD as at 3 July 2009

Gold	**$30.23/g**	**$940/oz Troy**
Zinc	**$0.7144/lb**	**$1,574.5/t**
Lead	**$0.7738/lb**	**$1,705.5/t**
Silver	**$0.4405/g**	**$13. 7/oz Troy**
Bismuth	**$5/lb**	**11,020/t**

The study has assessed the possible return on investment to shareholders of Frontier Resources Ltd. from exploitation of the Mineralised Resource shown in Table 1 and Table 2 at current metal prices shown in Table 3. The results using traditional process technology are shown in Table 4 including estimated Capital Investment in Table 5.

Also in Table 5, in blue, are the returns to the project of new technology which may be available at start up. These results are presented for comparison and if sustained in practice demonstrate advantages above the conventional process technology.

The advantage of the new technology is, if proven after a pilot plant study to the satisfaction of Frontier, that :-

1) All ores, whatever metals present, are processed simultaneously. In the case of the Narrawa & Stormont project, this means that by increasing throughput by 5tph to 45 tph, total ROM ore from each orebody is processed in one year of operations.

2) All metals contained in the ore can be selectively recovered. The metals are produced in a native state and are finely divided which we understand commands a better sale price in the Metal Markets. There are no Smelting and Refining costs in the traditional sense. Another advantage is the reduced power consumption for comminution of the ores.

3) As no reagents are used, Tailings are chemical free and harmless to the environment.

Please note that the cost structure for the possible new process is INDICATIVE at this time and is subject to revision. Similarly the value used for recovery has yet to be confirmed.

Table 4 : Summary of Net Cash Flow in AUD for the Inferred Mineralisation at Stormont and the Indicated and Inferred Resource at Narrawa before Tax for Discount 0% & 10%

Stormont First					
Quantity Mined & Processed			**Cash Operating Cost per Ounce**	**Discount 0%**	**Discount 10%**
Site	**In Situ t**	**ROM t**			
Stormont	**112.500**	**224,000**	**54.7**	**$14,000,000**	**$10,365,000**
Narrawa	**200,000**	**220,000**	**44.3**		
Narrawa First					
Narrawa	**200,000**	**220,000**	**44.3**	**$13,700,000**	**$10,400,000**
Stormont	**112,500**	**224,000**	**54.7**		
Narrawa & Stormont Material Processed Simultaneously					
	312,500	**344,000**	**40**	**$26,500,000**	**$20,700,000**

An assessment of Capital Investment is shown in Table 5.

Table 5: Estimated Capital Investment

Year -1	**Year 1**
Common Plant + Stormont CIP	**Narrawa Flotation**
$7,670,000	**$1,150,000**
Common Plant + Narrawa Flotation	**Stormont CIP**
$8,220,000	**$600,000**
Narrawa & Stormont	
Est. $8,760,000	**nil**

CONCLUSION

The philosophy of mining and processing at the Narrawa and Stormont Deposits is to adopt a simple approach utilising local workforce and contractors to mine and process the low tonnage Mineral Resources. The Conceptual Mining Study demonstrates the potential of a satisfactory investment, which will yield a future source of income to the shareholders of Frontier Resources Ltd.

OPERATING PARAMETERS : STORMONT & NARRAWA CREEK DEPOSITS

Mining parameters for Stormont and Narrawa Deposits are found in Table 6 and Table 7. Stormont will be mined and processed first as the plant is simple and process well established.

Due to a combination of fortuitous circumstances listed below, Waste Ore Ratio at Stormont is low at 0.5:1.

1. Overburden cover is minimal allowing easy access to the mineralised material from surface.
2. The shape of the synclinal structure is such that the contact (orthogonal to strike) between mineralised material and host rock has a low slope angle.

3. The sub dip coincident with strike of the orebody is about 8º, which permits access to the mineralised material without internal ramps.
4. The thickness of the structure is limited.

A limited exploration programme by a previous explorer (Jervois) estimated an Inferred Resource for Stormont of 135,000t at a grade of 3.44g/t gold and 0.21% Bismuth.

Table 6 : Stormont Deposit : Mining Parameters for Target Mineralisation

Length	160	
Thickness	Variable 5m to 16m	
Width	Variable 25m to 55m (estimated)	
SG	≈2.75	
Dip	8°	Tasmanian Government Map
Total Tonnes	See Table 1	Distributed in syncline structure
Grades	See Table 1	

At Narrawa, which is situated on a hillside dipping North at about 8%, the strike of the mineralised structure is orthogonal to dip on an East West axis strikes northwest, dipping 70^{o} north east with the slope. Historic mining removed about 4,000t of material and in doing so, have excavated overburden from above the mineralised structure and in particular, waste rock from the South wall of the open pit (upslope in relation to the mineralised structure). The bench and batter wall remains exposed and stable.

The new drilling program has increased the size of the mineralised structure extending in depth and along strike. Mining of the new mineralised structure will result in further excavation on all perimeters particularly to the South where the crest of the pit wall bisects the surface of the hillside. Whilst this is balanced to some extent with the bisection of the crest of the North wall lower on the hillside, waste volumes increase and the Waste Ore ratio at Narrawa is 1.0.

Table 7 : Narrawa Deposit : Mining Parameters for the Resource

Length	115	From 5875m to 5987.5E + extension >6000mE =130m
Thickness		Variable 5m to 20m
Depth		Variable 25m to 60m
SG	≈3.0	Assumes 17% sulphides
Dip	30°	Sectional analysis utilising MapInfo; add on Discover
Total tonnes	See Table 2	Distributed in four adjacent lodes (-4000t mined before WW11)
Grades	See Table 2	

Processing of the Stormont Resource will be via a grinding circuit and CIP plant installed at a suitable site relative to Narrawa so that mineralised material from both can be transported economically (downhill or horizontally) to the crusher bin. Tables 8 and 9 list operating parameters and results from metallurgical tests for each deposit.

At Stormont, metallurgical results for recovery, reagent consumption and BWI of 14kWhr/t favour a low cost operation. High gold recovery indicates a non-refractory gold mineralised material. Bismuth is not leached in the CIP process and other process routes are yet to be evaluated. Enquiries have indicated that the small quantity of bismuth does not affect gold recovery in the CIP metallurgical process.

Table 8 : Operating Parameters ; Stormont Deposit

Stormont			
Waste : Ore Ratio	0.5t Waste to 1.0t Ore		
Mining Dilution	10%		
Cut Off Grade AuEq g/t	**1.0g/t**		
	Metal	**Grade**	**Recovery**
Grades and Metallurgical Recoveries	Gold	3.94g/t	92.3%
	Bismuth	2718ppm	0.01%
	Silver	3.41g/t	80.5%

Power Cost at power cost $0.12/kWhr	$2.52/t, (BWI = 14kWhr/t + 50% Works Power)
Transport Stormont to Plant at Narrawa	Company owned and operated Trucks
Total Project	
State Royalty – All products	5% of metal Value
Smelting/Refining (Narrawa only)	5% of metal value (includes process losses}

At Narrawa, the mineralised material will be processed by flotation. This process is well established and reliable and a high quality concentrate is anticipated for smelting and refining.
The metallurgical results (from Amdel) can be used to derive the quantity of recovered metal for calculation of gross revenue.

Table 9 : Operating Parameters ; Narrawa Deposit

Narrawa			
Waste : Ore Ratio	1.0t Waste to 1.0t Ore		
Mining Dilution	10%		
Cut off Grade AuEq g/t	**1.0 g/t**		
	Metal	**Grade**	**Recovery %**
Grades and Metallurgical Recoveries	Gold	2.09g/t	96.7%
	Zinc	1.13%	98.5%
	Lead	1.33%	95.6%
	Silver	19.59g/t	92.4%
Flotation Reagent Costs	$1.71/t		
Power Cost at power cost $0.12/kWhr	$2.52/t, (BWI = 14kWhr/t + 50% Works Power)		
Transport to Risdon	Company owned and operated Trucks		

Alternatively, two other methods of extraction are being investigated utilising new and proven technology now available in Australia. The advantage of each new technology is that each metal is produced at site and there is no reliance on other parties for completion of the extraction process.

A : Metal specific absorbent resin pellets within a cyanide solution medium. This process is well established and can be tailored to recover all metals from ROM material. Viz At Stormont, the process will recover the Bismuth as well as the gold and silver. At Narrawa, the process will recover all metals in the ROM material. Previous experience indicates high recoveries of selected metals.

As with the CIP process proposed for the Stormont material, disposal of a cyanide solution at Narrawa Creek is a major problem. The area lies in a watershed draining into a hydroelectric reservoir (Lake Cethana) to the east and rich farmland to the north. There are methods of treatment of tailing effluent which neutralise the cyanide before possible release but these would affect profitability of the project.

Frontier will draw electrical power for processing mineralised material from scheme sources. Due to the modest BWI for rock material at both sites, power costs are not a significant component of operating costs in the plant. If a lower cost for supply of electrical power can be found then the project benefits. A High Voltage Transmission Line is situated about 1km north of the Narrawa Deposit site (Transend) and two Intermediate Voltage Transmission Lines (Aurora Energy) are situated near All Nations Gold Mine about 600m South of Narrawa and about 4km East of Stormont. A budget cost for power has been obtained at $0.12/kWhr which is lower than for diesel generation. We recommend that Transend and Aurora Energy Ltd both of Tasmania be approached with the objective of connection to this supply.

MINING
The Stormont Deposit contains gold and minor bismuth, whilst the Narrawa Deposit contains gold, zinc, lead and silver. Stormont is situated about 5km distant from the Narrawa Deposit at +80m greater elevation.
Both structures are situated on sloping hillsides, but the strike of the Stormont structure is sub-parallel to the slope of the hillside and the strike of the Narrawa Deposit structure is orthogonal to the slope of the hillside. Access to the working floor in both deposits is achieved via ramps exterior to the pit wall commencing from an appropriate elevation.

Stormont Deposit

The mineralisation at Stormont has been mined previously and is located on a slope with a sub dip to the South East at about 8º. A small open pit and a small adit is found at the lower north west end of the resource. Ground cover/overburden is thin in this lowermost part of the mineralised structure and increases gradually up slope to the south east.

Mineralisation is contained in a synclinal structure with a NW/SE trending axis, the distance between the mineralisation and surface is therefore variable on an orthogonal section with the result that the (South West and North East) wings of the syncline are closer to the surface than at the centre. A syncline axis parallel reverse fault on the northern syncline limb limits full extension of the North West limb of the syncline to surface.

The mining programme will commence with cleanup of the surface (anticipated to be mostly waste material) using a tractor. There is a small quantity of sidecast material which could be mineralised and require treatment. As the mineralised resource is situated on a gentle slope, access for drilling and blasting and subsequent loadout of mineralised material is provided by ramps bisecting the resource at chosen intermediate levels down dip. Experience during exploration drilling indicates a competent mineralised material and blasting will be either required or prudent.

Removal of overburden from the ground surface on the hangingwall of the mineralised structure will be carefully controlled to ensure access for mining. A balance must be maintained between sidecast material remaining and overburden removed so that a working platform is maintained for access to broken material for load and haul after blasting.

Drilling of the next part of the mineralised structure will proceed whilst removal of the overburden continues below the drilling level for unconstrained blasting. Due to the shallow dip of the mineralised structure, frequent intermediate levels or flitches may be required to ensure low dilution.

If the ore can be ripped, then no blasting will be required and the ore can be mined by ripping and/or careful breakout/stripping to the footwall using the grouser plate on the blade of tractor. (as developed and used for mining at the Telfer Deposit, WA).

Narrawa Deposit

Surface mineralisation at Narrawa was previously mined approximately 70years ago by open pit methods; now known as the Higgs Open Cut. The old workings are small and are situated on a hillside which dips to the north. The mineralised structure lies with a strike orthogonal to dip. The old workings have remained open since mining ceased without serious deterioration. A tunnel system is found in the floor of the pit parallel to strike and is connected to an adit which enters from the north (downslope from the workings).

Depth of mineralisation delineated by the exploration drilling programme, varies between surface to about 40m below the original surface. As the slope of the hillside is moderate and stable, the shallow depth of the resource allows for a pit design with steep walls and with the toe of the slope on the contact of the mineralisation with the waste rock. Mining regulations are met as long as there is a safety berm and the batters can be made safe within the reach of the bucket of the loading machine. Operating benches will be 10m to15m in depth according to the machinery available with a 5m berm for safe operation. On this basis, the Waste to Ore Ratio is 1.0.

The mining programme will commence with careful clean up of the sidecast from the old Higgs Open Cut on the slope below the open cut perimeter. Some of this material is mineralised and purchase of a portable XRF machine is recommended, which will enable direct readout of the grade of chip samples taken in the field. When mineralised material is identified, it can be picked up by an excavator and stockpiled for later processing.

Upon completion of clean up of the sidecast material, the surface above the footprint of the final open cut will be cleared to refusal using tractors (with blasting if necessary). The perimeter of this surface will extend 5m beyond the crest of the batter of the walls of the final pit and expose the surface expression of the open pit design in full. A perimeter drain will be cut 5m outside the crest and a safety bund dumped on the crest side of the drain.

On the south perimeter, mineralised material will be bisected 10m below the cleaned surface and will extend to about 40m below the cleaned surface extending east and west along strike. In the north, ore is found in the floor of Higgs open pit and as extensions 20m below and to the east from the current pit structure. Access to the mineralised material will be from exterior ramps designed to bisect the contact at say -10m and –20m from surface etc. In this way pit volumes are minimised. A Waste Ore Ratio of 1.0t Waste to 1.0t Ore has been estimated.

Narrawa Creek passes west to east down the north side of and partly over the resource. This will need to be diverted before mining. After discussions with Mr R Reid, (Frontier Exploration Manager) the cross section of the creek in full flood during the wet season is not great as the slope of the ground results in high velocity flow. The diversion is anticipated to be a simple channel of blade width cut by a tractor on the West side of the final open pit boundary from a suitable point above the open pit.

General

At both sites, a perimeter drain will be cut upslope of the open pit to catch/divert water from flowing over the crest where it may cause a wash out and endanger the operators. The floor of the drain will fall to the nearest watercourse, being Narrawa Creek and Stormont Creek respectively. A Safety Bund constructed from waste produced by construction of the drain will be located on the open pit side of the drain to prevent persons and water entering the open pit. A Safety fence will be located on/at the bund.

Mining will proceed during daytime only and utilise the services of a mining contractor selected, if the quotation is satisfactory, from a local source. Operators will live in their own homes in the locality and travel to work each day.

Dry hire is recommended and fuel and lubricants will be supplied by Frontier from the same fuel supply company used by Frontier. The contractor will be allocated a bowser at the mine for fuel supply (diesel) and that outlet will be charged to the mining contract at cost as part of the mining cost per tonne. Contractors will require an area for maintenance which should be convenient to the process plant thereby minimising the number of areas damaged by the operation. The cost of Drilling and Blasting, Fuel and Lubricants and all ancillary mining costs are included in the cost of mining Ore and Waste in Table 10 below. At many operations in Australia, dilution is less than 5% due entirely to strict supervision and awareness by operators of the importance of precision when operating their equipment. In this study, mining dilution is set at 10% (which should never be exceeded).

Survey Control of the mining operation is essential to ensure that excavations are strictly limited to plan and that care is taken to reduce waste dilution to a minimum. A local base GPS system operating on a mine grid will determine location with an accuracy of ± 0.25m which is sufficient for most practical purposes.

All mining plant and equipment will be supplied by the mining contractor including his own demountable units at site for office, ablutions and maintenance. Explosives will be mixed at site and the intrinsically safe Orica Nonel Detonating system (or similar) will be used.

Mining costs for Ore and Waste normally include allowance for ex pit transport to crushers and waste dumps. It is desirable that if Stormont commences operations first and the process plant is located to serve Stormont and Narrawa in the future, all infrastructure including the substantial concrete foundation necessary for heavy rotating machinery, e.g. ball mill, crusher, etc. and offices, warehouse, changerooms etc will be installed at Narrawa to serve both Mineralised resources and avoid need for reinstallation/relocation.

Correct location of the plant at Narrawa plant for processing mineralised material from Stormont and Narrawa will ensure that all trucks travelling fully laden with ROM Material will travel downhill or horizontal with minimum uphill haul thereby reducing operating costs and haul time. Operating cost and haul time are much reduced and excessive charges avoided. Frontier should be able to negotiate low cost per tonne of ore and waste. Table 11 shows anticipated Capital Expenditure for the mining operation.

METALLURGICAL PROCESSING

Common to processing of Mineralised material from Stormont and Narrawa is a conventional crushing and grinding plant. ROM material will be dumped on a grizzley above the primary Jaw Crusher and the underflow conveyed into a 50t Primary Bin. A brute force feeder will draw off material into a SAG Mill at 40tph. The outflow is pumped to an elevated DSM Screen or cyclone cluster and the correct size underflow is transferred by a launder to the next stage of the process, either the first tank in a CIP Plant (Stormont material) or a Conditioner Storage Vessel in a flotation plant (Narrawa Material). The overflow is transferred by launder to the SAG Mill for regrind. Provision for a secondary crusher or regrind mill is included in the Capex list in case a single SAG mill is unable to complete grinding of all components of the ROM feed to the desired size range of P80/75µ. It may be necessary to change the SAG Mill to a Ball Mill as well. The ore is of moderate hardness with a Bond Work Index ("BWI") of 14kWhr/t which may not be suitable to SAG milling. The following text is individual to each source of ROM Material.

Table 10 : Estimate of Mining Costs, Stormont Deposit

Stormont Deposit	**In Situ**	**ROM Diluted 10%**
Inferred Resource COG1.0g/t	112,500t	124,000t
Specific Gravity	2.75t /m^3	
Ore Mining Volume	41,000 m^3	67,000m^3(Broken)
Waste Material @ W:O Ratio = 0.5	20,500 m^3	33,000m^3(Broken)
Total Volume Broken Material	61,500 m^3	100,000m^3(Broken)
NB Assumes similar SG		**$**
Ore Mining Cost per tonne in situ	$6.5/t	805,000
Waste Mining Cost per cubic metre in situ	$7.0/$m^3$	231,000
Sub Total		**1,036,000**
Site Preparation and Access Roads		300,000
Transport Stormont Material to Plant		200,000
Mobilisation/Demobilisation Contractor		150,000
Total		**1,696,000**

Narrawa Deposit	**In Situ**	**ROM Diluted 10%**
Indicated and Inferred Resource COG 1.0g/t	200,000t	220,000t
Specific Gravity	Ore 3.0t/m^3	Waste 2.5t/m^3
Ore Mining Volume	70,000 m^3	112,000 m^3 (Broken)
Waste Material @ W:O Ratio = 1.0	80,000 m^3	128,000 m^3 (Broken)
Total Volume Broken Material		240,000 m^3 (Broken)
NB Assumes similar SG		**$**
Ore Mining Cost per tonne in situ	$6.5/t	1,300,000
Waste Mining Cost per cubic metre in situ	$7.0/ m^3	560,000
Sub Total		**1,860,000**
Site Preparation and Access Roads		300,000
Mobilisation/Demobilisation Contractor		150,000
Total		**2,310,000**

NB Tonnages rounded off

Table 11 : Capital Expenditure by Frontier for Supervision of Mine & Plant

Items delivered to site	**$**
*6 room Office at Narrawa Creek, ablutions (demountable units)	150,000
Air conditioning	20,000
2 room Office at Stormont, ablutions (demountable units) say	70,000
4 wheel drive Toyota Covered Tray tops x 4 for supervision/transport	150,000
Office furniture, computers, etc (8 Offices)	100,000
Portable XRF Machine for analysis of samples at site	50,000
Other – Consultants etc.	150,000
Total $	**600,000**

*inc. Prefabricated Mess for 15 persons + 5 ancillaries being guests, caterers, etc. Hot water is drawn from Solar systems for Laundry, Drying Room, etc

Stormont Deposit

Mineralised material containing Gold and minor Bismuth will be mined and processed to P80/75μ for leaching in a CIP plant. Sufficient tanks will be installed for 48 hours residence time with associated carbon transfer, lime and cyanide handling facilities. However, Atomaer oxygen enhancement is recommended which will increase reaction rate and reduce residence time and tankage volume. In WA, lime and cyanide are available premixed but if this service is not available in Tasmania, then bagged lime and cyanide will be used. A gold room will be established for recovery of bullion from the pregnant carbon. Stormont ore is the same hardness as Narrawa ore (14kWhr/t) and contains very low Copper, Lead, Zinc and Silver values. Recoveries and BWI shown in Table 8 have been determined by Amdel of Perth. Table 12 shows Gross Revenue from Stormont Material.

Table 12 : Stormont Deposit : Possible Gross Revenue from Target Tonne & and Grade

Estimated In-Situ Tons			**112,500t**	**Mining Dilution**	**10%**	**Currency Exchange Rate**	
Diluted Tons			**124,000t**			**USD 0.77 =**	**AUD 1.00**
	Grade		**In Situ**	**Estimate of**	**Value**	**Net Revenue**	
	In Situ	**Diluted**	**Metal**	**Metal Recovered**	**USD**	**USD**	**AUD**
Gold	3.94g/t	3.58g/t	443kg	410kg (92.3%)	940/oz	12,385,000	16,040,000
Silver	3.41g/t	3.10g/t	384kg	309.kg (80.5%)	13.7oz	136.000	177,000
Totals =						**12,555,000**	**16,217,000**

Table 13 lists reagents required for CIP extraction of gold and Table 14 lists Non Labour Operating Costs based on information from industry sources.

Table 13 : Stormont : List of Reagents, Consumption and Cost/t

Reagent	**Consumption kg/t**	**Cost/Tonne**	**$/t Ore Processed**
Carbon	0.3	3000	0.9
Lime	1.5	2200	3.3
Cyanide	2.0	1500	3.0
Oxygen	1.0	600	0.6
Total Cost Reagents			**7.8**

Table 14 : Stormont Deposit : Non Labour Operating Costs

Operating Costs - non Chemical	**Notes**	**$/t**
Grinding – Gowrie Park Basics	Accumulative accounts for	0.30
Mill Liners	purchase of wear metals	0.30
Maintenance Spares	Mechanical/Electrical Spares	0.50
Protective Clothing	Allowance	0.30
Light Vehicles	4 Units	0.10
Fuel Costs BWI = 18kWhr/t	Say 27 kWhr/t inc Works Power	2.52
Fuel Costs Remote Pumps	Tailings Dam Water Recovery	0.10
Maintenance of all Power Plant	+15% Fuel Cost	0.60
Analytical Supplies & Services	Wet and Dry analysis	1.00
Hire of Atomaer plant		0.30
Gold Room : Recovery of Metal		0.30
Tailings Disposal		0.06
Operating Costs – Non Chemical	Total	6.38
Reagents from Table 14		7.80
Total Operating Cost without Labour Costs $/t		**14.19**

Production of Dore

Gold Dore from Stormont will be produced in the Gold Room and refined to separate and recover the Silver from the Gold. The products will be sent with established security measures to the nearest refinery for processing.

Narrawa Deposit

Due to the location of the Narrawa Deposit relative to the smelting/refining complex at Risdon 250km South of Narrawa, it is necessary to minimise transport costs by processing ore material into a concentrate at site. Processing will utilise conventional crushing and milling of the ore followed by flotation to produce a concentrate which contains gold, lead, zinc, silver metals. Tailings will be discarded in a tailings dam at site. Table 15 shows gross revenue which has been calculated from the Metal prices and resource estimate shown in Table 2 and metallurgical characteristics of the mineralised material from the Narrawa Deposit are shown in Table 8 as established by Amdel, Perth.

Table 15 : Narrawa Deposit : Estimate of Gross Revenue

In Situ Tonnes 200,000t			**Diluted (ROM) Tonnes 222,000t**			**Ex Rate A$1 = US$0.77**	
	Grade		**In Situ**	**Estimate of**	**LME$/unit**	**Net Revenue**	
	In Situ	**Diluted**	**Metal**	**Metal Recovered**	**USD**	**USD**	**AUD**
Gold	2.09g/t	1.90g/t	419kg	404kg (96.7%)	940/oz	12,217,000	15,866,000
Zinc	1.13%	1.02%	2,260t	2,210 (98.5%)	1,574.5/t	3,500,000	4,520,000
Lead	1.33%	1.21%	2,660t	2,395t (90.0%)	1,705.5/t	4,086,000	5,306,000
Silver	19.60g/t	17.81g/t	3.92t	3.62t (92.4%)	**13. 7/oz**	**1,594,000**	**2,071,000**
					Total	**21,397,000**	**27,763,000**
Total Metal Content			**4,910**	**4,610t**	**Value/t**	**USD 3946/t**	**AUD5,125/t**

Reagents are added in the vessel ready for flotation. The Flotation circuit contains three stages being Roughing, Cleaning and Scavenging. The circuit is simple with recycling of underflow between stages as necessary to enhance recovery of all metals.

The concentrate is dewatered on a drum filter and then dried and stored in a weatherproof hopper. Concentrate is drawn off into reusable impermeable Bulka Bags which are sealed for transport from site direct to the smelting/refining facility. Concentrate production is about 40t/day to 50t/day and the company will own its own trucks for transport of loaded Bulka Bags to Risdon and of supplies to site. Note that road Transport regulations in Tasmania regarding axle loadings are very strict and are savagely enforced.

Table 16 : Narrawa Deposit ; List of Reagents, Consumption and Costs

Reagent	**Consumption g/t**	**Expenditure $/t**
Potassium Amyl Xanthate	150	0.45
Cytec reagent 3418A	15	0.08
Lime	740	0.74
Sodium Cyanide	15	0.03
Copper Sulphate	100	0.28
Aerofroth 65	30	0.12
Total Cost Reagents per Tonne Processed $		**1.70**

Table 17: Narrawa Deposit ; Non Labour Operating Costs

Operating Costs - non Chemical	**Notes**	**$/t**
Grinding - Gowrie Park Basics	Accumulative accounts for	0.20
Mill Liners	purchase of wear metals	0.20
Maintenance Spares	Mechanical/Electrical Spares	0.50
Protective Clothing	Allowance	0.30
Light Vehicles	4 Units	0.10
Fuel Costs BWI = 14kWhr/t	Say 21 kWhr/t inc Works Power	2.52
Fuel Costs Remote Pumps	Tailings Dam Water Recovery	0.20
Haulage Trucks Fuel + Tyres	600km Haul Return to Risden	3.50
Analytical Supplies & Services	Wet and Dry analysis	1.00
Tailings Disposal		0.65
Operating Costs – Non Chemical	Total	9.17
Reagents from Table 13		1.70
Total Operating Cost without Labour Costs $/t		**10.87**

Table 18 shows the effect of the new process on Non Labour Costs and Reagents. In particular, please note the decrease in Power Consumption. The costs apply to both Stormont & Narrawa.

Table 18: Stormont and Narrawa Deposits ; Non Labour Operating Costs including reagents

Operating Costs - non Chemical	**Notes**	**$/t**
Grinding - Gowrie Park Basics	Accumulative accounts for	0.20
Mill Liners	purchase of wear metals	0.20
Maintenance Spares	Mechanical/Electrical Spares	0.50
Protective Clothing	Allowance	0.30
Light Vehicles	4 Units	0.10
Power BWI = 5kWhr/t inc works power	$12/kwhr	0.60
Analytical Supplies & Services	Wet and Dry analysis	1.00
Tailings Disposal		0.65
Estimated Cost of Reagents		**0.50**
Total Operating Cost without Labour Costs $/t		**4.05**

Table 19 contains a list of probable process equipment for conventional processing of the mineralised material into a saleable product.

Table 19 : Installed Cost of Process Equipment : 40t/hr

			AUD x 1.0M	**AUD x 1.0M**
Narrawa	Conditioner/Storage Vessel	200t	0.300	
	Flotation Circuit	50tph capacity	0.500	
	Filter/Dryer Cyclones		0.150	
	Storage Shed with hopper and bagging appliance		0.200	
Stormont	50t Primary Bin + Feed Conveyor		0.150	
	Jaw Crusher 150mm x 150mm + Conveyor		0.450	
	SAG Mill or Ball Mill	100tph	1.000	
	Regrind Mill	40tph	0.500	
	Mill Recirculation Pumps		0.200	
	Classification Screen (DSM)		0.050	
	3 Tanks CIP Plant installed complete		0.350	
	Gold Room		0.250	
	Support Steelwork/pipework/Valves		0.400	**3.350**
Common	Concentrator Building & Offices etc.		0.350	
	Portable XRF		0.050	
	Electrical Switchgear & Distribution		0.400	
	Concrete 500/m^3 Delivered $300/$m^3$ (total plant)		0.150	
	Water Supply Dam and Pipeline (gravity supply)		0.050	
	Tailings Dam, water recovery pump and Pipeline		0.100	
	Site Preparation for mill		0.100	
	Design Construction Supervision		1.200	
	Diesel powered forklift		0.100	
	Prestart Wages for 21 days		0.220	
	2 x 30t Trucks hauling Concentrates/General supplies		0.800	
	**Mess + Laundry + Change rooms		0.200	
	Mining Capex (Table 8)		0.600	**4.320**
	Total			**$8,820M**

A contingency of say $2M should be held in reserve as part of the capital facility for this project.

At this time in Tasmania, there are many items of pre-owned equipment available for sale which should reduce the costs of some of the above items. Table 20 is a list of Capital Equipment and allocated costs of purchase for the new process

Table 20 : Estimated Cost of Equipment for New Process: 45t/hr ROM mineralised material

	AUD x 1.0M	AUD x 1.0M	50t
Primary Bin + Feed Conveyor	0.150		
Jaw Crusher 150mm x 150mm + Conveyor	0.450		
Support Steelwork/pipework/Valves/Concrete	0.400		
Concentrator Building & Offices/Facilities (Laboratory)	**0.250**		
Portable XRF	0.050		
Water Supply Dam and Pipeline (gravity supply)	0.050		
Tailings Dam, water recovery pump and Pipeline	0.100		
Site Preparation for mill	0.100		
Prestart Wages for 21 days	0.220		
2 x 30t Trucks hauling Product/General supplies	0.800		
Mess + Laundry + Change rooms	0.200		
Mining Capex (Table 8)	0.600		
Storage Shed with export container equipment	0.200		
Diesel powered forklift	0.100	3.670	
NEW PLANT FOR CRUSHING GRINDING METAL RECOVERY	4,000		
Including Design and Supervision of Construction	**1,000**	**5,000**	
Total		**$8.670,000**	

Processing Of Concentrates

Due to the complex polymetallic nature of the mineralised material at Narrawa Deposit, Toll Smelting and Refining is likely to be expensive but due to the unusually high quality of the concentrate, charges and losses are anticipated to be low at say about 5% of the value of contained metal compared to the normal charges approaching 7% to 10% of the value of the metal. The charge is deducted from gross revenue as a royalty by the Smelter Company.

PROCESS PLANT LABOUR REQUIREMENTS

Table 21 shows a list of Plant Labour based on a process time of 357 days per year (51 weeks/year) at 40t/hr.

Table 21 : Manpower (per annum basis) ; 360 days Processing

Production		3 Shifts	357	Days/Year
	Position	Number	Salary/annum	Total
	General Foreman	1	140,000	140,000
	Primary Crusher	2	110,000	220,000
	Concentrator + Tailings	3	110,000	330,000
	Mechanic/Fitter	1	120,000	120,000
	TA	2	110,000	110,000
	Metallurgical	1	150,000	150,000
	Analytical	1	120,000	120,000
		11	$	1,190,000
3	**Rotation Teams**	**7**	Days per week	3,570,000
Insurance/Taxes/Superannuation/Leave Loading			+27%	975,000
Total Establishment for Processing		**21**	**Annual Wages $**	**4,545,000**

The process plant will work continuously achieving a 21 hours of production per day (87% utilization). From experience at other operations, 9 men per 12 hour shift will be sufficient to operate and maintain the process plant. There will be 3 teams each working a 14 day on/7 day off shift regime with one team changing each week. Annual Leave of 28 days will be taken at the end of processing of the Narrawa Ore material.

Safety

Prior to start up, the workforce will begin with lectures on safety and procedures and 15 days wet operation of the process plant using mineralised material from cleanup and recovery of mineralised sidecast material from Stormont and Higgs Open Cut. Mechanics/Fitters will check bearing temperatures, alignments etc and correct as necessary before serious production commences.

All motors and switchgear will be confined to common size(s) and connected to the power supply by plugs and sockets. If a motor or switch unit fails then it can be changed out by the mechanic/fitter without an electrician.

PROCESSING SCHEDULE

Table 22 : Proposed Schedule for Processing Stormont Material followed by Narrawa Creek Material

Operation	**Resource**	**Rate**	**Hours/day**	**Days/year**	**3 Week Cycles**
Year 1 Safety/Training and Commissioning/Start up of Plant				21	**1**
Stormont	124,000t	40tph	21	148	**7 ±0.5**
Changeover to Flotation Circuit					**±0.5**
Year 1 & 2 Narrawa Creek	220,000t	40tph	21	262	**13**±0.5
Production Totals	**344,000t**	**40tph**	**21hours/day**	**431days = 1.25 years**	

NB Schedule allows 7 days for public holidays

MANAGEMENT

Part of the Management group listed in Table 21 will work with the Process Plant construction group to install plant and equipment inside a concentrator building, offices etc. Construction of these buildings provides weatherproof workspace for assembly of the plant and equipment.
A contract surveyor will mark out the open pit access roads and position structures and supply other survey services during construction and operations. The Group will monitor on behalf of Frontier, construction of the process plant.

Table 23 lists the management team who will be housed at Gowrie Park. Members will be employed as necessary for prestart works and construction leading to commissioning of the plant with the operator workforce who will be recruited at an appropriate time. The Manager will then cooperate with the Mining Contractor to ensure that purest ore will be delivered to the process plant at all times.

Table 23 : Management say 357 days per year at site

Position	**Number**	**Salary/annum**	**Total**
Manager	1	250,000	250,000
General Foreman	1	150,000	150,000
Contract Surveyor	1		2450,000
Assistant	1		
Road Haul Truck Drivers	3	120,000	360,000
Management Total	7	Annual Salaries $	1,000,000
+Insurance/Taxes/Superannuation/Leave Loading			1,300,000

After all mineralised material has been processed, the management team will remain for 3 months to supervise removal of all equipment from site and rehabilitate any damage or continue operations processing mineralised material from another part of the lease. Rehabilitation will require heavy earthmoving equipment supplied by the mining contractor.

HEAD OFFICE COSTS

Table 24 shows estimates of costs attributed to/administered by Head Office in Hobart.

Item of Expenditure	**Cost per Annum $**
Computer/Word Processor/Office Consumables	35,000
Power/Heating/Air Conditioning	50,000
Rental of Housing at Gowrie Park for Staff	120,000
Office Rent, Tel.	50,000
Country Rates	50,000
Comprehensive Insurance for the project	150,000
Salaries Expenses etc.	400,000

Total Head Office Costs	**855,000**

SUMMARY OF OPERATING COSTS

Table 25 and Table 26 list the estimated operating costs for each site and Table 27 lists the (reduced) Operating costs using the new technology.

Table 25 : Stormont Deposit : Summary of Operating Costs

Throughput	**124,000 tonnes per annum**	
Expenditure per Cost Centre	**Total Cost/annum**	**Total Cost/t**
Mining Costs	1,696,000	13.68
Operation Costs non Labour	1,770,000	14.27
Operating Labour = 169 days	1,884,000	15.19
Management = 169 days	827,000	6.66
Head Office	615,125	4.96
Totals	**6,792,000**	**55.00**

Table 26 : Narrawa Deposit : Summary of Operating Costs

Throughput	**211,500 tonnes per annum**	
Expenditure per Cost Centre	**Total Cost/annum**	**Total Cost/t**
Mining Costs	2,310,000	10.50
Operation Costs ex Labour	2,391,000	10.87
Operating Labour = 262 days	3,308,000	15.04
Management = 352 days	1,300,000	5.91
Head Office	855,000	3.89
Totals	**9,746,000**	**46.00**

Table 27: Stormont & Narrawa : Summary of Operating Costs

Throughput	**344,000 tonnes per annum**	
Expenditure per Cost Centre	**Total Cost/annum**	**Total Cost/t**
$	$	$
Mining Costs	4,006,000	11.65
Operation Costs non Labour	1,393,000	4.05
Operating Labour = 365 days	4,600,000	13.37
Management = 450days	2,206,000	6.41
Head Office	1,451,000	4.21
Totals $	**13,656,000**	**40.00**

Management includes allowance for Year -1 and Year 2 activities

FINANCIAL RETURN ON INVESTMENT

Table 28 shows the cash flows which result from mining and processing 112,500t in situ Inferred resource at Stormont using CIP technology and 200,000t of in situ Indicated and Inferred Resource at Narrawa using conventional process technology : Stormont material processed first :-

Table 28: Summary of Net Cash Flow; Sequential Process due to Flotation/CIP incompatibility

Item	**ROM t**	**Days**	**Year -1**	**Year 1**	**Year 2**
Training/Commissioning		21	$	$	$
Stormont	124,000	148		16,200,000	
Narrawa	220,000	262		19,000,000	8,800,000
Totals				**35,200,000**	**8,800,000**
- Government Royalty		5%		1,760,000	440,000
- Smelting Charges (Narrawa)		5%		960,000	440,000
Operating Costs		**Stormont**		6,792,000	
		Narrawa		7,293,000	2,871,000
Capital Costs (Investment)		**Stormont**	3,350,000		
		Common	4,320,000		

		Narrawa		1,150,000	
Total Debits			**7,670,000**	**17,955,000**	**3,751,000**
Net Cash Flow before Tax			**-7,670,000**	**17,955,000**	**3,751,000**
Net Present Value at		**Discount 0%**	**$14,000,000**	**Discount 10%**	**$10,700,000**

Table 29 shows the cash flows which result from mining and processing 200,000t of in situ Indicated and Inferred Resource at Narrawa using conventional process technology followed by112,500t in situ Inferred resource at Stormont using CIP technology : Narrawa material processed first :-

Table 29: Summary of Net Cash Flow; Sequential Process due to Flotation/CIP incompatibility

Item	**ROM t**	**Days**	**Year -1**	**Year 1**	**Year 2**
Training/Commissioning		21	$	$	$
Stormont	124,000	148		10,400,000	5,800,000
Narrawa	220,000	262		27,800,000	
Totals				**37,200,000**	**5,800,000**
Less Government Royalty		5%		1,860,000	290,000
Less Smelting Charges (Narrawa)		5%		1,390,000	
Operating Costs		**Stormont**		4,400,000	2,400,000
		Narrawa		10,164,000	
Capital Costs (Investment)		**Stormont**		600,000	
		Common	4,320,000		
		Narrawa	3,900,000		
Total Debits			**8,220,000**	**18,400,000**	**2,690,000**
Net Cash Flow before Tax			**-8,220,000**	**18,800,000**	**3,110,000**
Net Present Value at		**Discount 0%**	**$13,700,000**	**Discount 10%**	**$10,400,000**

Table 30 shows the cash flows which result from mining and processing 112,500t in situ Inferred resource at Stormont and 200,000t of in situ Indicated and Inferred Resource at Narrawa SIMULTANEOUSLY using new process technology if available and proven : Stormont material processed first :-

Table 30: Summary of Net Cash Flow; Simultaneous Processing of Mineralised Materials

FINANCIAL RESULTS : STORMONT AND NARRAWA PROJECTS					
NEW PROCESS					
Throughput		45	tph	Weeks/Year	51
Hours Available		21	hrs/day	Availability	87.5%
Days worked per Annum		357	days/year		
Production		Tonnes		Hours	days/year
Year -1 Training & Commissioning					21
Year 1 Stormont		124,000	`	2,755.6	131.22
Narrawa		220,000		4,888.9	232.80
	Totals	344,000		7,644.4	364.02
USD	.0.77	AUD 1.0			
AUD	1.2987	USD 1.0			
Mineralised Resource Stormont		ROM t =	124,000		Year 1
Recovered Value	98	% all metals	In Situ Metal	AUD	A$
		Au kg	443	39.26/g	22,135,355.
		Ag kg	384	0.57/g	279,590.
		Bi ppm	2718	0.0000/g	2,037.
Mineralised Resource Narrawa		ROM t =	220,000		
Recovered Value	98	% all metals	In Situ Metal	AUD	
		Au kg	419	39.26/g	16,120,000
		Zinc t	2,260	2,044/t	4,530,000
		Lead t	2,660	2,214/t	5,770,000
		Ag kg	3920	0.57	2,190,000
		Total Gross Income before Tax			51,027,000
Less Government Royalty	5%				2,550,000
Less Operating Costs		Stormont and Narrawa			13,656,000
Less Capital Costs		Stormont and Narrawa			8,760,000
Less Licensor Licence Fee		Stormont and Narrawa			75,000
	Income before Tax for Calculation of Royalty				26,000,000
		Licensor Royalty @ 5%			1,250,000
	Income before Tax for after payment of Royalty				$24,750,000
CASH FLOWS BEFORE TAX				Year -1	Year 1
Capital :: Stormont and Narrawa				-$ 8,760,000	nil
Income :: Stormont and Narrawa				nil	$ 34,760,000
ANNUAL CASH FLOWS				-$ 8,760,000	$ 34,760,000
LICENCE FEES AND ROYALTY TO LICENSOR			RETURN ON INVESTMENT		
ROYALTY		$1,250,000	NPV @ 0% discount		$ 26,500.000
LICENCE FEE		$ 75,000	NPV @ 10% discount		$ 20,700,000
TOTAL INCOME		$ 1,375,000	-0 -		- 0 -

APPENDIX 3

Petrological examination of rock samples from the Stormont Mine area, Moina

MRT Mineralogical/Petrology Laboratory

Job No. M09/010

An unpublished report for Frontier Resources Ltd.

R.S. Bottrill

18/3/09

SUMMARY

Four rock samples, from drilling near the Stormont mine, Moina, were examined and found to be calcic skarns. The primary skarns had andradite- diopside assemblages, and have been variably retrogressed to hastingsitic amphiboles. One contains some late stage, vein style bismuth and gold mineralisation with magnetite.

INTRODUCTION

Four rock samples from the above location were submitted for XRD and one for polished thin sectioning and brief petrography.

They were prepared and examined by XRD, transmitted and reflected polarised light and stereo-microscopic techniques in our laboratories.

Table 1: sample details.

Reg. No.	DDH/depth(m)	Treatment
G403494	SFD005 24.7 to 24.85m	XRD & PTS
G403495	SFD006 17.38 to 17.6	XRD
G403496	SFD004 9.83 to 9.95	XRD
G403497	SFD009 8.41 to 8.62	XRD

XRD ANALYSIS

X-ray diffraction analysis (XRD) was carried out on all the samples, for mineralogical analysis, and the results are included in appendix 1 below. The rocks appear to contain mostly calcic garnets (close to andradite), clinopyroxenes (close to diopside) and calcic amphiboles, with quartz and carbonates.

PETROLOGICAL EXAMINATION

Sample G403494 SFD005 24.7 to 24.85m

In hand specimen the rock sample is a fine grained, massive rock with greenish grey colour, with some fine sulphide veining (bismuthinite) and irregular splotches of a black mineral (magnetite?). There is no obvious foliation or lamination.

In thin section the rock is composed mostly of:

- **Garnet, medium grained (0.5-1mm), irregular to rounded and euhedral, colourless, disseminated and in veins, ~20%. It is strongly zoned and moderately birefringent but is not vesuvianite (from XRD, indicating calcic andradite).**
- **Diopside, fine-medium grained (0.01-0.5mm), blocky, subhedral, colourless, disseminated ~35%**
- **Amphibole fine-medium grained (0.05-2 mm long), highly fibrous to fine grained matted intergrowths, pale to deep green~20%**
- **Carbonate (calcite), fine - medium grained (<0.5mm), colourless, disseminated ~25%**
- **Magnetite, fine-medium grained (<0.5mm), irregular, poikiloblastic, disseminated ~2%**
- **Quartz, coarse grained (<2mm), in veins, ~1%**
- **Sulphides, disseminated and in veins; <5 mms, also colloidal, ~2%.**

The rocks appear to exhibit a primary assemblage of colourless, fine grained pyroxene in a groundmass of fine grained carbonate and amphibole (tremolite-actinolite?), with some disseminated coarser pyroxene and irregular patches and veins of coarse garnet. The primary assemblages were probably andradite- diopside-calcite (from XRD).
This assemblage is partly altered to cloudy patches of green amphiboles (hastingsite?) finely intergrown with magnetite and bismuthinite, probably largely replacing carbonate, associated with some veining. The veins comprise coarse garnet, quartz, amphibole and metallic minerals (see below).

Mineralisation comprises about 2% sulphides and metals and about 1% magnetite. Most occurs in and about a vein about 6mm wide. The vein contains most of the sulphides and the magnetite is mostly in a selvage to this. The main sulphide phase is bismuthinite, in a thin vein and grains to about 4mm, with a lamellar/twinned to prismatic structure. It is intergrown with a softer, whiter mineral, possibly lillianite (a

lead bismuth sulphide), to about 0.5mm. Native bismuth is also common as rounded, tarnished grains (< 0.15 mm), mostly on the edges of bismuthinite, suggesting replacement, but some also occurs as inclusions. Chalcopyrite grains are irregular and up to about 0.2mm in size. Numerous small rounded grains of gold (< 60 microns) also occur in the vein; it is rather pale and probably silver-rich. Most chalcopyrite and gold occur interstitial to garnet outside of the bismuthinite, but some grains are included in bismuthinite. The magnetite occurs as small irregular grains suffused with acicular amphiboles and carbonate grains in cloudy patches around the sulphide vein. It appears to be replacing carbonates? Some abundant but fine grained (<0.01mm) disseminated phases associated with this alteration may be monazite or cassiterite(?).

DISCUSSION AND INTERPRETATION

These rocks appear to contain mostly calcic garnets (close to andradite), clinopyroxenes (probably close to diopside) and calcic amphiboles, with quartz and calcite, so are interpreted as calcic skarns. The primary assemblages were probably andradite- diopside-calcite-tremolite assemblages, and have been variably altered or retrogressed to andradite, magnetite and amphiboles, with some sulphides and quartz, mostly vein-related.

The mineralisation is dominated by bismuth and copper sulphides, but also contains numerous small gold grains, with magnetite. The mineralisation is a typical epithermal style, granite-related mineralisation. The presence of native bismuth and magnetite indicates a low sulphur fugacity.

DEPARTMENT OF INFRASTRUCTURE ENERGY AND RESOURCES

Mineral Resources Tasmania

Client: R. Reid, Frontier Resources
Sample Location: Stormont
MRT Job Number: M010/09
Analysis: Mineralogy
Method: X-Ray Diffraction

Results:

Sample	*Minerals Identified*
G403494 (SFD005, 24.7-24.8)	major Garnet[1], Clinopyroxene[2], Amphibole, minor Quartz, Mg-Calcite
G403495 (SFD006, 17.38-17.6)	major Mg-Calcite, Amphibole, Garnet, Clinopyroxene[2], minor Quartz, Mica[3], trace Chlorite, Epidote
G403496 (SFD004, 9.83-9.95)	major Siderite, Garnet, Amphibole, minor Mg-Siderite, Quartz
G403497 (SFD009, 8.41-8.62)	major Amphibole, minor Quartz, trace Magnetite, ?

[1] possibly two Garnets present
[2] probably Hedenbergite
[3] probably not Muscovite
? unknown mineral/minerals, probably in brown parts of core (small peaks at 15.3Å, 10.93Å, 7.56Å, 3.71Å, 3.26Å, 3.09Å)
Garnet peaks in 403495 (some overlap with Quartz and Amphibole) – 4.25, 3.00/2.985, 2.675/2.660, 2.545/2.535, 2.445/2.430, 2.350/2.335, 2.180/2.170, 1.940/1.930, 1.725/1.715, 1.660/1.650, 1.600/1/590
Garnet peaks in 403495 (some overlap with Quartz and Amphibole) – 4.25, 2.995, 2.675, 2.545, 2.445, 2.345, 2.180, 1.940, 1.730, 1.660, 1.600

Analyst: R.N. Woolley
Date: 20 February 2009

APPENDIX 4

Metallurgical Testwork - gravity separation and leaching

The results of the gravity separation and leaching of the gravity separation products show an overall gold recoveries ranging from 80-85% with the highest recovery at the finest grind.

Recovery of gold into the gravity concentrate ranged from 22-29% with recovery increasing with fineness of grind.

The leach residues of the gravity tailings leach tests were high ranging from 1.85 to 1.4g/t reducing with finer of grind.

The gravity tailings leaching curves indicate that the gold is present in two phases i.e. a non-refractory fast leaching phase with 80-90% of the leachable gold recovered in the first two hours of the test and a slow leaching component with the remaining gold recovered in the following 46 hours.

The results indicate that this slow leaching component is still leaching at 48 hours.

AMDEL MINERAL LABORATORIES LTD

ABN: 30 008 127 802
64 Kurnall Rd, Welshpool, Western Australia, 6106
Telephone +61 8 9451 8477 Fax + 61 8 9451 4576
labsupport_mineral@amdel.com

Client:	Frontier Minerals
Project:	Stormont Deposit
Project No:	2922
Composite:	Master Composite @ P_{80} 150µm

RECONCILIATION - GRAVITY CONCENTRATE + GRAVITY TAILINGS LEACHING

Sample	Mass		Gold				Silver			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	86.7	4.33	55.3	22.1	90.4	20.0	7.1	12.3	78.9	9.7
Gravity Tailings	1913.3	95.67	8.82	77.9	79.0	61.6	2.3	87.7	56.6	49.6
Calculated Head Grade	2000.0	100.00	**10.8**	100.0		**81.6**	**3.1**	100.0		**59.3**
Assay Head Grade			**9.19**				**2.5**			

Sample	Mass		Bismuth			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	86.7	4.33	6060	12.8	0.0	0.0
Gravity Tailings	1913.3	95.67	1870	87.2	0.0	0.0
Calculated Head Grade	2000.0	100.00	**2052**	100.0		**0.0**
Assay Head Grade			**1830**			

AMDEL MINERAL LABORATORIES
ABN: 30 008 127 802
64-70 Kurnall Road, Welshpool, Western Australia, 6106
Phone +61 8 9451 8477 Fa x +61 8 9451 4576

CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Master Composite	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Gravity Conc	**Pulp Density:**	20
Amdel Project No:	2922	**Sample Weight (g):**	87	**Water Type:**	Perth Tap Water
Date:	12-May-09	**Grind Size:**	P_{80} 150µm	**NaCN (initial):**	20000
Metallurgist:	ASL			**NaCN (test):**	
Test Description:	Intensive Cyanide Leach			**pH:**	10.0 - 10.5
Test No:	**1A**			**Dissolved Oxygen:**	>4ppm

TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements pH Found	pH Left	DO (ppm)	NaCN (ppm)	Reagent Additions NaCN (g)	Lime (g)		Cum. Reagent Usage NaCN (kg/t)	Lime (kg/t)	Notes
Leaching	**0**	20.0										
	24	20.0	11.31									

TEST RESULTS

Time (hrs)	Solution Assays (mg/l) Au	Ag	Bi			Solids Assays (g/t) Au	Ag	Bi			Extracted Grade (g/t) Au	Ag	Bi			Recovery (%) Au	Ag	Bi		
0																				
24	12.50	1.4	0.04								50	6	0			**90.4**	**78.9**	**0.003**		
Residue Assays						5.3	1.5	6060												
Calculated Head Grades						**55.3**	**7.1**	**6060**												

AMDEL MINERAL LABORATORIES
ABN: 30 008 127 802
64-70 Kurnall Road, Welshpool, Western Australia, 6106
Phone +61 8 9451 8477 Fax +61 8 9451 4576

CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Gravity Tailings	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Master Composite	**Pulp Density:**	50% solids
Project No:	2922	**Sample Weight (g):**	1913	**Water Type:**	Perth Tap Water
Date:	11-May-09	**Grind Size:**	P_{80} - 150µm	**NaCN (initial):**	500
Metallurgist:	GL			**NaCN (test):**	300
Test Description:	Cyanide Leach Test			**pH:**	10-10.5
Test No:	**1B**			**Dissolved Oxygen:**	>4

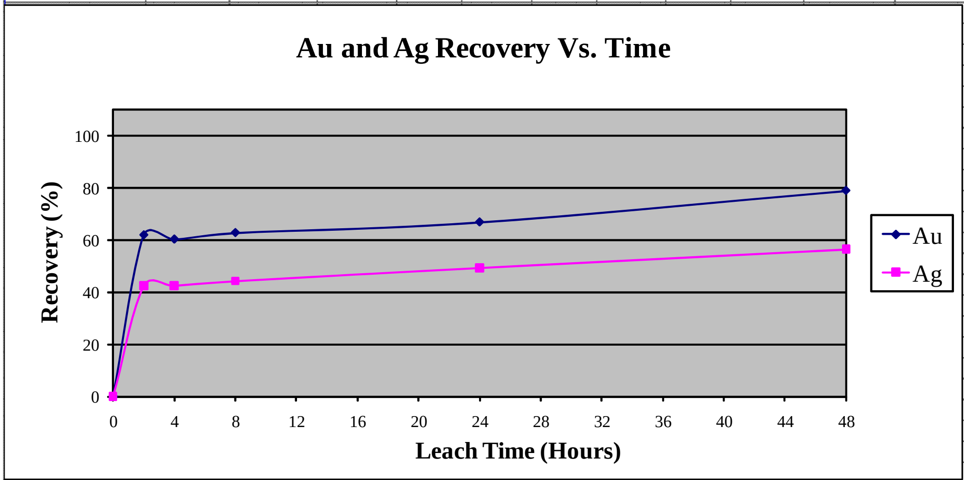
TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements				Reagent Additions			Cum. Reagent Usage		Notes
			pH Found	pH Left	DO (ppm)	NaCN (ppm)	NaCN (g)	Lime (g)	Other	NaCN (kg/t)	Lime (kg/t)	
Leaching	**0**	50.0	8.20	10.48	6.2		1.00	2.10			1.10	Test carried out using Perth tap water
	2	50.0	10.00	10.29	7.3	360	0.27	0.36		0.16	1.29	~50mls of slurry removed at each sampling time
	4	50.0	10.10	10.44	7.4	387		0.38		0.27	1.48	Solids returned & top-up solution added back in
	8	50.0	10.09	10.55	6.7	324	0.34	0.40		0.33	1.69	
	24	50.0	10.16	10.62	6.2	208	0.56	0.37		0.62	1.89	
	48	50.0	10.14		7.3	150				0.97	1.89	

Total NaCN added	2.17	gms
Nett added	2.14	gms (less NaCN in solution samples)

TEST RESULTS

Time (hrs)	Solution Assays (mg/l) Au	Ag	Bi	Solids Assays (g/t) Au	Ag	Bi	Extracted Grade (g/t) Au	Ag	Bi	Recovery (%) Au	Ag	Bi
0										0.0	0.0	
2	5.47	0.98					5.47	0.98	0	62.0	42.6	
4	5.26	0.97					5.33	0.98	0	60.4	42.7	
8	5.48	1.01					5.55	1.02	0	62.9	44.4	
24	5.69	1.10					5.91	1.14	0	66.9	49.5	
48	6.69	1.25	0.06				6.97	1.30	0	**79.0**	**56.6**	**0.003**
Residue Assays				1.85	1.0	1870						
Calculated Head Grades				**8.82**	**2.3**	**1870**						



AMDEL MINERAL LABORATORIES LTD
ABN: 30 008 127 802
64 Kurnall Rd, Welshpool, Western Australia, 6106
Telephone +61 8 9451 8477 Fax + 61 8 9451 4576
labsupport_mineral@amdel.com

Client:	Frontier Minerals
Project:	Stormont Deposit
Project No:	2922
Composite:	Master Composite @ P_{80} 106µm

RECONCILIATION - GRAVITY CONCENTRATE + GRAVITY TAILINGS LEACHING

Sample	Mass		Gold				Silver			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	81.4	4.07	61.3	23.5	71.8	16.8	8.2	13.4	75.5	10.1
Gravity Tailings	1918.6	95.93	8.49	76.5	79.9	61.1	2.2	86.6	55.4	48.0
Calculated Head Grade	2000.0	100.00	**10.6**	100.0		**78.0**	**3.2**	100.0		**58.1**
Assay Head Grade			**9.19**				**2.5**			

Sample	Mass		Bismuth			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	81.4	4.07	6531	13.8	0.0	0.0
Gravity Tailings	1918.6	95.93	1730	86.2	0.0	0.0
Calculated Head Grade	2000.0	100.00	**1926**	100.0		**0.0**
Assay Head Grade			**1830**			

AMDEL MINERAL LABORATORIES
ABN: 30 008 127 802
64-70 Kurnall Road, Welshpool, Western Australia, 6106
Phone +61 8 9451 8477 Fa x +61 8 9451 4576

CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Master Composite	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Gravity Conc	**Pulp Density:**	20
Amdel Project No:	2922	**Sample Weight (g):**	81	**Water Type:**	Perth Tap Water
Date:	12-May-09	**Grind Size:**	P_{80} 106µm	**NaCN (initial):**	20000
Metallurgist:	ASL			**NaCN (test):**	
Test Description:	Intensive Cyanide Leach			**pH:**	10.0 - 10.5
Test No:	**2A**			**Dissolved Oxygen:**	>4ppm

TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements				Reagent Additions			Cum. Reagent Usage		Notes
			pH Found	pH Left	DO (ppm)	NaCN (ppm)	NaCN (g)	Lime (g)		NaCN (kg/t)	Lime (kg/t)	
Leaching	**0**	20.0										
	24	20.0	11.25									

TEST RESULTS

Time (hrs)	Solution Assays (mg/l)					Solids Assays (g/t)					Extracted Grade (g/t)					Recovery (%)				
	Au	Ag	Bi			Au	Ag	Bi			Au	Ag	Bi			Au	Ag	Bi		
0																				
24	11.00	1.5	0.21								44	6	1			**71.8**	**75.5**	**0.01**		
Residue Assays						17.3	2.0	6530												
Calculated Head Grades						**61.3**	**8.2**	**6531**												

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CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Gravity Tailings	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Master Composite	**Pulp Density:**	50% solids
Project No:	2922	**Sample Weight (g):**	1919	**Water Type:**	Perth Tap Water
Date:	11-May-09	**Grind Size:**	P_{80} - 106µm	**NaCN (initial):**	500
Metallurgist:	GL			**NaCN (test):**	300
Test Description:	Cyanide Leach Test			**pH:**	10-10.5
Test No:	**2B**			**Dissolved Oxygen:**	>4

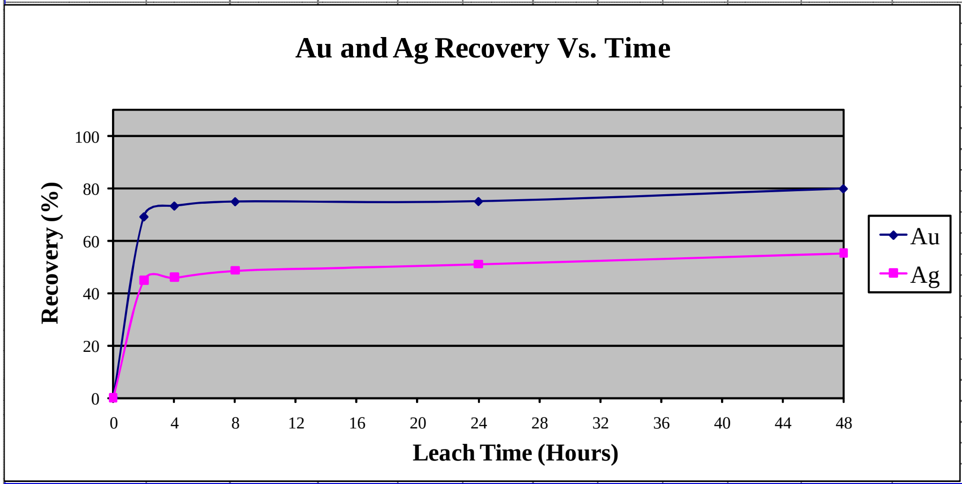
TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements: pH Found	pH Left	DO (ppm)	NaCN (ppm)	Reagent Additions: NaCN (g)	Lime (g)	Other	Cum. Reagent Usage: NaCN (kg/t)	Lime (kg/t)	Notes
Leaching	0	50.0	8.21	10.53	6.0		1.00	2.21			1.15	Test carried out using Perth tap water
	2	50.0	10.08	10.31	7.0	377	0.23	0.29		0.14	1.30	~50mls of slurry removed at each sampling time
	4	50.0	10.15	10.53	7.0	406		0.43		0.23	1.53	Solids returned & top-up solution added back in
	8	50.0	10.20	10.51	6.4	329	0.34	0.25		0.30	1.66	
	24	50.0	10.27	10.58	6.1	232	0.51	0.43		0.57	1.88	
	48	50.0	10.32		7.1	132				0.94	1.88	

Total NaCN added	2.08	gms
Nett added	2.05	gms (less NaCN in solution samples)

TEST RESULTS

Time (hrs)	Solution Assays (mg/l): Au	Ag	Bi	Solids Assays (g/t): Au	Ag	Bi	Extracted Grade (g/t): Au	Ag	Bi	Recovery (%): Au	Ag	Bi
0										0.0	0.0	
2	5.87	1.01					5.87	1.01	0	69.2	45.1	
4	6.15	1.02					6.22	1.03	0	73.3	46.0	
8	6.29	1.08					6.36	1.09	0	74.9	48.7	
24	6.14	1.11					6.37	1.15	0	75.0	51.2	
48	6.48	1.19	0.05				6.78	1.24	0	**79.9**	**55.4**	**0.003**
Residue Assays				1.71	1.0	1730						
Calculated Head Grades				**8.49**	**2.2**	**1730**						



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64 Kurnall Rd, Welshpool, Western Australia, 6106
Telephone +61 8 9451 8477 Fax + 61 8 9451 4576
labsupport_mineral@amdel.com

Client:	Frontier Minerals
Project:	Stormont Deposit
Project No:	2922
Composite:	Master Composite @ P_{80} 75µm

RECONCILIATION - GRAVITY CONCENTRATE + GRAVITY TAILINGS LEACHING

Sample	Mass		Gold				Silver			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	70.5	3.52	85	29.0	92.3	26.8	10.2	14.7	80.5	11.8
Gravity Tailings	1929.6	96.48	7.57	71.0	81.5	57.8	2.2	85.3	54.1	46.1
Calculated Head Grade	2000.0	100.00	**10.3**	100.0		**84.7**	**2.9**	100.0		**57.9**
Assay Head Grade			**9.19**				**2.5**			

Sample	Mass		Bismuth			
	(grams)	Dist (%)	Calc Head (g/t) wrt Product	Dist (%)	Leach Rec (%) wrt Product	Leach Rec (%) wrt Feed
Pan Conc	70.5	3.52	8471	14.3	0.0	0.0
Gravity Tailings	1929.6	96.48	1850	85.7	0.0	0.0
Calculated Head Grade	2000.0	100.00	**2083**	100.0		**0.0**
Assay Head Grade			**1830**			

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ABN: 30 008 127 802
64-70 Kurnall Road, Welshpool, Western Australia, 6106
Phone +61 8 9451 8477 Fax +61 8 9451 4576

CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Master Composite	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Gravity Conc	**Pulp Density:**	20
Amdel Project No:	2922	**Sample Weight (g):**	70	**Water Type:**	Perth Tap Water
Date:	12-May-09	**Grind Size:**	P_{80} 75µm	**NaCN (initial):**	20000
Metallurgist:	ASL			**NaCN (test):**	
Test Description:	Intensive Cyanide Leach			**pH:**	10.0 - 10.5
Test No:	**3A**			**Dissolved Oxygen:**	>4ppm

TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements				Reagent Additions			Cum. Reagent Usage		Notes
			pH Found	pH Left	DO (ppm)	NaCN (ppm)	NaCN (g)	Lime (g)		NaCN (kg/t)	Lime (kg/t)	
Leaching	0	20.0										
	24	20.0	11.45									

TEST RESULTS

Time (hrs)	Solution Assays (mg/l)					Solids Assays (g/t)					Extracted Grade (g/t)					Recovery (%)				
	Au	Ag	Bi			Au	Ag	Bi			Au	Ag	Bi			Au	Ag	Bi		
0																				
24	19.6	2.06	0.19								78	8	1			**92.3**	**80.5**	**0.01**		
Residue Assays						6.5	2.0	8470												
Calculated Head Grades						**84.9**	**10.2**	**8471**	**0**	**0**										

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ABN: 30 008 127 802
64-70 Kurnall Road, Welshpool, Western Australia, 6106
Phone +61 8 9451 8477 Fax +61 8 9451 4576

CYANIDE LEACH TEST REPORT

Client:	Frontier Resources	**Sample Description:**	Gravity Tailings	**Target Operating Conditions**	
Project:	Stormont Deposit	**Sample Ref:**	Master Composite	**Pulp Density:**	50% solids
Project No:	2922	**Sample Weight (g):**	1930	**Water Type:**	Perth Tap Water
Date:	11-May-09	**Grind Size:**	P_{80} - 75µm	**NaCN (initial):**	500
Metallurgist:	GL			**NaCN (test):**	300
Test Description:	Cyanide Leach Test			**pH:**	10-10.5
Test No:	**3B**			**Dissolved Oxygen:**	>4

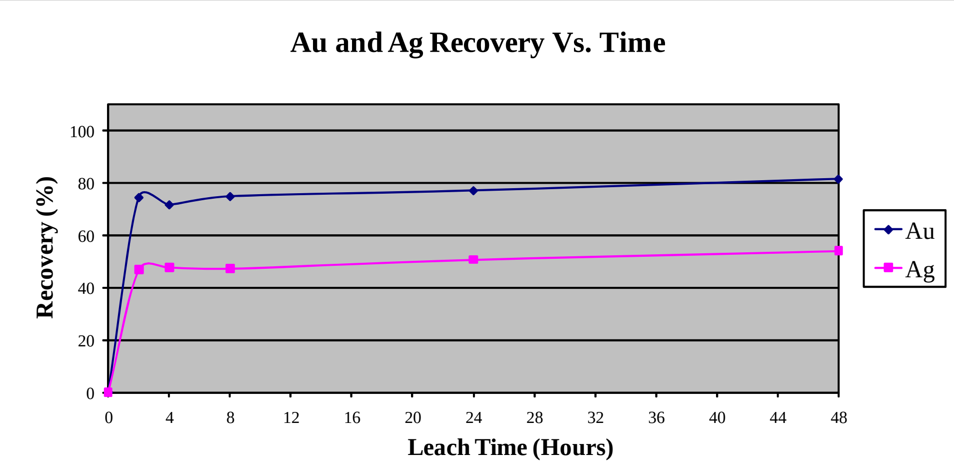
TEST CONDITIONS

Stage	Time (Hrs)	% Solids (w/w)	Pulp Measurements pH Found	pH Left	DO (ppm)	NaCN (ppm)	Reagent Additions NaCN (g)	Lime (g)	Other	Cum. Reagent Usage NaCN (kg/t)	Lime (kg/t)	Notes
Leaching	0	50.0	8.29	10.49	4.7		1.00	2.72			1.41	Test carried out using Perth tap water
	2	50.0	10.23	10.37	6.5	357	0.27	0.22		0.16	1.52	~50mls of slurry removed at each sampling time
	4	50.0	10.23	10.51	6.6	406		0.30		0.24	1.68	Solids returned & top-up solution added back in
	8	50.0	10.22	10.50	6.1	347	0.29	0.35		0.30	1.86	
	24	50.0	10.32	10.53	6.0	197	0.58	0.37		0.60	2.05	
	48	50.0	10.31		7.0	173				0.92	2.05	

Total NaCN added	2.14	gms
Nett added	2.11	gms (less NaCN in solution samples)

TEST RESULTS

Time (hrs)	Solution Assays (mg/l) Au	Ag	Bi	Solids Assays (g/t) Au	Ag	Bi	Extracted Grade (g/t) Au	Ag	Bi	Recovery (%) Au	Ag	Bi
0										0.0	0.0	
2	5.64	1.02					5.64	1.02	0	74.5	46.9	
4	5.37	1.03					5.43	1.04	0	71.7	47.8	
8	5.61	1.02					5.67	1.03	0	74.8	47.3	
24	5.65	1.07					5.84	1.10	0	77.1	50.7	
48	5.92	1.13	0.14				6.17	1.18	0	**81.5**	**54.1**	**0.01**
Residue Assays				1.40	1.0	1850						
Calculated Head Grades				**7.57**	**2.2**	**1850**						



APPENDIX 5

List of appended digital data files

RL 4-2005 River Lea Annual Report.doc
RL 4-2005 Test No 01 Master Comp @ P80 150µm - Gravity Separation + Products CN Leaching
RL 4-2005 Test No 02 Master Comp @ P80 106µm - Gravity Separation + Products CN Leaching
RL 4-2005 Test No 03 Master Comp @ P80 75µm - Gravity Separation + Products CN Leaching