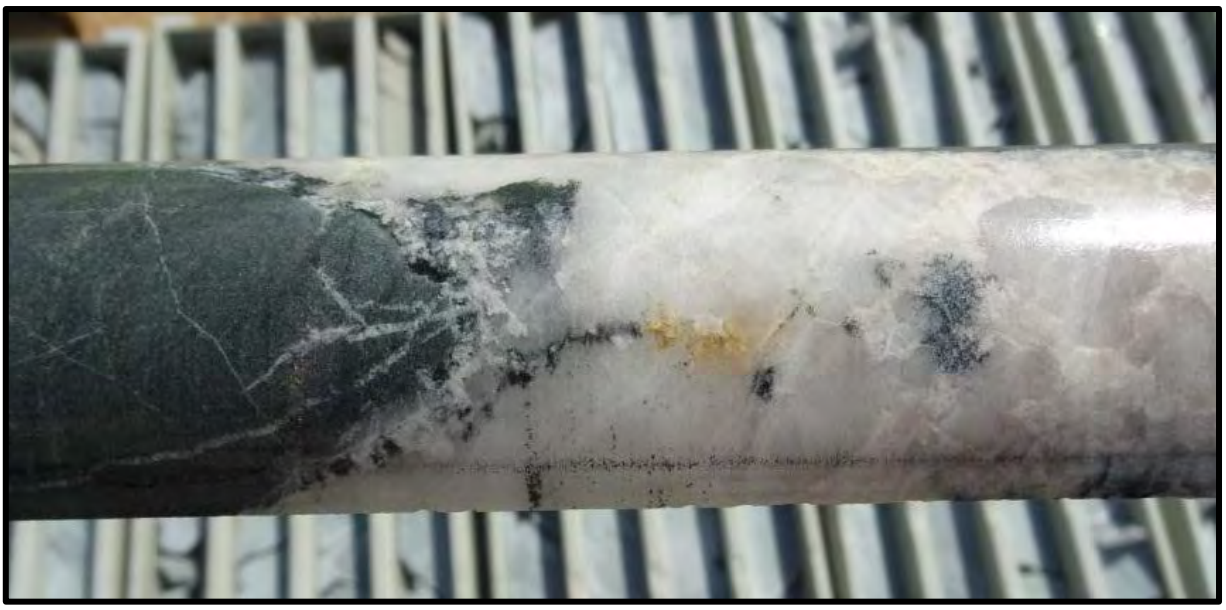
PRELIMINARY FEASIBILITY STUDY
SMARTS - HICKS OPEN PITS
KAROUNI GOLD PROJECT
GUYANA

TROY RESOURCES LIMITED

NI43-101 TECHNICAL REPORT



Visible Gold in Diamond Drill Core Smarts Deposit

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DATE AND SIGNATURE PAGE

The effective date of this Technical report, titled "NI 43-101 Technical Report, Troy Resources Limited, Preliminary Feasibility Study of Smarts – Hicks Open Pits - Karouni Gold Project, Guyana", is July 31, 2014.

"signed"

_____

Mr. Peter Doyle FAusIMM

Dated: September 8, 2014

"signed"

_____

Mr. Richard Maddocks FAusIMM

Dated: September 8, 2014

1. Summary

Troy Resources Limited (Troy) has prepared this technical report for the Karouni Gold Project (the Project) in accordance with National Instrument 43-101 "Standards of Disclosure for Mineral Projects" and in respect of a preliminary feasibility study (PFS) on the Karouni Gold Project. The PFS is preliminary in nature and is based on an open pit mine on the Smarts and Hicks Deposits that make up part of the Karouni Gold Project. It assumes processing on site. The outcome of the PFS will form the basis for ongoing exploration and development as well as construction of a mining operation.

1.1 Property Description and Location

The Karouni Gold Project is located in the central northern portion of Guyana and lies approximately 180km south-southwest of the capital of Guyana, Georgetown (see Figure 1-1).

The Karouni Gold Project covers approximately 40km along strike of the mineralised Omai - Smarts-Hicks-Whitehall (Kaburi) Corridor. This corridor also hosts the historical Omai Gold Mine which is located about 35km to the southeast of the Project. A summary of the key features of the Karouni Gold Project are:

- Hicks Deposit – a near surface Mineral Resource defined in 1996. Recent infill drilling has resulted in a new Mineral Resource and Reserve estimate which is documented in this report.
- Smarts Deposit – an artisanal bedrock gold anomaly where an Indicated and Inferred Mineral Resource was announced in August 2013. Recent infill drilling has resulted in a new Mineral Resource and Reserve estimate which is documented in this report.
- Larken Prospect – a small artisanal open pit in saprolite, approximately 50m wide by 120m in strike and less than 15m deep, that was mined in the mid to late 1990's. The pit lies approximately 700m to the northeast of the Hicks Zone and is likely to be a sub parallel structure to the one that hosts both the Hicks and Smarts Deposits. A new Inferred Resource estimate has been completed based on 2013 - 2014 drilling which is documented in this report.
- Numerous targets and alluvial workings which require further investigation.

1.2 Ownership

Through its wholly owned subsidiary Pharsalus Gold Inc. (Pharsalus Gold), Troy owns and manages the Karouni Gold Project which has an aggregate area of 113,581ha, comprising small, medium and large scale tenements. The permits grant the holders the right to explore for gold and in some cases diamonds and/or precious stones. The tenements have been acquired by either direct grant to Pharsalus Gold (10,517ha) or by vendor agreements (103,064ha). Apart from the Kaburi Agreement (11,794ha), which provides for Pharsalus Gold to earn a 90% interest, all other vendor agreements once options are exercised will provide Pharsalus Gold with an ultimate interest of 100%.

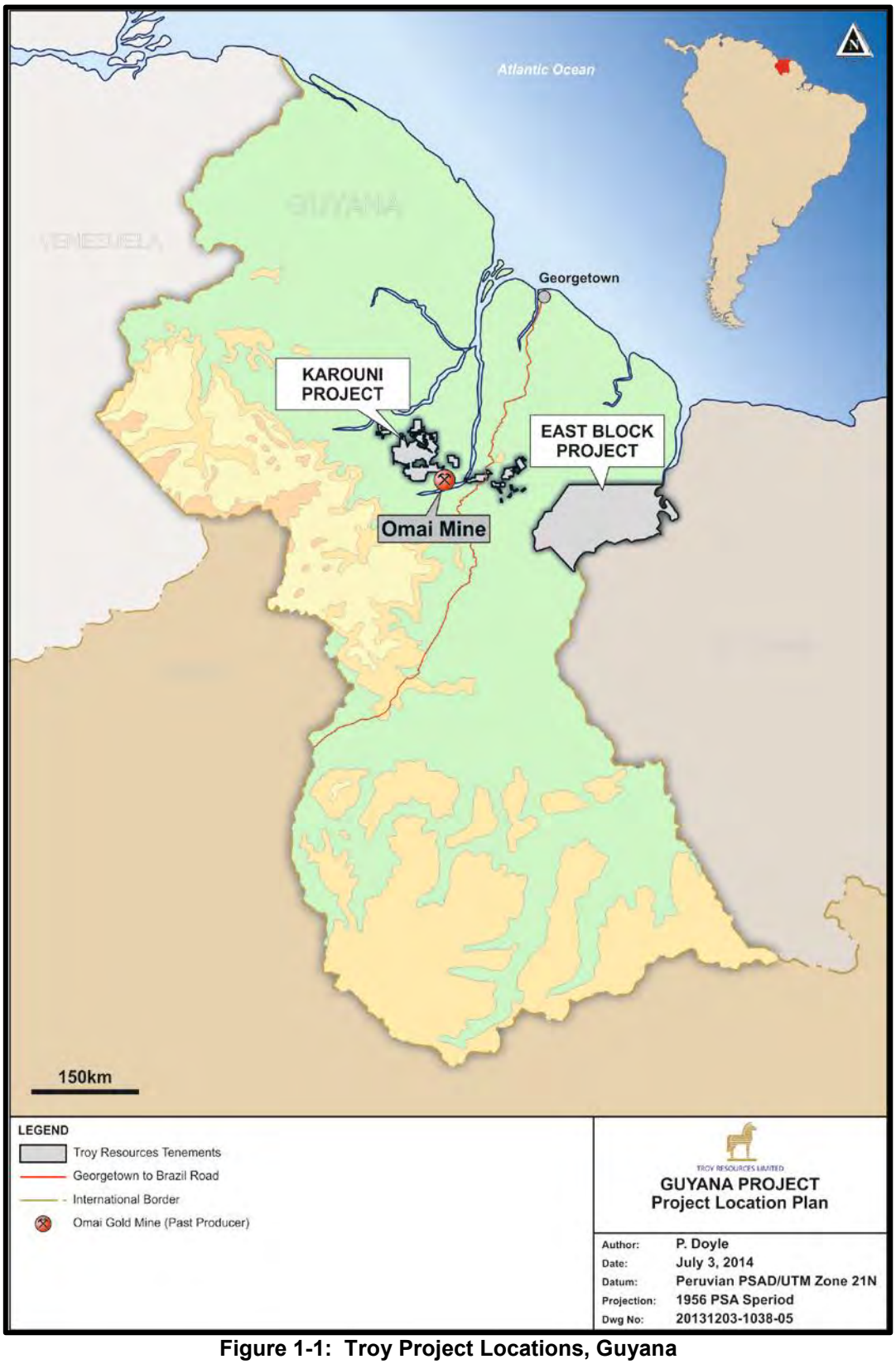

Figure 1-1: Troy Project Locations, Guyana

Troy holds the Karouni Gold Project rights through Pharsalus Gold Inc. (see Figure 1-2).

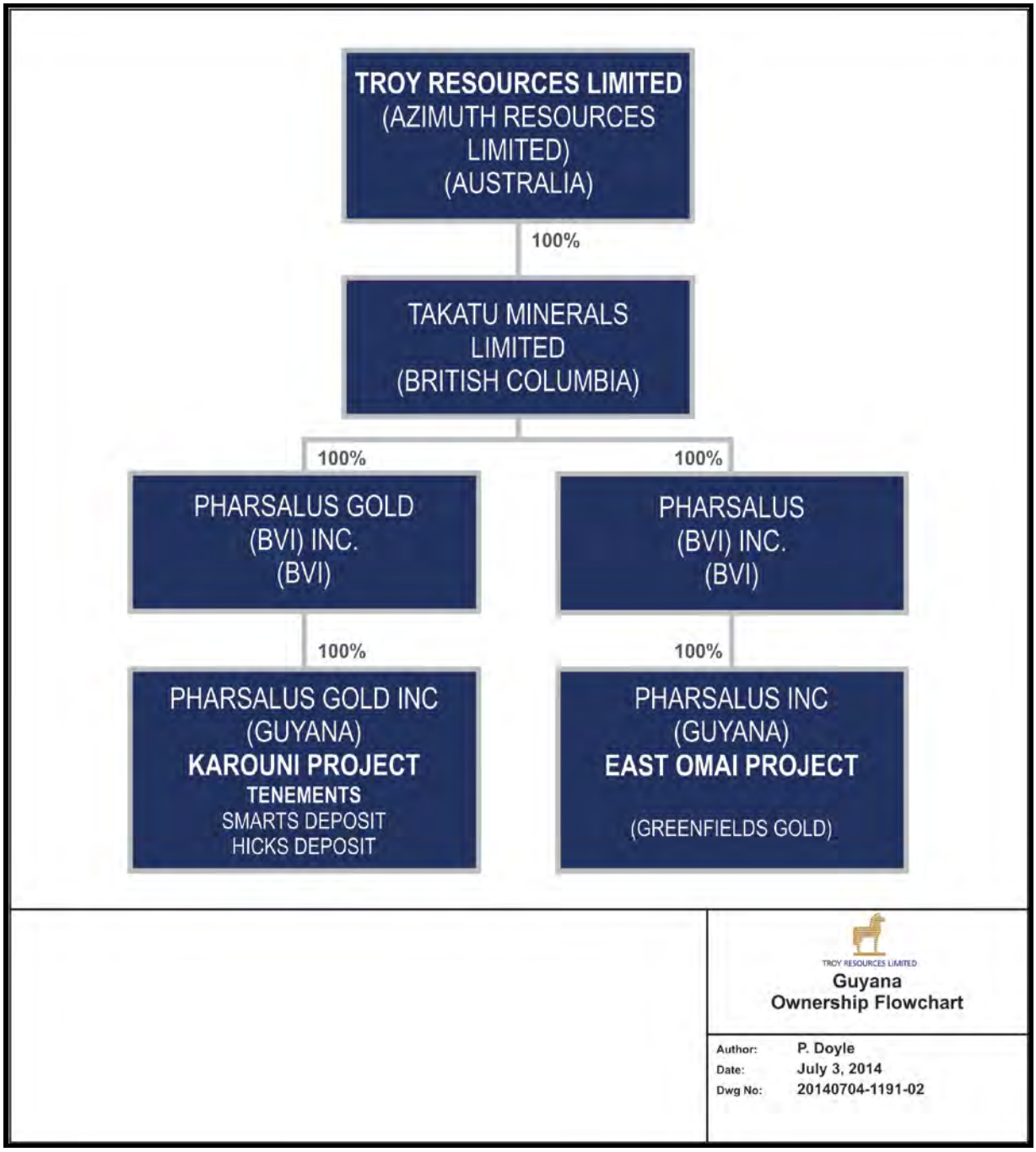

Figure 1-2: Corporate Structure for Guyana Assets - Troy Resources Limited

1.3 Geology and Mineralisation

Primary gold mineralisation is exposed at several localities within the Karouni Gold Project, the most notable being the Hicks, Smarts and Larken Zones along the northern extremity of the Project (see Figure 1-3). Here the White Sand Formation cover has been removed by erosion to expose the underlying Palaeoproterozoic greenstone successions which host a mineralised northwest trending, sub-vertical to steeply southwest dipping shear zone in the order of 40m to 60m in width. Extensive superficial cover of White Sand Formation within the central and southern portions of the project tenements masks the basement lithology and conceals any gold mineralisation. The evaluation of regional airborne geophysical data has however indicated that the Barama-Mazaruni Greenstone Belts and associated syntectonic intrusives persist at shallow depth beneath this cover.

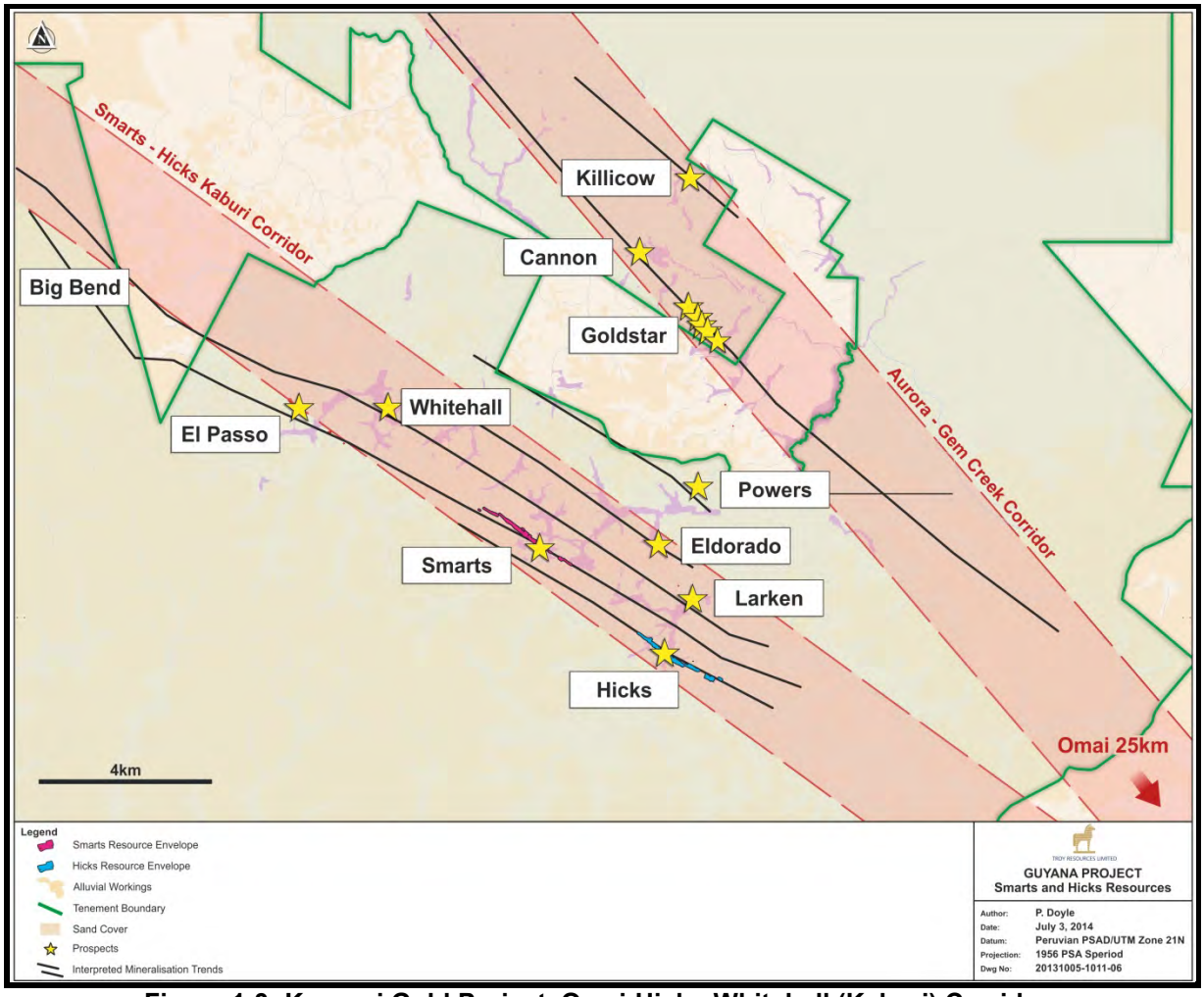

Figure 1-3: Karouni Gold Project, Omai-Hicks-Whitehall (Kaburi) Corridor

Secondary gold mineralisation at Eldorado and Whitehall (Kaburi) was mined from partially lithified, immature quartz conglomerates and arkoses (presumed to be of Tertiary age), which discontinuously overlie mineralised portions of the basement. Alluvial deposits are also being actively mined along the Kuribong and Potaro Rivers (and tributaries) that comprise the south-western project boundary.
No mineral production is officially recorded within the Karouni Gold Project area, however historic records indicate that the Eldorado Vein was worked in 1913 via limited shaft and drift development. The Whitehall (Kaburi) Pit is believed to have produced saprolite-hosted and alluvial gold to a depth of 5m to 15m although no official production records have been compiled.

1.4 Mineral Resource – Mineral Reserve Estimate

The reported Mineral Resource and Mineral Reserves for the Karouni Gold Project have been prepared by/or under the supervision of Mr. Peter Doyle and Mr. Richard Maddocks, who are full-time employees of Troy. Mr. Doyle has visited the Project site 8 times between May 2013 and July 2014. Mr. Maddocks visited the site from October 23 – 30, 2013.

Table 1-1 shows the Mineral Resource estimate for the Karouni Gold Project. This includes the Smarts Deposit, Hicks Deposit and Larken Prospect which are discussed individually later in this report. A significant portion of the Mineral Resource estimate has now been upgraded into the Measured Category reflecting the increased understanding of the geology and controls on mineralisation, especially at Smarts. This increased understanding has however also resulted in a reduction of the Inferred Resource particularly at depth in the Smarts Deposit. Resource outlines were constructed using three dimensional interpretations based on mineralised envelopes, constructed using logged geology, in particular the shear zone, and a nominal 0.5g/t gold cut-off grade with a minimum 2m downhole width and no edge dilution. Minor zones of internal dilution were included to maintain continuity of the resource wireframes. A LIDAR topographic survey was

used to create a topographic surface. Geological logging was used to create weathering surfaces for the top of fresh (tofr) and base of oxidation (boco) surfaces. A base of overburden surface and cross-cutting dolerite dyke were also interpreted. All modelling and wireframing were completed using Vulcan Software. The estimates for the deposits used standard Vulkcan block models using Ordinary Kriging (OK) grade interpolation for Hicks and Smarts and inverse distance squared for Larken, within wireframes, and the models have been reported at a 1.0g/t gold cut-off.

Table 1-1: Karouni Gold Project: June 30 2014 Mineral Resource Estimate									
Cut-off Grade	**0.5g/t Cut-off**			**1.0g/t Cut-off**			**2.0g/t Cut-off**		
Catagory	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Ounces**
Measured	2,510,000	4.4	359,000	2,205,000	5.0	351,700	1,680,000	6.0	326,700
Indicated	4,874,000	2.2	338,400	3,914,000	2.5	314,100	1,805,000	3.7	215,700
M & I	**7,384,000**	**2.9**	**697,400**	**6,119,000**	**3.4**	**665,800**	**3,485,000**	**4.8**	**542,400**
Inferred	8,048,000	1.8	470,200	5,692,000	2.2	411,500	2,303,000	3.5	259,300

The Mineral Reserve estimate is summarised in Section 15 of this report. The Mineral Reserve estimate is based on the Mineral Resource estimate described above. The Mineral Reserve estimate has only considered Open Pit mining at Smarts and Hicks Deposits.

The Mineral Resource estimate was optimised using proprietary software utilising Lerchs Grossman algorithms to produce a series of optimal pit shells. Cost inputs are summarised in the operating costs section of this report. The optimal pit shell was based on a gold price of US$1,300/oz.

The geotechnical parameters used in the optimisation are tabulated in Table 1-2. The optimisation assumed “ore loss” of 5% and mining dilution of 10%. These are appropriate factors considering the average ore width and dip.

A processing recovery of 94% was applied to all ore types.

Table 1-2: Karouni Gold Project Open Cut Proven & Probable Mining Reserves Estimate (June 30 2014)									
	Proven			**Probable**			**TOTAL**		
Ore Type	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Ounces**	**Tonnes (t)**	**Gold Grade (g/t))**	**Gold Ounces**
Oxide	194,000	5.0	31,200	251,000	2.0	15,700	445,000	3.3	46,900
Transitional	213,000	5.2	35,800	79,000	1.9	4,900	292,000	4.3	40,700
Fresh	1,331,000	4.6	197,400	550,000	2.2	38,300	1,881,000	3.9	235,700
Total	**1,738,000**	**4.7**	**264,400**	**880,000**	**2.1**	**58,900**	**2,618,000**	**3.8**	**323,300**

Notes: Mineral Reserve estimate effective as of 30 June 2014

1.5 Development

1.5.1 Overview

The PFS considers a combination of two open cut mines feeding a conventional carbon-in-leach (CIL) gold plant with a nominal capacity of 1Mtpa. The PFS assumes that a total of 2.61 million tonnes of material will be processed with an average grade of 3.84g/t gold with recovered gold production of 303,521 ounces over a 3 year mine life.

Highlights from the PFS, assuming a gold price for the base case of US$1,250/oz, are as follows (all figures in US$ unless otherwise stated):

- Three year open pit mine life with annual average gold production of 101,000 ounces and production in the first 12 months of 104,400 ounces.
- Conventional CIL plant augmented with gravity gold recovery treating a nominal 1Mtpa configured to allow easy low cost expansion at a later date.

- Approximately 2.6 million tonnes of material to be processed with an average grade of 3.84g/t gold. The sources in terms of tonnes are: Smarts - 68% and Hicks - 32% and in terms of contained gold, Smarts - 83% and Hicks - 17%.
- The Smarts Pit is expected to produce 1,774,000 tonnes of plant feed at 4.70g/t gold, have a mining strip ratio of 9.6:1 and be mined to a depth of 120m.
- The Hicks Pit is expected to produce 840,000 tonnes of plant feed at 2.02g/t gold, have a mining strip ratio of 4.6:1 and be mined to a maximum depth of 80m.
- Initial capital of $84.6 million (including pre-production mining costs of $11.3 million and contingency of $4.7 million) and sustaining capital and capital spares over the life of mine of $6.4 million.
- Assumed metallurgical recovery of 94%.
- LOM average C1 Cash Costs (excluding royalties) of $480/oz.
- LOM All in Cash Costs of $602/oz.
- After tax payback of 1.2 years.
- After tax NPV at 6% of $72.0 million.
- After tax IRR of 50.2%.

The Project licences' are considered highly prospective with only limited exploration having been conducted to date. Numerous drill targets have already been identified within trucking distance of the proposed plant site.

1.5.2 Project Parameters

The PFS assumes owner mining of the ore feeding a conventional CIL plant. Figure 1-4 shows the annualised gold production and mill feed grade. The Smarts Deposit will be mined as a four stage development with the deepest part of the pit being 120m below the surface.

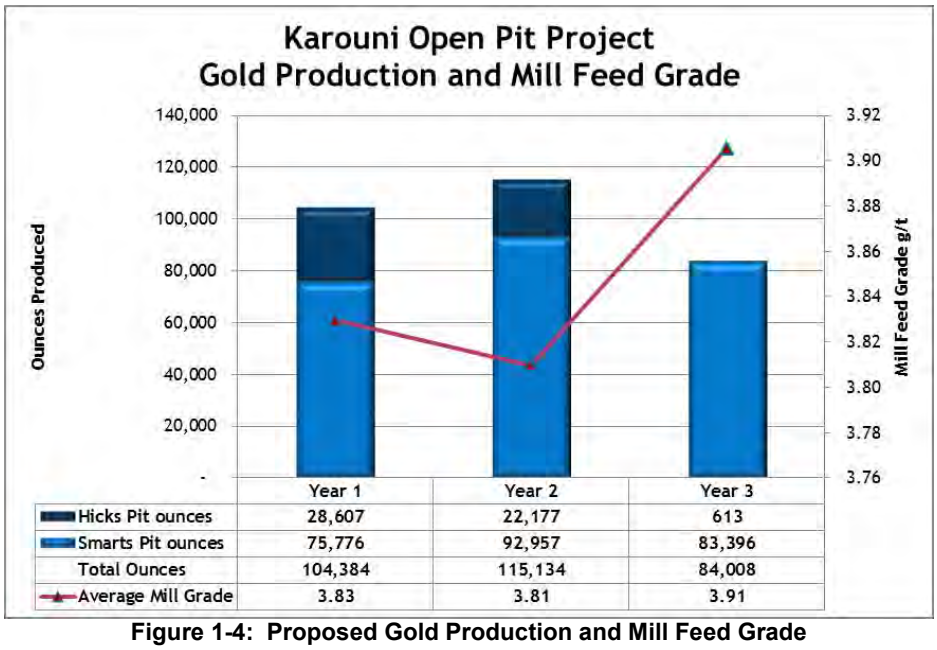

Figure 1-4: Proposed Gold Production and Mill Feed Grade

The lower grade Hicks Deposit will be mined via 3 small open pits, the deepest of which would extend 80m. Using a flexible mining fleet with a maximum of three excavators, mining will be scheduled between the Smarts and Hicks pits to optimise grade to the mill and earth moving requirements taking into account the various strip ratios. At this stage underground mining has not been included in the PFS. Drilling is continuing into the Smarts Deeps Target to further define the geometry and extent of the high grade shoots. The processing plant will be a conventional CIL with a nominal throughput capacity of 1MTPA. Ore will be fed through a primary jaw crusher to a secondary cone crusher. Crushed material will be fed to a 3.2 MW ball mill for grinding with material passing through a gravity circuit to recover coarse gold before the addition of cyanide. Gold solution will be stripped using

activated carbon and smelted to produce doré on site. The doré will be flown to the Guyana capital, Georgetown, for export to a refinery. Waste tailings will be pumped to a tailing facility.

Employees will be housed in an accommodation village located near the processing plant. Employees will work a rotating roster with transport provided to and from Georgetown. An airstrip will allow contract flights to and from Georgetown. Fresh water will be stored in tanks and sourced from rainwater. Fuel and other consumables will be trucked in from Georgetown and/or Linden along a dedicated access road.

1.5.3 Mining

Mining will commence approximately six months before the processing plant is scheduled to be commissioned. This will enable removal of sand overburden and adequate high grade stocks to be accumulated prior to the commencement of processing. An initial fleet comprising two excavators and up to 8 trucks will commence at the Smarts and Hicks Deposits. There is overburden consisting of white sand up to 30m thick covering much of the northern part of Smarts Deposit whereas the Hicks Deposit outcrops. Initial production will be predominantly sourced from the Hicks Deposit while Smarts overburden is removed. Production will then focus on the higher grade Smarts Deposit. An additional fleet will be mobilised after 3 months to provide further capacity for the Smarts Deposit. Mining will then progress with three excavator fleets until the pits are complete.

Mining will be done on 2.5m flitches over a 5m high blast. Initial sand overburden removal will be free dig with no blasting required (see Table 1-3 below).

Table 1-3: Open Pit Design Parameters

Batter Angle	**Overburden**	**35°**
	Oxide	**50°**
	Transitional	**55°**
	Fresh	**75°**
Batter Height	**20m vertical 10m in oxide**	
Ramp Width	**12m**	
Berm Width	**7m in fresh, trans 4m in oxide 3m overburden**	
Ramp Gradient	**1 in 9**	

1.5.4 Metallurgical Testwork and Recovery

Communition Testing

Unconfined Compressive Strength results showed that there would be no issues processing any of the material tested through a standard jaw crusher on the basis of required energy. The oxide material was classified as very weak, while the primary and shear materials were classed in the range of weak to medium-strong. The Crushing Work index results showed that there will be no identified issues through the crushing circuit. The results ranged from 0.4kWh/t to 17.9kWh/t for one of the sulphide samples, with the high sulphide result being classified as hard. The remaining sulphide tests returned values of soft to medium, indicating that this was an outlier. The Bond Work index results indicated that all samples were medium to hard, across all zones tested. The range of values was between 12.57kWh/t and 15.28kWh/t, with the highest value being recorded by the sulphide ore, and the lowest by the shear ore. Rod mill work index testing was conducted on the primary and shear master composites. Both results showed the material to be hard, with results of 19.9kWh/t and 17.5kWh/t for the primary and shear zones respectively. The primary material showed that there may be some build-up of critical sized material. Due to the lower grinding feed size and higher ball content it is not expected that this will be an issue for the grinding circuit.

Gravity – Cyanidation Leach Optimisation

A series of tests were conducted to determine the optimal grind size and cyanide dosage for the leaching process. As part of this testing process the effectiveness of the gravity stage was evaluated. Initial testing replicated the historical testwork, using a 72 hour leach time. It was determined, after the first round of testing, that a 24 hour leach time would be adopted due to negligible improvements in leach recovery being achieved from longer leach times. The testwork program, conducted on the oxide, primary and shear master composites, determined that a P80 of 63 microns would be used for subsequent testing and throughout the process plant, with a cyanide dosage of 250ppm. Under these conditions all samples displayed fast leaching kinetics and high recoveries. The gravity recovery varied between the three composites, with the oxide displaying a gravity recovery of around 30%, the shear 40% and the primary 60%. The initial evaluation of the master composites for the Project was to determine the grind size to be used for further testing. Following on the results observed during the historical testing, which indicated that there was an increase in gold extraction as the grind size was reduced, it was decided to test three grind sizes, being P80 106, 75 and 45 micron. This would allow for a comparison to the previous testing at P80 106 and 75 micron, and also allow for the results at the coarser grind size to be compared to a finer grind. The results from this series of testwork are summarised in Table 1-4 below.

Table 1-4: Grind Size - Cyanidation Leach Optimisation Testwork Summary

Composite	Grind Size (P80, µm)	Calculated Head Grade (g/t) Au	Gravity Gold Recovery	8h Gold Recovery	24h Au recovery	72h Au recovery	Lime consumption kg/t	NaCN Consumption (kg/t)
Oxide MC	106	3.15	28.4%	92.8%	94.4%	95.3%	1.76	0.32
Oxide MC	75	3.15	30.3%	94.5%	97.0%	95.6%	1.76	0.32
Oxide MC	45	2.78	23.7%	93.3%	92.8%	94.6%	1.88	0.34
Primary MC	106	5.78	56.6%	86.4%	89.9%	91.0%	0.93	0.23
Primary MC	75	6.48	58.7%	90.2%	91.8%	93.7%	0.95	0.21
Primary MC	45	8.98	62.2%	96.0%	96.8%	96.6%	1.09	0.23
Shear MC	106	6.09	41.9%	89.8%	93.1%	92.0%	1.08	0.31
Shear MC	75	6.67	42.4%	93.2%	94.9%	94.4%	1.28	0.37
Shear MC	45	5.58	37.5%	95.4%	96.0%	96.4%	1.30	0.33

Following the results of the initial optimisation testwork it was decided that additional work was required to further optimise the process. As seen in Table 1-5, at the 24 hour mark there was a 5% increase in gold extracted for the primary material, and a 1.1% increase in the shear material, between the P80 75 and 45 micron grind size. As a result of this it was decided that both a P80 63 and 53 micron grind would be tested on the primary material only, due to its greatest sensitivity to grind size. It was decided that the leach would be undertaken under similar conditions to the earlier master composites tests, with the only change, other than the grind sizes, being the leach time being limited to a 24 hour leach. The key data from the additional grind variation tests are shown in Table 1-5 below.

Table 1-5: Grind Variation Testwork Results

Composite	Grind Size (P80, µm)	Calculated Head Grade Gold (g/t) Au	Gravity Gold Recovery	8h Gold Recovery	24h Gold Recovery
Primary MC	75	6.48	58.7%	90.2%	91.8%
Primary MC	63	4.8	67.9%	93.3%	95.0%
Primary MC	53	4.91	66.1%	93.9%	94.8%

Primary MC	45	8.98	62.2%	96.5%	96.8%

The leach tests on the new grind sizes displayed a more consistent head grade than the previous tests, and also displayed a greater gravity recovery. The recoveries from gravity, and at all leach time intervals, between these two grind sizes however were very similar to each other. It is noted however that there is an increase in overall recovery similar to that seen at the P80 45 micron leach compared to the P80 75 micron leach. As the P80 75 micron leach showed a very similar recovery in this series of testwork compared to the historical testing, it is deemed that below 75 micron on the primary material there is a significant amount of gold that is unlocked and recoverable during the leach. Further grinding, below P80 63 microns, does show a greater amount of liberation of the gold, however not to the same significant increase seen when moving below 75 microns. Based on these findings it was determined that a P80 of 63 micron would be used for further testing on the Karouni material.

Variability Testing

Variability testing was conducted on 5 primary samples only, due to the amount of available mass for this ore zone, and the limited mass of the other ore zones. The variability samples were formed from high and low points in the primary zone, as well as over a range of gold grades to assess the effect of grade and depth variation. The results show that the gravity recovery ranged between 42% and 79%, almost independent of the head grade. Overall recovery ranged from 79% to 97%. There was a slight dependence on head grade, with the two lowest head grades displaying the lowest overall recovery. The cyanide consumption was low for all samples tested, while the lime consumption was at levels below concern. It is thus concluded that there is no undue reagent costs to be incurred for the primary ore.

1.5.5 Permitting

Necessary government approvals are in the process of being finalized. All required documentation has been submitted and is following the governmental internal process. The Minerals Agreement, which is the definitive agreement allowing for project exploitation, tax and royalty issues, importation allowances, physical and economic parameters for the extraction and sale of minerals, is in the final stages of its approval process.

1.5.6 Capital and Operating Costs

The estimated capital cost to construct and commission the Project is US$73.3 million, which includes a contingency of US$4.7 million. A further US$11.3 million is required for pre-production mining, including waste pre-strip and building a high grade stockpile ahead of treatment through the processing plant. Sustaining capital over the life of mine is estimated at US$2.4 million and a further US$4.0 million has been included for working capital spares. These costs include all procurement, delivery and construction direct and indirect costs. The estimates are based on the PFS design and are considered to have an accuracy of +10% to -10%.

Operating costs have been developed from first principles following a review of local market conditions and are shown in Table 1-6 below.

Table 1-6: Operating Cost Summary

Cost	Life of Mine (US$/oz)
Mining (ore and waste)	225
Processing	203
Administration	38
Refining	14
C1 Cash Costs	**480**

Royalties	113
Sustaining capital	8
All in Sustaining Cash Costs (AISC)	**US$602/oz**

1.5.7 Royalties and Taxes

The payment of gross production royalties are provided for by the Guyana Mining Act and the amount of royalty to be paid is prescribed by the Minister. As such royalties may be varied from time to time. Based on currently prescribed royalties and recent mineral agreements, royalties on gold production have been assumed as follows:

- 8% if the gold price is above US$1,000 per ounce; and
- 5% at a gold price of US$1,000 and less.

Certain tenements in the mining area have an additional vendor royalty of 2%.

The corporate tax rate in Guyana for gold companies is 30%.

1.5.8 Economic Analysis

Table 1-7 below shows the Pre and Post Tax NPV for the Project at various gold prices.

Table 1-7: Pre and Post Tax NPV for the Project at Various Gold Prices

Gold Price ($US/oz)	**$800**	**$1,000**	**$1,200**	**$1,250**	**$1,400**	**$1,600**
Pre-tax NPV at 6% ($US)	-$16,652,833	$33,871,579	$74,665,186	$86,890,839	$123,567,798	$172,470,409
IRR	N/A	26.19%	49.40%	56.17%	76.12%	102.11%
Gold Price ($US/oz)	**$800**	**$1,000**	**$1,200**	**$1,250**	**$1,400**	**$1,600**
Post-tax NPV at 6% ($US)	-$16,652,833	$33,871,579	$63,285,725	$71,959,221	$97,901,012	$132,361,257
IRR	N/A	26.19%	44.78%	50.17%	66.10%	86.71%

1.5.9 Sensitivity Analysis

Table 1-8 below, summarises the Project's sensitivity to head grade, recovery, operating cost, capital cost and gold price.

Table 1-8: Project Sensitivity Analysis

Post-tax NPV (US$M)	**-20%**	**-10%**	**Base**	**+10%**	**+20%**
Head Grade	25.47	50.07	71.96	93.69	115.35
Recovery (1)	N/A	67.34	71.96	76.57	N/A
Operating Costs	90.01	80.99	71.96	62.90	53.78
Capital Costs	89.47	80.72	71.96	63.20	54.45
Gold Price	33.87	50.18	71.96	93.58	115.14

(1) Sensitivity Analysis for Recovery is calculated at 92% for the downside and 96% for upside.

1.6 Exploration

The PFS only considered Open Pit Mineral Reserves at the Smarts and Hicks Deposits. Drilling is continuing into Smarts Deeps Target to infill and extend the existing Mineral Resources in this area. Drilling is also planned to infill and extend the Larken Inferred Mineral Resource in order to upgrade that Mineral Resource estamate to Indicated.

The remaining licences are also considered very prospective with numerous targets already identified within a 30km radius (see Figure 1-5 below) of the proposed plant site. Troy intends to recommence brownfields exploration later in calendar 2014 and believes there is good potential to identify additional Mineral Resources.

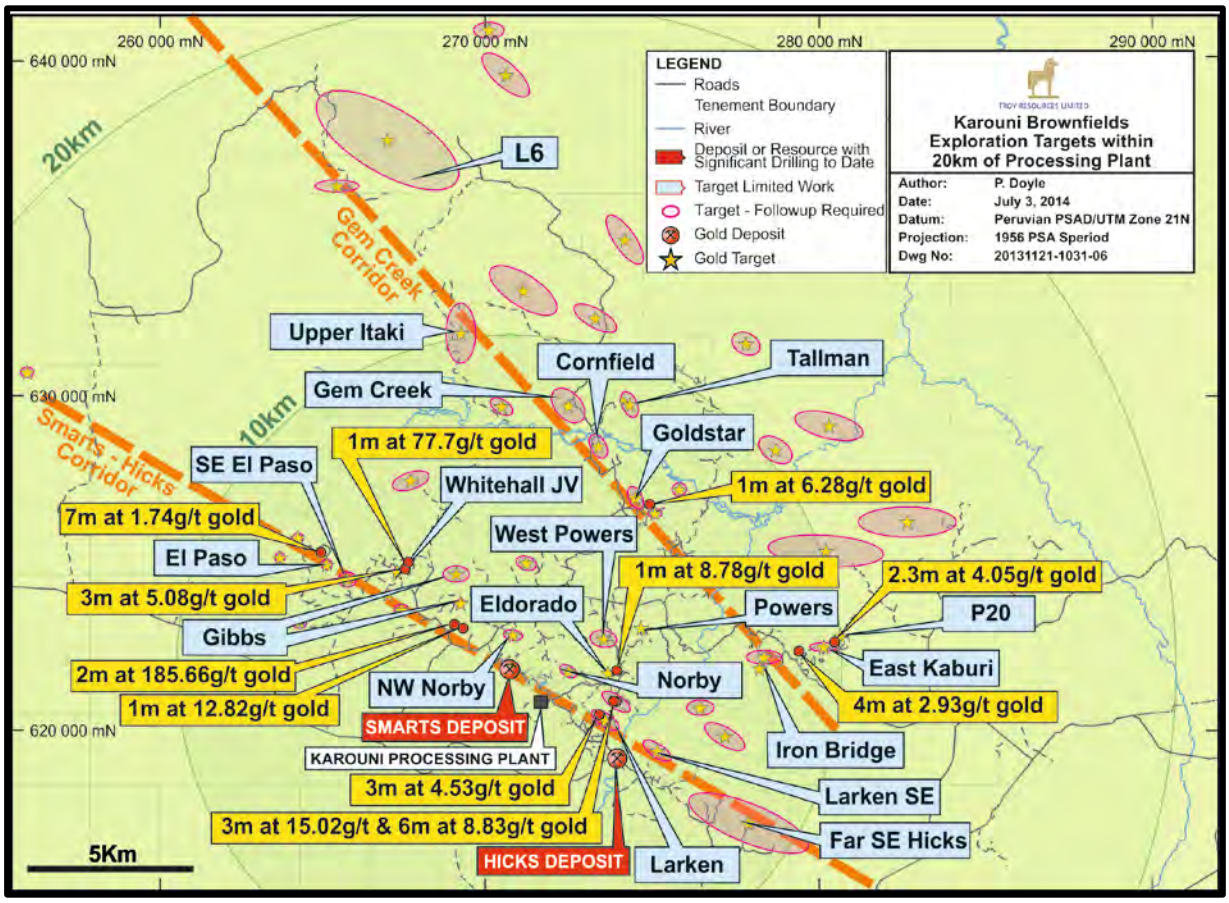

Figure 1-5: Karouni Brownfields Exploration Targets within 20km of Processing Plant

1.7 Conclusions and Recommendations

Given the positive results of the PFS it is recommended that the Smarts – Hicks Open Pit Mine be developed.

Additional drilling should be conducted to upgrade the Inferred Mineral Resources estimate on the Smarts Deposit so as to enable Mineral Reserves to be estimated.

The Karouni Gold Project area is prospective and under explored. Further exploration is warranted to investigate the project potential. This significant regional Brownfields potential warrants further systematic exploration.

2. Introduction

2.1 Terms of Reference

The Karouni Gold Project is owned and operated by Troy Resources Limited (Troy). Troy is a public company listed on the Australian Securities Exchange (ASX: TRY) and Toronto Stock Exchange (TSX: TRY), and registered in Australia.

Troy has prepared this Technical Report for the Karouni Gold Project in accordance with National Instrument 43-101 "Standards of Disclosure for Mineral Projects" following the completion of a Preliminary Feasibility Study covering the Hicks and Smarts Open Pit Gold Deposits (PFS).

2.2 Report Purpose

This report details the results of the PFS for the Karouni Gold Project as outlined in the July 28, 2014 press release by Troy.

2.3 Information Sources

This report relies extensively on information provided by Azimuth Resources Limited (Azimuth) as well as recent data generated since Troy completed its acquisition a takeover of Azimuth. Troy has engaged a number of specialist consultants and information from these reports prepared by previous independent consultants has been utilised in the compilation of this report. A full listing of the principal sources of information is included in Section 27 of this report.

2.4 Site Visits

Eight site visits to the Karouni Gold Project for the purpose of project updates, program planning and geological due diligence were undertaken by P. Doyle in 2013 (May 18 – 25; August 21 – 30; September 26 – October 3; October 23 – 30 and November 17 – 21) and 2014 (January 13 – 22, March 11 – 12 and June 25 – July 2). R. Maddocks visited the site from October 23 to October 30, 2013.

Previous site visits by Runge Pincock Minarco Global Consultants (RPM), undertaken for the purpose of geological due diligence, were conducted in October-November 2012 by Aaron Green (RPM), in December 2011 by Rod Ogilvie (RPM Associate) and October 2010 by Aaron Green.

3. Reliance on Other Experts

In respect of legal, environmental and taxation matters of this report Technical Report the authors have not relied on any external reports.

4. Property Description and Location

4.1 Property Area

The Karouni Gold Project covers an aggregate area of 113,581 hectares; comprising granted small, medium and large scale tenements west of the Essequibo River (see Figure 4-1).

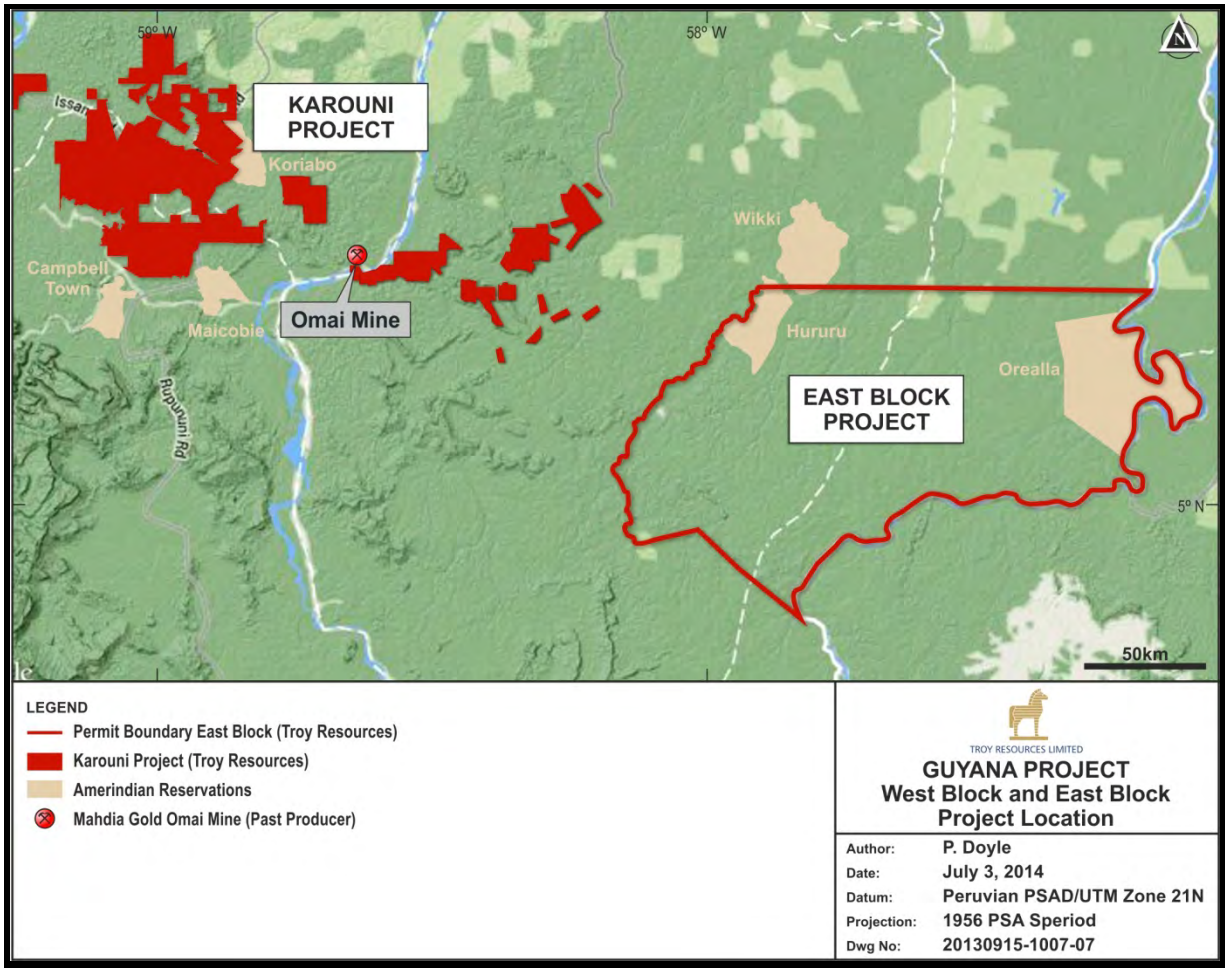

Figure 4-1: Location of the Karouni Gold Project and East Block Project Area Tenements

4.2 Property Location

The Karouni Gold Project is centred at 5º37'N and 59º05'W in the central-northern portion of Guyana (refer Figure 1-1). The project lies approximately 180km south-southwest of Georgetown.

4.3 Mineral Tenure

4.3.1 Guyana Mining Act

The ownership of all minerals within the Republic of Guyana is vested in the Co-operative Republic of Guyana. Correspondingly, the owners of surface title whose rights are governed by the State Lands Act (Chapter 61-01) have no mineral rights.

The Guyana Mining Act of 1989 ("Act"), and the regulations made under the Act ("Regulations") empower and define the duties of the Minister responsible for mining ("Minister") and the Guyana Geology and Mines Commission ("GGMC") to carry out the objects of the Act and Regulations including the grant of mineral title and supervising the conduct of mining and prospecting operations. The Act and Regulations also govern the rights, obligations and restrcitions imposed upon those granted mineral title. The Minister and GGMC enforce the procedures to be followed in the grant and regulation of all mineral titles in Guyana. At the present time the Minister responsible for mining is the Minister of Natural Resources and the Environment.

The types of mineral title granted under the Act that are relevant to Troy current operations are as follows:

- Prospecting permits for small scale operations which grant the permit holder the right to carry out prospecting operations (which is defined as operations carried for or in connection with exploration for any mineral) and locate and mark out claim licences;
- Claim licences which grant the claim holder the right to carry out small scale mining operations;
- Prospecting permits for medium scale operations which grant the permit holder the right to carry out prospecting operations and may be converted into mining permits for medium scale mining operations;
- Mining permits for medium scale operations which grant the permit holder the right to carry out medium scale mining operations;
- Prospecting licences which grant the licensee the right to carry out prospecting operations and may be converted into mining licences for large scale mining operations;
- Mining licences which grant the licensee the right to carry out large scale mining operations; and
- Permission for geological and geophysical survey ("PGGS") which grant the holder the right to conduct geological and geophysical surveys over a larger area of land than a prospecting licence.

The first four types of titles can only be granted to and held by Guyanese citizens or companies whose shareholders are exclusively Guyanese citizens. The last three types of titles may be granted to either foreign or Guyanese citizens or entities. Despite the restrictions on foreign ownership for the first three title types, it is possible for foreign citizens and entities to enter into exclusive commercial contractual arrangements to explore develop and mine within such tenements. The Civil Law of Guyana Act (Chapter 6-01, Section 3) determines that English Common Law applies as the common law of Guyana and accordingly all contractual commercial agreements are based on English Common Law.

The definition of small, medium and large scale operations is prescribed by the Regulations or from time to time by notice from the Minister. Currently, small scale operations are considered to be operations in which less than 200 cubic metres of material is mined per day, medium scale operations in which 200 to 1,200 cubic metres of material are mined per day and large scale operations are operations in which greater than 1,200 cubic metres of material are mined per day.

The period for which mineral titles and the conditions under which titles are granted and renewed are stipulated by the Act and are summarised as follows:

- Prospecting permits for small scale and medium scale operations are granted for one year and are essentially renewable indefinitely once application for renewal is made and prescribed fees paid to the GGMC. However, once the holder discovers a "mineral of anomalous occurrence" (a term which not defined under the Act) an application for a claim licence or mining permit must be made to the GGMC within 60 days (or such longer period as the GGMC may allow) of the discovery or the prospecting permit shall lapse.

- Claim licences for small scale operations are granted for one year and may be renewed indefinitely. However, the GGMC may not renew a claim licence if it has reasonable grounds to believe that an applicant for renewal does not intend to carry on in good faith, within the limits of their competence and resources, mining operations within the claim licence. The Act defines mining operations in broad terms as "operations carried out for or in connection with the mining of minerals".

- Mining permits for medium scale operations are granted for five years and may be renewed for successive periods of not greater than five years or the life of the mineral deposit whichever is shorter. However, the GGMC may not renew a mining permit if it has reasonable grounds to believe that an applicant for renewal does not intend to carry on in good faith, within the limits of their competence and resources, mining operations within the permit.

- Prospecting licences are granted for a period of three years and may be renewed twice for a period of one year. Grant and renewal of prospecting licences are subject to the applicant/licensee having sufficient financial and technical capacity to carry out prospecting operations, the applicant's/licensees proposed annual prospecting operations being considered adequate by the GGMC and the applicant/licensee having made adequate provision for the employment and training of Guyanese citizens. In addition the GGMC may grant or renew a prospecting licence under any special circumstances that it deems adequate or warranted by exceptional circumstances.

- Within 30 days of the discovery of "a mineral of anomalous occurrence" the holder of a prospecting licence must notify the GGMC of the discovery in writing and promptly undertake to quantify the mineral occurrence. Within a further 30 days after the initial discovery notice, a licensee must notify the GGMC whether or not he considers the discovery to be of commercial interest after which a licensee may at any time within one year from the notice of commercial interest apply for a mining licence. Notwithstanding the foregoing the GGMC with the Ministers approval may allow the licensee further time which the GGMC considers reasonable for the licensee to apply for a mining licence. If the licensee applies for a mining licence in line with this procedure and the application otherwise complies with the Act and Regulations the Minister must grant a mining licence to the licensee over the area applied for.

- Mining licences may be granted for a period of 20 years and subsequent renewals are for periods not exceeding 7 years. Renewals may be refused if the licensee is in default of the Act or the Regulations.

- An application for a mining licence requires the applicant to submit a detailed proposal for the establishment and conduct of mining operations, all supporting data and any other particulars as may be prescribed. A mining licence requires its holder to carry out development and mining operations in accordance with the forgoing detailed proposal, and duly notify the GGMC upon the commencement of production. Other conditions that are currently prescribed are:
    - approval of an environmental impact statement by the Guyana Environmental Protection Agency;
    - the submission of an adequate mine closure plan in accordance with the Regulations;
    - compliance with obligations to keep accurate production records; and
    - the submission of half-yearly and annual reports to the GGMC on all prospecting and mining operations.

- The Minister may grant a PGGS for the purpose of geological and geophysical surveys or any other investigation in Guyana or any part of it, which the Minister considers relevant for the prospecting or mining of minerals on terms and conditions that may be negotiated between the Minister and the applicant for the PGGS.

- Typically the holder of a PGGS has the exclusive right to file applications for a number of prospecting licences (as negotiated and defined by the terms of a PGGS) located within the area of a PGGS at any time prior to the expiry of the PGGS. The granting of a time extension to a PGGS may be negotiated but the grant of such extensions is solely at the discretion of the Minister.

For all of the above types of title, the mineral to be mined or prospected for must be specified by an applicant for mineral title and the title documents will state the mineral or minerals for which title is granted. Furthermore the holder of a prospecting permit or prospecting licence may in general only apply for a claim licence, mining permit or mining licence for the mineral or minerals the subject of the preceding prospecting permit or licence.

Regulations defining the size of and fees payable for each type of mineral title are prescribed by the Minister and may be varied from time to time. Currently the fees and prescribed sizes of the various mineral titles are as follows:

- Prospecting permits for small scale operations cost US$5.00 per year and allow prospecting operations for the location of claim licences for small scale operations within any part of any one of the five mining districts in Guyana.

- Claim licences may be no larger than 800 feet (243.8m) by 1,500 feet (457.2m) and may not be located within other previously existing granted mineral title. The annual fee for claim licences is US$5.00 per licence.

- Prospecting permits for medium scale operations may be between 150 and 1,200 acres. Annual fees commence at US$0.25 per acre and increase by US$0.10 per acre upon each subsequent annual renewal.

- Mining permits for medium scale operations may be between 150 and 1,200 acres. Annual fees are US$1.00 per acre.

- Prospecting licences may be no larger than 12,800 acres. Annual fees are US$0.50 per acre during the first year, US$0.60 per acre during the second year and US$1.00 per acre for the third year and for each of year of the two subsequent renewals. The holders of prospecting licences are required to submit annual work plans and budgets for approval by the GGMC. A performance bond of 10% of each annual work budget must be posted by the licensee each year.

- Mining licences require the payment of annual fees of US$3.00 per acre for Guyanese Companies and US$5.00 per acre. As most mining licences are granted subsequent to a prospecting licence they typically may be no larger than 12,800 acres.

- Permissions for geological and geophysical survey may be of any size and fees of any amount as negotiated between the Minister and the permit holder.

The payment of gross production royalties are provided for by the Act and the amount of royalty to be paid is prescribed by the Minister. As such royalties may be varied from time to time. Currently, the prescribed royalties on gold production are ad valoren of gross production sales at the following rates:

- For claim licences 5%;
- For mining permits for medium scale operations 5%;
- For mining licences 5%, however recent mineral agreements entered into stipulate a royalty of 8% if the gold price is above US$1,000 per ounce.

The Act makes broad provision for the GGMC, with approval of the Minister, to enter into mineral agreements with applicants for, or holders of, prospecting and mining licences with respect to:

- The conditions to be included in the licence as granted, applied for or renewed.
- The procedure to be followed by the GGMC while exercising any discretion conferred to it by the Act.
- Any matter incidental to or connected to a licence.

At present, it is usual for two mineral agreements to be entered into. One applicable to the early stages of prospecting operations and the other prior to the grant of a mining licence or commencement of mining operations.

Such mineral agreements provide incentives, define the fiscal regime and ensure the stability of the incentives and the fiscal regime for the entire term of a prospecting licence or for a set period (currently 15 years) for a mining licence. While it is possible for provisions of a mineral agreement to be negotiated it has been the practice of the GGMC and the Minister to provide the same terms and conditions at any one time for all holders of or applicants for prospecting and mining licences.

Currently the holders of prospecting licences are granted the following incentives under mineral agreements:

- Zero rating on all customs duties, VAT and excise taxes on the purchase and importation of all equipment and supplies to be used directly in prospecting operations with the exception of food and office equipment apart from computers.
- Exemption from all excise tax on fuel used in prospecting operations.
- 10% rating on consumption tax for aviation fuel.

Currently mineral agreements entered into prior to the grant of mining licences or the commencement of mining operations ensure the following incentives and fiscal regime:

- Zero rating on all customs duties, VAT and excise taxes on the purchase and importation of all equipment supplies, consumables, and spares to be used directly in mining operations.
- Exemption from all excise tax on fuel used in mining operations.
- 10% rating on consumption tax for aviation fuel.
- Consignment of the above incentives to contractors of the company who provide services directly in support of mining operations.
- Zero rating on VAT for payments to certain contractors such as drilling or mining contractors.
- Corporate tax rate of 30%.
- Exemption from withholding taxes on dividends and interest payments to the parent company or other offshore entities.
- Deductibility of royalty payments against corporate taxes.
- Offset of the cost of certain infrastructural works of general benefit to Guyana against the mineral royalty.
- Royalty of 5% if the gold price is below US$1,000 per ounce and 8% if the gold price is above US$1,000 per ounce.
- Capitalisation of prospecting costs.
- 20% straight line depreciation.
- Stability of royalties, incentives and tax rates for the term of the licence, the life of the deposit or for 15 years whichever is shorter.

4.4 Ownership

Azimuth, formerly Epsilon Energy Limited, announced positive completion of due diligence in relation to the acquisition of Takatu Minerals Limited (Takatu) on 3 November 2009 resulting in the acquisition by Epsilon of the projects controlled by Takatu; including, among others, the Karouni Gold Project (formerly known as the West Omai Gold Project) in central Guyana. In July 2013 Troy completed a takeover of Azimuth. Troy holds the Karouni Gold Project tenement rights through Pharsalus Gold Inc., an indirect, wholly owned subsdidiary of Troy (see Figure 4-2).

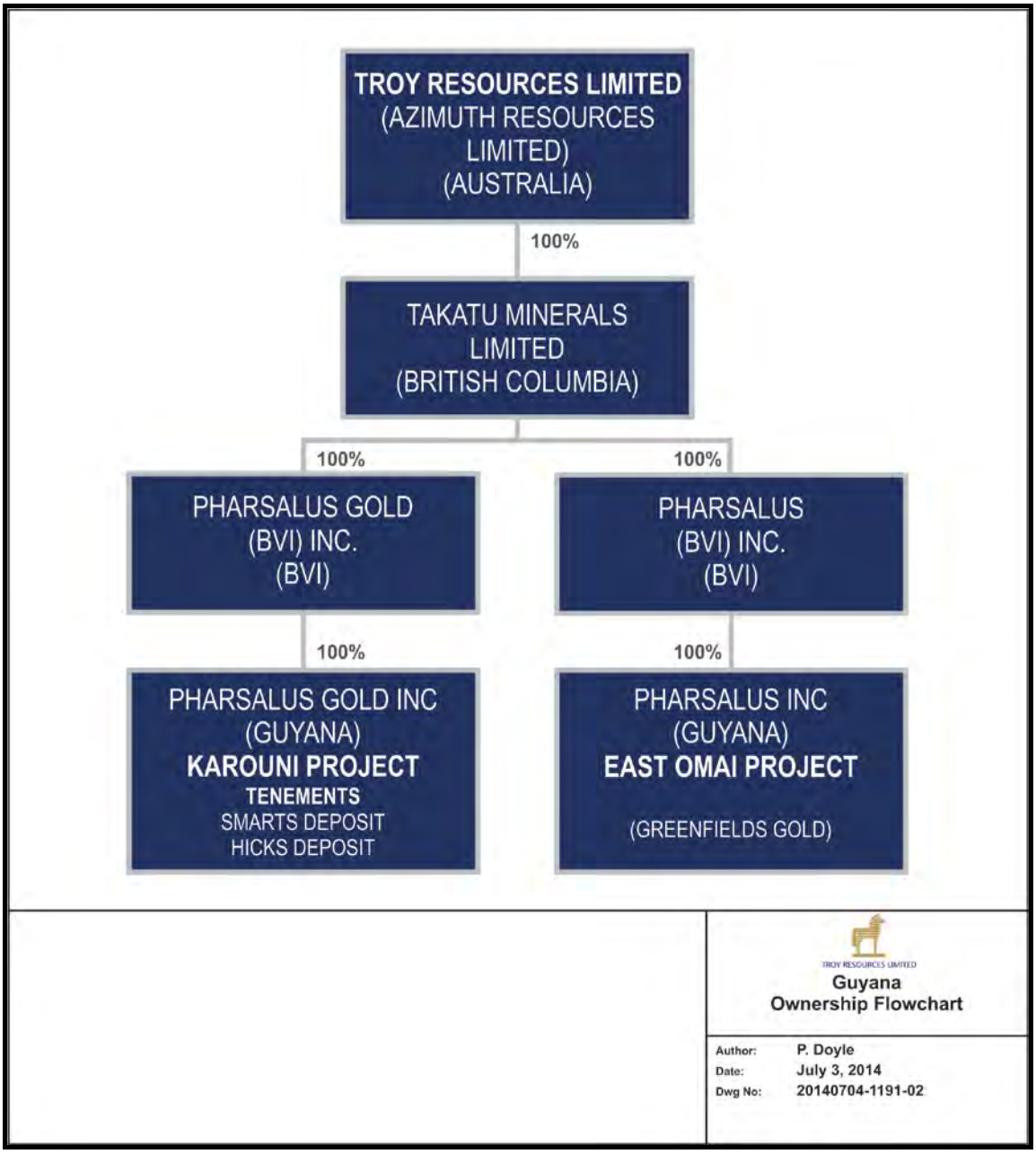

Figure 4-2: Corporate Structure for Troy's Guyana Assets held through Azimuth

The Karouni Gold Project tenements cover an aggregate area of 270,706 acres (113,581ha), granting the holders the right to explore for gold or gold and diamonds. The tenements have been acquired by either direct grant to (Pharsalus Gold) (25,990 acres /10,517ha) or by contractual agreements with tenement holders (254,668 acres 103,064ha). Apart from the Kaburi Agreement (29,143 acres 11,794ha), which provides for Pharsalus Gold to earn a 90% interest, all other vendor agreements provide Pharsalus Gold with the right to obtain an ultimate interest of 100%.

The Karouni Gold Project comprises a single (large scale) mining licence, 94 (small scale) claim licences, 220 (medium scale) prospecting and mining permits, and 6 (large scale) prospecting licences as shown in Table 4-1. All licences, permits and claims are granted for either gold or gold and diamonds. The (large scale) prospecting licences include three licences won by Pharsalus Gold at open auction on 22 November 2007 (GS14: P-18, P-19 and P-20) and which are owned 100% by the Company.

The tenements were granted and held in good standing at the time of this report.

Table 4-1: Karouni Tenement Schedule (Updated as of June 30, 2014)								
Tenement Holder	**Vendor Agreement**	**GGMC- File Number or claim HO number**	**Tenement Type**	**Area (Acres)**	**Grant Date**	**Expiry/Renewal / Re-grant Date**	**Troy's current interest**	**Maximum Interest to Be Earned**
Daniels, V.; Adams, J. & Adams, C.	Adams & Daniels	GS8: D-114/000/MP/2013	MSMP***	314	17-Mar-08	02-Jun-15	-*	100%
Daniels, V.; Adams, J. & Adams, C.	Adams & Daniels	GS8: D-114/001/MP/2013	MSMP***	897	17-Mar-08	02-Jun-15	-*	100%
Daniels, V.; Adams, J. & Adams, C.	Adams & Daniels	GS8: D-102/000/MP/2013	MSMP***	1,008	17-Mar-08	10-Mar-15	-*	100%
Daniels, V.; Adams, J. & Adams, C.	Adams & Daniels	GS8: D-102/001/MP/2013	MSMP***	1,126	17-Mar-08	10-Mar-15	-*	100%
Baksh, A.	Baksh 1	GS8:B-97/MP/001/2012	MSMP***	1,076	30-Nov-07	29-Nov-14	100%	100%
Baksh, A.	Baksh 1	GS8: B-294/001/2008	MSPP	845	06-Jan-09	05-Jan-15	100%	100%
Baksh, A.	Baksh 1	GS8: B-294/004/2008	MSPP	1,019	06-Jan-09	05-Jan-15	100%	100%
Baksh, A.	Baksh 1	GS8: B-380/000/2008	MSPP	762	05-Feb-09	04-Feb-15	100%	100%
Baksh, A.	Baksh 1	GS8:B-293/000/2007	MSPP	1,123	03-Sep-08	02-Sep-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/002/2008	MSPP	845	06-Jan-09	05-Jan-15	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/003/2008	MSPP	464	06-Jan-09	05-Jan-15	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/005/2007	MSPP	844	04-Sep-08	03-Sep-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/006/2007	MSPP	844	04-Sep-08	03-Sep-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/007/2007	MSPP	932	04-Sep-08	03-Sep-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-294/008/2007	MSPP	769	04-Sep-08	03-Sep-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-295/000/2007	MSPP	891	24-Jul-08	23-Jul-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-295/001/2007	MSPP	703	24-Jul-08	23-Jul-14	100%	100%
Baksh, A.	Baksh 1	GS8:B-296/003/2007	MSPP	1,052	30-Nov-07	29-Nov-14	100%	100%
Baksh, A.	Baksh 2	GS8:B-297/000/2008	MSPP	915	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/001/2008	MSPP	738	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/002/2008	MSPP	845	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/003/2008	MSPP	845	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/004/2008	MSPP	845	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/005/2008	MSPP	845	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-297/006/2008	MSPP	676	16-Oct-08	15-Oct-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-317/000/2008	MSPP	845	05-Nov-08	04-Nov-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-317/001/2008	MSPP	845	05-Nov-08	04-Nov-14	-*	100%
Baksh, A.	Baksh 2	GS8:B-382/000/2008	MSPP	1,175	06-Feb-09	05-Feb-15	-*	100%
Broomes, S.	Broomes 1	GS8:B-213/000/2005	MSPP	380	22-Jun-06	21-Jun-15	-*	100%
Broomes, S.	Broomes 1	GS8:B-213/001/2005	MSPP	1,129	22-Jun-06	21-Jun-15	-*	100%

Budhram, S.	Budhram 1	GS8: B235/000/06	MSPP	451	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/001/06	MSPP	1,031	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/002/06	MSPP	1,090	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/003/06	MSPP	970	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/004/06	MSPP	1,053	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/005/06	MSPP	1,059	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/006/06	MSPP	1,090	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/007/06	MSPP	1,073	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/008/06	MSPP	766	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/009/06	MSPP	1,090	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/010/06	MSPP	427	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 1	GS8: B235/011/06	MSPP	737	01-Feb-07	31-Jan-15	100%	100%
Budhram, S.	Budhram 2	GS8: B262/000/07	MSPP	1,075	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/001/07	MSPP	1,068	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/002/07	MSPP	974	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/003/07	MSPP	654	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/004/07	MSPP	979	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/005/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/006/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/007/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/008/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/009/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/010/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/011/07	MSPP	747	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/012/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/013/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/014/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/015/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/016/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/017/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/018/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/019/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/020/07	MSPP	1,156	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/022/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/023/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/024/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/025/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/026/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/027/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/028/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%

Budhram, S.	Budhram 2	GS8: B262/029/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/033/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/034/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/035/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/036/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/037/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/038/07	MSPP	1,200	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/040/07	MSPP	841	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/041/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/042/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/043/07	MSPP	1,090	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/044/07	MSPP	1,008	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 2	GS8: B262/045/07	MSPP	813	19-Jul-07	18-Jul-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/000/07	MSPP	1,043	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/001/07	MSPP	845	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/002/07	MSPP	845	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/003/07	MSPP	845	09-Oct-08	08-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/004/07	MSPP	726	09-Oct-08	08-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/005/07	MSPP	549	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/006/07	MSPP	691	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/008/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/009/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/010/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/011/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/012/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B300/013/07	MSPP	980	20-Aug-08	19-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B301/003/07	MSPP	891	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B301/009/07	MSPP	844	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B301/012/07	MSPP	772	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B301/018/07	MSPP	838	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B302/004/07	MSPP	1,200	06-Mar-09	05-Mar-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/000/07	MSPP	997	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/001/07	MSPP	779	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/002/07	MSPP	845	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/004/07	MSPP	844	08-Oct-08	07-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/005/07	MSPP	844	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/006/07	MSPP	845	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/009/07	MSPP	844	06-Oct-08	05-Oct-14	100%	100%

Budhram, S.	Budhram 3	GS8: B304/010/07	MSPP	845	19-Aug-08	18-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/011/07	MSPP	551	19-Aug-08	18-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/012/07	MSPP	481	19-Aug-08	18-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/013/07	MSPP	1,021	19-Aug-08	18-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/014/07	MSPP	694	19-Aug-08	18-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/018/07	MSPP	845	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/019/07	MSPP	676	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/022/07	MSPP	845	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/024/07	MSPP	676	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/025/07	MSPP	799	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/027/07	MSPP	535	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/028/07	MSPP	563	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/029/07	MSPP	1,091	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/030/07	MSPP	845	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/031/07	MSPP	676	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/032/07	MSPP	772	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/033/07	MSPP	845	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/034/07	MSPP	845	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/035/07	MSPP	745	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/036/07	MSPP	845	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/037/07	MSPP	201	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/038/07	MSPP	845	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B304/039/07	MSPP	676	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/001/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/004/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/007/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/009/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/010/08	MSPP	1,059	10-Oct-08	09-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/011/08	MSPP	1,200	10-Oct-08	09-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/013/08	MSPP	1,090	10-Oct-08	09-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/014/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/015/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/017/07	MSPP	1,090	21-Sep-09	20-Sep-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/018/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/019/07	MSPP	1,059	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/020/07	MSPP	1,090	13-Jul-09	12-Jul-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/021/07	MSPP	1,090	13-Jul-09	12-Jul-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/022/07	MSPP	1,090	13-Jul-09	12-Jul-14	100%	100%

Budhram, S.	Budhram 3	GS8: B305/023/07	MSPP	1,090	13-Jul-09	12-Jul-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/024/07	MSPP	1,090	13-Jul-09	12-Jul-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/025/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/026/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/027/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/028/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/029/08	MSPP	1,090	06-Nov-08	05-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/030/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/031/07	MSPP	1,090	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/032/07	MSPP	1,033	28-Aug-08	27-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/033/08	MSPP	831	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/034/08	MSPP	1,090	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/035/07	MSPP	1,090	22-Aug-08	21-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/036/07	MSPP	1,090	22-Aug-08	21-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/037/07	MSPP	1,090	11-Sep-09	10-Sep-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/038/07	MSPP	1,090	22-Aug-08	21-Aug-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/039/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/040/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/041/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/043/08	MSPP	701	07-Nov-08	06-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/044/08	MSPP	1,178	07-Nov-08	06-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/045/08	MSPP	1,090	07-Nov-08	06-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/046/08	MSPP	1,090	07-Nov-08	06-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/047/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/048/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/049/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/050/07	MSPP	1,090	06-Oct-08	05-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/055/08	MSPP	1,102	10-Nov-08	09-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/056/08	MSPP	1,134	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/057/08	MSPP	917	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/058/08	MSPP	1,102	10-Nov-08	09-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/059/08	MSPP	1,090	10-Nov-08	09-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/065/08	MSPP	455	10-Nov-08	09-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/066/08	MSPP	847	10-Nov-08	09-Nov-14	100%	100%
Budhram, S.	Budhram 3	GS8: B305/070/08	MSPP	847	15-Oct-08	14-Oct-14	100%	100%
Budhram, S.	Budhram 3	GS8: B306/000/07	MSPP	673	19-Sep-08	18-Sep-14	100%	100%
Campbell, R	Campbell	GS: C-83/000/2013	MSMP	224	06-Jun-13	06-Jun-15	100%	100%

Chase, B.(& Duncan Reynold)	Chase 1	31/1994/072	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Chase, B.(& Duncan Reynold)	Chase 1	31/1994/073	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Chase, B.(& Duncan Reynold)	Chase 1	31/1994/074	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Chase, B.(& Duncan Reynold)	Chase 1	31/1994/075	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Chase, B.(& Duncan Reynold)	Chase 1	31/1994/076	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Chase, B.	Chase 2	31/1994/071	SSCL	28	31-Mar-94	30-Mar-12	-*	100%
Chase, B.	Chase 2	31/1994/247	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Chase, B.	Chase 2	31/1994/248	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Chilcott, R.	Chilcott 1	31/1994/309	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Chilcott, R.	Chilcott 1	31/1994/310	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Chilcott, R.I	Chilcott 1	31/1994/311	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
David, N.	David 1	31/1993/088	SSPC	28	31-Mar-93	30-Mar-15	-*	100%
David, N.	David 1	31/1993/089	SSPC	28	31-Mar-93	30-Mar-15	-*	100%
David, N.	David 1	31/1993/090	SSPC	28	31-Mar-93	30-Mar-15	-*	100%
David, N.	David 1	31/1993/091	SSPC	28	31-Mar-93	30-Mar-15	-*	100%
David, N.	David 1	31/1993/092	SSPC	28	31-Mar-93	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	ML 02/2009 Formerly MGS 1/1998 Licence	LSML	354	21-May-09	20-May-16	-*	100%
George Hicks Mining Company Limited	Hicks	GS8: H57/MP/000/2012	MSMP***	910	31-May-13	31-May-15	-*	100%
George Hicks Mining Company Limited	Hicks	GS8: H57/MP/001/2012	MSMP***	1,017	31-May-13	31-May-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1988/020	SSCL	28	31-Mar-88	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/019	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/020	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/021	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/022	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/023	SSCL	28	31-Mar-81	30-Mar-15	-*	100%

Kaburi Development Company Limited	Hicks	31/1981/024	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/025	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/026	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/027	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/028	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/029	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/030	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/031	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/032	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/033	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/034	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/035	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/036	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/037	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/038	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/039	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/040	SSCL	28	31-Mar-81	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1981/041	SSCL	28	31-Mar-81	30-Mar-15	-*	100%

George Hicks Mining Company Limited	Hicks	31/1986/002	SSCL	28	31-Mar-86	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1986/003	SSCL	28	31-Mar-86	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1986/004	SSCL	28	31-Mar-86	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1986/039	SSCL	28	31-Mar-86	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1986/040	SSCL	28	31-Mar-86	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1987/035	SSCL	28	31-Mar-87	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1987/037	SSCL	28	31-Mar-87	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1987/038	SSCL	28	31-Mar-87	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1987/039	SSCL	28	31-Mar-87	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1987/040	SSCL	28	31-Mar-87	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1988/017	SSCL	28	31-Mar-88	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1988/018	SSCL	28	31-Mar-88	30-Mar-15	-*	100%
George Hicks Mining Company Limited	Hicks	31/1988/019	SSCL	28	31-Mar-88	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1991/083	SSCL	28	31-Mar-91	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1991/084	SSCL	28	31-Mar-91	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1991/085	SSCL	28	31-Mar-91	30-Mar-15	-*	100%
Kaburi Development Company Limited	Hicks	31/1991/086	SSCL	28	31-Mar-91	30-Mar-15	-*	100%
Campbell, R.	Jones & Newton	GS8: C-229/000/2005	MSPP	957	21-Jun-07	20-Jun-15	-*	100%
West Kaburi Mining Company Inc	Kaburi	GS14: W-10 (PL# 34/2009)	LSPP	4,391	12-Aug-09	11-Aug-14	-	90%

West Kaburi Mining Company Inc	Kaburi	GS8: H34-MP/2012	MSMP	1,040	22-Aug-00	21-Aug-14	-	90%
Harding, P, & Belle, C.	Kaburi	GS8: B-21/000/08 (old file number GS8:B-21/000/1994)	MSPP	820	07-May-09	06-May-15	-	90%
Harding, P, & Belle, C.	Kaburi	GS8: B-21/001/08 (Old no. B-21/001/1994)	MSPP	925	06-May-09	05-May-15	-	90%
Harding, P & Daniels, V.	Kaburi	GS8: D-32/000/94	MSPP	400	26-Nov-94	25-Nov-14	-	90%
Harding, P & Daniels, V.	Kaburi	GS8: D-32/001/94	MSPP	1,045	18-Nov-94	17-Nov-14	-	90%
Harding, P	Kaburi	GS8: H-1/000/05	MSPP	894	16-Mar-06	15-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-191/000/06	MSPP	701	26-Jan-07	25-Jan-15	-	90%
Harding, P	Kaburi	GS8: H-194/000/06	MSPP	1,023	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/001/06	MSPP	1,090	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/002/06	MSPP	1,090	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/003/06	MSPP	1,090	21-Mar-07	20-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/004/06	MSPP	826	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/005/06	MSPP	965	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/006/06	MSPP	1,089	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/007/06	MSPP	548	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/008/06	MSPP	1,090	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/009/06	MSPP	1,072	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/010/06	MSPP	793	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/011/06	MSPP	1,144	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/012/06	MSPP	320	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/013/06	MSPP	982	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-194/014/06	MSPP	724	22-Mar-07	21-Mar-15	-	90%
Harding, P	Kaburi	GS8: H-52/000/95	MSPP	807	19-Nov-04	18-Nov-14	-	90%
Harding, P	Kaburi	GS8: H-61/000/95	MSPP	1,200	27-Nov-04	26-Nov-14	-	90%
Harding, P	Kaburi	GS8: H-61/001/95	MSPP	846	27-Nov-04	26-Nov-14	-	90%
Harding, P	Kaburi	GS8: H-61/002/95	MSPP	901	26-Nov-04	25-Nov-14	-	90%
Harding, P	Kaburi	GS8: H99/000/0300/2000	MSPP	1,075	22-Aug-00	21-Aug-14	-	90%
Harding, P.	Kaburi	33/2005/143	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/144	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/145	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/146	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/147	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/148	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/149	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/150	SSCL	28	31-Mar-05	30-Mar-15	-	90%
Harding, P	Kaburi	33/2005/151	SSCL	28	31-Mar-05	30-Mar-15	-	90%

Lewis, O.	Lewis 1	L-11/MP/000/10	MSMP	1,090	28-Jun-10	27-Jun-15	100%	100%
Lewis, O.	Lewis 1	L-11/MP/001/10	MSMP	539	07-Jul-10	06-Jul-14	100%	100%
Lewis, O.	Lewis 2	31/2002/001	SSCL	28	31-Mar-02	30-Mar-15	-*	100%
Lewis, O.	Lewis 2	31/2002/002	SSCL	28	31-Mar-02	30-Mar-15	-*	100%
Lewis, O.	Lewis 2	31/2002/003	SSCL	28	31-Mar-02	30-Mar-15	-*	100%
Lewis, O.	Lewis 2	31/2002/004	SSCL	28	31-Mar-02	30-Mar-15	-*	100%
Lewis, O.	Lewis 2	31/2002/005	SSCL	28	31-Mar-02	30-Mar-15	-*	100%
Lewis, O.; Chase, B &.; Trotz, M.	Lewis-Chase-Trotz	GS14: L-6	LSPL	13,276	31-May-11	30-May-15	-*	100%
Pharsalus Gold Inc.	n/a	GS14: P-18 (PL14/2008) Licence	LSPL	4,958	31-Jul-08	30-Jul-14	100%	100%
Pharsalus Gold Inc.	n/a	GS14: P-19 (PL15/2008) Licence	LSPL	8,532	31-Jul-08	30-Jul-14	100%	100%
Pharsalus Gold Inc.	n/a	GS14: P-20 (PL16/2008) Licence	LSPL	12,500	31-Jul-08	30-Jul-14	100%	100%
Singh, D.	Singh	GS8:S-328/002/2008	MSPP	370	09-Oct-08	08-Oct-14	-*	100%
Singh, D.	Singh	GS8:S-328/005/2008	MSPP	275	04-Sep-08	03-Sep-14	-*	100%
Singh, D.	Singh	GS8:S-328/013/2007	MSPP	1,090	04-Sep-08	03-Sep-14	-*	100%
Singh, D.	Singh	GS8:S-328/016/2007	MSPP	1,090	04-Sep-08	03-Sep-14	-*	100%
Singh, D.	Singh	GS8:S-330/000/2007	MSPP	1,018	09-Oct-08	08-Oct-14	-*	100%
Smartt, L.	Smarts 1	31/1994/031	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 1	31/1994/037	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 1	31/1994/040	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 1	31/1994/133	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 1	33/1999/082	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Smartt, L.	Smarts 1	33/1999/083	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1984/042	SSCL	28	31-Mar-84	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1984/043	SSCL	28	31-Mar-84	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1984/044	SSCL	28	31-Mar-84	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1984/045	SSCL	28	31-Mar-84	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1984/046	SSCL	28	31-Mar-84	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1994/035	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/1994/036	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/005	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/006	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/007	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/008	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/009	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/010	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/011	SSCL	28	31-Mar-03	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	31/2003/012	SSCL	28	31-Mar-03	30-Mar-15	-*	100%

Smartt, L.	Smarts 2	33/1994/038	SSCL	28	31-Mar-94	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	33/1999/064	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	33/1999/079	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	33/1999/080	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Smartt, L.	Smarts 2	33/1999/081	SSCL	28	31-Mar-99	30-Mar-15	-*	100%
Trotz, M.; (Chilcott, R.)	Trotz #1	31/1994/312	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Trotz, M.; (Chilcott, R.)	Trotz #1	31/1994/308	SSCL	28	31-Mar-94	30-Mar-15	100%	100%
Broomes, S.	Broomes 1	GS8:B-215/000/2005	MSPP	533	22-Jun-06	21-Jun-15	-*	100%
Pereira Mining Company Limited	Pereira 1	ML 01/2012	LSML	5,105	01-Feb-12	02-Jul-19	-*	100%
Golden Eruption Resources (Guyana) Inc.	Pereira 1	PL 40/2011	LSPL	4,103	23-Dec-11	22-Dec-14	-*	100%
Chin, C.	Pereira 1	C-235/000/06 (PPMS/426/06)	LSPP	1,043	03-Jan-07	25-Jan-15	-*	100%
Pereira, J. & Pereira, R.	Pereira 1	P-16/MP/000/10	MSMP	1,086	28-Jun-10	27-Jun-15	-*	100%
Pereira, R.	Pereira 1	GS8:P-196/000/06 (PPMS/651/06)	MSPP	595	28-Mar-07	27-Mar-15	-*	100%
Pereira, R.	Pereira 1	GS8:P-196/001/06 (PPMS/652/06)	MSPP	861	28-Mar-07	27-Mar-15	-*	100%
Pereira, R.	Pereira 1	GS8:P-196/002/06 (PPMS/653/06)	MSPP	1,191	28-Mar-07	27-Mar-15	-*	100%
Pereira, R.	Pereira 1	GS8:P-190/000/06 (PPMS/382/06)	MSPP	963	14-Jul-06	13-Jul-14	-*	100%
Pereira, R.	Pereira 1	GS8:P-190/000/06 (PPMS/383/06)	MSPP	999	14-Jul-06	13-Jul-14	-*	100%
Gonsalves, N.	Gonsalves 1	G61:MP/000/2013	MSMP	555	30-Jan-13	30-Jan-15	100%	100%
Pereira, P.	Pereira 1	31/1973/590	SSPC	28	01-Feb-73	31-Mar-15	-*	100%
Pereira, P.	Pereira 1	31/1979/05	SSPC	28	08-Jan-79	31-Mar-15	-*	100%
Sookram, D.	Sookram 2	GS8:260/002/2007	MSPP	1,114	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/003/2007	MSPP	1.2	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/004/2007	MSPP	1,200	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/005/2007	MSPP	1,122	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/006/2007	MSPP	1,101	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/007/2007	MSPP	1,170	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/008/2007	MSPP	1,199	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/009/2007	MSPP	1,181	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/010/2007	MSPP	1,066	23-Jul-07	22-Jul-14	-*	100%
Sookram, D.	Sookram 2	GS8:260/011/2007	MSPP	1.166	23-Jul-07	22-Jul-14	-*	100%

Sookram, D.	Sookram 2	GS8:268/001/2007	MSPP	1,200	05-Sep-07	04-Sep-07	-*	100%
Sookram, D.	Sookram 2	GS8:268/002/2007	MSPP	1,200	05-Sep-07	04-Sep-07	-*	100%

Notes

*** Pharsalus Gold has the right to explore and acquire the tenements under various option agreements. The agreements require the future payment of cash and/or shares, and/or exploration expenditure commitments. Pharsalus Gold has not to date satisfied all the payment obligations under the respective agreements, or satisfied the minimum exploration expenditure commitments.**

**** The "Shani" small claim is subject to a title dispute between the registered tenement holder (Brian Chase) and the GGMC. Due to its location, prospectivity, and size, it is not considered material to the Karouni Gold Project.**
***** Medium Scale Prospecting Permits which have been converted to mining permits in 2013.**
LSML – Large Scale Mining Licence
LSPL – Large Scale Prospecting Licence
LSPP – Large Scale Prospecting Permits
MSMP – Medium Scale Mining Permit
MSPP – Medium Scale Prospecting Permit
SSCL – Small Scale Claim Licence
SSPC – Small Scale Prospecting Claim

4.5 Property Survey

The prospecting permits registered in the name of Shreemattie Budhram are presently defined by UTM co-ordinates (zones 20N and 21N 1956 Provisional South American Spheroid) and are partly surveyed and marked by notices or cut-lines where the permits adjoin and abut mineral titles not owned by Pharsalus Gold or abut the Bartica Issano Road. Where the permits have boundaries with other mineral title owned by Pharsalus Gold permit boundaries have not been surveyed and are not marked by notices or cut lines.

The survey status of the Hicks group of tenements, registered in the name of George Hicks Mining Company Limited and Kaburi Development Company Limited is as follows:

- The two medium scale prospecting permits (GS8H-44/000/98 & GS8H-44/001/98) are surveyed by GPS and are marked by cut-lines and corner boards. The single large scale mining licence number 2 of 2009, originally granted as MD9/39/16 No 641, has been surveyed via GPS, and is marked by a notice and defined by cut-lines.
- The 41 small scale claim licences have all been surveyed by GPS and are defined by cut-lines and corner boards.

All tenements associated with the Kaburi group, registered granted in the name of P. Harding, C. Belle & P. Harding, V. Daniels & P. Harding and the West Kaburi Mining Corporation Inc., have been surveyed, are defined by cut-lines and corner boards are established.

The three prospecting permits won at the GGMC auction held on 22 November 2007 are granted to Pharsalus Gold. These tenements are presently defined by UTM coordinates (zones 20N and 21N 1956 Provisional South American Spheroid) and have not been surveyed and demarcated on the ground. All small scale claims of the other groups are defined by unsurveyed cut lines, and marked by GPS surveyed corner boards.

4.6 Agreements and Encumbrances

The private agreements pursuant to which certain of the tenements comprising the Karouni Gold Project are described below:

Smart No 1. Agreement: Comprising 6 small scale claim licences (Pam 4, 10, 12, 16-18) whereby Pharsalus Gold has been granted an exclusive and irrevocable licence from the owner, Leslie Smart, to explore and mine in exchange for payments totaling US$90,000.

Smart No 2 Agreement: Comprises the option to acquire a 100% interest in 20 small scale claim licences (OK3, OK5-8, Pam 5, 6, 8, 9 &11 Bob, Bob1-2, May, May1-6) from the vendor Leslie Smart. The agreement was signed on July 26th, 2010 and entails the following option payments:-

- A payment of US$15,000 upon the signing of the agreement.
- Payments of US$15,000 on the 1st and 2nd anniversaries of the agreement.
- A final payment of $155,000 on the 3rd anniversary of the agreement.

The vendor has now received US$200,000 in payments satisfying all scheduled payments under the agreement to exercise the option.

Hicks Agreement: Comprises 41 small scale claim licences (Glory 1-7, Star 1-7, Midnight 1-6, Yan 1-12, Madrasi, Nugget, Produce, Victory & Simon), two medium scale prospecting permits (GS8H44/000/1998-001/1998) and one large scale mining licence ML number 2 of 2009, originally granted as MD9/39/16 No 641. The agreement is between Pharsalus Gold on one hand and George Hicks Mining Company & Kaburi Development Company on the other. The agreement covers the Hicks Prospect, was signed on 11 June 2007 and entails:-

- Cash payments totalling US$155,000 at and before the listing of Pharsalus Gold or Takatu, and shares to the value of US$150,000 in Takatu at listing.
- On-going annual cash payments of US$50,000 on each anniversary of the listing of Takatu, ceasing upon commencement of commercial production.
- A retained "net smelter return" (NSR) royalty of 2% on all commercial production. Royalties on the initial 200,000oz of gold produced from the tenements is to be paid to George Hicks Mining Company & Kaburi Development Company, after which the royalty is to be split 50:50 with Imperial Metals Corporation of Canada (a successor company to Cathedral Gold).
- The NSR royalty may be purchased from the vendors once 750,000oz of Mineral Reserves have been proven. The purchase price is US$5,000,000 at a spot gold price of US$500/oz, increasing at US$1,000,000 for each US$100/oz increment in the gold price.

Kaburi Agreement: Comprising nine small scale claim licences (Pearl and Pearl 1-8), 27 medium scale prospecting permits (GS8:B-21/000/08,B-21/001/08, GS8:D32/001/9,GS8:D32/002/94, GS8:H61/000/2000 to H61/002/2000, GS8:H99/000/0300/2000, GS8:H99/001/0301/2000, ,GS8:H52/000/95, GS8:H1/000/05, H194/000/06 toH191/014/06), and one large scale Prospecting Permit (W10). The tenements are variously owned by P. Harding, V. Daniels & P. Harding, C. Belle & P. Harding and the West Kaburi Mining Company Limited and Perial Limited. Under the terms of the agreement, signed and dated September 21, 2007, the Company (Pharsalus Gold) has the right and option to earn up to a 90% interest in the Kaburi tenements in several stages. Once a decision to mine is made, the vendors retain a free-carried net profits interest equal to the interest not yet earned by Pharsalus Gold. Pharsalus Gold may still increase its interest to 90% in accordance with the terms below following a decision to mine. The summary terms of the agreement are as follows:-

- Pharsalus Gold must pay the vendors US$550,000 in several staged payments, of which $450,000 has been paid as at the date of this report.
- Pharsalus Gold must commit to completing expenditure of US$2,500,000 before May 8, 2014. At 30 June 2014, Pharsalus Gold had spent US$2,191,000 in total on the tenements. Upon satisfying the vendor payments and expenditure commitment above, Pharsalus Gold will have earned a 50% interest in the Kaburi tenements.
- Thereafter Pharsalus Gold may earn an additional 10% interest in the tenements via further expenditure of US$2,400,000 and may elect to transfer title to Pharsalus Gold.
- Thereafter Pharsalus Gold may increase its interest in the tenements to 90%, in increments of 10% for each US$1,700,000 in completed expenditure.

Budhram No.1 Agreement: Comprising 12 medium scale prospecting permits (GS8B235/000/2006-011/2006). Under this agreement, the Company (Pharsalus Gold) is granted an exclusive, irrevocable licence by the owner, S. Budhram, to explore and mine in exchange for a one-off payment of US$2,500 and re-imbursement of all costs and fees associated with the grant of the permits. The agreement was entered into September 5, 2006.

Budhram No.2 Agreement: Comprising 41 medium scale prospecting permits, whereby the Company (Pharsalus Gold) has been granted an exclusive and irrevocable licence from the owner, S. Budhram, to explore and mine in exchange for a one-off payment of US$10,000 and re-imbursement of all costs and fees associated with the grant of the permits. The agreement was entered into August 31, 2007.

Budhram No.3 Agreement: Comprising 99 medium scale prospecting permits (as tabulated in Table 1-1), whereby Pharsalus has been granted an exclusive and irrevocable licence from the owner, Shreemattie Budhram, to explore and mine in exchange for a one-off payment of US$25,000 and re-imbursement of all costs and fees associated with the grant of the permits. The agreement was entered into May 8, 2008.

Baksh No 1 Agreement: Comprising the option to acquire a 100% interest in 14 medium scale prospecting permits (B-293/000/2007, B-294/001 – 003/2008, B-294/004 – 008/2007, B-295/000 – 004/2007, B-380/000/2008). The agreement was signed on June 23, 2010 and entails:-

- Payment of US$50,000 on the signing of the agreement.
- Payment of US$100,000 on the 1st anniversary.
- Payment of US$50,000 on the 2nd anniversary.
- A final payment of US$50,000 on the 3rd anniversary.

During the period of the option Pharsalus Gold had full rights to occupy, explore, prospect, mine or carry out related activities on the tenements. Pharsalus Gold has elected to exercise the option early via accelerated payment of the option price. The vendor, A. Baksh, has received payment in full and Pharsalus Gold is the holder of all of the rights and interest in these 14 medium scale prospecting permits. The vendor is entitled to a 2% NSR royalty, which Pharsalus Gold has the option to purchase at any time for a cash payment of US$1,000,000.

Baksh No 2 Agreement: Comprising the option to acquire a 100% interest in 57 medium scale prospecting and mining permits of which 10 prospecting permits (B-297/000-006/2008, B-317/000 – 002/2008 & B-382/000/2008) fall within the Karouni Gold Project. The agreement was entered into November 26, 2011 and entails:-

- Payment of US$500 on the signing of the agreement.
- Issue of 3,000,000 shares in Azimuth within 3 months of the signing of the agreement.
- Issue of 2,000,000 shares in Azimuth 2 years after the date of the agreement.
- Issue of 4,000,000 shares in Azimuth 4 years after the date of the agreement.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements. There is no subsisting royalty during or after the exercise of the option.

Lewis No 1 Agreement: Comprises the option to acquire a 100% interest in 2 medium scale mining permits (L-11/MP/000 & L-11/MP0001) from the vendor O. Lewis. The agreement was signed on April 8, 2011 and entails:-

- Payment of US$10,000 upon the signing of the agreement.
- Payments of US$20,000 on the 1st and 2nd anniversaries.

Pharsalus Gold has elected to exercise the option early via accelerated payment of the option price. The vendor has received payment in full and Pharsalus Gold is the holder of all of the rights and interest in these 2 medium scale prospecting permits. The vendor retains a 2% NSR royalty, which Pharsalus Gold may acquire for a payment of US$500,000.

Lewis No 2 Agreement: Comprises the option to acquire a 100% interest in 5 small scale claim licences (Peturla 2-6) from the vendor O. Lewis. This agreement was signed on July 6th, 2011 and entails:-

- Payment of US$25,000 upon the signing of the agreement.
- Payment of US$25,000 upon the first anniversary of the signing of the agreement.
- A final payment of US$125,000 upon the second anniversary of the signing of the agreement.

The vendor has now received US$50,000 in payments satisfying the first scheduled payment under the agreement and further payments of US$125,000 are required to exercise the option.

Jones and Newton Agreement: Comprising the option to acquire a 100% interest in 1 medium scale prospecting permit (GS8: C-229/000/05). The permit was granted in the name of R. Campbell on June 21, 2007. Full rights and ownership were transferred to S. Jones and C. Newton by Agreement dated 8th July, 2009. The agreement between S. Jones and C. Newton on the one hand and the Company (Pharsalus Gold) on the other was signed 6th July, 2011 and entails:-

- Payment of US$30,000 on the signing of the agreement.
- A final payment of US$30,000 on the 1st anniversary.

The signing of the agreement provides Pharsalus Gold with full rights to occupy, explore, prospect, mine or carry out related activities on the mining property. Upon exercise of the option the vendors are entitled to a 2% NSR royalty which Pharsalus Gold has the right to purchase at any time for a cash payment of US$500,000.

Chilcott Agreement: Comprises the option to acquire a 100% interest in 3 small scale claim licences (Troy-2, Debra-1, and Debra-2) from the vendor R. Chilcott. This agreement was signed on July 25th, 2011 and entails:-

- Payment of US$15,000 on the signing of the agreement.
- Payment of US$15,000 on the 1st anniversary.
- A final payment of US$75,000on the 2nd anniversary.

The vendor has now received US$105,000 in payments satisfying requirements to exercise the option.

Chase No 1 Agreement: Comprises the 5 small scale claim licences (Hakeem, & Gold Star 1-4) whereby Pharsalus Gold has been granted an exclusive and irrevocable licence from the owner, B. Chase, to explore and mine in exchange for payments totalling US$100,000. The agreement was executed on July 25, 2011.

Chase No 2 Agreement: Comprises the option to acquire a 100% interest in 3 small scale claim licences (Goldstar #5, Goldstar #6 and Shani #1) from the vendor B. Chase. This agreement was signed on July 6th, 2011 and entails:-

- Payment of US$15,000 on the signing of the agreement.
- Payment of US$15,000 on the first anniversary.
- A final payment of US$75,000 on the second anniversary.

The vendor has now received US$30,000 in payments satisfying the first scheduled payment under the agreement and further payments of US$75,000 are required to exercise the option. During the period of the option the vendors may operate up to 4 land dredges for the purpose of mining alluvial deposits.

Trotz Agreement: Comprises 2 small scale claim licences (Aubrey1 and Troy1), whereby Pharsalus Gold has been granted an exclusive and irrevocable licence from the owner, M. Trotz, to explore and mine in exchange for payments totalling US$40,000. The agreement was executed 1st March 2011.

Lewis, Chase, Trotz Agreement: Comprises the option to acquire a 100% interest in a single large scale prospecting licence (GS14: L-6) from the vendors O. Lewis, B. Chase and M. Trotz. The agreement was signed 8th April 2011 and entails:-

- A payment of US$25,000 on signing.
- A payment of US$25,000 on first anniversary.
- A payment of US$50,000 on second anniversary.
- A payment of US$100,000 on third anniversary.

The vendors have now received US$50,000 in payments satisfying the first scheduled payment under the agreement and further payments of US$150,000 are required to exercise the option. Upon exercise of the option the vendors will retain a 2% NSR royalty which may be purchased at any time for the sum of US$1,000,000.

Singh No 1 Agreement: Comprising the option to acquire a 100% interest in 5 medium scale prospecting permits. The agreement was signed on October 28th, 2011 and entails:-

- Payment of US$50,000 on the signing of the agreement.
- Payment of US$50,000 on the 1st anniversary.
- Payment of US$50,000 on the 2nd anniversary.
- A final payment of US$50,000 on the 3rd anniversary.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements. After the exercise of the option the vendor D. Singh is entitled to a 2% NSR royalty, which Pharsalus Gold has the option to purchase at any time for a cash payment of US$1,000,000.

Adams and Daniels Agreement: Comprising the option to acquire a 100% interest in 4 medium scale prospecting permits. The agreement was signed on March 1st, 2011 and entails:-

- Payment of US$50,000 on the signing of the agreement.
- Payment of US$50,000 on the 1st anniversary.
- Payment of US$50,000 on the 2nd anniversary.
- A final payment of US$50,000 on the 3rd anniversary.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements. After the exercise of the option the vendors, Jason Adams, Victor Daniels and Carl Adams are entitled to a 2% NSR royalty, which Pharsalus Gold has the option to purchase at any time for a cash payment of US$1,000,000.

Nelson David Agreement: Comprising the option to acquire a 100% interest in 5 small scale prospecting permits. The agreement was signed on May 15, 2012 and entails:-

- Payment of US$50,000 on the signing of the agreement.
- Payment of US$50,000 conditional upon the issuing of licences for the claims and the subsequent transfer of the licences to Pharsalus Gold.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements.

Simona Broomes Agreement: Comprising the option to acquire a 100% interest in 3 medium scale prospecting permits. The agreement was signed on June 20, 2012 and entails:-

- Payment of US$40,000 on the signing of the agreement.
- Payment of US$40,000 on the 1st anniversary.
- A final payment of US$40,000 on the 2nd anniversary.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements. After the exercise of the option the vendor, Simona Broomes is entitled to a 2% NSR royalty, which Pharsalus Gold has the option to purchase at any time for a cash payment of US$1,500,000.

Ryan Pereira Agreement: Comprising the option to acquire a 100% interest in 1 mining licence, 2 large-scale prospecting licences, 1 mining permit, 5 prospecting permits, and 2 small scale prospecting permits. The agreement was signed on September 5, 2012 and entails:-

- Payment of US$150,000 and the issuance of 500,000 (Azimuth) ordinary shares on the signing of the agreement.
- Payment of US$200,000 and the issuance of 500,000 (Azimuth) ordinary shares on the 1st anniversary.
- Payment of US$200,000 and the issuance of 500,000 (Azimuth) ordinary shares on the 2nd anniversary.
- A final payment of US$1,000,000 and the issuance of 1,500,000 (Azimuth) ordinary shares on the 3rd anniversary.

During the period of the option Pharsalus Gold has full rights to occupy, explore, prospect, mine or carry out related activities on the tenements, subject only to the vendors exclusive right to process mine tailings and to mine alluvial material situated within Mining Licence ML 01/2012 “Honeycamp” within a 4-year period from the signing of the agreement. Any gold which may be recovered from existing tailings by the company following the expiry of the 4-year exclusion period is subject to a 25% NSR payable to Ryan Pereira. After the exercise of the option the vendor, Ryan Pereira is entitled to a 2% NSR royalty, which Pharsalus Gold has the option to purchase at any time for a cash payment of US$10,000,000.

Norma Harris Agreement: Comprises a 12-month agreement for initial exploration work only, not including drilling, for which Pharsalus Gold has paid the licensee US$10,000, Before the expiry of the 12 months, Pharsalus Gold has the option but not the obligation upon payment of a further US$15,000 to extend the agreement for a further 5 years.

Imperial Metals has maintained an interest in the Hicks Zone to the present day and, under its agreement with Pharsalus Gold, retains a 1% net smelter return (NSR) royalty in the Project, applicable after the initial 200,000oz of gold production.

There are no other royalties, back-in rights, payments or other agreements or encumbrances to which the Project is subject.

4.7 Environmental Liabilities

The Environmental Protection Act of 1996 (Guyana) requires all mining operations to be subject to Environmental Impact Assessments (EIA) including provisions for public participation. Additional regulations

require holders of prospecting and mining permits to lodge 'environmental bonds' with the Guyana Geology and Mines Commission (GGMC) to cover the costs of restoring the environment in case of malpractice. Bonds are repayable if no damage is found. Social impact assessments are also required as part of the EIA. Existing environmental bonds held by GGMC to the credit of Pharsalus Gold in respect of mining licences, prospecting licences (large-scale), and mining permits under the control of Pharsalus Gold are detailed in Table 4-2 below:

Table 4-2: Existing Environmental Bonds - Karouni Block

ORIGINAL TENEMENT HOLDER	GGMC FILE NUMBER	TENEMENT TYPE	ENVIRONMENTAL BOND AMOUNT (USD)
Daniels, Victor	GS8: D-102/000/MP/2013	Medium Scale Mining Permit	500
Daniels, Victor	GS8: D-102/001/MP/2013	Medium Scale Mining Permit	500
Daniels, Victor	GS8: D-114/000/MP/2013	Medium Scale Mining Permit	500
Daniels, Victor	GS8: D-114/001/MP/2013	Medium Scale Mining Permit	500
Baksh, Azeem	GS8: B-97/MP/001/2012	Medium Scale Mining Permit	500
George Hicks Mining Company Limited	GS8: H-57/MP/000/2012	Medium Scale Mining Permit	500
George Hicks Mining Company Limited	GS8: H-57/MP/000/2012	Medium Scale Mining Permit	500
George Hicks Mining Company Limited	ML-02/2009	Large Scale Mining Licence	10,000
Lewis, Orin	GS8: L-11/MP/000/2010	Medium Scale Mining Permit	500
Lewis, Orin	GS8: L-11/MP/001/2010	Medium Scale Mining Permit	500
Lewis, Orin	GS14: L-6	Large Scale Prospecting Licence	10,120
Pharsalus Gold, Inc.	GS14: P-18	Large Scale Prospecting Licence	5,200
Pharsalus Gold, Inc.	GS14: P-19	Large Scale Prospecting Licence	5,200
Pharsalus Gold, Inc.	GS14: P-20	Large Scale Prospecting Licence	5,200
Campbell, Roxanne	GS8: C-83/MP/000/2013	Medium Scale Mining Permit	500
Gonsalves, Neil	GS8: G-61/MP/000/2013	Medium Scale Mining Permit	500

Notes:

- **Bonds for applicable West Kaburi (Harding) & Pereira tenements were lodged by, and are still registered in the name of the original tenement holder, and are not held by GGMC to the credit of Pharsalus Gold.**

Upon granting of licences/permits in each of the above tenement categories, the GGMC enters into a formal standard environmental management agreement ("EMA") with the tenement holder. At this time, the tenement holder must lodge an environmental bond in cash with the GGMC. The EMA sets out specific protocols for licence/permit holders in relation to activities such as exploration disturbances, mining excavation, limited deforestation, removal and stockpiling of topsoil, sediment loading in adjacent waterways, and the control of hazardous wastes. Once the area disturbed is restored / rehabilitated to the standard required and approved by GGMC, the environmental bond will normally be refunded.

Troy is aware of a potential environmental issue related to alluvial mining surface distrubance caused by local illegal miners on the P20 tenement. Troy is working with government authorities to resolve the issue. The matter is currently under review by the GGMC and discussions with governmental officals are ongoing. Troy is not aware, nor has been made aware, of any other environmental liabilities associated with the Karouni Gold Project. The title documents include no specific environmental requirements beyond those required under the Guyana Mining Act and associated regulations.

The site visit to Karouni Gold Project identified significant environmental disturbance associated with areas of historic and current formal and informal small scale mining, particularly at the Hicks Deposit, Eldorado and Whitehall Prospects. It is understood that the various agreements between Pharsalus Gold and the vendors incorporate a clause leaving existing environmental liabilities with the vendors. In all other cases, environmental liability under the Mining and Environmental Protection Acts will lie with Troy.

Troy is progressing through the established, prescribed permitting process to obtain all of the required environmental approvals to develop the Project.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation, Vegetation and Land Use

The Karouni Gold Project is dominated by gently undulating terrain at elevations ranging from 50m to 100m above sea level. The area is drained by a dendritic network of moderately incised small streams and creeks. The Project area is veneered entirely by tropical lowland forest (see Figure 5-1) with the canopy height ranging from approximately 20m to 50m. Aside from logging tracks, the forest cover has essentially been preserved by very selective logging practices.

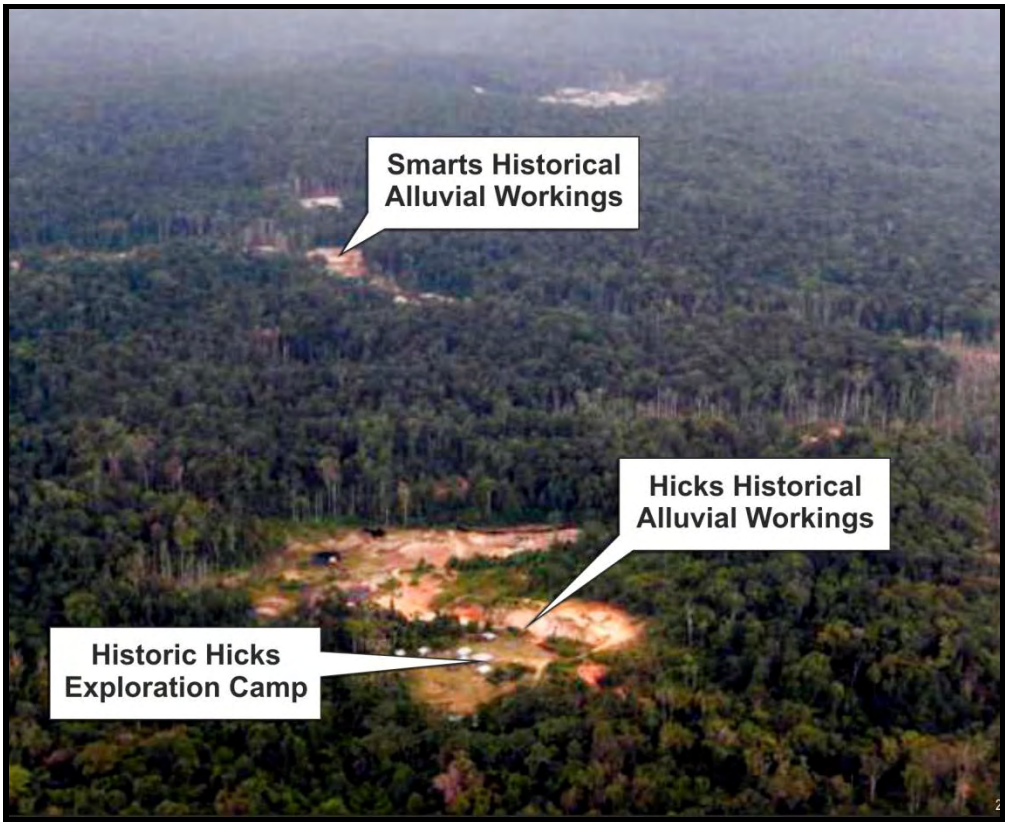

Figure 5-1: Karouni Gold Project Looking Northwest

5.2 Climate

The Karouni Gold Project experiences a tropical monsoonal climatic regime, consistent with Guyana's location between 2º and 8º north of the equator. The climate is characterised by hot, wet summers and warm, damp winters. The annual average rainfall approximates 2,500mm, concentrated in two wet seasons respectively extending from May to July and November to January. Average maximum temperatures show little seasonal variation, averaging 27ºC, tempered by year-round trade winds. Maximum temperatures rarely exceed 28ºC on the more elevated Pakaraima Plateau 100km to the west. The length of exploration field season is all year round with occasional interruptions by rain during the wet season.

5.3 Access

The Karouni Gold Project is located approximately 180km south-southwest of Georgetown. The Project is situated in a remote area with no established population centres within the Project area aside from small portable mining alluvial camps/operations. The nearest town is Linden to the north-northest on the road route from Georgetown to site (5 - 6hr drive from Hicks Camp). The Project can be accessed by 4WD car or truck form either Bartica or Georgetown. Access from Bartica is via the Bartica-Potaro Road, taking the Issano Branch Road 135km from Bartica to the small settlement of 14 Mile, thence via a further 5.5km of bush track to Hicks Camp. The journey from Bartica to the Hicks Deposit takes approximately seven hours. Alternative access from Georgetown is gained via Sherima Crossing, a journey of approximately 7-10 hours. The Project

can also be accessed via helicopter to a pad constructed at the Hicks Camp. Topography and existing road access to the Karouni Gold Project are displayed on Figure 5-2. In early 2014, Troy completed the construction and permitting of the site airstrip.

Construction of an access road connecting the proposed Amalia hydropower project site to the existing Bartica-Potaro Road started in the first quarter of 2011. The completed road runs through the Karouni Gold Project area. Troy has completed road access from the Hicks Camp to the new Amalia Falls Road.

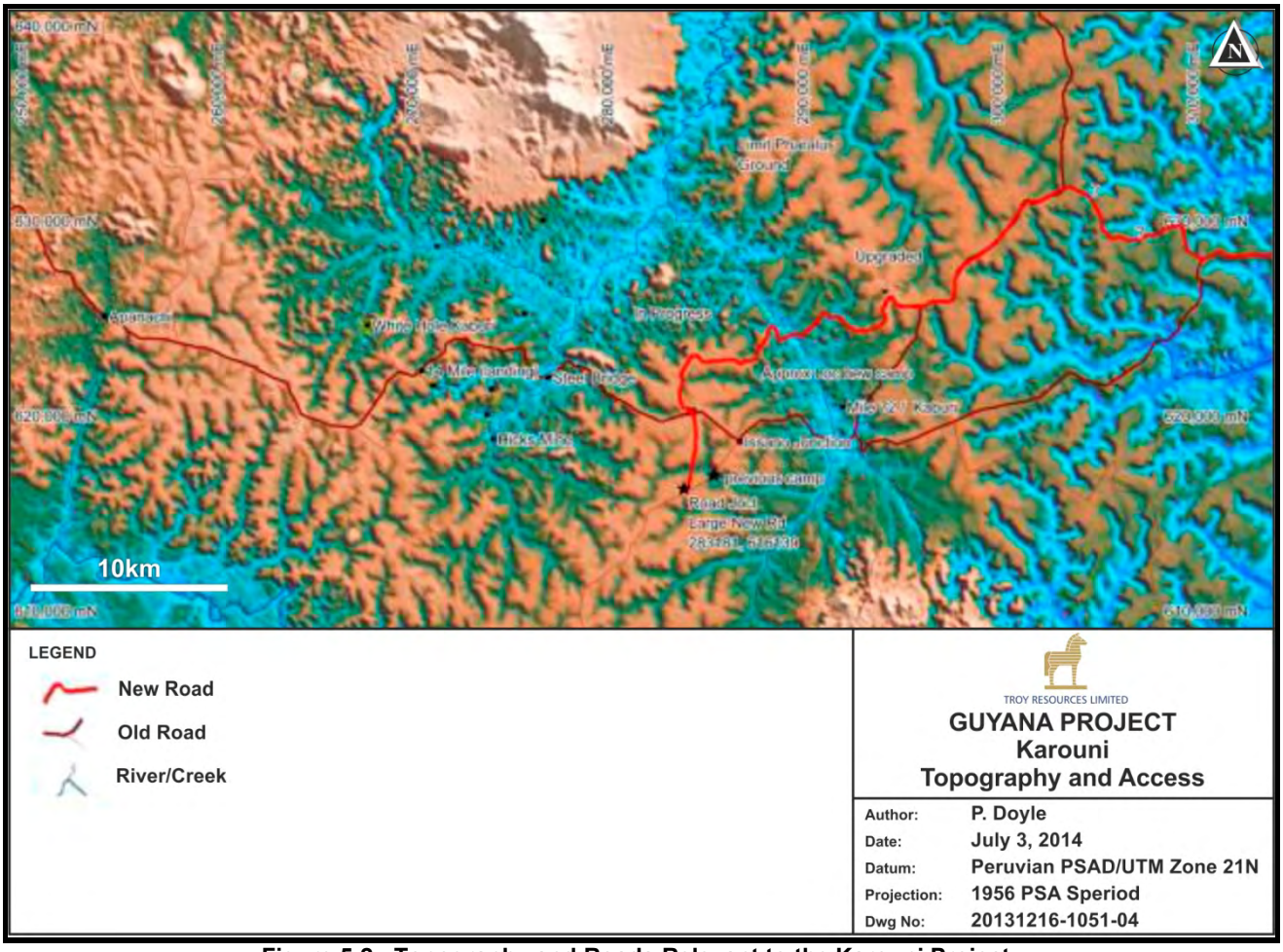

Figure 5-2: Topography and Roads Relevant to the Karouni Project

5.4 Infrastructure and Services

The population within the Project area is limited to small gold mining camps. Aside from the road and helicopter access described above, infrastructure within the Project area is essentially non-existent. The Project is serviced by an exploration camp located at the Hicks Zone that can accommodate up to 100 people (see Figure 5-3).The new mine development will be operated on diesel generated power.

The main administration office and warehouse will be located adjacent to the processing facility. The office will house all administration and technical staff. The warehouse will carry all necessary spare parts and consumables for daily operational requirements. An accommodation village for 300 personnel will be constructed to the south of the processing facility and adjacent to the airstrip. The village will have facilities for accommodating 30 personnel in ensuited type units. The village will include messing facilities for meals and access to satellite television and telephones.

The processing plant will require electrical energy and water to operate. The expected level of electrical energy use is ~5.7MW with electricity supplied from a Troy owned and operated diesel powered generating facility. The power generation capability will be made up of 5 diesel gensets of 2MW capacity each. The powerhouse will be located adjacent to the processing plant. Diesel will be trucked in from Georgetown and stored in a dedicated storage facility onsite.

A facility for storing 160,000 liters of diesel fuel will be constructed adjacent to the processing plant. The facility will be fully bunded and will deliver fuel to the powerhouse. In addition, there will be provision for refuelling light vehicles. Another storage facility will be located at the laydown and workshop area. This facility will be utilised for refuelling the open pit.

A fresh water storage facility will be constructed to hold potable water. In addition, a process water facility will also be constructed. The water usage will initially be high until the tailings return water has stabilized. Expected use of process water is ~800,000m3/a to cover wash down, camp, mine and other uses. Approximately 45% of the process water is expected to be returned from the TSF. The remainder will be derived from pit de-watering, a bore field and possibly small dams. Guyana is a very high rain fall area and a lack of water is not seen as a potential issue.

Waste rock dumps will be located to the north-east of the Hicks and Smarts Deposits. The Smarts Deposit dump will consist of one main dump whereas the Hicks Deposit dumps will be located adjacent to each of the pits. Top soil and oxidised waste material will be stored until completion of the dumps and then spread over them in order to enable rehabilitation and revegetation of the completed dumps.

The tails storage facility will be accommodated in a natural valley located to the south-west of the main Smarts and Hicks Deposits. A compacted, earthern embankment will contain the tails. The tails dam will be designed to hold all of the planned tails discharge from the processing plant over the forecast 3 year mine life.

Figure 5-3: Aerial View of the Hicks Exploration Camp

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6. History

6.1 Historical Background

Portions of the Karouni Gold Project have been held more or less continuously by small family gold mining syndicates (locally termed 'Pork Knockers') since the 1960's. This situation persists to the present day.

Portions of the current Project area were variously held under option to purchase agreements by Cominco (1974-1975), Overseas Platinum Corporation (1988) and Cathedral Gold Corporation (1993-2002).

In 1999, Cathedral Gold corporate joint ventured the property to Cambior Inc., then owner and operator of the Omai Gold Mine located 40km to the east, with a view to processing the Hicks mineralisation through the Omai processing facility. Cambior intended to use its existing mining fleet, rather than road trains, to haul mill feed from the Hicks Zone. Execution of this approach proved uneconomic and disruptive to the mining schedule at Omai itself. No further work was undertaken and the joint venture was terminated in 2000.

In 2002, Cathedral Gold Corporate became a service company to the oil and gas sector and spun its gold and base metals assets into a new company called Imperial Metals Inc. Imperial Metals has maintained an interest in the Hicks Zone to the present day and, under its agreement with Troy, retains a 1% net smelter return (NSR) royalty in the Project, applicable after the initial 200,000oz of gold production.

Azimuth, formerly Epsilon Energy Limited, announced the acquisition of Takatu Minerals Limited (Takatu) on 3 November 2009 resulting in the acquisition by Epsilon of the Projects including the Karouni Gold Project (formerly known as the West Omai Gold Project) in central Guyana. In July 2013, Troy completed a takeover of Azimuth. Troy holds the Karouni Gold Project tenement rights through Pharsalus Gold Inc., an indirect, wholly owned subsdidiary of Troy.

6.2 Past Exploration Work

The Eldorado Vein was worked in 1913 via limited shaft and drift development. A review of historic reports indicate that a single vein some 0.5m in width and grading between 23g/t gold and 37g/t gold was worked to a depth of 12m along a strike length of around 90m.

In 1933, the Geological Survey Department of British Guiana investigated an area of approximately 600 square miles along an area to the West of the Kaburi River. The exact boundary of this survey is not shown in the final report (Grantham et. al., 1933). The investigation included field mapping, soil sampling and washing and collection of rock specimens. A total of 70 microscope slide were prepared from the rock samples and a further 30 slides from sands and washed sample concentrate.

In 1947, the Geological Survey drilled three Diamond Core holes at the Eldorado Target spaced at approximately 120m intervals along strike. Core recoveries were reportedly extremely poor but nonetheless defined a 2m wide zone of veining and silicification at the contact between "white clay" and carbonaceous clay. No core was preserved and no assays are recorded.

In 1959, the Eldorado Target was described by a geological survey team as quartz veinlets and stringers developed over a width of at least 30m, hosted by black carbonaceous clay.

In 1962, the Larken Zone was described by a geological survey team as a 1.2m wide zone, extending over a strike length of 150m to 180m, exposed in several short trenches. Panning of crushed material revealed considerable gold, however no assays are recorded.

In 1963, the British Guyana Geological Survey completed a geochemical survey over the Hicks Zone defining a 60m wide zone of high chromium values over a strike length of at least 400m. This anomaly is interpreted to reflect the high magnesian basalt incorporated within the basement greenstone succession comprising the Hicks Zone, and is consistent with the presence of fuchsite alteration.

In 1968 the Hicks – Eldorado - Whitehall (Kaburi) Goldfield was investigated with the dominant control on mineralisation reported as a band of carbonaceous meta-sediments which contained "good grades throughout" and that elevated chromium values (presumably associated with fuchsite alteration) provided a reliable guide to higher gold values.

During 1974 - 1975, Cominco completed geological mapping and geochemical surveys, and drilled 64 25/8" auger holes (1,442m) and two AX Diamond Core holes (240m) at the Hicks Zone.

In 1988, Overseas Platinum Corporation (OPC) completed a program of geochemical and geophysical surveys, geological mapping, shallow trenching, auger drilling and 10 Diamond Core holes. This exploration was primarily designed to assess the platinum potential of the chrome anomaly associated with the Hicks Zone, at which time the company identified and refocused exploration on gold mineralisation.

In 1989, Denison Mines Guyana Limited through option agreements with local vendors conducted a systematic mapping, stream sediment program (1,002 samples), soil sampling program (1,646 samples) and airborne magnetometer survey. Soil samples were collected on a 200m x 50m grid and delineated a broad region of approximately 1.5km by 4km of anomalous soils in the vicinity of the Whitehall - Kaburi Target (see Figure 6-1).

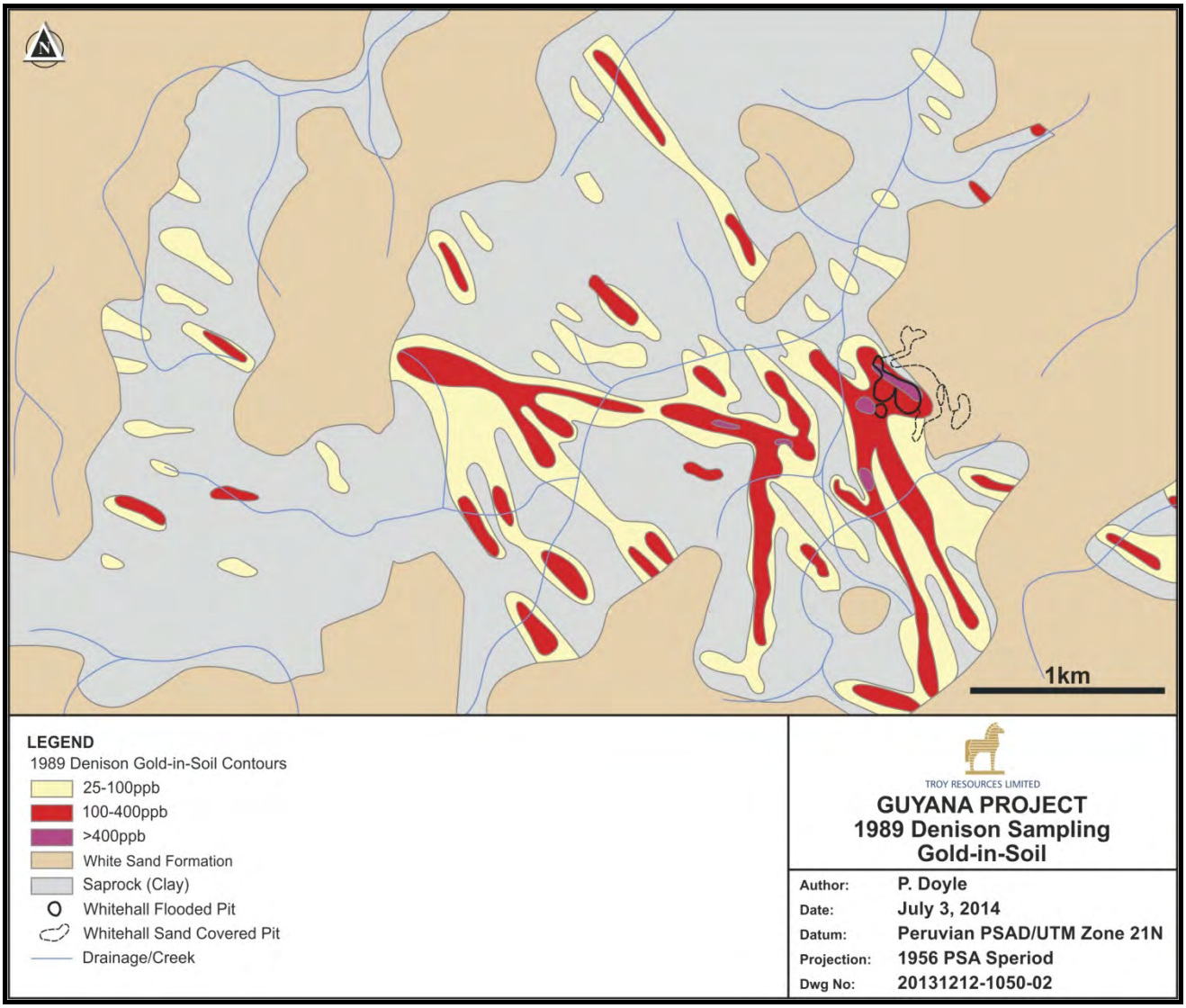

Figure 6-1: Denison 1989 Whitehall Kaburi Soil Sampling Results (White Sand Formation shown in tan)

In 1993, Cathedral Gold Corporation (Cathedral Gold) optioned the Whitehall (Kaburi) - Eldorado Targets (including the Hicks Zone), completing an initial program of surface trenching, mapping and sampling, resulting in the excavation of 30 trenches at 25m to 150m spacing, and to an average depth of approximately 2m, over the Hicks Zone. This work defined a northwest trending zone of semi-continuous gold mineralisation over a 1,600m strike length, within a zone of deformation and alteration averaging some 44m in width. Significant intervals recorded from trench channel sampling appear to represent the only valuable information remaining (see Figure 6-2).

The only information available on trench channel sampling completed by OPC and Cathedral Gold is that consistent 1m sample lengths were reportedly adopted, with the single exception of Cathedral Gold trench 93-19, where there are several intervals of 1.75m in length. While the sample preparation, analysis and security aspects associated with the Cathedral Gold trench channel sampling and diamond drilling are again not formally documented, it is reasonably presumed that the approach was identical to that adopted for the "super pit" samples described below, however this cannot be substantiated.

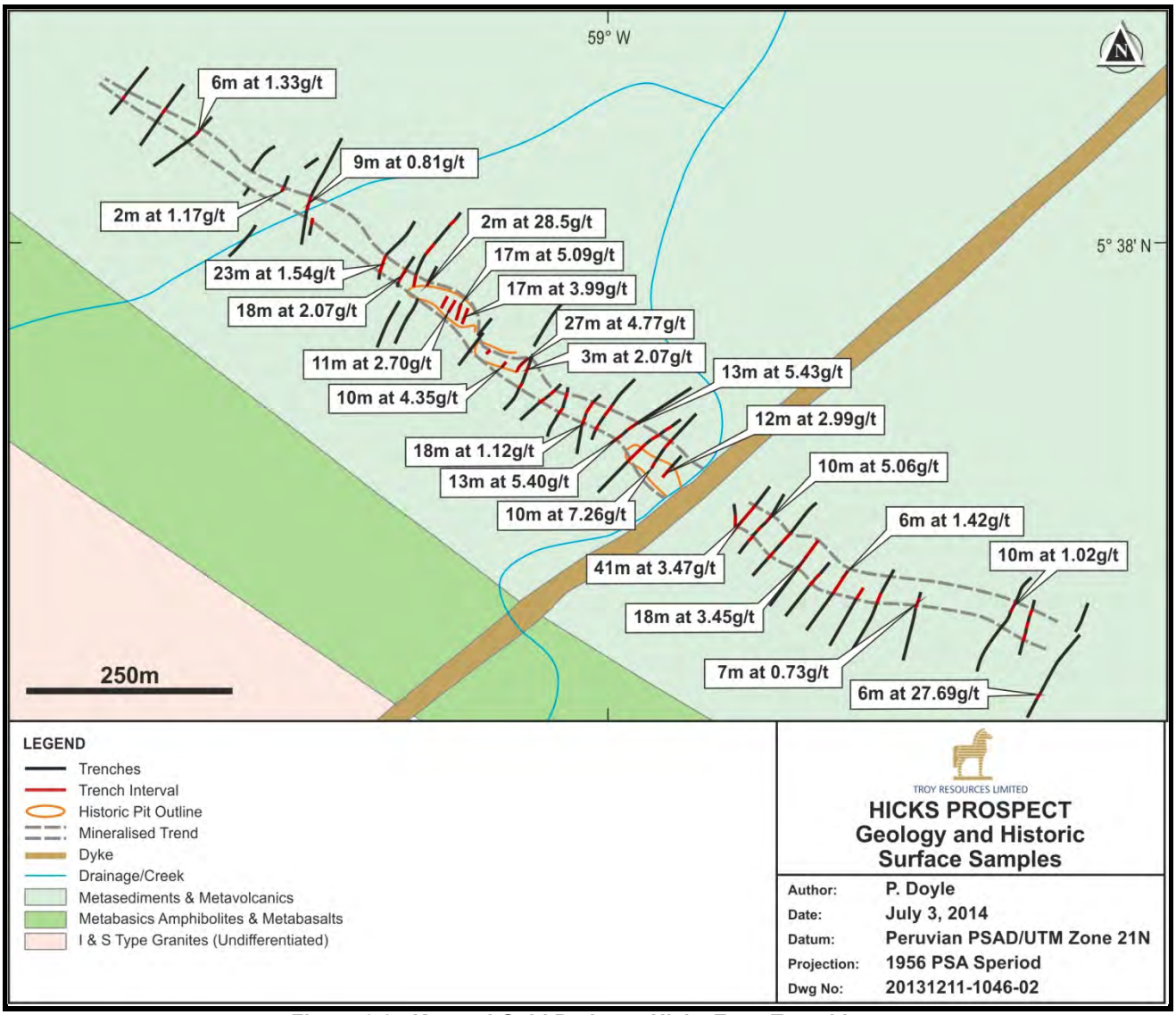

Figure 6-2: Karouni Gold Project – Hicks Zone Trenching

In 1994, Cathedral Gold completed 39 Diamond Core holes (3,901.7m) at the Hicks Zone on 100m spaced traverses, with two holes completed on each line. Additional trenching was carried out on the southeast and northwest ends of the Hicks Zone and followed up by limited drill testing where warranted. Other programs included magnetic and IP geophysical surveys and hand auger/soil sampling.

In 1995, Cathedral Gold completed a further 19 Diamond Core holes in two campaigns at the main Hicks Zone, designed to infill selected sections on a 50m line spacing. The orientation of mineralised zones comprising the Hicks Zone is well established, and it is estimated that the true width of mineralised intercepts reported in Table 6-1 represents some 60% to 70% of the downhole intervals. Additional ground magnetic surveys were completed over the entire property at 100m line spacing, infilling to a 50m line spacing over the Hicks Zone. Previously untested magnetic anomalies, primarily within the Eldorado Batholith and at the southeast extremity of the Hicks Zone, were followed up with infill hand-auger sampling, trenching and drilling.

The majority of the Cathedral Gold Diamond Core is located within collapsed core racks and scattered over the site at the Hicks Camp. This drill core cannot be salvaged in any meaningful manner and is unsuitable for further use. Examination of the Cathedral Gold Diamond Core remaining at the Hicks Camp indicates that selected mineralised intervals were cut using a diamond saw, with half-core bagged for analysis. The drilling database indicates a considerable range of sample lengths up to a general maximum of 2m, with very occasional intervals (presumably internal waste) up to a maximum of 7.06m in length. For details of the 1994 – 1995 Cathedral Gold Diamond Drilling Gold Assay intercepts refer to Section 6 of the NI-43101 Technical Report "Preliminary Economic Evaluation West Omai Gold Project dated February 28, 2014.

Also in 1995, Cathedral Gold excavated a pit, termed the 'Super Pit', at the Hicks Zone adjacent to the existing Blackwater Pit previously worked by artisanal miners. The pit measured 80m in length by 14m in width by 3.8m in depth and approximately 4,000m^3 (6,800t) of material was excavated and placed on the periphery of the pit to preserve the mineralised material and protect the pit from rainwater influx and the adjacent hydraulic mining operations. The pit floor was gently sloped, and a drain installed inside the periphery of the pit to preserve the floor for sampling.

Considerable mapping detail and sampling within the "Hicks Super Pit" was undertaken and the resulting maps and sample locations are still available. The sampling maps provide an indication of saprolite grades and the continuity of gold mineralisation in oxide material. Mapping results from the "Hicks Super Pit" shows that mineralisation (in plan-view) occurs in lenses; while historical drillhole 94-30 which intercepts this zone at depth illustrates that the zones are also lensoidal with depth.

In January 1996, a preliminary mineralisation estimate was undertaken for the Hicks Zone. Further exploration included limited regional exploration to identify additional mineralisation to compliment that identified at the Hicks Zone, however exploration was largely suspended during 1997 due to a declining gold price and an inability to access capital.

In 1998, Cathedral Gold joint ventured the property to Cambior Inc., owner and operator of the Omai Gold Mine located 35km to the east, with a view to processing the Hicks Zone mineralisation through the Omai processing facility. Cambior constructed an access road between the two projects and completed 22 confirmatory Diamond Core holes at Hicks (1,118m), prior to preparing a Mineral Resource estimate and undertaking preliminary "internal" mining studies. Cambior Inc. intended to use its existing mining fleet, rather than road trains to haul mill feed from the Hicks Zone, however execution of this approach proved uneconomic and disrupted the mining schedule at Omai Mine itself. No further work was undertaken and the joint venture was terminated in 2000. A total of 9,463m of trench and channel sampling and 17,191m of Diamond Core drilling in 93 holes were completed by Cathedral Gold and Cambior Inc. during the period 1992 to 2000.

Between 2000 and 2003, the GGMC completed two separate (yet systematic) investigative programs covering the current Karouni Gold Project (see Figure 6-3). These programs involved GIS database compilation, geological mapping, stream sediment sampling, rock chip sampling and petrology. Stream sediment sampling a sampling density of one site per 6.5km^2, with -30# BLEG and panned concentrate samples collected from each site. The samples were analysed for gold, silver, copper, platinum and palladium. While an extensive cover of White Sand Formation masked the basement geochemical response over much of the Project area, clusters of gold values >200ppb gold from the -80# samples are clearly associated with known gold occurrences wherever the Barama-Mazuruni Greenstone Belts are exposed (see Figure 6-3). These clusters coincide with the Hicks, Eldorado and Whitehall (Kaburi) Prospects, where respective recorded -80# sample maxima are 2,670ppb gold, 1,850ppb gold and 4,530ppb gold.

Similarly, the BLEG results generated local sample maxima of 2,330ppb gold, 63ppb gold and 376ppb gold at the Hicks, Eldorado and Whitehall (Kaburi) Targets respectively. Local BLEG maxima associated with tributaries along the Kuribong River, representing the southern project boundary, include 53ppb gold, 171ppb gold and 68ppb gold. These results may reflect alluvial material associated with the Kuribong River itself.

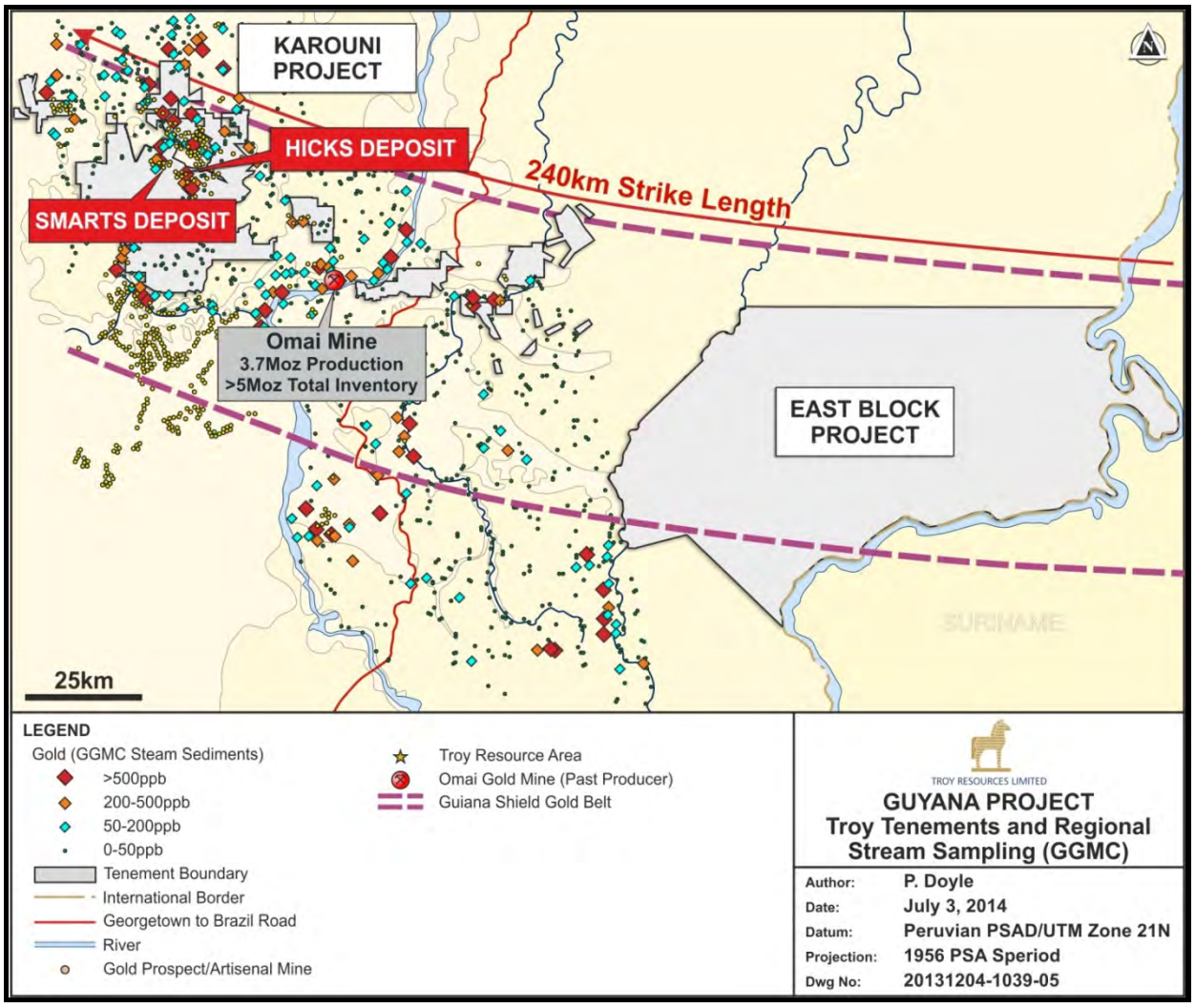

Figure 6-3: Karouni Gold Project – GGMC Stream Sediment Gold Geochemistry

Azimuth, formerly Epsilon Energy Limited, announced the acquisition of Takatu Minerals Limited (Takatu) on 3 November 2009 resulting in the acquisition by Epsilon of the Projects including the Karouni Gold Project (formerly known as the West Omai Gold Project) in central Guyana. Takatu Minerals had assembled the land package that is the Karouni Gold Project but did not conduct any field exploration.

Modern exploration resumed on the Karouni Gold Project in 2010 when Azimuth commenced a program that included; prospecting, geological mapping, geochemical sampling, rock sampling, trenching, channel sampling as well as Reverse Circulation (RC), Aircore (AC) and Diamond Core (DC) drilling.

Azimuth's drilling campaigns to date have targeted mainly the Hicks and Smarts Deposits with limited tests of satellite targets such as further campaigns at Whitehall (Kaburi), Larken and a number of other prospects. For the purposes of this report the database close-out is considered to be April 30, 2014. Since March 2013, Troy has managed the drilling with all of the drilling focussed within the Mineral Resource areas at Smarts, Hicks and Larken Zones. Azimuth commenced drilling of the Hicks and Smarts Zones in August 2010 and this continued through to July 2013. Azimuth has completed 1002 RC drillholes all are five inch diameter holes and range in length from 6m to 396m for a total of 89,013m of drilling. The majority of RC drilling was completed at Smarts (596 holes for 47,020m), Hicks (187 holes for 17,580m), Larken (31 holes for 3,322m) and Whitehall (Kaburi) (40 holes for 5,627m). Minor RC drilling campaigns have also been carried out at Eldorado, Croft, Far SE Hicks, Gibbs, Kaburi West, Kanhai, Norby, El Paso, Gold Star and Powers.

Under Azimuth, Diamond Core drilling commenced at the Smarts Zone in August 2010 and at Hicks Zone in September 2012. The majority of core is of HQ and HQ3 size and minor PQ was used to gain better recovery through the oxide zone. Diamond holes range in length from 31m to 480m for a total of 32,826m of drilling. Azimuth drilled 94 holes at the Smarts Zone for 16,528m of drilling, 19 holes at the Hicks Zone for 5,148m, 2

holes at Larken for 308m, and 2 holes at Kaburi for 348m. Detailed core logging has been completed with core sampling carried out over visually mineralised sections with visible gold associated with quartz veins in sheared chlorite +/- sericite +/- carbonate altered, strongly-sheared Mafic to Intermediate Volcanics and Volcanoclastics.

Azimuth also drilled 93 air-core (AC) holes for 5,193m which were drilled at the Smarts and Hicks Deposits and for further regional exploration purposes.

Auger sampling has been carried out by Azimuth and previous companies. Data is available for 16,700 samples of which 11,100 were collected by Azimuth.

In July 2013, Troy completed a takeover of Azimuth. Troy holds the Karouni Gold Project tenement rights through Pharsalus Gold Inc., an indirect, wholly owned subsdidiary of Troy.

The current Karouni Gold Project database contains all available historical trench and drilling data. A total of 166 trenches appear in the database. Of these, 32 trenches have unknown coordinates and 19 have not been surveyed. A total of 125 trenches have both known co-ordinates and surveys. Information is not available from the historical log sheets, but the majority of trenches were excavated between 1993 and 1995 and correspond with Cathedral Gold trench maps so it can be reasonably assumed that the majority of trenches were excavated by Cathedral Gold.

A total of 134 historical drill holes appear in the database and are summarised in Table 6-1. Of these 13 Diamond Core drill holes have unknown collar coordinates. Troy re-surveys old drill collars as they are located and verified.

Table 6-1: Summary of Historical Drilling - Karouni Project (Drilling completed prior to Azimuth's Acquistion of the Project)

Hole Prefix	Prospect	Type	Number	Metres	Company	Hole Sequence
H	Hicks	DC	2	240.47	Cominco	H-65 to H-66
W	Nine Mile	DC	5	671.63	Cominco	W-10, W-21 to W-24
EH88	Eldorado	DC	10	964.1	Overseas Platinum Corporation	EH88-01 to EH88-10
DH94	Hicks Eldorado	DC	45	9,323.51	Cathedral Gold	DH94-01 to DH94-41
DH95	Hicks Eldorado Tallman	DC	36	7,583.75	Cathedral Gold	DH95-01 to DH95-71
DH96	Tallman	DC	6	563.61	Cathedral Gold	DH96-06 to DH96-11
EKP96	Kaburi East	DC	6	641.03	Cathedral Gold	EKP96-01 to EKP96-06
NMDH96	Nine Mile	DC	2	198.42	Cathedral Gold	NMDH9601 to NMDH96-02
OCG-98	Hicks	DC	22	1,117.90	Cambior	OCG-98-72 to OCG-98-93

6.2.1 Historical Bulk Density Test work Pre-Azimuth

Bulk density test work was conducted by Cathedral Gold and later Cambior. Bulk density was determined for 1,074 core samples from drill holes DH94-01 to DH94-32 and DH95-32 to DH95-50. All historical diamond holes are located on the Hicks Deposit. The average total density of 2.84t/m^3 is similar to the primary bedrock value of 2.75t/m^3 used by Cathedral in 1996 and 2.75t/m^3 used by Cambior in 1998. These measurements represent dry bulk density values for in-situ material. Neither Cathedral Gold nor Cambior conducted bulk density measurements for the weathered saprolite material. This reflected a lack of available drill core to undertake bulk density test work on.

6.3 Production

No mineral production is officially recorded within the Karouni Gold Project however gold continues to be mined by small private syndicates operating within the property or within tenements excised from the property on both an informal and formal basis.

Historic records indicate that the Eldorado Vein was worked in 1913 via limited shaft and drift development. A single vein some 0.5m in width and grading between 23g/t gold and 37g/t gold was reportedly worked to a depth of 12m along a strike length of around 90m. Inspection of the Eldorado Target in 1923 by the Geological Survey of Guyana, found that the workings had collapsed, but reported that approximately 100 tonnes of stockpiled quartz, grading around 30g/t gold, remained.

In 1959 the Eldorado Target was being worked by several small parties. The clay was washed for its direct gold content and the vein material preserved for later crushing and additional gold recovery.

In 1963, a small sluicing operation was commenced at the Eldorado Target, however production ceased shortly afterwards following the death of the owner.

The Whitehall (Kaburi) Vein was reportedly worked sporadically by manual methods between 1916 and 1926. Mining is understood to have comprised the development of a small open pit some 10m wide, 10m long and 15m deep before the walls collapsed and operations ceased.

The vein zone was again worked in 1933 by about 20 men, developing numerous shallow quartz veins and stringers over an unrecorded width, along strike from the earlier workings.

Between 1996 and 2006, private operator Patrick Harding exploited the Whitehall (Kaburi) Vein Zone and associated alluvials via hydraulicking and sluicing. A D6 bulldozer and 30t excavator were utilised in overburden removal. Operations during the initial two years focussed on alluvial deposits along a 3km length of Lucky Strike Creek and other watercourses, which drain the primary source of mineralisation. As mining progressed upstream, the Vein Zone was ultimately exposed in the bed of Lucky Strike Creek and progressively extended. The operation excavated a pit approximately 200m in diameter to a depth of 5m to 15m into saprolite through superficial White Sand Formation cover. The depth of mining was limited by inefficient gravel pumps, rather than the base of saprolite mineralisation. Production records filed with the Mines Commission indicate that approximately 18,000 ounces of gold were produced between 1996 and 2007. Presently the area is not being worked and the pit is partially backfilled with white sand overburden and flooded where it remains open.

The majority of production from the Karouni tenements continues to be derived from alluvial and eluvial sluicing operations, predominantly associated with the Hicks, Eldorado and Whitehall (Kaburi) Targets.

7. Geological Setting and Mineralisation

7.1 Regional Geology

7.1.1 Stratigraphy

Guyana is located on the northeast coast of South America, and is entirely comprised or underlain by rocks of the Precambrian Guiana Shield. The Guiana Shield lies between the Orinoco and Amazon Rivers and encompasses all or portions of neighbouring countries Suriname, French Guiana, Venezuela, Brazil and Columbia, aggregating to an area of 1.6 million km^2.

From oldest to youngest, the Guiana Shield and cover sequences comprise the following principal stratigraphic successions (see Figure 7-1).

- The Imataca Complex, an Archaean complex composed of 3.4Ga - 2.7Ga protolith, on which are superimposed high grade metamorphic rocks of Trans-Amazonian age (2.2Ga - 2.0Ga) exposed in the Ciudad Bolivar region of Venezuela.
- High grade granulites and gneisses of the Central Guyana Granulite Belt of Palaeoproterozoic age (2.4Ga - 2.3Ga).

- A widely developed Palaeoproterozoic granite-greenstone succession, which was deformed and metamorphosed during the Trans-Amazonian tectono-thermal event (2.2Ga - 2.0Ga). The Trans-Amazonian age rocks stretch across much of the northern third of the shield, incorporating a succession of older gneissic granitoids, with the two principal occurrences in Guyana referred to as the Bartica Gneiss and Kanuka Group. These gneissic complexes are considered equivalent to the Cape Coast Suite of West Africa and are enveloped by a less deformed succession of basic to acid volcanics, sediments and syntectonic granitoids. In Guyana, the greenstones are referred to as the Barama-Mazaruni Supergroup, in Venezuela the Pastora Supergroup, while in Surinam and French Guiana the term Maroni Supergroup is used. The greenstones are intruded by syntectonic granitoids (2.25Ga - 2.0Ga) equivalent to the Dixcove Suite of West Africa, providing an upper age limit for the Trans-Amazonian Orogeny, similar to that recorded for the Eburnean Orogeny of West Africa.
- The Uatuma Supergroup comprises a mostly undeformed and unmetamorphosed Mesoproterozoic (1.9Ga - 1.5Ga) sequence of supracrustals deposited above a major regional unconformity with the underlying Palaeoproterozoic basement. From oldest to youngest, these include 1,000m - 1,600m thick continental sandstones and conglomerates of the Muruwa Formation, and the 1,000m - 2,000m thick felsic volcanics, terrigenous sediments and sub-volcanic intrusives comprising the Iwokrama Formation.
- The Uatuma Supergroup is unconformably overlain by the topographically distinctive Roraima Group, which forms the Pakaraima Plateau bounded by a near vertical escarpment. The Roraima Group comprises a 1,000 – 3,600m thick sequence of relatively flat lying, unaltered, conglomerates and sandstones with subordinate siltstones and felsic volcanics. The age of the Roraima is constrained by the underlying basement and mafic intrusives to between 1.9Ga - 1.6Ga.
- Intruding the Roraima and Trans-Amazonian basement are dykes and sills of gabbro and dolerite of the Avanero Suite. These intrusives are typically of substantial thickness and may extend laterally for hundreds of kilometres. The Avanero Suite intrusives are undeformed and unmetamorphosed. Ages of the Avanero suite have been determined by radiometric dating to 1.84Ga - 1.61Ga.
- Other basic intrusives, younger than the Avanero Suite, which cluster at ages of 1330, 1213, 907, 700-570 and 430-500 million years before present. These dykes are designated the PAPA dykes, an acronym standing for Post-Avanero Pre-Apaotoe.
- Opening of the Atlantic during the Mesozoic resulted in the formation of intracratonic rifts within the Guiana Shield, which were filled with basaltic lavas and terrestrial sediments. In Guyana these Mesozoic rifts are represented by the Takutu Graben, an east-northeast trending rift structure, and the mostly offshore Guyana coastal basin.
- Associated with the opening of the Atlantic Rift was the emplacement of the dyke swarms known as the Apatoe Suite. Dykes of this suite are typically narrow, fine grained doleritic dykes of great strike extent, which predominantly trend north-northeast to east-northeast.
- Quaternary to Recent sediments of the Corentyne Group veneer much of the basement geology in northern Guyana. Clean shallow marine to estuarine quartz sands of the Berbice Formation (also referred to as the Mackenzie or White Sand Formation) are most prevalent, increasingly overlain by reducing fluviatile clays of the Demerara Formation towards the northern coast.

The Guiana Shield has been correlated with the Leo-Man Shield of West Africa, and it is generally accepted that prior to the opening of the Atlantic during the Mesozoic the two shields formed a contiguous craton. The Archaean Imataca Complex can be correlated with the Archaean Liberian Province, the Central Guyana Granulite Belt with the Dimbroko Zone in Ivory Coast, the Barama-Mazaruni greenstones with the Birimian greenstones and the Trans-Amazonian tectono-thermal event with the Eburnean Orogeny.

7.1.2 Structure

The boundary between the Imataca Complex and the Barama-Mazaruni Supergroup in Guyana is represented by the Guri Faults, which collectively comprise a distinct linear east-northeast trending crustal break.

The greenstone belts of the Barama-Mazaruni Supergroup are strongly deformed, with at least two episodes of tectonism evident in many areas. The metamorphic grade of the greenstone succession is typically greenschist facies, but may reach amphibolite facies close to belt margins and granitoids.

The Muruwa and Iwokrama Formations of the Uatuma Supergroup are weakly deformed, typically comprising broad open folds.

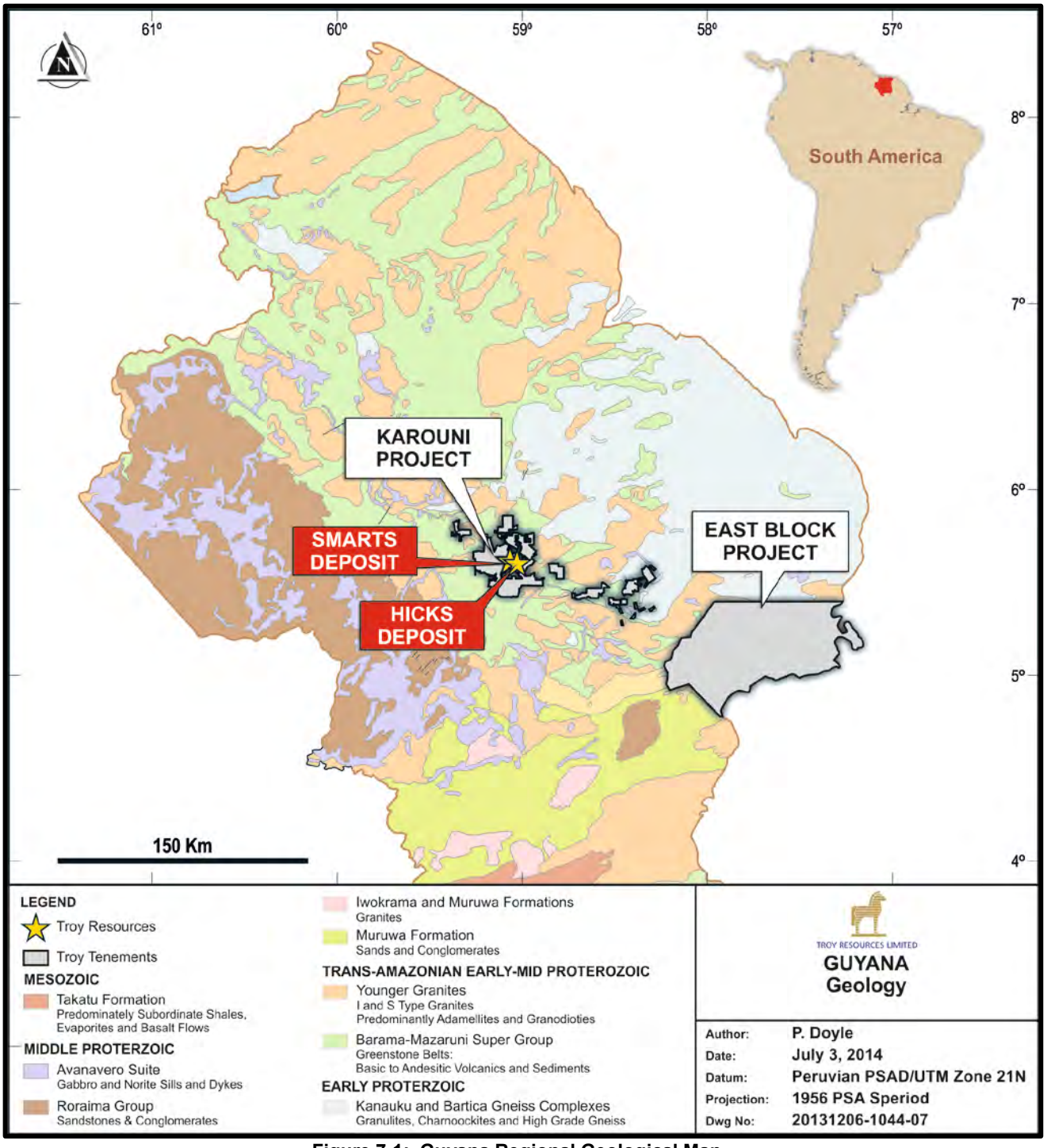

Figure 7-1: Guyana Regional Geological Map

7.1.3 Mineralisation

The Imataca Complex incorporates substantial iron ore deposits in Venezuela, comparable to similar deposits within Archaean portions of the Leo-Man Shield in Liberia, Guinea and Sierra Leone of West Africa.

Just as the Birimian Series of West Africa is noted for its substantial gold endowment, the lesser explored Barama-Mazaruni Greenstones of Guyana and their equivalents in adjacent countries include several significant gold occurrences such as Omai in Guyana, El Callo, Las Christinas and Brisas in Venezuela, Gross Rosebel in Suriname, and Camp Caiman, Dorlin and Yaou in French Guiana (see Figure 7-2). Similarly, the greenstones are also known for their significant deposits of bauxite, manganese and diamonds, along with lesser copper, molybdenum, lead-zinc, chrome and nickel. During the Tertiary and Quaternary periods, the Guiana Shield has undergone continuous passive subsidence along its oceanic margin, creating deep (2,500m - 4,800m) and mostly offshore sedimentary basins prospective for oil. The substantial bauxite deposits of the Guiana Shield typically occur at the contact between recent unconsolidated sediments of the Berbice Formation and the basement successions.

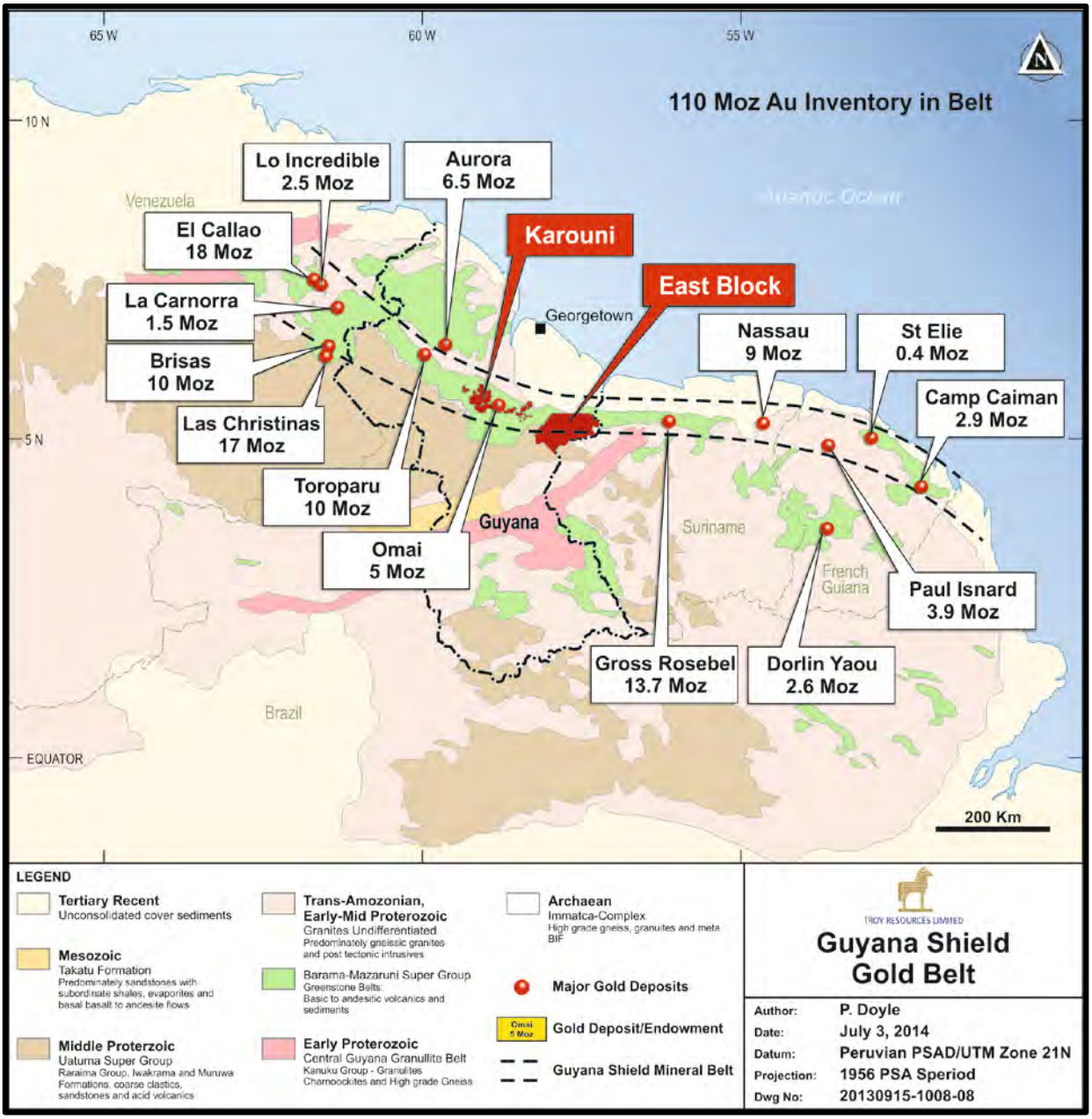

Figure 7-2: Troy Project Locations and Key Gold Projects of the Gold Guiana Shield Belt

7.2 Local Geology

Although partially obscured by a veneer of Tertiary to Holocene sedimentary cover, the basement geology of the Karouni Project is dominated by west-northwest trending greenstone belts of the Palaeoproterozoic Barama-Mazaruni Supergroup, interspersed with extensive Granitoid Batholiths also of Trans-Amazonian age.

The northern tenements straddle the northeast limb of an interpreted shallowly plunging, northwest trending anticline (see Figure 7-3). The core of the anticline is occupied by relatively undeformed syntectonic Granitoids, notably Granodiorite and Monzonite, around which is wrapped greenstone stratigraphy. The greenstones locally comprise a basal succession of high magnesian basalts, tholeiitic basalts and andesites, which are locally intruded by a Felsic to Intermediate Porphyry, overlain by a succession of Intermediate Volcanoclastics and Sediments, incorporating more persistent Carbonaceous Shale horizons. This stratigraphy reasonably is consistent with that recorded at the Omai Mine, 35km to the southeast.

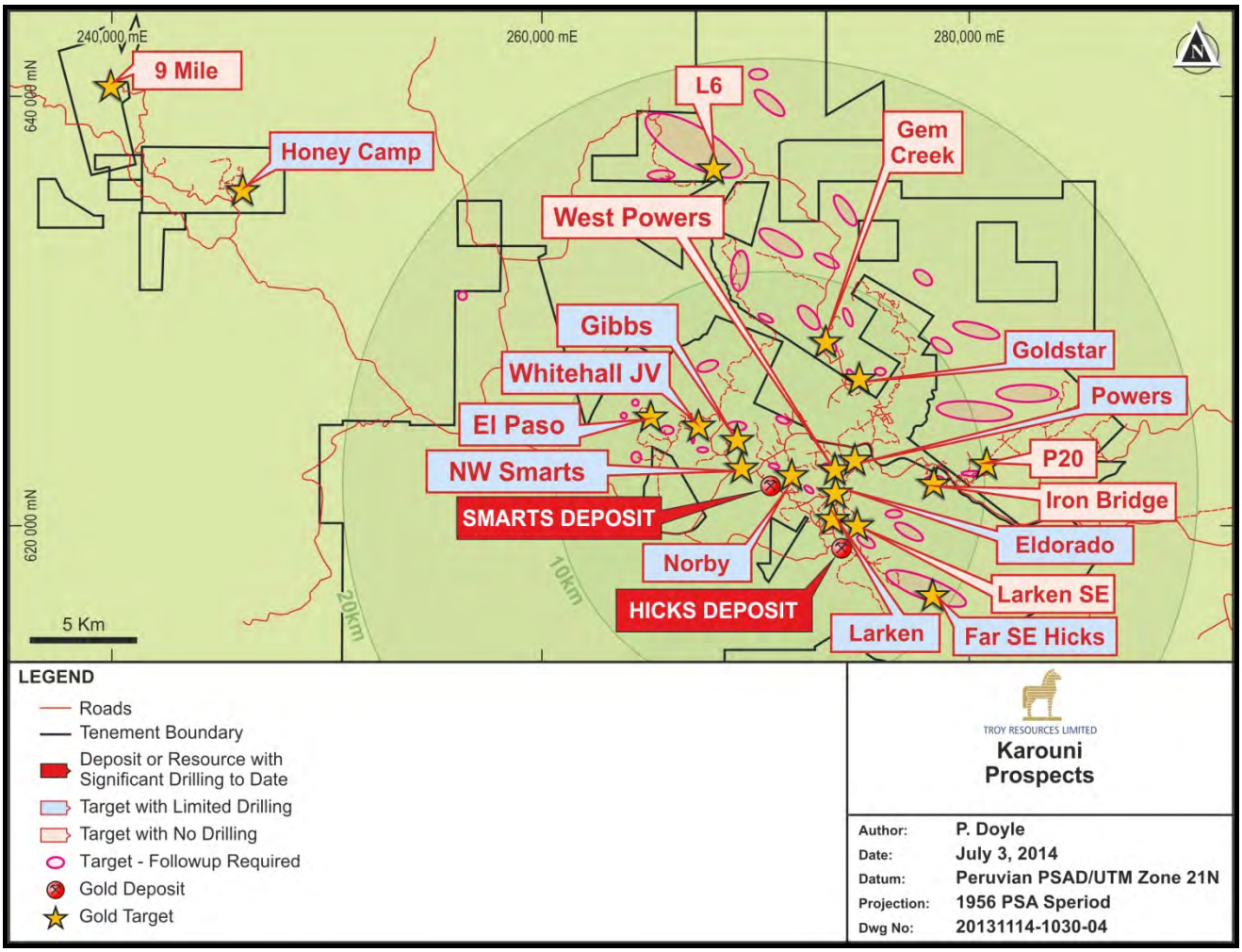

Figure 7-3: Karouni Project Geology – Main Prospect Areas

The Greenstone succession is variously metamorphosed to Upper Greenschist Facies, increasing to Amphibolite facies along the granite contact. The basement is intruded by a series of northeast trending dolerite dykes of the Avanero Suite, one of which intersects mineralisation associated with the Hicks Zone immediately east of Blackwater Creek.

Most of southwest portion of the tenement block that hosts the Hicks, Eldorado and Whitehall (Kaburi) Zones, is veneered by Tertiary to Holocene Sediments. The Lower Cover Sequence mantling the basement is less persistent and comprises a cream coloured, poorly sorted, Oligomictic Conglomerate with angular quartz fragments cemented by Quartzose Sand or Arkose. This Conglomerate is un-named, but is presumed to be of Tertiary age. It is mildly silica indurated and frequently strongly mineralised with gold where proximal to basement mineralisation. Where exposed, the unit appears to be up to 8m in thickness.

The Basal Conglomerate is overlain by more persistent cover of the Corentyne Group, more specifically, the Pleistocene to Oligocene White Sand (or Berbice) Formation. As the name suggests, the White Sand Formation comprises distinctively unconsolidated clean white sand of littoral or shallow marine origin that veneers much of the coastal plain of Guyana. The unmineralised White Sand Formation ranges up to 40m in thickness and obscures much of the basement geology (see Figure 7-4).

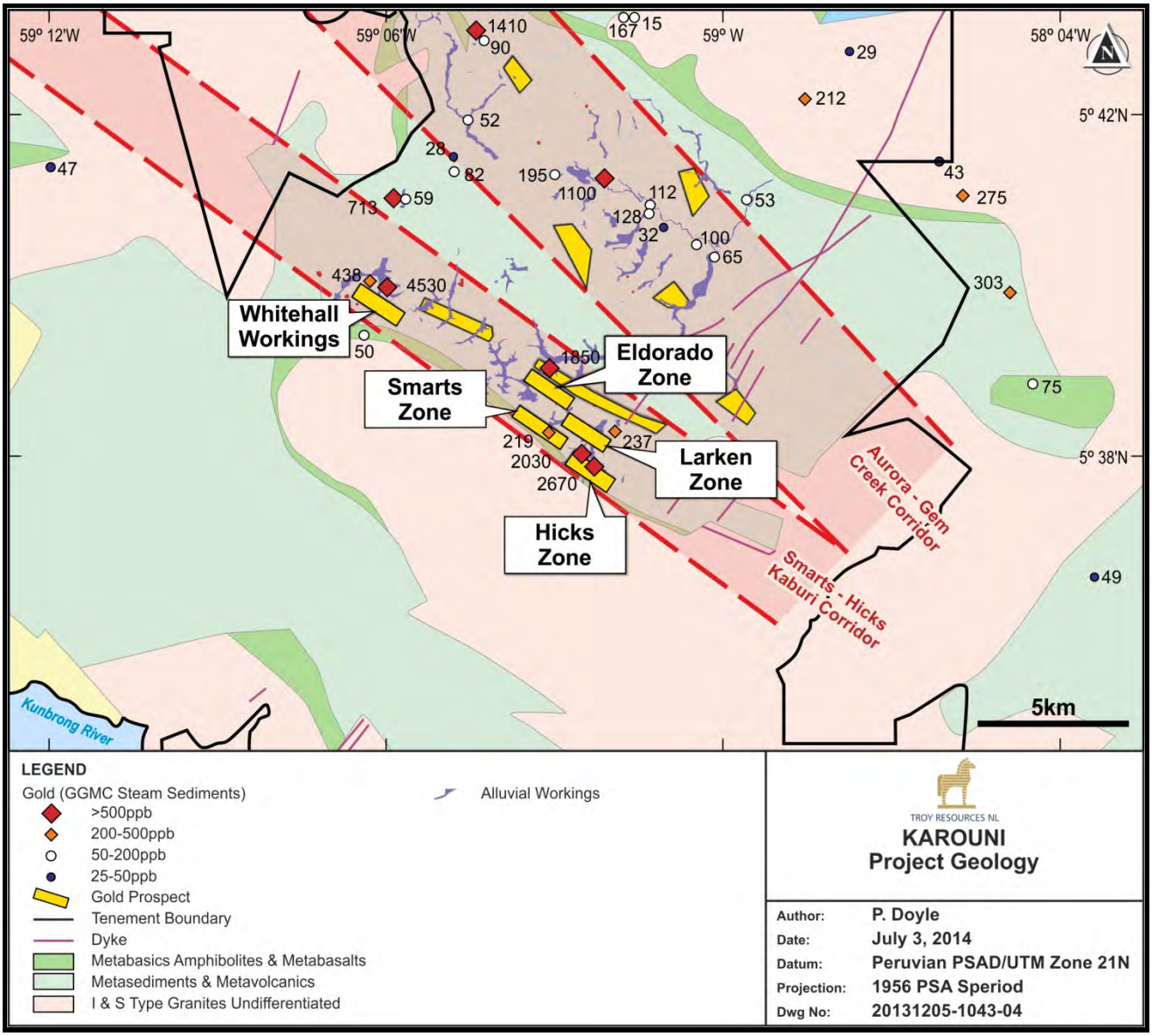

Figure 7-4: Karouni Project Geology & Main Prospect Areas

All surface "bedrock" in the area is typically altered to saprolite to depths of several metres to several tens of metres (see Figure 7-5). The stratigraphy is locally overlain by unconsolidated marine and Deltaic Sands plus Clays of the Cretaceous-Tertiary Berbice Formation.

Figure 7-5: Overhead View Historic Workings Smarts Pit (Looking North)

7.3 Property Geology and Mineralisation

Primary gold mineralisation is exposed at several localities within the Karouni Project, the most notable being the Hicks, Smarts and Larken Zones along the northern extremity of the project (see Figure 7-6 and Figure 7-7). Here the White Sand Formation cover has been removed by erosion to expose the underlying mineralised Palaeoproterozoic Greenstone successions of the Trans-Amazonian Barama-Mazaruni Group. Extensive superficial cover of White Sand Formation within the central and south portions of the project tenements masks the basement lithology and would conceal any gold mineralisation. The acquisition and evaluation of airborne geophysical data has however indicated that the Barama-Mazaruni Greenstone Belts and associated syntectonic intrusives persist at shallow depth beneath this cover.

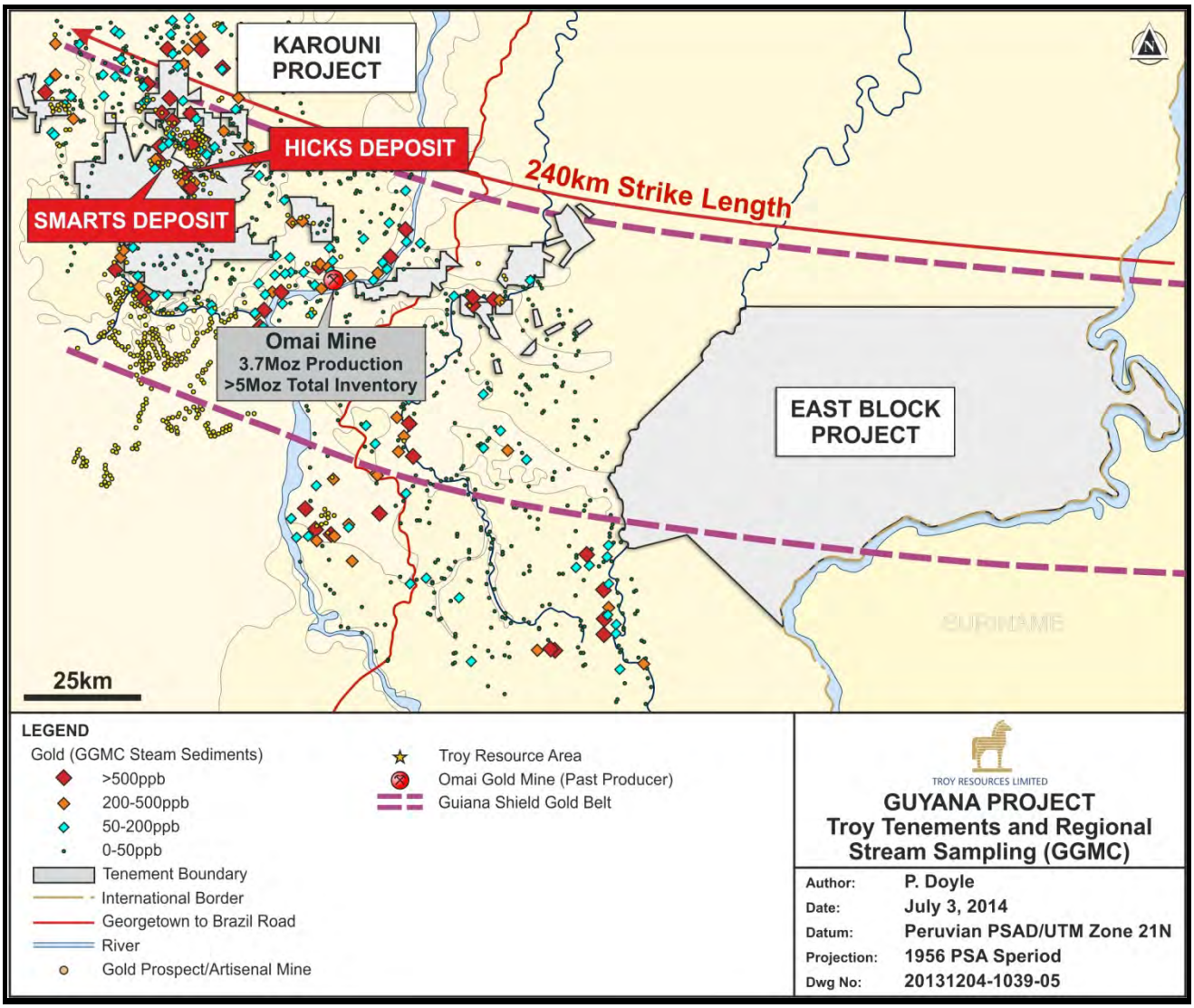

Figure 7-6: Karouni and East Block Project Corridor

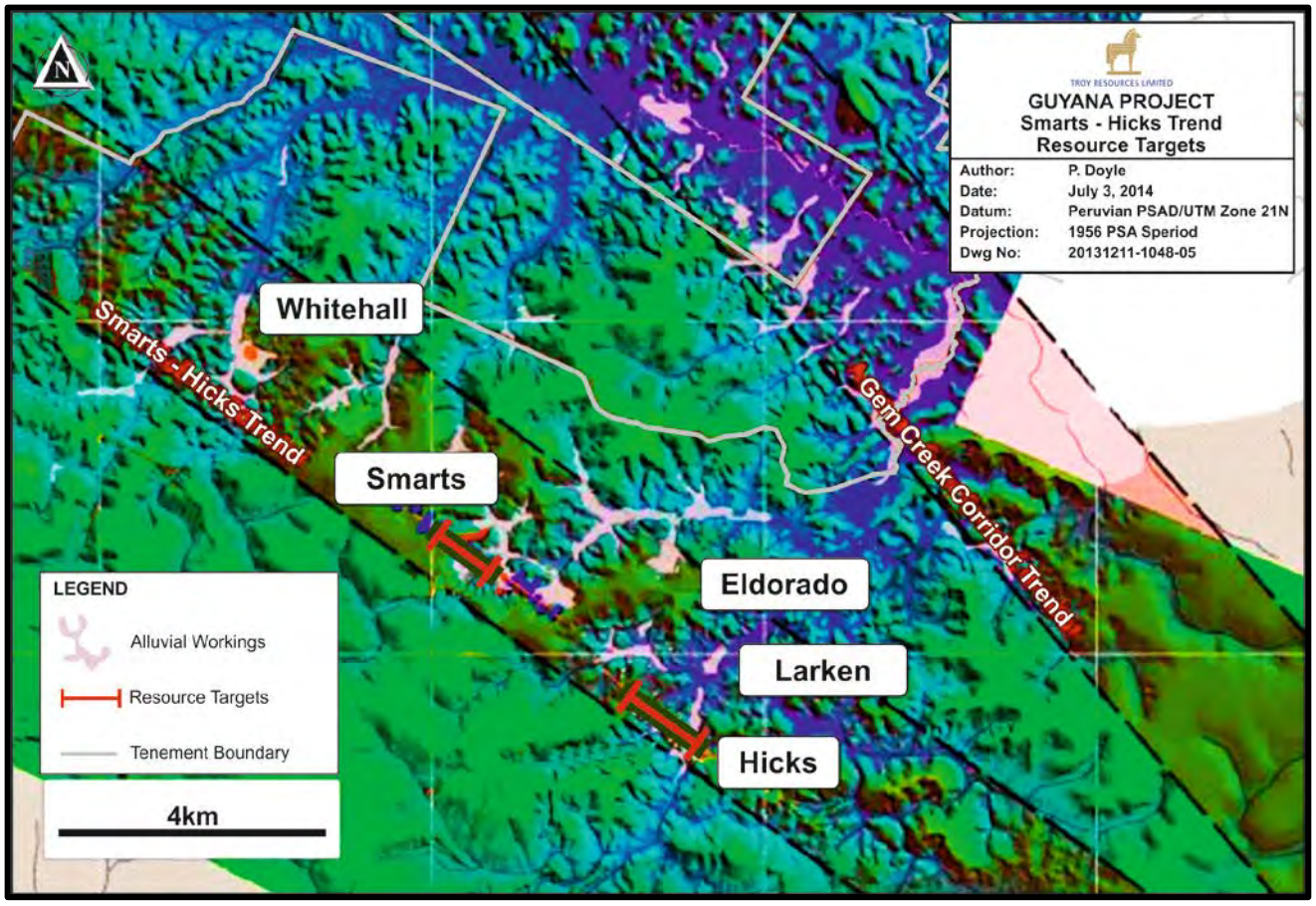

Figure 7-7: Karouni Gold Project, Omai-Hicks-Kaburi Corridor

Hicks Deposit

Gold mineralisation at the Hicks Deposit is hosted by a northwest trending, sub-vertical to steeply southwest dipping shear zone some 2,500m in strike length and up to 60m wide in places. The shear has developed within Basalts and Andesites comprising the footwall Greenstone Succession along the north-eastern limb of a shallowly northwest plunging anticline. Auriferous mineralisation is also noted at the contacts of Porphyry-Granite intrusives. The shear is comprised of semi-continuous zones of quartz lenses and quartz-carbonate veining or brecciating.

Numerous, moderately well-defined gold-rich lenses, up to 15m wide, occur within the shear and are characterized by anomalous quartz veining, quartz flooding, shearing, chloritization, seritisation and pyritisation. Visible gold and the majority of gold values typically occur within and along margins of quartz veins, in silicified Granitic Dykes, and in adjacent, pyritic, often sheared Meta-Andesite. Pyrite is common at up to 3% by volume, with local, trace amounts of molybdenite, galena and sphalerite, associated with auriferous quartz veins. Mineralisation is variously accompanied by silica-sericite-chlorite-carbonate-pyrite-tourmaline alteration, while fuchsite is developed within porphyry intrusives in contact with high magnesian basalts and along shear zones.

In general, the northwest trending mineralised shear has developed within a steeply inclined sequence of metamorphosed Andesitic to Basaltic Volcanics, adjacent to the northwest margin of the Eldorado (Granodiorite) Batholith. The volcanic stratigraphy is cut by a multiple series of Andesitic to Felsic intrusive phases that precede and post-date the introduction of gold.

Smarts Deposit

Exploration has identified extensive gold mineralisation at the Smarts Deposit. Gold is hosted by a northwest trending, sub-vertical to steeply southwest dipping shear zone 2,800m in strike length and up to 60m wide. The shear zone has developed within Basalts and Andesites comprising the footwall Greenstone Succession along the northeast limb of a shallowly northwest plunging anticline. Auriferous mineralisation is also noted at

the contacts of porphyry-granite intrusives. The shear zone is comprised of semi-continuous zones of quartz lenses and quartz-carbonate veining or brecciation.

Numerous, moderately well-defined gold-rich lenses, up to 15m wide, occur within the shear zone and are characterized by anomalous quartz veining, quartz flooding, shearing, chloritization, seritisation and pyritisation (see Figure 7-8 and Figure 7-9). Visible gold and the majority of gold values typically occur within and along margins of quartz veins, in silicified Granitic Dykes, and in adjacent, pyritic, often sheared Meta-andesite. Pyrite is common at up to 3% by volume associated with auriferous quartz veins. Mineralisation is variously accompanied by silica-sericite-chlorite-carbonate-pyrite-tourmaline alteration. In general, the northwest trending mineralised shear zone has developed within a steeply inclined sequence of metamorphosed Andesitic to Basaltic Volcanics, adjacent to the northwest margin of the Eldorado (Granodiorite) Batholith. The volcanic stratigraphy is cut by a multiple series of Andesitic to Felsic intrusive phases that precede and post-date the introduction of gold.

Figure 7-8: Typical Smarts Mineralisation in Core (SDD025)

Figure 7-9: Typical Smarts Shear Zone in Core

Larken Zone

The Larken Zone is located 2km northwest of the Hicks Zone and 2km southeast of the Eldorado Target. These three mineralised occurrences essentially represent parallel zones arranged in an en echelon series across the stratigraphy.

In 1962, the Larken Vein was described as being 1.3m in width and 150m to 180m in strike length. At this time the vein was exposed in several short trenches, and panning of crushed vein material is reported to have revealed considerable gold, however no assays are recorded. Parts of the vein have been sporadically worked by artisanal means since 1962.

On 10th January 2013 Azimuth reported RC drilling results from the Larken Zone. These results included 12m at 4.8g/t gold from 25m (LRC007) and 5m at 2.5g/t gold from 91m (LRC009).

In July 2013 the Company completed a limited Diamond Core and Reverse Circulation drill program to test of the continuity of the mineralisation along the Larken Zone as defined by earlier auger gold-in soil sampling, trench channel sampling and wide spaced drilling. This target is located just north of the Smarts-Hicks Trend on a parallel northwest – southeast striking shear. The drilling confirmed continuity of mineralisation and although the best grades are associated with narrow intervals, additional drilling is warranted. Significant assay intervals are as follows (see Figure 7-10, Figure 7-11 and Table 7-1): 6m at 3.78g/t gold from 34m, including 2m at 9.69g/t gold from 36m; 4m at 1.59g/t gold from 49m; 1m at 12.18g/t gold from 39m; 1m at 7.45g/t gold from 103m; 1m at 3.59g/t gold from 76m and 1m at 2.30g/t gold from 67m. Further drilling is warranted.

Table 7-1: Larken Zone Diamond and Reverse Circulation Drilling Assay Results

Hole ID	Easting (m)	Northing (m)	Elevation (m	Depth (m)	Azimuth	Dip	Gold Intercept Interval (m at g/t gold)
LDD001	273522	620420	71	158.7	-60.7	215	**2m at 2.95g/t gold from 67m**
LDD002	273585.461	620399	73	148.9	-57	215	**1.0m at 7.45g/t gold from 103m**
LRC026	273677	620321	57	121.0	-58.6	215.2	**6.00m at 1.55g/t gold from 102.00m**
LRC028	273625	620328	64	79.0	-57.00	215.00	**1.0m at 12.18g/t gold from 39m 2m at 2.85g/t gold from 49m**
LRC029	273465	620452	64	91.0	-54.8	215	**6.00m at 3.78g/t gold from 34.00m**
LDD001	273522	620420	71	158.7	-60.7	215	**2m at 2.95g/t gold from 67m**
LDD002	273585	620399	73	148.9	-57	215	**1.0m at 7.45g/t gold from 103m**

Figure 7-10: Larken Zone View Looking North

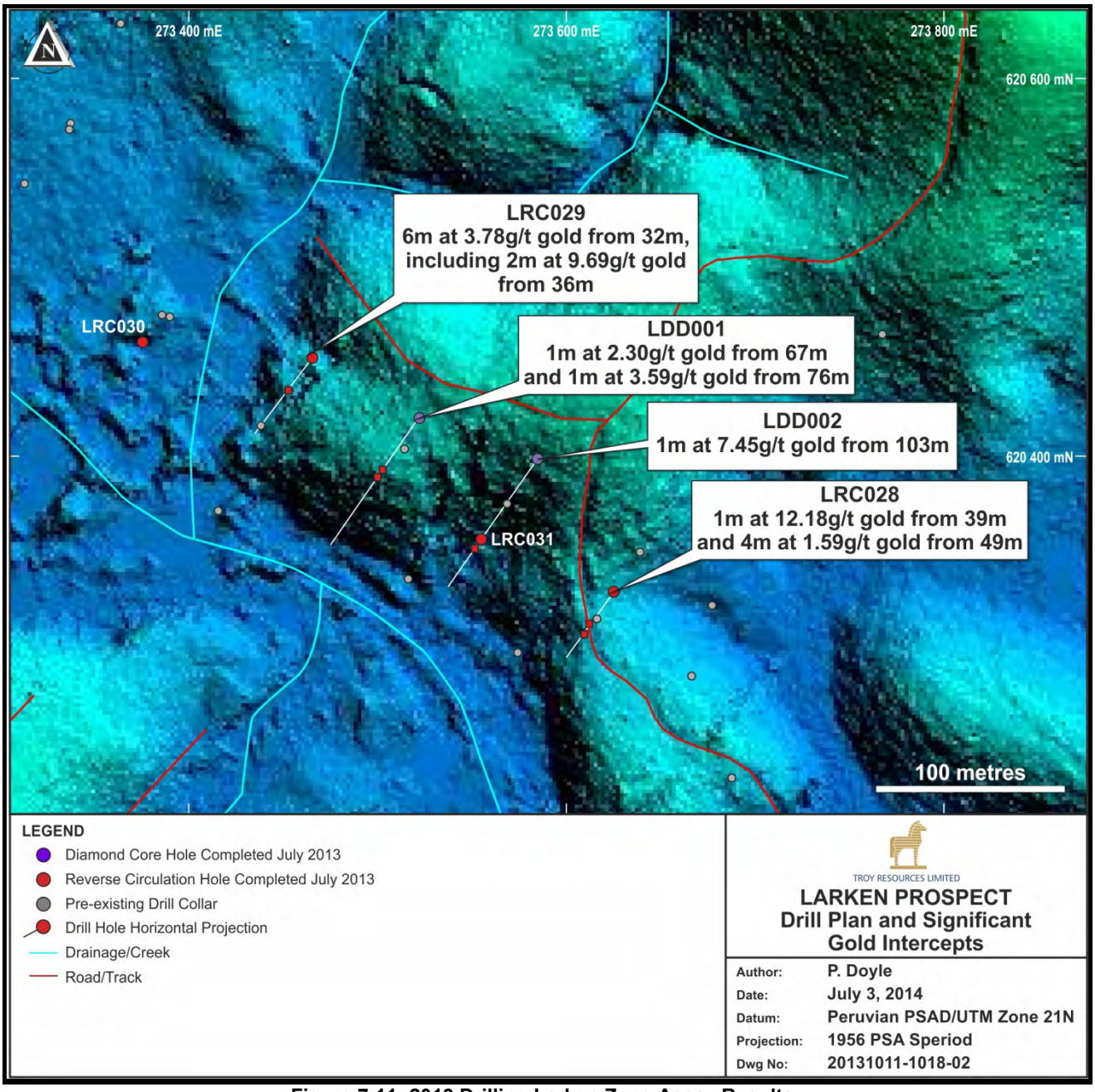

Figure 7-11: 2013 Drilling Larken Zone Assay Results

Whitehall (Kaburi) Target

The Whitehall (Kaburi) Target is located some 10km to the northwest of the Hicks Zone. The Main Vein consists of several in-situ sub-parallel stockworks and sheeted quartz vein zones exposed over a collective width of at least 200m and a strike length of at least 500m (see Figure 7-12). Gold mineralisation is associated with black, vitreous quartz exhibiting 'crack-seal' laminations and breccia textures reflecting episodic pulses of hydrothermal mineralisation. Individual zones range from 5m to 30m in width, are sub-vertical and strike northwest.

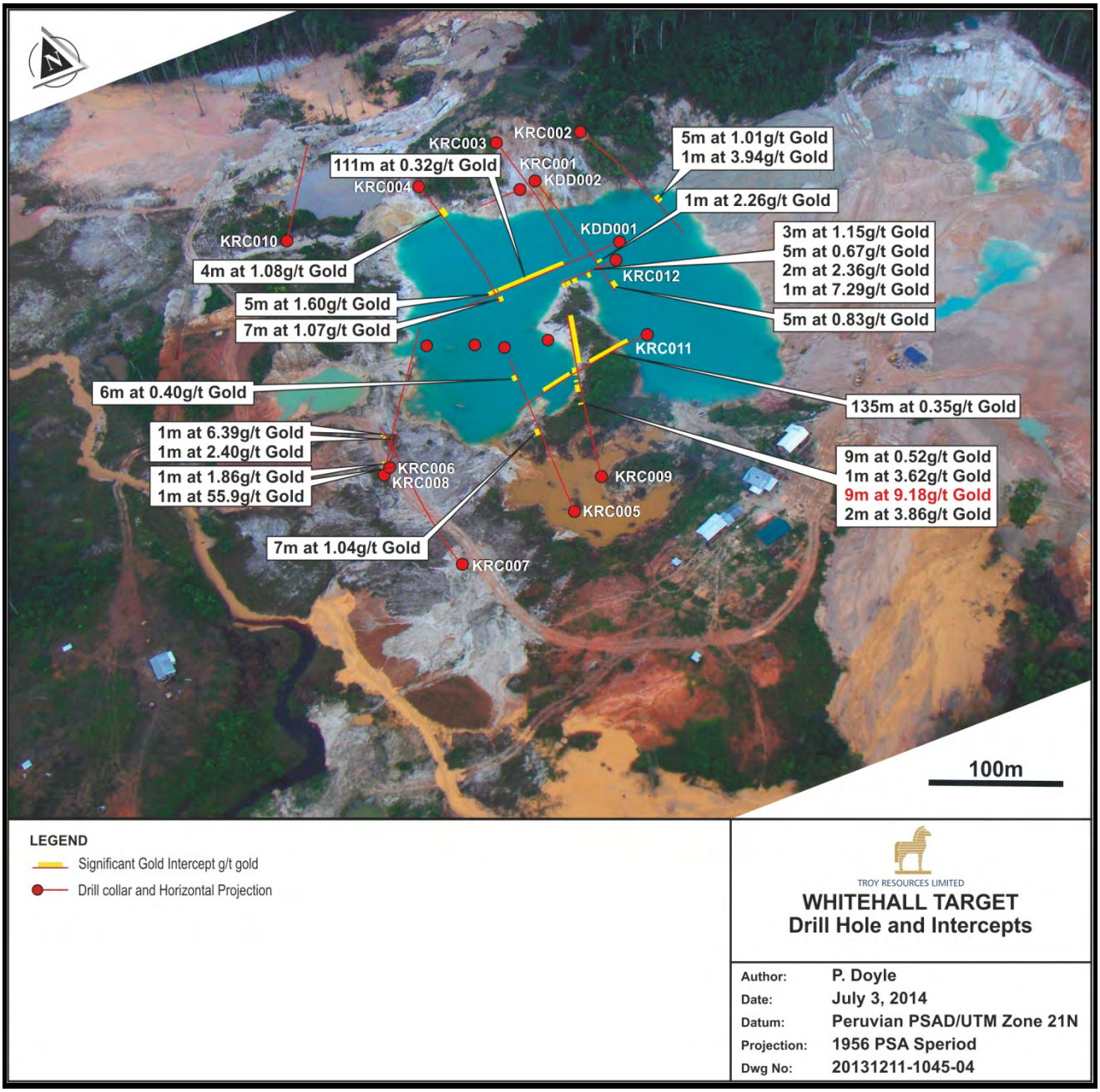

Figure 7-12: Karouni Project - Whitehall (Kaburi) Prospect Location Drilling Assay Intercepts

The host sequence is dominated by Intermediate Volcanics and Volcanoclastics intercalated with Siltstones, and Carbonaceous Shales (similar to the Eldorado Target), along with associated Quartz Feldspar Porphyry Dykes. Stockworks and sheeted vein zones appear to be preferentially developed within the more brittle units along the margins of the Carbonaceous Shale.

Saprolitic mineralisation has been worked by sluicing to a depth of approximately 10m to 15m (see Figure 7-13). Mineralisation remains open in all directions beneath White Sand Formation cover 5m to 20m in thickness.

In late 2011, Azimuth drilled 12 Reverse Circulation (RC) scout holes for 2,141m and 2 Diamond Drill holes for 349m to test potential mineralisation. As reported December 19, 2011, drilling intersected a small altered and mineralised Granitoid which intrudes Mafic-Intermediate Volcanics and Volcanoclastics. The drilling results suggest there maybe 3 three styles of mineralisation:

- Scattered narrow gold mineralisation associated with quartz veining in Mafic-Intermediate Volcanics and Volcanoclastics that are the host rock of the small Granitoid Intrusive exemplified by KRC005 which intersected 7m at 1.04g/t gold from 106m, KRC002 which intersected 5m at 1.1g/t gold from 148m and KRC008 which intersected 1m at 56g/t gold from 5m;

- Pervasive mineralisation within a small Granitoid Intrusive exemplified by KDD001 which intersected 111m at 0.32g/t gold from 69m and KRC011 which intersected 135m at 0.35g/t gold from 32m;

- Higher tenor mineralisation focused on the contacts of the Granitoid Intrusive such as KRC009 which intersected 9m at 9.2g/t gold from 118m (including 1m at 71g/t gold) and KRC012 which intersected 1m at 7.3g/t gold from 73m (with the hole ending in mineralisation).

Figure 7-13: Whitehall Target - Shallow Alluvial Mine Workings

Alluvial Deposits

Secondary gold mineralisation at Eldorado and Whitehall (Kaburi) Targets was mined from partially lithified, immature Quartz Conglomerates and Arkoses (presumed to be of Tertiary age), which discontinuously overlie mineralised portions of the basement. Alluvial deposits are also being actively mined along the Kuribong and Potaro Rivers (and tributaries) that comprise the southwest project area boundary, as shown in Figure 7-14.

Figure 7-14: Potaro River Alluvial Workings

8. Deposit Types

8.1 General

The Karouni Gold Project deposits are situated on the west-northwest trending Greenstone Belts of the Palaeoproterozoic Barama-Mazaruni Supergroup interspersed with extensive Granitoid Batholiths also of Trans-Amazonian age. Basement lithologies in the project area are partially obscured by a veneer of Tertiary to Holocene sedimentary cover.

Primary gold mineralisation is exposed at several localities within the Karouni Project, the most notable being the Hicks, Smarts and Larken Zones along the northern extremity of the project. Here the White Sand Formation cover has been removed by erosion to expose the underlying mineralised Palaeoproterozoic greenstone successions of the Trans-Amazonian Barama-Mazaruni Group.

8.2 Geology Model

Gold mineralisation within the Karouni Gold Project comprises mesothermal quartz, quartz-carbonate, quartz-tourmaline structurally controlled lodes, veins, stockworks and breccia zones variously hosted by Mafic to Intermediate Volcanics and Volcanoclastics, Carbonaceous Shales and syntectonic Granitoid Stocks. Gold is commonly accompanied by pyrite and less commonly arsenopyrite, in the latter instance particularly where hosted by Carbonaceous Shales and Phyllites. In essence, the style of mineralisation is entirely consistent with similar gold-bearing Archaean and Palaeoproterozoic greenstone terrains world-wide. Gold workings associated with recent alluvial deposits are notable along virtually all streams and rivers throughout the project area, where the barren White Sand Formation cover has been removed by erosion.

8.3 Exploration Model

The Company is primarily targeting large tonnage, medium to high grade gold deposits along north-west south-east trending mineralised shear zone within Omai – Hicks – Smarts – Whitehall (Kaburi) Corridor (see Figure 7-6). Most exploration undertaken to date has been on prospects that have had either historic exploration or recent alluvial mining.

The Company has an exploration strategy which will test the mineralised shear zone through regional trenching, auger drilling, geophysical and LiDAR surveys through to extensional and infill Reverse Circulation (RC) or deeper diamond drilling.

This exploration strategy has resulted in the definition of gold resources at the Hicks Zone and Smarts Zone where RC and diamond core drilling by Troy (and previous explorers at Hicks) has outlined zones of semi-continuous gold mineralisation over a strike length of 2,900m and 2,700m respectively.

Troy is continuing its exploration program, targeting resource extensions, the Larken Zone and other prospective targets within the Karouni tenements.

8.4 Hicks

The Hicks Deposit is the southernmost deposit defined to date along Omai – Hicks – Smarts – Whitehall (Kaburi) Corridor. Gold is hosted by a northwest trending, sub-vertical to steeply southwest dipping shear zone some 2,900m in strike length and up to 60m wide in places with average grade mineralisation of approximately 2g/t gold.

8.5 Smarts

The Smarts Deposit was discovered in January 2011 when Azimuth used RC drilling to follow-up earlier encouraging trench results around the local workings known as ‘Smarts Pit’. The prospect is located 4km to the northwest of the Hicks Zone. Gold is hosted by a northwest trending, sub-vertical to steeply southwest dipping shear zone, that has been observed in drill holes for some 2,800m of strike length and up to 200m wide in places with average grade mineralisation of approximately 4g/t gold - 5g/t gold.

The nature of mineralisation has been identified as having the following key features;

- Gold mineralisation occurs within the Smarts Shear, and also in the footwall and hangingwall.
- Gold mineralisation also occurs in the hangingwall and footwall of the porphyritic dacite marker unit.

- There is also evidence for an early low grade gold phase associated with silicification, hematite, and pyrite.
- A late-stage dominant gold phase with pyrite, pyrrhotite, chlorite, quartz and carbonate. This has the following styles;
    - Laminated veins within the Smarts Shear,
    - Disseminated sulphides within reactive host rocks,
    - Foliation parallel veins,
    - Stratigraphically controlled visible gold-bearing vein arrays within finer grained volcanoclastic sediments that cross cut bedding and foliation and,
    - Small scale discrete east-dipping reverse shears with ultra-high grade gold (with visible gold, pyrite, carbonate and possible tellurides).
- The most economically significant laminated veins occur within the Smarts Shear. The highest grade zones of laminated veins within the shear occur at a region where there is a dip change on the Smarts Shear beneath a lithic tuff unit. Areas of the Smarts Shear with the massive mafic unit in the footwall or hangingwall also correlated with elevated gold grades.
- The highest gold grades within veins outside the Smarts Shear are associated with veins that transect the dominant foliation and bedding. These styles of veins are commonly associated with the finer grained unit within the larger package defined as coarse and fine grained volcanoclastic sediments. This style of veining is interpreted to be stratabound (i.e. it dominantly occurs within this host rock).
- A major set of steeply northwest-dipping veins that transect bedding and foliation occur in the footwall of the Smarts Shear on section 2850 (this vein set has underground potential). This vein set is hosted within a stratigraphic lens of fine grained sediments in the footwall of the Smarts Shear that does not exist on the other logged section. The significance is that major laterally continuous vein arrays in this unit may abruptly stop along strike if the critical stratigraphic unit lenses out. Likewise there is the potential for an entirely new vein array within a stratigraphic lens of this stratigraphy along strike from a non-mineralised section that lacks the critical host rock. More isolated lenses of this key unit are likely to occur in the footwall of the Smarts Shear within the foliated footwall volcanoclastic unit.
- The foliated grey volcanoclastic unit is a very poor host rock. The lithic tuff and porphyritic rhyodacite also appear to be poor host rock types for gold. Significant gold mineralisation occurs in all other stratigraphic units.

Host rocks play an important role in the gold deposition at Smarts, Hicks and Larken. Key stratigraphic units identified in core logging to date include (see Figure 8-1, Figure 8-2 and Figure 8-3)

Porphyritic Rhyodacite (Marker Unit): This unit has clear plagioclase crystals and is an excellent marker bed. This unit has an interval ranging from 0.4m to 3.2mwith most intervals measuring approximately 1.5m. Early in the project's history the term "Rhyodacite" was the most common classification used by the geologists on site, but has not been confirmed by petrography or geochemistry. The unit is conformable with bedding (and may not be extrusive or intrusive in nature).

Interlayered Fine and Coarse Grained Volcanoclastic Sediments: These units have clearly defined bedding with bedding thicknesses ranging from 1cm to 30cm. These units are normally light green to green in colour. The coarser grained units commonly have chlorite and disseminated pyrite in mineralised zones, the finer grained units are commonly silicified and host vein arrays. This unit contains the Porphyritic Rhyodacite marker unit. The definition of "coarse grained" and "fine grained" reflects the current observable texture and probably reflects modification via metamorphism and alteration, and not the primary state. There are major variations in the distribution of the fine and coarse grained units, in some areas the fine grained unit dominates. In the logging program it was not possible to correlate these variations from drill hole to drill hole and so they have been grouped into one unit (the significance is that the areas dominated by the finer grained unit commonly have mineralised vein arrays).

Massive Mafic Unit (Foliated and Unfoliated): individual examples of this unit commonly have a thickness of 5m to 10m. Bedding cannot be readily identified within this unit (and at the Hicks Deposit this unit can have blue quartz). The interpretation is that this unit is a basalt or a mafic intrusive. In some areas this unit is foliated, in other areas it isn't. There is also a range of alteration over printing it with examples of silicification, chloritisation and pyritisation. When strongly silicified it is hard to definitively ascertain if the rock was derived from a mafic protolith (potential for misidentification as a mafic unit in some of the logging).

Lithic Tuff (Commonly Silicified): This unit is strongly silicified tuff with clearly defined lithic fragments, the term is also the most common classification used by the geologists on site. In some areas this unit is strongly foliated.

Foliated Footwall Volcanoclastic Unit (Extensive Carbonate): This is the dominant footwall sequence east of the Smarts Shear. The unit is strongly foliated and can in places be difficult to distinguish from the Lithic Tuff and the foliated grey Volcanoclastic unit. The distinguishing feature of this unit is ubiquitous distinctive carbonate lenses and layers – the lenses are commonly 1cm to 10cm in length. The layers also extend across the entire length of core. In some areas the foliated footwall volcanoclastic unit is in conformable contact with the finer grained sequences of the interlayered fine and coarse grained volcanoclastic sediments.

Foliated Grey Volcanoclastic Unit (Coarse Carbonate Porphyroblasts): This unit is grey to dark grey in colour and is a strongly foliated unit with distinctive carbonate porphyroblasts that can be elongate within the foliation plane.

Sulphide-Rich Sediments: Small zones (less than 1m thick) of thinly bedded sulphidic sediments were identified intermittently but could not be correlated between drill holes.

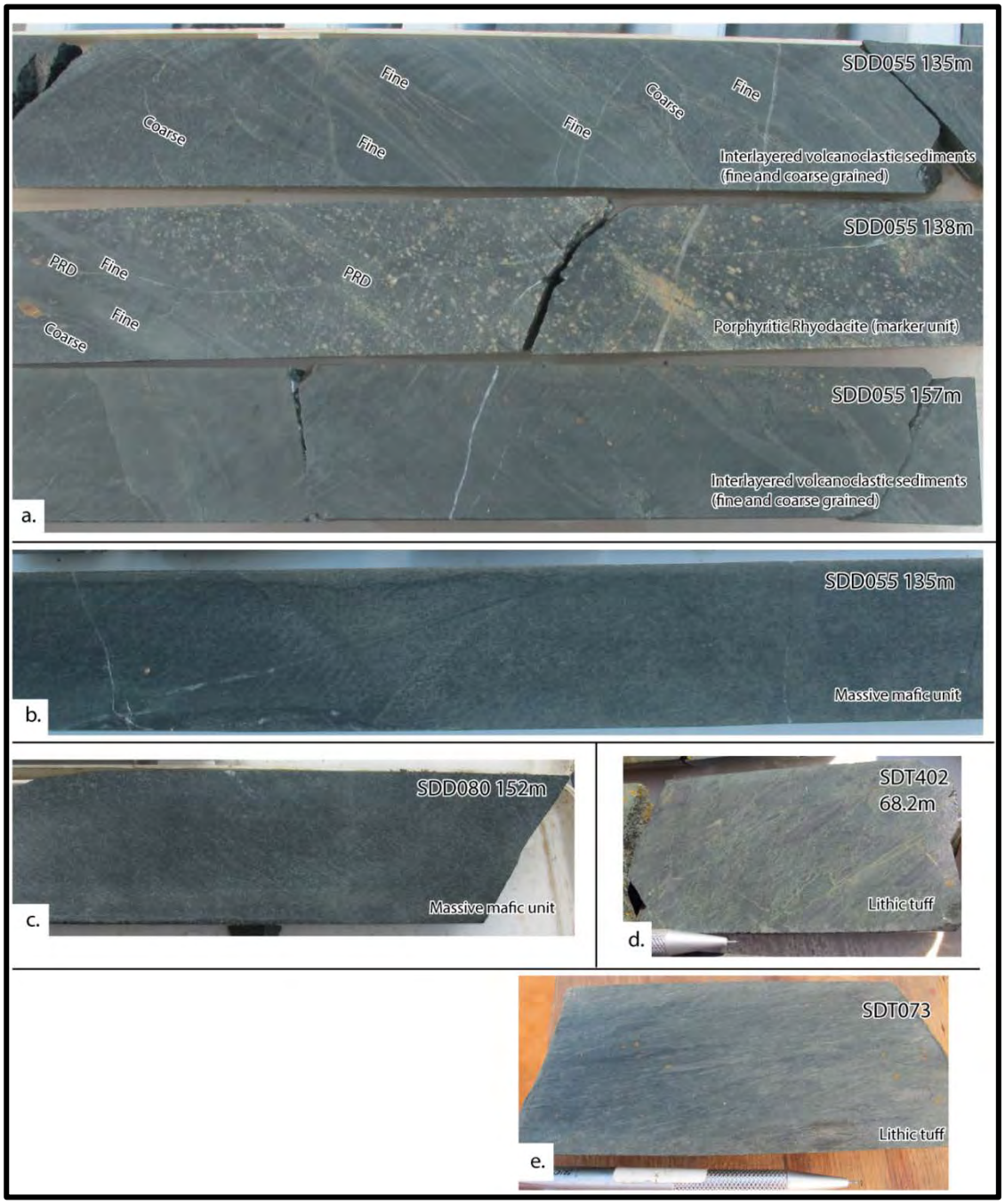

Figure 8-1: Smarts Zone Key Lithologies

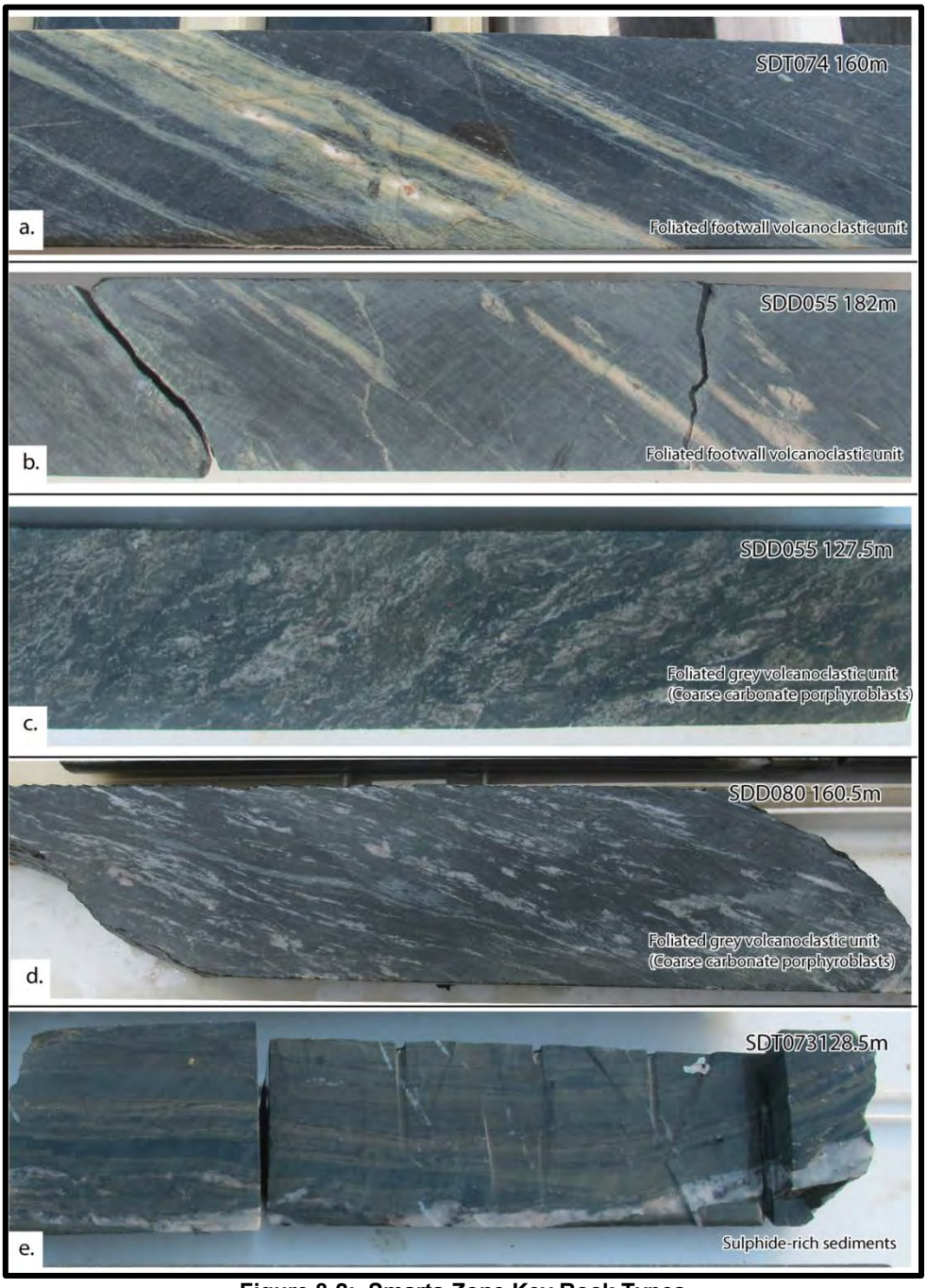

Figure 8-2: Smarts Zone Key Rock Types

There are major lateral facies variations in stratigraphy both within a section and also between sections. Examples are the variation in the coarse and fine grained Volcanoclastic Sediments and also the Lithic Tuff. There is a major lens of the coarse and fine grained Volcanoclastic Sediments in the footwall of the Smarts Shear that up section grades into the foliated footwall Volcanoclastic sequence. Drillhole SDD034 is located in the transitional zone and both of these units occur and are conformably interlayered, this appears to mark a transitional zone between the two units. Only one major lens of the massive Mafic Unit was defined on Section 2850, but multiple lenses were defined on Section 2380. The foliated grey Volcanoclastic Unit was most commonly associated with the massive Mafic Unit.

There are major internal variations in the distribution of the fine and coarse grained sequences in the unit defined as “Interlayered fine and coarse grained Volcanoclastic Sediments”, in some areas the fine grained unit dominates.

The Porphyritic Dacite unit was utilised as a marker horizon in the drillcore and could be correlated across multiple drillholes. The correlation of this unit across multiple drillholes indicates the stratigraphy is steeply west dipping. This Porphyritic Dacite marker unit is conformable with bedding in drillcore.

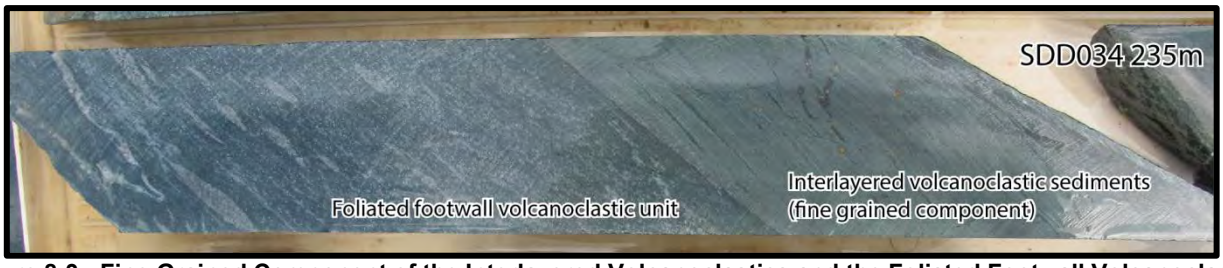

Figure 8-3: Fine Grained Component of the Interlayered Volcanoclastics and the Foliated Footwall Volcanoclastic Unit in the Transitional Zone between the Two Units

Structural Features

A dominantly steeply west dipping foliation observed throughout the deposit. A late crenulation cleavage was identified in some areas. In drillcore foliation and stratigraphic contacts are in most cases close to parallel (due to the stratigraphy being flatted during foliation development). However, logging of the stratigraphic units highlights that these packages must be transected by the foliation in areas due to the major lateral facies variations along strike and in section.

The major distinctive structural feature within the deposit is the Smarts Shear (see Figure 8-4, Figure 8-5, Figure 8-6 and Figure 8-7). The Smarts Shear is dominantly steeply west-dipping, but can dip to the east in some areas. This dip change is most notable on Section 2380 and appears to correlate with the appearance of the Lithic Tuff on the eastern side of the shear. In core the Smarts Shear is a strongly foliated zone, with foliation parallel carbonate and quartz veins. The shear is 50cm to 2m wide, and in some areas it has smaller subsidiary shears. Late stage brittle brecciation is common. There are major differences in the amount of veining within the Smarts Shear. In some areas there is no major veining, in other areas there are laminated quartz veins with high gold values. Areas with no major veins within the shear commonly correlate with areas the foliated footwall Volcanoclastic Unit is in fault contact with the foliated grey Volcanoclastic Unit in the hangingwall. Wide zones of quartz veining (up to 4m) within the Smarts Shear occur where the shear dips back to east. Veins also develop adjacent to the massive mafic units, and adjacent to the interlayered coarse and fine grained volcanoclastic sediments. The geometry suggests the veining is related to dilation on the lower angle segment.

On section 2850 there is a repetition of the Porphyritic Dacite Marker unit and the interlayered fine and coarse grained Volcanoclastic Sediments across the Smarts Shear implying a major component of reverse movement. The repeated massive mafic units on Section 2380 may reflect fault imbrication.

There are multiple sets of veins parallel and oblique to the dominate foliation and stratigraphic contacts. Extensive boudinage of veins parallel to the foliation with the development of internal tension veins implies a component of vertical stretch. This is consistent with reverse movement and imbrication of stratigraphy across the Smarts Shear.

There is a clear host rock control on the development of vein arrays that are oblique to the dominant steeply west-dipping foliation. The finer grained silicified units within the Volcanoclastic Sediments are the most common host rock. In some cases these veins terminate against the coarser units. This indicates some of the veins are tensional in nature and are not shear veins – this is relevant for both mineralised and non-mineralised veins. Many of the veins are foliation parallel/ steep dipping and some have 30 degrees dips due to making a 90 degree core axis angle).

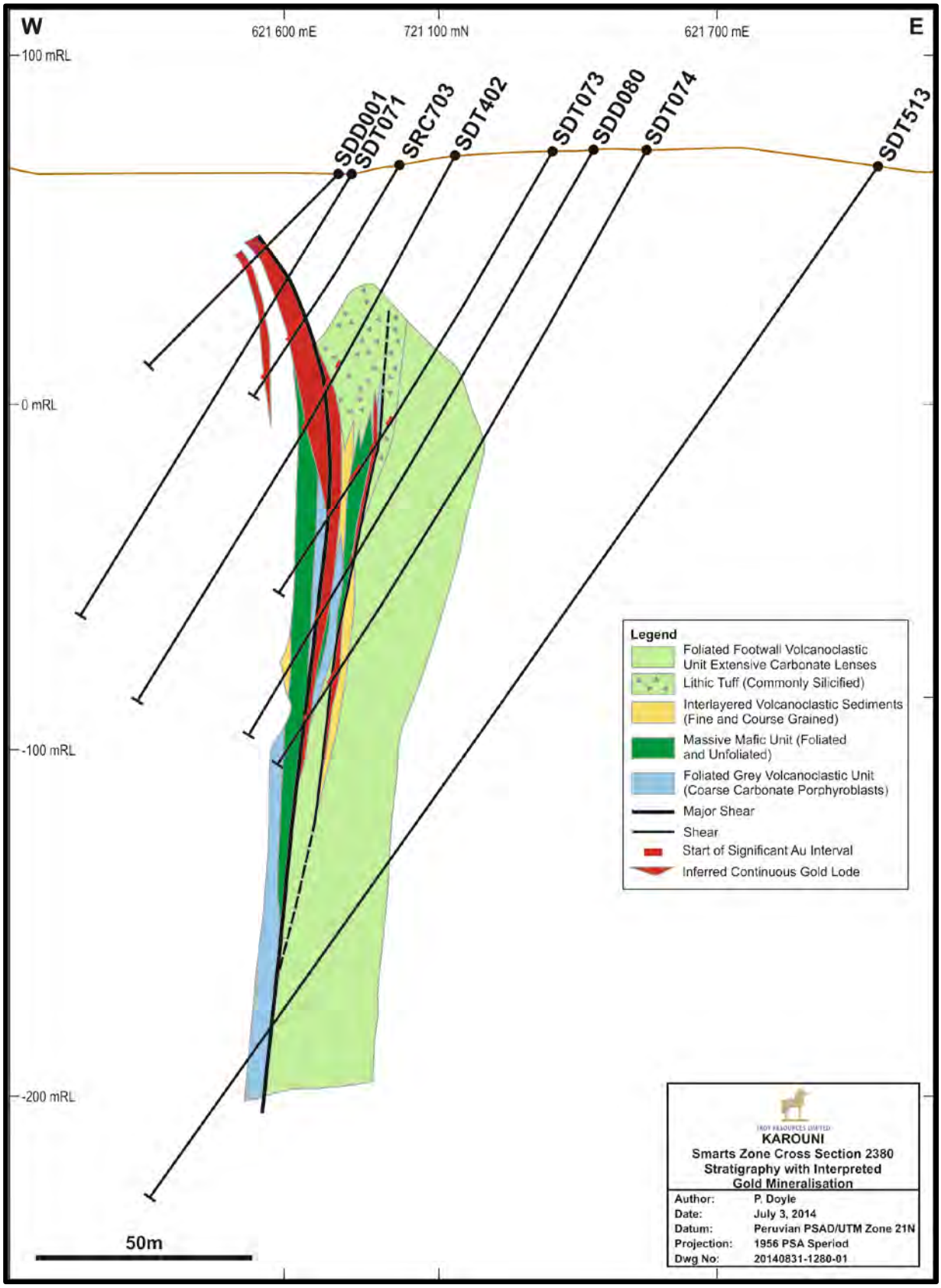

Figure 8-4: Smarts Cross-Section 2380 Stratigraphy with Interpreted Gold Mineralisation

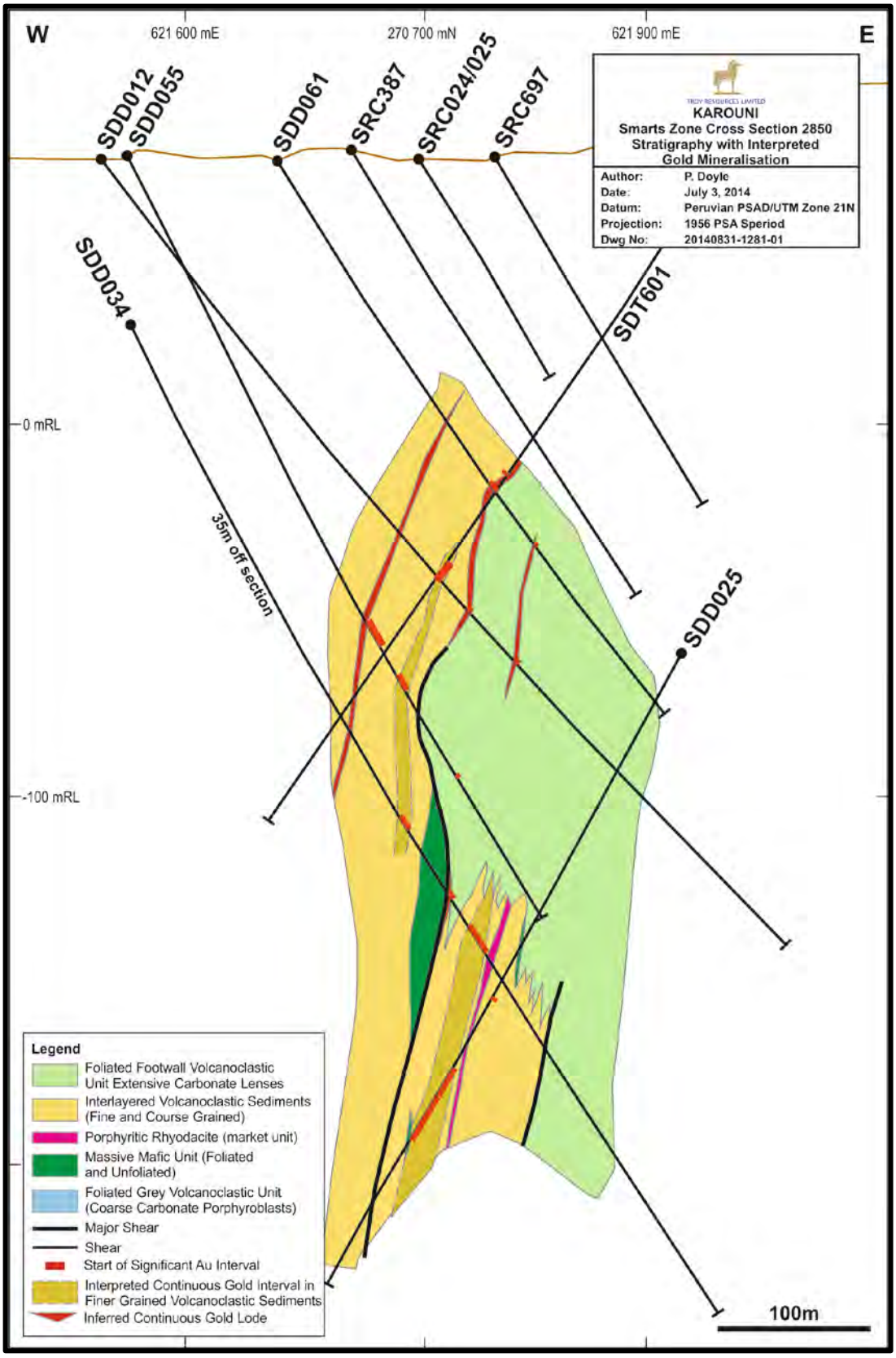

Figure 8-5: Smarts Cross-Section 2850 with Interpreted Mineralisation

Figure 8-6: Intersection through the Smarts Shear with no major Quartz Veining (SDT513)

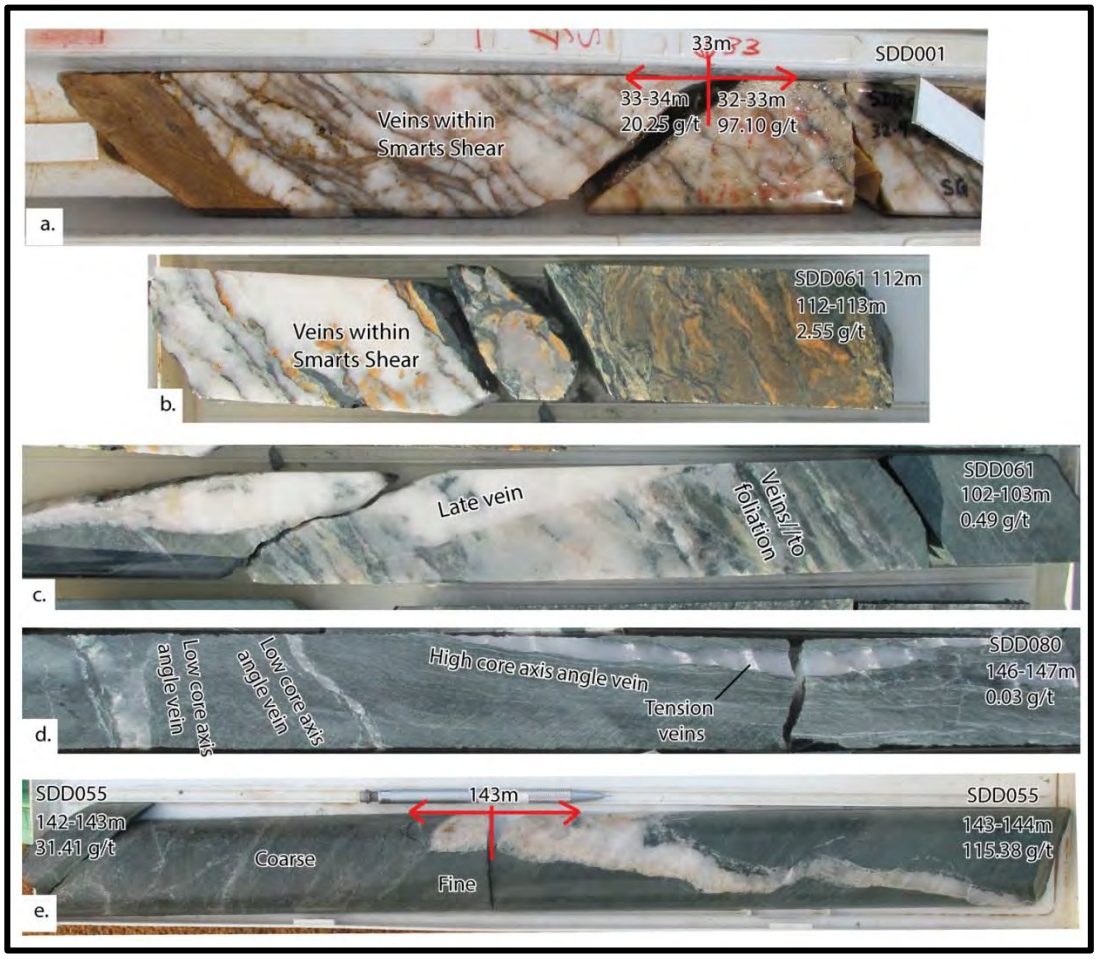

Figure 8-7: Quartz Veining within the Smarts Deposit

Gold Mineralisation

The nature of mineralisation has been identified as having the following key features;

- An inferred low grade early gold phase with hematite, pyrite with extensive silicification.
- A late-stage dominant gold phase with dominant pyrite, pyrrhotite, chlorite, quartz and carbonate assemblages. This has the following styles;
  - Laminated veins within the Smarts Shear;,
  - Disseminated sulphides within reactive host rocks;
  - Foliation parallel veins;
  - Stratigraphically controlled visible gold-bearing vein arrays within finer grained Volcanoclastic Sediments small scale discrete reverse shears with visible gold, pyrite.
- Gold mineralisation occurs within the Smarts Shear, and also in the footwall and hangingwall.
- Significant gold mineralisation also occurs in the hangingwall and footwall of the Porphyritic Dacite Marker Unit (this unit itself was not a prospective host rock).
- The foliated grey Volcanoclastic unit is a very poor host rock type in all intervals logged.
- The Lithic Tuff also appeared to be a poor host.
- Significant gold mineralisation was logged in all other stratigraphic units.

The above relationships have been used to correlate gold mineralisation identified in the drilling which is detailed below; see Figure 8-8 and Figure 8-9.

Early Hematite Associated Veins: Low angle vein arrays associated with hematite alteration were observed in several areas. These veins are associated with extensive silicification and low gold grades were observed in all areas they could be positively identified.

Laminated Veins: Laminated veins were observed within ductile shears in several locations, but the most economically significant occur within the Smarts Shear. The highest grade zones of laminated veins within the shear are at the region where there is a dip change in the Smarts Shear beneath the Lithic Tuff. This geometry of veining on a lower angle segment suggests the lodes were linked to reverse movement with the formation of dilational jogs. At deeper levels, in areas with a steeply west-dipping geometry, lower grades occur in the current drilling data. Apart from the dip of the shear there also appears to be a host rock control on the formation of mineralised veins within the Smarts Shear. Areas of the Smarts Shear with the massive Mafic Unit in the footwall or hangingwall correlated with elevated gold grades. Areas with no major veins within the shear commonly correlated with areas the foliated footwall Volcanoclastic Unit was in fault contact with the foliated grey Volcanoclastic Unit in the hangingwall.

Disseminated Sulphides within Reactive Host Rocks: There is extensive evidence for sulphidation of the wallrock, particularly by pyrite. The extent of gold mineralisation associated with this is hard to estimate due to the dominance of high grade gold in quartz veins. The presence of higher gold grades within the Smarts Shear adjacent to the massive Mafic Unit is inferred to reflect it is a reactive host rock with potential for laterally continuous mineralisation. There are host rock variations linked to the amount of sulphidation i.e. the foliated Mafic Unit versus the Tuff or the fine versus the coarse grained units in the Volcanoclastic Sediments. There is also a foliation control on the sulphidation with evidence for sulphidation along the foliation. This most likely reflects permeability and fluid flow along the foliation. The grey foliated Volcanogenic Unit with overgrowth of carbonate porphyroblasts is a very poor host rock with little to no sulphidation observed.

Stratigraphically Controlled Visible Gold-Bearing Vein Arrays and Veins Parallel to Foliation: In drill core there are major variations in the orientations of the mineralised veins. A set of mineralised quartz-carbonate veins have developed sub-parallel to, or close to, the foliation. These veins are associated with carbonate, chlorite and pyrite. In general these veins have lower grades in comparison to veins that transect the foliation. The highest gold grades appear to be associated with veins that transect the foliation and bedding. These styles of veins were most commonly associated with the finer grained unit within the coarse and fine grained Volcanoclastic Sediments; this unit is commonly highly silicified. This style of veining is interpreted to be stratabound in that it dominantly occurs within this host rock.

The orientations of the veins that transect bedding within the Fine Grained Unit vary. Furthermore, in some cases high grade gold intercepts of these vein types could be correlated between drillholes, but in other cases they could not. Some examples of these are as follows;

- A set of high grade gold bearing veins (115 g/t) within the hangingwall of the Smarts Shear that occur within the Fine Grained Unit. Some of these dip at about 60 degrees to the east and are offset by small reverse shears. This set of veins could not be correlated updip or downdip (even though the stratigraphy could).
- A set of steeply northwest-dipping veins occurring in the footwall of the Smarts Shear. These veins have consistent core axis angles in both of the drill holes they were intersected in, which are interpreted to mean that they are a vein array with a similar trend. In plan view the veins have a north-easterly strike (assuming the foliation is north-trending in mine grid). The veins do not project up into the foliated footwall Volcanoclastic Unit. This vein set is hosted within a stratigraphic lens in the footwall of the Smarts Shear. The significance of this is that major correlatable lateral vein arrays in this unit may abruptly stop along strike if the stratigraphy lenses out. Likewise there is the potential for an entirely new vein array within a stratigraphic lens of this stratigraphy along strike from a non-mineralised section that lacks the critical host. There is major potential for more of these vein arrays within the Deposit.
- An inferred lens of correlated mineralisation in the footwall of the Smarts Shear has been interpreted on section 2850 between SDD012 and SDD061). The interpretation is that the mineralisation is controlled by a stratigraphic variation in the footwall Volcanoclastic Unit (in this case the likely presence of a fine grained unit intercalated within the foliated footwall Volcanoclastic Unit).
- An inferred lens of correlated mineralisation in the hangingwall of the Smarts Shear has been interpreted on section 2850 between SDD061, SDT601, SDD055, and SDD034). The interpretation is that the mineralisation is controlled by the presence of a laterally continuous Fine Grained Unit within the interlayered fine and coarse grained Volcanoclastic Sediments. This interpreted stratabound zone of mineralisation is parallel to the Porphyritic Rhyodacite Marker Bed.

Discrete Reverse Shears with High Grade Gold: Narrow discrete steeply east-dipping shears exist in the foliated footwall Volcanoclastic sequence. These contain visible gold within dilation jogs and also pyrite, carbonate, and possible tellurides with strong hydrothermal alteration of the wallrock. These shears also provide evidence that the mineralisation was associated with reverse movement on the shears. These high grade structures cannot currently be correlated between drill holes or sections.

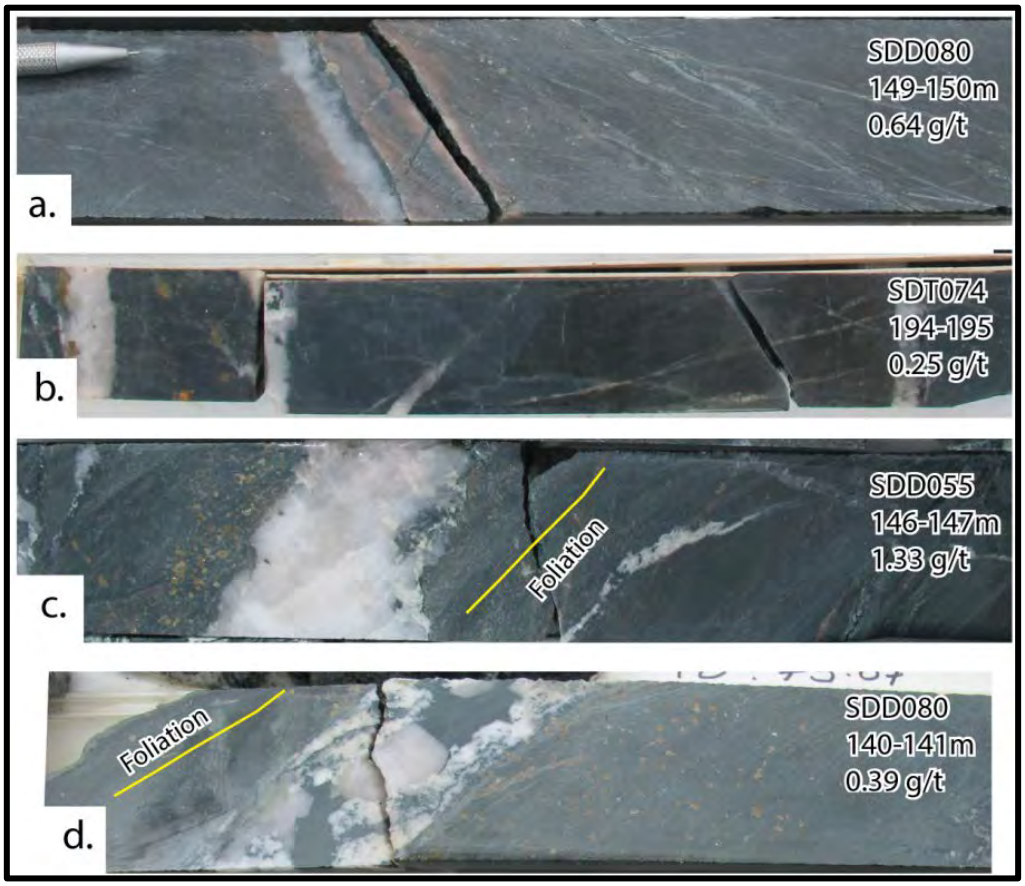

Figure 8-8: Smarts Gold Bearing Quartz Veining Parallel to the Foliation

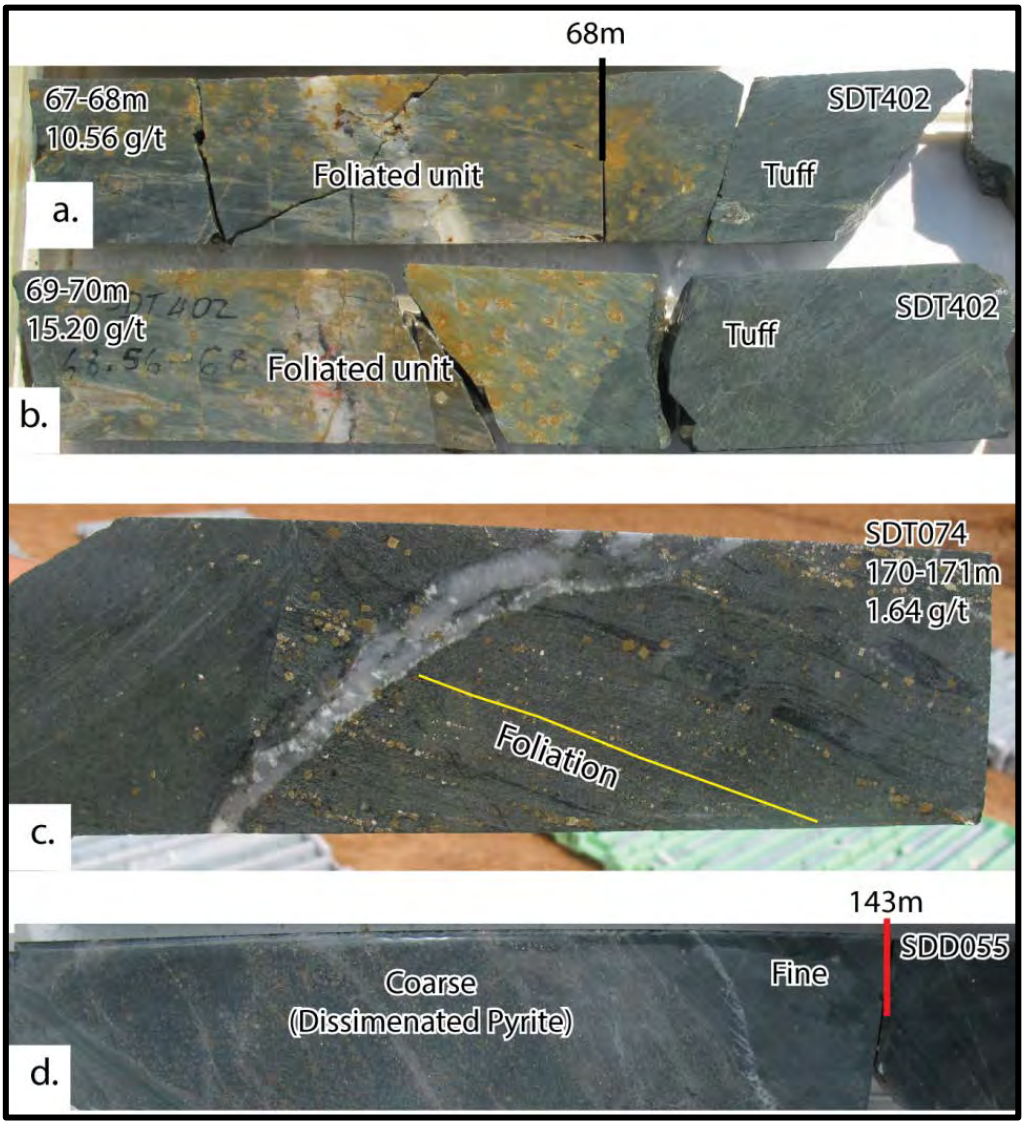

Figure 8-9: Disseminated Sulphides within the Smarts Deposit
Coarse = Coarse Grained Unit within Interlayered Volcanoclastic Sequence
Fine = Fine Grained Unit within Interlayered Volcanoclastic Sequence

Structural Model:
The following observations were used to constrain a simple and preliminary kinematic/ structural model for the Smarts Deposit;

- The mineralisation is late-stage (the mineralised veins are not obviously folded or flattened by the foliation development). This event is interpreted to have reactivated an existing steeply dipping set of shears and stratigraphic units.
- Evidence for larger laminated veins on lower angle segments of the Smarts Shear implies the formation of dilational jogs via reverse movement.
- There are isolated steep east-dipping high grade reverse shears with possible tellurides.
- The presence of mineralised low angle veins is indicative of compressional deformation (normally associated with reverse movement).
- The presence of stratabound vein arrays controlled by the fine grained volcanoclastic sediments.
- Gold associated with sulphidation of wall rocks (e.g. the massive mafic unit).
- The lithic tuff correlates with the dip change on the Smarts Shear – this either may reflect it being a control on the orientation of the shear. It may also have acted as a seal to mineralising fluids.
- A component of oblique movement cannot be excluded.

The current model (see Figure 8-10) is that shortening during the main stage of gold mineralisation resulted in the reactivation of pre-exising steeply dipping shears in a dominantly reverse manner. The mineralisation is

complicated by marked variations in host rocks both up and down dip, and along strike. Lower angle segments of the Smarts Shear developed major dilational jogs, with a possible hydrothermal fluid seal provided by the over-lying lithic tuffs. Discrete east-dipping shears developed in the footwall and could merge with the main Smarts Shear (these contain ultra-high grade gold with possible tellurides). Gold is associated with the massive mafic unit adjacent to the Smarts Shear, and also within stratabound vein arrays within fine grained sedimnetary packages that reflect low strain shearing of these units (in both the footwall and hangingwall of the Smarts Shear).

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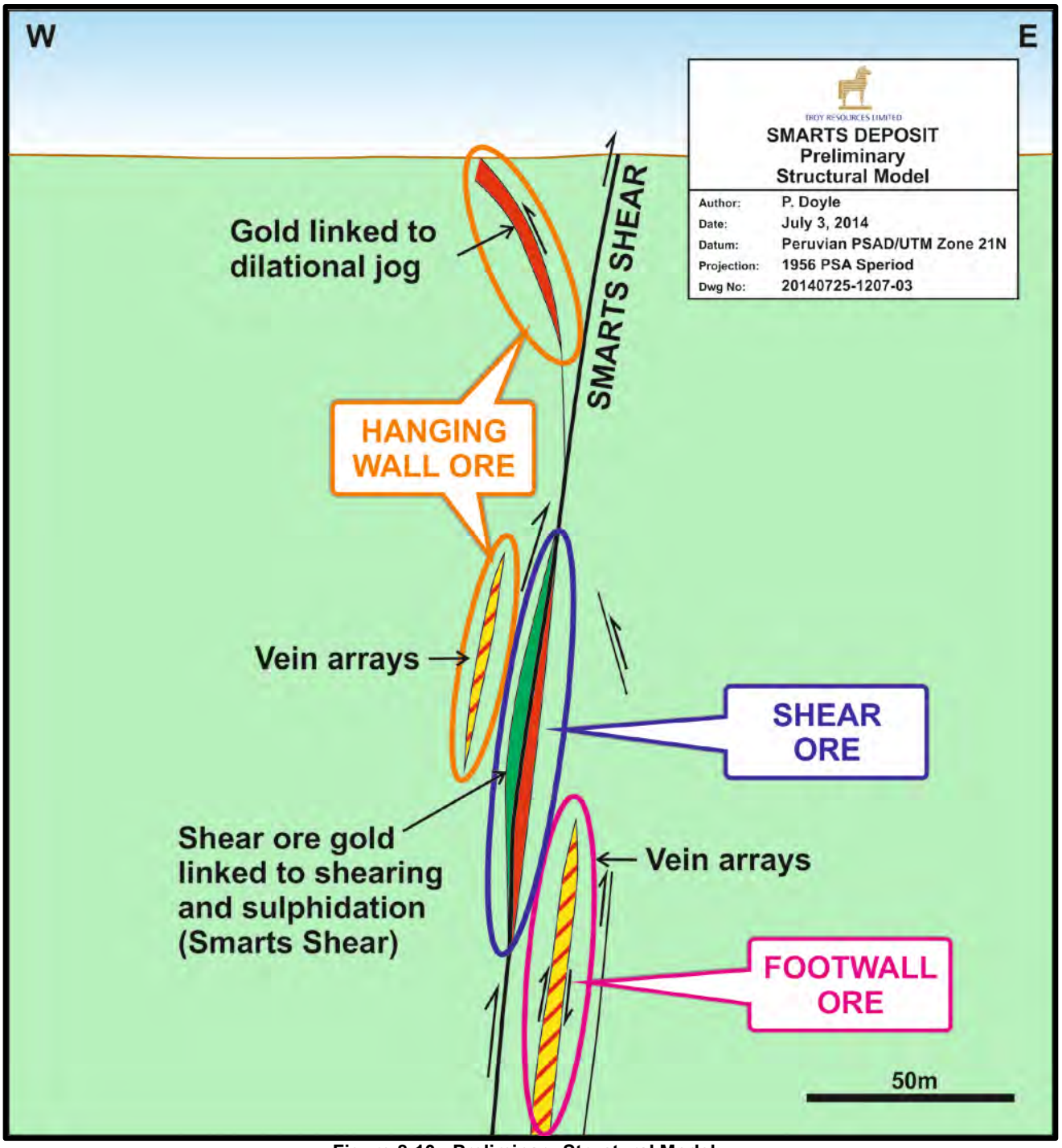

Figure 8-10: Preliminary Structural Model

9. Exploration

9.1 Introduction

Exploration work between March and July 2013 documented in this report was managed by Azimuth and conducted using funding provided by Troy with technical input and oversight by Troy management. Troy acquired Azimuth in late July 2013 and assumed control of all in-country field based and support /logistics activities.

The majority of exploration work undertaken by Troy has consisted of Reverse Circulation (RC) and Diamond Core drilling of the Smarts, Hicks and Larken Zones which is described in Section 10 below. Earlier Azimuth managed drilling occurred within the Karouni Gold Project and included Whitehall (Kaburi), Larken and other prospects. Other work includes regional field reconnaissance, airborne geophysical surveys, geological mapping, prospecting, rock chip sampling, trenching, channel sampling.

Geobase Australia Pty Ltd (Geobase) was commissioned by Azimuth to compile, validate and manage the historical exploration dataset for the Karouni Gold Project. Information on historical exploration is outlined separately within Section 6. The Karouni Gold Project was the main exploration focus for Azimuth exploration and the initial scope of work included:-

- Initial compilation of available current and historic exploration data (drilling, trenching and surface geochemical sampling).
- Establishing an auditable and robust Exploration Data Management System (EDMS).
- Validation of critical historical exploration data.

These objectives were met and Geobase and the historic data is now part of the active project database for the Karouni Gold Project.

Since July 2013, Troy has managed exploration of the Karouni Gold Project with work consisting of Diamond Core and Reverse Circulation Resource infill drilling of the Smarts Deposit, Hicks Deposit and the Larken Zone. The Karouni Gold Project database has been maintained by Troy personnel on site at Hicks Camp with back-ups in Georgetown and Perth offices.

9.2 Karouni Survey Grids

Troy has reviewed the historical local grids in use within the Project and assigned database codes as listed below.

- BKLG - Black Local Grid.
- ELLG - Eldorado Local Grid.
- HILG - Hicks Local Grid.
- KELG - Kaburi East Local Grid.
- LALG - Larken Local Grid.
- NMLG - Nine Mile Local Grid.
- RALG - Ravines Local Grid.

At the time of initial drill database compilation in November 2011, an interim 2 point local grid transformation to the PSAD56_21N grid system was created where possible. Differential GPS surveys at the Hicks Prospect during 2011 have located several historical holes with a corresponding local grid coordinate. A review of these drillholes in relation to the standardised grid coordinates still gives some inconsistency although it is thought historical drillholes have been determined to within +/-5m of actual location on the PSAD56_21N grid system. The database retains the original local grid survey for historical drillholes and also the DGPS survey pickup (for located holes only), until further survey analysis provides a more refined grid transformation. More recent drillholes retain transformation coordinates based on the interim 2 point transformation between the local grid and PSAD56_21N grid system.

Within the historical data there is very little Meta data available for down-hole survey control. Meta data includes such information as the grid system of survey, type of survey, instrument used and dates. If available this data has been added to the database and any unknown details flagged accordingly. The DGPS collar survey, grid transformation and survey analysis was carried out by Troy. Troy is responsible for validating the data entered in the database.

Troy will extend the existent Ground Magnetics Survey to NW and to SE existent survey that covers Hicks area.

The NW part of the survey will cover 10,000m of the Smarts-Hicks Corridor and parallel mineralised trends which include Smarts, West Powers, Norby, NW Smarts, Gibbs, Whitehall and El Paso Targets, as shown on Figure 9-1.

The Ground Magnetic Survey totals 375km of survey lines spaced at 100m and 10m station intervals. 310km of lines of the first grid is opened.

Troy is also contracting IP/Resistivity Survey, Dipole-Dipole Array to be carried out firstly on orientation lines over known deposits to be further applied on the targets.

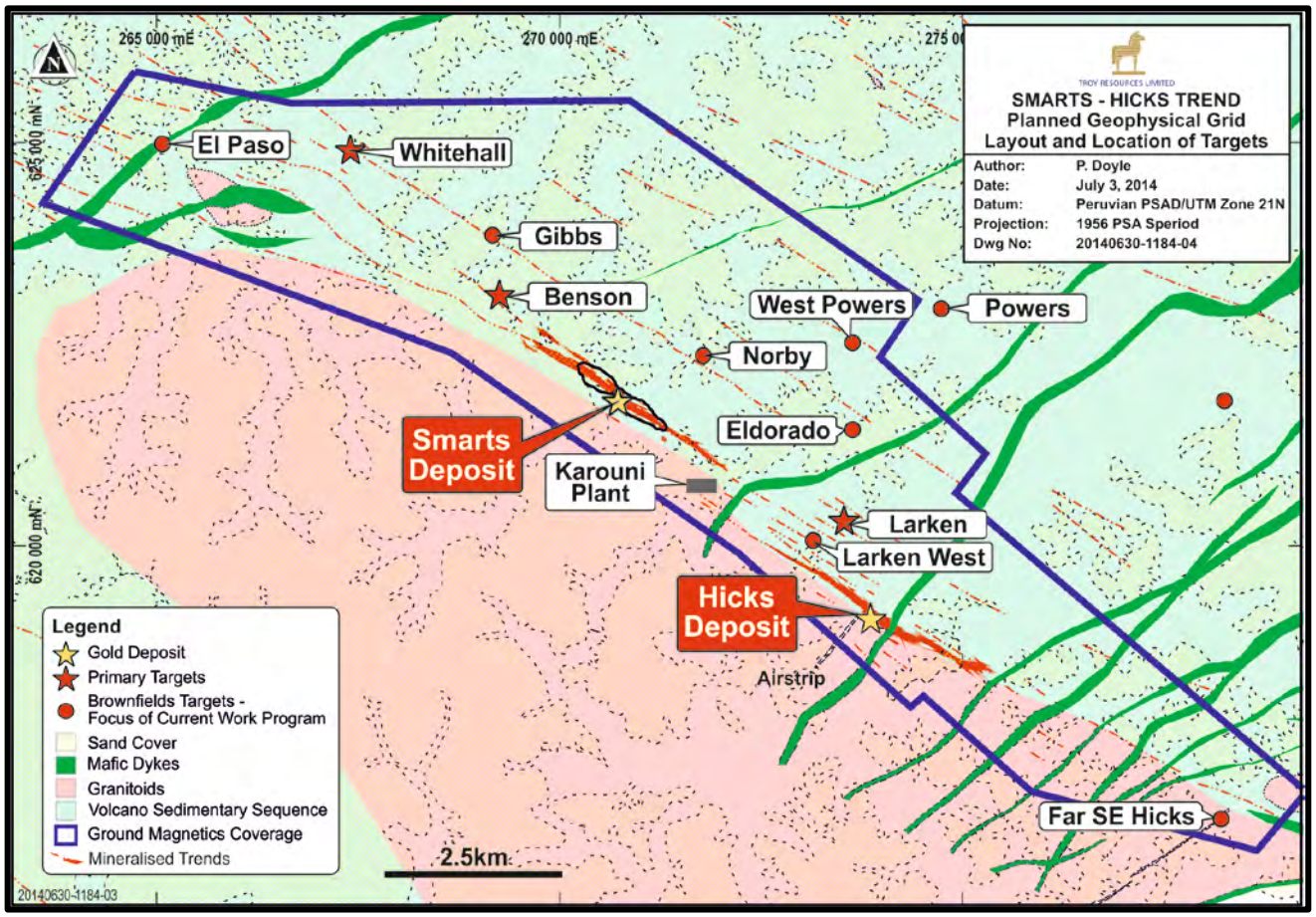

Figure 9-1: Planned Geophysical Grid Layout and Location of Targets

10. Drilling

10.1 Introduction

For the purposes of this report the database close-out is considered to be April 30, 2014. Since March 2013, Troy has managed the drilling conducted by Azimuth with all of the drilling focussed within the Mineral Resource areas at the Smarts, Hicks and Larken Zones. A total of 1,403 holes for 150,545m have been completed to this date. Drilling of the Hicks and Smarts Zones commenced in August 2010 and continued through to February 2014. All drilling completed by Troy and Azimuth on the Karouni Gold Project is summarised in Table 10-1. Complete drillhole collar information listing drillhole locations is included within Table 10-2 and Table 10-5 is a complete listing of significant drillhole assay intercepts of the drilling completed under Troy.

Since July 2013, Troy has continued to employ both Reverse Circulation and Diamond Core drill rigs to infill and upgrade the existing Mineral Resources. In terms of total Troy drilling; to date 123 holes have been drilled at the Smarts Zone for 16,895m of drilling and 62 holes at the Hicks Zone for 6,476.5m. The drilling metreage breakdown is as follows; Reverse Circulation; 69 holes have been drilled at the Smarts Zone for 6,050m of

drilling and 19 holes at the Hicks Zone for 2,079m, as well as Diamond Core; 54 holes have been drilled at the Smarts Zone for 10,845m of drilling and 43 holes at the Hicks Zone for 4,397.5m.

Troy also drilled 15 sterilization Reverse Circulation holes for 1623m at the proposed tailings containment area, to processing plant site and the new camp site.

Troy is compiling information from recent auger holes and validating this against the historical data set. The anticipated completion date of the auger database validation and re-compilation is the last calendar quarter of 2014. An orientation auger program was carried out by Troy in 2014 at Hicks and Smarts with the objective of studying the gold and multi-elements dispersion in saprolite. 72 Auger holes at Hicks and 126 Auger holes at Smarts were completed by Troy in May 2014.

Table 10-1: Summary of Troy - Azimuth Drilling Karouni Gold Project (as of April 30, 2014)

Type	Hole Prefix	Prospect	Number	Metres	Hole Sequence
Reverse Circulation (RC)	ARC	Airstrip	8	456	ARC001 to ARC008
	SRC	Croft	2	204	SRC553 to SRC554
	EPRC	El Paso	14	2,142	EPRC001 to EPRC014
	ERC	Eldorado	8	683	ERC001 to ERC008
	FRC	Far SE Hicks	11	1,275	FRC001 to FRC011
	SRC	Gibbs	17	1803	SRC225, SRC339 to SRC346, SRC681-688
	GRC	Gold Star	11	1,676	GRC001 to GRC011
	HRC	Hicks	172	15,957	HRC001 to HRC127, SRC598 to SRC599, SRC609 to SRC611, SRC634 to SRC644, SRC676 to SRC680, HRC128 to HRC161
	KRC	Kaburi	40	5,627	KRC001 to KRC012, KRC024 to KRC051
	RRC	Kaburi West	2	158	RRC001 to RRC002
	KRC	Kanhai	11	923	KRC013 to KRC023
	LRC	Larken	31	3,322	LRC001 to LRC031
	SRC	Norby	46	3,830	SRC066 to SRC068, SRC527 to SRC571
	PRC	Powers	9	1,275	PRC001 to PRC009
	SRC	Smarts	596	47,015	SRC001 to SRC526, SRC572 to SRC706, SRC707 to SRC712, SRC712A, SRC713 to SRC740, SRC740A, SRC741 to SRC749, SRC749A, SRC750 to SRC760, SRC760A, SRC761 to SRC771
	WPRC	West Powers	9	1,039	WPRC001 to WPRC009
TOTAL RC HOLES			**987**	**87,385**	
Diamond Core (DC)	HDD	Hicks	56	9,401	HDD001 to HDD062
	KDD	Kaburi	2	348	KDD001 to KDD002
	LDD	Larken	2	308	LDD001 to LDD002
	RDD	Regional	3	124	RDD001 to RDD003

	SDD	Smarts	140	26,962	SDD001 to SDD085, SDD094 to SDD102, SDD102A, SDD103 to SDD146
TOTAL DC HOLES			**203**	**37,143**	
DCT	SDT	Smarts	91	18,640	11,899m of RC precollars (SRC holes) with 6,741m of DC tail (SDT holes)
TOTAL SDT HOLES			**91**	**18,640**	
Sterilization	HRC, HDD and SDD	Tailings Facilities, Mill and Camp Sterilization and Hydrogeology	29	2,179	HRC142 to HRC151, HRC157 to HRC16, HDD020 to HDD025, SDD086 to SDD093
TOTAL HOLES			**29**	**2,179**	
Aircore (AC)	SAC	Gibbs	12	758	SAC001 to SAC012
	HAC	Hicks	2	149	HAC001 to HAC002
	RAC	Regional	65	3,554	RAC001 to RAC064
	SAC	Smarts	14	732	SAC013 to SAC026, SAC231
TOTAL AC HOLES			**93**	**5,193**	
ALL DRILLING TOTALS			**1,403**	**150,540**	

Notes:

1. **Smarts RC Drilling Total does not include the 91 SDT pre-collars for 6736m these were include with DCT metres**
2. **For DCT Holes: Depth from surface to end of diamond tail (includes RC pre-collar)**

Table 10-2: Karouni Gold Project Drilling Summary Troy Drilling (from July 13, 2013)

Hole ID	Northing (m)	Easting (m)	Elevation (m)	Depth (m)	Dip	Azimuth	Prospect	Type
HDD026	619544.4	273300.1	61.8	135	-62	35	Hicks	DD
HDD027	619416.1	273593	60.1	86.6	-51	215	Hicks	DD
HDD028	619452.9	273607.6	60.2	146.5	-55	215	Hicks	DD
HDD029	619410.3	273640.2	62.2	119.7	-55	215	Hicks	DD
HDD030	619371.4	273672.9	69.9	80.4	-51	215	Hicks	DD
HDD031	619359.5	273729.3	76.8	151.7	-54	215	Hicks	DD
HDD032	619258.9	273793.5	80.1	95.5	-51	215	Hicks	DD
HDD033	619292.4	273683.7	77.9	48	-51	35	Hicks	DD
HDD034	619267.3	273666.5	77.2	150.3	-51	35	Hicks	DD
HDD035	619288.4	273681	77.9	92.9	-51	35	Hicks	DD
HDD036	619266.1	273732.1	82.8	89.5	-50	35	Hicks	DD
HDD037	618832.8	274212.8	62.9	176.5	-55	35	Hicks	DD
HDD038	618881.8	274179.3	61.6	140.5	-50	35	Hicks	DD
HDD039	618890.3	274131.7	61.5	160.5	-58	35	Hicks	DD
HDD040	619061.3	274041.8	65.9	70.5	-50	215	Hicks	DD

HDD041	619072.8	274049.8	64.5	101.4	-51	215	Hicks	DD
HDD042	619080	274008.5	73	122.5	-55	35	Hicks	DD
HDD043	618997.7	274049.1	57.1	122.4	-56	35	Hicks	DD
HDD044	619008.8	274062.5	57.3	89.5	-50	35	Hicks	DD
HDD045	619017	273965	67.6	182.4	-55	35	Hicks	DD
HDD046	619038.8	273919.7	70.7	164.7	-58	35	Hicks	DD
HDD047	619107.6	273912.3	64.1	74.5	-50	35	Hicks	DD
HDD048	619131	273930.7	60.5	83.4	-49	35	Hicks	DD
HDD049	619136.1	273857.1	73.4	77.3	-51	35	Hicks	DD
HDD050	619120.6	273834.5	76.2	142.8	-62	35	Hicks	DD
HDD051	619154.3	273790.8	81.3	121	-54	35	Hicks	DD
HDD052	619113.5	273781.8	81.8	185.3	-51	35	Hicks	DD
HDD053	619564.7	273318.3	63.7	80.1	-54	35	Hicks	DD
HDD054	619563.9	273399.7	63	62.3	-50	215	Hicks	DD
HDD055	619607	273417	65.8	149.4	-54	215	Hicks	DD
HDD056	619532	273440.1	63	80.5	-55	215	Hicks	DD
HDD057	619550.1	273454.5	69.4	110.3	-56	215	Hicks	DD
HDD058	619569.4	273465.6	70.9	146.5	-56	215	Hicks	DD
HDD059	619510.3	273485.4	60	80.5	-55	215	Hicks	DD
HDD060	619552.2	273507.7	67.5	173.5	-55	215	Hicks	DD
HDD061	618491	274988.2	84.7	68.5	-59	35	Hicks	DD
HDD062	618471.4	274976.6	85.9	90	-54	35	Hicks	DD
HRC128	620113.2	272347.6	67	103	-56	35	Hicks	RC
HRC129	620056.8	272398.7	77.4	103	-57	35	Hicks	RC
HRC130	620341.1	271993.5	99.3	97	-56	35	Hicks	RC
HRC131	620171.1	272234.8	66.1	97	-55	35	Hicks	RC
HRC132	620204.6	272168	68.3	103	-54	35	Hicks	RC
HRC133	620453.7	271820.3	105.6	129	-56	35	Hicks	RC
HRC134	620504.2	271733.5	103.6	109	-55	35	Hicks	RC
HRC135	620565.6	271650.9	101.4	91	-55	35	Hicks	RC
HRC136	620625.6	271561.1	101.9	130	-54	35	Hicks	RC
HRC137	620085.4	272327.8	67.5	109	-53	35	Hicks	RC
HRC138	620535.8	271691.7	102	109	-55	35	Hicks	RC
HRC139	620550	271640.5	99.4	124	-54	35	Hicks	RC
HRC140	620593.8	271609.9	101.1	109	-54	35	Hicks	RC
HRC141	620672.4	271474.2	102.1	115	-54	35	Hicks	RC
HRC152	620696.3	271488.3	104.7	121	-57	35	Hicks	RC
HRC153	620754.3	271413.4	107.8	91	-57	35	Hicks	RC

HRC154	620578.4	271678.6	104.6	115	-56	35	Hicks	RC
HRC155	620615.9	271705.4	107.5	121	-56	35	Hicks	RC
HRC156	620656.7	271734.7	108.3	103	-55	35	Hicks	RC
SDD094	621779.4	270961.1	85.3	233.7	-60	215	Smarts	DD
SDD095	621569.7	271015.6	65	173.6	-61	35	Smarts	DD
SDD096	621648.5	271059.7	64.7	89.9	-67	35	Smarts	DD
SDD097	621625	270940	66.9	119.8	-65	35	Smarts	DD
SDD098	621690.3	270869.6	75.3	80.5	-60	215	Smarts	DD
SDD099	621760.1	270944.7	85.9	152.2	-61	215	Smarts	DD
SDD100	621568.2	271139.3	69.4	51.6	-66	215	Smarts	DD
SDD101	621569.6	271016.3	65	80.5	-55	35	Smarts	DD
SDD102	622061.3	270405.8	103.2	21.6	-64	35	Smarts	DD
SDD102A	622065.5	270409.2	103.2	125.6	-67	35	Smarts	DD
SDD103	621983.7	270480.3	97.1	142.8	-66	35	Smarts	DD
SDD104	621891.9	270476.7	82.8	268.8	-63	35	Smarts	DD
SDD105	622152.1	270267.7	101.9	155.5	-64	35	Smarts	DD
SDD106	622060.3	270480	103.6	130	-68	35	Smarts	DD
SDD107	622165.8	270481.9	96.1	135.7	-66	215	Smarts	DD
SDD108	622019.5	270514.7	102.9	150	-62	215	Smarts	DD
SDD109	621696.4	270771	70.4	240	-63	35	Smarts	DD
SDD110	621812	270655	74.9	3.2	-61	35	Smarts	DD
SDD111	621815.9	270657.2	75.9	235.6	-64	35	Smarts	DD
SDD112	621823.1	270930.1	80.5	240	-55	215	Smarts	DD
SDD113	621911.3	270489.1	88.8	221.7	-62	35	Smarts	DD
SDD114	621720.6	271058.9	81.6	230	-65	215	Smarts	DD
SDD115	621960.9	270464.1	91.9	209.8	-63	35	Smarts	DD
SDD116	621760.1	270563.1	70.9	360	-61	35	Smarts	DD
SDD117	621975.5	270407.3	91.8	230.6	-63	35	Smarts	DD
SDD118	621761.2	270682.5	72.4	250	-61	35	Smarts	DD
SDD119	622021	270369.3	101.1	233.7	-65	35	Smarts	DD
SDD120	621698.9	271105.3	76.9	240	-59	215	Smarts	DD
SDD121	622036	270319.2	103.2	230	-62	35	Smarts	DD
SDD122	621689	271155.8	71.2	265.1	-60	215	Smarts	DD
SDD123	621698.3	270712.3	69.7	300	-60	35	Smarts	DD
SDD124	621047.3	271859.4	76.2	107.7	-66	35	Smarts	DD
SDD125	621710.5	270648.3	72.8	344.5	-60	35	Smarts	DD
SDD126	621843.1	270561.2	84.7	221.7	-61	35	Smarts	DD
SDD127	621811.9	270597.7	80	300	-61	35	Smarts	DD

SDD128	621765.2	270621.9	71.7	300	-61	35	Smarts	DD
SDD129	622060	270265	103.1	240	-63	35	Smarts	DD
SDD130	621874.3	270523.8	88	225	-62	35	Smarts	DD
SDD131	621720.9	270591.3	71	389.7	-61	35	Smarts	DD
SDD132	622003.9	270297.4	103	84.3	-64	35	Smarts	DD
SDD133	621979.2	270339.5	96	360	-64	35	Smarts	DD
SDD134	621680	270627.1	74.6	389.5	-63	35	Smarts	DD
SDD135	621954.1	270395	83	239.6	-60	35	Smarts	DD
SDD136	621647.9	270680.2	72.8	167.5	-62	35	Smarts	DD
SDD137	621837.1	270499.3	79	305.6	-62	35	Smarts	DD
SDD138	621651	270682.9	72.6	353.5	-63	35	Smarts	DD
SDD139	621793.8	270527.8	73.4	275.7	-56	35	Smarts	DD
SDD140	621658.6	270745.3	69.3	305.5	-60	35	Smarts	DD
SDD141	622010.3	270297.4	103.2	281.7	-66	35	Smarts	DD
SDD142	621061.4	271873.1	86.2	72	-55	35	Smarts	DD
SDD143	621143.6	271787.5	68.1	55	-54	35	Smarts	DD
SDD144	621082.7	271847.8	73.9	70	-57	35	Smarts	DD
SDD145	621149.3	271760.2	69.3	75	-56	35	Smarts	DD
SDD146	621161.6	271742.8	72.2	80	-56	35	Smarts	DD
SRC707	621603.2	271170.3	66.3	85	-55	215	Smarts	RC
SRC708	621588.4	271164.9	67.6	90	-50	200	Smarts	RC
SRC709	621560.6	271158.8	66.8	70	-53	215	Smarts	RC
SRC710	621578.4	271111	68.9	61	-54	215	Smarts	RC
SRC711	621600.8	271125.1	68	85	-55	215	Smarts	RC
SRC712	621622.7	271140	68.9	49	-55	215	Smarts	RC
SRC712A	621619.4	271138.2	68.8	120	-55	215	Smarts	RC
SRC713	621657.3	270984.1	75.8	61	-55	215	Smarts	RC
SRC714	621687.1	270941.5	79.2	61	-55	215	Smarts	RC
SRC715	621771.7	270927.4	85.1	79	-55	215	Smarts	RC
SRC716	622004.7	270528	102	79	-52	35	Smarts	RC
SRC717	621654.1	271104.7	72.1	120	-54	215	Smarts	RC
SRC718	621629.8	271087.8	68.1	90	-56	215	Smarts	RC
SRC719	621606.3	271071.3	64.9	67	-55	215	Smarts	RC
SRC720	621668.1	271054.4	74.4	115	-61	215	Smarts	RC
SRC721	621654	271041.6	69	90	-59	215	Smarts	RC
SRC722	621635.3	271030.7	65.7	61	-50	215	Smarts	RC
SRC723	621679.8	270998.5	79.2	109	-55	215	Smarts	RC
SRC724	621704.3	271016.2	81.8	127	-55	215	Smarts	RC

SRC725	621708.9	270955.2	82.5	109	-56	215	Smarts	RC
SRC726	621729.4	270968.7	85.5	133	-56	215	Smarts	RC
SRC727	621747.4	270981.1	86.9	91	-55	215	Smarts	RC
SRC728	621713.7	270887	78.4	97	-55	215	Smarts	RC
SRC729	621737.6	270904	81.1	121	-56	215	Smarts	RC
SRC730	621754.5	270915.9	82.9	121	-60	215	Smarts	RC
SRC731	621771.2	270862.1	75.3	109	-55	215	Smarts	RC
SRC732	621789.3	270874.4	76.5	127	-55	215	Smarts	RC
SRC733	621807.6	270887.5	78.4	97	-55	215	Smarts	RC
SRC734	621825.8	270828.6	80.7	121	-58	215	Smarts	RC
SRC735	621829.6	270694.9	70.3	103	-55	35	Smarts	RC
SRC736	621851.8	270711.2	71.3	79	-54	35	Smarts	RC
SRC737	621887.2	270735.5	75.7	115	-54	215	Smarts	RC
SRC738	621886	270707	76.5	91	-54	35	Smarts	RC
SRC739	621867	270667.3	71.5	67	-54	35	Smarts	RC
SRC740	621886.5	270680.5	73.4	79	-55	35	Smarts	RC
SRC740A	621885.6	270679.7	73.6	13	-55	35	Smarts	RC
SRC741	621901.6	270692.1	78.7	73	-53	35	Smarts	RC
SRC742	621927.7	270650.8	89.6	79	-55	35	Smarts	RC
SRC743	621894.3	270627.8	86.3	91	-55	35	Smarts	RC
SRC744	621924.4	270589.2	94.1	111	-58	35	Smarts	RC
SRC745	621940.1	270601.2	94.8	103	-59	35	Smarts	RC
SRC746	621959.9	270617	96.6	115	-54	35	Smarts	RC
SRC747	622029.3	270572.2	103.9	79	-54	215	Smarts	RC
SRC748	621967.5	270501.4	97.7	127	-54	35	Smarts	RC
SRC749	621981.8	270511.3	99.3	25	-55	35	Smarts	RC
SRC749A	621982.6	270511.8	99.3	31	-55	35	Smarts	RC
SRC750	621995.7	270520.5	100.5	79	-45	35	Smarts	RC
SRC751	622000.2	270465.3	98.9	127	-54	35	Smarts	RC
SRC752	622018.2	270477.7	100.9	97	-57	35	Smarts	RC
SRC753	622035.8	270489.6	103.1	73	-56	35	Smarts	RC
SRC754	622035.4	270421.9	102.7	127	-56	35	Smarts	RC
SRC755	622055.5	270434.5	103.7	93	-54	35	Smarts	RC
SRC756	622073.3	270447.2	103.4	73	-55	35	Smarts	RC
SRC757	622075.2	270377.4	103.1	121	-56	35	Smarts	RC
SRC758	622096.5	270391.3	102.5	85	-56	35	Smarts	RC
SRC759	622114.5	270404.1	101.9	61	-54	35	Smarts	RC
SRC760	622138	270389	110	28	-63	35	Smarts	RC

SRC760A	622133.8	270386.8	101.2	49	-63	35	Smarts	RC
SRC761	622113.5	270342.5	102.2	113	-55	35	Smarts	RC
SRC762	622131.4	270354.6	101.6	75	-55	35	Smarts	RC
SRC763	622147.4	270366.6	100.6	67	-54	35	Smarts	RC
SRC764	622167.9	270278.3	101.4	103	-59	35	Smarts	RC
SRC765	622182.4	270288.6	101	85	-59	35	Smarts	RC
SRC766	622199.7	270300.8	100	73	-60	35	Smarts	RC
SRC767	621980.3	270510.4	99.1	115	-54	35	Smarts	RC
SRC768	621994.7	270519.5	100.4	91	-55	35	Smarts	RC
SRC769	621911	270637.5	88.7	73	-56	35	Smarts	RC
SRC770	621010.6	271915.8	86.3	85	-56	35	Smarts	RC
SRC771	621161.6	271742.8	72.2	31	-55	35	Smarts	RC

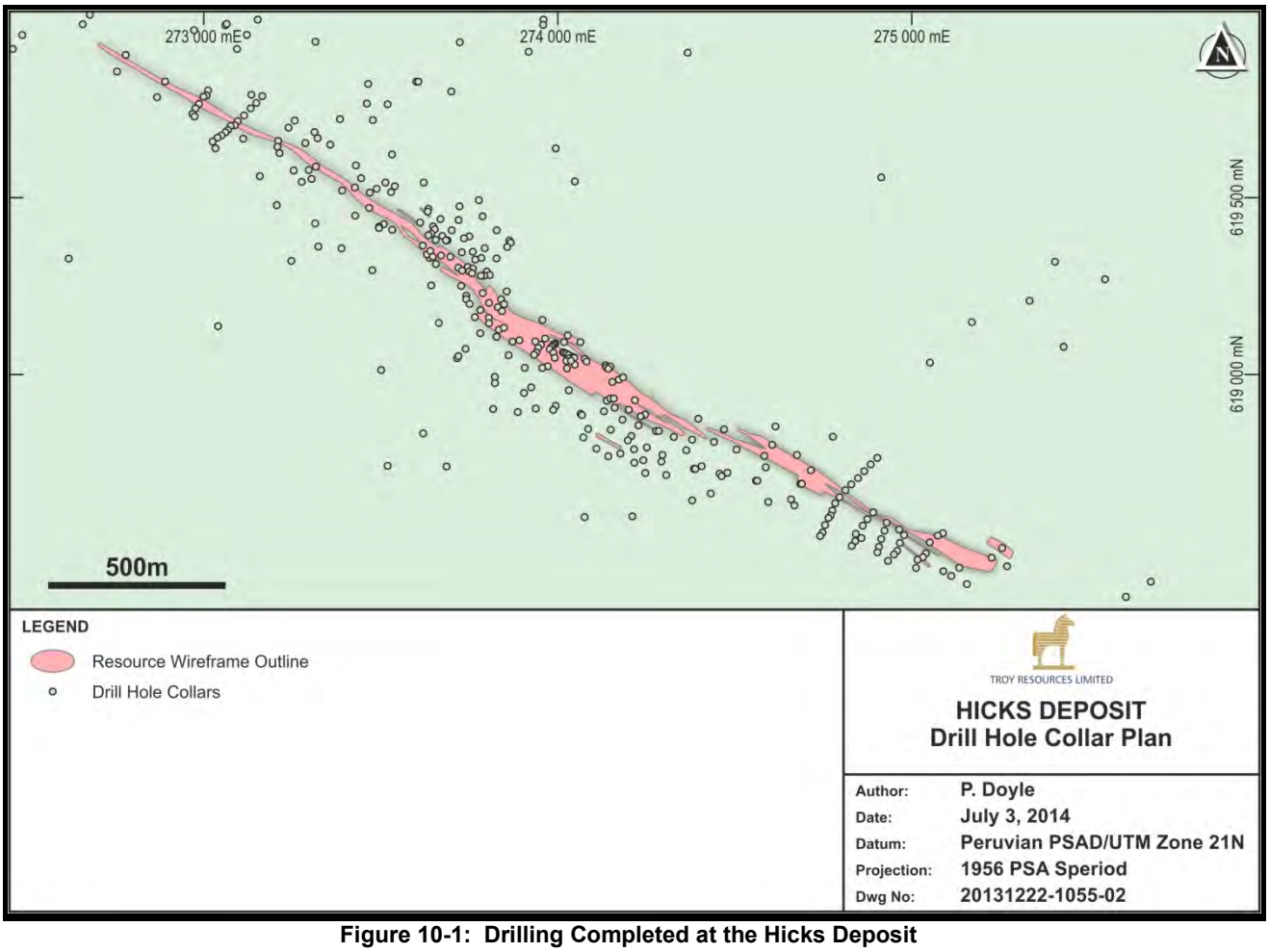

Figure 10-1: Drilling Completed at the Hicks Deposit

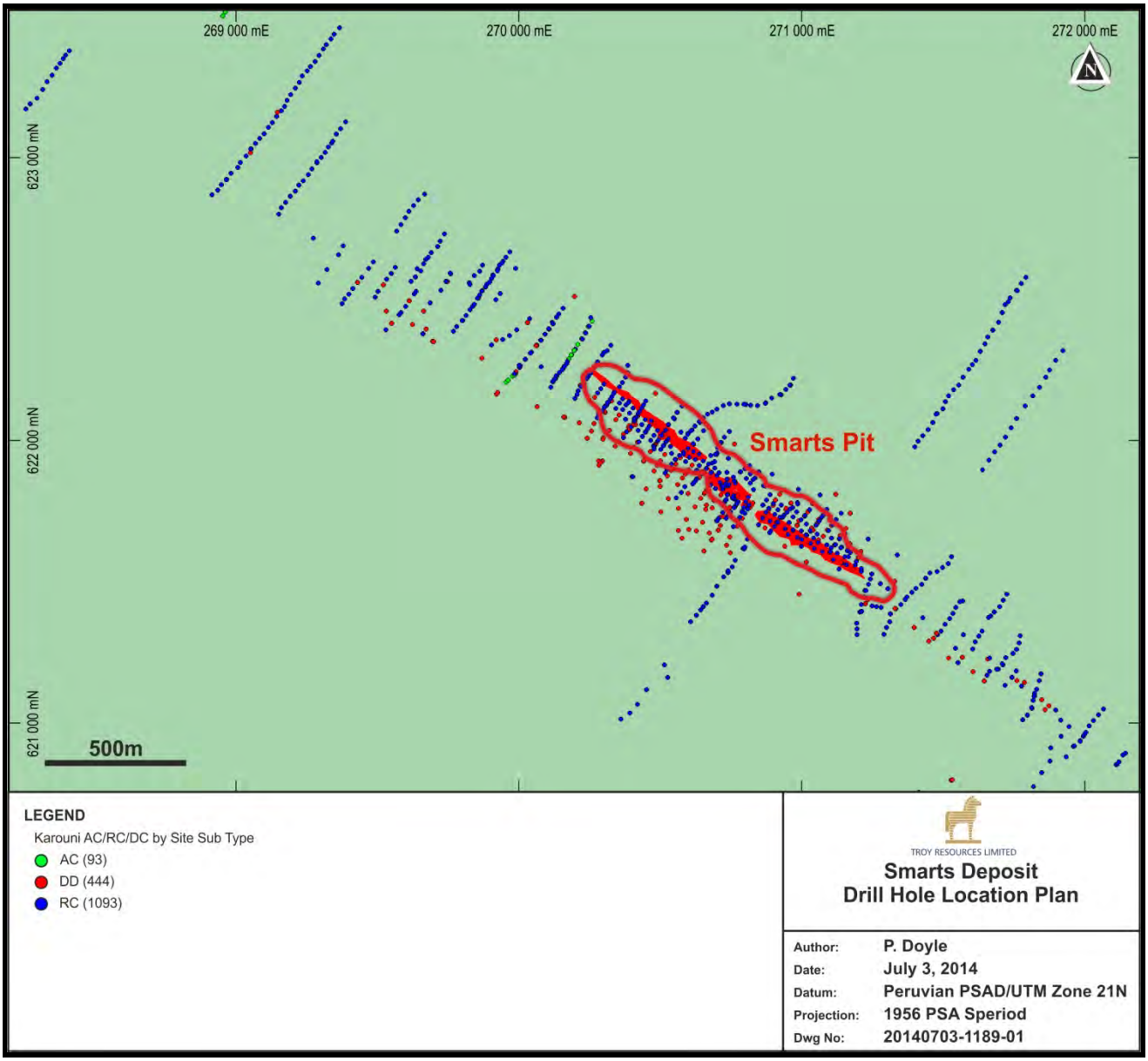

Figure 10-2: Drilling Completed at the Smarts Deposit

10.2 Collar and Downhole Survey

RC and Diamond Core holes at Smarts were drilled at 20m to 80m spacings on 25m to 220m spaced north-east southwest orientated drill sections. At Hicks, the drill spacing is 25m to 60m on 15m to 100m spaced drill sections of the same orientation. These spacings are sufficient to show a continuity of mineralisation in the main mineralised zones of both deposits. The mineralisation at Hicks and Smarts is generally steep dipping and holes drilled from the surface are generally not orthogonal to the mineralisation down-dip. Consequently the true thickness of intersections is generally less than the downhole thickness and typically 60% to 80% of the downhole length as a guideline. Troy has orientated the drill holes to be generally orthogonal to the northwest striking mineralisation.

Drillhole collars are surveyed with Differential GPS by Troy personnel and compared to a 2011 LiDAR surface for verification. All holes are inclined holes with the dip ranging between 40° and 75°. Holes are inclined from collar positions outside of the mineralised zone and intersect the mineralised zone at a high angle. (refer to Figure 10-3 and Figure 10-4).

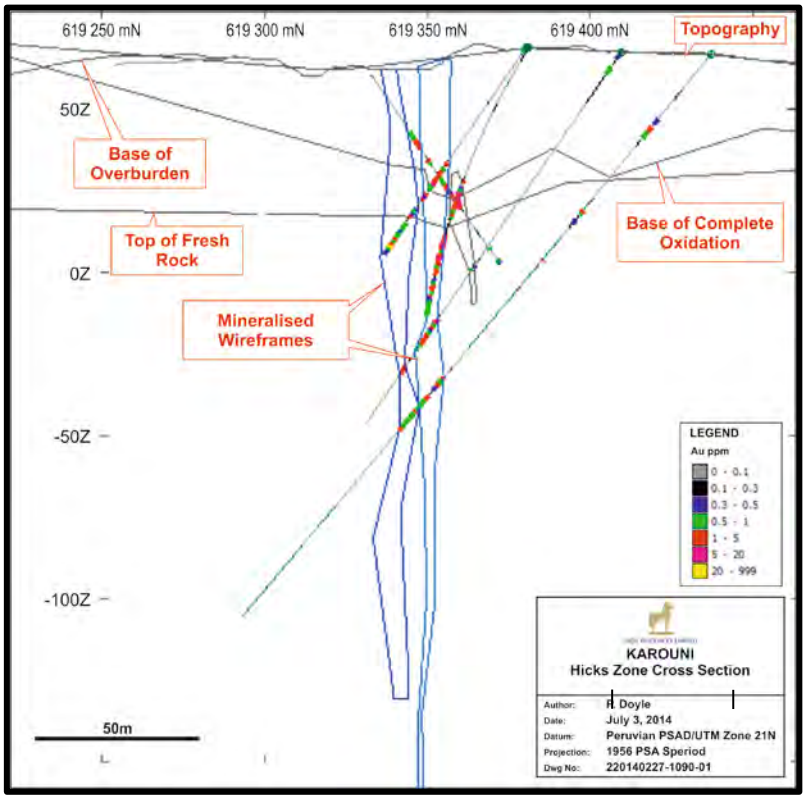

Figure 10-3: Typical Cross Section for the Hicks Deposit

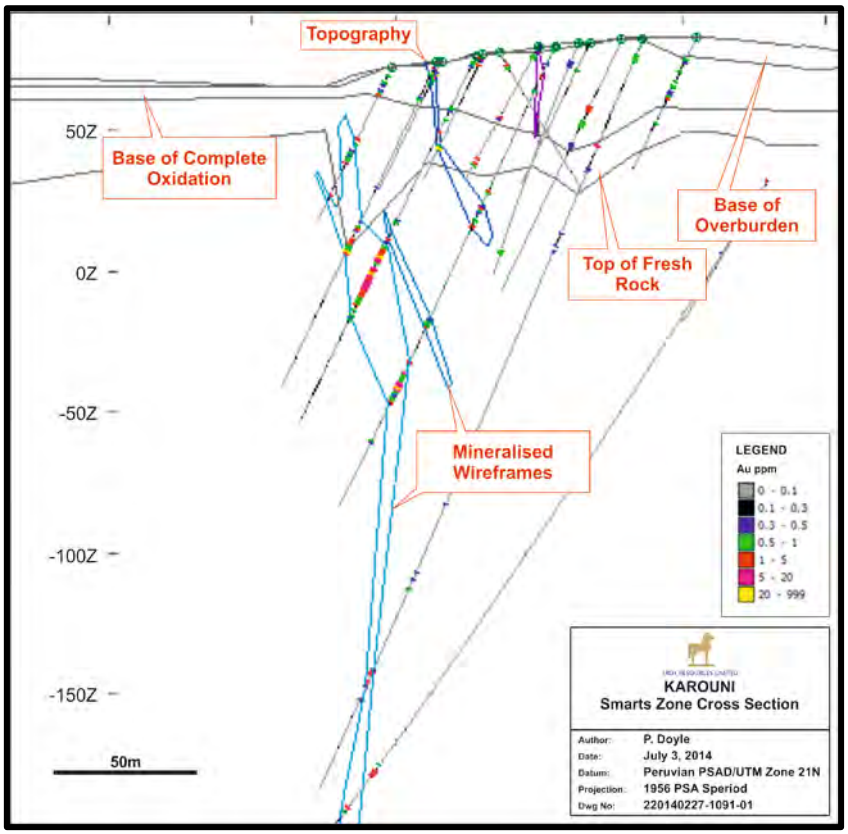

Figure 10-4: Typical Cross Section for the Smarts Deposit

10.3 Bulk Density

10.3.1 Bulk Density Test work

10.3.2.1 Methodology – Drill Core

Troy conducts density test work using the water immersion technique. The work has been undertaken on recent Smarts and Hicks Diamond Core drilled by Azimuth/Troy. All density test work is currently conducted on-site as follows:-

- Testwork locations are marked on drill core by the logging geologist. The samples are generally 10 to 15cm in length and are marked with a unique sample identification number. Samples are collected from intact whole core and are chosen to be broadly representative of the main lithological and mineralisation units (see Figure 10-5). Core samples are chosen at a rate of one sample every two core boxes or more frequently where required.

Figure 10-5: Core Samples for Bulk Density Test Work

- Samples are weighed in grams on a digital scale to an accuracy of one decimal place. A 200g standard weight is tested in air and water each morning before test work commencement to ensure the scale is calibrated correctly. The scale calibration test is repeated at the commencement of test work for each drillhole.
- Samples are sealed in candle wax or clear film after drying and before re-weighing in water. Wax coating or film wrap is undertaken immediately after drying to ensure samples do not re-absorb moisture from the atmosphere.

10.3.2.2 Methodology – Surface Sample

The Company has additionally carried out bulk density test work for the unconsolidated sand overburden. A surface sample was collected for density test work as only minor core drilling has been carried out in the oxide

and overburden zone. The test work procedure uses the volume displacement method with minor variation depending on surface topography. The procedure can be outlined as follows: -

- Flat topography
  - Surface sample is collected from a nominally sized 30cm by 30cm by 30cm hole.
  - A plastic bag is placed in the hole and filled water to surface. The bagged water is then measured using a marked beaker.
  - The removed material is also bagged and transported to site.
  - Bagged samples are oven dried and weighed to allow calculation of a dry bulk density value.
- Steep to vertical topography
  - A 2" PVC spear of known length and dimensions is used to collect a sample. The spear is driven into the surface until completely filled at which point the sample is retrieved and bagged.
  - Bagged samples are oven dried and weighed to allow calculation of a dry bulk density value.

10.3.2.3 Results

Statistical analysis of the core density data by weathering zone and inside/outside the mineralisation domain was conducted. Bulk density results are shown below in Table 10-4. A review of the bulk density for barren sand overburden was also carried out. An average density was obtained by material type. Outlier values were excluded using the following criteria listed below and in Table 10-3:-

- Overburden: - Samples rejected where <1.2 or >2.0.
- Oxide: - Samples rejected where >2.7 (6 samples).
- Transition: - One obvious outlier was rejected.
- Fresh: - Samples rejected where <1.5 or >3.5 (3 samples).

Table 10-3: Density Values by Material Type and Deposit (t/m³)

Material Type	No. Samples	Average Density	Min. Density	Max. Density
Hicks Oxide Mineralised	0	-	-	-
Hicks Oxide Waste	64	1.77	1.26	2.7
Hicks Transitional Mineralised	0	-	-	-
Hicks Transitional Waste	52	2.81	1.93	3.17
Hicks Fresh Mineralised	98	2.76	2.37	3.03
Hicks Fresh Waste	1,281	2.86	1.57	3.49
Smarts Oxide Mineralised	43	1.82	1.38	2.68
Smarts Oxide Waste	153	1.69	1.24	2.59
Smarts Transitional Mineralised	17	2.29	2	2.72
Smarts Transitional Waste	149	2.29	1.49	3.14
Smarts Fresh Mineralised	188	2.73	1.7	3.1
Smarts Fresh Waste	1,020	2.88	1.49	4.52
All Overburden	12	1.82	1.58	2.12
All Oxide Mineralised	43	1.82	1.38	2.68
All Oxide Waste	215	1.71	1.24	2.59
All Transitional Mineralised	17	2.29	2	2.72
All Transitional Waste	201	2.43	1.49	3.17
All Fresh Mineralised	286	2.74	1.7	3.1
All Fresh Waste	2,301	2.87	1.49	4.52

Note: - The bulk density test work for the Smarts Transitional Waste material type is currently considered more correct than the historical test work compiled for the Hicks Transitional Waste material type.

10.4 Moisture Testwork

The bulk density has been applied on a dry basis.

10.5 Soil/Saprolite Auger Sampling

For auger sampling, material is collected in one metre intervals using a “Dutch” or Edelman auger and homogenized on a clean plastic sheet. Most auger samples were collected by hand, excluding a brief period where a mechanized auger was used. Auger depths have varied, but are often as much as 9m and reach a maximum depth of 15m. A representative 1kg - 2kg sample is extracted by coning and quartering either from a 1m or 2m sample interval. A blank control sample, a field duplicate and a standard are inserted every 40-50 samples.

Lithology type, colour and the presence of sulphide, quartz veins or alluvial material is recorded for each metre upon sampling and subsequently cross checked by the supervising geologist. Data entry has been carried out in the same way as for drill data to ensure consistency in information collection.

Two types of auger sampling have been carried out with the majority of auguring collected at 25m intervals on regularly spaced 035^0 trending grids, joined by a local baseline. Each collar site is also surveyed using a GPS, and in some cases has been surveyed using DGPS. In some areas where valleys are filled with alluvium and tailings, ridge and spur sampling at 25m intervals has been carried out using a similar methodology.

Auger sampling results are used to guide further exploration such as RC and diamond drilling. The auger data is not included in the resource estimation process.

10.6 Drilling Conditions

The drill pad area is maintained at a good house-keeping standard to ensure equipment is in good working order for the drill campaign. This includes level working conditions and weather protection.

The cyclone is cleaned before drilling starts for each RC hole and is kept clean throughout drilling and sample collection Samples for each metre are spread evenly across the splitter and care is taken to ensure the sample does not spill over the edge of the splitter. Any material adhering to the splitter represents a source of potential contamination between intervals (as is the case for the cyclone). The splitter is regular checked for corrosion or any damage and the sample may be assisted through the splitter with the aid of a rubber mallet. Wet samples were collected and allowed to dry before splitting.

Recent drilling by Troy has been conducted by two Contractors; Orbit Drilling and Drillex. Multiple drill rigs are in operation including: Orbit Diamond Core (DC) rig, Orbit Reverse Circulation (RC rig) and Drillex 6000 Multipurpose. Since late 2012 through to April 2014 only the Orbit Contract rigs were in operation (see Figure 10-6 and Figure 10-7).

Figure 10-6: 2012 Contract Reverse Circulation (RC) Drill Rig at NW Smarts - Karouni Gold Project

Figure 10-7: Contract Drill Diamond Core (DC) Rig – Smarts Zone

10.7 Sample Quality and Recovery

The sampling methodology described in this section relates specifically to the post-2009 drilling campaigns by Troy (Azimuth now Troy). Pre-2009 diamond core drilling was cut in half using a core saw and sampled to 1m intervals or to geological boundaries, although some larger composites were taken in barren zones.

10.7.1 Sample Collection of RC Drilled Intervals

All RC drilling to date has been completed by using a 5" to 5 ½" face hammer with samples collected mainly at 1m or 3m intervals. The rig geologist determines if the sampling is to be done at 1m increments or composited into a 3m interval. Sample pertaining to mineralised composites returning >0.25g/t gold are retrieved and these 1m samples are dried, re-split and submitted for assay.

10.7.2 RC Drill Core Recovery

The RC drilling has an average drill sample recovery of 18kg. Theoretical sample weights are 20kg weight for 100% recovery in oxide material and a 36kg weight for 100% recovery in fresh material. The theoretical drill sample weights include 5% moisture content.

Sample recovery appears acceptable for the RC drilling after allowing for varying density by weathering. The sample weight is relatively low in near-surface intervals; however some recovery loss would be expected in unconsolidated material which forms a relatively minor rock type.

10.7.2.1 Weight of RC Sample Dispatched to Assay Laboratory

Sample submission was supervised by a Company geologist and also checked by the receiving laboratory to ensure sufficient sample mass was obtained for assay analysis. Troy maintains an ongoing continuous QAQC Program. Troy completed a QAQC quality assurance program during 2011-12 whereby the laboratory sample weight was recorded. As the laboratory sample is a representative sub-lot of the sample mass at a constant split from the two-tier or three-tier riffle splitter, a measure of overall sample quality and representivity is provided by this analysis.

The date of the sample, water content (as wet, moist or dry) and sample identification were recorded for 15,733 samples mainly from the Smarts and Hicks drilling. This represents approximately 27% of the assay database as at 13 July 2013. Approximately 22% of the samples were recorded as wet with the remainder recorded as dry or moist. Summary statistics for the original field sample by sample length are compiled in Table 10-4 below.

Table 10-4: Summary Weight of Original Field Samples (SRC collar prefix)

Sample Length	Sample Moisture	Sample Count	Sample Weight (Ave. kg)	Sample Weight (Min. kg)	Sample Weight (Max. kg)	Sample Weight (SD kg)
1m	**Dry to moist**	**7,962**	**2.54**	**0.01**	**7.00**	**0.95**
1m	**Wet**	**2,156**	**2.18**	**0.08**	**6.56**	**0.99**
2m	**Dry to moist**	**106**	**3.65**	**0.08**	**6.54**	**1.19**
2m	**Wet**	**100**	**4.10**	**0.12**	**8.24**	**1.54**
3m	**Dry to moist**	**12,426**	**4.33**	**0.01**	**12.58**	**1.62**
3m	**Wet**	**3,753**	**4.12**	**0.01**	**9.28**	**1.75**
4m	**Dry to moist**	**4**	**4.52**	**3.32**	**5.72**	**1.70**
4m	**Wet**	**12**	**5.08**	**1.60**	**7.18**	**3.04**
All	**All**	**26,519**	**3.58**	**0.02**	**9.90**	**1.38**

A sample weight comparison between the original sample and the field duplicate sample is presented in Figure 10-8 for 429 samples. Results are reasonable in both cases. It is recommended sample weights continue to be recorded on an 'as dry' basis for sample recovery analysis. Site visits conducted by Troy's QPs have concluded that the field procedures in place are suitable and well maintained.

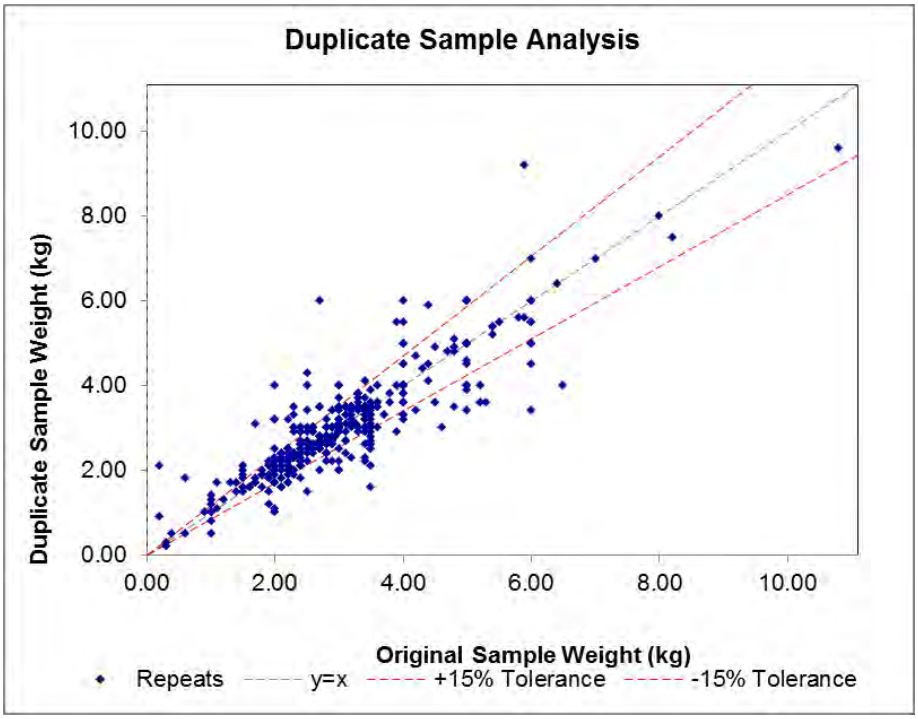

Figure 10-8: Sample Weight Comparison between Original and Duplicate Field Samples

10.7.3 Sample Collection of Diamond Drilled Intervals

Diamond Core drilling is mainly done HQ and HQ3 sized equipment and minor PQ, with samples collected mainly at 1m lengths. Diamond Core is retrieved from the drill site twice daily by a geologist and secured in the back of a Landcruiser pick up using ratchet straps.

10.7.4 Diamond Core Drilling Recovery

The Diamond Core recovery has been calculated for historical drill core and results have been entered into the database. Average core recovery for the diamond drilling is 91%. Of note is that approximately 3% of the sample intervals record a recovered core recovery above 100%. The reason for this discrepancy is unknown, however, it is thought this largely reflects a data translation error. Other additional errors are thought to reflect inexact locations of core depth marker blocks or possible drill core lifter problems with minor core from one core barrel run being included in the next core barrel run.

Unfortunately historical drillcore is considered lost due to poor preservation from an inappropriate storage facility used by previous explorers. It is considered likely that any impact from the data translation errors would be of minor importance and of minor risk.

The diamond drill core recovery has also been calculated for core drilled up to July 2013 and core recovery was found to be excellent in fresh rock. Average core recovery for the Diamond Core drilling is 99%. Visual inspection of core during the site visits supported the quality and recovery of the core (see Figure 10-9). There are no known core recovery factors which would materially impact the accuracy and reliability of assay results.

Figure 10-9: Example of Good Core Recovery from Diamond Drill Core

10.8 Geological Controls

Important geological controls or domain boundaries for gold mineralisation relate to the flat-lying weathering or barren overburden horizons and the sub-vertical, mineralised shear zones. In this respect, the structurally related shear zones provide the most important parameter in selecting the sample interval length. The mineralised shear zones vary from several metres wide to a width of tens of metres in places. The mineralisation boundary is sharply defined over an interval of decimetres and the waste separation between mineralised zones is also in the order of several metres and up to ten metres width in places. The inclined 1m sample interval is sufficient to define the mineralisation and domain boundaries at a scale suitable for further block model and optimisation studies.

Gold mineralisation downhole is variable and significantly higher grade intersections do occur adjacent to mineralisation boundaries and/or lower grade intersections. Studies to date on the better drilled Smarts deposit indicate that the coefficient of variation (CV) is 2.96 for mineralised intersections. This suggests that standard industry techniques such as Ordinary Kriging with minor top-cuts can be used for the block model estimate and that the quality of the estimate will be sufficient for optimisation studies.

10.9 True Thickness of Intersections

Troy has released the results of exploration drilling conducted during 2012 - 2014 as assay results have returned. The visual mineralisation is recognised as iron-oxide stained quartz veins within a shear zone and has been identified in the majority of holes. The orientation of mineralised zones comprising each prospect is well established, and it is estimated that the true width of mineralised intercepts reported in the tables below of significant intersections represents some 60% to 80% of the downhole intervals. Significant intersections for the recent drilling at the Smarts and Hicks Deposits, as well as some of the local prospects are listed below in Table 10-5.

Table 10-5: Smarts, Hicks and Larken - Significant Intercepts (July 13, 2013 – April 30, 2014)

Hole	Easting (m)	Northing (m)	Elevation (m)	Prospect Name	Depth From (m)	Depth To (m)	Length (m)	Gold Grade (g/t) Au
HDD026	273300	619544	62	Hicks	94	95	1	6.76
					95	96	1	3.99
					96	97	1	2.50
					97	98	1	2.52

					98	99	1	1.59
					99	100	1	5.31
					103	104	1	4.68
					104	105	1	1.18
					108	109	1	1.19
HDD027	273593	619416	60	Hicks	8.1	9	0.9	2.43
					9	10	1	2.87
					15	18	3	2.46
					20	21	1	1.47
					21	22	1	1.59
					63	64	1	2.73
					64	65	1	17.36
HDD028	273608	619453	60	Hicks	82	83	1	1.87
					84	85	1	3.43
					85	86	1	2.49
					86	87	1	2.28
					87	88	1	2.47
					114	115	1	3.29
					115	116	1	1.28
					117	118	1	4.89
					118	119	1	6.33
HDD029	273640	619410	62	Hicks	55	56	1	1.74
					56	57	1	4.87
					58	59	1	1.69
					62	63	1	1.56
					64	65	1	2.21
HDD030	273673	619371	70	Hicks	22	23	1	20.05
					23	24	1	5.67
					30	31	1	2.56
					31	32	1	1.00
					39	40	1	1.00
					41	42	1	1.66
					62	63	1	1.52
					63	64	1	2.62
HDD031	273729	619359	77	Hicks	69	70	1	1.40
					70	71	1	1.20
					71	72	1	1.64
					95	96	1	3.84
					96	97	1	1.55
					97	98	1	2.35
HDD032	273793	619259	80	Hicks	44	45	1	2.28
					46	47	1	37.82
					47	48	1	3.37
					48	49	1	1.39
					50	51	1	1.01
					51	52	1	6.26
					52	53	1	1.83
					56	57	1	2.28
					57	58	1	2.53
					58	59	1	3.85
					59	60	1	1.58
					60	61	1	1.03
HDD034	273667	619267	77	Hicks	100	101	1	5.43
					101	102	1	3.23
					102	103	1	5.28

					103	104	1	2.20
					105	106	1	1.34
					119	120	1	8.45
					122	123	1	1.75
					123	124	1	3.24
					124	125	1	1.43
HDD035	273681	619288	78	Hicks	67	68	1	1.21
					69	70	1	1.59
					74	75	1	1.89
					75	76	1	4.18
					76	77	1	1.84
HDD036	273732	619266	83	Hicks	47	48	1	1.31
					48	49	1	1.05
					49	50	1	2.56
					50	51	1	1.14
					51	52	1	3.04
					52	53	1	2.82
					57	58	1	2.98
					59	60	1	1.16
					63	64	1	4.86
					64	65	1	1.53
					65	66	1	8.58
					66	67	1	7.96
					67	68	1	2.83
					69	70	1	1.34
					80	81	1	1.16
HDD037	274213	618833	63	Hicks	102	103	1	1.11
					103	104	1	1.49
					107	108	1	1.37
					108	109	1	2.73
					109	110	1	1.80
					110	111	1	2.02
					111	112	1	1.56
					112	113	1	1.14
					113	114	1	1.21
					115	116	1	2.91
					122	123	1	1.92
HDD038	274179	618882	62	Hicks	75	76	1	1.10
					76	77	1	11.15
					77	78	1	2.41
					81	82	1	1.21
HDD039	274132	618890	61	Hicks	77	78	1	2.34
					78	79	1	5.30
					79	80	1	1.91
					84	85	1	2.26
					85	86	1	3.28
					99	100	1	1.32
					103	104	1	5.03
					104	105	1	4.20
					115	116	1	2.16
					116	117	1	2.24
HDD040	274042	619061	66	Hicks	21	22	1	2.02
					22	23	1	5.80
					28	29	1	1.32
					29	30	1	1.61

					30	31	1	1.11
					31	32	1	1.09
					42	43	1	0.72
					43	44	1	1.32
HDD041	274050	619073	65	Hicks	48	49	1	2.10
					50	51	1	1.52
HDD042	274009	619080	73	Hicks	29	30	1	1.51
					31	32	1	1.16
					32	33	1	1.32
					92	93	1	3.73
HDD043	274049	618998	57	Hicks	26	27	1	1.36
					30	31	1	5.31
					35	36	1	2.98
					36	37	1	2.15
					37	38	1	1.61
					38	39	1	1.64
					39	40	1	1.58
					40	41	1	28.03
					41	42	1	1.88
					44	45	1	2.47
					45	46	1	1.50
					47	48	1	1.72
					53	54	1	1.29
					54	55	1	2.26
					55	56	1	3.89
					56	57	1	1.57
					58	59	1	1.02
					108	109	1	1.69
HDD044	274063	619009	57	Hicks	7	9	2	4.29
					9	10	1	0.74
					10	11	1	6.36
					11	12	1	6.06
					15	16	1	2.79
					16	17	1	3.01
					18	19	1	1.53
					20	21	1	2.34
					23	24	1	2.58
					24	25	1	1.05
					25	26	1	2.12
					26	27	1	4.39
					27	28	1	2.51
					28	29	1	3.33
					29	30	1	2.35
					54	55	1	3.00
					76	77	1	1.55
HDD045	273965	619017	68	Hicks	84	85	1	2.48
					85	86	1	1.57
					86	87	1	18.22
					89	90	1	5.03
					90	91	1	10.91
					91	92	1	2.12
					92	93	1	6.67
HDD046	273920	619039	71	Hicks	87	88	1	1.22
					88	89	1	1.61
					89	90	1	10.12

					90	91	1	6.38
					123	124	1	4.00
					124	125	1	4.83
					137	138	1	1.35
					138	139	1	1.75
HDD047	273912	619108	64	Hicks	35	36	1	8.87
					36	37	1	5.97
					38	39	1	1.95
					39	40	1	3.02
					42	43	1	7.86
					43	44	1	1.54
					45	46	1	1.08
					53	54	1	2.02
					55	56	1	1.27
					60	61	1	3.88
					61	62	1	0.92
HDD048	273931	619131	61	Hicks	2	3.4	1.4	3.18
					5	6	1	1.08
					18	19	1	3.80
					19	20	1	4.58
					20	20.7	0.7	1.16
					20.7	21.4	0.7	1.56
					22	23	1	1.14
					23	24	1	1.62
HDD050	273834	619121	76	Hicks	80	81	1	1.22
					82	83	1	1.42
					83	84	1	1.85
					84	85	1	1.93
					102	103	1	1.00
					122	123	1	10.82
					123	124	1	10.10
HDD051	273791	619154	81	Hicks	110	111	1	1.84
HDD052	273782	619113	82	Hicks	103	104	1	3.32
					120	121	1	1.83
					137	138	1	14.01
					138	139	1	1.56
					145	146	1	5.50
					146	147	1	5.62
					147	148	1	4.37
					149	150	1	1.17
					150	151	1	2.29
HDD053	273318	619565	64	Hicks	51	52	1	3.25
					52	53	1	2.74
					53	54	1	1.21
					54	55	1	0.38
					55	56	1	1.41
					56	57	1	1.05
HDD054	273400	619564	63	Hicks	20	20.8	0.8	1.49
HDD055	273417	619607	66	Hicks	98	99	1	2.80
					100	101	1	2.14
					102	103	1	1.84
					103	104	1	3.50
					104	105	1	2.37
					105	106	1	2.59
					106	107	1	2.02

					107	108	1	2.16
					108	109	1	3.04
					109	110	1	2.50
					110	111	1	2.08
					112	113	1	3.25
					113	114	1	2.25
					114	115	1	1.50
					115	116	1	1.22
					119	120	1	2.41
					120	121	1	2.75
					121	122	1	3.12
HDD056	273440	619532	63	Hicks	13	14	1	2.54
HDD057	273454	619550	69	Hicks	61	62	1	3.10
					77	78	1	3.96
					84	85	1	4.66
					85	86	1	3.30
HDD058	273466	619569	71	Hicks	96	97	1	2.22
					97	98	1	1.97
					98	99	1	2.84
					102	103	1	2.08
					107	108	1	1.07
					112	113	1	2.01
					113	114	1	1.69
					127	128	1	2.66
					128	129	1	1.05
					129	130	1	1.19
					131	132	1	1.16
HDD060	273508	619552	68	Hicks	126	127	1	1.51
					127	128	1	1.22
HDD061	274988	618491	85	Hicks	36	37	1	1.68
					39	40	1	1.78
					40	41	1	1.50
					41	42	1	2.52
HRC128	272348	620113	67	Hicks	20	21	1	8.51
HRC132	272168	620205	68	Hicks	30	31	1	1.57
HRC135	271651	620566	101	Hicks	78	79	1	2.38
					79	80	1	2.61
					80	81	1	1.49
					81	82	1	2.33
HRC137	272328	620085	67	Hicks	71	72	1	1.90
HRC138	271692	620536	102	Hicks	62	63	1	4.75
					94	95	1	2.61
HRC139	271640	620550	99	Hicks	79	80	1	2.44
HRC140	271610	620594	101	Hicks	63	64	1	3.70
					80	81	1	1.63
					85	86	1	1.33
					96	97	1	2.09
LDD001	273522	620420	71	Larken	67	68	1	2.30
					76	77	1	3.59
LDD002	273585	620399	73	Larken	103	104	1	7.45
LRC028	273625	620328	64	Larken	39	40	1	12.18
					49	50	1	4.12
					50	51	1	1.58
LRC029	273465	620452	64	Larken	34	35	1	1.19
					36	37	1	10.37

					37	38	1	9.00
					39	40	1	1.77
SDD060	270500	621927	92	Smarts	146	147	1	9.74
					152	153	1	6.63
					153	154	1	2.16
					154	155	1	6.60
					155	156	1	5.43
SDD061	270664	621825	77	Smarts	103	104	1	1.48
					110	111	1	1.63
					111	112	1	1.54
					112	113	1	2.55
SDD062	270516	621950	96	Smarts	118	119	1	1.74
					119	120	1	9.07
					121	122	1	2.21
					122	123	1	2.82
					126	127	1	3.37
					127	128	1	1.70
SDD063	270882	621754	79	Smarts	31	32	1	2.01
					32	33	1	5.57
					33	34	1	8.95
					34	35	1	1.75
					35	36	1	2.04
					100.8	102	1.2	2.86
SDD064	270538	621893	92	Smarts	150	151	1	3.19
					160	161	1	6.53
SDD067	270925	621726	83	Smarts	38	39	1	1.09
SDD069	270943	621754	86	Smarts	6	7	1	2.54
					7	8	1	2.01
					36	37	1	7.01
					70	71	1	19.02
					115	116	1	2.31
					116	117	1	9.18
					120	121	1	6.81
					121	122	1	6.13
					122	123	1	7.93
					123	124	1	30.76
					124	125	1	1.47
					125	126	1	3.20
					127	128	1	2.97
					128	129	1	7.12
					129	130	1	1.05
					133	135	2	1.87
					133	134	1	1.06
					134	135	1	2.67
SDD070	270582	621956	98	Smarts	43	44	1	6.15
SDD071	270989	621715	84	Smarts	68	69	1	1.44
					91	92	1	1.12
					93	94	1	34.16
					94	95	1	24.84
					98	99	1	126.54
					100	101	1	41.98
					101	102	1	1.45
					102	103	1	2.11
					103	104	1	4.83
					106	107	1	6.31

					108.4	109	0.6	3.10
					109	110	1	1.66
					111	112	1	1.40
					157	158	1	1.04
SDD072	270578	621866	87	Smarts	138	139	1	2.66
					139	140	1	1.74
					150	151	1	5.03
					153	154	1	2.12
SDD073	271035	621690	79	Smarts	12	16	4	2.60
					54	55	1	2.24
					64	65	1	3.72
					68	69	1	1.60
					96	97	1	3.52
					101	102	1	1.04
					102	103	1	9.68
					103	104	1	8.03
					104	105	1	1.02
					105	106	1	2.27
					106	107	1	4.32
					108	109	1	5.48
SDD074	270588	621880	88	Smarts	134	135	1	4.97
					157	158	1	5.26
SDD075	271040	621683	78	Smarts	16	20	4	1.47
					20	24	4	1.07
					52	53	1	1.16
					56	57	1	1.14
					57	58	1	1.65
					64	65	1	1.13
					67	68	1	2.19
					89	90	1	2.30
					90	91	1	2.49
					95	96	1	1.56
					96	97	1	8.81
					97	98	1	1.58
					99	100	1	4.81
					103	104	1	1.49
					105	106	1	1.38
					106	107	1	1.56
					107	108	1	1.06
SDD076	270600	621896	89	Smarts	95	96	1	18.39
					96	97	1	13.87
					97	98	1	7.49
					98	99	1	3.81
					108	109	1	3.04
					164	165	1	2.80
					174	175	1	1.53
SDD077	271075	621661	72	Smarts	20	21	1	2.85
					30	31	1	4.90
					75	76	1	8.39
					76	77	1	1.11
					77	78	1	4.55
					82	83	1	4.14
					84	85	1	3.17
					87	88	1	4.02
					88	89	1	15.43

					89	90	1	1.85
					90	91	1	11.14
					91	92	1	1.96
					92	93	1	4.06
					93	94	1	1.38
					95	96	1	9.89
					108	109	1	2.22
SDD078	271089	621682	75	Smarts	106	107	1	1.14
					113	114	1	3.25
					129	130	1	2.56
					130	131	1	7.26
					131	132	1	19.10
					132	133	1	1.24
SDD079	270611	621912	91	Smarts	83	84	1	1.76
					92	93	1	1.13
					150	151	1	10.19
SDD080	271136	621664	72	Smarts	12	16	4	1.41
					98	99	1	2.28
					99	100	1	3.00
					100	101	1	1.06
					134	135	1	10.24
					150	151	1	1.53
					151	152	1	2.89
					152	153	1	7.99
					153	154	1	1.84
SDD081	270639	621869	79	Smarts	58	59	1	3.70
					59	60	1	2.92
					97	98	1	2.62
SDD082	271165	621620	66	Smarts	71	72	1	3.03
					101	102	1	6.42
					124	125	1	27.44
					125	126	1	27.08
					126	127	1	11.10
					132	133	1	2.09
					149	150	1	4.78
					151	152	1	1.27
SDD083	270654	621890	78	Smarts	15	16	1	1.65
					64	65	1	2.31
					104	105	1	6.61
SDD084	271206	621599	65	Smarts	125	126	1	1.59
					126	127	1	1.38
					150	151	1	1.44
SDD085	270413	622056	103	Smarts	121	122	1	1.29
					122	123	1	1.21
					123	124	1	2.79
					124	125	1	2.34
					125	126	1	10.43
					126	127	1	13.92
					128	129	1	9.77
					129	130	1	29.26
					130	131	1	3.19
					131	132	1	2.41
					132	133	1	1.45
SDD094	270961	621779	85	Smarts	137	138	1	12.92
					144	145	1	2.70

					145	146	1	1.58
					147	148	1	1.26
					148	149	1	1.39
					150	151	1	2.11
					151	152	1	2.90
SDD095	271016	621570	65	Smarts	116	117	1	1.17
					125	126	1	1.53
					143	144	1	1.41
					148	149	1	2.13
SDD096	271060	621649	65	Smarts	16	17	1	1.51
					28	29	1	1.52
SDD097	270940	621625	67	Smarts	64	65	1	0.73
					71	72	1	1.68
					72	73	1	3.81
					73	74	1	1.46
					74	75	1	15.92
					111	112	1	1.17
					112	113	1	2.32
SDD099	270945	621760	86	Smarts	104	105	1	1.01
					115	116	1	2.99
SDD100	271139	621568	69	Smarts	17	18	1	1.90
SDD101	271016	621570	65	Smarts	61	62	1	1.34
SDD102A	270409	622065	103	Smarts	99.7	100	0.3	1.83
					100	101	1	2.69
SDD104	270477	621892	83	Smarts	206	207	1	4.26
					207	208	1	1.44
					209	210	1	1.20
SDD105	270268	622152	102	Smarts	115	116	1	1.00
					116	117	1	2.04
					118	119	1	4.12
SDD106	270480	622060	104	Smarts	25	26	1	27.58
					26	27	1	7.71
					30	31	1	3.10
					33	34	1	1.70
					34	35	1	1.19
					44	45	1	1.57
					47	48	1	5.23
SDD109	270771	621696	70	Smarts	140	141	1	17.01
					141	142	1	14.66
					142	143	1	1.94
					192	193	1	2.03
SDD111	270657	621816	76	Smarts	95	96	1	3.00
					96	97	1	3.18
					97	98	1	89.33
					98	99	1	3.02
					105	106	1	1.74
					122	123	1	102.64
					133	134	1	1.66
					134	135	1	1.43
					135	136	1	11.57
					142	143	1	2.20
					154	155	1	7.70
					155	156	1	1.78
SDD113	270489	621911	89	Smarts	170	171	1	28.96
					171	172	1	3.81

					172	173	1	2.59
					174	175	1	1.13
					179	180	1	12.97
SDD115	270464	621961	92	Smarts	161	162	1	9.10
					164	165	1	1.27
					165	166	1	30.70
					166	167	1	63.53
					167	168	1	2.60
SDD116	270563	621760	71	Smarts	138	139	1	1.91
					297	298	1	9.44
					300	301	1	4.44
					325	326	1	3.01
					327	328	1	2.84
					328	329	1	7.95
					330	331	1	3.09
SDD118	270683	621761	72	Smarts	197	198	1	2.51
					213	214	1	4.65
					214	215	1	1.33
					215	216	1	6.94
					216	217	1	1.72
					217	218	1	2.53
					218	219	1	3.50
					219	220	1	121.56
					229	230	1	1.35
					235	236	1	1.11
					244	245	1	3.15
SDD119	270369	622021	101	Smarts	195	196	1	1.11
SDD120	271105	621699	77	Smarts	156	157	1	1.46
					161	162	1	1.11
					167	168	1	22.27
					168	169	1	7.22
					169	170	1	1.63
					170	171	1	2.54
					171	172	1	1.07
					173	174	1	1.12
					181	182	1	3.33
SDD121	270319	622036	103	Smarts	198	199	1	11.59
SDD122	271156	621689	71	Smarts	221	222	1	5.28
SDD123	270712	621698	70	Smarts	216	217	1	3.84
					245	246	1	2.54
					246	247	1	9.60
					247	248	1	3.24
					248	249	1	2.37
					249	250	1	1.80
					252	253	1	3.94
					261	262	1	9.99
SDD124	271859	621047	76	Smarts	74	75	1	2.76
SDD125	270648	621711	73	Smarts	273	274	1	15.00
					277	278	1	1.45
					278	279	1	6.20
					279	280	1	2.55
					280	281	1	10.12
					281	282	1	20.21
					282	283	1	1.56
					283	284	1	6.43

					284	285	1	7.22
					285	286	1	1.30
					286	287	1	1.43
					292	293	1	12.99
					297	298	1	3.17
					298	299	1	8.59
					299	300	1	2.21
SDD126	270561	621843	85	Smarts	173	174	1	3.77
					182	183	1	1.61
					186	187	1	1.82
SDD127	270598	621812	80	Smarts	185	186	1	1.13
					186	187	1	7.12
					197	198	1	3.87
					251	252	1	3.93
					255	256	1	2.52
					269	270	1	1.20
SDD128	270622	621765	72	Smarts	207	208	1	12.11
					208	209	1	38.15
					212	213	1	1.17
					213	214	1	30.51
					214	215	1	7.31
					215	216	1	5.01
					217	218	1	1.67
					218	219	1	1.51
					221	222	1	1.26
					222	223	1	5.78
					223	224	1	1.20
					225	226	1	21.03
					226	227	1	2.11
					230	231	1	7.39
					231	232	1	2.38
					232	233	1	3.27
					233	234	1	1.15
					244	245	1	1.60
					245	246	1	37.59
					248	249	1	2.54
					251	252	1	5.77
					256	257	1	3.28
					257	258	1	9.82
					258	259	1	27.96
					259	260	1	3.97
					261	262	1	2.65
					267	268	1	12.68
					268	269	1	7.36
					269	270	1	10.82
					270	271	1	3.66
					271	272	1	2.15
SDD129	270265	622060	103	Smarts	214	215	1	5.28
					215	216	1	10.31
					218	219	1	3.24
					219	220	1	7.58
					220	221	1	4.28
SDD130	270524	621874	88	Smarts	190	191	1	9.18
SDD131	270591	621721	71	Smarts	304	305	1	1.52
					308	309	1	1.17

					312	313	1	16.33
					316	317	1	1.15
					317	318	1	21.19
					318	319	1	2.50
					319	320	1	1.19
					320	321	1	2.03
					321	322	1	7.20
					322	323	1	1.45
					327	328	1	4.75
SDD133	270339	621979	96	Smarts	257	258	1	3.02
					340	341	1	4.11
SDD134	270627	621680	75	Smarts	328	329	1	23.17
					329	330	1	9.90
					330	331	1	8.46
					331	332	1	2.61
					333	334	1	4.21
					334	335	1	6.64
					335	336	1	2.90
					336	337	1	3.84
					356	357	1	2.35
					357	358	1	1.36
					359	360	1	3.52
SDD135	270395	621954	83	Smarts	211	212	1	5.80
SDD137	270499	621837	79	Smarts	258	259	1	4.21
					268	269	1	3.00
					275	276	1	1.21
					287	288	1	14.35
					288	289	1	6.25
					289	290	1	4.92
SDD138	270683	621651	73	Smarts	336	337	1	1.16
					337	338	1	3.97
					338	339	1	5.74
					341	342	1	2.67
					342	343	1	6.01
SDD139	270528	621794	73	Smarts	229	230	1	4.92
					236	237	1	5.92
					238	239	1	1.88
					244	245	1	40.99
					245	246	1	8.38
					262	263	1	2.88
SDD140	270745	621659	69	Smarts	251	252	1	3.63
					252	253	1	3.96
					258	259	1	3.71
					259	260	1	2.71
					261	262	1	1.33
SDD142	271873	621061	86	Smarts	49	50	1	1.67
SDD143	271787	621144	68	Smarts	27	28	1	1.75
SDD145	271760	621149	69	Smarts	42	43	1	6.12
					43	44	1	7.08
					44	45	1	8.08
					48	49	1	1.14
SDD146	271743	621162	72	Smarts	35	36	1	20.50
					36	37	1	4.59
					38	39	1	3.77
					39	40	1	5.12

SRC691	270407	622076	103	Smarts	68	69	1	1.08
					69	70	1	4.89
					70	71	1	3.03
					71	72	1	13.01
					72	73	1	5.52
					73	74	1	3.29
					74	75	1	9.35
					75	76	1	2.48
					76	77	1	11.04
					77	78	1	1.50
					78	79	1	1.58
					79	80	1	1.56
SRC693	270536	621979	100	Smarts	32	33	1	7.35
					66	67	1	1.90
					67	68	1	4.06
					68	69	1	5.80
					82	83	1	9.48
SRC694	270623	621932	92	Smarts	34	35	1	55.02
					35	36	1	41.85
					36	37	1	66.64
					37	38	1	5.60
					38	39	1	32.35
					39	40	1	3.96
					51	52	1	24.75
					54	55	1	6.43
					66	67	1	1.26
SRC695	270637	621952	95	Smarts	83	85	2	0.96
					83	84	1	0.66
					84	85	1	1.27
					88	89	1	3.36
SRC696	270670	621916	81	Smarts	11	12	1	0.66
					22	23	1	11.90
					66	67	1	4.65
					88	91	3	10.87
SRC697	270699	621875	73	Smarts	34	35	1	6.50
					35	36	1	4.68
					36	37	1	1.90
					37	38	1	12.81
					38	39	1	10.92
					39	40	1	14.75
					40	41	1	1.81
					73	76	3	1.38
SRC698	270743	621843	71	Smarts	10	11	1	9.40
					11	12	1	66.77
					12	13	1	10.47
					13	14	1	4.76
					14	15	1	1.13
					15	16	1	1.63
					16	17	1	1.82
					27	28	1	1.10
					28	29	1	1.47
					30	31	1	1.15
					72	73	1	3.25
					86	89	3	1.63
SRC699	270582	621952	98	Smarts	48	49	1	1.97

					49	50	1	1.02
					70	73	3	2.88
SRC700	270678	621947	92	Smarts	39	42	3	1.14
SRC701	270922	621728	83	Smarts	38	39	1	1.74
					41	42	1	1.86
					76	79	3	2.73
					95	96	1	2.84
					96	97	1	3.33
					97	98	1	2.82
					98	99	1	7.89
					99	100	1	3.05
SRC702	270952	621667	76	Smarts	84	85	1	2.98
SRC703	271104	621620	69	Smarts	28	29	1	2.44
					53	54	1	10.05
					54	55	1	9.78
					55	56	1	6.42
					58	59	1	7.21
					59	60	1	11.45
					60	61	1	1.53
					69	70	1	11.25
					70	71	1	1.32
					71	72	1	3.92
					72	73	1	1.63
SRC704	271058	621639	65	Smarts	4	7	3	1.09
					24	25	1	2.49
					42	43	1	2.63
					52	53	1	2.62
					53	54	1	8.66
					54	55	1	3.60
					55	56	1	2.41
					56	57	1	1.38
SRC705	271044	621618	65	Smarts	11	12	1	37.64
					12	13	1	4.41
					21	22	1	3.70
SRC706	271192	621571	64	Smarts	37	38	1	1.72
					38	39	1	1.28
					47	48	1	1.38
					113	114	1	1.07
SRC707	271170	621603	66	Smarts	5	7	2	4.33
					52	53	1	9.86
					53	54	1	1.81
					55	56	1	1.61
					57	58	1	1.42
					61	62	1	2.19
					64	65	1	2.05
SRC708	271165	621588	68	Smarts	30	31	1	4.72
					35	36	1	2.55
					48	49	1	6.97
					49	50	1	102.81
					50	51	1	9.02
					52	53	1	1.39
SRC709	271159	621561	67	Smarts	17	18	1	7.42
					18	19	1	4.06
					19	20	1	1.33
					20	21	1	5.16

					21	22	1	1.79
					22	23	1	1.24
					23	24	1	3.21
					24	25	1	1.33
					32	33	1	1.21
SRC710	271111	621578	69	Smarts	11	12	1	4.39
					15	16	1	10.02
					16	17	1	7.32
					18	19	1	4.46
					19	20	1	8.32
SRC711	271125	621601	68	Smarts	45	46	1	3.07
					46	47	1	12.32
					47	48	1	84.02
					48	49	1	62.34
					49	50	1	5.38
					50	51	1	1.71
					57	58	1	2.37
					58	59	1	3.32
SRC712	271140	621623	69	Smarts	13	14	1	1.40
					32	33	1	2.96
SRC712A	271138	621619	69	Smarts	29	30	1	2.56
					70	71	1	2.26
					74	75	1	1.72
					79	80	1	1.22
					80	81	1	3.28
					81	82	1	1.57
					84	85	1	5.93
					96	97	1	2.07
					99	100	1	4.26
SRC713	270984	621657	76	Smarts	17	18	1	1.89
					18	19	1	8.29
					22	23	1	12.90
SRC714	270942	621687	79	Smarts	9	10	1	4.59
					10	11	1	1.25
					11	12	1	14.02
					13	14	1	1.25
					20	21	1	3.19
SRC716	270528	622005	102	Smarts	43	44	1	2.91
					44	45	1	4.49
					51	52	1	3.80
					52	53	1	12.37
					53	54	1	71.24
					54	55	1	2.80
					56	57	1	10.44
					57	58	1	4.26
					58	59	1	4.12
					61	62	1	5.67
					66	67	1	3.96
SRC717	271105	621654	72	Smarts	40	41	1	1.94
					41	42	1	1.37
					84	85	1	1.19
					85	86	1	3.25
					86	87	1	10.59
					87	88	1	2.77
					88	89	1	3.14

					91	92	1	3.13
					92	93	1	1.69
					93	94	1	2.52
					95	96	1	1.65
					96	97	1	2.09
SRC718	271088	621630	68	Smarts	46	47	1	13.24
					47	48	1	2.44
					48	49	1	11.12
					49	50	1	2.23
					50	51	1	2.86
					51	52	1	1.77
					52	53	1	2.95
					53	54	1	4.68
					54	55	1	1.90
					55	56	1	6.45
					56	57	1	2.88
					57	58	1	2.11
					67	68	1	54.70
					68	69	1	10.88
					69	70	1	3.42
					70	71	1	1.64
					84	85	1	1.31
					86	87	1	1.43
					88	89	1	1.28
SRC719	271071	621606	65	Smarts	14	15	1	2.67
					18	19	1	1.81
					19	20	1	1.91
SRC720	271054	621668	74	Smarts	14	15	1	1.16
					15	16	1	3.47
					47	48	1	23.31
					48	49	1	16.62
					49	50	1	5.84
					51	52	1	3.54
					68	69	1	1.29
					75	76	1	5.07
					77	78	1	3.56
					79	80	1	1.28
					86	87	1	1.93
					87	88	1	1.72
					88	89	1	1.84
					89	90	1	3.82
					90	91	1	7.63
					91	92	1	10.73
SRC721	271042	621654	69	Smarts	10	11	1	28.38
					11	12	1	3.56
					31	32	1	2.09
					50	51	1	1.21
					64	65	1	1.53
					65	66	1	1.60
					66	67	1	2.11
SRC722	271031	621635	66	Smarts	13	14	1	3.10
SRC723	270998	621680	79	Smarts	49	50	1	1.00
					50	51	1	1.82
					51	52	1	2.03
					52	53	1	9.93

					53	54	1	2.81
					64	65	1	1.82
					65	66	1	1.91
					67	68	1	1.01
					83	84	1	1.99
					84	85	1	4.59
					86	87	1	10.94
					87	88	1	35.90
					88	89	1	1.49
					91	92	1	1.93
					92	93	1	1.92
					93	94	1	2.10
					94	95	1	1.54
					98	99	1	2.63
					99	100	1	2.06
					102	103	1	1.19
					103	104	1	3.86
SRC724	271016	621704	82	Smarts	8	9	1	2.27
					36	40	4	1.64
					36	37	1	1.44
					37	38	1	2.16
					39	40	1	2.43
					54	55	1	2.41
					62	63	1	1.85
					91	92	1	1.82
					94	95	1	10.48
					95	96	1	23.07
					96	97	1	8.73
					97	98	1	2.13
					98	99	1	3.93
					99	100	1	9.80
					100	101	1	1.32
					104	105	1	1.13
					105	106	1	3.01
					106	107	1	2.80
					108	109	1	1.99
					109	110	1	1.58
SRC725	270955	621709	82	Smarts	7	8	1	5.68
					8	9	1	3.14
					9	10	1	2.62
					93	94	1	4.45
					94	95	1	66.53
					95	96	1	138.55
					96	97	1	27.42
					97	98	1	4.12
					101	102	1	2.07
					102	103	1	1.18
					104	105	1	1.16
					105	106	1	1.75
					108	109	1	1.03
SRC726	270969	621729	86	Smarts	21	22	1	1.05
					22	23	1	2.22
					24	25	1	1.48
					44	45	1	1.47
					88	89	1	2.33

					94	95	1	2.32
					95	96	1	3.40
					96	97	1	1.98
					99	100	1	2.35
					100	101	1	3.66
					102	103	1	5.68
					105	106	1	13.41
					106	107	1	17.16
					107	108	1	10.39
					108	109	1	2.26
					110	111	1	9.62
					113	114	1	4.50
					114	115	1	7.21
					115	116	1	2.87
					121	122	1	1.38
					122	123	1	13.94
					127	128	1	3.96
					128	129	1	17.86
					129	130	1	3.78
SRC727	270981	621747	87	Smarts	61	62	1	2.30
					62	63	1	9.57
					63	64	1	2.67
					70	71	1	6.87
					73	74	1	11.56
					76	77	1	2.77
					81	82	1	1.15
SRC729	270904	621738	81	Smarts	25	26	1	1.37
					26	27	1	7.04
					29	30	1	11.83
					73	74	1	3.81
					99	100	1	2.57
					100	101	1	1.86
SRC730	270916	621755	83	Smarts	11	12	1	5.71
					12	13	1	1.32
					13	14	1	1.55
					15	16	1	1.71
					88	89	1	10.80
					89	90	1	3.45
					90	91	1	3.28
					101	102	1	3.18
					102	103	1	3.53
					103	104	1	24.57
					104	105	1	17.46
					105	106	1	17.57
					106	107	1	1.45
					107	108	1	8.20
					108	109	1	5.38
					109	110	1	1.39
					110	111	1	0.27
					111	112	1	5.28
					112	113	1	1.72
					113	114	1	1.72
					114	115	1	48.86
					115	116	1	1.68
					116	117	1	11.90

					117	118	1	4.71
					118	119	1	0.65
					119	120	1	1.66
SRC731	270862	621771	75	Smarts	20	21	1	10.54
					21	22	1	9.01
					22	23	1	1.44
					25	26	1	5.72
					62	63	1	5.88
SRC732	270874	621789	76	Smarts	18	19	1	3.17
					19	20	1	1.58
					23	24	1	1.56
					53	54	1	7.94
					54	55	1	2.04
					55	56	1	1.60
					56	57	1	1.19
					57	58	1	1.40
					92	93	1	2.97
					96	97	1	1.81
					114	115	1	1.72
					120	121	1	2.31
SRC733	270888	621808	78	Smarts	26	27	1	5.86
					28	29	1	1.96
					30	31	1	3.16
					63	64	1	2.42
					73	74	1	2.20
SRC734	270829	621826	81	Smarts	14	15	1	1.41
					34	35	1	393.15
					35	36	1	28.35
					36	37	1	9.93
					37	38	1	27.73
					39	40	1	1.57
					55	56	1	4.08
					56	57	1	10.93
					57	58	1	51.90
					58	59	1	1.97
					63	64	1	1.79
					76	77	1	17.44
					79	80	1	11.41
					80	81	1	3.81
					81	82	1	15.55
					82	83	1	1.41
					84	85	1	4.47
					90	91	1	1.76
					91	92	1	25.32
					92	93	1	1.88
					94	95	1	12.11
					97	98	1	13.84
					108	109	1	11.09
					112	113	1	3.03
					113	114	1	62.69
					114	115	1	4.43
					118	119	1	1.75
					119	120	1	5.07
					120	121	1	1.33
SRC735	270695	621830	70	Smarts	37	38	1	1.61

					40	41	1	25.19
					41	42	1	3.33
					62	63	1	1.61
					73	74	1	26.99
					74	75	1	4.01
					75	76	1	3.54
					76	77	1	2.04
SRC736	270711	621852	71	Smarts	44	45	1	2.19
					52	53	1	2.61
SRC738	270707	621886	77	Smarts	17	18	1	3.68
					19	20	1	1.04
					42	43	1	1.44
					61	62	1	1.16
					66	67	1	5.21
					79	80	1	1.65
					80	81	1	1.34
					90	91	1	1.36
SRC739	270667	621867	71	Smarts	64	65	1	1.77
SRC741	270692	621902	79	Smarts	13	14	1	2.62
					14	15	1	2.17
					16	17	1	2.20
					53	54	1	2.42
					54	55	1	2.02
SRC742	270651	621928	90	Smarts	69	70	1	5.53
SRC743	270628	621894	86	Smarts	35	36	1	3.52
					36	37	1	32.39
					37	38	1	2.63
					38	39	1	1.50
					40	41	1	6.70
					43	44	1	6.44
					45	46	1	1.83
SRC744	270589	621924	94	Smarts	87	90	3	1.59
					91	92	1	1.60
					92	93	1	1.77
					93	94	1	3.07
SRC745	270601	621940	95	Smarts	40	41	1	3.40
					41	42	1	2.57
					91	92	1	2.29
SRC746	270617	621960	97	Smarts	46	47	1	3.01
					49	50	1	3.71
SRC747	270572	622029	104	Smarts	37	38	1	3.64
					40	41	1	1.51
					42	43	1	8.04
					43	44	1	37.73
					44	45	1	5.24
					48	49	1	3.07
					55	56	1	2.12
					56	57	1	1.31
					57	58	1	1.03
					58	59	1	4.91
					59	60	1	23.44
					60	61	1	1.66
SRC748	270501	621967	98	Smarts	86	87	1	2.27
					87	88	1	53.30
					88	89	1	10.80

					89	90	1	13.24
					90	91	1	2.12
					91	92	1	2.77
					92	93	1	17.73
					93	94	1	9.77
					94	95	1	9.79
					95	96	1	1.59
					105	106	1	1.29
					106	107	1	4.16
					108	109	1	1.62
					109	110	1	1.84
					110	111	1	1.98
					111	112	1	8.25
SRC750	270521	621996	100	Smarts	38	39	1	1.25
					48	49	1	5.06
					50	51	1	5.06
					54	55	1	4.05
					57	58	1	4.35
					58	59	1	1.07
					59	60	1	9.15
SRC751	270465	622000	99	Smarts	98	99	1	12.33
					99	100	1	76.13
					102	103	1	1.32
					104	105	1	2.41
					106	107	1	25.41
					107	108	1	50.38
					108	109	1	3.24
					109	110	1	2.55
					113	114	1	1.38
SRC752	270478	622018	101	Smarts	76	77	1	1.46
					80	81	1	8.41
					81	82	1	13.54
					82	83	1	1.31
					87	88	1	2.15
SRC753	270490	622036	103	Smarts	27	28	1	1.78
					28	29	1	2.35
					29	30	1	1.93
					33	34	1	10.72
					44	45	1	2.23
					53	54	1	8.34
					57	58	1	6.66
					58	59	1	1.29
SRC754	270422	622035	103	Smarts	89	90	1	24.00
					90	91	1	8.89
					91	92	1	2.81
					96	97	1	9.88
					97	98	1	3.86
					98	99	1	1.28
					100	101	1	11.03
					101	102	1	3.77
					104	105	1	17.97
					105	106	1	44.72
					107	108	1	3.86
					108	109	1	12.59
					109	110	1	25.80

					111	112	1	14.18
					112	113	1	1.47
					114	115	1	1.08
					115	116	1	2.50
SRC755	270434	622056	104	Smarts	46	47	1	2.14
					47	48	1	10.21
					49	50	1	1.14
					56	57	1	2.00
					57	58	1	6.67
					58	59	1	14.01
					63	64	1	12.11
					78	79	1	2.72
					80	81	1	1.59
SRC756	270447	622073	103	Smarts	28	29	1	2.07
					33	34	1	2.26
					34	35	1	5.26
					40	41	1	1.55
					41	42	1	7.63
SRC757	270377	622075	103	Smarts	101	102	1	11.58
					102	103	1	4.47
					103	104	1	5.12
					104	105	1	2.23
SRC758	270391	622097	103	Smarts	62	63	1	4.14
					63	64	1	19.96
					64	65	1	5.96
					65	66	1	1.62
					66	67	1	2.84
					70	71	1	11.85
SRC759	270404	622114	102	Smarts	27	28	1	26.34
					28	29	1	9.34
					29	30	1	8.40
					30	31	1	4.44
					31	32	1	41.68
					32	33	1	15.48
					33	34	1	11.79
					34	35	1	3.25
					35	36	1	7.44
					36	37	1	15.93
					37	38	1	1.24
					38	39	1	3.25
					39	40	1	1.38
					43	44	1	4.67
					44	45	1	2.44
SRC760A	270387	622134	101	Smarts	28	29	1	3.37
					29	30	1	1.85
					30	31	1	2.19
					31	32	1	8.67
					32	33	1	71.53
					33	34	1	30.80
					34	35	1	53.69
					35	36	1	51.56
					36	37	1	13.47
					37	38	1	4.01
					43	44	1	3.35
SRC761	270343	622114	102	Smarts	28	29	1	1.03

					86	87	1	1.99
SRC762	270355	622131	102	Smarts	44	45	1	3.61
					45	46	1	44.48
					46	47	1	40.01
					47	48	1	24.14
					48	49	1	28.15
					49	50	1	2.18
					51	52	1	3.57
					52	53	1	9.15
					53	54	1	1.21
					55	56	1	1.56
SRC763	270367	622147	101	Smarts	25	26	1	92.34
					26	27	1	23.40
					27	28	1	19.96
					28	29	1	14.18
					29	30	1	2.43
					30	31	1	9.63
					31	32	1	10.21
					32	33	1	3.12
					33	34	1	3.21
					36	37	1	1.93
					37	38	1	1.82
					38	39	1	1.79
					41	42	1	1.57
SRC764	270278	622168	101	Smarts	48	49	1	2.02
					80	81	1	1.13
					81	82	1	1.00
SRC765	270289	622182	101	Smarts	54	55	1	1.18
					55	56	1	4.69
					56	57	1	15.50
					57	58	1	27.92
					58	59	1	4.59
					59	60	1	1.36
					62	63	1	5.94
SRC766	270301	622200	100	Smarts	24	25	1	1.33
					26	27	1	10.51
					27	28	1	2.11
					30	31	1	1.12
					69	70	1	2.19
					70	71	1	1.11
SRC767	270510	621980	99	Smarts	30	31	1	1.71
					65	66	1	2.95
					66	67	1	1.82
					83	84	1	1.79
					84	85	1	2.31
					86	87	1	3.78
					87	88	1	4.70
					88	89	1	1.38
					90	91	1	2.79
					91	92	1	3.90
SRC768	270519	621995	100	Smarts	31	32	1	1.18
					32	33	1	6.12
					33	34	1	1.45
					34	35	1	1.82
					35	36	1	27.45

					36	37	1	74.98
					37	38	1	5.80
					38	39	1	14.58
					39	40	1	1.70
					41	42	1	44.45
					42	43	1	3.59
					43	44	1	1.32
					48	49	1	16.25
					73	74	1	2.51
SRC769	270638	621911	89	Smarts	42	43	1	8.41
					53	54	1	3.95
					54	55	1	9.63
					55	56	1	33.82
					56	57	1	12.14
SRC770	271916	621011	86	Smarts	46	47	1	1.43
					52	53	1	3.34
					60	61	1	3.16
					61	62	1	8.63
					62	63	1	1.62

10.10 Data Excluded

Trench information was used to guide mineralisation wireframe interpretation however was excluded from the estimation process at both Hicks and Smarts. This reflected concerns with the quality, age and location of the data. In general the trench information agreed with the structural interpretation from drilling at deeper levels.

At Smarts RC drillhole SRC321 was also excluded from the estimation as results conflicted with surrounding holes SRC009 and SDT008.

11. Sample Preparation, Analyses and Security

The sampling preparation, analyses and security described in this section relates specifically to the drilling campaigns since July 2013 by Troy.

11.1 Sample Interval Selection and Mineralisation Controls

A sample interval of 1m has been selected for the RC and Diamond Core drilling with proximity to gold mineralisation (buffer zone). This sample spacing ensures a representative sample weight is collected at a scale sufficient to define geological and mineralisation boundaries. The 1m samples are assayed at 1m intervals in visibly conspicuous mineralisation or otherwise composited to 3m intervals before assay. Any low grade internal zones are also assayed at 1m intervals and a sample buffer is placed before and after the mineralisation boundary to ensure the assays do not begin or end within high-grade mineralisation. The original 1m samples are sent for assay where any significant gold assay grades are recorded for the 3m composite samples.

The use of a 1m sample interval was selected after consideration of the following:

- Consideration of previous sampling methodology.
- The RC drilling method and sample collection process for current drill campaigns.
- A representative sample weight suitable for transport, laboratory preparation and analysis.
- The lithological thickness of the White Sands Formation and underlying basement lithology.
- A mineralisation zone thickness ranging from several metres to tens of metres.
- Suitability for statistical analysis. A standard sample length ensures all assay results are treated on equal support when reviewing assay statistics (before sample compositing for geostatistical analysis and resource estimation).
- The Diamond Core and RC drilling method will in general provide superior sample collection compared to open-hole drill methods (e.g. auger or RAB) and reduce the possibility of down-hole grade smearing or contamination.

11.2 RC Drilling Sampling Methodology

All RC drilling to date used a 5 to 5 ½" face hammer with samples collected mainly at 1m or 3m intervals. The rig geologist determines if the sampling is to be done at 1m increments or composited into a 3m interval. Sample pertaining to mineralised composites returning >0.25g/t gold are retrieved and these 1m samples are dried, presplit and submitted for assay.

A sample sheet is completed on the rig by the geologist that corresponds to a sample ticket book. Data such as down hole interval, sample identification number, sample method, sample quality, bulk bag weights, water content and QA/QC sample checks are recorded on both the sample sheet and ticket book. This includes the name of the sampling supervisor.

The rig geologist determines the placement of duplicate samples, standards and blanks to be inserted. Sample identification numbers designated for standards or blanks are written on the bags with the bags left empty. These QA/QC check samples are inserted at frequency of no less than 5% standard, 5% blanks and 10% duplicate.

The downhole samples are recovered from the drill rig cyclone and collected in UV-stabilized plastic sample bags. Each plastic bag is labelled with the relevant drillhole name and sample interval. An identification plate made from non-perishable aluminium is placed inside each bag and the bag is sealed to prevent spillage.

Each sample bag is then split at the drill site to provide a representative sample for assay analysis. The representative sample is collected in a pre-numbered calico bag after passing through a three tier riffle splitter supplied by Troy. The sample splitters are made in Australia and certified to industry standards. In cases of wet samples Troy practice was to collect a "spear" sample as moist samples clog the splitter. Troy has implemented a system of drying all moist samples and then splitting with riffle splitter.

Holes are then assayed at 1m intervals in visibly conspicuous mineralisation or otherwise composited to 3m intervals. While not always weighed, samples sent to the laboratory are in the order of three to four kilograms. The coarse field reject is stored on-site until assay results are received and validated. Troy recognises that spearing is a relatively poor quality method of splitting compared to riffle or cone splitting. Therefore spear

samples which return assays >0.2g/t gold are re-assayed by drying the coarse field reject, riffle splitting a sample when dry and re-sending for laboratory analysis.

All samples are allocated a unique sample identification number. The number is written on the cloth sample bag and a weather proof paper tear out from the ticket book with the same identification number is placed within each cotton sample bag. The cotton sample bags are lined with plastic to avoid leakage and possible contamination issues from wet or damp samples. Samples are then weighed and range from 1kg minimum to 7kg maximum.

All samples are collected from the drill site each day and taken to the sample processing shed at Hicks exploration camp (see Figure 11-1). Typically, one hole is collated into one assay batch run.

Figure 11-1: RC Samples Awaiting Dispatch at the Hicks Exploration Camp

The geologist creates a chain-of-custody dispatch before off-site sample transport. Included in these documents is the sample batch numbers loaded on the transport truck. The transport driver signs off on the dispatch which is also checked and signed for by the receiving laboratory. This signed copy identifying that all samples sent off-site were also received by the laboratory is then sent back to Troy for record keeping.

11.3 Diamond Drilling Sampling Methodology

Diamond Core drilling is mainly done using HQ and HQ3 sized equipment and minor PQ, with samples collected primarily at 1m lengths. Diamond Core is retrieved twice daily from the drill site by a geologist and secured in the back of a Landcruiser pick up using ratchet straps. All core boxes are identified with the drillhole number and down hole from and to intervals in permanent marker pen.

After geological and geotechnical logging takes place the core is photographed wet and dry. Photos include sample intervals and assay sample numbers identified from tickets in the core trays. All photos are checked for quality before the core is cut.

The core is placed in the saw cradle facing downhole (see Figure 11-2). If the core is oriented, the core is cut perpendicular to the orientation line. For consistency the orientation line is placed on the left hand side of the saw blade with the right hand side of the core taken for assay sample. The ½ core marked by the orientation line is retained in the core box. If there is no orientation line for reference, the core is still placed in the saw cradle facing down hole with the right hand side taken for assay and the left retained in the core box.

Once the core is halved, it is sampled every 1m (unless otherwise noted). Blanks and standards are inserted at a rate of not less than one every 20 samples. Halved core samples for analysis are placed in a cotton sample bag and identified with the sample number (see Figure 11-3). A sample number ticket is also taken from the sample ticket book and placed inside each bag. Sample ticket books are retained on site at the Camp. Samples are laid out in rows by sample identification and grouped by drill hole number.

All core boxes with the retained ½ core are stacked according to drill hole number and retained on site at the camp.

Figure 11-2: Core Cutting at the Hicks Exploration Shed

Figure 11-3: Core Samples Awaiting Dispatch at the Hicks Exploration Shed

Historic Diamond Core was cut in half using a core saw and sampled to 1m intervals or to geological boundaries, although some larger composites were taken in barren zones. Detailed sampling procedures for the historic core drilling are not available.

11.4 Auger Sampling Methodology

Most auger sampling programs are conducted from camps remote from the Hicks Camp and accordingly, auger samples are collected on a weekly basis. When the auger samples arrive at the sample shed, the samples and data sheets are handed to a senior crew leader for dispatch preparation.

The hole number and sample range is written on a tri-pod and placed at the head of the batch run. These are checked against the data sheets and ticket books by the supervising sample crew leader and checked a second time by a senior geologist. Concurrent with the checking of sample intervals and sample numbers, blanks and standards are inserted into the empty numbered cloth bags by the geologist. All samples are placed in large plastic lined poly-weave bags that have been pre-labelled with the sample range and bag number and the assay lab address.

11.5 Sample Chain of Custody and Transport from Camp

All samples are loaded into large plastic bags on site and freighted to the laboratory under the supervision of Troy personnel. The plastic bags are sealed with metal ties and sample transport is provided by contractors. A Chain of Custody form is filled in by the Senior Geologist as a summary of all samples leaving site on each sample dispatch. Included in this form is the batch number, sample range, the date of dispatch, laboratory destination and a check box. This form is signed by the supervising geologist and also signed by the transport contractor. The vehicle registration is also noted. All samples are subsequently loaded onto the truck and are transported to Georgetown. All documentation is emailed to the destination assay laboratory. The assay laboratories are independent of Troy.

Pharsalus Gold has previously employed two sample preparation laboratories within Guyana to prepare samples for analysis: -

1. Acme Laboratory, located at Lot 13 Plantation Non Pariel, East Coast Demerara; and

2. ActLabs Guyana, located at 27/28 Parcel Beterverwagting, Industrial Area, East Coast Demerara.

More recently, Troy has contracted the ActLabs Guyana laboratory has been used for sample preparation.

On receipt of samples, the preparation laboratories are required to check that each sample on the Dispatch Requisition has been received and match the authorized Chain of Custody document. The laboratory undertakes reconciliation between physical sample numbers and received paperwork before sample crushing. Troy is advised by the laboratory of any issues such as missing or additional samples, insufficient sample mass for analysis or problems with sample identification. The preparation laboratory manager then authorizes the batch by signing the Chain of Custody document and returns it to Troy's office in Georgetown for filing.

Quality assurance samples are inserted by Troy personal prior to sample transport to the preparation laboratories. Quality assurance is undertaken on a systematic basis with blanks, standards and field duplicates inserted at regular intervals. Troy's, and previously Azimuth's, Quality Assurance and Quality Control (QAQC) protocol entails two QAQC samples are inserted for every 10 samples. The QAQC samples consist of a field duplicate sample and either a standard or blank. The standards and blanks are inserted in an alternating sequence.

Chain of custody and sample transport procedures are incomplete for historic drilling at the Project. As such Troy is not aware of any issues that may be present for historic drilling. The current chain of custody and sample transport procedures developed for the Karouni Gold Project drilling and exploration to be adequate.

11.6 Assay Sample Preparation from Drilling Samples

For the Hicks Deposit, sample preparation was mainly conducted by ActLabs Guyana at their Georgetown facility (East Coast Demerara). Some of the earlier Hicks samples were prepared at ACME Guyana at their Georgetown facility (East Coast Demerara). For the Smarts Deposit, sample preparation was conducted by ActLabs Guyana at their Georgetown facility (East Coast Demerara).

Batches of individual samples are loaded into sacks and sealed on site, and transported by Company vehicle to the appropriate sample preparation facility. Dispatch sheets accompany each shipment, with a copy retained on site. The laboratory undertakes sample reconciliation with the dispatch sheet upon arrival of the samples. Any significant differences are flagged with Troy for resolution. The laboratory is responsible for assay sample preparation and the subsequent assay procedure.

In the author's opinion the sample preparation, security and analytical procedures are appropriate for the deposit style, at industry standard and will provide results suitable for Resource estimation.

11.7 Methodology of Sample Assay Analysis for Drilling Samples

Prior to 2012, assays of drilling samples for the Smarts and Hicks Deposits were mainly performed by ActLabs in Chile. In January 2012 ActLabs commissioned their assay laboratory in Guyana and ISO certification was received in June 2012 (see Figure 11-4 and Figure 11-5). The same assay methodology has been used at both ActLabs laboratories. P. Doyle visited the ActLabs laboratories in May 2013 and again in March 2014.

Figure 11-4: Furnaces at ActLabs Guyana

Samples were initially assayed using a 30g aliquot for fire assay with AAS finish. Assayed samples returning >3g/t gold were repeat fire assayed using a 30g charge with gravimetric finish. The fire assay with gravimetric finish is the preferred result for subsequent use. Since mid-year 2013 a 50g fire assay with AAS finish has be adopted as the standard analytical method and assayed samples returning >10g/t gold were repeat fire assayed using a 50g charge with gravimetric finish.

When the logging geologist notes the presence of visible gold or geological characteristics suggesting high gold grade in mineralised drill intervals a screen fire assay using a 500g charge is undertaken. These results supersede all other assay methods. The screen fire assay is undertaken on the logged interval with additional samples also taken for screen fire assay. Typically additional samples are collected from a distance of 5m to 10m on either side of the interval containing visible gold. The screen fire assay method is considered to provide more reliable assay results when compared to the initial fire assay or fire assay with gravimetric finish methods.

The selection of the assay method is similar for 1m samples and 3m composite samples.

The following information for sample preparation and analysis is listed for ActLabs:

- Record sample batch received.
- Crush, pulverise to pulp material and split sample. A size fraction check is carried out every 8th sample or to customer requirements (e.g. 90% passing 12 mesh). Internal laboratory blanks are inserted every 35th sample in addition to cleaning of crushing and pulverising equipment with compressed air after preparation of each sample. These blanks are analysed and reported to the client as part of laboratory quality control. A barren silica wash is also carried out at the end of each assay batch.
- Assay via fire assay (FA) and AAS finish on a 30g test weight. A repeat, second fire assay with gravimetric finish is undertaken where the initial FA assay is >3g/t gold. The second assay is considered more correct for use. The laboratory undertakes analysis of a sample duplicate every 35th

sample and reports the results to the client as a part of laboratory quality control. In addition, the laboratory inserts certified and in-house standards which are of similar grade and material matrix to the client samples. The in-house laboratory standards are subjected to round robin testing from other laboratories to determine the accepted value and confidence intervals.

- Internal laboratory controls (standards and blanks) are carried out for each sample batch and reported with sample assay results. All results are stored in a dedicated Laboratory Information Management System (LIMS) database.

Internal laboratory controls (standards and blanks) are carried out for each sample batch and reported with sample assay results.

Assay procedures for the historical Diamond Core drilling are not well documented.

Figure 11-5: AAS Section at ActLabs, Guyana

11.8 Assay Quality Control Measures (QAQC) for Drilling

In-house and laboratory QAQC data for the Karouni Gold Project have been reviewed for analytical data collected between July 2013 and April 2014.

11.8.1 Introduction

All drilling programs carried out at the Karouni Gold Project have included ongoing QAQC procedures. These procedures include the use of certified standards and blanks, internal laboratory quality control and submission of field duplicate samples.

The Rig Supervising Geologist determines the placement of standards, blanks and duplicate samples to be inserted in the sample stream. Sample identification numbers designated for standards or blanks are written on the bags with the bags left empty. These QAQC check samples are inserted at frequency of no less than 5% standard, 5% blanks and 10% duplicate. A summary of sample percentage by QAQC type for Hicks and Smarts is presented in Table 11-1 and Table 11-2.

A QAQC assessment of the data is completed on a regular basis by Company geologists. The QAQC report provided by Geobase (Lloyd, 2013) concluded that all sampling, sample preparation, sample security and analytical procedures conform to industry best practice and are adequate to give a representative picture of

the nature of the mineralized body and its host rocks. Since mid-year 2013 monthly QAQC reports are generated by Troy staff. Consulting Geochemist. Dr. Mark Fedikow; Ph.D. P.Eng. P.Geo. C.P.G. of Mount Morgan Resources Ltd Consultancy; 627 Manchester Blvd. North, Winnipeg, Manitoba, Canada was engaged by Troy to review QAQC results as well as current procedures and protocols.

Table 11-1: Hicks QAQC by Sample Type for Drilling

Check Type	Sample Count	% Total Sample Population	% Actlabs Sample Population	% Acme Sample Population	% ALS Sample Population
Field Derived QAQC					
Standards	557	4.91	5.02	2.87	1.69
Blanks	461	4.07	4.09	3.69	3.39
Field Duplicates	571	5.06	5.19	2.87	0
Lab Introduced QAQC					
Sample Prep Repeats	112	0.99	0.91	2.87	0
Pulp Repeats in Lab	297	2.62	2.57	3.07	8.47
Element Repeat Analysis	188	1.66	1.66	0	15.25
External Lab Checks					
Pulp Checks	39	0.34	0	0	100

Table 11-2: Smarts QAQC by Sample Type for Drilling

Check Type	Quantity	% Total Sample Population	% Actlabs Sample Population	% Acme Sample Population	% ALS Sample Population
Field Derived QAQC					
Standards	2,656	4.85	4.89	4.49	2.13
Blanks	2,540	4.63	4.67	4.37	1.06
Field Duplicates	2,112	3.85	3.46	7.83	0
Lab Introduced QAQC					
Sample Prep Repeats	812	1.48	1.43	2.00	0
Pulp Repeats in Lab	1,849	3.37	3.41	2.92	8.51
Element Repeat Analysis	1,344	2.45	2.66	0	24.47
External Lab Checks					
Pulp Checks	94	0.17	0	0	100

11.8.2 Standards and Blanks

The following quality control procedures are currently in use for blanks and standards:

- Blanks – consisting of uniform material containing no gold. Troy uses blanks consisting of silica sand and a second described as gabbro GT siftings. Blanks are utilized to track possible sources of contamination during sample preparation.
- Standards – currently there are up to 21 different standards in use by Troy. The standards have been sourced from Geostats Pty Limited and Rocklabs Limited, and are all certified gold reference materials. The standards have been selected on the basis of their grade to match the average grade range of gold mineralisation. Matrix matching in standards is outstanding.

The standards are supplied as pre-crushed powders and their use provides an indication of the ability to determine the content of the metal of interest in an accurate and reproducible manner in the assay procedure.

The use of field introduced blank standards is primarily to assess any sequence contamination introduced during sample preparation and analysis. Blank material consists of non-certified in-house blank material of barren rock types. The material used for Blank 1 has been collected from white sand on-site while Blank 2 has been collected from gabbro siftings. The field preparation of blank material has not been documented. Blank 2 is utilized to demonstrate the lack of contamination of the samples in the laboratory analytical stream.

The certified standard reference materials used by Troy and reviewed in this report are summarised in Table 11-3. Only those standards with >25 replicate analyses are displayed in this Table. Twenty-five replicates are considered the minimum number of assays upon which to base significant observations and conclusions. The silica sand blank Blank2 has returned results within acceptable limits. A small number of elevated spikes in the replicate analyses of blank Blank2 are documented (Figure 11-6) however these are interpreted as spurious and not indicative of a particular pattern of contamination (Figure 11-7). Additional under-reporting occurs for standards G900-5 (Figure 11-8) and G909-5 (Figure 11-9). Control charts for standards G903-9, G904-8 and G909-1 are given in Figure 11-10, Figure 11-11 and Figure 11-12 respectively. Accuracy and reproducibility as demonstrated by the standards is excellent.

Table 11-3: Summary of Standards

Standard	Assay Method	Certified Value (Gold ppm)	Std. Deviation	Number Used
G900-5	Fire Assay	3.21	0.13	216
G903-9	Fire Assay	11.26	0.41	62
G904-8	Fire Assay	5.53	0.18	134
G907-2	Fire Assay	0.89	0.06	986
G909-1	Fire Assay	1.02	0.06	735
G909-5	Fire Assay	2.63	0.1	442

The results of 287 replicate analyses of Blank2 indicate the relative lack of contamination measured in this blank (see Figure 11-6). Exceptions occur at several points during the analytical time frame for these measurements but these are neither systematic nor indicative of serious contamination in the assay laboratory.

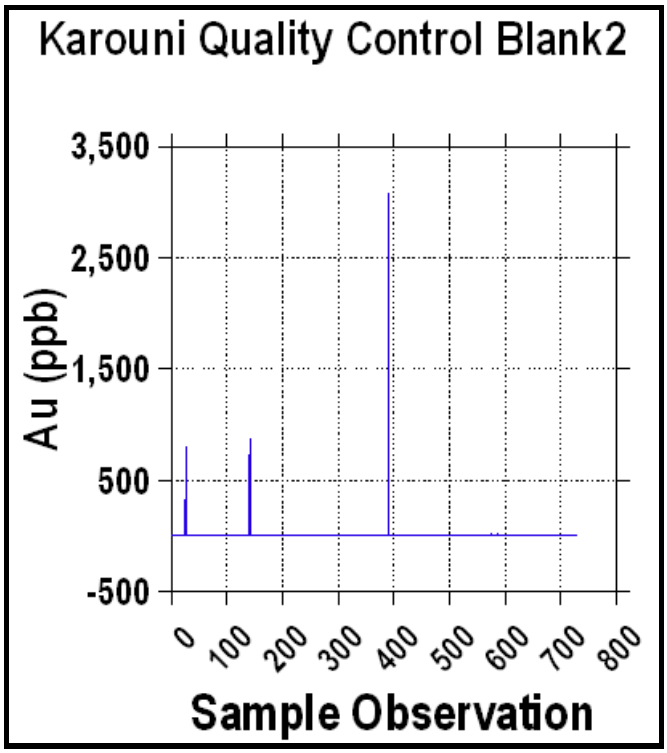

Figure 11-6: Silica Sand Blank "Blank2" - Plot based upon 729 Observations

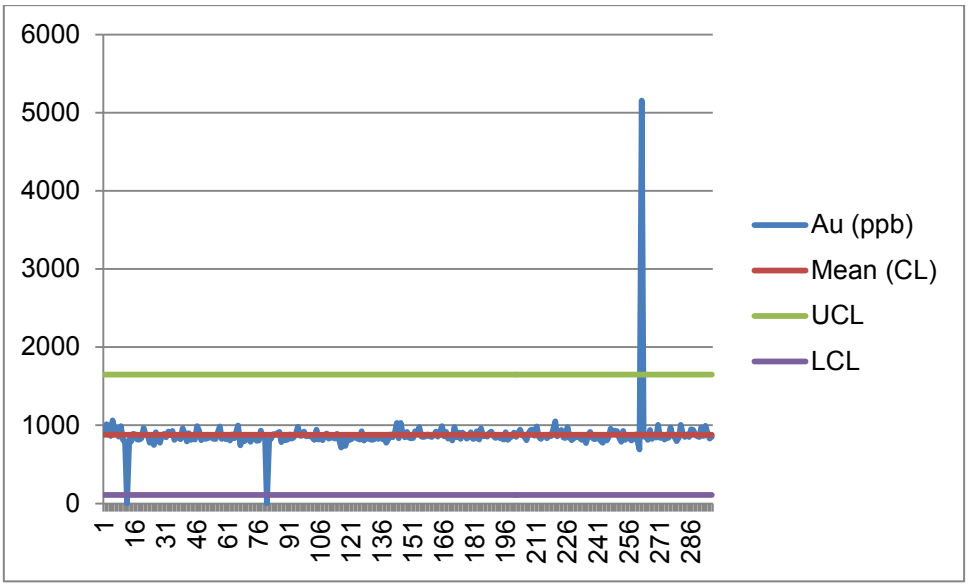

Figure 11-7: Standard G907-2 Control Chart, Karouni Project (n=296) - Au by Fire Assay with AAS Finish

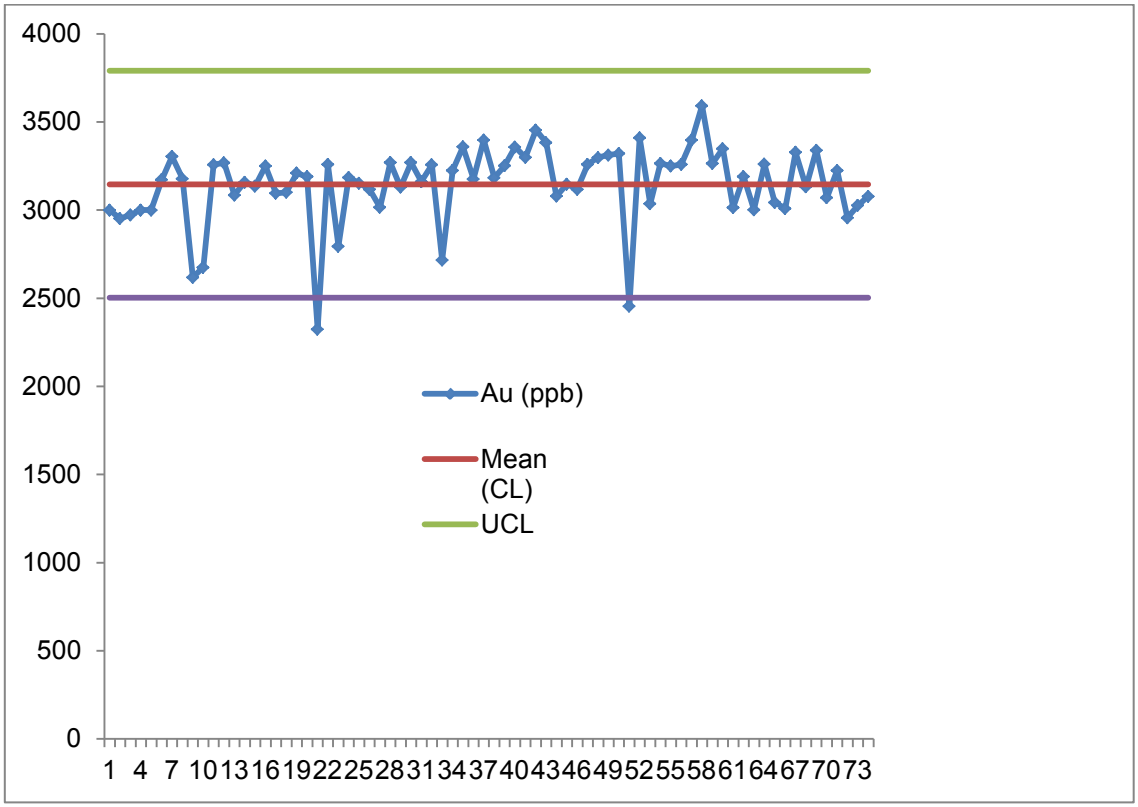

Figure 11-8: Standard G900-5 Control Chart, Karouni Project (n=79) - Au by Fire Assay with AAS Finish

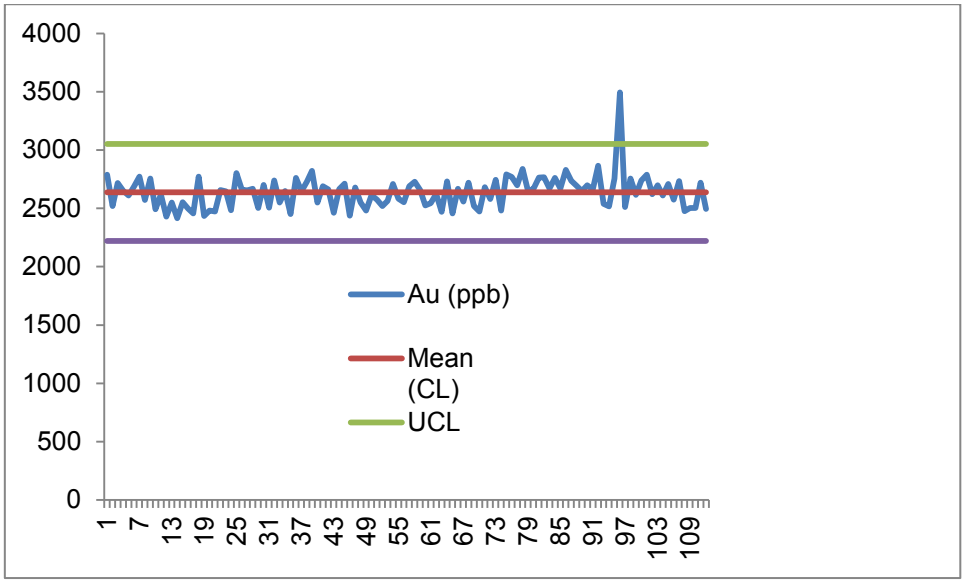

Figure 11-9: Standard G909-5 Control Chart, Karouni Project (n=112) - Au by Fire Assay with AAS Finish

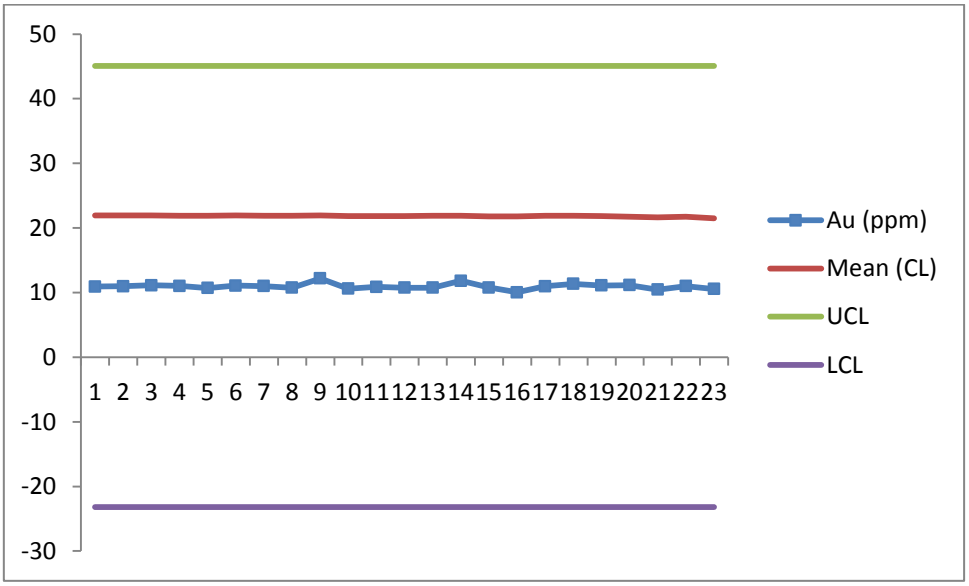

Figure 11-10: Standard G903-9 Control Chart, Karouni Project (n=24) – Au by Fire Assay with Gravimetric Finish

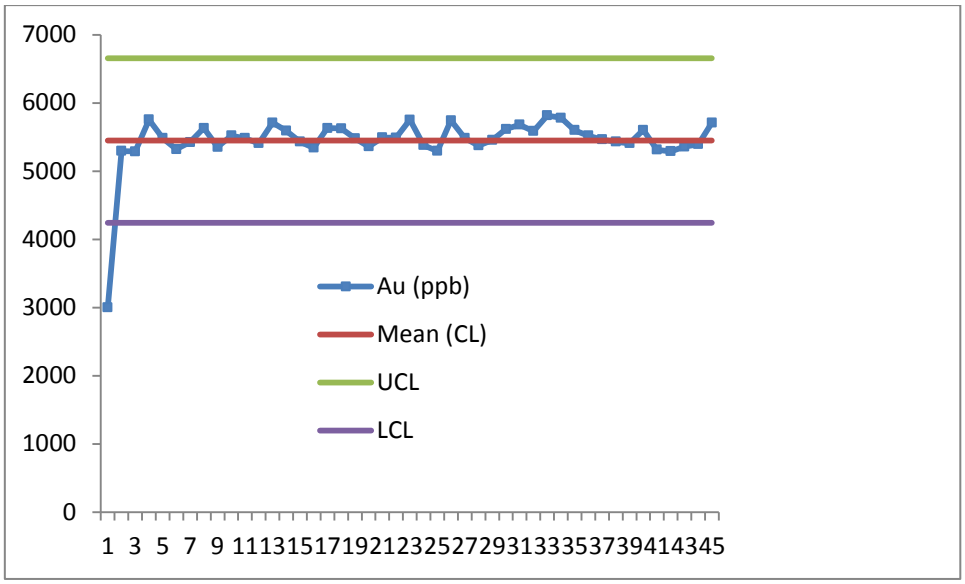

Figure 11-11: Standard G904-8 Control Chart, Karouni Project (n=45) - Au by Fire Assay with AAS Finish

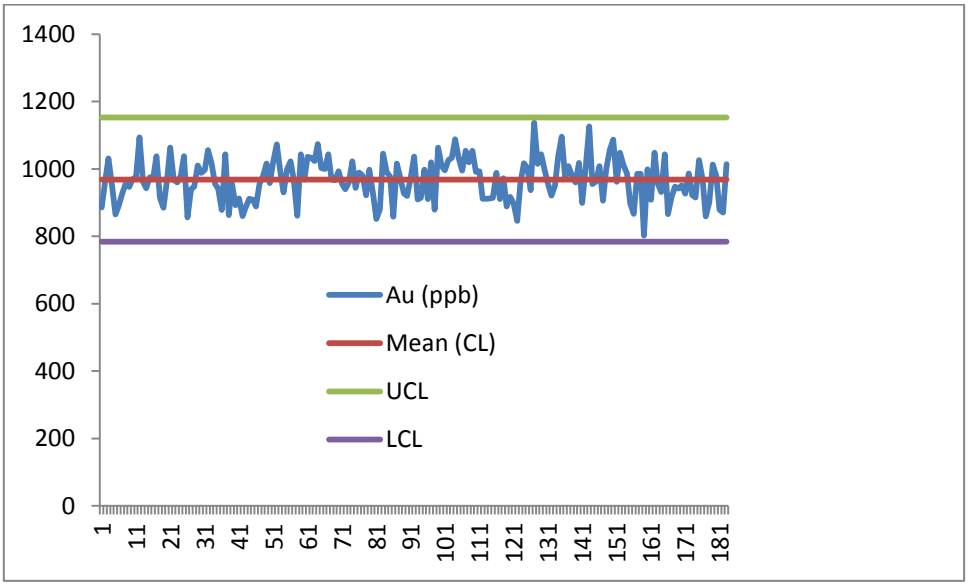

Figure 11-12: Standard G909-1 Control Chart, Karouni Project (n=183) - Au by Fire Assay with AAS Finish

11.8.3 Laboratory Duplicate Samples

Duplicates for Spear and Three Tier riffled core samples indicate good linear relationships between the original and the repeat analysis for both Spear and Three Tier riffled samples (see Figure 11-13 and Figure 11-14)

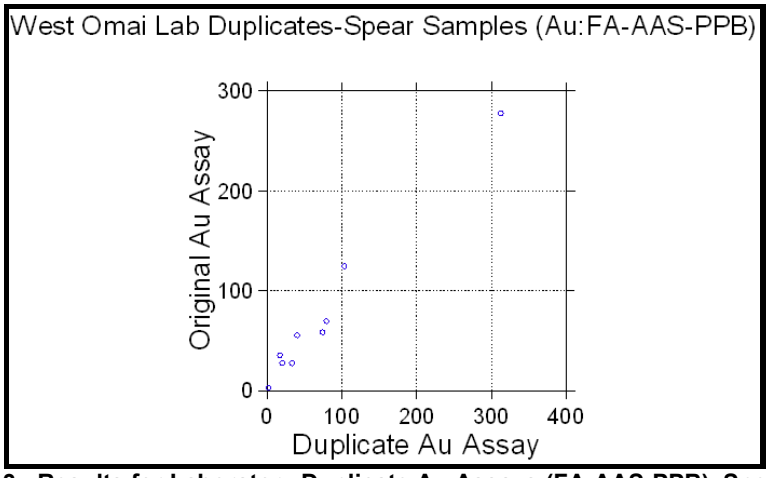

Figure 11-13: Results for Laboratory Duplicate Au Assays (FA-AAS-PPB), Spear Samples

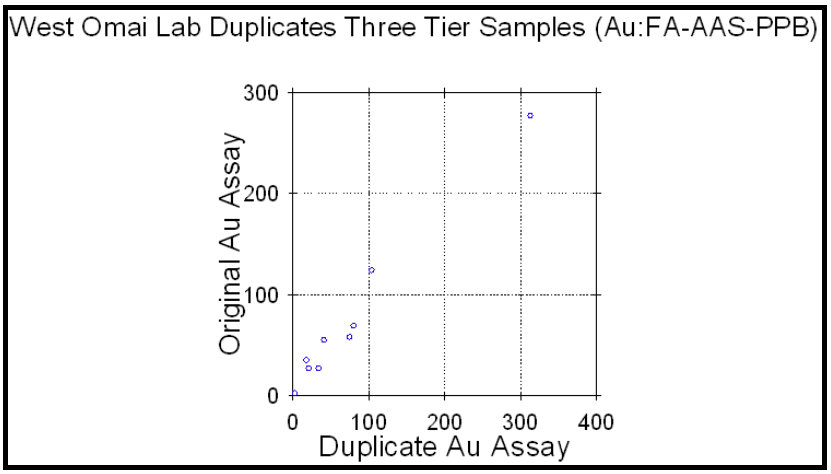

Figure 11-14: Results for Laboratory Duplicate Au Assays (FA-AAS-PPB), Three Tier Samples

Sample Size in Laboratory Duplicates: Gold assay results for quarter core and half core are compared in Figure 11-15. It is clear the appropriate sample size has been utilized for the Karouni assays. Quarter core results indicate a significantly less linear association than do the results for the half core samples. There is a suggestion of nugget effect in the Three Tier assays at elevated concentrations.

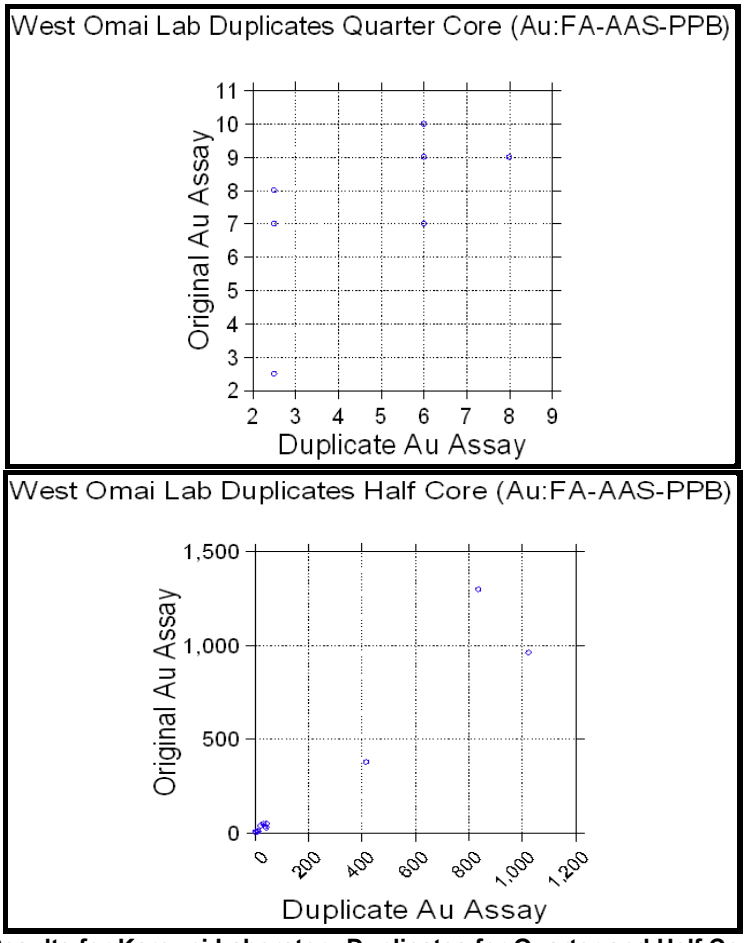

Figure 11-15: Results for Karouni Laboratory Duplicates for Quarter and Half Core Sample Sizes

Field Duplicates: The results of simple linear regression for both Spear and Three Tier samples indicates scattered data indicative of possible nugget effects in the Spear samples and highly variable data for the Three Tier samples. The cause for the variability is likely an inhomogenous sample resulting from nugget effect. Interestingly, in over-range sample assays both Spear and Three Tier samples appear to be less

variable than assays at lower concentrations. This may be the difference between fire assay results and those for AAS (see Figure 11-16, Figure 11-17 and Table 11-4)

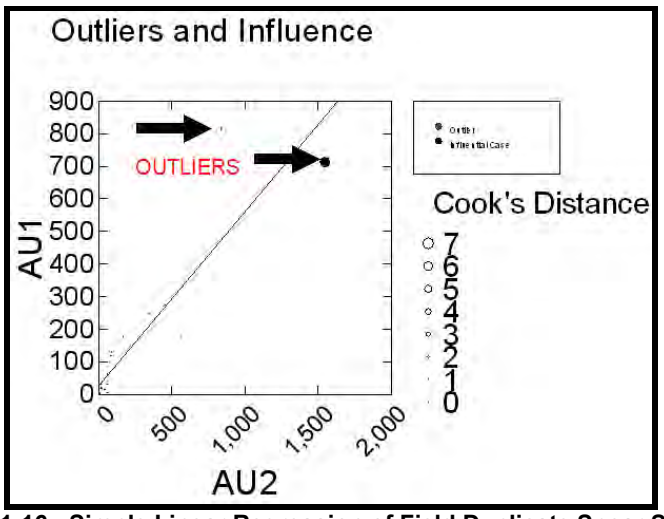

Figure 11-16: Simple Linear Regression of Field Duplicate Spear Samples.

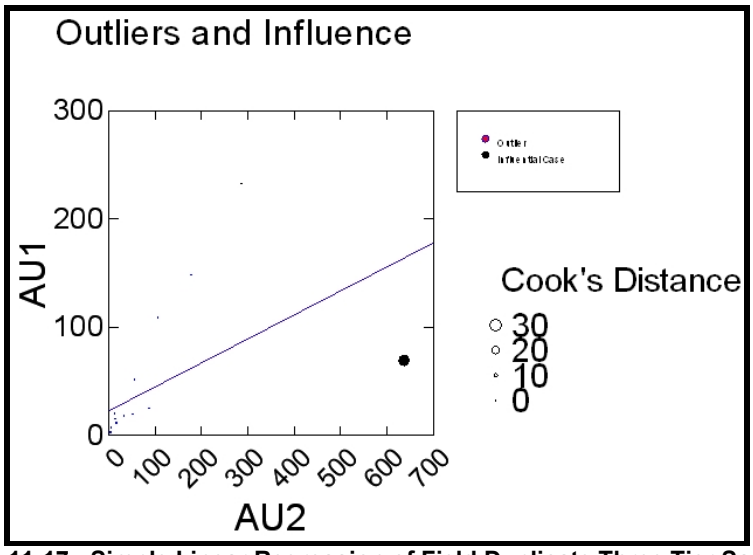

Figure 11-17: Simple Linear Regression of Field Duplicate Three Tier Samples.

Table 11-4: Over-Range Gold Assay Duplicates for Spear and Tier 3 Samples			
Karouni Field Duplicates-Tier3 and Spear Over-Range Assays.			
Sample Type	Field Duplicate Pair	Gold Fire-Assay-(GRAVIMETRY-PPM)	
		Original	Duplicate
Spear Samples	59725-59726	7.89	8.73
Tier 3 Samples	59864-59865	3.92	3.82
	54125-54126	12.22	15.86

Laboratory Replicates: Simple linear regression analysis of laboratory assay duplicates based on Fire Assay-AAS finish indicate good reproducibility with elevated variability at higher concentrations (see Figure 11-18). The increased variability is likely attributable to nugget effect.

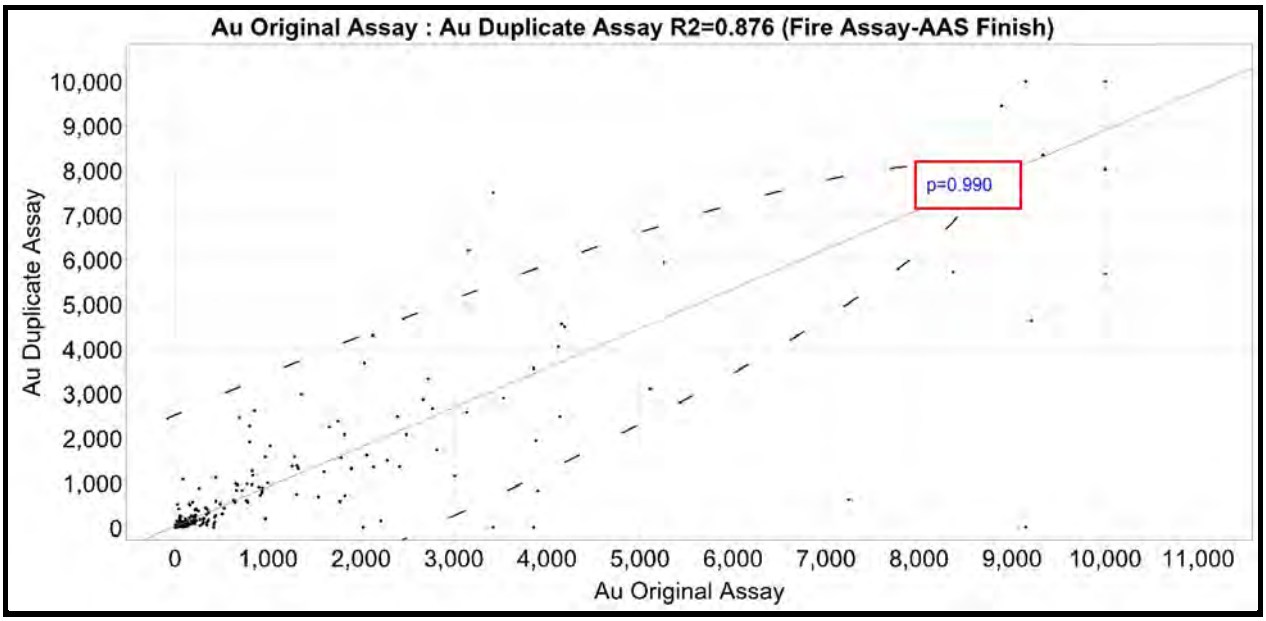

Figure 11-18: Simple Linear Regression of Original vs. Duplicate Assay Data Based on Fire Assay with AAS Finish

Sample points outside of the 99th percentile ellipse can be regarded as outliers. Plot based on 390 duplicate pairs. Assay results depicted as parts per billion.

11.9 Adequacy of Sample Preparation, Analyses and Security

Notwithstanding the lack of information regarding sample preparation, security and analytical procedures for historic drilling at the Project the assay/analytical procedures implemented for modern, recent drilling are considered appropriate and adequate for the deposit style and the environment within which exploration is being undertaken. The approaches are based on best practice industry standards and results are suitable for comparison with historic results and ongoing assessments. As part of future quality control measures at Karouni an examination of the distribution of particulate and/or sulphide bound gold might alleviate some of the reproducibility issues observed in field duplicate samples.

11.9.1 Summary of Observations and Conclusions to Date

Duplicates

Field Duplicates: There is significant scatter at all levels of concentration in this data particularly for the gold assays determined by Fire Assay with atomic absorption ("AA") finish (ppb). In fact there are 118 samples out of 420 (28%) that fall outside the error limits. This is a reflection of sample inhomogeneity and the difficulty in selecting a representative sample for assay that ensures a reliable distribution of gold or gold-bearing particles in the sample.

There is significantly less scatter in the Field Duplicates for gold determined by Fire Assay-Gravimetry (ppm) although there are only 21 assays used to assess this. Nevertheless, 11 out of 21 assays fall outside of the error limits for a 52% failure rate. The inhomogeneity in the field duplicates is also reflected in this approach to gold assay. Gravimetry or AA-based analytical approaches cannot make up for an inhomogenous sample.

Lab Duplicates

Labdup_lp: Laboratory duplicates reflect far less variability in analyses for gold assays determined by AA or Gravimetry. In the AA data (ppb) there is relatively good reproducibility observed with 39 of 278 samples falling outside of the error limits for a success ratio of 14%. In the Gravimetric determination of gold (ppm) there are only 4 samples observed and one of these falls outside of error limits (25%). These data are considered superior to the Field Duplicates in terms of reproducibility.

Labdup_p: Despite a large number of gold determinations by Fire Assay and AA finish (n=350; ppb) for these duplicate samples many fall below the lower limit of detection ("LLD") for the analytical method (206/350=59%). Of those samples >LLD 72 (20%) are outside of the error limits. The gold assays by Gravimetry (ppm) are few (n=11) however none of these exceed the error limits and on this basis, albeit a small sample population, appear to offer a more accurate and reproducible gold assay.

Lab Replicates: Gold analyses by Fire Assay and AA finish (ppb) are based on 436 samples of which 276 are below the limits of detection for this analytical method and 74 of 436 (17%) are outside of the error limits. Variability is significant but with 274 of 436 or 63% of the samples <LLD it is difficult to appreciate just how variable these data are. There is only one duplicate pair analyzed by Fire Assay with Gravimetry and as such is not commented upon further.

Blanks

Blanks: Blank2 is the only blank for which data is available. Only 3 samples exceed the error limits for 902 analyses for an excellent 0.3%. The 3 samples outside of the error limits are considered to be spurious. There are however 3 distinctive periods where some minor gold contamination was documented at very low levels. These periods occurred during the first two weeks of September (2013), during the period September 30 to November 16 and onwards from February 13 (2014). These periods of slightly elevated gold in Blank2 are derived from a moving average of samples assayed during these periods and as such may not be as consistent as initially thought. There are no significant issues identified in Blank2.

Standards

The analysis of gold in standards is reviewed for two analytical approaches: Fire Assay-Atomic Absorption finish and Fire Assay-Gravimetric finish. Observations based on both are given below for most standards.

G900-5: Au-Fire Assay-Atomic Absorption (ppb): A slow but progressive decrease in gold concentration is documented for these analyses. It is likely however this pattern or trend is strongly influenced by 6 very low assay results produced after December 24/13. There are only 6 of 82 measurements (7%) are outside of the error limits. The additional assay approach of Fire Assay-Gravimetry for these standards indicates low to very low variability with no assays outside of the error limits. Both analytical approaches indicate excellent results for G900-5.

G903-9: There were no samples plotted for gold assays by FA-AA (ppb). However the plot for gold assays for G903-9 by FA-Gravimetry documents no samples outside of the error limits but with a slow and steady decrease in concentration in assay results from November 22/13 to March 4/14.

G904-8: There are no samples assayed for gold by FA-AA (ppb) that fall outside of the error limits for Au-FA-AA (ppb). There is however a couple of general trends within the plotted data. These are a small progressive decrease in gold abundance between December 10/13 and February 11/14. At the end of this trend the opposite occurs with a slow progressive increase in gold content of the standard between February11/14 through March 1/14. In the Au-FA-Gravimetry (ppm) determinations no samples fall outside the error limits but like the FA-AA Au determinations those for Gravimetry demonstrate a steady increase between July 16/13 and September 6/13. It is noted that only 8 assays were undertaken with Gravimetry finish.

G907-2: Based on 386 determinations for gold on this standard by FA-AA (ppb) only 3 samples lie outside the error limits (0.7%). The results are well-constrained. There were no results reported for gold by FA-Gravimetry.

G909-1: Like standard G907-2 the analytical results based on gold assays by FA-AA (ppb) G909-1 are well constrained with only 1 sample outside of the error limits for 226 determinations (0.4%). There were no patterns of variability observed within the data. There were no results reported for gold by FA-Gravimetry.

G909-5: The analytical results based on for gold assays by FA-AA (ppb) in standard G909-5 are well constrained with only 1 sample outside of the error limits for 128 determinations (0.7%). There were no patterns of variability observed within the data. The FA-AA Au determinations based on Gravimetry for only 3 samples indicate none lay outside of the error limits.

Conclusions:

1. The Field Duplicates are by far the most variable dataset for gold analyses regardless whether they are assayed with AA finish or Gravimetry. This is understandable given the collection and homogeneization of a field sample is likely to be less rigorous than one prepared for assay in the laboratory. The variability for assays by AA-Gravimetry is less because the magnitude of the measurement is in parts per million and not parts per billion.
2. Laboratory duplicates are generally excellent in terms of accuracy and reproducibility. Laboratory replicates are tough to assess because 63% of the assays are <LLD.
3. Standards are performing well. There are minor variabilities with the standards over time but none of this variability is of significance.
4. The quality of the gold assays is therefore interpreted to be very good based on accuracy and reproducibility.
5. As part of future quality control measures at Karouni an examination of the distribution/siting of particulate and/or sulphide bound gold might alleviate some of the reproducibility issues observed in field duplicate samples.

12. Data Verification

Database verification was carried out during on-site reviews and Mineral Resource estimation. The due diligence included field checks on the collar and downhole orientation as well as verification of site and database procedures. Regular spot check of the database; collar, downhole survey, assay and geology tables against original files are undertaken. As a result of the verification process, the authors are confident the quality of the data is of a high standard suitable for use in this report.

12.1 Historical Drill Hole Database Pre –Troy

No assay certificates are available for any exploration completed by the British Guyana Geological Survey (GGMC), Overseas Platinum Company (OPC) or Cathedral Gold. While these results cannot therefore be verified, documentation in relation to trenching, diamond drilling and pit sampling completed by Cathedral Gold is sufficient to suggest that there is no reason to doubt their validity.

Similarly, stream sediment sampling completed by the GGMC appears to have been undertaken in a rigorous manner to best international standards at the time. While no assay certificates are available for this work, the procedures and results are well documented and there is no reason to doubt their validity.

At the Azimuth's request Geobase Consultants undertook cross referencing of the compiled assay data with the original assay laboratory files. A number of inconsistencies were noted. These inconsistencies, in conjunction with a lack of metadata available on assay techniques, deemed it necessary to reload all of the original data. In addition, random checks of assay data from original drill paper logs were also undertaken. No significant issues were identified. Samples of unknown origin were given a Lab ID of "Unknown". If the date of analysis was unknown a default value of 01/01/1900 was used.

12.2 Recent Exploration

In the past Azimuth outsourced all its on-going database management services to an independent company, Geobase. All exploration derived data such as drilling, surface geochemistry and point data is stored within a SQL database model designed for the storage of exploration derived data (Geomodel). This provides a flexible and efficient environment for validating, storing and assessing data from diverse software applications and sources.

Access to the database is restricted to personnel only who have been trained in database management. All appropriate and valid changes requested from site are made only by the Database Administrator. Site personnel do not have the ability to directly access the database which allows the integrity of the data to be maintained.

Since mid-2013, Troy has managed the database as part of its global database management system. Troy has set up a comprehensive suite of procedures for drilling, sampling and assaying. The procedures include standardised templates for geological logging and sample selection (including insertion of standards, blanks and duplicates). A spreadsheet based log is completed for each drillhole and then submitted to the Database Administrator for verification and uploading to the drillhole database. The capture of exploration data is outlined within Figure 12-1 below. Standard lithological codes for the major rock types are presented within Figure 12-2. The table structure of the database for drilling and trenching is presented below in Figure 12-3.

The preferred means of data capture is to record data digitally, however it has been necessary in some cases to complete hardcopy paper logs which are then entered into Excel based tables at a later stage. The spreadsheet also allows for the recording of comments made in relation to the captured meta-data (e.g. drilling conditions, sample recovery, abandoned holes, etc.). Database verification includes validation based on code checks (e.g. geology legend), multi-table (e.g. drillhole collar or end of hole) and spatial validation. Database queries flagged by database Administrator are then sent to the field based team for resolution and correction before database finalisation. The validation check list is comprehensive and includes:

- Checks for non-standard character codes.
- A check of numeric values against a minimum and maximum range.
- Missing coordinates, height and drillhole depth in the collar file.
- Large deviations in both dip and azimuth in the survey file.

- Missing intervals in the down-hole interval file.
- Duplication of drillhole numbers or overlapping intervals in the down-hole interval file.
- Records in the interval file that exceeds the total depth in the collar file.
- Missing holes in either the collar file or down-hole interval file.
- Missing intervals inserted and classified accordingly to knowledge base.
- Down-hole survey dip and azimuth data checks (Magnetic vs. True North vs. Local).
- Spatial distribution of mineralisation.
- Spatial distribution of down-hole geology.

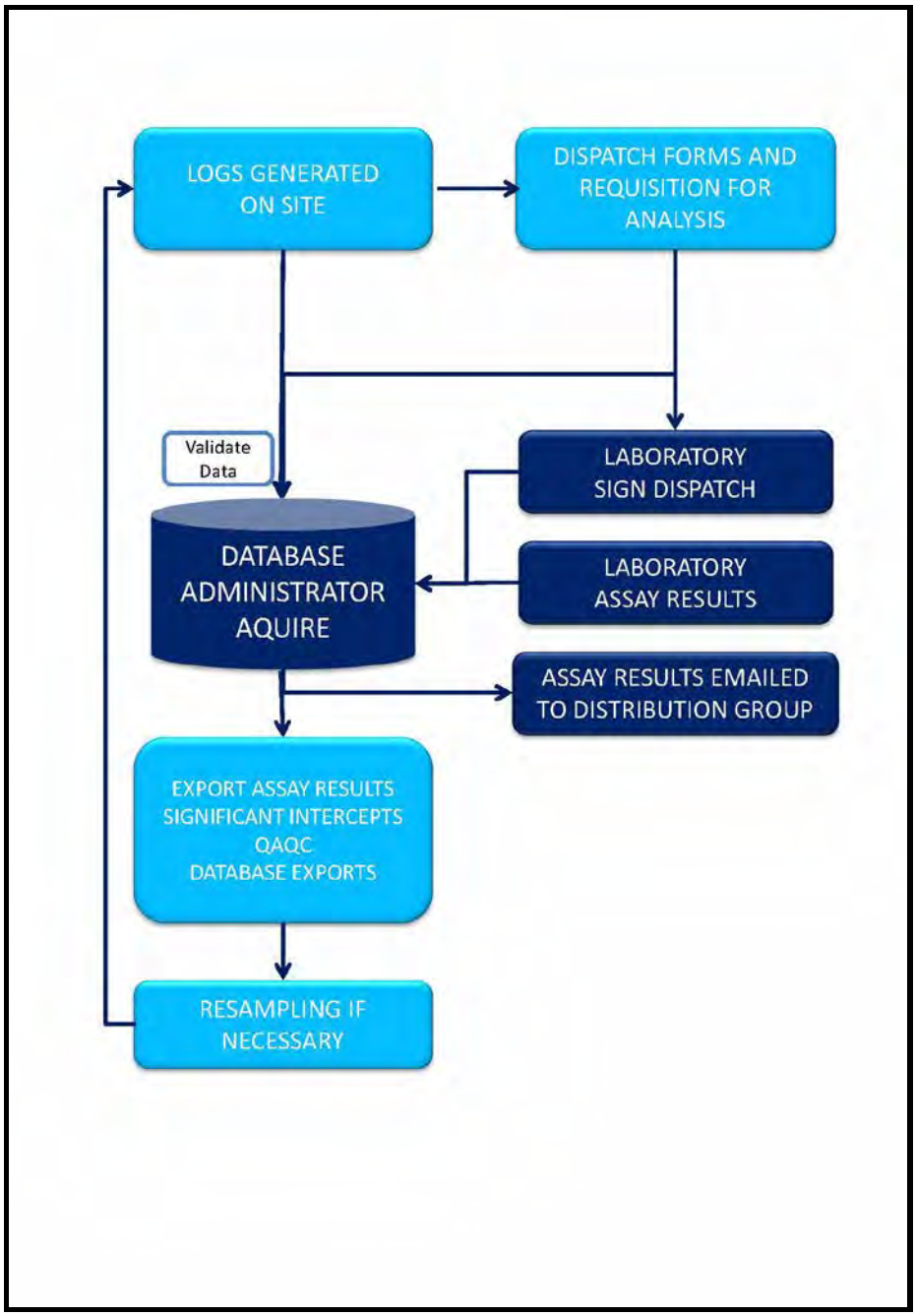

Figure 12-1: Exploration Project Data Flow

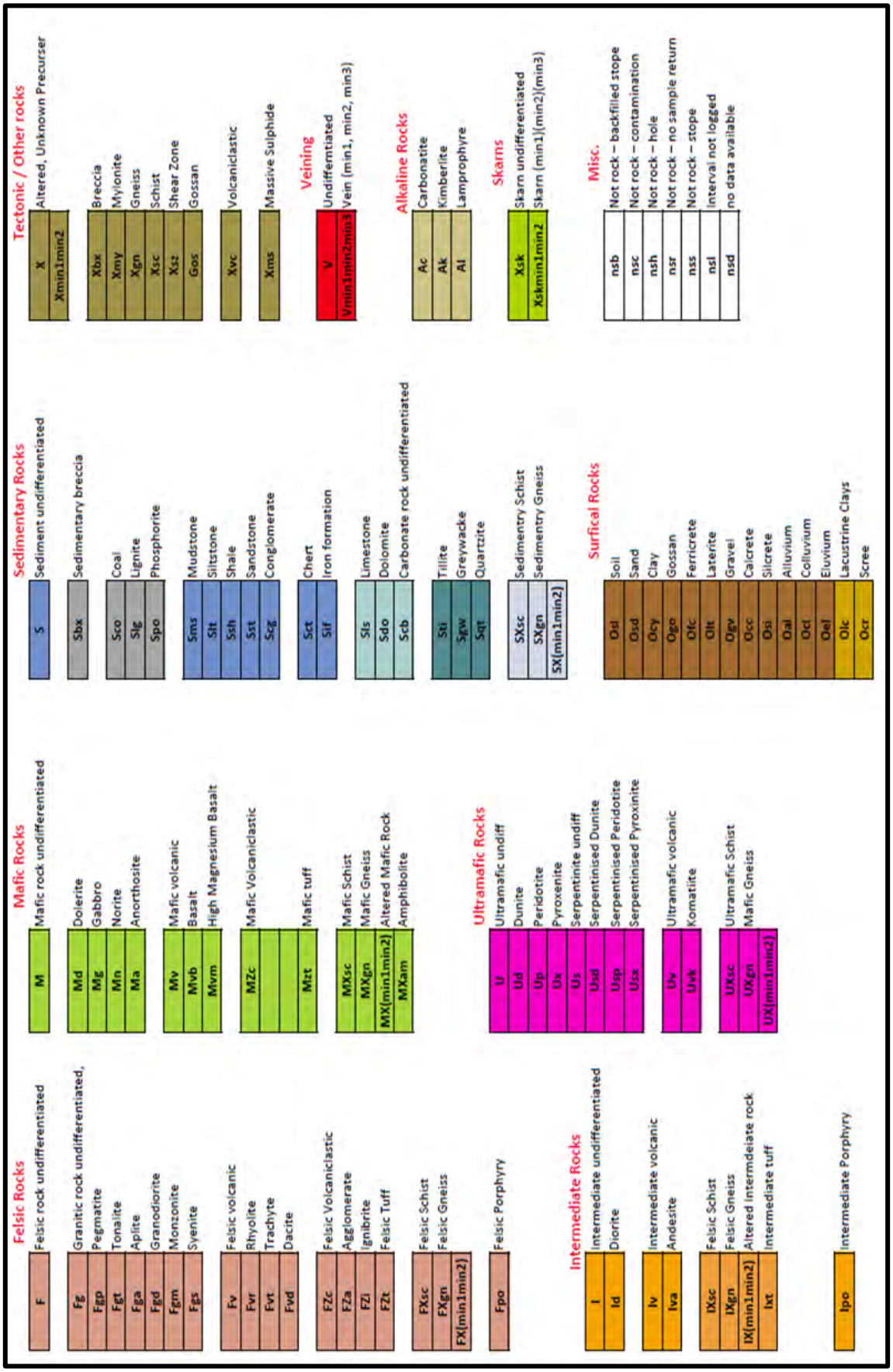

Figure 12-2: Standard Geological Codes

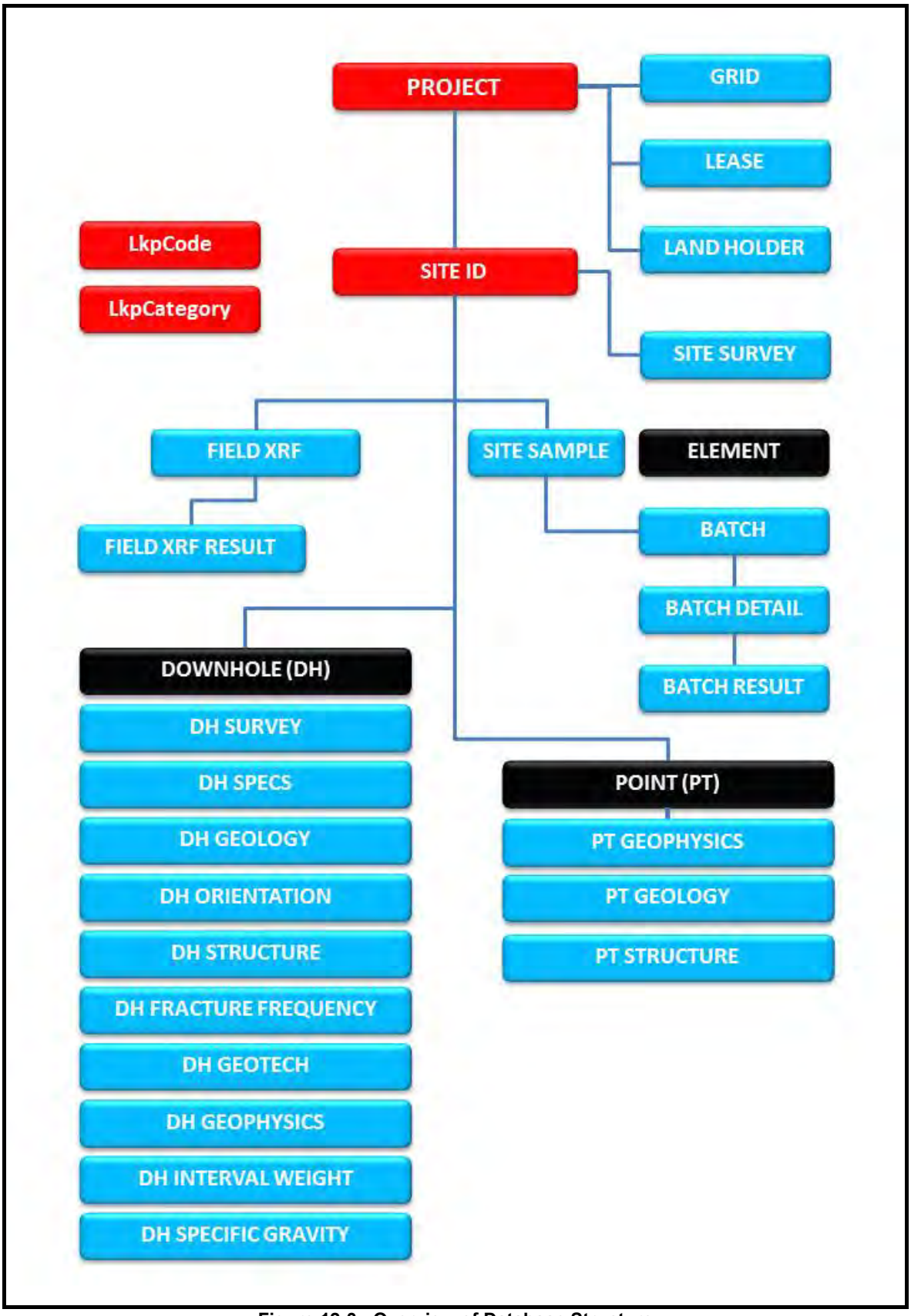

Figure 12-3: Overview of Database Structure

12.3 Twinned Drillholes or Sample Lines

The drilling of twinned drill holes refers to an industry practice of repeat drilling an earlier drillhole. The repeat drillhole trace is generally designed to pass within 5m of the existing drillhole and aims to provide additional confidence on items such as:

- Intersections of high grade mineralisation or nugget effect
- Geological boundaries
- Historical information
- Drilling method

12.3.1 Twinned Drillholes

Azimuth carried out a twin drillhole program at the Smarts Deposit to compare RC versus twin RC and RC versus twin diamond drilling. A total of six sets of twin drill holes occur throughout the Smarts Deposit and were compiled by RPM for comparison. All twin drilling was carried out between May 2011 and January 2012.

The twin drill comparison data set relates only to drilling conducted and was restricted to within the gold mineralisation domain objects. The average distance between hole trace for the twin holes was approximately 4m with the maximum separation distance being 7m. All twin drillholes are parallel to sub-parallel.

As documented in earlier technical reports, the comparison results in terms of overall gold grade average between original drill holes is 4.73g/t gold for 10.0m average mineralisation length versus twin drill holes at 4.73g/t gold for 8.7m average mineralisation length. This provides additional confidence that the position of geological boundaries and mineralisation information is robust on a global basis.

Comparison results for diamond versus RC and RC versus RC drill twins show greater difference. Also considerable variation can be noted on a twin hole by hole examination. The variation partly reflects the small data set and is considered reasonable for the style of the deposit and nuggety nature of mineralisation.

12.4 Database Verification

12.4.1 Historical Data

The drill data queries raised during the Mineral Resource estimation process were resolved promptly and a validated database re-issued, if required. There are no significant queries outstanding from this review process.

During 2011, RPM Consultants undertook a random spot check of 20 database assay results. The assay results were from Azimuth conducted drilling activities during mid-2011. The review comparison was between the drillhole database assay table and digital assay laboratory files over random assay batch numbers. No errors were noted in the database records.

A further data validation process was carried in April 2012 and involved the comparison of the database for 17 holes against the data files. The data validation was undertaken for both the Hicks and adjacent Smarts Deposits. This included the assay, geology, collars and downhole survey information. The validation process noted the following:

- No issues were found with assay information.
- No major issues were found with collar information.
- No issues were found with geology information. Variation between hardcopy and database geological logging codes was noted. The database contains the correct legend. Site geologists are using the correct geological logging legend.
- No issues were found with downhole surveys.

Table 12-1 summarizes the validation results.

Table 12-1: Drill Holes Validated Against Azimuth Hardcopy Data (May 2012)

Hole ID	Validation			
	Assays	Collars	Geology	DH Surveys
HRC028	No Issues	Updated*	No Issues	No Issues
HRC030	No Issues	No Issues	No Issues	No Issues
HRC031	No Issues	No Issues	No Issues	No Issues
HRC032	No Issues	No Issues	Different codes**	No Issues
HRC033	No Issues	Updated*	No Issues	No Issues
HRC054	No Issues	No Issues	No Issues	No Issues
HRC055	No Issues	No Issues	No Issues	No Issues
HRC056	No Issues	Updated*	No Issues	No Issues
HRC075	No Issues	No Issues	No Issues	No Issues
HRC076	No Issues	Updated*	No Issues	No Issues
HRC072	No Issues	No Hardcopy	Different codes**	No Hardcopy
HRC082	No Issues	Updated*	Different codes**	No Issues
HRC083	No Issues	Updated*	Different codes**	No Issues
HRC084	No Issues	Updated*	No Issues	No Issues
HRC085	No Issues	Updated*	Different codes**	No Issues
HRC086	No Issues	Updated*	No Issues	No Issues
HRC087	No Issues	Updated*	Different codes**	No Issues
SDT438	No Issues	Updated	No Hardcopy	No Issues
SDT477	No Issues	Updated	Different codes**	No Issues
SDT479	No Issues	Updated	Different codes**	No Issues
SDT489	No Issues	Updated	No Hardcopy	No Issues
SDD002	No Hardcopy	No Issues	Different codes**	Different Dip
SDD003	No Hardcopy	No Issues	No Issues	No Issues
SDD004	No Hardcopy	No Issues	No Issues	No Issues
SDD005	No Issues	Different X-Y	No Issues	No Issues
SRC319	No Issues	Updated	Different codes**	No Issues
SRC390	No Issues	Updated	Different codes**	No Issues
SRC477	No Issues	Updated	Different codes**	No Issues
SRC310	No Hardcopy	No Issues	Different codes**	No Issues
SRC323	No Issues	No Issues	Different codes**	Different Azimuth
SRC327	No Issues	No Issues	No Issues	No Issues
SRC371	No Issues	No Issues	Different codes**	No Issues
SRC372	No Issues	No Issues	Different codes**	No Issues
SRC402	No Issues	No Issues	No Issues	Different Azimuth
*** Database collar / survey updated for more accurate survey. No issues unresolved.**				
****Hardcopy logging updated to match correct database legend codes. No issues unresolved.**				

12.4.2 2013 Drilling Campaign

Data validation of the recent drilling involved the analysis of database records for both diamond drill holes and RC holes. All original data files were uploaded to the database for comparison against the Resource data. The process included change detection for the assay, geology, collars and downhole survey information. Conclusions from the data verification are listed below:

- No issues were found with assay information.
- No major issues were found with collar information. Collar RL information was compared to a LiDAR surface by the Company prior to finalisation of the database, all records correspond to the LiDAR elevation.
- No issues were found with geology information. The database contains the correct legends, with lithology and weathering codes for each interval.
- No issues were found with downhole surveys.

A summary of the data validated for the Hicks and Smarts Deposits is shown in Table 12-2 and Table 12-3.

Table 12-2: Hicks Drill Holes Validated Against Hardcopy (December 2012)

Hole ID	Validation			
	Assays	Collar	Geology	DH Surveys
HDD001	No Issues	No Issues	No Issues	No Issues
HDD003	No Issues	No Issues	No Issues	No Issues
HDD004	No Issues	No Issues	No Issues	No Issues
HDD010	No Issues	No Issues	No Issues	No Issues
HDD011	No Issues	No Issues	No Issues	No Issues
HDD012	No Issues	No Issues	No Issues	No Issues
HDD014	No Issues	No Issues	No Issues	No Issues
HDD015	No Issues	No Issues	No Issues	No Issues
HDD016	No Issues	No Issues	No Issues	No Issues
HDD017	No Issues	No Issues	No Issues	No Issues
HRC088	No Issues	No Issues	No Issues	No Issues
HRC095	No Issues	No Issues	No Issues	No Issues
HRC115	No Issues	No Issues	No Issues	No Issues
HRC117	No Issues	No Issues	No Issues	No Issues
HRC120	No Issues	No Issues	No Issues	No Issues

Table 12-3: Smarts Drill Holes Validated Against Hardcopy

Hole ID	Validation			
	Assays	Collars	Geology	DH Surveys
SDD009	No Issues	No Issues	No Issues	No Issues
SDD011	No Issues	No Issues	No Issues	No Issues
SDD013	No Hardcopy	No Issues	No Issues	No Issues
SDD014	No Issues	No Issues	No Issues	No Issues
SDD016	No Issues	No Issues	No Issues	No Issues
SDD017	No Issues	Different X-Y-Z	No Issues	No Issues
SDD019	No Issues	No Hardcopy	No Issues	No Issues
SDD024	No Issues	No Issues	No Issues	No Issues
SDD026	No Issues	No Issues	No Issues	No Issues
SDD027	No Issues	No Issues	No Issues	No Issues
SDD028	No Issues	No Issues	No Issues	No Issues
SDD029	No Issues	No Issues	No Issues	No Issues
SDD030	No Issues	No Issues	No Issues	No Issues
SDD031	No Issues	No Issues	No Issues	No Issues
SDD034	No Issues	No Issues	No Issues	No Issues
SDD035	No Issues	No Issues	No Issues	No Issues
SDD039	No Issues *	No Hardcopy	No Issues	No Issues
SDT417	No Issues	No Issues	No Issues	No Issues
SDT442	No Issues	No Issues	No Issues	No Issues
SDT513	No Issues	No Issues	No Issues	No Issues
SDT590	No Issues	No Issues	No Issues	No Issues
SDT591	No Issues	No Issues	No Issues	No Issues

SDT593	No Issues	No Issues	No Issues	No Issues
SRC625	No Issues	No Issues	No Issues	No Issues
SRC648	No Issues	No Hardcopy	No Issues	No Issues
SRC656	No Issues	No Issues	No Hardcopy	No Hardcopy
SRC669	No Issues	No Issues	No Hardcopy	No Hardcopy
SRC672	No Issues	No Issues	No Hardcopy	No Hardcopy
SRC675	No Issues	No Issues	No Hardcopy	No Hardcopy
*** Lab results came after database closure during Dec 2012. Database has no assays for SDD039**				

12.4.3 Results of Data Verification

Through the processes of the site visits, data review, analysis of QAQC data and validation of digital data against original drilling records the exploration data compiled in the drillhole database and the exploration procedures undertaken by Troy has been verified.

The digital data correctly represents the original data gathered from field activities. Differences found between the digital data and field location or hard copy data for previous drilling were resolved. These differences reflect updated information correctly located within the database. Data validation for recent drilling discovered no discrepancies. It is the author's opinion that the data is sufficient for the purposes of this technical report and that the data is suitable for further geological studies.

13. Mineral Processing and Metallurgical Testing

13.1 Hicks Deposit

An initial metallurgical program comprising eleven samples from the Hicks Deposit was completed at ACME Metallurgical Laboratories in Vancouver, Canada in February 2012. The metallurgical program tested the amenability of gold-bearing material to both cyanide leach and gravity recovery methods. The composites were chosen to represent a range of grades from both oxidised and fresh material. In addition, one composite of fresh material representing a spread of grades from spatially diverse locations within the deposit was also tested.

The leach tests were conducted for 72 hours at a target grind size of P80 = 75 microns and at 40% solids pulp density with industry standard reagent addition rates maintained. For the leach program, the average recovery for the 11 composites was 93%. The high extraction rates (recoveries) achieved in both oxidised and fresh materials indicates that gold mineralisation at the Hicks Deposit is not refractory and both oxide and fresh material will be amenable to treatment by conventional CIP or CIL. Test data is shown in Table 13-1 and Table 13-2 hereafter.

The Smarts Deposit metallurgical test-work also comprised 11 composites and was tested at ACME Metallurgical Laboratories in Vancouver in October 2011. The metallurgical program targeted definition of Leaching and Gravity performance as separate processes. The composites were chosen to represent a range of grades from both oxidised and fresh material. In addition, two composites each of oxide and fresh material representing a spread of grades from spatially diverse locations within the deposit were also tested. The leach tests were conducted for 72 hours at a target grind size of P80 = 75 microns and at 40% solids pulp density.

For the leach program, the average recovery for the 11 composites was 93%. Not all samples displayed complete leaching at 72 hours, indicating the possible presence of free gold. This presents opportunities for further improvement of recoveries with the integration of gravity and leaching processes. Test data is shown in Table 13-3 hereafter.

Further metallurgical testwork was also undertaken at AMMTEC in Perth during 2013 as a part of the Scoping Study. The Karouni gold ore was found to be free milling and gravity gold recovery was necessary to achieve high overall gold recoveries. A gravity gold recovery of 40 to 50% could be expected. The ore has approx. 0.5% sulphur is low in silver and contains no deleterious elements. The oxide ore is relatively soft whilst the primary ore has a range of Bond Ball Work Indices from 14kWh/t 6 to 18.4kWh/t which is moderately hard. The Abrasion Index was low so crushing liners and ball consumption should be low.

Table 13-1: Hicks Deposit Azimuth Metallurgical Testwork ACME Labs October 2011				
Test	**Type and Location of Material**	**Gold Head Grade (g/t) Au**	**Gold Recovery (72 hours)**	**NaCN Consumption (kg/t)**
C1	Oxide - HRC007 (32-33m)	0.79	91.4	1.4
C2	Oxide - HRC007 (27-31m)	2.44	89.8	1.6
C3	Oxide - HRC001 (6-9m)	0.10	94.0	1.0
C4	Oxide - HRC007 (17-22m)	2.23	96.0	1.0
C5	Oxide - HRC101 (9-15m)	1.29	95.5	0.9
C6	Fresh - HRC079 (61-62m)	0.57	93.0	1.0
C7	Fresh - HRC018 (113-114m)	1.84	96.5	0.6
C8	Fresh - HRC004 (72-75m)	4.90	89.4	0.3
C9	Fresh - HRC004 (69-70m)	4.01	88.0	0.7
C10	Fresh - HRC079 (60-61m)	0.66	94.3	0.5
C11	Fresh - Bulk Rejects Compilation	0.63	95.8	0.4
Average: All Samples		1.77	93.1	0.9
Average: Oxide Samples		1.37	93.3	1.2
Average: Fresh Samples		2.10	92.8	0.6

Hicks Cyanide Leach Program Results								
Test	**Composite**	**Gold Grade (g/t) Au**			**Gold Recovery (%)**		**Mass Pull (%)**	
		Gold Head Grade	**Gravity Con**	**Pan Con**	**Gravity Con**	**Pan Con**	**Gravity Con**	**Pan Con**
G1	Oxide - HRC007 (32-33m)	0.41	2.4	8.8	21.2	3.3	3.6	0.2
G2	Oxide - HRC007 (27-31m)	2.97	13.9	83.0	24.6	3.2	5.1	0.1
G3	Oxide - HRC001 (6-9m)	0.05	0.8	3.6	51.3	17.5	3.7	0.3
G4	Oxide - HRC007 (17-22m)	2.31	9.0	141.8	23.5	6.5	6.1	0.1
G5	Oxide - HRC101 (9-15m)	1.42	3.7	67.3	24.1	4.5	7.4	0.1
G6	Fresh - HRC079 (61-62m)	0.35	3.8	48.5	55.0	14.9	3.8	0.1
G7	Fresh - HRC018 (113-114m)	1.56	8.5	237.8	28.3	9.6	5.9	0.1
G8	Fresh - HRC004 (72-75m)	5.32	43.4	230.6	54.1	21.0	6.1	0.4
G9	Fresh - HRC004 (69-70m)	3.91	37.7	284.6	45.7	6.7	4.7	0.1
G10	Fresh - HRC079 (60-61m)	0.53	5.4	185.0	55.0	19.6	6.4	0.1
G11	Fresh - Bulk Rejects Compilation	0.61	5.6	144.2	62.5	26.7	5.6	0.1
Average: Oxide Samples					40.5	12.1	5.3	0.2
Average: Fresh Samples					28.9	7.0	5.2	0.2
Average: All Samples					50.1	16.4	5.4	0.2

Table 13-2: Hicks Deposit Azimuth Gravity Amenability Testwork ACME Labs October 2011				
Hicks Gravity Amenability Program Results				
Test	**Type of Material**	**Head Gold Grade (g/t Au)**	**Gold Recovery (72 hours)**	**NaCN Consumption (kg/t)**
C1	High Grade Oxide	14.98	92.7	1.37
C2	High Grade Fresh	6.59	92.7	1.32
C3	Medium Grade Oxide (no quartz veining)	2.01	87.4	1.25

C4	Low Grade Oxide (no quartz veining)	0.86	95.9	1.05
C5	Medium Grade Fresh	3.61	90.1	1.05
C6	Medium Grade Fresh	2.54	92.0	1.28
C7	Low Grade Oxide (with quartz veining)	0.63	95.7	0.62
C8	Higher Grade Fresh	4.05	94.8	1.13
C9	Low Grade Fresh (with quartz veining)	0.87	88.8	1.35
C10	Oxide Composite of Low, Medium and High Grades	9.75	96.9	1.19
C11	Fresh Composite of Low, Medium and High Grades	4.34	96.4	0.96
Average: All Samples		4.57	93.04	1.14
Average: Oxide Samples		5.64	93.72	1.1
Average: Fresh Samples		3.67	92.47	1.18

Table 13-3: Smarts Deposit Azimuth Cyanide Leach Testwork

Smarts Cyanide Leach Program Results

Test	Type of Material	Head Gold Grade (g/t Au)	Pan Concentrate Mass (%)	Gold Recovery (%) (to concentrate)
G1	High Grade Oxide	13	0.1	11.6
G2	High Grade Fresh	8.3	0.2	49.8
G3	Low Grade Oxide (no quartz veining)	1.3	0.2	46
G4	Low Grade Oxide (no quartz veining)	0.7	0.1	33.5
G5	Medium Grade Fresh	2.1	0.2	24.3
G6	Medium Grade Fresh	2	0.3	21.2
G7	Low Grade Oxide (with quartz veining)	0.6	0.2	55.8
G8	Higher Grade Fresh	3.4	0.1	25.2
G9	Low Grade Fresh (with quartz veining)	0.6	0.2	20.8
G10	Oxide Composite	7.8	0.1	44.8
G11	Fresh Composite	3.6	0.1	57.5
Average: Oxide Samples		4.68	0.14	38.3
Average: Fresh Samples		3.33	0.18	33.1
Average: All Samples		3.94	0.16	35.5

Following a gap analysis, further metallurgical testing is currently under way at Bureau Veritas in Perth, WA to test the potential for increasing recovery and to aid in equipment sizing and operating parameters.

A program which was conducted by Bureau Veritas in Perth Western Australia, investigated the response of the three ore zones consisting of primary, shear and oxide, to a range of processing parameters. This included comminution, leaching, and processing characterisation.

The testwork samples were selected based on geological data and review of the core which was provided by Troy for the testwork program. Testing was staged in order to confirm gold containing intervals as soon as possible prior to the continuation of gold recovery testwork. Initially only comminution testing that required material larger than 3.35mm was conducted, prior to assaying the material on a per metre basis. Composites were formed based on this assay data for the remainder of the testing, both for variability and for master composites.

Comminution Results

Comminution testing was conducted on all zones of the ore body. Depending on the sample mass available and condition of the core provided, not all zones were could be tested for all comminution parameters.

Unconfined Compressive Strength (UCS)

UCS results showed that there should be no concerns in processing any of the ores tested through a standard jaw crusher, based on the required energy input. The oxide material was classified as very weak, whilst the primary and shear materials were classed in the range of weak to medium-strong.

The UCS results are presented in Table 13-4 below, showing the ranking, average, maximum, minimum values and the standard deviation for each hole and zone tested.

Table 13-4: UCS Results Summary

Hole	Zone	UCS Ranking	Average	Max	Min	Standard Deviation
99	Primary	Medium-Strong	25.65	45.62	12.93	14.29
100	Oxide	Very Weak	2.75	3.30	2.20	0.55
100	Primary	Weak	8.85	14.35	3.50	4.43
101	Shear	Medium-Strong	29.53	80.57	8.37	27.08
102a	Primary	Weak	21.76	39.56	10.59	7.52
102a	Shear	Weak	16.67	45.50	4.48	14.60
103	Primary	Medium-Strong	28.14	32.52	20.75	5.26

Crushing Work Index (CWi)

An average of the results of the CWi tests showed that all material could be classified as “very soft”. There will be no identified issues as a result of any of these ores being treated through the crushing circuit as presented in the design. The results ranged from 0.4kWh/t to 11.2kWh/t for one of the shear samples. These results show that there should be no issue for processing through a standard jaw crusher.

The CWi results are presented in Table 13-5 below, showing the ranking, average, maximum, minimum values and the standard deviation for each hole and zone tested.

Table 13-5: CWi Results Summary

Hole	Zone	# CWi	CWi (kWh/t, average)	Max	Min	Standard Deviation	Ranking
99	Primary	21	3.13	4.79	2.39	0.73	Very Soft
100	Oxide	20	1.46	3.52	0.37	1.05	Very Soft
101	Primary	15	3.87	6.01	3.38	0.73	Very Soft
101	Shear	20	4.64	9.62	3.77	1.80	Very Soft
102a	Primary	30	4.24	7.25	3.44	1.12	Very Soft
102a	Shear	15	5.14	11.21	2.41	2.18	Very Soft
103	Primary	21	4.42	9.51	2.64	1.97	Very Soft
103	Shear	19	5.12	7.69	2.36	1.43	Very Soft

SAG Mill Comminution (SMC)

SMC testing is conducted to further evaluate ore hardness and competency for size reduction. This testing uses a smaller particle starting size than that of the crushing tests. This testing is used to determine the power requirements and ore performance through the grinding circuit.

The oxide material was not tested through SMC testing due to insufficient mass of material of suitable competency and of the required size fraction being available. Testing was conducted on the primary and shear material.

The shear material was seen to be the least competent, being classified as soft - medium. The primary material was seen to be more competent, being rated as hard to very hard. These rankings were achieved on the basis of the A*b values (material hardness), and the ta values (resistance to abrasion).

The SMC results are presented in Table 13-6 below, showing the SMC A&b values, and A*b and ta ranking for each hole and zone tested.

Table 13-6: SAG Mill Comminution (SMC) Results Summary

Sample ID	A	b	A*b	Rank	ta	Rank
SDD102a - Primary	58.7	0.63	37.0	**Hard**	0.35	**Hard / Mod Hard**
SDD103 - Primary	54.6	0.53	28.9	**Very Hard**	0.28	**Very Hard**
SDD103 - Shear	51.9	0.95	49.3	**Medium**	0.48	**Medium**
SDD102a - Shear	46.3	1.81	83.8	**Soft**	0.83	**Soft**

The shear material should not pose any comminution issues, whilst the harder primary material will likely result in a higher power consumption in the grinding circuit than the shear material. Simulations show however that this should not be a problem for the process plant. Should any significant issues arise, it is probable that blending of the shear and primary materials will assist in negating any increased energy requirement for the primary material.

Bond Ball Mill Work Index (BWi)

The Bond ball mill work index is a standard test for determining the grinding energy required to grind the material undergoing testwork. This test was conducted on the primary and shear material, using the resulting products from the SMC testing, which was -3.35mm which is the particle size required for this test.

The BWi results indicated that all samples were medium to hard, across all zones tested. The range of values was between 12.57kWh/t and 15.28kWh/t, with the highest value being recorded by the primary ore, and the lowest by the shear ore. The BWi results are presented in Table 13-7 below, showing the measured BWi and the ranking for each hole and zone tested.

Table 13-7: BWi Results Summary

Sample	BWi (kWh/t)	Ranking
SDD102a Primary	15.28	Hard
SDD102a Shear	12.57	Medium
SDD103 Primary	13.54	Medium
SDD103 Shear	14.64	Hard

The results of these tests show that there is no expectation that any processing issues will result from the ore grindability. These BWi values were used during the grinding modelling and simulating of the project comminution circuit.

Bond Rod Mill Work Index (RWi)

RWi testing is conducted on coarser material than the BWi test, and this testwork program was conducted primarily to acquire information to compare the RWi and BWi indices. This comparison can be used as an indication of potential grinding performance problems, particularly the build-up of critical sized material in SAG/AG milling.

Rod mill work index testing was conducted on the primary and shear master composites. Both results showed the material to be hard, with results of 19.9kWh/t and 17.5kWh/t for the primary and shear zones respectively.

The RWi results are presented in Table 13-8 and Table 13-9 below, showing the measured RWi and the ranking for each hole and zone tested.

Table 13-8: RWi Results Summary

Sample	RWi (kWh/t)	Ranking
SDD102a Primary	19.9	Hard
SDD102a Shear	17.5	Hard

Table 13-9: RWi Results Summary

Sample	RWi (kWh/t)	BWi (kWh/t)	RWi:BWi
SDD102a Primary	19.9	15.28	1.30
SDD103 Shear	17.5	14.64	1.20

The primary material test results showed that there may be some build-up of critical sized material in a SAG milling circuit. Due to the smaller grinding feed size and higher ball content it is not expected that this will be an issue for the grinding circuit as planned. Should there be any build-up of critical sized material during processing there will be a requirement to recycle any scats discharge from the ball mill.

Bond Abrasion Index (Ai)

The Ai test uses standard testing equipment in order to determine how abrasive an ore is and hence indicate how much wear will be expected in the crushing and grinding circuits, as well as in any transfer chutes.

The primary and shear master composites were tested. The primary material returned a result of 0.13 which classifies this ore as moderately abrasive. The shear material returned a value of 0.09, which is classified as non-abrasive. See Table 13-10 for the results.

Table 13-10: Ai Results Summary

Sample	Ai
SDD102a - Primary	**0.1277**
SDD103 - Shear	**0.0892**

These results indicate that there should be no undue wear experienced in the plant, though the primary material will cause a higher level of wear than that of the shear.

Head Assays

A total of five variability samples were selected from the primary ore zone. The head assays for the 5 variability samples are shown in Table 13-11 below. The variability samples are designated as follows:

- P-V1, Primary Variability Sample 1
- P-V2, Primary Variability Sample 2
- P-V3, Primary Variability Sample 3
- P-V4, Primary Variability Sample 4
- P-V5, Primary Variability Sample 5

Table 13-11: Variability Head Assays

Species	Basis	P-V1	P-V2	P-V3	P-V4	P-V5

Gold (average)	g/t	2.65	14.0	7.5	9.80	5.37
Ag	ppm	0.65	5.65	0.75	0.35	0.25
Cu	ppm	108	40	96	104	64
Zn	ppm	118	104	64	70	82
Ni	ppm	124	428	56	198	246
As	ppm	1	-1	2	3	4
Pb	ppm	4	5	5	6	4
Si	%	25.1	27.3	31.2	25.7	25.4
Fe	%	9.3	8.18	5.94	5.83	7.91
Al	%	7.37	5.76	6.75	6.69	6.58
Ca	%	0.95	0.5	0.65	4.04	2.87
Mg	%	4.06	6.44	2.01	4.21	4.94
Ti	%	0.89	0.56	0.59	0.47	0.65
Na	%	1.05	0.02	3.26	2.6	1.97
K	%	0.42	0.24	0.25	0.32	0.19
Hg	ppb	-10	-10	-10	-10	-10
S	%	1.04	0.28	1.06	0.43	0.41
C	%	0.08	0.12	0.16	1.08	0.82

The variability samples were formed on a per hole basis and were composited to form a range of head grades, from low to high, but around the expected plant feed grade. This would allow the performance of the leaching circuit to be evaluated through a range of gold concentrations. The variability composites were subjected to bottle roll testing at the optimal leach conditions as determined by the master composite testing.

The gold grades of the samples ranged from 2.6g/t gold through to 14.0g/t gold. It can be seen that there is no noticeable correlation between the gold grade and the silver grade present in the samples, whilst the sample with highest silver grade is also the sample with the highest gold grade, the lowest silver grade is seen in P-V4 which has the second highest gold grade at 9.8g/t gold.

Most samples exhibit copper, nickel and zinc levels below that of concern (300ppm each, or 1000ppm total). P-V2 however does have elevated nickel levels at 428ppm nickel. This may cause a slight increase in cyanide consumption during leaching, depending on the nickel species, i.e. whether they are cyanide soluble or not.

Iron grades seen in the samples are below 10% iron. On this basis it is not expected that any issues will be experienced with cyanide consumption, depending on the iron form.

All samples showed very low levels of both arsenic and mercury, with mercury being below detection limits in all samples. This will be beneficial in terms of the occupational health and safety risk, particularly during gold smelting.

Both P-V4 and P-V5 show slightly elevated levels of carbon and so may be a source of “preg robbing” during the leaching and adsorption process. The use of the carbon in leach process will negate this issue to a large extent should it arise.

The total sulphur levels are low overall in all samples, with P-V1 and P-V3 showing the highest levels of total sulphur at just over 1%. Though a high sulphide analysis indicates a possiblility of some reduction in gold

extraction should the gold be sulphide locked, and possibly a slight increase in cyanide consumption, the indication is that this should not be a major cause for concern for the Project.

The master composites formed were submitted for a head assay. Select species from these head assays are shown in Table 13-12 below.

Table 13-12: Master Composite Head Assay Summary

Species	Basis	Primary MC	Shear MC	Oxide MC
Gold (Average)	g/t	11.1	5.9	3.5
Ag	ppm	1.3	0.6	1.4
Cu	ppm	88.0	99.0	100.0
Zn	ppm	71.0	71.0	101.0
Ni	ppm	152.0	417.0	405.0
As	ppm	1.5	2.5	1.5
Pb	ppm	4.0	5.0	5.0
Si	%	28.8	22.9	32.4
Fe	%	6.3	6.9	9.6
Al	%	5.6	5.1	5.7
Ca	%	2.6	5.7	0.1
Mg	%	3.6	6.2	1.0
Ti	%	0.5	0.5	0.5
Na	%	1.8	0.6	0.0
K	%	0.3	0.1	0.2
Hg	ppb	45.0	-10	45.0
S	%	0.5	0.5	0.1
C	%	0.8	1.8	0.2

The summary table was formed from all results received. It should be noted that the values displayed, particularly the gold grades, are reported as averages of those determined by duplicate assays. Where a minus value is shown, this indicates that concentration was below the species detection limit.

It is noted that in the duplicate assays for the primary and shear materials, both returned an outlier on the gold grade in the raw testwork results. Discussions with the laboratory determined that it is likely that these results were switched between the two composites involved. Given the reasonably strong correlation between the other duplicate assays on the gold grade the decision was taken to remove the outliers from the average and use only the conforming values returned. This decision would be validated once the leach work calculated grades are available. This problem is considered to be due to the nuggetty nature of the gold and its effect on the assays. This was shown repeatedly during the extraction testwork and the consolidation of the calculated and assayed head grades.

13.1.1 Primary Master Composite

The primary master composite displayed a high gold grade, with 11.1g/t gold being returned as the average grade, with the outlier as mentioned above removed. Minimal silver was seen within the sample assay, which aligned with the variability sample head assays.

The 88ppm copper, 52ppm nickel and 71ppm zinc were all below levels of concern, and it is not expected that any increased cyanide consumption will be experienced as a result of these species.

The iron level in the sample assay is 6.3%. At these levels it is not expected that reagent consumption increases will be seen due to cyanide soluble iron.

Both the arsenic and mercury levels within the sample assay were below levels of concern, being 1.5ppm arsenic and 45ppb mercury respectively. This is encouraging from a reduced occupational health and safety risk point of view for the process plant.

The carbon level within the sample is low (0.8%), and is hence not expected to cause issues with preg-robbing.

The sulphur grade of the sample is low (0.5%). It is not expected that increased cyanide consumption or an excessive amount of gold locking will be seen as a result of this.

13.1.2 Shear Master Composite

The shear master composite displayed a gold grade of 5.9g/t gold, following the same averaging process as that used for the primary master composite. This sample also has a low silver grade (0.6g/t gold).

The assayed levels of 99ppm copper and 71ppm zinc are below levels of concern, however the nickel concentration is elevated at 417ppm nickel. This higher nickel concentration may cause some issues of increased cyanide consumption during leaching.

The iron content of the sample is higher than the primary sample, however it is still considered to be low at 6.9% iron. It is not expected that this will cause any undue effect on the reagent consumption.

The arsenic levels are very low at 2.5ppm arsenic, and the mercury levels are below detection limits. As with the primary material this indicates a reduced occupational health and safety risk.

The carbon is slightly elevated at 1.8%. There will be a requirement during plant operation therefore to check for any preg-robbing occurrences.

The sulphide level is low, as with the primary ore and it is not expected any issues will be seen during the processing of the ore as a result of this.

13.1.3 Oxide Master Composite

The oxide master composite displayed a gold grade of 3.5g/t gold, using the same averaging method as for the other master composites. The silver level was low, with a grade of 1.4g/t silver.

The copper and zinc levels were below levels of concern, at 100ppm and 101ppm respectively. The nickel content was elevated at 405ppm. This may cause some increase in cyanide consumption during leaching.

The iron content of the sample was 9.6%, which was below levels of concern and should not cause issues with increased cyanide consumption.

The arsenic was low at 1.5ppm, and the mercury was also low at 45ppb. This should present no occupational health and safety issues even though there will be natural concentration of the mercury in the leaching and carbon absorption stages.

The oxide material assays returned the lowest values for both the sulphur grade and carbon grade, at 0.1% and 0.2% respectively, for all the master composites. Hence it is not expected that there will be issues with preg-robbing or with gold locking due to sulphide association.

Master Composite Testing

Master composites of primary, shear and oxide ores were formed. While not all tests were conducted on all composites, the primary, shear and oxide master composites were used to assess optimal leaching conditions and determine predicted process performance. The optimal leaching conditions were determined initially, with these conditions being used for the variability testing.

Gravity – Cyanidation Leach Optimisation

A series of tests were conducted to determine the optimal grind size and cyanide dosage for the leaching process. As part of this testing process the effectiveness of the gravity stage was evaluated. Initial testing replicated the historical testwork, using a 72 hour leach time. It was determined, after the first round of testing, that a 24 hour leach time would be adopted due to negligible improvements in leach recovery being achieved from longer leach times.

The testwork program, conducted on the oxide, primary and shear master composites, determined that a P_{80} of 63 microns would be used for subsequent testing with a cyanide dosage of 250 ppm. Under these conditions all samples displayed rapid leaching kinetics and high recoveries.

The grind sensitivity results as a function of grind size (106, 75 and 45 micron) are presented in Table 13-13 below, showing the calculated head grade, gravity recovery, leach recoveries at 8, 24 & 72 hours, and the lime and NaCN consumption.

Table 13-13: Grind Sensitivity Results Summary as a Function of Grind Size

Composite	Grind Size (P80, um)	Calculated Head Gold Grade (g/t Au)	Gravity Gold Recovery	8h Gold Overall Recovery	24h Gold Overall Recovery	72h Gold Overall Recovery	Lime Consumption (kg/t)	NaCN Consumption (kg/t)
Oxide MC	**106**	**3.15**	**28.4%**	**92.8%**	**94.4%**	**95.3%**	**1.76**	**0.32**
Oxide MC	75	3.15	30.3%	94.5%	97.0%	95.6%	1.76	0.32
Oxide MC	45	2.78	23.7%	93.3%	92.8%	94.6%	1.88	0.34
Primary MC	106	5.78	56.6%	86.4%	89.9%	91.0%	0.93	0.23
Primary MC	75	6.48	58.7%	90.2%	91.8%	93.7%	0.95	0.21
Primary MC	45	8.98	62.2%	96.0%	96.8%	96.6%	1.09	0.23
Shear MC	106	6.09	41.9%	89.8%	93.1%	92.0%	1.08	0.31
Shear MC	75	6.67	42.4%	93.2%	94.9%	94.4%	1.28	0.37
Shear MC	45	5.58	37.5%	95.4%	96.0%	96.4%	1.30	0.33

The grind sensitivity results as a function of cyanide concentration in the slurry are presented in Table 13-14 below, showing the gravity recovery, leach recoveries at 8, 24 & 72 hours, and the lime and NaCN consumption.

Table 13-14: Grind Sensitivity Results Summary (as a Function of Cyanide Concentration)

Grind Size	Cyanide Concentration (ppm)	Gravity	8 Hours	24 Hours	Leach Only	NaCN (kg/t)	Lime (kg/t)
Oxide							
75 µm	500	30.3%	94.5%	97.0%	66.7%	0.24	1.53
63 µm	250	27.9%	94.9%	97.1%	69.1%	0.14	1.33
63 µm	150	35.6%	88.0%	98.2%	62.6%	0.13	1.51

Shear							
75 µm	500	42.4%	93.2%	94.9%	52.5%	0.24	0.93
63 µm	250	50.4%	90.9%	96.7%	46.3%	0.26	0.77
63 µm	150	55.0%	94.6%	97.9%	43.0%	0.07	0.82
Primary							
63 µm	500	67.6%	93.3%	95.0%	27.5%	0.12	0.88
63 µm	250	63.2%	91.0%	93.4%	30.3%	0.09	0.79
63 µm	150	68.5%	89.7%	91.6%	23.1%	0.05	0.85

The gravity recovery varied between the three composites with the oxide material displaying a gravity recovery of around 30%, the shear 40% and the primary 60%. Given the high performance of the gravity stage, it is recommended that a gravity stage be incorporated into the processing circuit.

Mineralogy – QEMSCAN

QEMSCAN was undertaken on heavy liquid separation sinks and floats following desliming of a sub-sample of the primary master composite material.

The floats contained a majority of quartz, feldspar and other silicates. Based on work conducted in the testwork program, the feldspar and silicates present show a negligible negative effect on the rheological properties of the primary ore.

The analysis of the sinks showed that the sulphides identified in the head assays were present as pyrite. Other major minerals present in the sinks include iron oxides and hydroxides, as well as iron silicates. These results give basis to the assumption that the unleached gold remaining in the leach residue may be locked in pyrite. A diagnostic leach could be used to confirm this.

The QEMSCAN results showed no major cause for concern for the processing of these ores.

Oxygen Uptake Testing

Oxygen uptake testing is used to determine how much air is required in the leaching circuit, or if there is a requirement for oxygen sparging due to a higher consumption of oxygen. Oxygen forms an integral part of the chemical reactions in the leaching process.

The oxygen consumption ranged (Table 13-15) between maximum values of -0.024mg/l/min for the oxide, and -0.035 mg/l/min as a maximum value seen for both the shear and primary ore types.

Table 13-15: Oxygen Uptake Results Summary

Sample	Oxygen Uptake Results		Average	Maximum
	Average	Max		
Oxide MC	-0.013	-0.024	Air Sparging Adequate	Air Sparging Adequate
Primary MC	-0.027	-0.035	Air Sparging Adequate	Air Sparging Adequate
Shear MC	-0.024	-0.035	Air Sparging Adequate	Air Sparging Adequate

Oxygen consumption levels under -0.05 mg/l/min typically require air sparging only, with pure oxygen being added in some cases to improve leach kinetics. As the maximum uptake results are far under the levels that require oxygen sparging (-0.15 mg/l/min) it is not expected that there will be any requirement for oxygen during leaching, with slurry agitation assumed to be sufficient to provide the oxygen required for all ore types.

Carbon Loading Testing

Carbon loading tests were conducted on the primary and shear master composites, with the slurry from a bulk leach test on gravity tailings being used to generate a representative carbon loading feed. As a result, the primary material showed a lower gold concentration in solution being used in the tests than the shear, due to the greater amount of gold in the primary ore being recovered by gravity. The slurry was split into three equal parts for a three stage carbon contact test.

The carbon loading tests generate two slurry dependant constants, k and n, which are used in the Fleming equation to evaluate carbon loading performance. Suitable performance is achieved with k values over 200, and n values over 0.6.

The primary material had a k value of 177, and an n value of 0.74. This indicates that while there is good mixing efficiency with the slurry, the rate and efficiency of the carbon loading is lower than would be preferable. Reviewing the results it was seen that the gold concentration in the feed solution was low, due to a high percentage of the gold being recovered by gravity and hence not present in the CIL circuit. The final gold concentration was very low (0.016 mg/l), and hence it is not deemed that there will be any issue with carbon loading of the gold for the primary material and therefore no significant anticipated gold losses in the plant tailings.

The shear material had a higher feed gold concentration, due to the lower gravity recovery when compared to the primary material. The k and n values for the shear test were 202 and 0.70 respectively, resulting in a final gold concentration of 0.048 mg/l. This indicates therefore that there should be no processing issues.

Rheology Testwork

Rheology testing was conducted on all three master composites. Rheology testing analyses the viscosity of the slurry under a range of shear rates, indicating the viscosity of the slurry under high and low shear conditions. High viscosity values can result in increased pumping or some difficulties in slurry transport Based on the expected design of the process plant, the tests were conducted at a P_{80} of 63 micron, at 40, 50 and 60% solids by weight. A range of shear rates was applied to the slurries, with shear rates of 5 s^{-1} and 200 s^{-1} being taken as low shear and high shear conditions respectively for the analysis.

Viscosities under 3000 mPa.s for a low shear, and 300 mPa.s for high shear conditions, are taken as recommended limits and were used in the analysis of results. Viscosities above these levels indicate that there may be difficulty in pumping and/or mixing, or that excessive power requirements may be needed during pumping (high shear) or tank agitation (low shear) conditions (see Table 13-16).

Table 13-16: Rheology Results Summary

Sample		High (mPa.s)	Low (mPa.s)	Viscosity Issues	
				High	Low
Oxide	40%	13.05	92.34	No	No
	50%	65.26	1077	No	No
	60%	610	8311	Yes	Yes
Primary	40%	7.678	0	No	No
	50%	15.36	154	No	No
	60%	62.19	462	No	No
Shear	40%	6.91	0	No	No
	50%	19.96	154	No	No
	60%	117	1385	No	No

It was seen that the oxide material had viscosity issues at 60% solids, with the lower slurry densities (40 and 50% solids) being below the recommended limits, and hence there should be no issue when processing the oxide. There would however, be a requirement for additional power input when pumping the oxide slurry under high density conditions, such as the cyclone feed pumps. This may be avoided through blending with other ore zones, depending on the flexibility of the mining program. As it is not expected that slurry densities at or

above 60% solids are expected in the leach circuit, there is no expectation that any high viscosity slurries will cause issues with tank agitation.

The shear and primary material, under low shear conditions and 40% solids, displayed viscosities of 0 mPa.s. This indicates that in these situations the material was settling out of the slurry. In order to maintain correct agitation the leaching circuit will need to be kept above this slurry density. This issue was not seen at the 50 or 60% solids situations, nor were there any issues with viscosities at levels above the recommended limits.

Based on the current process plant design, with optimal thickener control, it is not expected that any viscosity issues will be seen throughout the process plant.

Thickening Tests - Column

Thickening testwork is used to determine slurry settling rates for thickeners and for determining optimal flocculant dosing. To conduct these tests the slurry was conditioned to pH 10.5. As the material consisted of RC chip samples and therefor already very fine, no additional grinding was conducted.

The tests were conducted on RC chip samples (SRC730 and SRC711) provided by Troy, with three flocculants being tested. It was determined that Magnafloc 351 and 345 had the most potential for the ores tested, due to the clarity of the overflow. The initial testing was conducted at dosing rates of 25g/t.

Both Magnafloc products showed good solid settling performance, with both showing rapid settling kinetics and clear overflows. The results in Table 13-17 show that for both samples tested the Magnafloc 351 showed higher underflow solids density. This flocculant also displayed a higher hindered settling velocity than the 345 product. It was hence recommended to proceed with the Magnafloc 351 as the preferred flocculant in the design work until such time that the Outotec thickening testwork was conducted.

Table 13-17: RC Chip Settling Results

Property	SRC 730		SRC711	
	Magnafloc 351	Magnafloc 345	Magnafloc 351	Magnafloc 345
Free Settling Velocity m/h	25.20	27.06	25.88	27.85
Hindered Settling Velocity m/h	1.14	0.78	0.72	0.60
Underflow % Solids w/w	55.3%	51.6%	56.2%	44.9%

Outotec Thickening Testwork

Dynamic thickener testing was conducted on a sub sample of the master composite material. This material, as in the bulk leach testwork below, was formed from the master composite, and ground to a P_{80} of 63 microns. As the processing plant was to operate with a pre-leach thickener, the dynamic thickening tests were conducted on material representative of cyclone overflow, as opposed to leach tailings. The slurry fed to the thickener was to be conditioned to pH 10.5 prior to testing to represent the expected plant slurry. Outotec also provided recommendations for thickener sizing based on the continuous settling tests. This unit sizing test was based on a flowrate of 125tph.

Particle Size Distribution Results

As part of the testwork program, a laser sizing particle size distribution analysis was undertaken on the samples tested. The d_{80} determined from this size distribution was used by Outotec. This was reported as the P_{80} of the sample in the Outotec report supplied, resulting in some variation in the size expected (P_{80} 63 micron) and the size determined, due to the different sizing methods used. The samples tested however were all within plus or minus 5 micron of the required P_{80} size when determined by screening.

Flocculant Screening

Flocculant screening was undertaken by Outotec prior to undertaking the dynamic thickener testing. Five flocculants were tested, including the previously tested Magnafloc 351. Based on the screening process Magnafloc 155 was selected for the dynamic testing. Although not the fastest settling flocculant, Magnafloc 155 provided the best combination of good settling rates and overflow clarity.

Dynamic testing was subsequently conducted on all three samples (oxide, shear and primary). A range of settling flux rates (m^2/t/hr) and flocculant dosages were trialled during the program, with the underflow solids density and the overflow clarity being recorded at each level.

The oxide and shear ores were shown to perform best at 30g/t flocculant dosage, while the primary material was tested at a lower rate of 20g/t. As shown in Table 13-18, all samples tested produced a range of underflow densities, depending on the dosage rate and the flux rate. With a flux rate of 1m^2/t/h, typical for a high rate thickener, the underflow densities are also provided with an upscale from the dynamic test thickener through to the full size unit as defined by Outotec. In this instance all samples tested displayed underflow densities above the required 50%w/w solids.

The primary ore showed a greater amount of solids in the overflow compared to the other samples tested. It is noted that with an increased flocculant dosage from 20g/t to 30g/t, the overflow solids content drops from around 350mg/l down to 210mg/l at a lower flux rate. All samples tested display overflow clarities below 500mg/l, which will be suitable for plant operation and process water recycle.

Table 13-18: Outotec Settling Results

Parameter	Oxide Ore	Shear Ore	Primary Ore
Feed Density (%w/w)	15	15	15
Flocculant dosage (g/t)	30	30	20
U/F density range (%w/w)	47.4-53.2		
U/F density (%w/w), flux of 1	48.9 (51.9 scaled up)	56.5 (59.5 scaled up)	57.3 (60.3 scaled up)
Overflow clarity (mg/l)	~100	~100	~350
High Rate Thickener diameter (m)	13	13	13

The testing also provided an indicated thickener diameter, with all samples tested showing suitable performance in a 13m diameter high rate thickener. It is hence not expected that difficulties will be experienced with any of the ore types using the current 25m thickener selected for the process plant.

Bulk Leach and Detoxification

In order to conduct the detoxification testwork a 10kg sample from the primary and shear master composites was used as feed. To generate the tailings required two bulk leaches were conducted, one for each master composite tested (see Table 13-19). The tests were run under the optimal conditions as determined by the optimisation testwork, being:

- P_{80} 63 micron
- pH 10.5
- Free cyanide 250ppm
- 24 hour leach time
- Gravity pre-leach stage

Table 13-19: Bulk Leach Results

Composite	Grind Size (P80, um)	Calculated Head Gold Grade (g/t) Au	Gravity Gold Recovery (%)	8h Gold Recovery (%)	24h Gold Recovery (%)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)	Final NaCN Conc (ppm)
Primary MC	63	5.74	65.56%	91.51%	94.30%	0.24	0.57	123

Shear MC	63	6.60	40.37%	88.97%	93.94%	0.26	0.60	122

The primary master composite displayed a gravity recovery of 65% in this test, with the overall recovery being 94.3%. These results were similar to those seen in other testing, with the head grade of the feed to the gravity and cyanidation leach test being 5.74g/t. Due to the bulk leach being conducted in a vat leach rather than a smaller bottle roll leach, as expected, the cyanide consumption increased slightly to 0.24kg/t, where the lime was fell to 0.57kg/t. These results are deemed to be closer to those expected for the plant operation.

The bulk leach test on the shear material was run under the same conditions as the primary material, with a feed grade of 6.60g/t. The gravity recovery was, as expected, lower than the primary material at 40%, with an overall recovery of 93.9%. The cyanide and lime consumptions were almost identical to that of the primary, being 0.26kg/t and 0.60kg/t respectively.

The final cyanide concentrations for both leach tests were 122ppm - 123ppm. To better simulate plant operation, no cyanide was added to the leach after the 8 hour mark. The leach slurry was taken directly from the bulk leach and split into equal sizes for use in the detoxification testwork (see Table 13-20 and Table 13-21).

To prevent degradation of the cyanide, the three addition ratios of SO_2:CN to be tested (ratios of 2,3,4) were run in parallel.

The primary material showed strong performance at all addition ratios, with the cyanide level, both free and WAD (weak acid dissociable) cyanide, being below 1ppm after 2 hours. The shear material was unable to achieve free or WAD cyanide levels below 1ppm after 8 hours at the lowest ratio, however at ratios of 3 and 4 both the free and WAD cyanide were seen to be below 1ppm.

Table 13-20: Primary Detox Results

Time (Hours)	SO_2:CN Ratio, 2		SO_2:CN Ratio, 3		SO_2:CN Ratio, 4	
	Concentration (ppm)					
	Free CN^-	$CN_{(WAD)}$	Free CN^-	$CN_{(WAD)}$	Free CN^-	$CN_{(WAD)}$
0	57	60	57	60.00	57	60.00
2	0.54	0.91	0.002	0.02	0.002	0.01
5	0.38	0.59	0.002	0.01	0.002	0.02
8	0.044	0.13	0.002	0.02	0.002	0.02

Table 13-21: Shear Detox Results

Time (Hours)	SO_2:CN Ratio, 2		SO_2:CN Ratio, 3		SO_2:CN Ratio, 4	
	Concentration (ppm)					
	Free CN^-	$CN_{(WAD)}$	Free CN^-	$CN_{(WAD)}$	Free CN^-	$CN_{(WAD)}$
0	55	66	55	66.00	55	66.00
2	6.6	9.8	0.21	0.61	0.002	0.06
5	2.6	6	0.17	0.63	0.002	0.06
8	2.3	5.8	0.002	0.03	0.002	0.02

(a) Variability Testing

Variability testing was conducted on 5 primary samples only, due to the amount of available mass for this ore type. The other ore zones had limited mass and were hence not tested for variability. The variability samples

were formed from high and low points in the primary ore, as well as over a range of gold grades to assess the effect of grade and depth variation.

The results (summarised below in Table 13-22) show that the gravity recovery ranged between 42% and 79%, almost independent of the head grade. Overall recovery ranged from 79% to 97%. There was a slight dependence on head grade, with the two lowest head grades displaying the lowest overall recovery.

The cyanide consumption was low for all samples tested, whilst the lime consumption was at levels below concern. It is thus concluded that there is no need for reagent costs to be incurred for the primary ore.

Table 13-22: Variability Leach Results Summary

Sample	Gold Head Grade (g/t) Au	Gold Recovery (%) Leach Time (Hours)					NaCN Consumption (kg/t)	Lime Consumption (kg/t)
		0	2	4	8	24		
P-V1	2.59	58.4	76.1	77.9	79.5	80.4	0.09	1.25
P-V2	11.12	42.6	78.8	87.3	92.4	92.8	0.12	1.15
P-V3	3.94	43.5	70.0	75.5	76.4	79.2	0.08	0.91
P-V4	7.80	79.3	88.4	92.2	95.6	97.4	0.10	0.83
P-V5	4.58	58.2	78.5	84.8	87.9	95.2	0.07	1.03

14. Mineral Resource Estimates

14.1 Overview

Resource outlines were constructed using cross sectional interpretations prepared using logged geology and a nominal gold cut-off grade of 0.5g/t gold however some lower grade intersections were included. A minimum downhole intersection of 2m was used with no edge dilution and minor zones of internal dilution were included to maintain continuity of the Resource wireframes.

The LiDAR survey was used to create a topographic surface. Geological logging was used to create weathering surfaces for the top of fresh (TOFR) and base of oxidation (BOCO) surfaces as well as a base of transported overburden surface. A cross-cutting dolerite dyke at the Hicks Prospect was also interpreted.

All modelling and wireframing were completed using Vulcan Software Version 9. Statistical analysis and variography were completed using Vulcan software. The estimates for the deposits used standard Vulcan block models with Ordinary Kriging (OK) grade interpolation and inverse distance squared for Larken, within wireframes and the models have been reported at a 1.0g/t gold cut-off. The Mineral Resource estimate for the Karouni Project is summarised below Table 14-1.

Table 14-1: Karouni Gold Project Mineral Resource Estimate (Effective Date: June 30th 2014)

SMARTS ZONE (1g/t Cut-off)

Type	Measured			Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces
Oxide	186,000	5.7	34,000	15,000	2.8	1,300	15,000	2.5	1,200
Transitional	209,000	5.8	39,200	16,000	2.5	1,300	21,000	3.2	2,100
Fresh	1,810,000	4.8	278,500	940,000	3.7	111,000	2,232,000	2.5	181,700
Total	**2,205,000**	**5.0**	**351,700**	**971,000**	**3.6**	**113,600**	**2,268,000**	**2.5**	**185,000**

HICKS ZONE (1g/t Cut-off)

Type	Measured			Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces
Oxide	-	-	-	252,000	2.2	17,900	31,000	1.6	1,600
Transitional	-	-	-	75,000	2.1	5,000	35,000	1.6	1,800
Fresh	-	-	-	2,591,000	2.1	175,800	3,106,000	1.9	193,000
Total	**-**	**-**	**-**	**2,918,000**	**2.1**	**198,700**	**3,172,000**	**1.9**	**196,400**

LARKEN ZONE (1g/t Cut-off)

Type	Measured			Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces
Oxide	-	-	-	-	-	-	11,000	3.9	1,400
Transitional	-	-	-	-	-	-	20,000	5.6	3,600
Fresh	-	-	-	-	-	-	221,000	3.5	25,100
Total	**-**	**-**	**-**	**-**	**-**	**-**	**252,000**	**3.7**	**30,100**

TOTAL KAROUNI GOLD PROJECT (1g/t Cut-off)

Type	Measured			Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces

Oxide	186,000	5.7	34,000	267,000	2.2	19,200	57,000	2.3	4,200
Transitional	209,000	5.8	39,200	91,000	2.2	6,300	76,000	3.1	7,500
Fresh	1,810,000	4.8	278,500	3,531,000	2.5	286,800	5,559,000	2.2	399,800
Total	**2,205,000**	**5.0**	**351,700**	**3,889,000**	**2.5**	**312,300**	**5,692,000**	**2.2**	**411,500**

Note: Numbers may not add up due to rounding.

It is not anticipated that this Mineral Resource estimate will be materially affected, to any extent, by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors.

14.2 Hicks Deposit

14.2.1 Preparation of Wireframes

14.2.1.1 Resource Mineralisation

The interpreted sectional outlines were manually triangulated to form wireframes as shown in Figure 14-1. Geological information from trenches was used to guide the interpretation of mineralisation boundaries. Information from drilling was used in preference where generally minor contradictions in trench and drill data were noted. This reflects the historical nature of the trench information and the more uncertain location of the trench sample lines.

To form ends to the wireframes, the end section strings were copied to a position midway to the next section or to approximately 20m depending on sample information along strike or down dip. The end section strings were adjusted to match the dip, strike and plunge of the zone. The wireframed objects were validated using Vulcan software and set as solids.

A total of 7 mineralisation domain wireframes were created and used to select the sample data to be used for grade estimation, and to constrain the block model for estimation purposes. The mineralisation wireframes were treated as hard boundaries for all estimation purposes, that is, only assays from within each wireframe were used to estimate blocks within that wireframe.

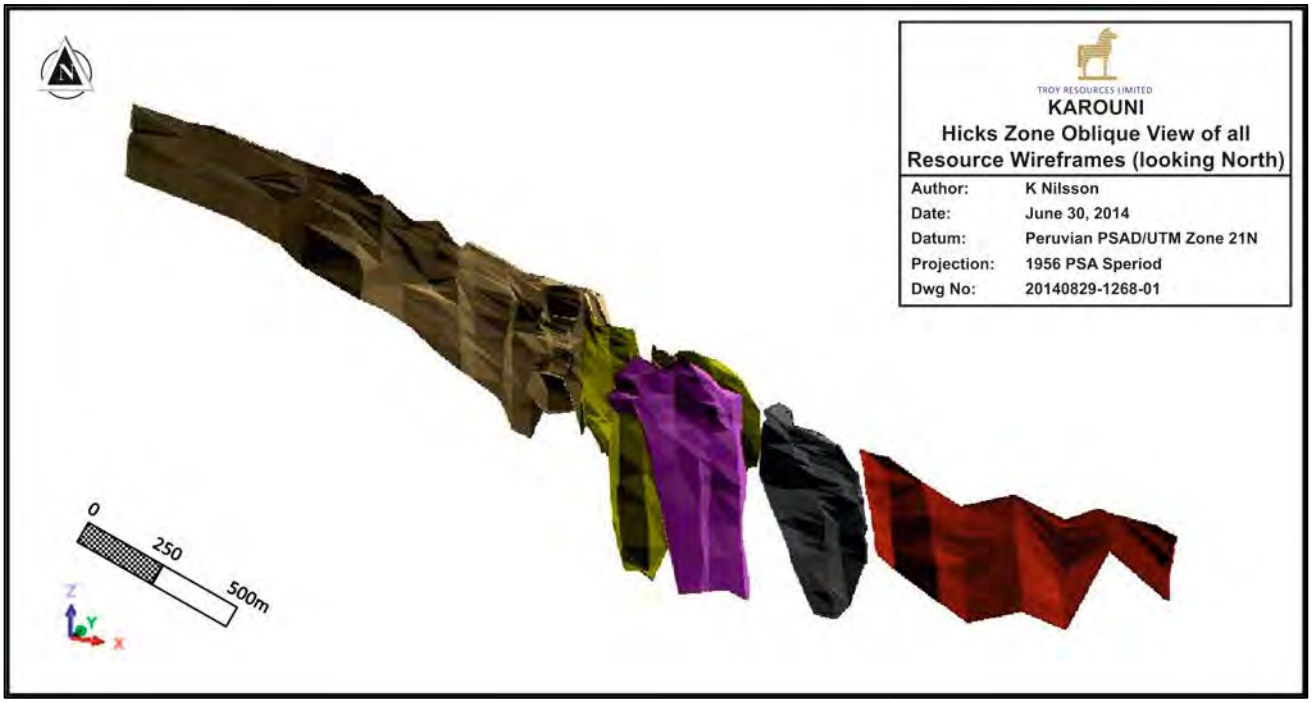

Figure 14-1: Oblique View of all Hicks Resource Wireframes (View Looking North)

14.2.1.2 Weathering Surfaces

Weathering surfaces were interpreted on cross section and extended to cover the block model limits. Weathering codes and degree of oxidation were included in the geology database and used to define the base of complete oxidation (*boco_2014.00t*) and top of fresh rock (*tofr_2014.00t*) surfaces.

The database geological codes used to define the weathering surfaces are presented in Table 14-2. The surfaces were mainly defined from the database field "Regolith".

Conversely, both the regolith zone and the transition zone (i.e. base of complete oxidation) are defined by geological codes starting with 'R'. The regolith unit is generally composed of the Rpa (pallid zone), Rmo (mottled zone), Rlt (laterite), Rfe (ferricrete), and Rsp (saprolite). The weathering of these units may vary, but generally spans from moderately weathered (mw) to extremely weathered (ew). It should be noted that the degree of weathering is not directly linked to the degree of oxidization, but to the competency of the rock.

The transition zone refers to the transition between the regolith and fresh rock. The database geological code used is Rsr (saprock). Since this does define the transition between regolith and fresh rock, this unit tends to be weakly weathered (ww).

Table 14-2: Hicks Zone Geological Codes for Weathering

Type	Database field "Regolith" (generalised)	Database field "Weathering" (generalised)	Block Model Code
Above Surface	-	<empty>	air
Sand Overburden	Osd, Ocy, Osl, Oal	<empty> to ww	ob
Oxide (Regolith)	Rpa, Rmo, Rlt, Rfe, Rsp	mw to ew	ox
Transition	Rsr	ww to mw	tr
Fresh	Fr	Fr	fr

14.2.1.3 Geological Surfaces

The overburden unit is defined by geological codes starting with 'O', and is generally composed of Osd (overburden sand) and Ocy (overburden clay) (see Table 14-2). The overburden surface (*overburden_2014.00t*) was interpreted on cross section and extended to cover the block model limits.

14.2.1.4 Topographic Surface

The surface elevation is moderately undulating and ranges from 55m to 90m in elevation above sea level. This reflects the incisement of overburden and basement lithologies by numerous streams.

The current topographic surface was coded from a LiDAR survey undertaken in late 2011 (*contour_wo_resource_2m.00t*). The survey was undertaken by Digital World Mappine Inc. and is nominally accurate to +/- 0.3m on flat terrain. The surface was thinned and re-contoured to reduce file size, however trench excavation lines are still generally visible and the surface is considered to be of excellent quality.

Drill collars, trench excavation lines and the LiDAR topographic surface generally show a reasonable correlation. To ensure consistency drill collars were adjusted in the Z direction (elevation) to match the LiDAR surface. Collar adjustments were generally minor or reflected corrections to either historical drill collars or where a hand held GPS had been used.

14.2.1.5 Dolerite Dyke

A near-vertical barren dolerite dyke of approximately 80m true thickness orthogonally cross-cuts the mineralisation at the Hicks Zone. Drilling and trench assay and logging information was used to constrain the position of the dyke boundaries (see Figure 14-2). The orientation of several drillholes were designed to intersect the dyke position and this information, along with field mapping of the dyke, where exposed in creek beds has accurately fixed the position of the hanging wall and footwall positions of the dyke.

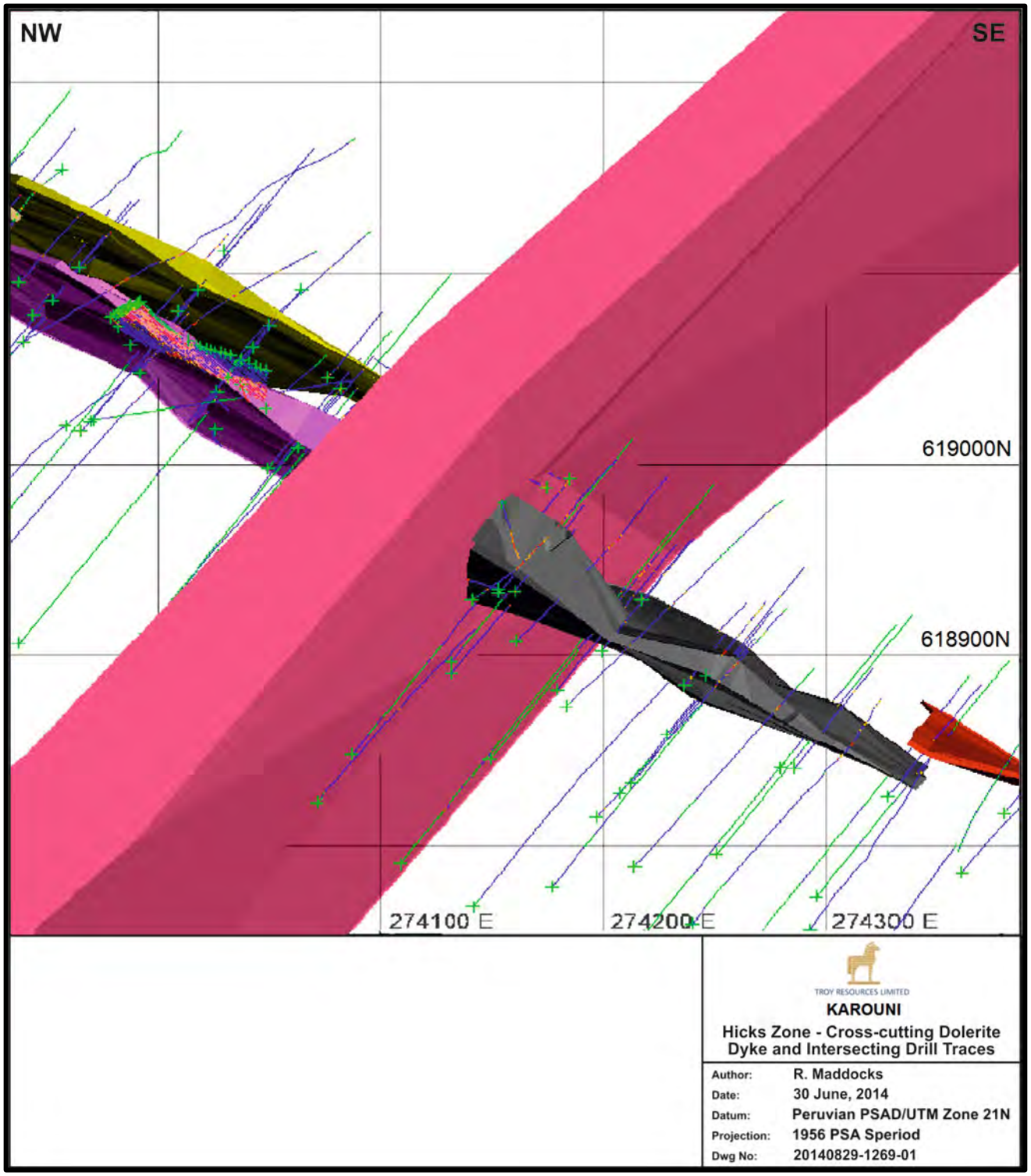

Figure 14-2: Hicks Deposit Cross-cutting Dolerite Dyke and Intersecting Drill Traces

14.2.2 Sample Statistics

14.2.2.1 General

The wireframes of the mineralised zones were used to define the Resource intersections. The drill samples used to interpret the mineralisation are dominantly of 1m length although some sample interval variation does occur to reflect geological boundaries. A 1m interval length was selected for compositing and further geostatistical analysis. This composite length reflects the majority of sample intervals and is at a suitable

scale for the width of the Resource wireframes. Any residual composites of less than 0.5m in length were added to the preceding composite.

Vulcan software was then used to extract 1m down-hole composites within the intervals coded as Resource intersections. A small number of negative assays representing assay detection limit were converted to 0.005g/t grades.

The composites were checked for spatial correlation with the objects, the location of the rejected composites and zero composite values. A master composite file was created including each of the individual Resource objects in the wireframe model.

14.2.2.2 Deposit Statistics

The composite sample data for the Hicks Deposit was grouped by ore zone object. Summary statistics for gold are shown in Table 14-3 for all the ore zone solid triangulations.

An analysis was conducted on the statistics of the different modelled weathering zones. All gold sample data was used for this comparison which showed that there is only minor difference in grade population distribution between the oxide, transitional and fresh samples (Figure 14-3). A statistical review indicated that some gold enrichment of the weathering and transition zones has occurred at the Hicks Deposit as indicated in Table 14-4, however the impact on the overall Mineral Resource estimate is minimal given the relatively small number of tonnes associated with the oxide and transition zones compared to the fresh mineralisation.

It was concluded that an estimation hard boundary at the weathering surfaces was not necessary.

Table 14-3: Hicks Zone Summary Statistics by Domain for Gold

Ore Zone	Volume	No of Composites	Low	High	Mean	Variance	Standard Deviation	CV
A	1,066,731	822	0.005	109.77	1.93	29.22	5.41	2.80
B	656,171	354	0.003	37.41	1.69	11.27	3.36	1.98
C	487,335	312	0.000	27.69	2.22	12.47	3.53	1.59
D	506,052	375	0.005	20.91	1.99	9.27	3.04	1.53
E	434,461	130	0.005	19.52	1.57	7.81	2.80	1.78
F	9,323	18	0.000	8.87	2.16	7.30	2.70	1.25
G	75,224	86	0.000	19.98	1.82	11.52	3.39	1.86

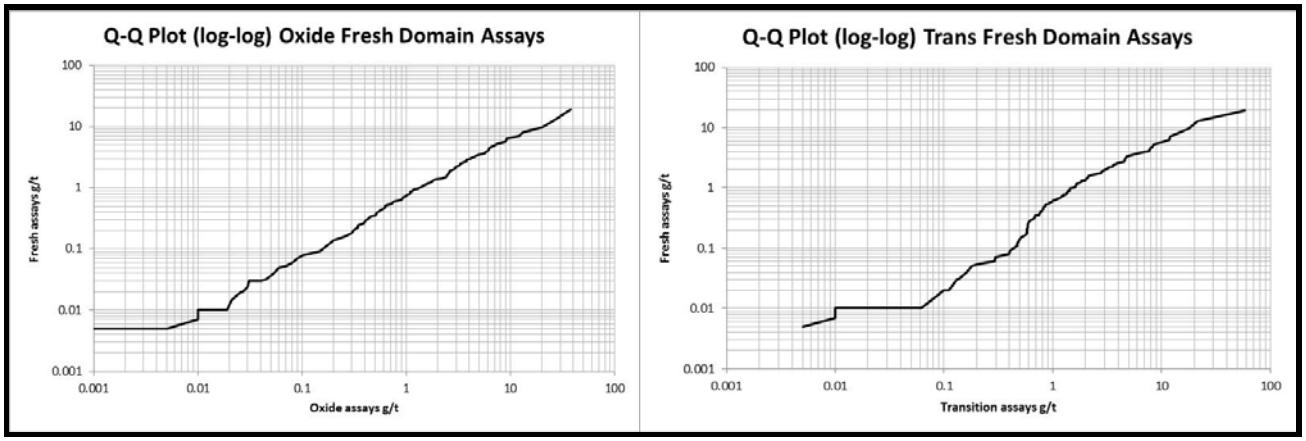

Figure 14-3: Hicks Zone Q-Q Plot by Weathering Domain

Table 14-4: Hicks Zone Statistics by Weathering Domain

Domain	Samples	Minimum	Maximum	Mean	Stand. Dev.	CV
Oxide	279	0.001	37.41	2.40	4.40	1.83

Transition	113	0.001	58.14	2.96	6.32	2.14
Fresh	1,024	0.001	109.77	1.79	4.44	2.48

14.2.3 High Grade Cuts

To assist in the selection of appropriate high grade cuts, the composite data was interrogated and a population histogram, log probability plot and the coefficient of variation (CV) statistics were generated for the modelled ore zone domains. High grade cuts were determined by noting distinct breaks in the shape of the distribution on the log probability plot and population histogram. High CV values also indicate the need for high grade cuts.

It was determined that high grade cuts were required for four of the seven objects. High grade cut statistics for each applicable object are listed in Table 14-5 below.

Table 14-5: Hicks Zone High Grade Cuts by Domain

Object	Samples	Before HG Cut		HG Cut	No. Cut	Percentage (%)	After HG Cut	
		Mean	CV				Mean	CV
A	822	1.93	2.80	20	7	0.9%	1.70	1.69
B	354	1.69	1.98	20	3	0.8%	1.62	1.71
C	312	2.22	1.59	20	3	1.0%	2.18	1.51
D	375	1.99	1.53	20	3	0.8%	1.98	1.52
E	130	1.57	1.78	20	0	0.0%	1.57	1.78
F	18	2.16	1.25	20	0	0.0%	2.16	1.25
G	86	1.82	1.86	20	0	0.0%	1.82	1.86

The Mineral Resource estimate is reported using cut gold values.

14.2.4 Geostatistical Analysis

Downhole and three orthogonal variograms were modelled. Directions were based on the strike and dip direction of the modelled ore zone domains. The ore zones all display long major axes along the NW-SE direction and shorter semi-major and minor axes.The variograms displayed reasonable structure.

The experimental variograms are normalised against the sample variance so that the sill value is 1 and the structures are viewed as ratios or proportions of the sill.

A two structured nested spherical model was found to model the experimental variogram well. The downhole variogram provides the best estimate of the true nugget value which is 0.1.

The orientation of the plane of mineralisation was aligned with the interpreted wireframes. The experimental variograms were calculated with the first aligned along the main mineralisation continuity while the second is aligned in the plane of mineralisation at 90° to the first orientation. The third is orientated perpendicular to the mineralisation plane, across the width of the mineralisation. See Figure 14-4.

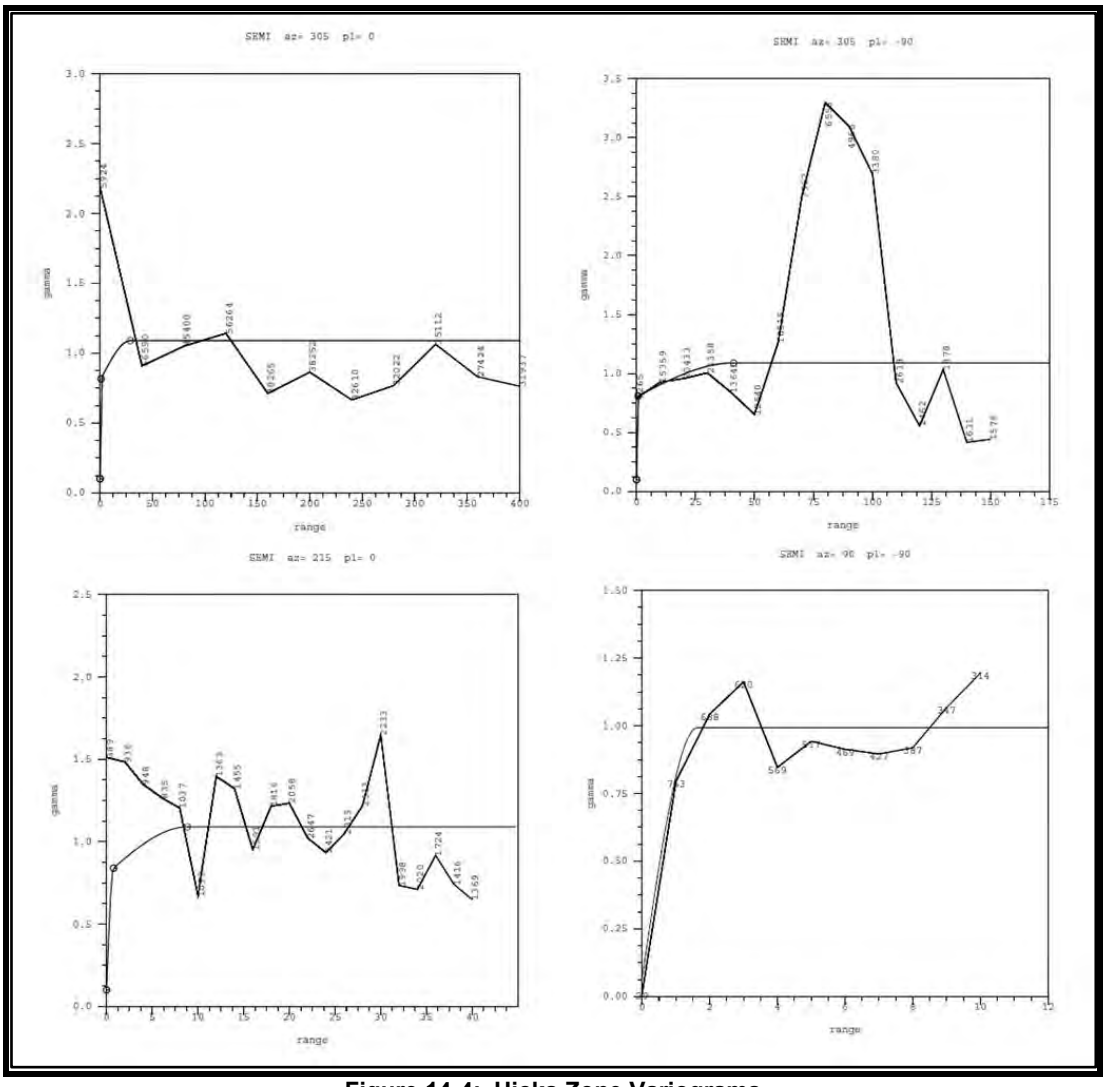

Figure 14-4: Hicks Zone Variograms

14.2.5 Kriging Parameters

The gold composite grades were interpolated into a Vulcan block model using the ordinary kriging (OK) algorithm using the nugget, sill values and ranges determined from the variogram models discussed in Section 14.2.4. The ranges obtained from the variogram models were used as a guide in determining the search ellipse parameters used in the resource estimate. The kriging parameters resulting from analysis of the main lode (a) were applied to the relevant adjacent minor lodes.

14.2.6 Block Model

A Vulcan block model was created to encompass the full extent of the Zone. A list of block model parameters is displayed in Table 14-6 below.

The block model used a primary block size of 20m along strike (NW-SE) by 4m across strike by 10m vertical with sub-cells of 5m by 1m by 2.5m. The block model was rotated to 305 degrees on the horizontal plane to match the strike direction of mineralisation. Block model co-ordinates are on the national grid PSAD56-UTM Zone 21N to match the trench and drillhole database.

The parent Y block size was selected on the basis of approximately half of the average drill section spacing in better drilled areas of the Hicks Deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. In addition, the selection of a 10m model cell height reflected the spacing of exploration drilling down-dip.

Table 14-6: Hicks Zone Block Model Parameters			
Model Name	**Hicks_Nov 2013.bmf**		
	Y	**X**	**Z**
Minimum Coordinates	**617,690**	**275,330**	**-400**
Maximum Coordinates	**621,490**	**276,130**	**150**
Block Size (Sub-blocks)	**20 (5)**	**4 (1)**	**10 (2.5)**
Rotation	305° (horizontal plane)		
Attributes:			
Au_cut	Estimated gold grade from high grade cut composites (g/t) - Reportable		
Class	Resource Classification - measured (mes), indicated (ind), inferred (inf).		
lith	Rock type, waste, ore. dyke		
Ave_dis	Average distance to interpolate Au.		
Auid	Inverse distance squared grade		
Num_sam	Number of samples to interpolate Au_cut.		
kv	Kriging variance from interpolation of Au_cut.		
Pass	Interpolation pass number from interpolation of Au.		
Bd	Bulk Density.		
weath	Weathering type, fr, tr, ox, ob		
Type	Geological Domain – air(air), overburden (ob), oxide (ox), transition (tr), fresh (fr).		

14.2.7 Grade Interpolation Parameters

For all zones in the Hicks Zone, the wireframe objects were used as hard boundaries in the interpolation. That is, only grades inside each wireframe object were used to interpolate the blocks inside the coded object. The Ordinary Kriging (OK) algorithm was selected for grade interpolation. This was to allow the measured spatial variation of gold mineralisation to be incorporated into the model.

An orientated 'ellipsoid' search was used to select data for interpolation. The ellipse was oriented to the average strike, dip and plunge of the mineralised zones, and varied accordingly within each object. The maximum search radius was set at a distance similar to the variogram range for the major direction or as required to ensure all blocks were estimated in the final kriging pass. The major to semi-major and the major to minor search ratios were determined from the variogram ranges.

A two-pass estimation search was used to complete estimation for gold within the domain objects. A first pass search radius of 50m and a second pass search radius of 200m along strike were used with a minimum number of samples of 10 for the first pass and 5 for the second.

Estimation parameters used in the estimate are listed in Table 14-7.

Table 14-7: Hicks Zone Ordinary Kriging (OK) Estimation Parameters for Gold		
Parameter	**Pass 1**	**Pass 2**
Search Type	**Ellipsoid**	**Ellipsoid**
Bearing	305°	305°
Plunge	0°	0°
Dip	-90°	-90°
Major axis search radius	50m	200m
Semi-Major axis search radius	20m	50m

Minor axis Search Radius	50m	100m
Minimum Samples	10	5
Maximum Samples	40	40
Block Discretisation	3 X by 4 Y by 4 Z	3 X by 4 Y by 4 Z
Percentage Blocks Filled	49%	51%

14.2.8 Bulk Density and Material Type

Bulk density values were assigned to each weathering material type based on values compiled from historical and recent bulk density test work for the Hicks and Smarts Zones. Domain values for bulk density were further separated by locality related to the mineralisation object wireframes (Section 10.3). The assigned bulk density values are detailed below in Table 14-8 and are considered to be consistent with nearby deposits in the region.

Material type was also defined on the basis of weathering state. Material type and the corresponding bulk density were coded into the block model in the 'type' and 'bd' attributes respectively.

Table 14-8: Hicks Zone Material Type and Bulk Density

Material Type	Bulk Density t/m^3 (Dry)	Range
Overburden	1.82	Topographic Surface to Base of Overburden
Oxide Mineralised	1.82	Base of Overburden to BOCO
Oxide Waste	1.71	Base of Overburden to BOCO
Transitional Mineralised	2.29	BOCO to TOFR
Transitional Waste	2.43	BOCO to TOFR
Fresh Mineralised	2.76	Below TOFR
Fresh Waste	2.86	Below TOFR
Dolerite Dyke	2.65	Within Cross-cutting Dolerite Dyke

14.2.9 Resource Classification

The Hicks Zone shows reasonable continuity of mineralisation within well defined geological constraints. Drillholes are located at 25m to 60m spacings on 15m to 100m spaced north-east orientated drill sections.

Trench information was used to guide mineralisation wireframe interpretation but was excluded from the estimation process. This reflected sample quality concerns with the trench information. However, in general the trench information agreed with the structural interpretation from drilling at shallow levels.

The drill spacing is sufficient to allow the grade intersections to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the thickness and grade of the domains.

The authors consider that geological and mineralisation continuity has been demonstrated with sufficient confidence to allow the Hicks Zone to be classified as either Indicated or Inferred Mineral Resource. Indicated Mineral Resources are defined by drilling density and confidence in continuity of mineralisation between drillholes. This generally corresponds to the 50m drilling sections whereas the 100m drilling sections are classified as Inferred Mineral Resources. Mineralisation generally forms a series of coherent gold envelopes along strike and is open down-dip.

The Resource block model has an attribute "class" for all blocks within the Resource wireframes coded as "Inf" or "Ind".

14.2.10 Results

A summary of the estimate reported at a 1.0g/t gold cut-off grade is presented in Table 14-9.

Table 14-9: Hicks Zone June 30th 2014 Mineral Resource Estimate (1.0g/t Gold Cut-off)

Type	Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces
Oxide	277,000	2.2	19,700	31,000	1.6	1,600
Transitional	75,000	2.1	5,000	35,000	1.6	1,800
Fresh	2,591,000	2.1	175,800	3,106,000	1.9	193,000
Total	**2,943,000**	**2.1**	**200,500**	**3,172,000**	**1.9**	**196,400**

Note rounding errors may occur in compiled totals

Figure 14-5 displays the block model for the all the Hicks Deposit ore zones, in longitudinal section coloured by cut gold grade (g/t).

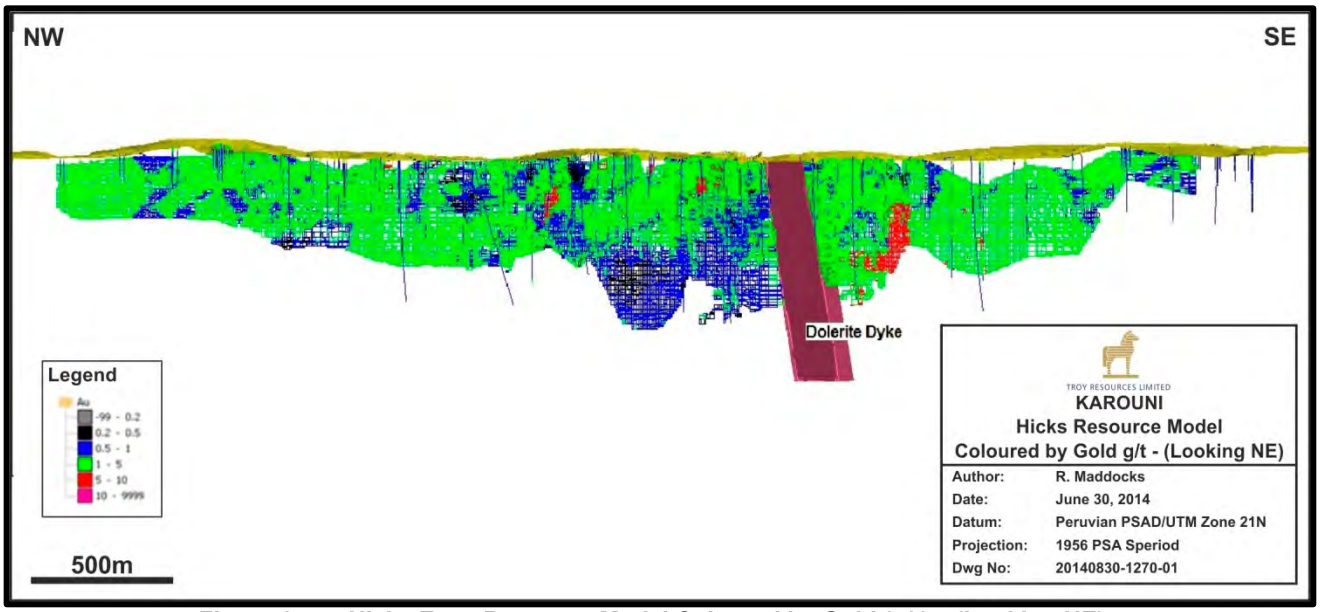

Figure 14-5: Hicks Zone Resource Model Coloured by Gold (g/t) – (Looking NE)

To show the tonnage and grade distribution throughout the entire deposit, a bench breakdown has been prepared for gold by weathering type material. The material type breakdown is shown in Figure 14-6.

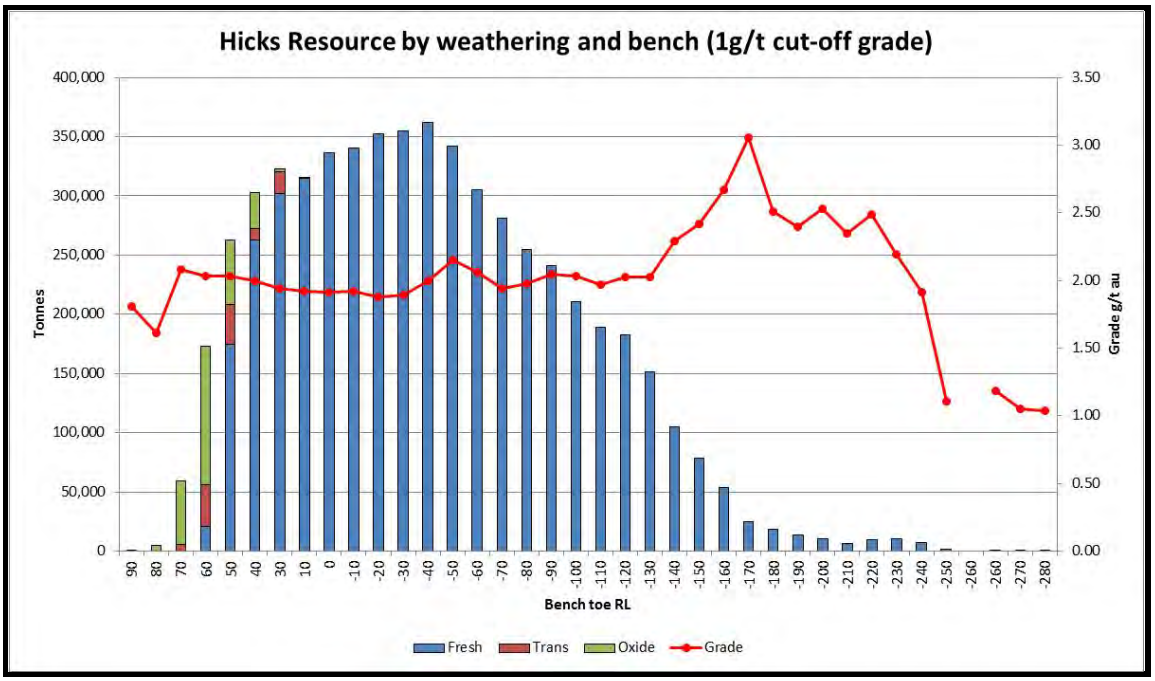

Figure 14-6: Hicks Zone - Global Bench Tonnage by Material Type

Mineral Resource tonnes and grade per 10m bench are graphed below in Figure 14-7. The tonnage decrease at depth represents estimation constraints as drill density decreases rather than a geological limit to the Mineral Resource boundary. Gold grade is relatively consistent for all levels although a moderate grade increase is noted for the upper levels and a relatively sharp increase for lower levels of the Hicks Deposit however this is in the Inferred Mineral Resource category with relatively few tonnes. The grade tonnage curves for the Mineral Resource are compiled below in Figure 14-8.

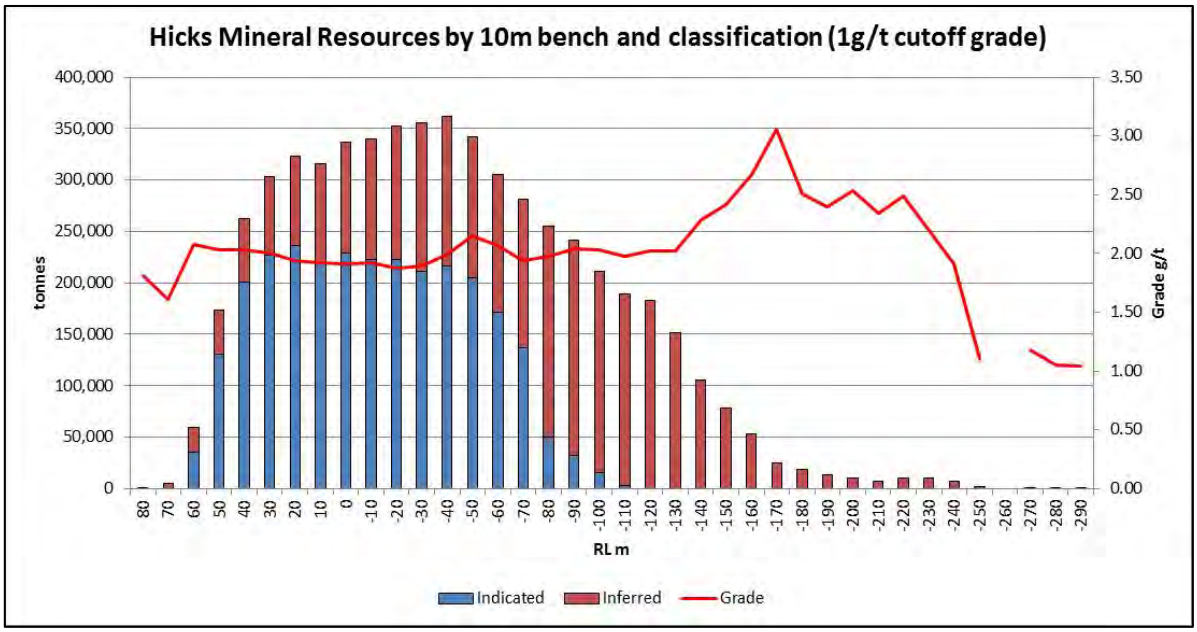

Figure 14-7: Hicks Zone Resource – Tonnes and Grade per 10m Bench

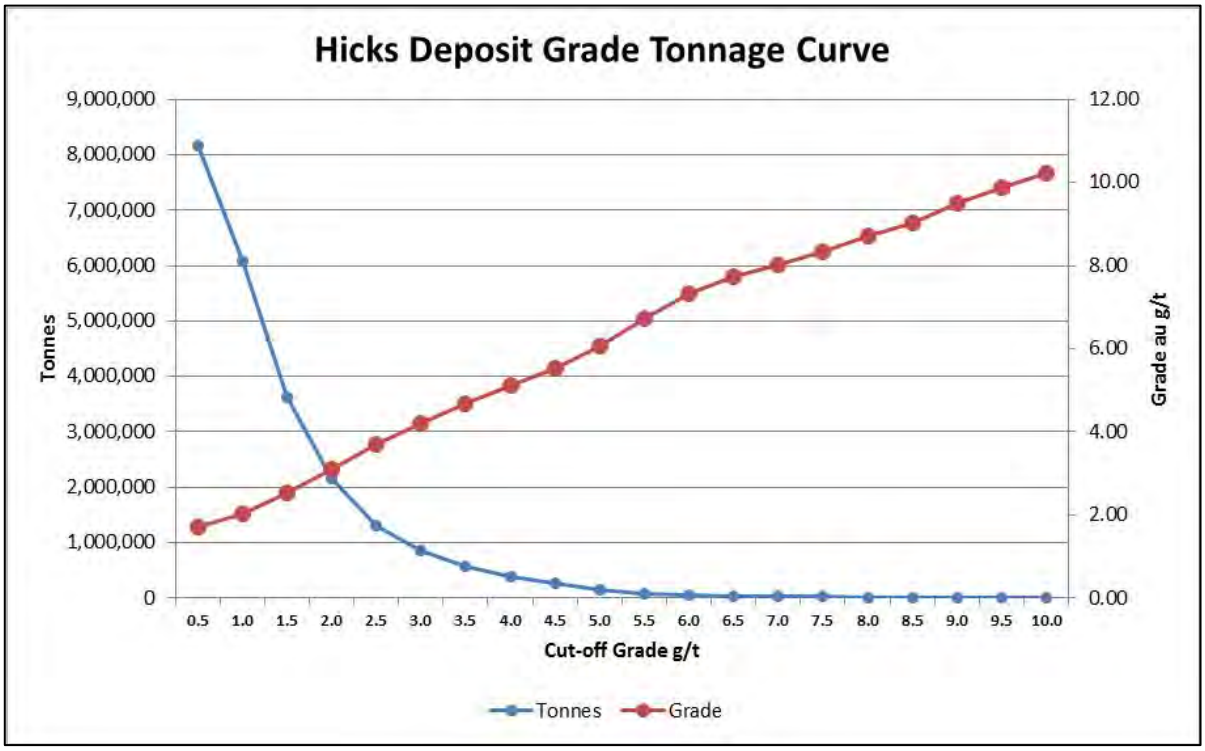

Figure 14-8: Hicks Zone Resource – Grade Tonnage Curve

14.2.11 Model Validation

A three step process was used to validate the Hicks Zone estimate. Firstly a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. Overall the assessment indicated that the trend of the modelled grade appeared consistent with the drillhole grades.

A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. The results of the comparison are presented in Table 14-10.

Table 14-10: Hicks Zone Average Composite Input v Block Model Output (By Ore Zone)

Ore Zone	Wireframe Volume	Model Volume	Block Model Gold Grade	No of Composites	Composite Uncut Mean	Composite Cut Mean
A	1,066,731	1,066,731	1.55	822	1.93	1.70
B	656,171	656,171	1.08	354	1.69	1.62
C	487,335	487,335	1.46	312	2.22	2.18
D	506,052	506,052	2.04	375	1.99	1.98
E	434,461	434,461	1.47	130	1.57	1.57
F	9,323	9,323	1.69	18	2.16	2.16
G	75,224	75,224	1.80	86	1.82	1.82

As a further check that the interpolation of the block model correctly honoured the drilling data, a trend analysis was completed by comparing the interpolated blocks to the sample composite data. The trend analysis was completed for 100m northings and for elevation in 10m bench heights. All objects were combined and used for the model validation.

Results are summarised in Figure 14-9 and Figure 14-10.

The validation plots show good correlation between the composite grades and the block model grades for the comparison by northing and elevation, variations are where limited intercepts have defined individual objects. The trends shown by the raw data are honoured by the block model.

The elevation plot shows the effect of extending the wireframes at depth beyond the last drill hole intercept resulting in no sample composites to support the estimated block grades.

The comparisons show the effect of the interpolation, which results in smoothing of the block grades, compared to the composite grades.

An additional check was conducted with the estimation of grades using an inverse distance squared methodology. Search directions were along the main ore zones axes and search distances were determined from drill spacing. The results from this estimation were within about 2% for contained metal.

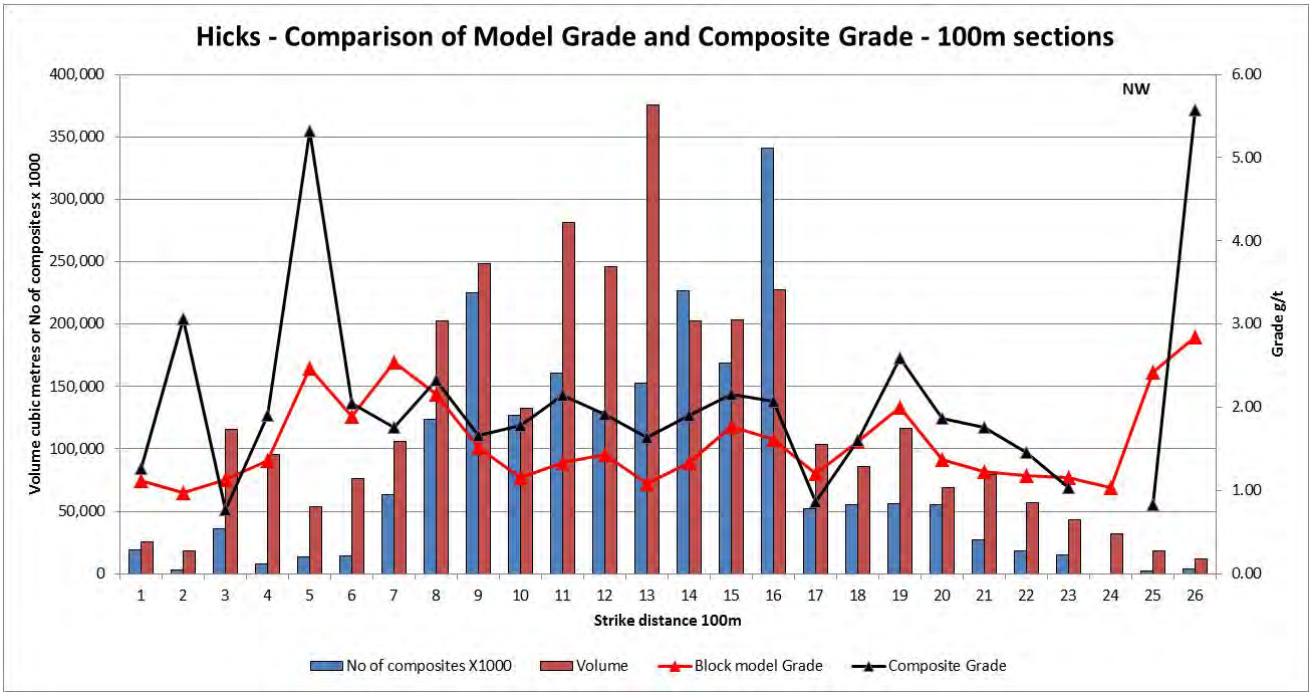

Figure 14-9: Hicks Zone Resource Validation by NE-SW Trending Panels

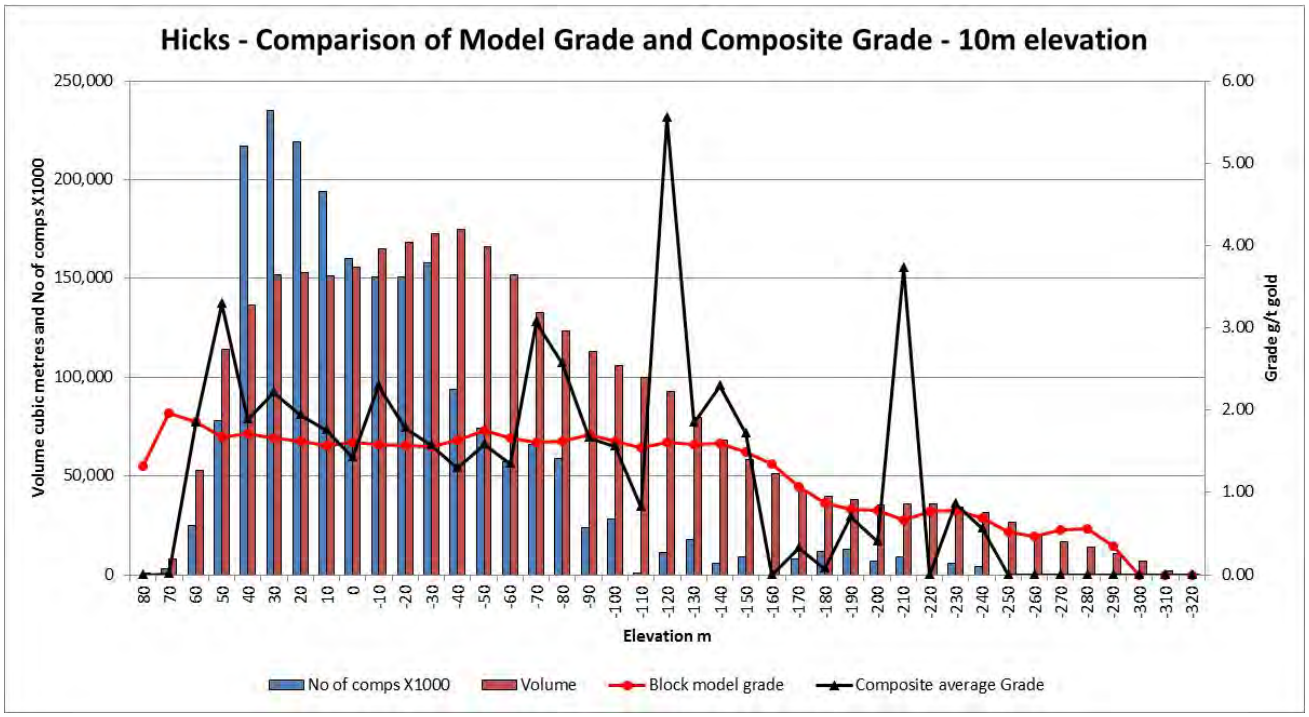

Figure 14-10: Hicks Zone Resource Validation by Elevation

14.3 Smarts Zone

14.3.1 Preparation of Wireframes

14.2.1.1 Resource Mineralisation

The interpreted sectional outlines were manually triangulated to form wireframes as show in Figure 14-11.

Geological information from trenches was used to guide the interpretation of mineralisation boundaries. Information from drilling was used in preference where generally minor contradictions in trench and drill data were noted. This reflects concerns with the sample quality of the trench information.

To form ends to the wireframes, the end section strings were copied to a position midway to the next section or to approximately 25m depending on sample information along strike or down dip. The end section strings were adjusted to match the dip, strike and plunge of the zone. The wireframed objects were validated using Vulcan software and set as solid triangulations.

Modelling was conducted on two main geological domains; the ore zones and the shear zones. A total of 22 ore zones were modelled and 6 shear zones were modelled as shown in Figure 14-12.

The mineralisation wireframes were treated as hard boundaries for all estimation purposes, that is, only assays from within each wireframe were used to estimate blocks within that wireframe.

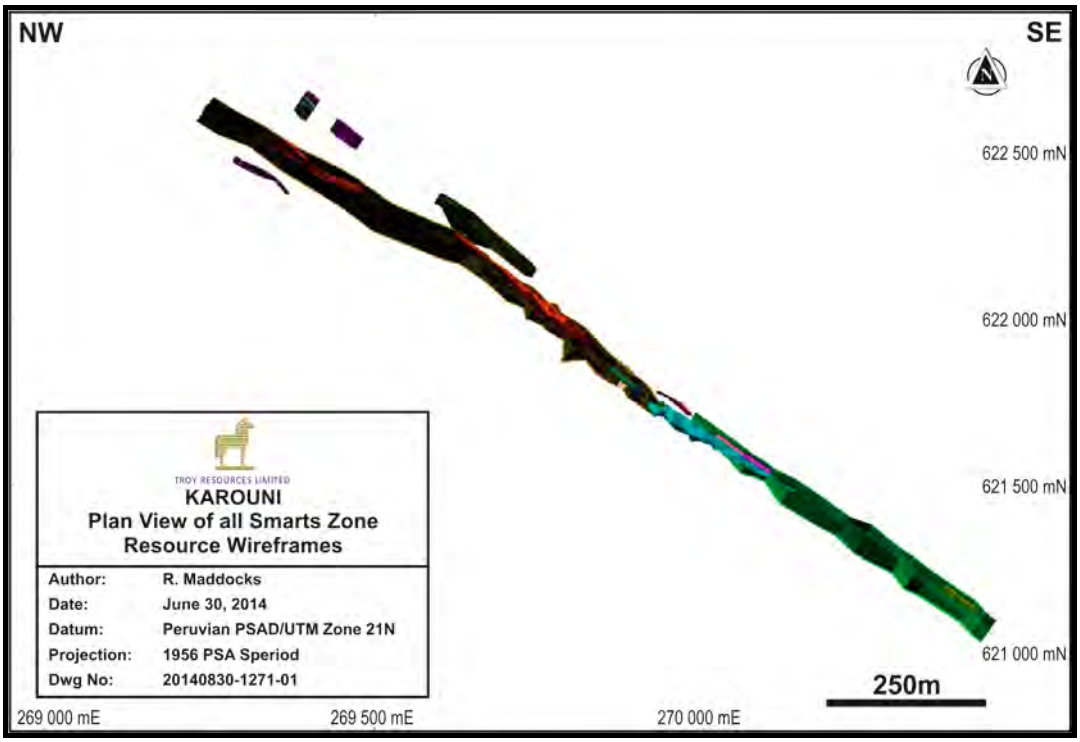

Figure 14-11: Smarts Zone Plan View of All Resource Wireframes

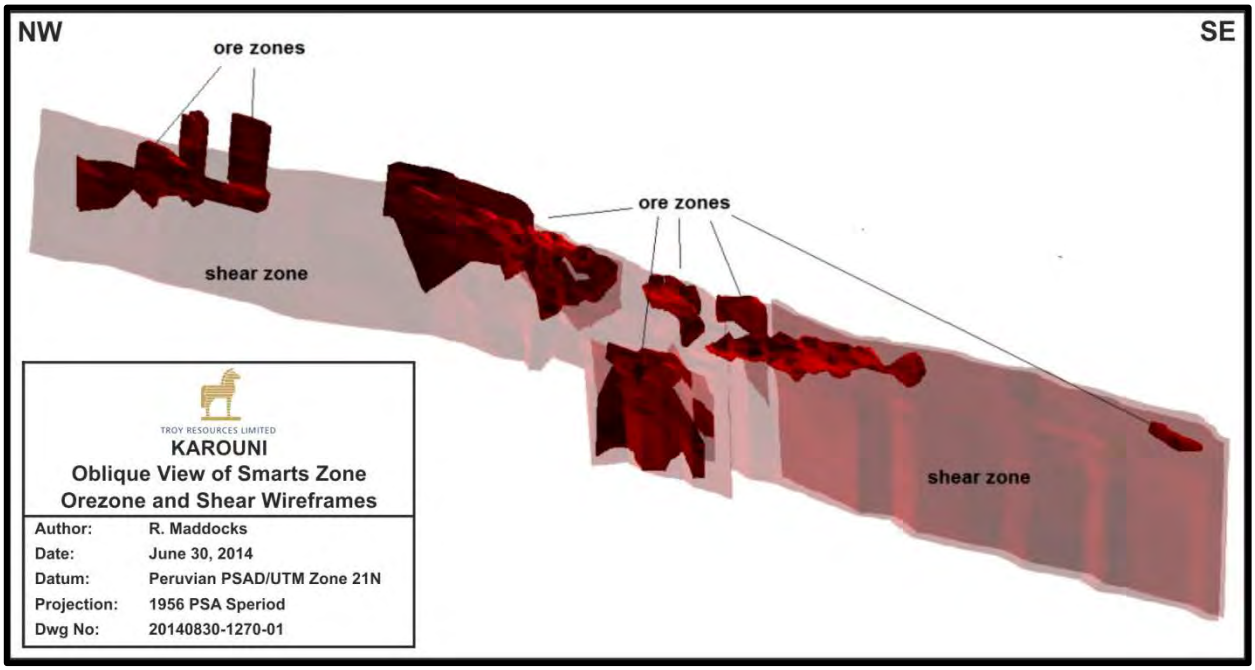

Figure 14-12: Smarts Zone Oblique View of Resource Wireframes

14.2.1.2 Weathering Surfaces

Weathering surfaces were interpreted on cross-section and extended to cover the block model limits. Weathering codes and degree of oxidation were included in the geology database and used to define the base of complete oxidation (*boco_2014.00t*) and top of fresh rock (*tofr_2014.00t*) surfaces.

The database geological codes used to define the weathering surfaces are presented in Table 14-2. The surfaces were mainly defined from the database field "Regolith".

Conversely, both the regolith zone and the transition zone (i.e. base of complete oxidation) are defined by geological codes starting with 'R'. The regolith unit is generally composed of the Rpa (pallid zone), Rmo (mottled zone), Rlt (laterite), Rfe (ferricrete), and Rsp (saprolite). The weathering of these units may vary, but generally spans from moderately weathered (mw) to extremely weathered (ew). It should be noted that the degree of weathering is not directly linked to the degree of oxidization, but to the competency of the rock.

The transition zone refers to the transition between the regolith and fresh rock. The database geological code used is Rsr (saprock). Since this does define the transition between regolith and fresh rock, this unit tends to be weakly weathered (ww).

14.2.1.3 Geological Surfaces

The overburden unit is defined by geological codes starting with 'O', and is generally composed of Osd (overburden sand) and Ocy (overburden clay). The overburden surface (*overburden_2014.00t*) was interpreted on cross section and extended to cover the block model limits.

14.2.1.4 Topographic Surface

The surface elevation is moderately undulating and ranges from 55m to 90m in elevation above sea level. This reflects the incisement of overburden and basement lithologies by numerous streams.

The current topographic surface was coded from a LiDAR survey undertaken in late 2011 (*contour_wo_resource_2m.00t*). The survey was undertaken by Digital World Mapping Inc. and is nominally accurate to +/- 0.3m on flat terrain. The surface was thinned and re-contoured to reduce file size, however trench excavation lines are still generally visible and the surface is considered to be of excellent quality.

Drill collars, trench excavation lines and the LiDAR topographic surface generally show a reasonable correlation. To ensure consistency drill collars were adjusted in the Z direction (elevation) to match the LiDAR surface. Collar adjustments were generally minor or reflected corrections to either historical drill collars or where a hand held GPS had been used.

A typical cross section from the Smarts Zone highlighting the weathering surfaces and the mineralisation wireframes is shown in Figure 14-13. The section location can be referred to in Figure 14-11.

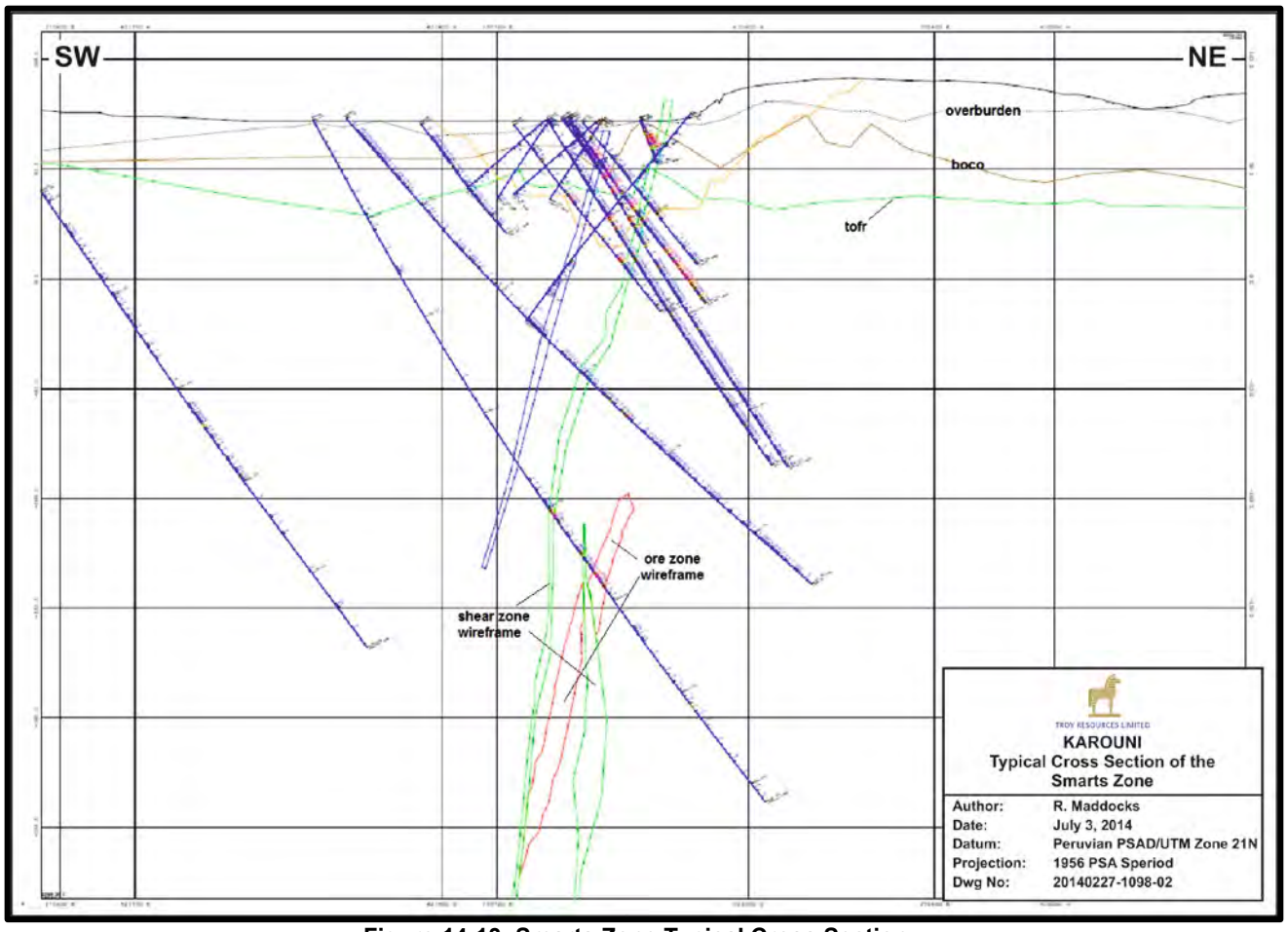

Figure 14-13: Smarts Zone Typical Cross Section

14.3.2 Sample Statistics

14.2.2.1 General

The wireframes of the mineralised zones were used to define the Resource drillhole intersections.

The drill samples used to interpret the mineralisation are dominantly of 1m length.

A 1m interval length was selected for compositing and further geostatistical analysis. This composite length reflects the majority of sample intervals and is at a suitable scale for the width of the Resource wireframes.

A small number of negative assays representing assay detection limit were converted to 0.005g/t grades (half the detection limit).

The composites were checked for spatial correlation with the objects, the location of the rejected composites and zero composite values. A master composite file was created for the deposit with composites within each sold wireframe flagged.

14.2.2.2 Deposit Statistics

The composite sample data for the deposit was grouped by Resource object for analysis. Summary statistics for gold are shown in Table 14-11 for all ore zones and shear zones.

Table 14-11: Smarts Zone Summary Statistics for Mineralisation Wireframes

Ore Zone	Volume	No of Composites	Gold Low (g/t)	Gold High (g/t)	Gold Mean (g/t)	Variance	Standard Deviation	CV
1	423,122	767	0.0020	126.54	4.18	119.67	10.94	2.62
2	17,295	33	0.0010	8.01	1.48	3.60	1.90	1.28
3	16,852	55	0.0001	10.27	1.29	5.42	2.33	1.81
4	218,355	74	0.0050	25.56	4.50	34.94	5.91	1.31
5	42,374	39	0.0300	121.56	8.63	443.86	21.07	2.44
6	276,128	721	0.0001	146.20	7.87	273.57	16.54	2.10
7	29,043	171	0.0050	107.64	4.84	166.46	12.90	2.67
8	28,911	113	0.0001	168.00	9.78	674.50	25.97	2.66
9	6,035	6	0.6500	5.74	2.45	3.96	1.99	0.81
10	18,168	53	0.0100	62.69	4.49	97.84	9.89	2.20
11	15,784	16	0.1600	60.70	10.51	303.88	17.43	1.66
12	17,115	19	0.0150	18.10	2.78	17.35	4.17	1.50
13	124,768	55	0.0010	66.52	6.13	109.44	10.46	1.71
14	23,717	15	0.0010	147.76	14.12	1432.59	37.85	2.68
15	121,302	39	0.0030	10.13	1.12	3.72	1.93	1.72
16	10,024	21	0.0100	20.50	5.45	37.26	6.10	1.12
17	30,859	105	0.0100	11.56	1.27	4.46	2.11	1.66
18	31,558	33	0.0030	12.72	2.65	10.44	3.23	1.22
19	28,383	13	0.1500	14.02	2.24	14.14	3.76	1.68
20	23,389	6	0.5200	8.06	3.17	8.69	2.95	0.93
21	1,632	17	0.1450	29.03	6.45	71.12	8.43	1.31
22	4,292	6	0.8500	12.68	6.25	23.22	4.82	0.77
Shear1	1,539,473	939	0.0001	52.05	1.54	20.27	4.50	2.93
Shear2	125,846	41	0.0050	6.24	0.69	1.86	1.36	1.97
Shear3	48,343	95	0.0001	15.68	1.15	7.54	2.75	2.40
Shear4	976,005	320	0.0001	53.94	0.64	12.23	3.50	5.43
Shear5	1,161,576	435	0.0001	27.22	0.33	3.65	1.91	5.78
Shear6	4,090	3	0.0030	0.74	0.25	0.18	0.42	1.69

14.3.3 High Grade Cuts

To assist in the selection of appropriate high grade cuts, the composite data was analysed and population histograms, log probability plots and the coefficient of variation (CV) statistic was assessed for each Resource object.

Following a review of the population histograms and the CV by domain, it was determined that high grade cuts were required for some objects. High grade cut statistics for each applicable object are listed in Table 14-12.

Table 14-12: Smarts Zone High Grade Cuts by Domain

Object	Samples	Before High Grade (HG) Cut		High Grade Cut	No. Cut	Percentage (%)	After High Grade Cut	
		Mean	CV				Mean	CV
1	767	4.18	2.62	80	6	0.8%	4.03	2.38
2	33	1.48	1.28	10	0	0.0%	1.48	1.28
3	55	1.29	1.81	10	1	1.8%	1.28	1.80
4	74	4.50	1.31	20	5	6.8%	4.31	1.23
5	39	8.63	2.44	50	1	2.6%	6.79	1.80
6	721	7.87	2.10	50	27	3.7%	6.92	1.71
7	171	4.84	2.67	50	4	2.3%	4.30	2.27
8	113	9.78	2.66	50	6	5.3%	6.74	1.95
9	6	2.45	0.81	20	0	0.0%	2.45	0.81
10	53	4.49	2.20	15	4	7.5%	3.34	1.53
11	16	10.51	1.66	25	3	18.8%	7.11	1.32
12	19	2.78	1.50	20	0	0.0%	2.78	1.50
13	55	6.13	1.71	15	4	7.3%	4.72	1.06
14	15	14.12	2.68	15	2	13.3%	4.11	1.24
15	39	1.12	1.72	15	0	0.0%	1.12	1.72
16	21	5.45	1.12	20	1	4.8%	5.43	1.11
17	105	1.27	1.66	20	0	0.0%	1.27	1.66
18	33	2.65	1.22	15	0	0.0%	2.65	1.22
19	13	2.24	1.68	15	0	0.0%	2.24	1.68
20	6	3.17	0.93	15	0	0.0%	3.17	0.93
21	17	6.45	1.31	10	3	17.6%	4.37	0.89
22	6	6.25	0.77	50	0	0.0%	6.25	0.77
Shear1	939	1.54	2.93	15	22	2.3%	1.30	2.33
Shear2	41	0.69	1.97	15	0	0.0%	0.69	1.97
Shear3	95	1.15	2.40	15	1	1.1%	1.14	2.38
Shear4	320	0.64	5.43	15	4	1.3%	0.52	3.76
Shear5	435	0.33	5.78	15	3	0.7%	0.29	4.95
Shear6	3	0.25	1.69	15	0	0.0%	0.25	1.69

14.3.4 Geostatistical Analysis

Geostatistical analysis was carried out on the three representative ore zones with sufficient data points for such an analysis. Ore Zones 1 and 6 and Shear Zone 1 were used as the basis for all other zones.

Downhole and three orthogonal variograms were modelled. Directions were based on the strike and dip direction of the modelled ore zone domains. The ore zones all display long major axes along the NW-SE direction and shorter semi-major and minor axes. A two structured nested spherical model was found to model the experimental variogram well.

The orientation of the plane of mineralisation was aligned with the interpreted wireframes. The variograms were calculated with the first aligned along the main mineralisation continuity while the second is aligned in the plane of mineralisation at 90° to the first orientation. The third is orientated perpendicular to the mineralisation plane, across the width of the mineralization (see Figure 14-14, Figure 14-15 and Figure 14-16).

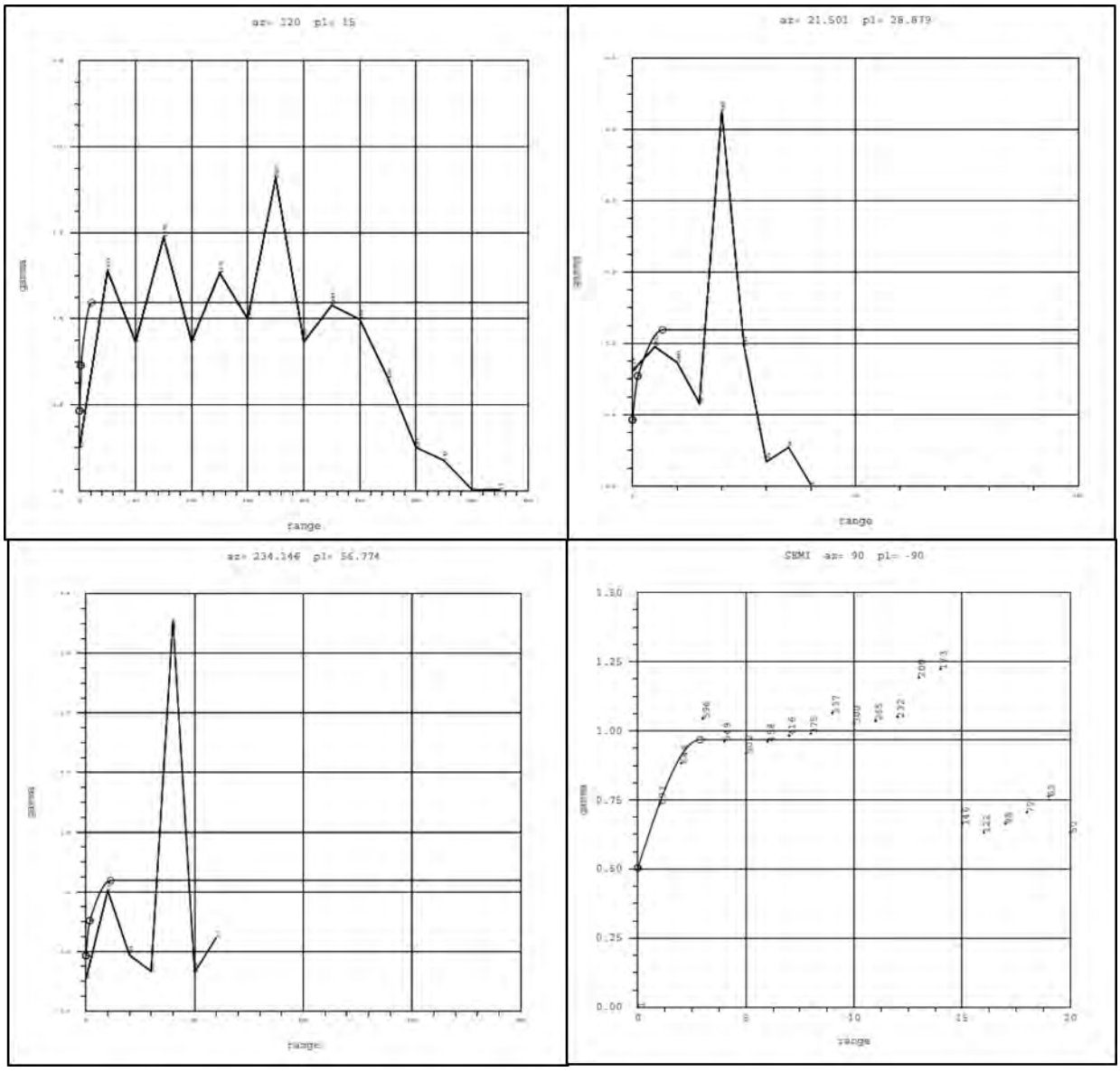

Figure 14-14: Smarts Zone Orthogonal and Downhole Variograms for Ore Zone 1

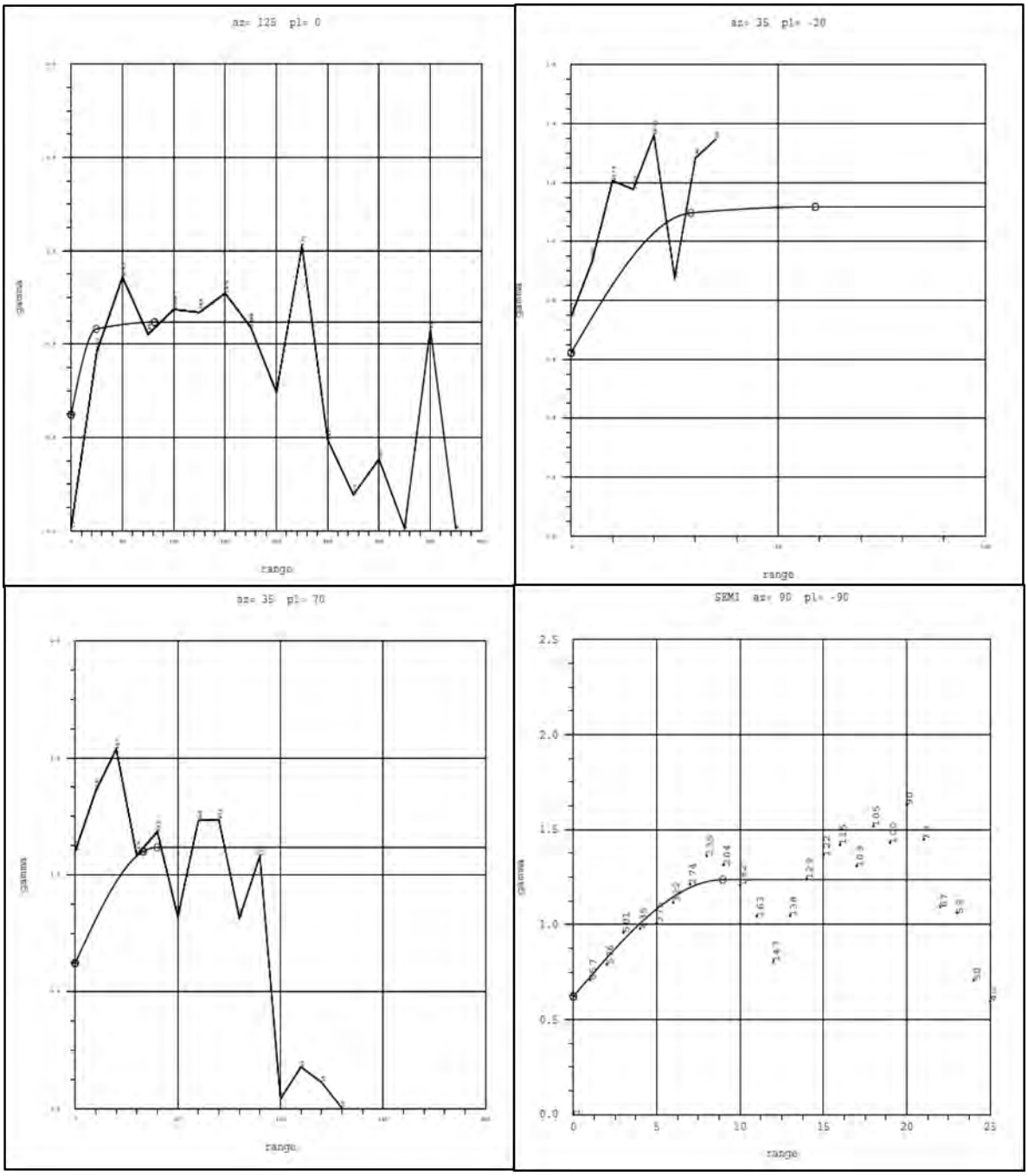

Figure 14-15: Smarts Zone Orthogonal and Downhole Variograms for Ore Zone 6

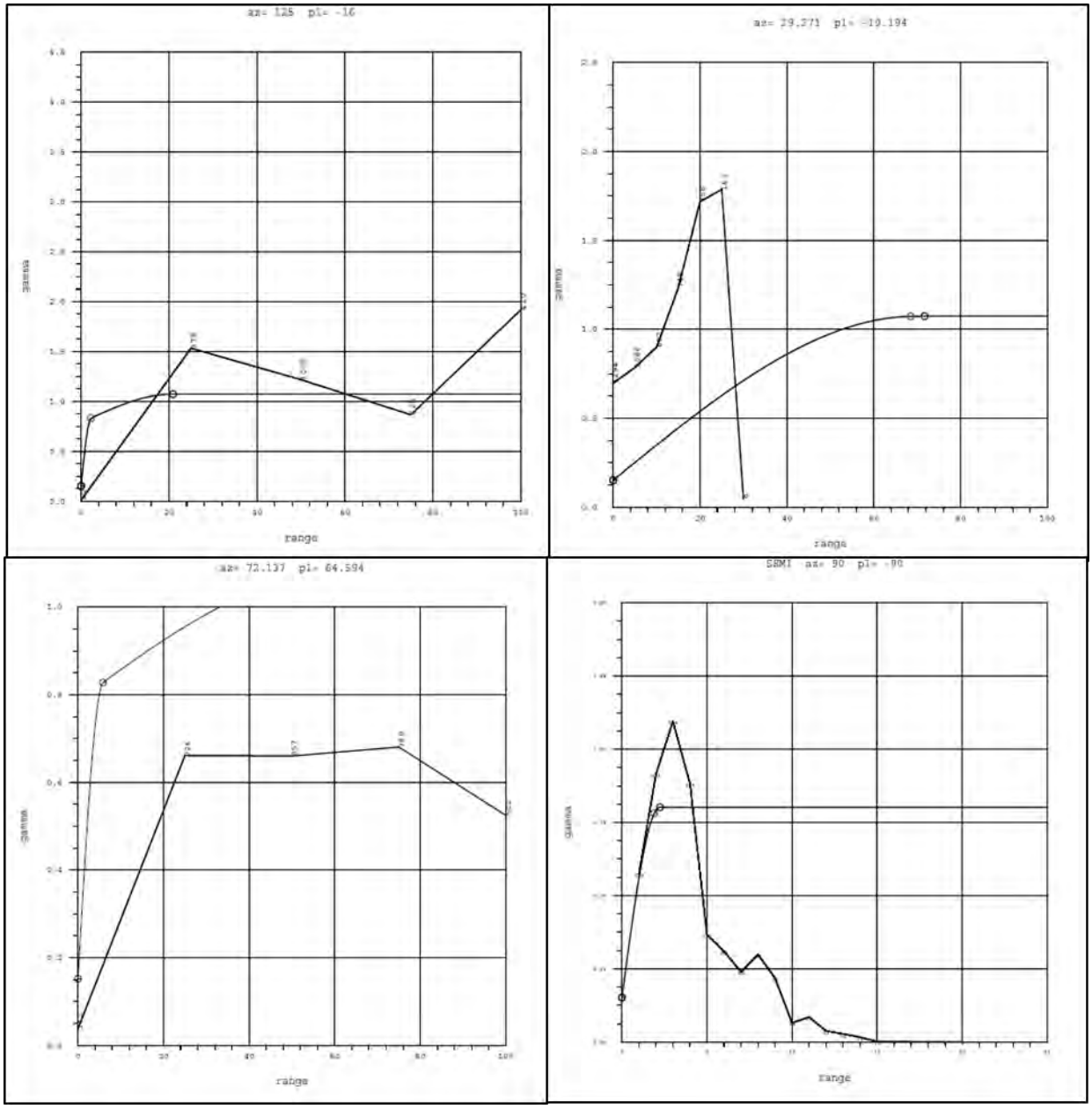

Figure 14-16: Smarts Zone Orthogonal and Downhole Variograms for Shear Zone 1

14.3.5 Kriging Parameters

The gold grades were interpolated into a Vulcan block model using the Ordinary Kriging algorithm using the nugget, sill values and ranges determined from the variogram models. The ranges obtained from the variogram models were used as a guide in determining the search ellipse parameters used in the Mineral Resource estimate. The kriging parameters resulting from analysis of the two main ore zones (1 and 6) and shear zone 1 were applied to the relevant adjacent minor lodes. All parameters are summarised in Table 14-13.

Table 14-13: Smarts Zone Resource Kriging Parameters

Ore Zone	Model Type	Structure	Nugget Co	Ci	Azimuth	Plunge	Dip	Major	Semi-Major	Minor
1	Nested Spherical	1	0.464	0.194	116.00	-46.00	10.00	15.20	1.11	1.01

		2	0.464	0.437	107.00	-58.00	-73.00	93.10	19.50	5.28
2	Nested Spherical	1	0.464	0.194	116.00	-46.00	10.00	15.20	1.11	1.01
		2	0.464	0.437	107.00	-58.00	-73.00	93.10	19.50	5.28
3	Nested Spherical	1	0.464	0.194	116.00	-46.00	10.00	15.20	1.11	1.01
		2	0.464	0.437	107.00	-58.00	-73.00	93.10	19.50	5.28
4	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
5	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
6	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
7	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
8	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
9	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
10	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
11	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
12	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
13	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
14	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
15	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
16	Nested Spherical	1	0.464	0.194	116.00	-46.00	10.00	15.20	1.11	1.01
		2	0.464	0.437	107.00	-58.00	-73.00	93.10	19.50	5.28
17	Nested Spherical	1	0.622	0.65	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
18	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
19	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
20	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
21	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.431	91.00	-69.00	-18.00	139.00	25.10	23.20
22	Nested Spherical	1	0.622	0.065	343.00	-20.00	-44.00	106.00	101.00	29.30
		2	0.622	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
Shear Zone	**Model Type**		**Nugget Co**	**Ci**	**Azimuth**	**Plunge**	**Dip**	**Major**	**Semi-Major**	**Minor**

1	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
2	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
3	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
4	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
5	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07
6	Nested Spherical	1	0.152	0.288	19.00	-52.00	45.00	704.00	118.00	19.30
		2	0.152	0.634	43.00	-73.00	-76.00	431.00	173.00	2.07

14.3.6 Block Model

A Vulcan block model was created to encompass the full extent of the Smarts Zone. A list of block model parameters is displayed in Table 14-14.

Table 14-14: Smarts Zone Block Model Summary

Model Name	**Smarts_June2014.bmf**		
	Y	**X**	**Z**
Minimum Coordinates	620,660	272,000	-400
Extend	3,200	552	540
Block Size (Sub-Blocks)	20 (5)	4 (1)	10 (2.5)
Rotation	305° (horizontal plane)		
Attributes:			
Au	Estimated gold grade from composites with high grade cut (g/t).		
Class	Resource Classification - measured (mes), indicated (ind), inferred (inf).		
Mined	Above surface = y, below surface = n.		
Rock	Rock type, waste, ore, shear		
Numsamp	Number of smaples to interpolate Au.		
Numhole	Number of drillholes to interpolate Au.		
Kvar	Kriging variance from interpolation of Au.		
Pass	Interpolation pass number from interpolation of Au.		
Density	Bulk Density.		
Weath	Weathering fr-fresh, tr-transitional, ox-oxide, ob-overburdenair-air.		

The block model used a primary block size of 20m along strike (NW-SE) by 4m across strike by 10m vertical with sub-cells of 5m by 1m by 2.5m. The block model was rotated to 305 degrees on the horizontal plane to match the strike direction of mineralisation. Block model co-ordinates are on the national grid PSAD56-UTM Zone 21N to match the trench and drillhole database.

The parent Y block size was selected on the basis of at least half of the average drill section spacing in better drilled areas of the Smarts Deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and downdip direction. In addition, the selection of a 10m model cell height reflected the spacing of exploration drilling downdip.

14.3.7 Grade Interpolation Parameters

For all zones in the Smarts Zone, the wireframe objects were used as hard boundaries in the interpolation. That is, only grades inside each wireframe object were used to interpolate the blocks inside the coded object. The Ordinary Kriging algorithm was selected for grade interpolation. This was to allow the measured spatial variation of gold mineralisation to be incorporated into the model.

An orientated 'ellipsoid' search was used to select data for interpolation. The ellipse was oriented to the average strike, dip and plunge of the mineralised zones, and varied accordingly within each object. The maximum search radius was set at a distance similar to the variogram range for the major direction or as required to ensure all blocks were estimated in the final kriging pass. The major to semi-major and the major to minor search ratios were based on the variogram ranges.

A three-pass estimation search was used to complete estimation for gold within the domain objects. The second pass was about twice the distance as the first pass and the third pass was double again.

Estimation parameters used in the estimate are listed below in Table 14-15.

Table 14-15: Smarts Zone Grade Interpolation Parameters

	Pass1			Pass 2			Pass 3			Search Direction		
Ore Zone	**Major**	**Semi Major**	**Minor**	**Major**	**Semi Major**	**Minor**	**Major**	**Semi Major**	**Minor**	**Bearing**	**Plunge**	**Dip**
1	25	10	20	50	20	40	100	30	60	125	15	-30
2	25	10	20	50	20	40	100	30	60	125	15	-30
3	25	10	20	50	20	40	100	30	60	125	15	-30
4	25	20	30	50	40	60	100	60	90	125	0	12
5	25	20	30	50	40	60	100	60	90	125	0	12
6	25	20	30	50	40	60	100	60	90	125	0	15
7	25	20	30	50	40	60	100	60	90	125	0	-10
8	25	20	30	50	40	60	100	60	90	125	0	13
9	25	20	30	50	40	60	100	60	90	125	0	10
10	25	20	30	50	40	60	100	60	90	125	0	21
11	25	20	30	50	40	60	100	60	90	125	0	21
12	25	20	30	50	40	60	100	60	90	125	0	13
13	25	20	30	50	40	60	100	60	90	125	0	20
14	25	20	30	50	40	60	100	60	90	125	0	16
15	25	20	30	50	40	60	100	60	90	125	0	19
16	25	10	20	50	20	40	100	30	60	125	15	35
17	25	20	30	50	40	60	100	60	90	125	0	10
18	25	20	30	50	40	60	100	60	90	125	0	19
19	25	20	30	50	40	60	100	60	90	125	0	19
20	25	20	30	50	40	60	100	60	90	125	0	19
21	25	20	30	50	40	60	100	60	90	125	0	-85
22	25	20	30	50	40	60	100	60	90	125	0	12
Shear Zone	**Major**	**Semi Major**	**Minor**	**Major**	**Semi Major**	**Minor**	**Major**	**Semi Major**	**Minor**	**Bearing**	**Plunge**	**Dip**
1	20	20	30	40	40	40	100	60	150	125	-16	15
2	20	20	30	40	40	40	100	60	150	125	-16	0
3	20	20	30	40	40	40	100	60	150	125	-16	17
4	20	20	30	40	40	40	100	60	150	125	-16	15
5	20	20	30	40	40	40	100	60	150	125	-16	15

6	20	20	30	40	40	40	100	60	150	125	-16	20

14.3.8 Bulk Density and Material Type

Bulk density values were assigned to each weathering material type based on values compiled from historical and recent bulk density test work for the Smarts Zone. Domain values for bulk density were further separated by inside/outside the mineralisation object wireframes. The assigned bulk density values are detailed below in Table 14-16. These values are considered to be consistent with nearby deposits in the region.

Table 14-16: Smarts Zone Material Type and Bulk Density

Material Type	Bulk Density t/m^3 (Dry)	Range
Overburden	1.82	Topographic Surface to Base of Overburden
Oxide Mineralised	1.82	Base of Overburden to BOCO
Oxide Waste	1.82	Base of Overburden to BOCO
Transitional Mineralised	2.29	BOCO to TOFR
Transitional Waste	2.29	BOCO to TOFR
Fresh Mineralised	2.73	Below TOFR
Fresh Waste	2.88	Below TOFR

Bulk density information is relatively sparse in the oxide and transition zones, reflecting the lower amount of available drill core for density test work. These two weathering zones comprise a relatively small proportion of the mineralisation total, however these zones will be the first areas exposed in mining.

Material type was also defined on the basis of weathering state. Material type and the corresponding bulk density were coded into the block model in the 'type' and 'bd' attributes respectively.

14.3.9 Resource Classification

The Smarts Zone shows good to reasonable continuity of mineralisation within well-defined geological constraints. The drill hole spacing throughout the project is approximately 50m along strike with recent infill drilling targeting 25m by 25m spaced drilling within the proposed open pits. Drill section spacing increases to greater than 100m on more sparsely drilled areas generally on the margins of the zone. Drill spacing downdip is of similar dimensions.

The drill spacing is sufficient to allow the grade intersections to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the thickness and grade of the domains.

The Mineral Resource was classified on the basis of sample spacing and continuity of the interpreted zones. The Measured Mineral Resource was defined within areas of 25m by 25m spaced drilling due to the good continuity and predictability of the lode positions. Successive infill drilling campaigns in the open pit area from 100m spacing to 50m and finally to 25m has confirmed the geological and grade model for this part of the Resource, accordingly the 25m spaced areas have generally been classified as Measured Resource. The Indicated Mineral Resources was generally defined by 50m x 50m spaced drilling. The Inferred Mineral Resource included areas of the Mineral Resource where sampling was greater than 50m by 50m and small isolated pods of mineralisation outside the main mineralised zones. The Mineral Resource classification for the Ore Zone wireframes is shown in Figure 14-17 and for the Shear Zone wireframes in Figure 14-18.

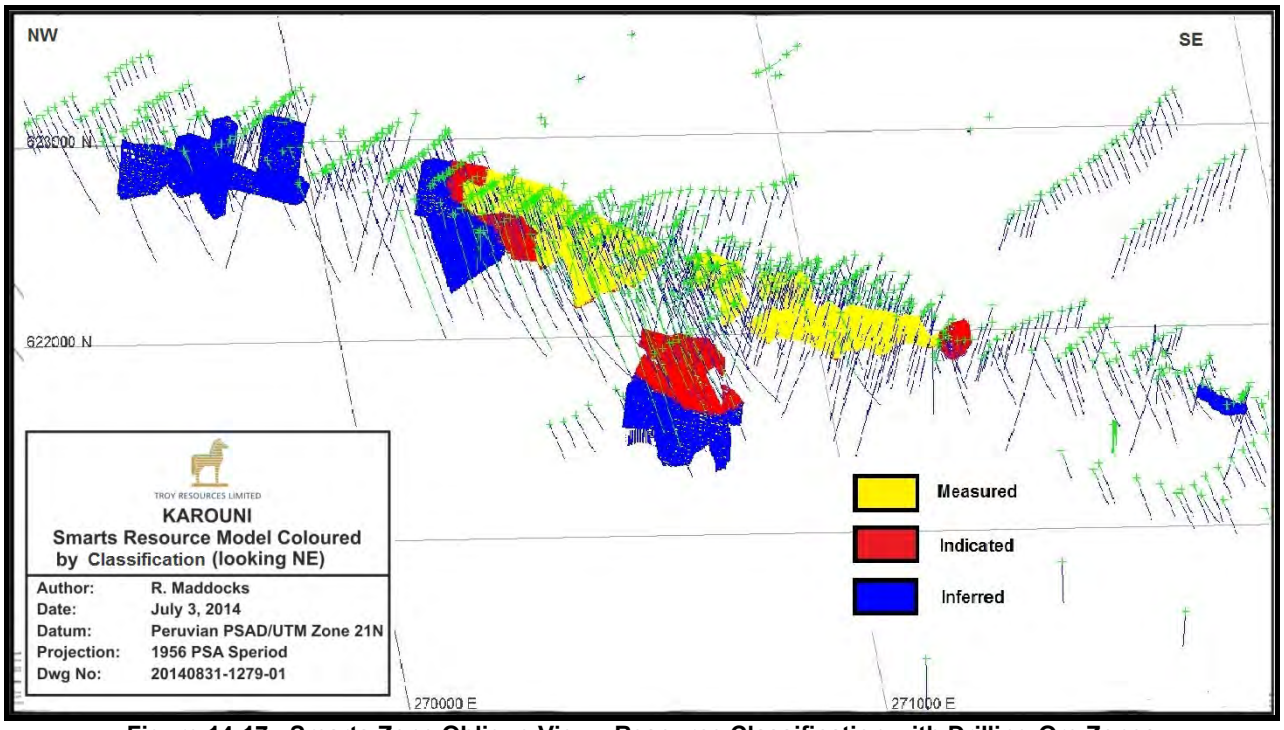

Figure 14-17: Smarts Zone Oblique View - Resource Classification with Drilling-Ore Zones

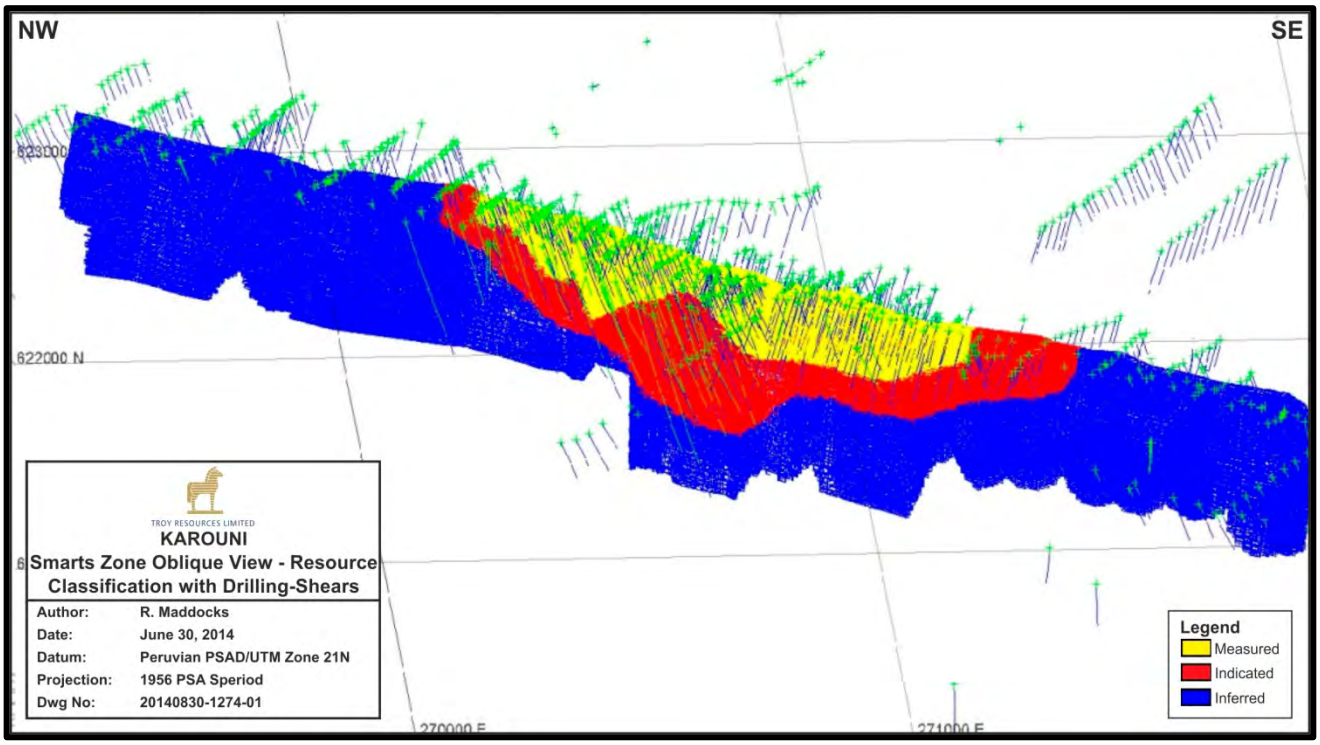

Figure 14-18: Smarts Zone Oblique View - Resource Classification with Drilling-Shear Zones

The Resource block model has an attribute “class” for all blocks within the Resource wireframes coded as “Mea”, “Ind” or “Inf”.

14.3.10 Results

A summary of the estimate reported at a 1.0g/t gold cut-off grade is shown in Table 14-17.

Table 14-17: Smarts Zone June 2014 Mineral Resource Estimate (1.0g/t gold Cut-off)									
Type	Measured			Indicated			Inferred		
	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces	Tonnes (t)	Gold Grade (g/t)	Gold Contained Ounces
Oxide	186,000	5.7	34,000	15,000	2.8	1,300	15,000	2.5	1,200
Transitional	209,000	5.8	39,200	16,000	2.5	1,300	21,000	3.2	2,100
Fresh	1,810,000	4.8	278,500	940,000	3.7	111,000	2,232,000	2.5	181,700
Total	**2,205,000**	**5.0**	**351,700**	**971,000**	**3.6**	**113,600**	**2,268,000**	**2.5**	**185,000**

Note: Rounding errors may occur in compiled totals.

Figure 14-19 shows the block model for the Smarts Ore Zone wireframes in longitudinal section coloured by cut gold grade (g/t) and displaying the topography. Figure 14-20 shows the same view but with the gold grades in the Shear Zones.

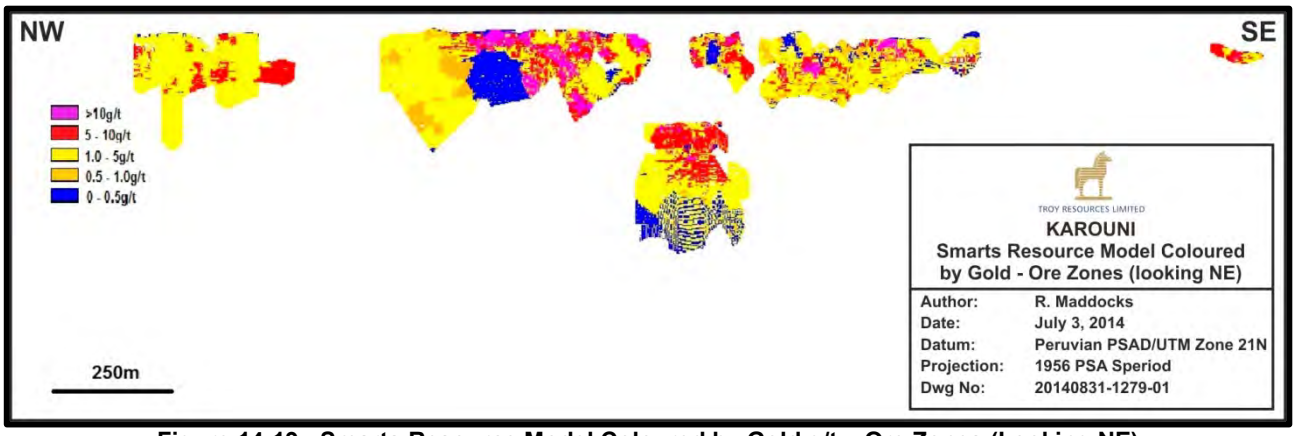

Figure 14-19: Smarts Resource Model Coloured by Gold g/t – Ore Zones (Looking NE)

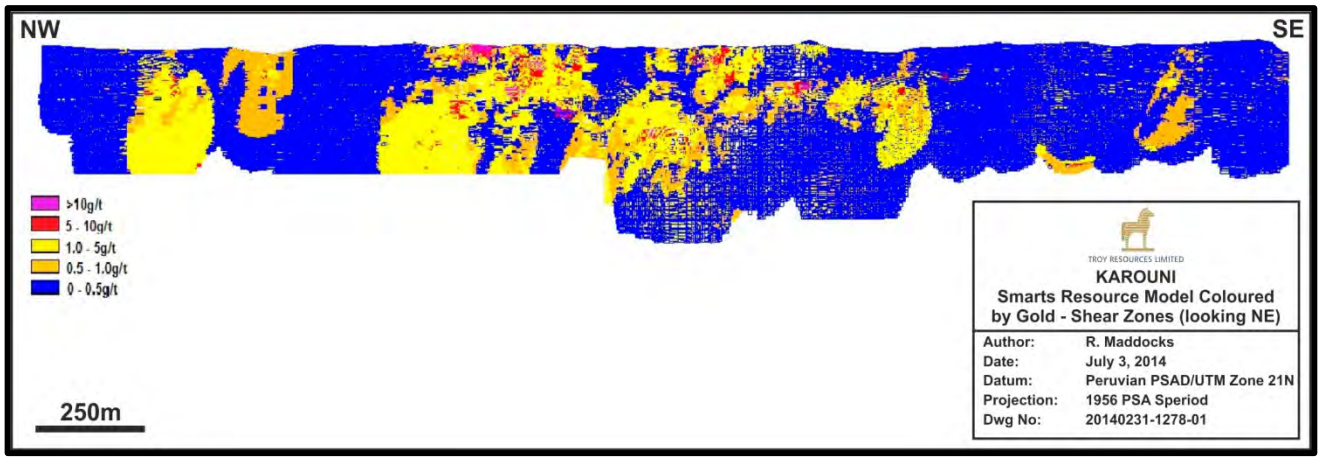

Figure 14-20: Smarts Resource Model Coloured by Gold g/t – Shear Zones (Looking NE)

To show the tonnage and grade distribution throughout the entire Smarts Zone, a bench breakdown has been prepared for gold by weathering type material. The material type breakdown is shown in Figure 14-21.

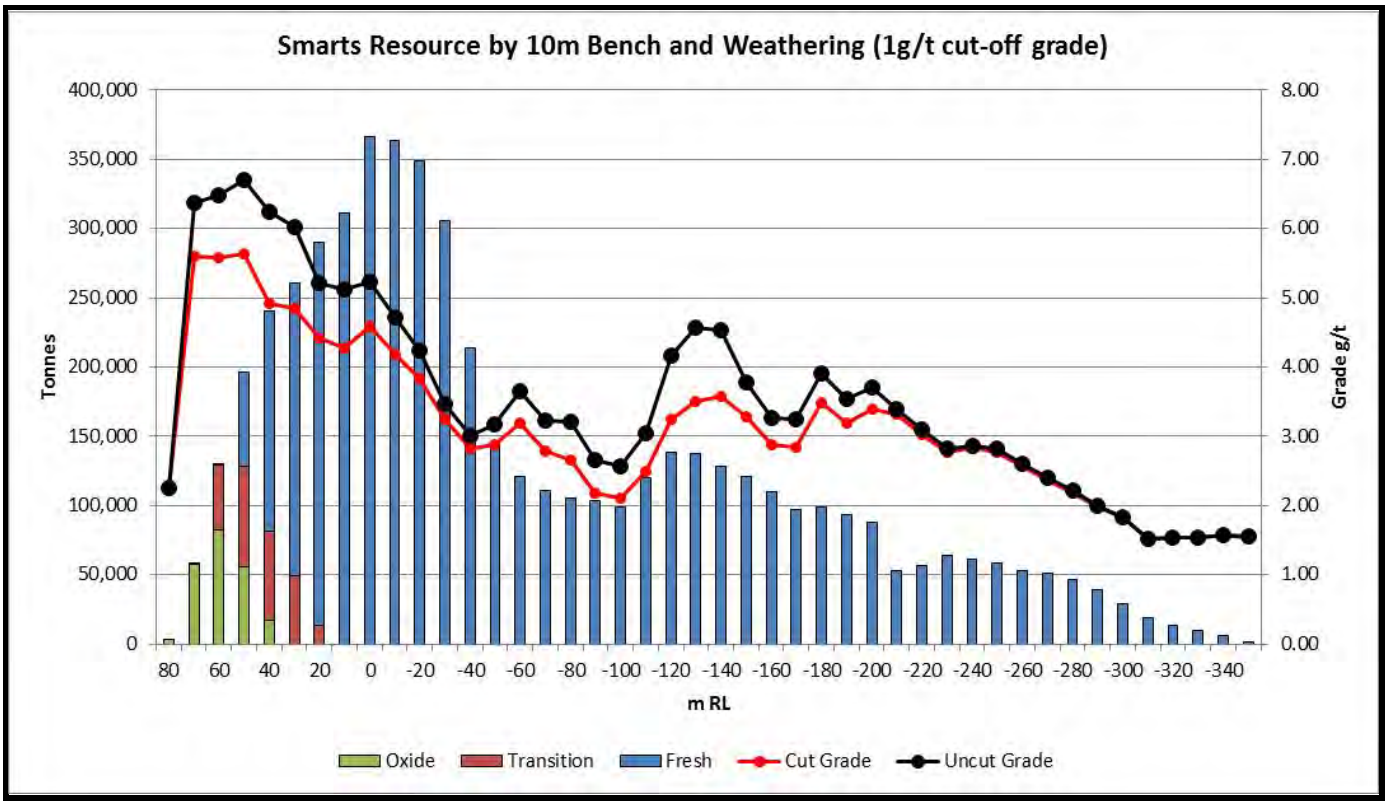

Figure 14-21: Smarts Zone Resource – Global Bench Tonnage by Weathering

Mineral Resource tonnes and grade per 10m bench are graphed below in Figure 14-22. The grade lines reflect the high grade shoots contained within the Smarts Zone with the shallow "high grade" shoots contained within the pit clearly identifiable as is the lower "high grade" shoot between levels -100mRL and -160mRL.

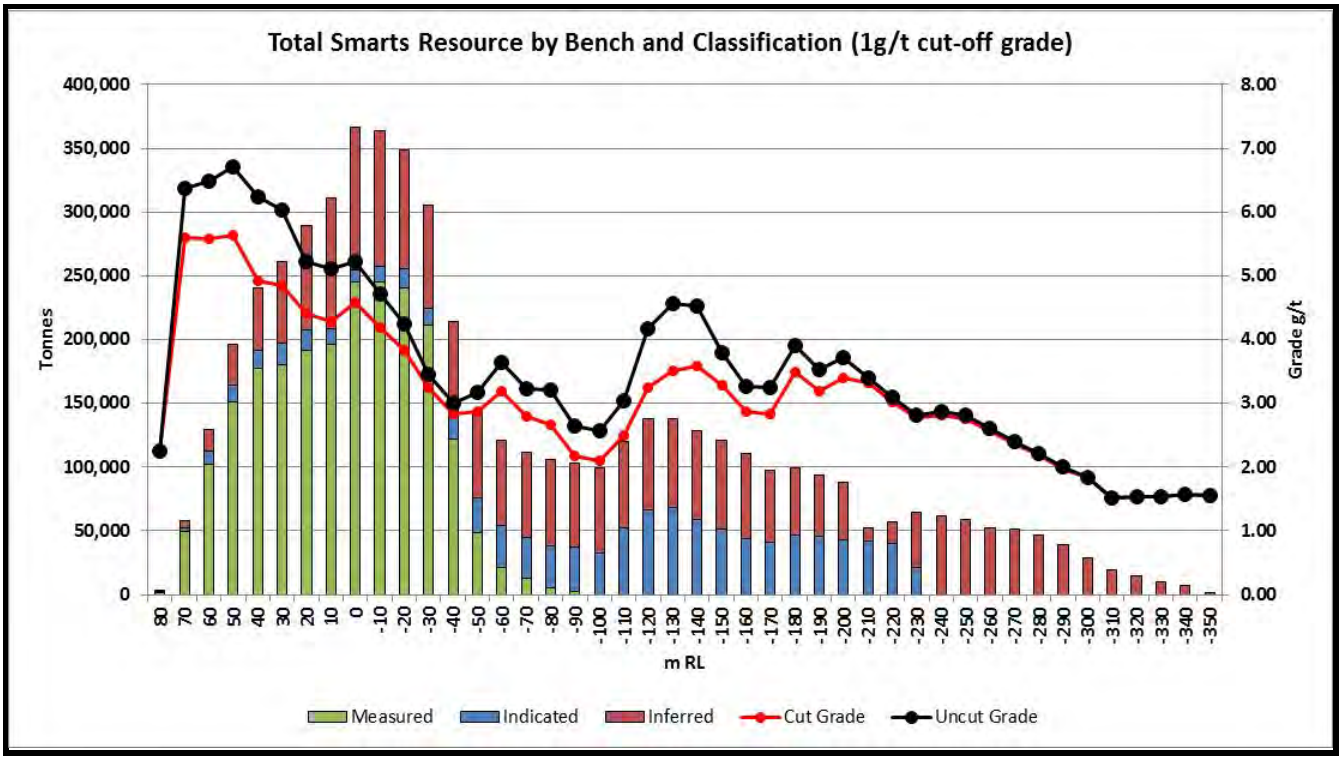

Figure 14-22: Smarts Zone Resource – Tonnes and Grade per 10m Bench

The grade tonnage curves for the Mineral Resource are compiled below in Figure 14-23.

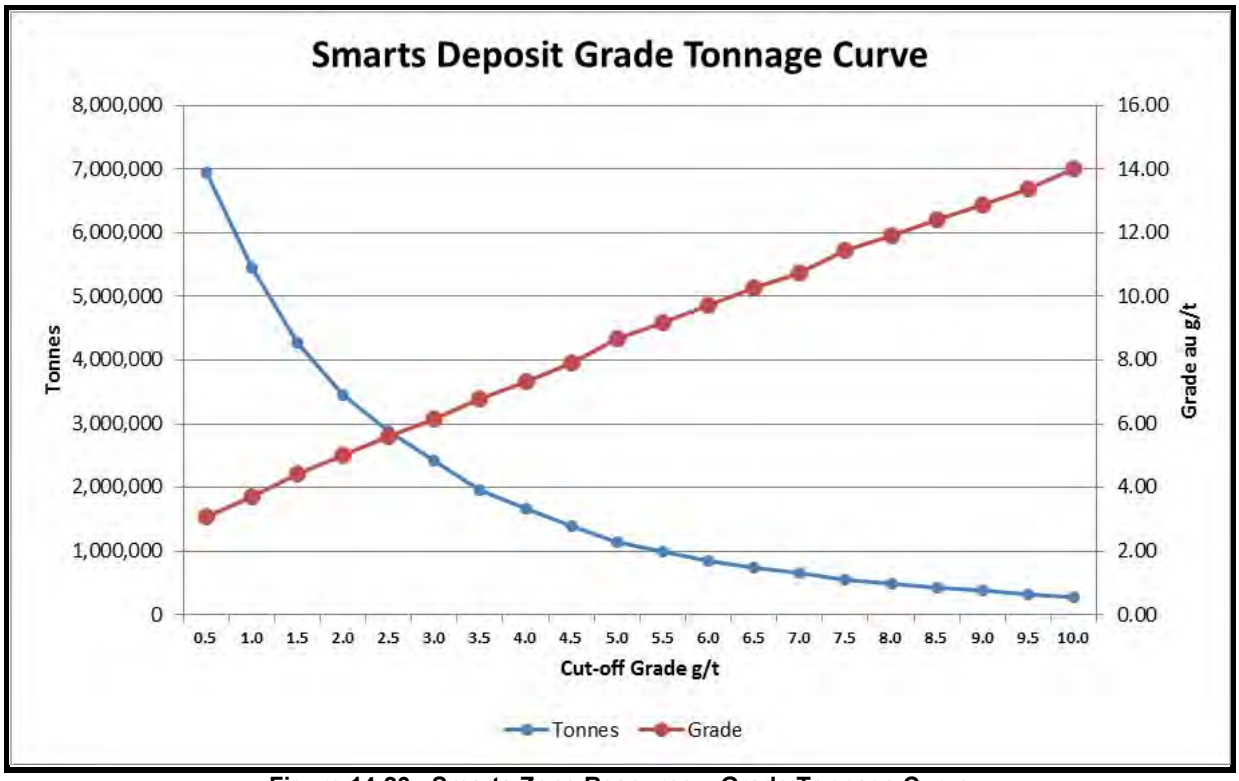

Figure 14-23: Smarts Zone Resource – Grade Tonnage Curve

14.3.11 Model Validation

A three step process was used to validate the Mineral Resource estimate. Firstly a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. Overall the assessment indicated that the trend of the modelled grade appeared consistent with the drillhole grades. A representative section is shown in Figure 14-24 below.

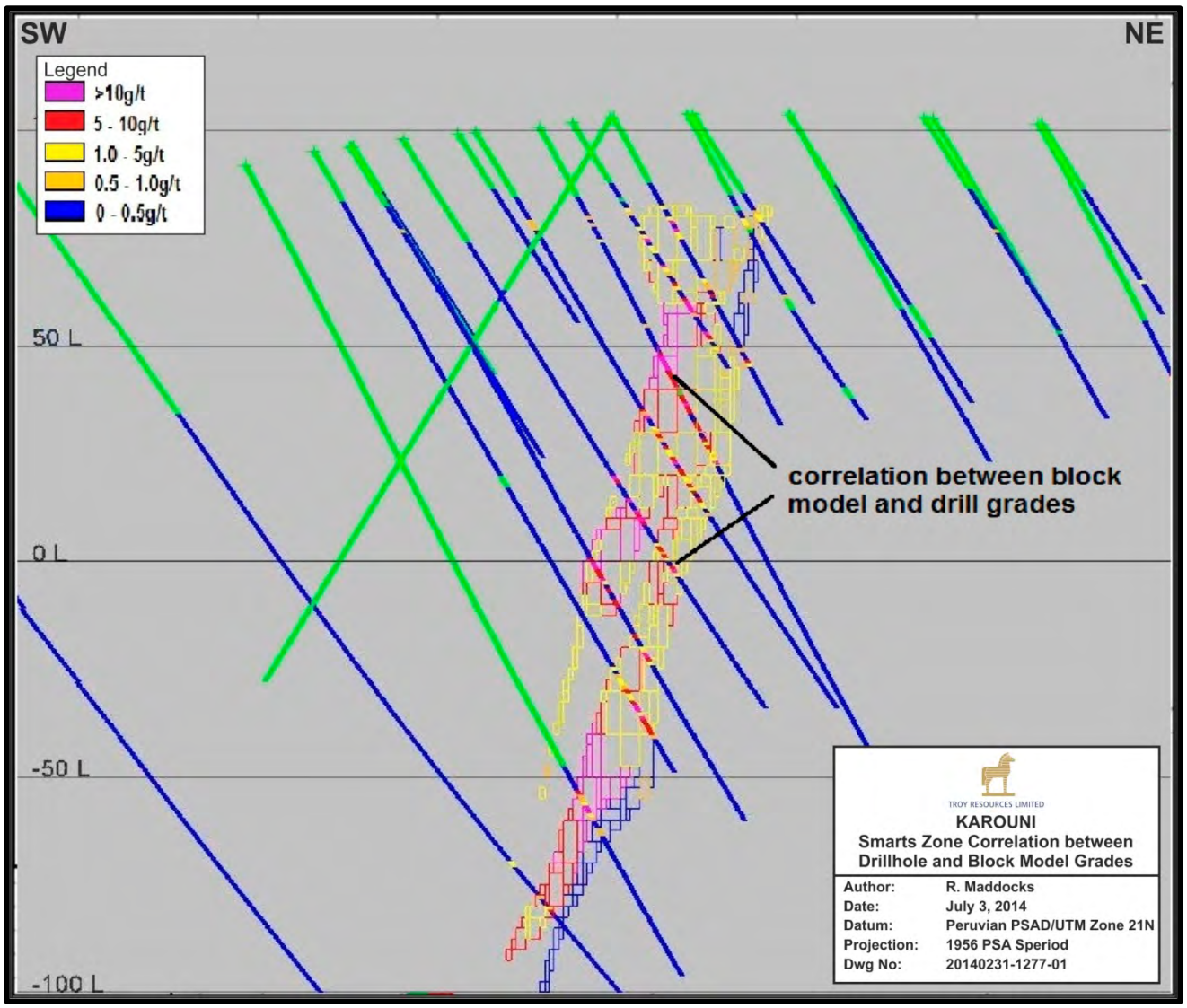

Figure 14-24: Smarts Zone Example of Correlation between Drillhole Grades and Block Model Grades

To check that the interpolation of the block model correctly honoured the drilling data, further validation was carried out by comparing the interpolated blocks to the sample composite data. Resource validation plots by easting panels and elevation are presented in Figure 14-25 and Figure 14-26 respectively.

The validation plots show acceptable correlation between the composite grades and the uncut and cut block model grades. The trends shown by the raw data are generally honoured by the block model and generally the model grades are lower than the composite grades.

The comparisons also show the effect of the interpolation, which results in some smoothing of the block grades compared to the composite grades. This would be expected where drill sample data density decreases.

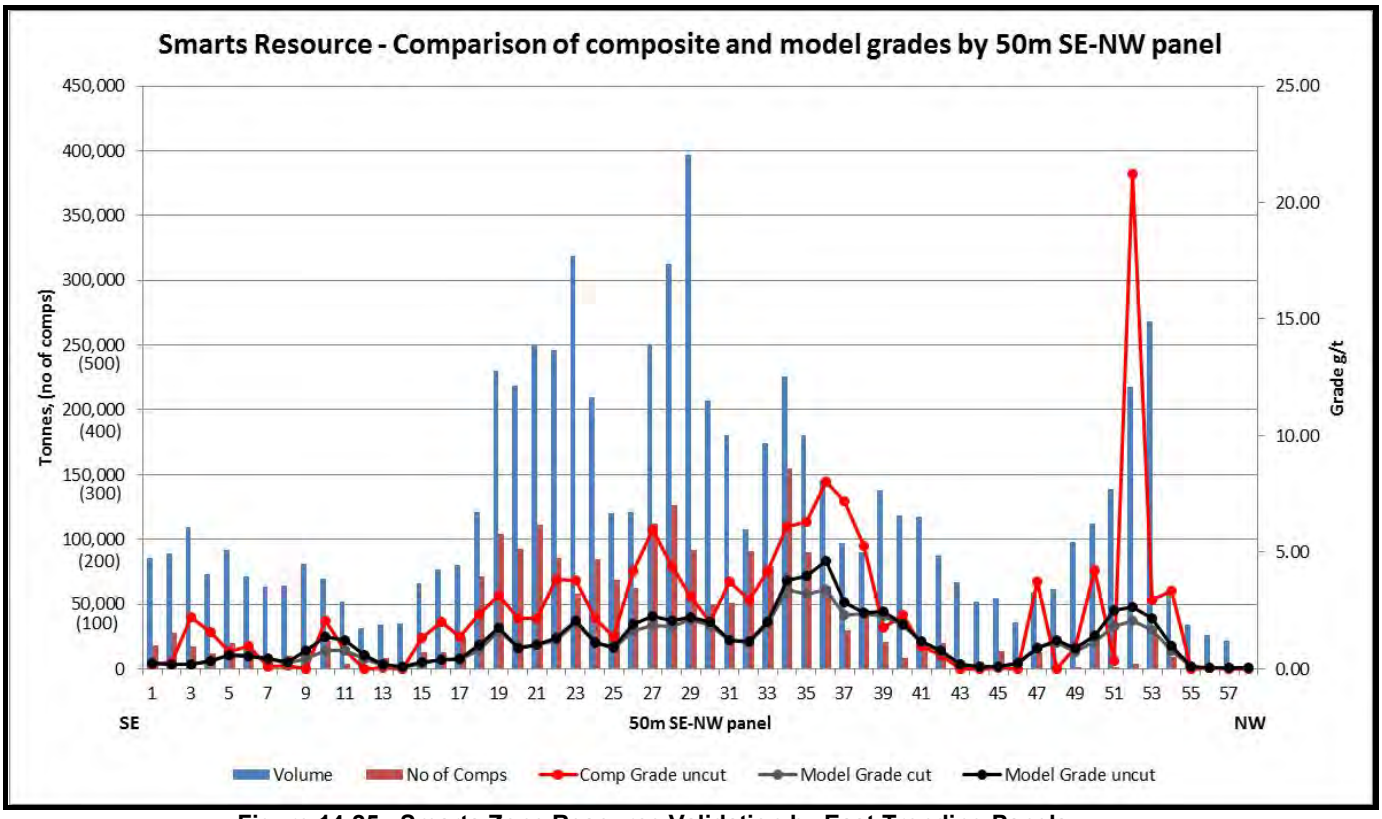

Figure 14-25: Smarts Zone Resource Validation by East Trending Panels

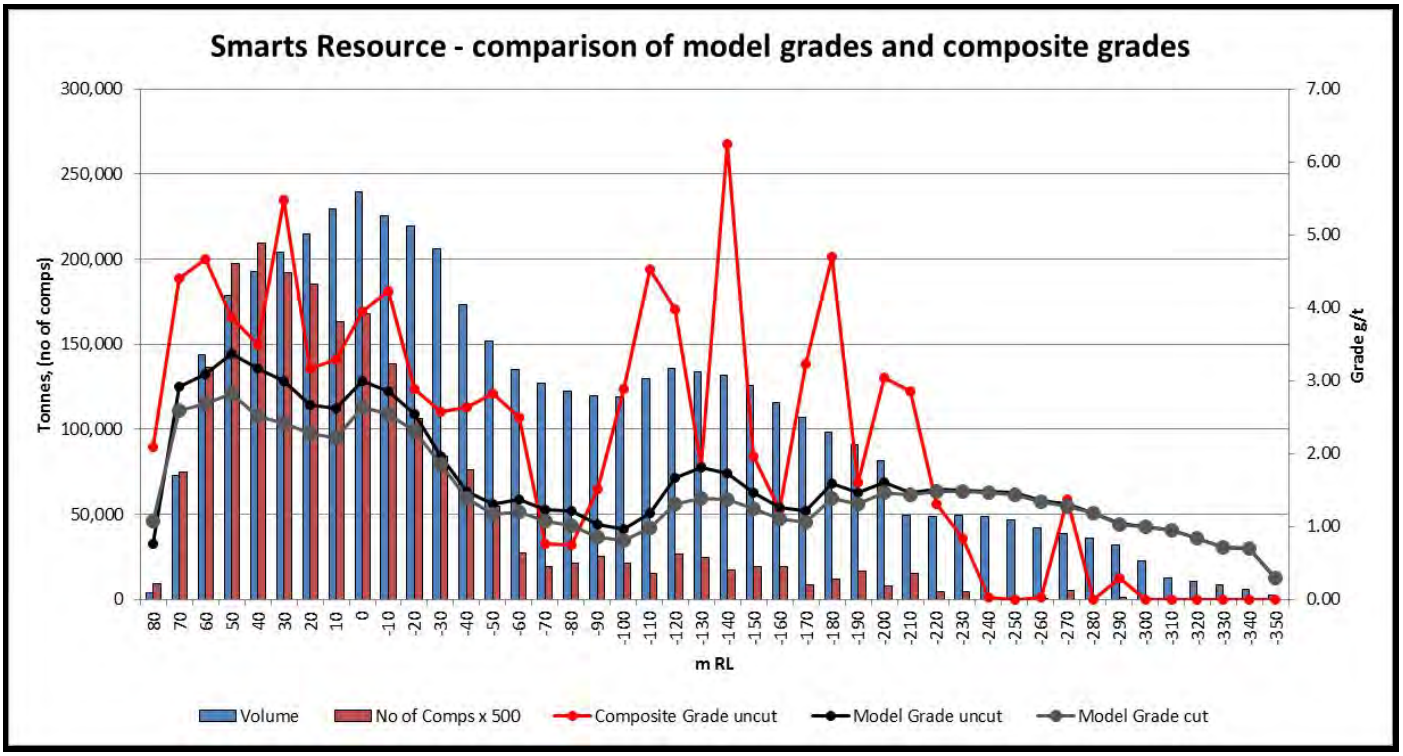

Figure 14-26: Smarts Zone Resource Validation by Elevation

14.4 Larken Zone

14.4.1 Preparation of Wireframes

14.4.1.1 Resource Mineralisation

The interpreted sectional outlines were manually triangulated to form a single wireframe as shown in Figure 14-27.

To form ends to the wireframes, the end section strings were copied to a position approximately midway to the next section or no greater than 25m. The end section strings were adjusted to match the dip, strike and plunge of the Larken Zone mineralisation. The wireframed objects were validated using Vulcan software and set as solids.

The mineralisation wireframe was treated as a hard boundary for all estimation purposes, that is, only assays from within the wireframe was used to estimate blocks within the wireframe.

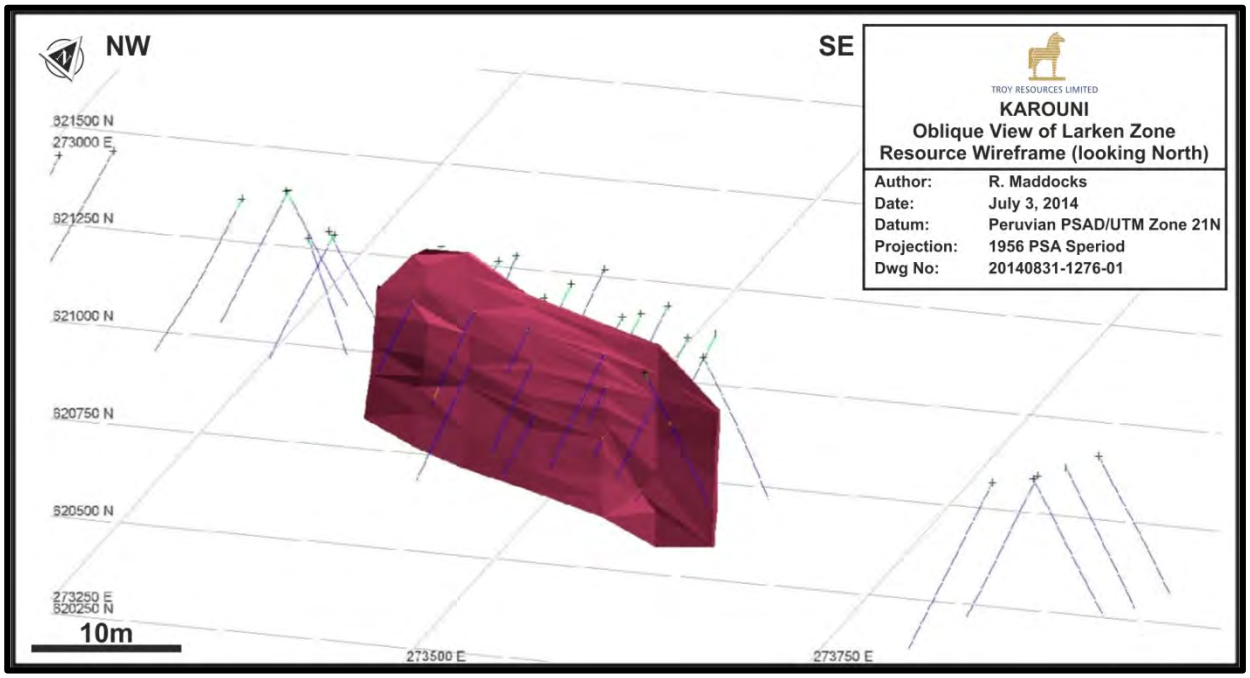

Figure 14-27: Oblique View of Larken Resource Wireframe (View Looking North)

14.2.1.2 Weathering Surfaces

Weathering surfaces were interpreted on cross section and extended to cover the block model limits. Weathering codes and degree of oxidation were included in the geology database and used to define the base of complete oxidation (*boco_2014.00t*) and top of fresh rock (*tofr_2014.00t*) surfaces.

The database geological codes used to define the weathering surfaces are presented in Table 14-2. The surfaces were mainly defined from the database field “Regolith”.

Conversely, both the regolith zone and the transition zone (i.e. base of complete oxidation) are defined by geological codes starting with ‘R’. The regolith unit is generally composed of the Rpa (pallid zone), Rmo (mottled zone), Rlt (laterite), Rfe (ferricrete), and Rsp (saprolite). The weathering of these units may vary, but generally spans from moderately weathered (mw) to extremely weathered (ew). It should be noted that the degree of weathering is not directly linked to the degree of oxidization, but to the competency of the rock.

The transition zone refers to the transition between the regolith and fresh rock. The database geological code used is Rsr (saprock). Since this does define the transition between regolith and fresh rock, this unit tends to be weakly weathered (ww).

14.2.1.3 Geological Surfaces

The overburden unit is defined by geological codes starting with 'O', and is generally composed of Osd (overburden sand) and Ocy (overburden clay) (Table 14-2). The overburden surface (*overburden_2014.00t*) was interpreted on cross section and extended to cover the block model limits.

14.2.1.4 Topographic Surface

The surface elevation is moderately undulating and ranges from 55m to 90m in elevation above sea level. This reflects the incisement of overburden and basement lithologies by numerous streams.

The current topographic surface was coded from a LiDAR survey undertaken in late 2011 (*contour_wo_resource_2m.00t*). The survey was undertaken by Digital World Mappine Inc. and is nominally accurate to +/- 0.3m on flat terrain. The surface was thinned and re-contoured to reduce file size, however trench excavation lines are still generally visible and the surface is considered to be of excellent quality.

Drillcollars, trench excavation lines and the LiDAR topographic surface generally show a reasonable correlation. To ensure consistency drill collars were adjusted in the Z direction (elevation) to match the LiDAR surface. Collar adjustments were generally minor or reflected corrections to either historical drill collars or where a hand held GPS had been used.

14.4.2 Sample Statistics

14.4.2.1 General

The wireframe of the mineralised zone was used to define the Resource intersections. The drill samples used to interpret the mineralisation are dominantly of 1m length. A 1m interval length was selected for compositing and further geostatistical analysis. This composite length reflects the majority of sample intervals and is at a suitable scale for the width of the Resource wireframes. Any residual composites of less than 0.5m in length were added to the preceding composite.

Vulcan software was then used to extract 1m down-hole composites within the intervals coded as resource intersections. A small number of negative assays representing assay detection limit were converted to 0.005g/t grades.

The composites were checked for spatial correlation with the objects, the location of the rejected composites and zero composite values. A master composite file was created including each of the individual Resource objects in the wireframe model.

14.4.2.2 Deposit Statistics

Summary statistics for gold are shown in Table 14-3 for the ore zone solid triangulation. Summary statistics by domain for gold for the Larken Zone are shown below in Table 14-18.

Table 14-18: Larken Zone Summary Statistics by Domain for Gold

Ore Zone	Volume	No of Composites	Gold Grade Low	Gold Grade High	Gold Grade Mean	Variance	Standard Deviation	CV
Larken	132,965	72	0.001	41.56	2.50	31.98	5.65	2.26

14.4.3 High Grade Cuts

Due to the lack of data points within the mineralization solid shape is was not possible to statistically determine the appropriate top cut. The Larken Zone is very similar geologically to the nearby Smarts Deposit so a top cut of 20g/t was applied which is the same as several of the Smarts Deposit mineralised wireframes.

The Mineral Resource estimate is reported using cut gold values.

14.4.4 Geostatistical Analysis

A detailed geostatistical analysis was not possible due to the lack of data points within the solid shape.

14.4.5 Grade Interpolation Parameters

The gold composite grades were interpolated into a Vulcan block model using inverse distance squared (ID2) algoithm. The search direction was based on the orientation of the mineralization and the ranges based on the drill spacing. All parameters are summarised in Table 14-19.

Table 14-19: Larken Zone Grade Interpolation Parameters

Parameter	Pass 1
Search Type	Ellipsoid
Bearing	305°
Plunge	0°
Dip	-90°
Major axis Search Radius	50m
Semi-Major Axis Search Radius	20m
Minor axis Search Radius	50m
Minimum Samples	10
Maximum Samples	40
Block Discretisation	3 X by 4 Y by 4 Z
Percentage Blocks Filled	49%

14.4.6 Block Model

A Vulcan block model was created to encompass the full extent of the zone. A list of block model parameters is displayed in Table 14-20 below.

The block model used a primary block size of 20m along strike (NW-SE) by 4m across strike by 10m vertical with sub-cells of 5m by 1m by 2.5m. The block model was rotated to 305 degrees on the horizontal plane to match the strike direction of mineralisation. Block model co-ordinates are on the national grid PSAD56-UTM Zone 21N to match the trench and drillhole database.

The parent Y block size was selected on the basis of approximately half of the average drill section spacing in better drilled areas of the deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. In addition, the selection of a 10m model cell height reflected the spacing of exploration drilling down-dip.

Table 14-20: Larken Zone Block Model Parameters

Model Name	Larken_June 2014.bmf		
	Y	X	Z
Minimum Coordinates	620,150	273,690	-150
Maximum Coordinates	620,650	273,870	150
Block Size (Sub-blocks)	20 (2.5)	20 (2.5)	20 (2.5)
Rotation	305° (horizontal plane)		
Attributes:			
Gold (Au)	Estimated gold grade from cut composites (g/t)		
Density	Bulk density		
Ore/Waste	1=waste, 2=mineralisation		
Rock Type	General rock type code		

Table 14-20: Larken Zone Block Model Parameters			
Model Name	**Larken_June 2014.bmf**		
	Y	**X**	**Z**
Minimum Coordinates	620,150	273,690	-150
Maximum Coordinates	620,650	273,870	150
Block Size (Sub-blocks)	20 (2.5)	20 (2.5)	20 (2.5)
Rotation	305° (horizontal plane)		
Weathering	Ob=overburden, ox=oxide, tr=transition,fr=fresh		

14.4.7 Bulk Density and Material Type

Bulk density values were assigned to each weathering material type based on values compiled from historical and recent bulk density test work for the Hicks Zone and Smarts Zone. The assigned bulk density values are detailed below in Table 14-21 and are considered to be consistent with nearby deposits in the region.

Table 14-21: Larken Zone Material Type and Bulk Density		
Material Type	**Bulk Density t/m^3 (Dry)**	**Range**
Overburden	1.82	Topographic Surface to Base of Overburden
Oxide	1.82	Base of Overburden to BOCO
Transitional	2.29	BOCO to TOFR
Fresh	2.75	Below TOFR

14.4.8 Resource Classification

The Larken Zone shows reasonable continuity of mineralisation but due to the lack of drilling data it has been classified as 100% Inferred Mineral Resource. Drillholes are located at about 70m spaced north-east orientated drill sections.

The drill spacing is sufficient to allow the grade intersections to be modelled into a coherent wireframe. Reasonable consistency is evident in the thickness and grade of the domain.

The authors consider that geological and mineralisation continuity has been demonstrated with sufficient confidence to allow the Larken Zone to be classified as Inferred Mineral Resource.

14.4.9 Results

A summary of the estimate reported at a 1.0g/t gold cut-off grade is presented in Table 14-22.

Table 14-22: Larken Zone June 30th 2014 Mineral Resource Estimate (1.0g/t Gold Cut-off)			
	Inferred		
Classification Type	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces (oz)**
Oxide	11,000	3.9	1,400
Transitional	20,000	5.6	3,600
Fresh	221,000	3.5	25,100
Total	**252,000**	**3.7**	**30,100**

(Note rounding errors may occur in compiled totals)

Figure 14-28 displays the block model for the Larken Zone, in longitudinal section coloured by cut gold grade (g/t).

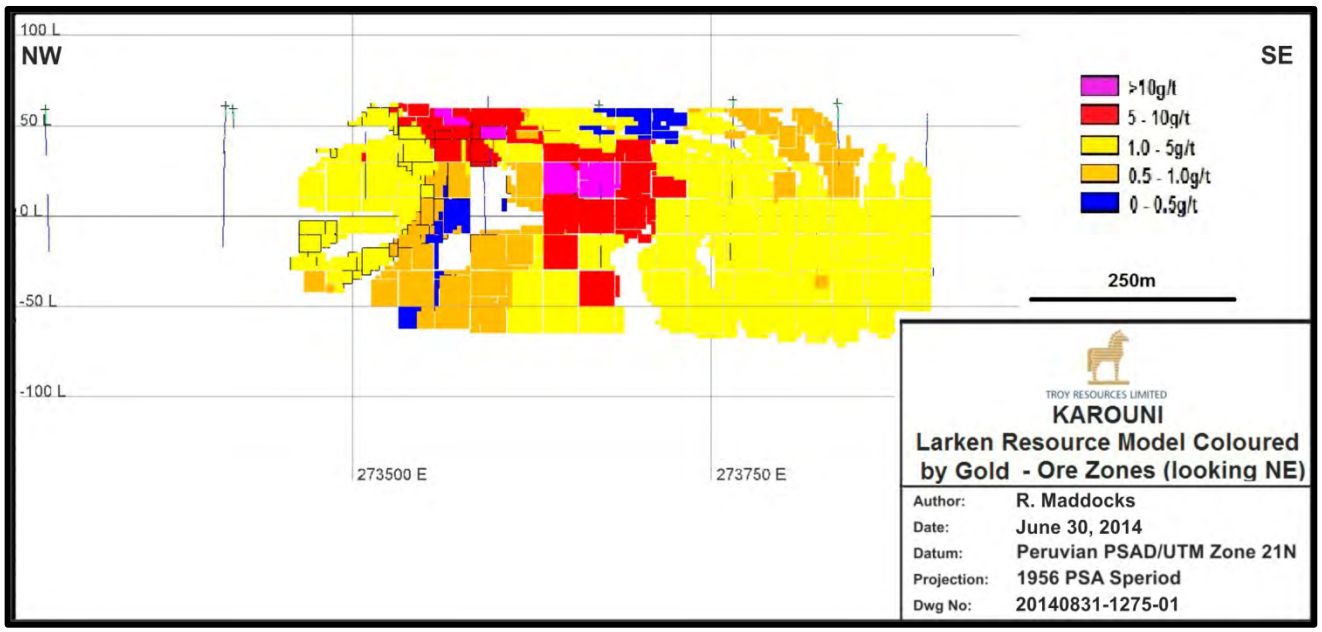

Figure 14-28: Larken Zone Resource Model Coloured by Gold g/t – (Looking NE)

To show the tonnage and grade distribution throughout the entire Larken Zone, a bench breakdown has been prepared for gold by weathering type material. The material type breakdown is shown in Figure 14-29. Grade tends to be higher in the transition and oxide zones.

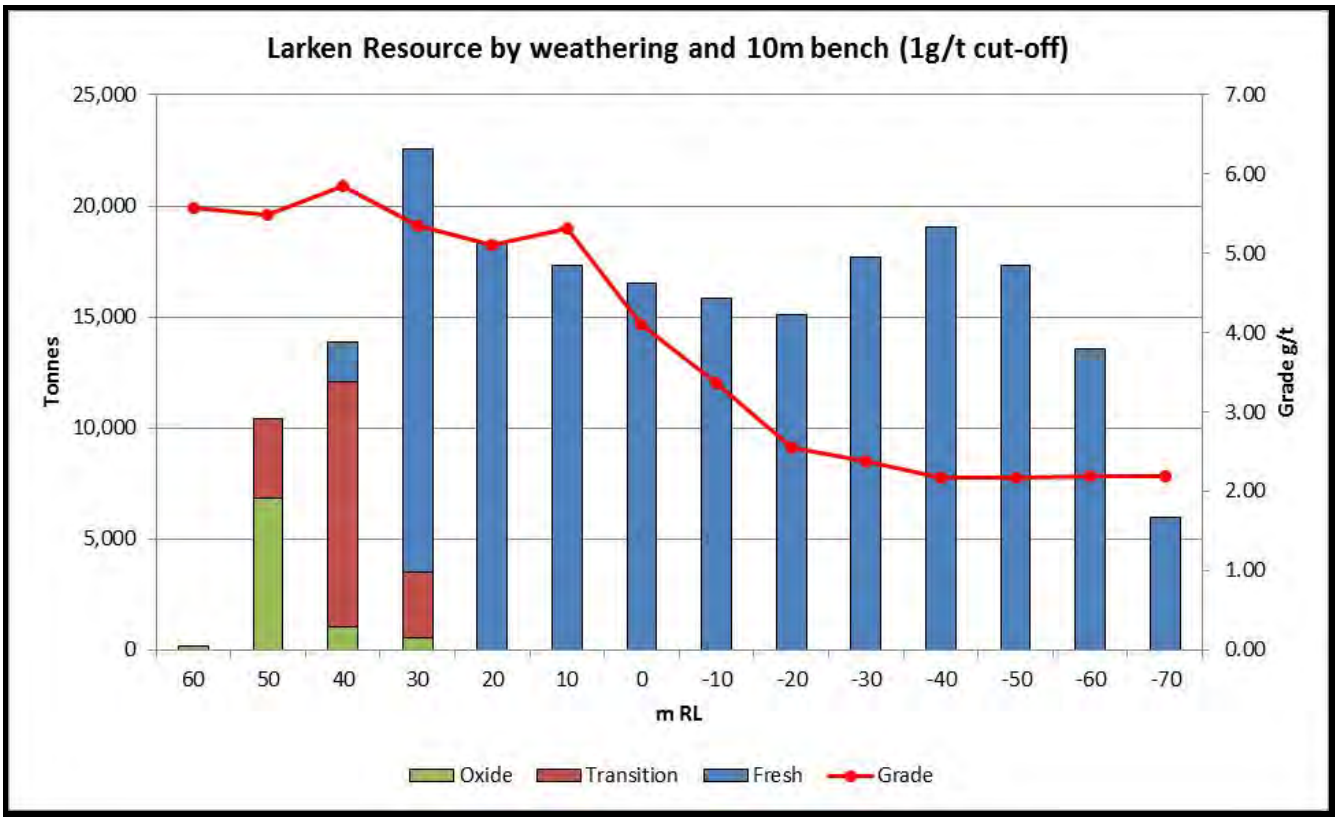

Figure 14-29: Larken Zone Resource – Global Bench Tonnage by Material Type

14.4.10 Model Validation

Due to lack of data points normal validation processes using swarth plots were not assessed. A qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. Overall the assessment indicated that the trend of the modelled grade appeared consistent with the drillhole grades.

15. Mineral Reserves

Mineral Reserves have been estimated for open pits only at the Smarts and Hicks Zones. Future underground operations will be estimated at a later date following the completion of deeper drilling and subsequent modelling.

15.1 Open Pit Mineral Reserves

Mineral Reserves were estimated based on the Mineral Resources that have been presented in Section 14 of this Technical Report. Mineral Resource estimates are inclusive of Mineral Reserves. The Mineral Resource block models were imported onto Whittle™ software and a series of pit shells generated. The input parameters used in the optimization are shown in Table 15-1. The optimal pit shell was chosen based on the economic analysis and the relative volumes. The optimization was conducted with underground mining costs included so as to limit the pit depth should underground mining become more economically attractive. This did not impact the pit volumes or reserves significasntly due to the geometry and location of high grade shoots. The lower high grade shoot that will be the subject of a future mining study was too deep to be exploited by open pit mining methods at the gold price and cost scenarios considered.

Table 15-1: Pit Optimisation Parameters

Parameter	Unit	Values
Slope Angles - Overburden	angle from horizontal	45
Slope Angles - Oxide	angle from horizontal	45
Slope Angles - Transition	angle from horizontal	55
Slope Angles - Fresh	angle from horizontal	70
Open Pit Mining Recovery	%	5%
Open Pit Ore Dilution	%	10%
Mining Production Limit	tonnes per annum	9,000,000
Processing Limit	tonnes per annum	1,000,000
Gold Price	US dollars per ounce	1,300
Processing Cost	US dollars per tonne treated	23.46
Open Pit Mining Cost	US dollars per tonne mined	5.00
Underground Mining Cost	US dollars per tonne mined	100.00
Processing Recovery	%	94
Government Royalty	% NSR	8

15.2 Mineral Reserves Estimate

The Mineral Reserves estimate is shown in Table 15-2.

Table 15-2: Mineral Reserves Statement Karouni Open Pit Project June 30 2014 Ore Reserve Estimate

	Proved			Probable			TOTAL		
Type	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**
Oxide	**194,000**	**5.0**	**31,200**	**251,000**	**2.0**	**15,700**	**445,000**	**3.3**	**46,900**
Transitional	**213,000**	**5.2**	**35,800**	**79,000**	**1.9**	**4,900**	**292,000**	**4.3**	**40,700**

Fresh	**1,331,000**	**4.6**	**197,400**	**550,000**	**2.2**	**38,300**	**1,881,000**	**3.9**	**235,700**
Total	**1,738,000**	**4.7**	**264,400**	**880,000**	**2.1**	**58,900**	**2,618,000**	**3.8**	**323,300**
Smarts Deposit									
June 30 2014 Ore Reserve Estimate									
	Proved			**Probable**			**TOTAL**		
Type	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**
Oxide	194,000	5.0	31,200	11,000	2.9	1,000	205,000	4.9	32,200
Transitional	213,000	5.2	35,800	9,000	2.9	800	222,000	5.1	36,600
Fresh	1,331,000	4.6	197,400	20,000	4.0	2,500	1,351,000	4.6	199,900
Total	**1,738,000**	**4.7**	**264,400**	**40,000**	**3.4**	**4,300**	**1,778,000**	**4.7**	**268,700**
Hicks Deposit									
June 30 2014 Ore Reserve Estimate									
	Proved			**Probable**			**TOTAL**		
Type	**Tonnes (t)**	**Gold (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**	**Tonnes (t)**	**Gold Grade (g/t)**	**Gold Contained Ounces**
Oxide	0	0.0	0	240,000	1.9	14,700	240,000	1.9	14,700
Transitional	0	0.0	0	70,000	1.8	4,100	70,000	1.8	4,100
Fresh	0	0.0	0	530,000	2.1	35,800	530,000	2.1	35,800
Total	**0**	**0.0**	**0**	**840,000**	**2.0**	**54,600**	**840,000**	**2.0**	**54,600**

The Karouni Gold Project is made up of the Hicks Deposit and the Smarts Deposit. These are reported with an effective date of June 30th 2014.

Mineral Reserves by definition have taken into account environmental, permitting, legal title, taxation, socio-economic, marketing and political factors and constraints, as discussed in Section 20 and Section 21 of this PFS. The Mineral Reserves are adequate to support mine planning.

16. Mining Methods

The Karouni Gold Project will be developed initially as an open pit operation at the Smarts and Hicks Deposits. The total mine life for the pits is three years. Underground mining will be assessed later.

16.1 Open Pit Mining Operations

Mining will commence six months before the processing plant is commissioned. This will enable adequate high grade stocks to be accumulated prior to the commencement of processing.

An initial fleet comprising two excavators will commence at the Smarts and Hicks Deposits. There is sand overburden up to 30m thick covering much of the Smarts Deposit whereas the Hicks Deposit outcrops. Initial ore production will be sourced from the Smarts Deposit with production at the Hicks Deposit commencing in month 5. Ore production will then focus on the higher grade Smarts Deposit. An additional fleet is mobilised after 5 months to provide additional capacity for the Hicks Deposit. This fleet is then demobilised in month 30 with a second fleet demobilised in month 29 and open pit mining concluded in month 36.

16.1.1 Smarts

The Smarts Deposit will be mined as a 4 stage pit. The first two stages will target shallow, higher grade pods with the final two stages amalgamating the pits into a single final pit. Mining will be via conventional backhoe configured excavators loading 30t rigid trucks. Mining will be done on 2.5m flitches over a 5m high blast. Initial sand overburden removal will be free dig with no blasting required.

Oxidation/weathering is of moderate depth at the Smarts Deposit with average depths to fresh rock of about 50m below natural surface. Blasting will however be required in the transitional zone and deeper parts of the oxidised profile in order to maintain excavator dig rates.

Final vertical depth of the stage 4 Smarts Pit is 140m. Total volume excavated will be 8,614,000m3. The final pit design incorporates the following design parameters see Table 16-1 and Figure 16-1:

Table 16-1: Smarts Pit Design Parameters		
Batter Angle	Overburden	45°
	Oxide	45°
	Transitional	55°
	Fresh	70°
Batter height	20m vertical in transitional and fresh, 10m in oxide and overburden	
Ramp width	12m single lane 18m double lane	
Berm width	5m	
Ramp gradient	1 in 9	

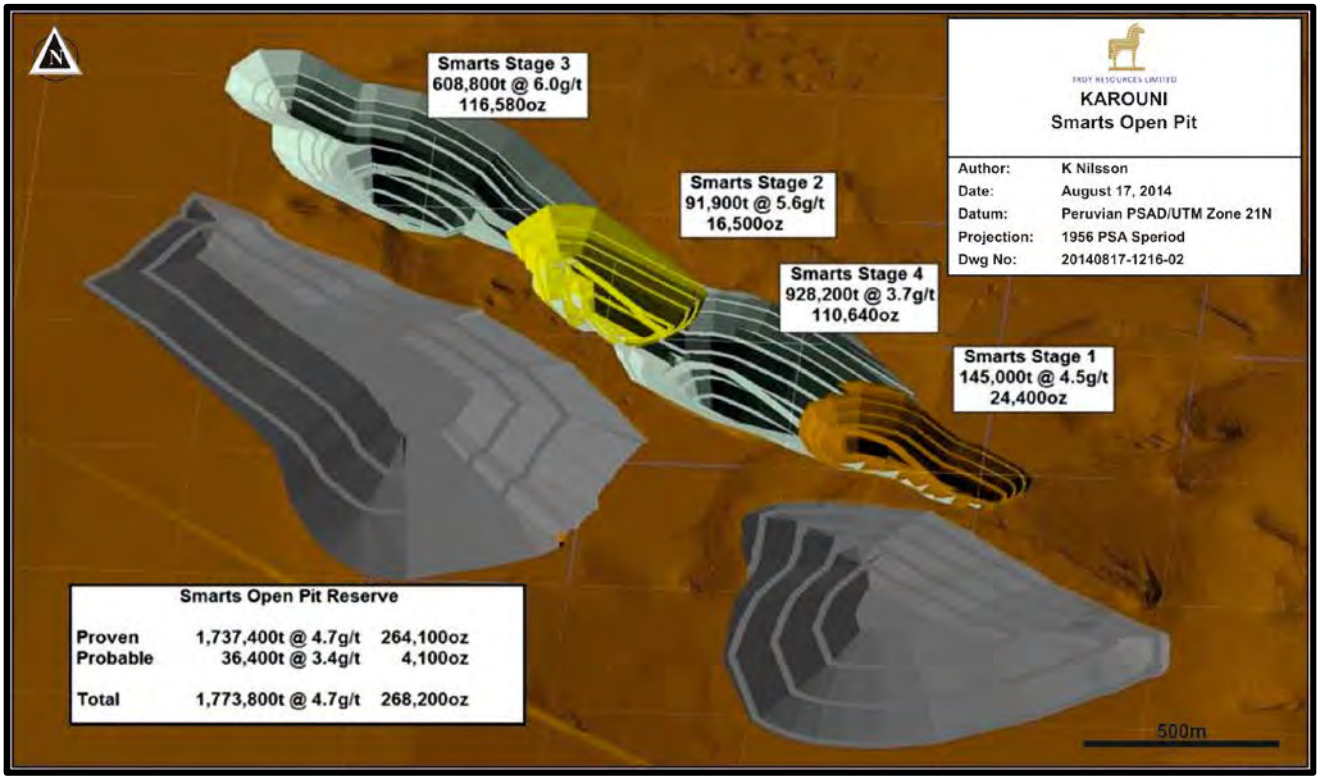

Figure 16-1: Smarts Pit Mining Plan

Drilling has encountered water accumulating at the base of the sand overburden. It is proposed to divert this water away from the pit crest via drains. Any naturally occurring water courses will also be diverted around the excavation using purpose built drainage channels.

Water in the pit will be pumped out using conventional in-pit sumps excavated as the pit progresses and water pumped to the pit crest for dispersal utilising the previously mentioned surface drainage infrastructure.

16.1.2 Hicks

The Hicks Deposit will be developed as a series of three discrete pits along the Hicks line of mineralisation (see Figure 16-2). Open pit mining will target higher grade, shallow mineralisation. The deepest pit, stage 2, extends to 90m below natural surface.

There is no overburden masking mineralisation at the Hicks Deposit. Mining will commence in month 5 after mining has started at the Smarts Deposit with the first two mining fleets.

Mining parameters will be the same as those used for designing the Smarts Pits.

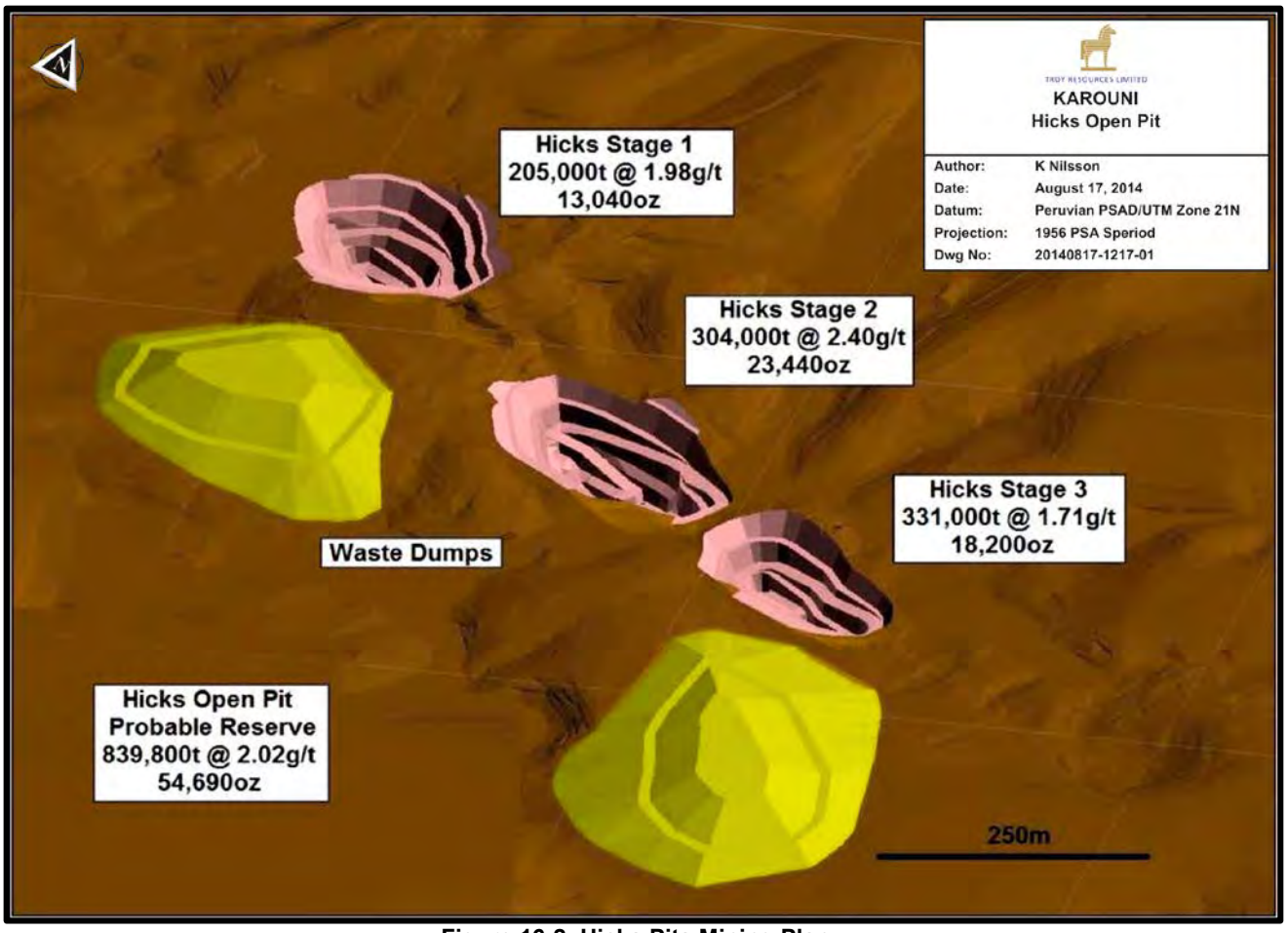

Figure 16-2: Hicks Pits Mining Plan

16.2 Open Pit Summary and Schedule

The proposed open pit and mining schedule is detailed below in Table 16-2 and Figure 16-3.

Table 16-2: Open Pit Summary and Schedule

	2014	2015	2016	2017	TOTAL
Total Ore Mined	**27,767**	**889,167**	**856,869**	**839,853**	**2,613,657**
Grade	**5.43**	**3.22**	**4.43**	**3.85**	**3.84**
Contained Ounces	**4,847**	**92,149**	**122,006**	**103,903**	**322,906**
SMARTS DEPOSIT					
High Grade Tonnes Mined	25,485	301,670	502,840	729,987	1,559,983
Grade	5.83	6.03	6.03	4.27	5.20
Contained Ounces	4,775	58,518	97,423	100,123	260,839
Low Grade tonnes mined	2,282	35,764	65,898	109,866	213,811
Grade	0.97	1.08	1.09	1.07	1.07
Ounces	71	1,236	2,301	3,780	7,388
HICKS DEPOSIT					
High Grade Tonnes Mined	-	368,386	250,565	-	618,951

Grade	-	2.20	2.60	-	2.36
Ounces	-	26,101	20,916	-	47,017
Low Grade Tonnes Mined	-	183,347	37,566	-	220,912
Grade	-	1.07	1.13	-	1.08
Ounces	-	6,295	1,366	-	7,661
ORE PROCESSED					
Tonnes	-	737,543	993,469	882,645	2,613,657
Grade	-	3.52	4.21	3.70	3.84
Contained Ounces	-	83,441	134,526	104,939	322,906
Recovery	-	0.94	0.94	0.94	0.94
Recovered Ounces	-	78,434	126,454	98,643	303,531

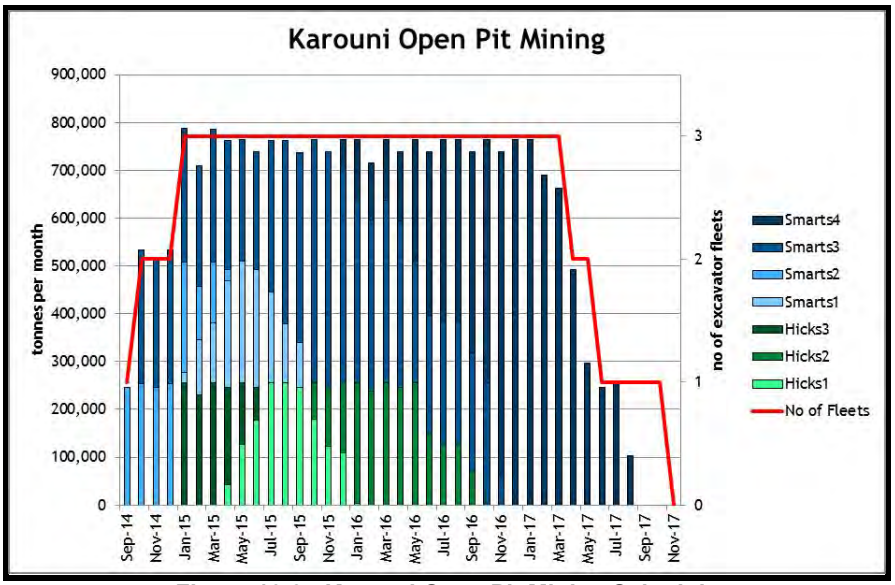

Figure 16-3: Karouni Open Pit Mining Schedule

17. Recovery Methods

The proposed method of gold recovery is via a conventional CIL gold processing facility. This will involve crushing, grinding, leaching and adsorption followed by elution and electrowinning. Final product will be gold in the form of doré bars. The plant throughput rate is designed to treat 1,000,000 tonnes per annum of run of mine ore delivered to the primary crusher.

17.1 Description and Flow Sheet of Process Plant

The process plant design is based on a conventional CIL plant with primary and secondary crushing to produce mill feed for the grinding circuit where water is added. The mill product will be controlled by the use of hydro-cyclones to produce the desired feed size to the leach circuit. A pre-leach thickener will be used to control leach feed density. Oversize material returning from the cyclones for further grinding will pass over screens to remove the +2mm material. The +2mm material will overflow the screen back to the grinding mill, while the -2mm undersize is fed to two centrifugal concentrators for recovery of the coarser gravity gold. The concentrate from the centrifugal concentrators provides a high gold grade feed to an intensive leach reactor system. The expected free or coarse gold is anticipated to be of the order of 40% – 50% of the total gold entering the plant. The high tenor gold solution from the intensive leach reactor (ILR) will be sent to the gold refinery section for electrowinning and smelting. The doré bars produced will be transported to an external refinery for final processing.

The leach feed will be processed in a series of agitated tanks comprising one leach tank and six carbon tanks, with sodium cyanide being added into the first four tanks as required by the process. Compressed air will also be added to assist in the leaching process, as required, in the first three tanks. Activated carbon will be added to the adsorption circuit and it will adsorb the gold with the loaded carbon moving counter current to the slurry flow in the circuit until sufficiently loaded for stripping, when the carbon will be sent to an elution circuit for gold recovery by electrowinning and smelting. The stripped carbon will be regenerated and returned to the adsorption circuit.

The leached ore (tailings) will pass through a series of three detoxification tanks where sodium metabisulphite (SMBS) and copper sulphate will be added together with air to drive the detoxification reaction. Lime slurry will also be added for pH control to improve the destruction kinetics. The detox circuit will remove any free and WAD cyanide driving the contained CN^- ion concentration down to or lower than the required levels. The detoxified tailings are then pumped to a large tailings storage facility (TSF). As tails settle any water recovered will pumped back to the processing plant or stilling ponds along with any rain water retained.

17.2 Plant Design, Equipment Characteristics

The processing plant has been designed for an annual processing rate of 1,000,000t/a as a start-up configuration, with the capacity to increase production when or if required. The crushing circuit is based on the use of a Metso C 120 jaw primary crusher followed by a HP 500 secondary cone crusher and a double deck vibrating screen with the screen oversize from both decks being returned to the cone crusher for further crushing. The crushed product will be transferred via a conveyor to a covered mill feed stockpile, which has a live capacity sufficient for 24hr of production. The mill feed will be drawn from the stockpile using an open slot and a belt feeder with variable speed control. There is an emergency mill feed system in place to allow for crusher overhauls and down time. The emergency mill feed system utilises a feed hopper mounted over the mill feed conveyor belt. The mill feed conveyor will have a weightometer to record production and to control the mill feed tonnage rate. Lime slurry will be added to the mill feed chute as required to maintain a high pH (10.5) in the milled ore, which will prevent cyanide losses by volatilisation.

The grinding circuit will consist of a 3.2MW ball mill with the grind size being controlled by a bank of six hydrocyclones of which four will be in use with two on standby. Grinding balls will be added to the mill, manually as required.

A pre-leach thickener will be used to control the leach feed density. The leach circuit will consist of 7 x 600m^3 steel tanks, each with agitators. Compressed air inlets will be available on the first four tanks and utilised as required. Carbon transfer pumps will be installed in the CIL tanks. The tailings from the leach circuit will be treated using the INCO SO_2 Air process reducing any free and WAD cyanide down to permitted levels. The tailings will be deposited in a TSF built to contain the total calculated volume for the life of mine. Excess water will be pumped back to the plant for re-use, with excess rain water passing through stilling ponds prior to being discharged from the site. The control of and measurement of feed and discharge volumes and density of milled material is carried out by way of nucleonic density meters fitted to the leach circuit feed and discharge lines respectively. Total plant operation will be controlled by an Emerson Delta V computerized process control system to ensure control over feed, discharge, chemical levels, tank level control and to promote maximum efficiency.

Gold recovery from the coarse gold fraction during the grinding and classification process will be by way of centrifugal concentrators followed by an intensive leach reactor system with the final pregnant leach solution going to the refinery for electrowinning and smelting.

The elution and refining circuit will strip the carbon of gold by way of a pressurised strip circuit containing a carbon acid wash column, gold stripping column, integrated pressurized electrowinning system and a carbon transfer and regeneration system. The production of gold doré bars will be achieved by cleaning the electro-won gold sludge, and drying and smelting the product with appropriate fluxes. The gold room will be a high security facility. See Figure 17-1 below.

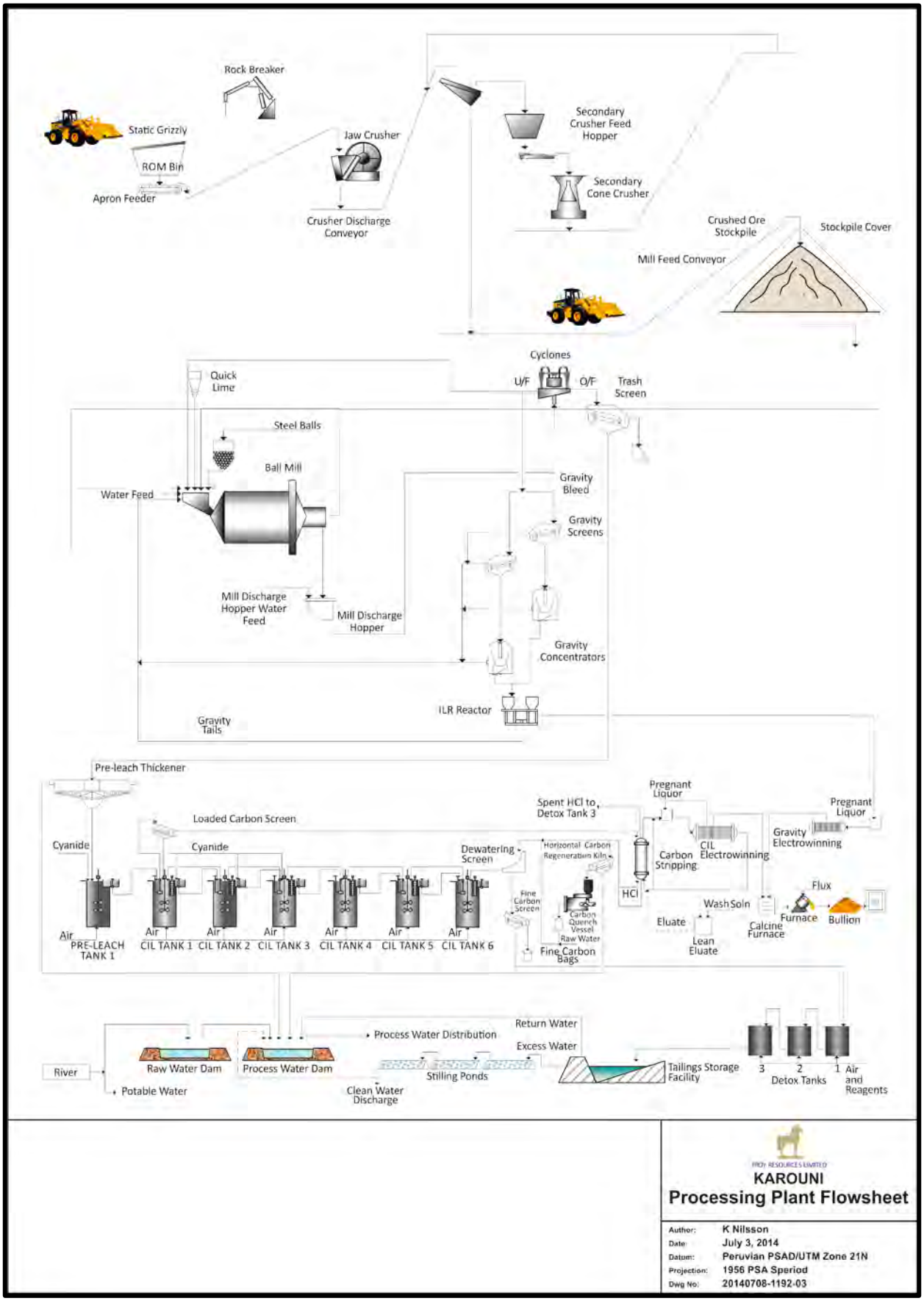

Figure 17-1: Karouni Project Processing Plant Flowsheet

17.3 Current or Projected Requirements for Energy, Water and Process Materials

The processing plant will require chemicals, electrical energy and water to operate. The expected level of electrical energy use is ~5.7MW with electricity supplied from a company owned and operated diesel powered generating facility of ~10MW capacity consisting of a number of smaller units, delivering power to be distributed over a 13,800 volt network at 60Hz. This will provide backup power and room for expansion.

The water usage will initially be high until the tailings return water has stabilized. Expected use of process water is ~800,000m3/a to cover wash down, camp, mine and other uses. Approximately 45 % of the process water is expected to be returned from the TSF. The remainder will be derived from pit de-watering, a bore field and possibly small dams. Guyana is a very high rain fall area and a lack of water is not seen as a potential issue.

In terms of chemicals, the Karouni Gold Project will require approximately 500t of CN, 1,600t of lime, 700t of grinding balls, 17,675t of diesel fuel, 480t of SMBS, 100m3 of 32% hydrochloric acid, 30t of flocculant, 30t of caustic soda and a number of smaller quantities of other chemicals each year. None of the required products are available in Guyana so these will need to be imported. See proposed plant layout in Figure 17-2.

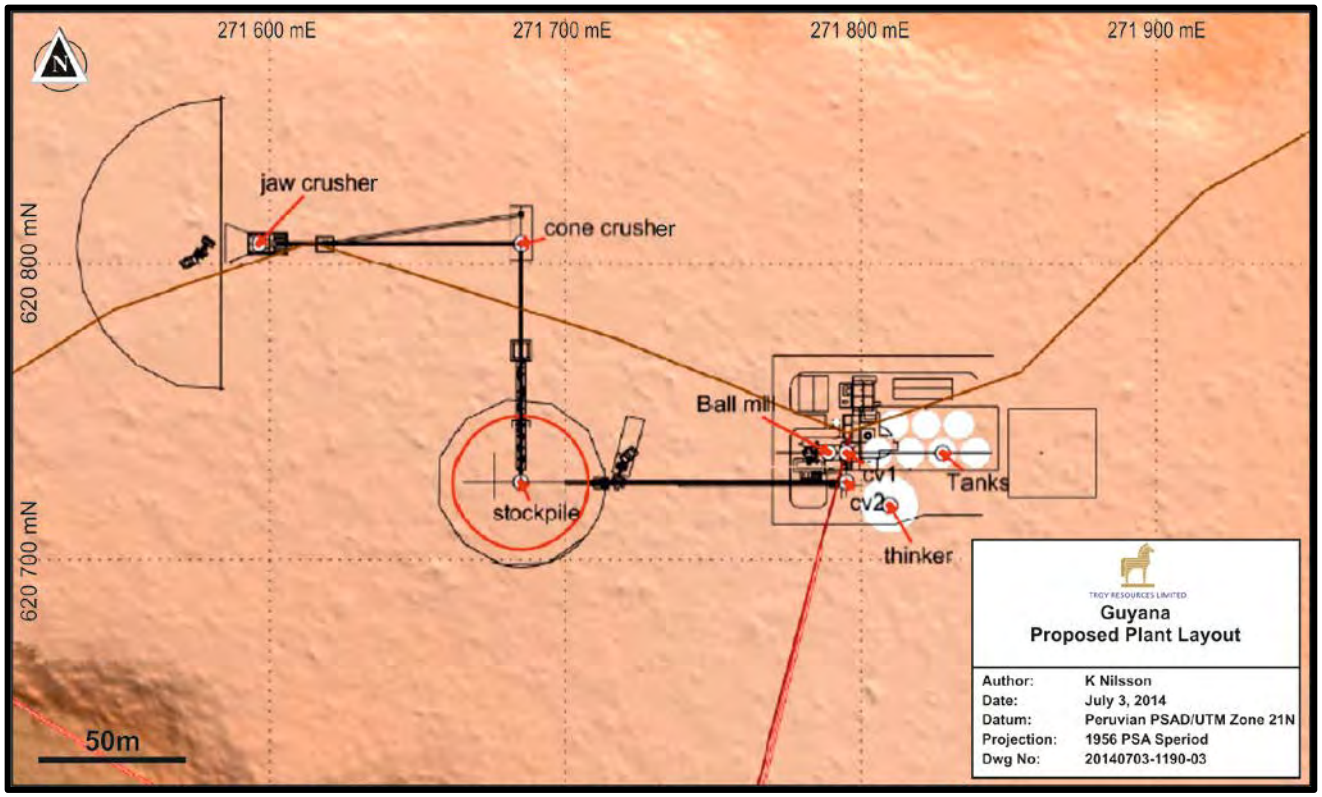

Figure 17-2: Proposed Plant Layout

18. Project Infrastructure

The following infrastructure is proposed for the Karouni Gold Project see Figure 18-1:

i) Processing plant
ii) Power generation facility
iii) Offices and warehouse
iv) Fuel storage facility
v) Laydown and workshop facility

vi) Accommodation village
vii) Tails storage facility
viii) Water storage
ix) Haul roads
x) Main access road
xi) Airstrip
xii) Waste rock dump

18.1 Processing Plant

The processing plant is proposed to be located centrally between Smarts and Hicks Deposits.

18.2 Power Generation

The power generation capability will be made up of 5 diesel gensets of 2MW capacity each. The powerhouse will be located adjacent to the processing plant. Diesel will be trucked in from Georgetown and stored in a dedicated storage facility onsite.

18.3 Offices and Warehouse

The main administration office and warehouse will be located adjacent to the processing facility. The office will house all administration and technical staff. The warehouse will carry all necessary spare parts and consumables for daily operational requirements.

18.4 Fuel Storage

A facility for storing 160,000 liters of diesel fuel will be constructed adjacent to the processing plant. The facility will be fully bunded and will deliver fuel to the powerhouse. In addition there will be provision for refuelling light vehicles.

Another storage facility will be located at the laydown and workshop area. This facility will be utilised for refuelling the open pit.

18.5 Laydown and Workshop Area

An area dedicated to heavy vehicle maintenance will be located near the main pit haulage road. This area will contain an undercover maintenance facility for the servicing of heavy equipment and offices for maintenance personnel. It will also have a diesel storage facility for the refuelling of mining vehicles.

18.6 Accommodation Village

An accommodation village for 300 personnel will be constructed to the south of the processing facility and adjacent to the airstrip. The village will have facilities for accommodating 30 personnel in ensuited type units. The village will include messing facilities for meals and access to satellite television and telephones.

18.6 Tails Storage Facility

The tails storage facility will be accommodated in a natural valley located to the south-west of the main Smarts and Hicks Deposits. A compacted, earthern embankment will contain the tails. The tails dam will be designed to hold all of the planned tails discharge from the processing plant over the forecast 3 year mine life.

18.7 Water Storage

A fresh water storage facility will be constructed to hold potable water. In addition, a process water facility will also be constructed.

18.8 Haul Roads

A network of haul roads will be constructed to transport material from the mines to the processing facility. These roads will not be accessible to the public. The haul roads will be sprayed with water to minimise any potential dust.

18.9 Main Access Road

The main access road is largely in place. Access is via the Amalia hydro-electricity access road.

18.10 Airstrip

An airstrip for light aircraft has been completed and will be utilised for transport of personnel to and from Georgetown.

18.11 Waste Rock Dump

Waste rock dumps will be located to the north-east of the Hicks and Smarts Deposits. The Smarts Deposit dump will consist of one main dump whereas the Hicks Deposit dumps will be located adjacent to each of the pits. Top soil and oxidised waste material will be stored until completion of the dumps and then spread over them in order to enable rehabilitation and revegetation of the completed dumps.

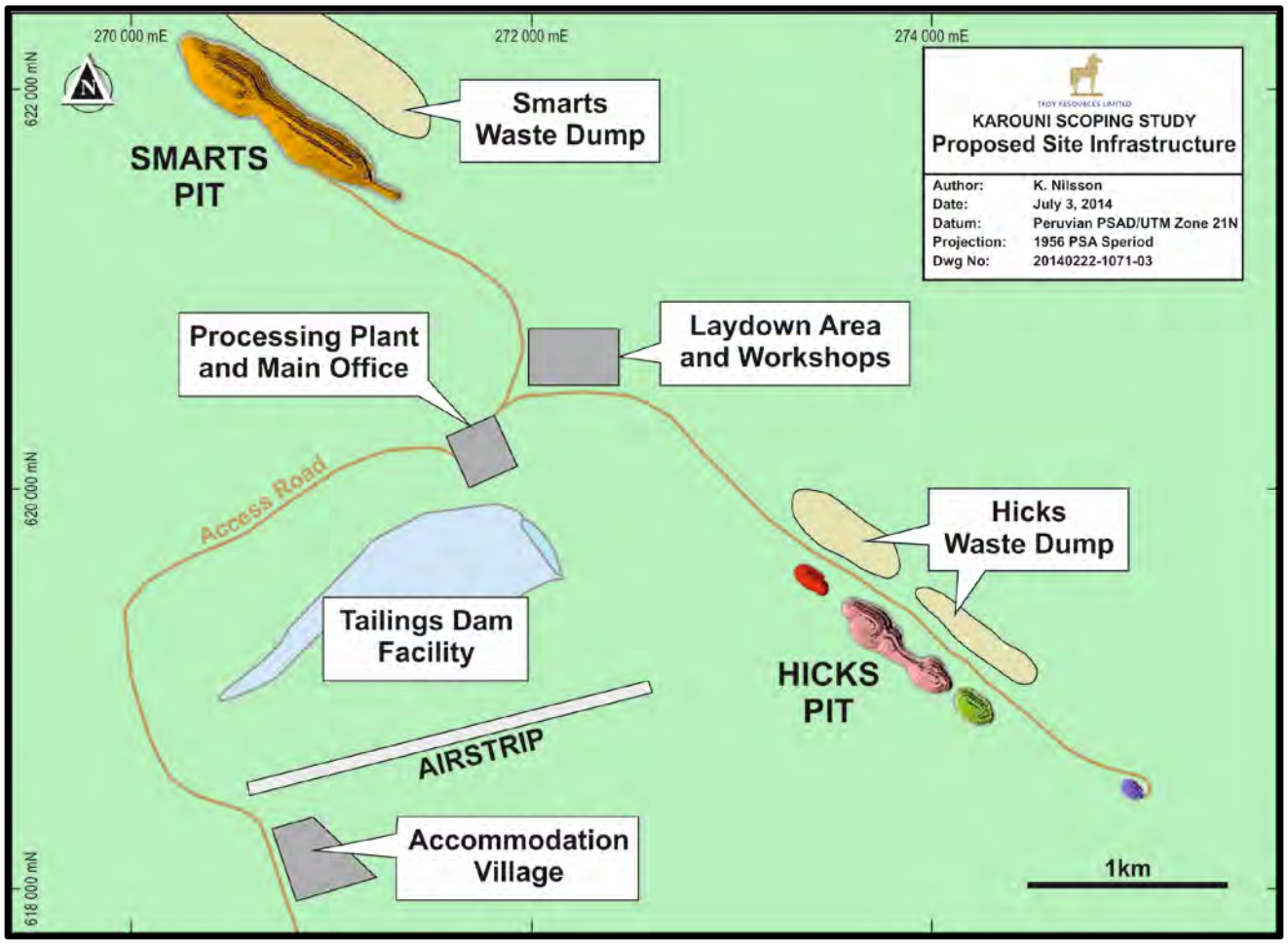

Figure 18-1: Karouni Scoping Study Proposed Site Infrastructure

19. Market Studies and Contracts

No market studies were carried out for this technical report. The final product of the Karouni Gold Project will be gold doré bars. These will be easily sold and absorbed in the current market at prevailing global gold prices.

20. Environmental Studies, Permitting and Social or Community Impact

A draft Environmental and Social Impact Assessment (ESIA) has been completed for the Karouni Gold Project and lodged with the Guyana Environmental Protection Agency (EPA). The ESIA has been prepared to conform to guidelines established by the EPA and as detailed in the EPA publication titled Environmental Impact Assessment Guidelines, Volume 3 Mining, Version 1, August 2000. In accordance with those

guidelines the ESIA is comprised of three components, the Environmental Baseline Study, the Environmental and Social Assessment and the Environmental and Social Impact Management Plan.

The Environmental and Social Assessment identifies and assesses the impacts of the proposed action and its alternatives taking into consideration the measures that are proposed to mitigate and/or enhance the probable impacts. The Environmental and Social Assessment identifies the project-environment interactions during design, construction and operation and after closure of the operation. The planned implementation schedule, associated with mitigation of environmental effects, is also detailed as part of the ESIA.

Final requirements and plans for waste and tails disposal, site monitoring and water management were determined as part of the PFS process. Permitting is progressing and will include rehabilitation and bonding requirements. Mine closure planning, including remediation and reclamation, will be finalised as part of the permitting process.

20.1 Approach

The primary reports available for the Karouni Gold Project are the baseline characterization report and the Preliminary Scoping Report. These reports were prepared over the period 2012 to 2013. The baseline characterization report details the physical, ecological and sociocultural baseline data for the project area and its immediate vicinity. The Preliminary Scoping Study (PSS) presents details on the mine site infrastructure, the process plant and plans for project implementation. The ESIA incorporates both the baseline characterization data and information presented in the PSS.

The overall ESIA process followed is depicted in Figure 20-1 below.

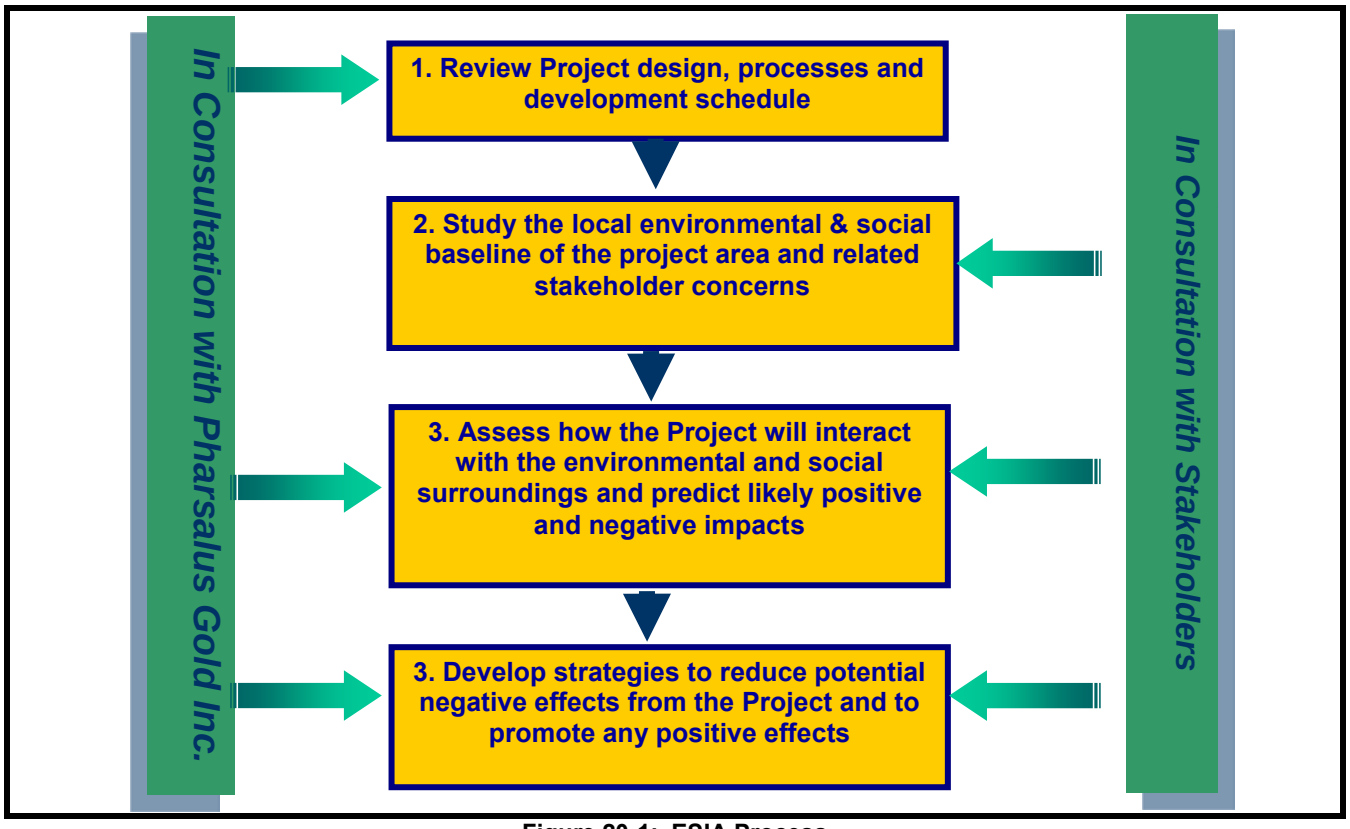

Figure 20-1: ESIA Process

20.2 ESIA Methodology

The methodology incorporated the following tasks which are described in more detail below:

Review of existing documentation;
Assessment of policy, legislative and regulatory framework;
Evaluation of historic baseline data;
Determination of scope of assessment;

Public meetings;
2012 & 2013 field activities and surveys;
Impact assessment and rating;
Development of environmental and social management plan;
Institutional analyses for project implementation;
Development of monitoring program;
Preparation of emergency response procedures; and
Preparation of closure guidelines

20.3 Review of Existing Documentation

The ESIA reviewed available historic data for the mine site and related areas. The review examined data on environmental, economic and social variables which included site geology, hydrogeology, surface water flow, biodiversity, conservation priorities, the proximity of indigenous communities, mining operations now underway and those formerly taking place in the area, and the likely presence of archaeological resources in the area. Examples of documents reviewed include:

- Geological reports for the area;
- Checklists of Plants, Fish, and Terrestrial Fauna of the Guiana Shield Region;
- 2009 IUCN Red List of Threatened Species;
- NatureServe and IUCN species distribution maps and GIS data;
- Alliance for Zero Extinction data;
- Guiana Shield Initiative studies and GIS data;
- Forestry Studies for the area;
- Data on Aquatic Resources of the area;
- Indigenous Groups reports for the mining sector;
- NGO information, including Conservation International (CI) and World Wildlife Fund (WWF);
- World Bank (IFC) guidelines on Baseline Characterization;
- World Bank (IFC) guidelines on Environmental Assessment;
- Guyana Environmental Protection Act;
- Guyana Town and Country Planning Act;
- Guyana Forestry Commission Guidelines;
- Guyana EPA Guidelines on Environmental Management Plans;
- Guyana EPA Guidelines on Human and Environmental Risk Assessment; and
- Guyana EPA Guidelines on Air Emissions Modeling.

Guidelines for the mining industry (e.g., ICMM), including those for the management of cyanide and other hazardous chemicals and standards for the construction of roads in mining areas, were also reviewed to determine their relevance to the project.

20.4 Assessment of Policy, Legislative and Regulatory Framework

A review of the applicable Guyanese Regulatory Framework regarding environmental and social requirements of the Karouni Gold Project was undertaken. A comparison was made with requirements of the IFC Performance Standards (PSs) and Environmental, Health and Safety (EHS) standards. In those instances where Guyana regulations differed from the levels and measures presented in the IFC EHS guidelines, the more stringent of the two requirements formed the basis of this ESIA.

20.5 Evaluation of Historic Baseline Data

Physical, biological, and socio-cultural baselines were completed for the Karouni Gold Project by Ground Structures Engineering Consultants (GSEC) in 2012. The baseline compilation program included biological field surveys, recovery and analyses of surface water samples, visits to communities within proximity of the mine site and compilation of data for the Karouni Gold Project area and its vicinity. This ESIA is based on a review of that baseline information and data gathered from additional field programs to fill data gaps and to confirm data acquired by the initial baseline studies.

20.6 Determination of Scope of Assessment

Based on the review and analysis of existing information, the scope of the assessment was determined, including the methodologies to be followed for the additional baseline field studies including hydrogeology, biodiversity, and socio-economic. Scoping identified:

- The appropriate boundaries of the Karouni Gold Project (e.g., project area of influence);
- Major issues and concerns;
- The information necessary for decision-making;
- The significant effects and factors to be considered; and
- The scope and methodology of the assessment.

20.7 Public Meetings

Representatives of PGI and GSEC attended mandatory government agency sector and community stakeholder meetings chaired by the Environmental Assessment Board under the Guyana EIA legislation. A Government Agency Sector Review meeting was held in Georgetown on November 15, 2013 and an advertised public meeting was held by the EPA with a presentation of the Karouni Gold Project and proposed activities by PGI at the Kaburi Village Benab, the regional seat of Region 7, on November 16, 2013. Concerns and issues raised by attendees at the meeting have been incorporated into the analysis.

20.8 Field Activities and Surveys

The baseline data was expanded to include ecology information for the proposed new access road to the mine site. The initial baseline ecology was acquired by utilizing various methodologies including field surveys, direct observation, and where appropriate, through interviews and questionnaires. Data collection for the physical environment focused on surface and groundwater water hydrology and quality and soil type. The biological environment data collection was focused on flora and fauna, rare/endangered species, sensitive and critical natural habitats, species of commercial importance, migratory species and the biodiversity of the area.

20.9 Hydrogeological Data Acquisition

Five (5) well nests were drilled within the mine site area. Each nest was comprised of three wells, one each in alluvials, saprolite and bedrock. A groundwater monitoring well was installed in each well at each well nest. One borehole was located within the footprint of the area identified for the tailings pond and another within the footprint of the area identified for the process plant. Ground water levels were recorded at each monitoring well to provide data for hydrogeological evaluation.

Groundwater samples were collected from each the appropriate monitoring well at each well nest after development. Each ground water sample was analyzed for the presence and concentrations of heavy metals, and other constituents to conform to Guyana EPA guidelines. The minimum list of parameters for which all samples were analyzed coincided with those identified by the World Bank for cyanidation operations for the recovery of gold.

20.10 Surface Water

Water samples were recovered from several streams in the Karouni Gold Project area and tested to determine the concentration of chemical constituents. Snap shots estimates of surface water flows, on major streams/creeks/rivers upstream and downstream of the mine site were measured and coupled with surveyed cross-sections of those streams to develop discharge estimates for surface water modeling. The slope of each stream was also recorded at various locations to provide additional data for surface water modeling. Surface water samples were also recovered from major streams/creeks/rivers upstream and downstream of the mine site. Sediment samples were also recovered from point coincident with surface water sampling points to confirm to Guyana EPA guidelines. Both surface water and sediment samples were analyzed for the presence and concentrations of heavy metals, and other constituents to conform to Guyana EPA guidelines. The minimum list of parameters for which all samples were analyzed coincided with those identified by the World Bank for cyanidation operations for the recovery of gold.

An analytical model was utilized to assess potential impacts to surface water from various discharge, spill, and release scenarios during mine construction and operation. The primary objective for the ESIA was to assess the duration, spatial extent, and dilution of accidental discharges, potentially containing cyanide, to the Kaburi River. Both small spills that may occur from time to time and catastrophic releases were assessed for the proposed Karouni Gold Project.

20.11 Biological Resources

Two field survey of biological resources for the mine site area and its environs was conducted, one each for a single wet and a single dry season. During the surveys all biological features at the mine site and its environs were sampled. Unknown varieties were sampled, preserved and returned to Georgetown for identification. In addition, a forester prepared estimated quantities of the most dominant species by measuring the breast height, diameters and other data necessary to determine plant biomass.

20.12 Socio-Economic Resources

The socio-economic baseline was compiled by both stakeholder mapping and information gathering. Focused consultations were held with the Ministry of Amerindian Affairs and with Indigenous Peoples NGOs (GOIP, APA and TAAMOG) to gather information on the presence of indigenous communities in the Karouni Gold Project area. Data was compiled on the presence of human communities in the area and agricultural, forest production, fishing and hunting practices. The socio-economic baseline also established resource use patterns in the area and examined land use such as agriculture, forestry and mining. The presence of social infrastructure such as schools, medical facilities, communications/transport networks and markets was established. Data was also compiled on public health and on health risks and diseases in the vicinity of the project.

A fact finding or information seeking exercise was undertaken of small rural communities in the vicinity of the project through consultations. The key objective was to assess the baseline of the communities and identify impacts (if any) that the Karouni Gold Project could potentially have on the human environment. The main approach therefore was stakeholder 'consultation' focused and was aimed at participatory engagement of stakeholders in project conception, design and implementation.

An information seeking/research oriented exercise was also undertaken with the communities in the Karouni Gold Project Area of influence to profile the socioeconomic baseline conditions of these communities and the potential impacts (if any) that might arise as a result of the Karouni Gold Project. The following activities were carried out in the communities:

- Discussion with community leaders - informal discussion to provide information on the Karouni Gold Project and to understand community concerns;
- Focus Group Discussions - open ended discussions with groups or sub groups within each communities such as women, youth, elderly people; and
- Community survey forms - these forms were completed in each community to gain basic information on the natural setting, governance, local customs and economy, social development, infrastructure, administrative and economic conditions.

20.13 Impact Assessment and Rating

An impact rating system was developed for the ESIA to enable a systematic ranking of identified environmental and social impacts and presentation of a summary table which can be used by the Karouni Gold Project to prioritise mitigations according to ranking. The impacts section presented in this report ranks impacts using the criteria presented in the following sections.

The impact rating is the product of two elements: (1) the severity of the potential impact and (2) the likelihood of the impact or "event" occurring.

20.14 Severity and Enhancement Criteria

Firstly, the severity or enhancement of each impact was rated using the criteria identified in Table 20-1, Table 20-2, Table 20-3 and Table 20-4. Note that colours are used to assist the reader in reviewing the impacts and their relative magnitude. As such, they should not be considered definitive.

Table 20-1: Severity Criteria (Negative Environment Impacts)	
Low	Affects environmental conditions, species, and habitats over a short period of time, is localized and reversible.
Medium	Affects environmental conditions, species and habitats in the short to medium term. Ecosystems integrity will not be adversely affected in the long term, but the effect is likely to be significant in the short or medium term to some species or receptors. The area/region may be able to recover through natural regeneration and restoration.
High	Affects environmental conditions, species and habitats for the long term (i.e., over the life of the Project) may substantially alter the local and regional ecosystem and natural resources, and may affect sustainability. Regeneration to its former state would not occur without intervention. Affects environmental conditions or media over the long term, has local and regional affects and/or is irreversible.

Table 20-2: Severity Criteria (Negative Social or Health Impacts)					
Ranking	**Duration**	**Extent**	**Ability to Adapt**	**Socio-cultural Outcome**	**Health Outcome**
Low	Short-term <1 year low frequency	Individual/ Household	Those affected will be able to adapt to the changes with relative ease, and maintain pre-impact livelihoods, culture, quality of life and health.	Inconvenience but with no consequence on long-term livelihoods, culture, quality of life, resources, infrastructure and services.	Event resulting in annoyance, minor injury or illness that does not require hospitalization
Medium	Medium-term 1-6 years Medium or intermittent frequency	Small number of household	Those affected will be able to adapt to change, with some difficulty, and maintain pre-impact livelihoods, culture, quality of life and health but only with a degree of support	Primary and secondary impacts on livelihoods, culture, quality of life, resources, infrastructure and services	Event resulting in moderate injuries or illness, which many require hospitalization
High	Long-term/ Irreversible > 6 years Constant frequency	Large part of/ full settlement	Those affected will not be able to adapt to changes and continue to maintain pre-impact livelihood	Widespread and diverse primary and secondary impacts likely to be impossible to reverse or compensate for.	Catastrophic event resulting in loss of life, severe injuries or chronic illness requiring hospitalization.

Table 20-3: Enhancement Criteria (Positive Environmental Effects)				
Ranking	**Duration**	**Extent**	**Degree of Change**	**Focus/ Sensitivity**
High level of enhancement	Benefits will be sustained over the long term.	Benefits will extend beyond local environment (i.e., linkage of fragmented habitat, e.g., regional corridor)	Direct benefits to species or resources will provide significant opportunities for sustainability.	Benefits will pertain to species, habitats and natural resources that are degraded, or are sensitive, rare, or in need of protection..
Medium level of enhancement	Benefits will be measurable in the short term and possibly longer.	Benefits to species, habitats and natural resources in the local environment and beyond.	Moderate benefits to species, habitat, and natural resources that may provide some opportunities for sustainability.	Benefits will pertain to species, habitats and natural resources that have some level of degradation, sensitivity, or rarity.
Low level of enhancement	Benefits will be short term.	Benefits to a few species, associated habitat, and resources in the local environment only.	Minor benefits to species, habitat, and natural resources that may provide minor opportunities for sustainability.	Benefits will pertain to species, habitats and natural resources that are not , sensitive or rare.

Table 20-4: Enhancement Criteria (Positive Social and Health Impacts)				
Ranking	**Duration**	**Extent**	**Degree of Change**	**Focus/ Sensitivity**
High level of enhancement	Benefits will be lasting and sustained over the long term i.e.: more than 7 years	Benefits throughout the local community and beyond to State/ National level.	Direct benefits to individuals and communities will provide significant opportunities for leveraging secondary benefits and significantly improving livelihoods for themselves and others	Benefits will pertain to vulnerable groups and those that would have otherwise have been 'losers' as a result of the project.

Table 20-4: Enhancement Criteria (Positive Social and Health Impacts)				
Ranking	**Duration**	**Extent**	**Degree of Change**	**Focus/ Sensitivity**
Medium level of enhancement	Benefits will be felt for a medium period of time (1 to 7 years) or be intermittent over the longer term	Benefits to many individuals and households in the local community and beyond	Moderate benefits to individuals and communities which will provide some opportunities for furthering themselves and improving livelihoods	Benefits will possibly pertain to vulnerable groups and those that might have been 'losers' from the project
Low level of enhancement	Benefits will be short-term (up to a year)	Benefits to a few individuals and households either in the local area and/or further afield.	Some benefits to individuals and communities, potentially improving opportunities for furthering themselves and improving livelihoods	Benefits will not pertain to vulnerable groups and will only benefit those that would have otherwise benefited from the project.

Likelihood Criteria

Likelihood categories were divided into the following:

- Low likelihood - Not known in the mining sector;
- Medium likelihood - Has occurred in mining projects; and
- High likelihood – high likelihood of the event occurring in mining projects.

Determining the Rating

The overall rating of the impacts was determined by using the following matrix Table 20-5.

Table 20-5: Rating Matrix			
Impact Rating	**Likelihood**		
Severity/ Enhancement	**Low**	**Medium**	**High**
High Level of Enhancement	**Moderate**	**Major**	**Major**
Medium Level of Enhancement	**Minor**	**Moderate**	**Major**
Low Level of Enhancement	**Minor**	**Minor**	**Moderate**
Low Severity	**Minor**	**Minor**	**Moderate**
Medium Severity	**Minor**	**Moderate**	**Major**
High Severity	**Moderate**	**Major**	**Major**

It should be noted that these matrices act as a guide and there may be situations where their rigid application is inappropriate and where stakeholder perceptions and feedback have a significant role to play. For specific impacts where this is the case, the rating is clearly explained in the evaluation of the impact.

20.15 Limitations of the ESIA

The ESIA has the following limitations:

Timing of the study- the ESIA was undertaken while the Karouni Gold Project PFS was being prepared. The Karouni Gold Project design details and facility location are expected to evolve and change during the PFS. This draft ESIA is not consequently based on definitive positions for several of the proposed site facilities including waste disposal and the water management pond.

The analysis and findings of this draft ESIA are therefore based on the status of the Karouni Gold Project description as of 30 September 2013. Any changes or additions to the Karouni Gold Project design and plans subsequent to September 2013 would not necessarily be covered under this draft ESIA or associated management plans.

Necessary government approvals are in the process of being finalized. All required documentation has been submitted and is following the governmental internal process. The Minerals Agreement, which is the definitive agreement allowing for project exploitation, tax and royalty issues, importation allowances, physical and economic parameters for the extraction and sale of minerals, is in the final stages of its approval process.

21. Capital and Operating Cost Estimates

21.1 Capital Costs

The estimated capital cost to construct and commission the Karouni Gold Project is US$73.3 million, including a contingency of US$4.7 million. A further US$11.3 million is required for pre-production mining, including waste pre-strip and building a high grade stockpile ahead of treatment through the processing plant. Sustaining capital over the life of mine is estimated at US$2.4 million and a further US$4.0 million for working capital spares.

These costs include all procurement, delivery and construction direct and indirect costs. The estimates are based on the PFS design and are considered to have an accuracy of +10 to -10%. Table 21-1 shows the capital cost summary.

Table 21-1: Capital Cost Summary

Cost	US$M
Drilling & Metallurgical Testwork	0.05
Design & Engineering	2.03
Concrete and Road Construction	3.41
Power Plant	4.69
Plant & Site Infrastructure	31.08
Mining Equipment – Open Cut	9.44
Dams & Water	5.19
Salaries, Oncosts & Overheads	8.19
Unallocated Sundry Costs	3.01
Start-up Consumables	1.50
Subtotal:	68.59
Contingency:	4.73
Total Cost to Construct & Commission:	**US$73.32**

21.2 Operating Costs

Operating costs have been developed from first principles following a review of local market conditions and are shown in Table 21-2 below.

Table 21-2: Operating Cost Summary

Costs	Life of Mine (US$/oz)
Mining (Ore and Waste)	225
Processing	203
Administration	38
Refining	14
C1 Cash Costs	**480**
Royalties	113
Sustaining Capital	8
All in Sustaining Cash Costs	**US$602/oz**

21.3 Taxes & Royalties

The payment of gross production royalties are provided for by the Guyana Mining Act and the amount of royalty to be paid is prescribed by the Minister. As such royalties may be varied from time to time. Based on

currently prescribed royalties and recent mineral agreements, royalties on gold production have been assumed as follows:

- 8% if the gold price is above US$1,000 per ounce;
- 5% at a gold price of US$1,000 and less; and
- Additional 2% vendor royalties on some parts of the Karouni Gold Project.

The corporate tax rate in Guyana for gold companies is 30%.

22. Economic Analysis

Table 22-1 below shows the Pre and Post Tax NPV for the Karouni Gold Project for various gold prices.

Table 22-1: Pre and Post Tax NPV for the Project

Gold Price ($US/oz)	$800	$1,000	$1,200	$1,250	$1,400	$1,600
Pre-tax NPV at 6% ($US)	-$16,652,833	$33,871,579	$74,665,186	$86,890,839	$123,567,798	$172,470,409
IRR	N/A	26.19%	49.40%	56.17%	76.12%	102.11%
Gold Price ($US/oz)	**$800**	**$1,000**	**$1,200**	**$1,250**	**$1,400**	**$1,600**
Post-tax NPV at 6% ($US)	-$16,652,833	$33,871,579	$63,285,725	$71,959,221	$97,901,012	$132,361,257
IRR	N/A	26.19%	44.78%	50.17%	66.10%	86.71%

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Such Mineral Resources have not been considered in this economic assessment.

SENSITIVITY ANALYSIS

Table 22-2 summarises the Karouni Gold Project's sensitivity to head grade, recovery, operating cost, capital cost and gold price. Table 22-3 summarises the Project cash flow forecast.

Table 22-2: Sensitivity Analysis

Post-tax NPV (US$M)	-20%	-10%	Base	+10%	+20%
Head Grade	25.47	50.07	71.96	93.69	115.35
Recovery (1)	N/A	67.34	71.96	76.57	N/A
Operating Costs	90.01	80.99	71.96	62.90	53.78
Capital Costs	89.47	80.72	71.96	63.20	54.45
Gold Price	33.87	50.18	71.96	93.58	115.14

(1) Sensitivity analysis for recovery is calculated at 92% for the downside and 96% for upside.

The sensitivity analysis indicates that the Project is economically robust with reasonable variations to major cost and revenue factors. For example a 20% reduction in gold price from the base case of $1,250/oz still results in an NPV of $33.87m.

Table 22-3: Project Cash Flow Forecast

Period Ending			31-Dec-14	31-Dec-15	31-Dec-16	31-Dec-17	Total
Mining							
	Open Pit						
	Total Movement	t	1,828,000	9,080,000	9,026,000	3,514,000	23,447,000
	Waste Tonnes Mined	t	1,803,000	8,206,000	8,169,000	2,674,000	20,851,000
	Ore Tonnes Mined	t	28,000	889,000	857,000	840,000	2,614,000
	Grade Mined	g/t	5.43	3.22	4.43	3.85	3.84
Processing							
	Tonnes Processed	t	-	738,000	1,003,000	873,000	2,614,000
	Grade Processed	g/t	-	3.83	3.94	3.74	3.84
	Contained oz	oz	-	91,000	127,000	105,000	323,000
	Recovery	%	-	94.00%	94.00%	94.00%	94.00%
	Recovered oz	oz	-	86,000	120,000	99,000	304,000
Revenue							
	Spot Revenue	US$	-	106,733,000	149,218,000	123,456,000	379,407,000
	Royalties	US$	-	9,543,000	14,789,000	10,055,000	34,387,000
	Refining	US$	-	1,085,000	1,718,000	1,421,000	4,224,000
Net Revenue		US$	-	96,105,000	132,711,000	111,980,000	340,795,000
Costs							
	Mining	US$	-	25,779,000	30,687,000	11,946,000	68,412,000
	Processing	US$	-	17,835,000	23,156,000	20,545,000	61,537,000
	General Administration	US$	-	3,533,000	4,240,000	3,887,000	11,660,000
	Capital Expenditure	US$	72,942,000	15,969,000	750,000	1,375,000	91,036,000
Total Project Cost		US$	72,942,000	63,117,000	58,833,000	37,753,000	232,645,000
Project Cash Flow Before Tax			(72,942,000)	32,988,000	73,877,000	74,227,000	108,150,000
	Corporate Tax	US$	-	-	2,874,000	14,721,000	17,595,000
Project Cash Flow		US$	(72,942,000)	32,988,000	71,003,000	59,506,000	90,555,000
Cumulative Cash Flow		US$	(72,942,000)	(39,954,000)	31,049,000	90,555,000	

23. Adjacent Properties

The Omai Gold Mine, which was previously owned and operated by Cambior Inc., is located 160km south of the capital Georgetown and 35km east of the Karouni Gold Project.

Between 1990 and 2004, Denison Mines Ltd., Cambior Inc., and subsequently Omai Gold Mines Ltd completed exploration consisting of stream sediment geochemistry, profile and grid auger drilling and MMI (Mobile Metal Ion) geochemical sampling in several stages around and fringing the Wenot and Fennel Pits and extending to the east. Auger drilling indicated several gold anomalies attributed to the Wenot Shear. Cambior Inc., (Val d'Or, Quebec) a Canada-based gold mining company, replaced Placer Dome at the Omai project and, as project operator, completed a "bankable feasibility study" and subsequently secured a mining license. A total of 394 diamond drill holes (60,486m) were drilled in the Fennel area up to 2006. Omai Gold Mines Ltd conducted open pit mining from two separate deposits (Wenot and Fennel) which commenced in March, 1994 and ceased in September, 2005. Between 1993 and 2005, the Omai open pit mining and CIL processing operation produced 3.7Moz gold (Cambior Inc. Annual Report, 2005). The mill production was sourced from mineralised material grading approximately 1.3g/t under the management of Cambior Inc. The operation comprised two adjacent deposits, being the Fennell and Wenot pits. The Fennell Pit was the largest contributor, generating some 49Mt of mill feed at 1.5g/t Au for 2.37Moz.

Mineralisation at the Fennel Deposit comprised stacked sub-horizontal vein sets developed within a broad, steeply dipping quartz diorite intrusive. The Fennel Deposit was mined to a depth of 250m before bottoming out on a 180m thick unmineralised dolerite dyke.

Mineralisation at the Wenot Deposit was hosted by a succession of basaltic and andesitic volcanics, underlain by a polymictic conglomerate and overlain by sediments variously comprising shales, siltstones and

greywackes. The volcanics are intruded by a conformable quartz-feldspar porphyry dyke and subsidiary rhyolite dykes. Mineralisation is spatially associated with a 100m wide zone of ductile deformation termed the Wenot Shear, along which the quartz-feldspar porphyry intrudes. The dyke has been dated at 2120 ± 2Ma, and its appearance and position is highly consistent with that observed at the Hicks Zone at the Karouni Project.

The majority of deformation, which appears to post-date the dykes, has induced several well-developed zones of brittle fracturing and mineralised quartz veining. These zones are preferentially developed with the dykes and andesitic volcanics immediately adjacent to the Wenot Shear. Mineralisation was mined over a strike length of 2,000m and the mineralised zone was up to 200m in width.

The Wenot Deposit was depleted in April 2002 and the last blast in the Fennell Deposit took place in October 2004. Processing of low-grade hard rock from the stockpiles accumulated during the initial years of production ceased in 2005. Over the course of its life, the mine produced over 3.7 million ounces of gold, 1.6 million ounces in excess of the production originally forecasted by the feasibility study.

On July 25, 2011, Mahdia Gold Corp (Mahdia) submitted an application to the Guyana Geology & Mines Commission for a prospecting license over the former Omai Mine area, a concession of 7,244 acres located in the central part of Guyana. On January 25, 2012, the government of Guyana granted Mahdia Gold Corp. (Guyana) Inc. (Mahdia Gold), a wholly-owned subsidiary of Mahdia, the Prospecting License (GS14: M-39) PL 01/2012 (the Prospecting License, or PL), covering an area of 7,244 acres (2,931.54 hectares).

The information included within this section and relating to adjacent properties is separate to the property that is the subject of the technical report. The adjacent property (Omai Gold Mine) and the property of this technical report (Karouni Gold Project) are separated by 15 kilometres.

Information relating to the Omai Gold Mine property was sourced from publically released documents by mine operator Cambior Inc. and from final exploration reports on the Omai licences lodged with the Guyana Geology and Mines Survey. These exploration reports are available to the public through the CN Barron Library.

The Qualified Person has been unable to verify the information and that the information is not necessarily indicative of the mineralisation on the Karouni Gold Project that is the subject of this report.

24. Other Relevant Data and Information

There is no other data or information regarding the Karouni Gold Project relevant to this report that has not been disclosed elsewhere in the document.

25. Interpretation and Conclusions

The Karouni Gold Project covers approximately 40km along strike of the mineralised Omai-Hicks-Smarts-Whitehall (Kaburi) Corridor. A summary of the key features of the Karouni Gold Project are:

- Hicks Deposit – Troy has conducted extensional RC and Diamond Core drilling along strike and at depth to an historic near-surface resource defined for the Hicks Zone in 1996 and 1998. Drilling has combined with the historic results to allow the reporting a significant low to medium grade Inferred Mineral Resource.

- Smarts Deposit – artisanal bedrock gold anomaly where Azimuth discovered significant primary gold mineralisation during an RC drilling program in 2011. Follow-up RC and diamond core drilling has defined a significant medium to high grade Indicated and Inferred Mineral Resource.

- Larken Zone - a small artisanal open pit in saprolite, approximately 50m wide by 120m in strike and less than 15m deep, that was mined in the mid to late 1990s. The pit lies approximately 700m to the northeast of the Hicks Deposit and is likely to be a sub parallel structure to the one that hosts both Hicks and Smarts Deposits.

- Numerous alluvial workings along strike of the mineralised corridor.

Drilling has confirmed the continuation of mineralisation underneath partial sand cover at the Hicks and Smarts Deposits and semi-continuous gold mineralisation has been identified along shear zone with individual shears averaging 15m in width. Weathered material occurs to an average depth of 35m and the mineralisation is open at depth below oxide material and along strike.

Drilling completed since 2011 at the Hicks Deposit has confirmed historical drilling and trenching information where previous diamond drilling had defined a persistent zone of mineralisation extending at least 2,500m along strike. Troy has drilled the overlying weathered zones to replace trenching and pit information and is completing further infill drilling to confirm the continuity of mineralisation based on historical diamond drilling at depth.

The Smarts Deposit is located at a shallow artisanal open pit in saprolite approximately 50m wide by 120m in strike. The pit lies approximately 4km to the northwest of the Hicks Deposit and is likely on strike with a lateral offset from the Hicks Deposit or a parallel zone to the Hicks Deposit. Visually the mineralisation is recognised as iron-oxide stained quartz veins within the shear zone. Assay results from the drilling have extended the known strike of mineralisation to 2,800m at the Hicks Deposit and 2,900m at the Smarts Deposit. The mineralised shear zone remains open along strike, at depth and across strike to a maximum likely width of 200m. Recent drilling campaigns completed by Troy have aided the constraint of the mineralisation.

Comprehensive drilling and QA/QC protocols have been employed by Azimuth and now by Troy and the recent RC and Diamond Core data is considered acceptable for use in this Resource estimation.

The Hicks and Smarts Mineral Resources represent geologically well-defined zones of low to medium grade (Hicks) and medium to high grade (Smarts) gold mineralisation. The zones are orientated in a north-west direction and significant enrichment or depletion within oxide material has not occurred. The mineralised domains show variation in thickness and geometry along strike, however, the drill density has allowed the delineation of coherent bodies of mineralisation.

The drill spacing is sufficient to allow the grade intersections to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the thickness and grade of the domains. The authors consider that geological and mineralisation continuity has been demonstrated with sufficient confidence to allow the Hicks and Smarts Deposits to be classified as Inferred and/or Indicated Mineral Resource. Further drilling is recommended to increase confidence in the Mineral Resource.

The Hicks and Smarts Deposits appears to have reasonable potential for profitable exploitation by a medium to large scale open pit and possibly underground mining.

The Whitehall (Kaburi) Target is a multiple sheeted vein and stockworks occurrence over a collective width of 400m and strike length of at least 500m represents a high priority target for immediate surface sampling and further. An RC and Diamond Core campaign conducted during 2012 confirmed the presence of lower grade continuous gold mineralisation.

Elsewhere along the Smarts - Hicks- Whitehall (Kaburi) mineralised trend, much of the remaining basement potential is obscured by superficial cover of the barren White Sand Formation. It is considered plausible that extensions to known mineralisation or mineralised repetitions along this trend exist and that exploration should investigate this area. Prospective basement structures obscured by the White Sand Formation may be defined and prioritised via ground magnetic surveys which would infill the airborne survey. More prospective corridors could then be followed up by pattern Aircore drilling, with the objective being to sample the interface between the cover sequence and basement, and normalising the results for iron content.

Many of the Karouni Gold Project tenements are almost entirely veneered by White Sand Formation cover; however the area is essentially surrounded by alluvial deposits derived from the restricted basement exposures. It is highly likely that additional basement mineralisation exists beneath this cover and exploration using the above approach is equally applicable and justified.

At present, Troy is continuing to explore the Karouni Gold Project, focusing on the shear zone that hosts the Smarts and Hicks Deposits as well as shear zones parallel to this structure.

During 2014, systematic field evaluations consisting of prospecting, geological mapping, geophysical surveys and geochemical sampling are planned to rank the various regional targets for drilling within the Karouni Gold Project.

26. Recommendations

26.1 Exploration Strategy

The Karouni Gold Project worthy of further exploration and notes the project forms the newest focus of exploration for Troy. An initial Mineral Resource estimate at the Hicks and Smarts Deposits was reported in April 2012 with a subsequent Resource update in January 2013 and most recently in July 2014. Troy intends to conduct further staged exploration strategy over the Karouni Gold Project as discussed below:

- In 2011, a significant amount of airborne radiometric and magnetic surveys were flown over the area. Interpretation and further processing of the data from that exercise will provide better definition of geological units and structural features in the underlying basement rock to better define targets.
- Further mapping, prospecting and regional auger drilling to investigate targeted areas will be undertaken. Exploration will extend along strike from drilled out areas and mapping will be carried out on areas of interest.
- Detailed Ground Magnetic and Dipole-Dipole Induced Polarization surveys will be completed over the Smarts – Hicks Trend in attempt to characterize the mineralisation and develop a geophysical "thumbprint" for gold mineralisation that is outcropping and/or under shallow sand cover.
- Detail soil auger sampling of both the Smarts and Hicks Deposits is planned with samples analysed for a multi-element suite of elements in an attempt to identify pathfinder elements associated with the gold mineralisation.
- Scout drilling using Diamond Core (DC) or possibly Aircore (AC) and Reverse Circulation (RC) rigs aimed at targets within the Karouni Gold Project area that have been identified through ground investigations (mapping and geochemical sampling) and geophysical techniques (airborne Magnetics / Radiometrics) has been conducted simultaneously with the infill drilling at the Smarts and Hicks Deposits. Further scout drilling will focus on identifying new structures and mineralised corridors as well as targeting extensions to known mineralised zones.
- Infill drilling will be carried out at the Smarts Deeps (Underground) and the Hicks Deposit areas to further improve geological confidence and scale of these deposits and to build upon the Mineral Resource

Troy intends to continue a systematic exploration approach with progressive work programs to not only grow its resource base, but also to identify other future targets in the project area. The magnetic, radiometric and LiDAR surveys should be particularly effective at focusing exploration on the identified (and any new) targets. The survey will also be useful in defining the continuation of the mineralised corridor under the barren White Sands Formation. Surface exploration would be scheduled to fit in with the exploration field season. The proposed strategy builds on available data and geological interpretation, and should continue to prove effective.

26.2 Recommendations – Exploration Program

Troy has a continuous improvement program in place for exploration campaigns and these recommendations form a part of that strategy. These recommendations are outlined for the Karouni Gold Project. Recommendations are listed below as:

- Infill drilling at Smarts Deeps (Underground) Resource to upgrade Inferred Mineral Resources (7,000m) to at least Indicated to enable the estimation of updated Mineral Resources and Mineral Reserves. **Estimated Cost: USD$2,500,000;**
- Detailed independent review of regional and brownfields area prospectivity including target identification and ranking. **Estimated Cost: USD$70,000;**
- Brownfields area targets will be the focus ongoing field assessment consisting of geological mapping, prospecting and regional auger drilling to investigate targeted areas. Exploration will extend along strike from drilled out areas and mapping will be carried out on areas of interest. **Estimated Cost:USD:$500,000;**
- Detailed Ground Magnetic and Dipole-Dipole Induced Polarization surveys will be completed over the Smarts – Hicks Trend in attempt to characterize the mineralisation and develop a geophysical "thumbprint" for gold mineralisation that is outcropping and/or under shallow sand cover. **Estimated Cost: USD$350,000**

- Detail soil auger sampling of both the Smarts and Hicks Deposits is planned with samples analysed for a multi-element suite of elements in an attempt to identify pathfinder elements associated with the gold mineralisation. **Estimated Cost: USD$30,000**
- Scout drilling using Diamond Core (DC) or possibly Aircore (AC) and Reverse Circulation (RC) rigs aimed at targets within the Karouni Gold Project area that have been identified through ground investigations (mapping and geochemical sampling) and geophysical techniques (airborne Magnetics / Radiometrics) has been conducted simultaneously with the infill drilling at Smarts and Hicks. Further scout drilling will focus on identifying new structures and mineralised corridors as well as targeting extensions to known mineralised zones. **Estimated Cost: USD$7,500,000**
- Key elements to be considered in planning and execution of on-going exploration work include;
    - All samples are collected by a properly maintained riffle splitter or cone splitter (including field duplicates).
    - Zones of mineralisation intercept drill holes at intervals without assay values. Twinned holes are required to confirm the position of the zones. Twinned holes will also aid in the validation of assay results, specifically at the Hicks Deposit.
    - The frequency of any faulting and fault geometry has not been defined in the Mineral Resource estimate. This may create potential for tonnage and overall geometry variations in the model.
    - A small, close-space “Grade Control” program should be carried out as the project develops. The drill program would investigate short-range grade continuity as a part of estimation studies and would ideally be located in areas with potential for open-pit mining.
    - Additional drilling in the resource area may define new zones of mineralisation both down dip and along strike.
    - Sparse drilling downdip has been interpreted into continuous zones of mineralisation in some areas. Infill drilling may modify this interpretation.
    - Comprehensive drilling and QA/QC protocols have been employed, however the authors strongly recommend consistent monitoring of QA/QC data and the implementation of inter-laboratory testing to further enhance the confidence in assay data quality.

26.3 Recommendation - Development

The positive results of the PFS warrant the commencement of mine development of the Karouni Gold Project.

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28. Glossary of Technical Terms

-150# Mesh: Material fraction passing a screen size equivalent to approximately 150 apertures per inch, equating to an aperture size of 105µm.

-80# Mesh: Material fraction passing a screen size equivalent to approximately 80 apertures per inch, equating to an aperture size of 180µm.

AAS - Atomic Absorption Spectrophotometry: a method for analysing elemental concentrations in a sample that has been dissolved in acid.

Ag: The chemical abbreviation for the element silver.

Airborne Radiometric Survey: Data collected from a survey undertaken by helicopter or fixed-wing aircraft for the purpose of recording natural radioactivity emitted by rocks.

Aircore: Drilling method employing a drill bit which yields sample material which is delivered to the surface inside the rod string by compressed air.

Alluvial Deposits: Silt, sand and gravel material, transported and deposited by rivers.

Amphibolite Facies: An assemblage of minerals formed at moderate to high temperatures during regional metamorphism.

Andesite / Andesitic: An intermediate volcanic rock composed of andesine and one or more mafic minerals.

Anticline: A fold structure in rocks in which strata dip in opposite directions away from the central axis.

Archaean: The oldest rocks of the Precambrian era, older than about 2,500 million years before the present.

Arkose: A poorly sorted sedimentary rock of granular texture composed essentially of quartz and feldspar.

Arsenopyrite: Mineral -an iron, arsenic sulphide, FeAsS.

Artisanal: Referring to a skilled manual worker or a craftsperson.

Basaltic lavas: Fine to medium grained volcanic flow rocks composed primarily of ferromagnesian minerals and plagioclase feldspars.

Bauxite: An aluminium-bearing rock-type composed of hydrated aluminium oxides.

Bi: The chemical abbreviation for the element bismuth.

BLEG - Bulk Leach Extractable Gold: An analytical method for accurately determining low levels of gold.

Breccia: Rock comprising angular fragments enclosed in a matrix.

Cainozoic: The youngest Era of geologic time spanning between 65 million years ago and the present.

Carbonaceous: Containing carbon, often in the form of graphite.

Chalcopyrite: A copper iron sulphide mineral with a copper to iron ratio of 1.

Chlorite: A green coloured hydrated aluminium-iron-magnesium silicate mineral (mica) common in metamorphic rocks.

Cu: The chemical abbreviation for the element copper.

Deformation: A general term for the process of folding, faulting, shearing, compression or extension of rocks as a result of stress.

Dendritic: Branching like a tree, a form of crystalline gold.

Dextral: Differential lateral movement across a fault such that the block away from the observer appears to have been displaced to the right.

Diabase: A mafic intrusive rock equivalent to a diorite.

Dilution: The inclusion of waste material into ore during the mining process, normally expressed as a percentage of ore tonnes mined ie 10%.

Dolerite: A medium grained mafic intrusive igneous rock composed mostly of pyroxenes and sodium-calcium feldspar.

Dyke: A tabular body of intrusive igneous rock, crosscutting the host strata at an oblique angle.

Eluvial: Formed by the weathering of rock in place to a greater or less depth, with no significant transport of material.

En echelon: Repeating parallel, but offset, structures, veins or lenticular bodies.

Fault: A fracture or fracture zone in the earth's crust along which displacement of opposing sides has occurred.

Felsic: Light colour rocks containing an abundance of feldspars, feldspathoids and silica.

Felsic Volcanic: A volcanic extrusive rock which has a high proportion of silica, potassium and sodium and low iron and magnesium.

Fluviatile: Material transported and deposited in rivers and streams.

"g/t": Grams per tonne = 0.03215 troy ounces per metric tonne.

Gabbro: A fine to coarse grained, dark coloured, igneous rock composed mainly of calcic plagioclase, clinopyroxene and sometimes olivine.

Galena: A lead sulphide mineral.

GIS: Geographic information system.
Gneiss: A coarse grained, banded, high grade metamorphic rock.
Goethite: A hydrated iron oxide mineral.
Granite: A coarse-grained igneous rock containing mainly quartz and feldspar minerals and subordinate micas.
Granitoids: Intrusive rocks which include granites and granodiorites.
Granodiorite: A coarse-grained igneous rock containing mainly quartz and feldspar minerals with biotite and or hornblende.
Granulite: Metamorphic rocks formed by high temperature and/or pressure metamorphism.
Gravimetric: Measured by weight.
Greenschist Facies: A facies of metamorphic rock characterised by the development of the mineral chlorite.
Greenstone: Term commonly applied to low metamorphic grade rocks of basic composition and comprised of the minerals chlorite and amphibole. This term is commonly applied to Archaean rock sequences dominated by these rock types.
Greywacke: Sandstone composed dominantly of rock and mineral fragments.
Ground Magnetic Surveys: Surface based measurements of the earth's magnetic field intensity to locate concentrations of magnetic minerals.
Hematite: An iron oxide mineral with the chemical formula Fe_2O_3.
Holocene: The epoch that covers the last 10,000 years of time, and the later sub-division of the Cenozoic era and Quaternary period.
Hornfels: A fine grained dark rock formed by thermal metamorphism.
Hydrothermal: Pertaining to hot aqueous solutions having temperatures up to 400°C. The solutions transport and deposit metals and chemicals in solution.
ICP - Inductively Coupled Plasma: Analytical method for determining concentrations of elements.
Illmenite: An iron titanium oxide mineral.
Intrusive: Refers to a magma which has solidified before reaching the earth's surface.
"kg": Kilogram, a standard metric unit for weight.
"kt": One thousand metric tonnes.
Lithological: Pertaining to the character of the rock.
Mafic: Rocks composed dominantly of magnesium, iron and calcium-rich rock-forming silicates.
Magnetite: An iron oxide mineral with the chemical formula Fe_3O_4.
Mesothermal: Mineral deposits formed (precipitated) at moderate temperatures, characteristically between 200ºC and 350ºC.
Mesozoic: Era of geological time between 245 million years before present and 65 million years ago.
Metamorphism: The mineralogical, chemical and structural changes imposed on a solid rock which has been subjected to conditions of heat and or pressure which are different to those under which it formed.
Mineralisation: The concentration of metals and their compounds in rocks, and the processes involved therein.
Monzonite: A granular plutonic rock containing approximately equal amounts of orthoclase and plagioclase and thus, intermediate between syenite and diorite.
"Mt": One million metric tonnes
Ore Loss: The amount of ore that is not recovered, or is lost, during the mining process, normally expressed as a percentage of total ore tonnes ie 95%.
Orogeny: The process by which structures in fold belt mountains formed including folding, faulting, metamorphism and intrusion of igneous rocks.
Overburden: The soil and other material that lies above a specific geologic feature.
"oz": Ounces.
Paleo-placer: An ancient alluvial deposit commonly hosting gold mineralisation.
Palaeoproterozoic: The period of the earth evolution between 2,500 million years and 1,600 million years ago.
Palaeozoic: An Era of geologic time spanning between 570 and 225 million years ago.
Pegmatite: A very coarse grained intrusive felsic igneous rock which commonly occurs in dyke-like bodies containing lithium-boron-fluorine-rare earth bearing minerals.
Permian: The youngest period of geologic time in the Palaeozoic, spanning between 280 and 225 million years ago.
Phyllite: A metamorphic rock intermediate in grade between a slate and a schist with mineralogy usually dominated by fine grained micas and chlorite.
Pleistocene: The current period of geologic time, from approximately 1.8 million years to the present.

Porphyry: An igneous rock of any composition which contains scattered large crystals of one or more minerals set in a fine grained groundmass.
"ppb" -Parts Per Billion: A measure of low level concentration.
"ppm" - Parts Per Million: A quantitative equivalent of grams per tonne (g/t).
Preg-robbing: Term used to describe the loss of gold by the re-adsorption of dissolved gold cyanide onto carbonaceous ore components. This robbing of gold from the pregnant liquor (i.e. the gold cyanide solution) can be a significant cause of low gold recovery in conventional carbon-in-pulp processes.
Proterozoic: An era of geological time spanning the period from 2,500 million years to 570 million years before present.
Protolith: The parent rock prior to alteration or metamorphism.
Pyrite: An iron sulphide mineral with the chemical formula FeS2
Quartz: A mineral composed of silicon dioxide, SiO2.
Quartz Diorite: Igneous rock consisting of an aggregate of the minerals plagioclase feldspar, hornblende and pyroxene, and containing in excess of 10% quartz by volume.
Quaternary: The younger of the two geologic periods in the Cenozoic era, between 1.8 million years before present and the present day.
RAB – Rotary Air Blast: An open-hole Rotary Air Blast drilling technique.
Reverse Circulation (RC) Drilling: Drilling method employing a percussive action to break the rock, and in which sample material is delivered to the surface inside the rod string by compressed air.
Recent: The later of the two geologic epochs comprised in the Quaternary period, in the classification generally used; Holocene. Also, the deposits formed during that epoch. Recent includes the geologic time and deposits from the close of the Pleistocene (glacial) epoch until and including the present.
Rhyolite: Fine-grained felsic igneous volcanic rock containing a high proportion of silica and feldspar, similar in composition to granite.
Saprolite: A part of the rock weathering profile where the mineralogy is dominated by clay minerals and silica.
Schist: A strongly foliated crystalline metamorphic rock formed by dynamic metamorphism.
Sediment: A rock formed of particles deposited from suspension and transport in water, wind or ice.
Sericite: A mica mineral very common as an alteration product in metamorphic and hydrothermally altered rocks.
Serpentinised: The conversion of the ferromagnesian minerals or rocks into aggregates of the serpentine minerals.
Shale: A fine grained, laminated sedimentary rock formed from clay, mud and silt.
Siliceous: Containing silica.
Sills: Sheets of igneous rock which is flat lying or has intruded parallel to stratigraphy.
Siltstone: A rock intermediate in character between a shale and a sandstone that is composed of silt sized grains.
Sinistral: Differential lateral movement across a fault such that the block away from the observer appears to have been displaced to the left.
Sluicing: Methodology for concentrating heavy minerals by washing unconsolidated material through a box (sluice) equipped with riffles that trap the heavier minerals on the floor of the box.
Sphalerite: A zinc and iron sulphide mineral
Stock: A small intrusive mass of igneous rock, usually possessing a circular or elliptical shape in map view.
Stockworks: An intersecting network of planar veins and veinlets
Syenite: An intrusive igneous rock composed essentially of alkali feldspar and little or no quartz and ferromagnesian minerals.
Syntectonic: Occurring or forming at the same time as deformation
"t ": Tonne.
Tectonism: The processes of deformation in the earth's crust that produce its continents, ocean basins, mountains, and major folds and faults.
Tourmaline*:* aluminium silicate mineral containing boron.
Ultramafic*:* Igneous rocks consisting essentially of ferromagnesian minerals with trace quartz and feldspar.
Volcanic*:* Pertaining to igneous material poured out on the surface of the earth in a molten state and to fragmental material of all sizes erupted from volcanic vents.