OCEANAGOLD

Technical Report for the MACRAES PROJECT

Located in the province of Otago, NEW ZEALAND

Prepared by
OceanaGold Corporation
and Oceana Gold (New Zealand) Limited

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February 12th, 2010

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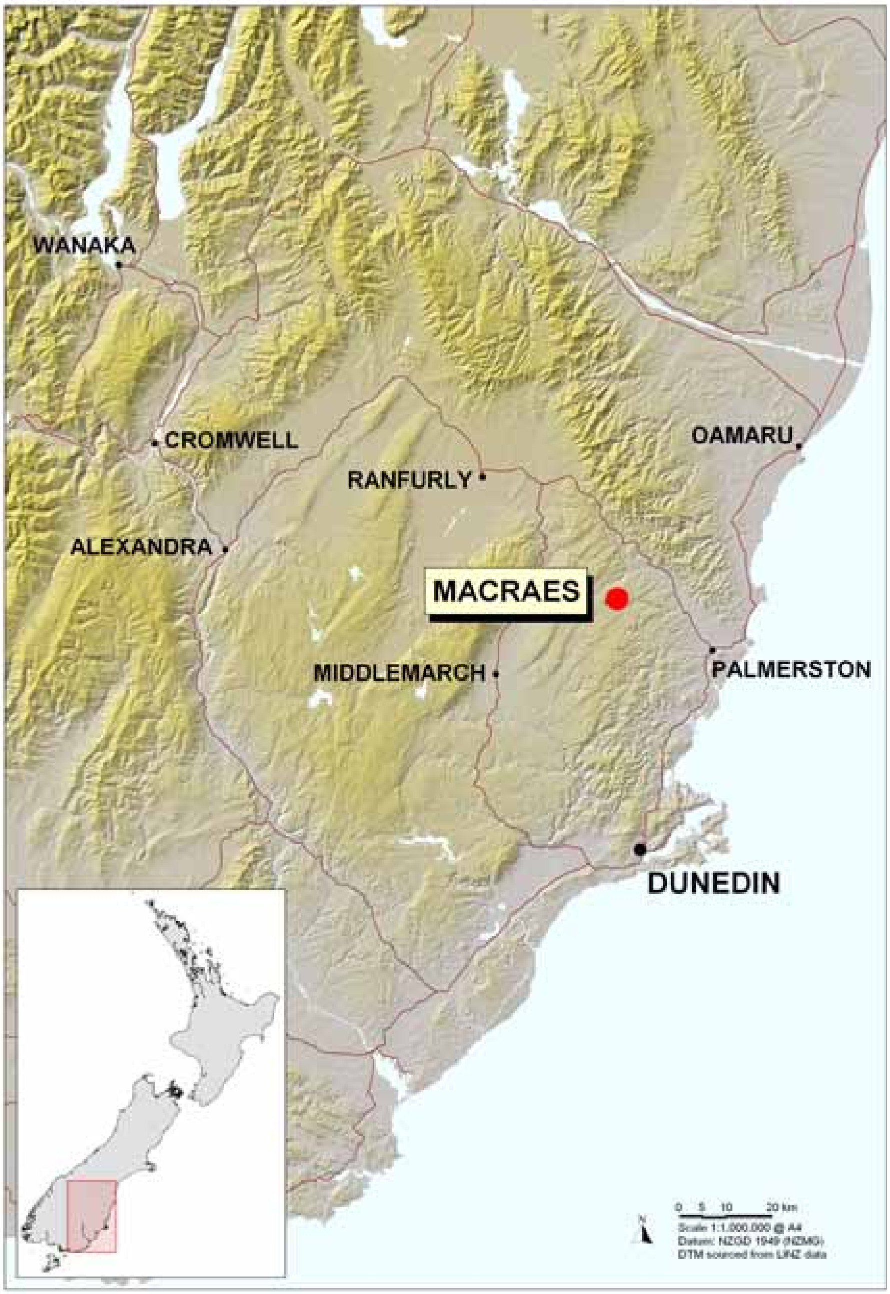
1 SUMMARY

1.1 Description and Location of Property

The Macraes Project is located approximately 30 kilometres (km) to the northwest of Palmerston in the Otago Region of the South Island, New Zealand (NZ). The mining operation occurs 1-2km to the east of the Macraes Flat township and is predominantly surrounded by farmland.

The Macraes mining and exploration tenements cover a contiguous area of 27,492 hectares.

Figure 1.1: General Location of the Macraes Project



1.2 Ownership

The Macraes Project is controlled by OceanaGold Corporation through its wholly owned subsidiary Oceana Gold (New Zealand) Limited (“Oceana”).

1.3 Geology and Mineralization

The Macraes gold deposits are located within a low-angle (~15-20°) shear zone, the Hyde Macraes Shear Zone (HMSZ), which has been traced for at least 30km along strike. The mine is developed within this regionally continuous shallowly east dipping structure. The HMSZ consists of variably altered, deformed, and mineralized schist up to 150m thick, known as the Intrashear Schist. The thickest part of the shear zone consists of several mineralized zones stacked on metre-thick shears. These shears have ductile deformation textures overprinted by cataclasis (Craw et al., 1999). The Hangingwall shear can be up to 25m thick and is commonly darker coloured due to fine grained graphite and sheared sulphide minerals (McKeag et al., 1989).

There is a strong empirical correlation between gold, arsenic, scheelite, silicification and strain intensity within the HMSZ. Gold-scheelite-pyrite-arsenopyrite mineralization is associated with replacement and fissure quartz veins within D4 post-metamorphic shear zones. Shear parallel quartz veins and cataclastic shears contain the highest gold and scheelite grades (Lee et al. 1989).

The following four types of mineralization occur within the HMSZ at Macraes (Mitchell et al., 2006):

1) Mineralized schist. This style of mineralization involved hydrothermal replacement of schist minerals with sulphides and microcrystalline quartz. Mineralization was accompanied by only minor deformation.

2) Black sheared schist. This type of schist is pervaded by cm to mm scale anastamosing fine graphite and sulphide bearing microshears. This type of mineralization is typically proximal to the Hangingwall Shear. Scheelite mineralization occurs in the silicified cataclastic shears.

3) Shear-parallel quartz veins. These veins lie within and/or adjacent to the black sheared schist, and have generally been deformed with the associated shears. The veins locally cross-cut the foliation in the host schist at low to moderate angles. Veins are mainly massive quartz, with some internal lamination and localized brecciation. Sulphide minerals are scattered through the quartz, aligned along laminae and stylolitic seams. These veins range from 1cm to > 2m. Scheelite mineralization is associated with quartz veining in some areas.

4) Stockworks. These veins occur in localized swarms that are confined to the Intrashear Schist. Individual swarms range from c. 100 to 2000m^2 in area and consist of numerous (10 – 100) subparallel veins. Most of these veins formed subperpendicular to the shallow east dipping shear fabric of the Intrashear Schist. Stockwork veins are typically traceable for 1-5m vertically with most filling fractures that are 5 – 10cm thick, but can be up to 1m thick. Swarms of stockwork veins within the Intrashear Schist were lithologically controlled by the dimensions and locations of more competent pods of Intrashear Schist.

Gold is closely associated with pyrite and arsenopyrite in all of the above styles of mineralization. Rarely free gold up to 300µm occurs in quartz veins, but most gold is as 1-10µm scale blebs hosted in and near sulphide grains (Angus, 1993).

1.4 Exploration Concept

There is good potential for discovery at Macraes. The current exploration focus is two-pronged:

- for near-surface mineralisation at the northern end of the Macraes Line of Strike (MLOS) where little previous exploration has been conducted; and

- elsewhere for extensions down-dip of areas previously mined by open pit and underground methods.

1.5 Status of Exploration and Resources

Exploration at the Macraes Project will continue with the objective of discovering further open pit resources along the MLOS. A combination of field mapping and soil sampling will be used to define drill targets for near-surface mineralization.

A combination of drilling collared from both surface and underground locations will target areas of high underground potential.

Resource estimates for the Coronation, Deepdell, Golden Point, Round Hill, Frasers, Golden Bar and Taylors deposits comprising the Macraes Project have been generated by Oceana.

The reintroduction of the Round Hill, Southern Pit and the northern portion of the Innes Mills resources has resulted from increased gold prices. These resources were not included in the December 31, 2008 inventory.

The resource models were derived via geological and mineralization zone modelling of the individual deposits. Estimation involved the application of a variety of techniques including polygonal methods, Ordinary Kriging and Multiple Indicator Kriging. Technique selection was based on the quantity and spacing of available data, and the interpreted controls on, and styles of, mineralization under review.

The table below represents the Macraes Project Mineral Resource Statements as at December 31, 2009 reported in accordance with Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects of December 2005 (the Instrument) and the classifications adopted by Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Council in December 2005. Furthermore, the resource classification is also consistent with the Australasian Code for the Reporting of Mineral Resources and Ore Reserves of December 2004 (the Code) as prepared by the Joint Ore Reserves Committee (JORC) of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia. A detailed subdivision of these resources by deposit and material type, indicating the applicable cut-off grade in each case, is provided in section 17 of this report.

Table 1.1: Macraes Project Mineral Resource Statement as at December 31, 2009

Resource Cut-off	Resource Area	Measured		Indicated		Measured & Indicated			Inferred Resource		
		Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
0.5 g/t	Coronation	.	.	1.23	1.18	1.23	1.18	0.05	2.98	1.1	0.11
0.5 g/t	Deepdell	0.23	1.67	.	.	0.23	1.67	0.01	0.32	1.0	0.01
0.5 g/t	Golden Point	.	.	.	.	.	.	.	1.48	2.6	0.12
0.4 g/t	Round Hill	4.06	0.98	20.18	0.93	24.24	0.94	0.73	18.38	1.2	0.72
0.5 g/t	Southern Pit	0.52	0.90	2.90	0.85	3.41	0.85	0.09	.	.	.
0.5 g/t	Innes Mills	0.04	1.94	0.74	1.08	0.79	1.13	0.03	0.23	0.7	0.01
0.5 g/t	Frasers Pit	12.14	1.53	28.36	0.91	40.50	1.10	1.43	9.41	0.7	0.21
No cut-off	Frasers Underground P1 & P2	1.38	3.10	4.88	2.21	6.26	2.41	0.48	4.62	2.0	0.30
No cut-off	Frasers Underground Panel2 Deeps	.	.	0.35	3.56	0.35	3.56	0.04	0.54	3.7	0.06
0.5 g/t	Golden Bar	0.09	1.56	1.18	1.40	1.27	1.42	0.06	4.96	1.4	0.22
0.5 g/t	Taylors	.	.	0.28	1.50	0.28	1.50	0.01	0.41	1.1	0.01
0.5 g/t	Stockpiles	5.69	0.64	.	.	5.69	0.64	0.12	.	.	.
	Macraes Total	**24.15**	**1.30**	**60.09**	**1.06**	**84.25**	**1.13**	**3.05**	**43.34**	**1.3**	**1.77**

Mineral resources are inclusive of all ore reserves.

1.6 Development and Operations

The physical and financial projections presented in this report are based upon the Life of Mine Plan as at November, 2009 (LOMP09) with the subsequent inclusion of the proposed Southern Pit reserve. LOMP09 has been depleted for mining as at December 31, 2009.

The Macraes Project is the largest gold producing operation in NZ and has been in operation since 1990. To December 31, 2009, approximately 2.8 million ounces of gold have been produced. The operation consists of a large-scale surface mine, an underground mine and an adjacent process plant inclusive of an autoclave for pressure oxidation of the ore. Since 2007, flotation concentrate from the Reefton mine has been transported by rail and road to utilise surplus autoclave capacity.

The Frasers Stages 4C and 5 are the only open pit stages currently being mined, and supply approximately 4.8Mt of ore per annum, while the Frasers Underground (FRUG) mine supplies a further 0.9Mt of ore per annum. Stockpiles provide supplementary feed when required. The combined Macraes production for the twelve months, ended December 31, 2009 was 213koz.

The Macraes process plant is capable of treating approximately 5.6Mtpa of ore and incorporates a semi-autogenous grinding (SAG) mill, flotation circuit, autoclave for pressure oxidation of the concentrate, CIL plant and smelting facilities.

The current combined open pit, stockpile and underground reserves of 1.55Moz support a mine life at Macraes extending to 2017. The mine life at Macraes is expected to increase in subsequent Life of Mine (LOM) plans.

A breakdown of open pit, stockpile and underground reserves is shown in Table 1.2. The reserves are reported by category using a 0.5 g/t cut-off for open pit and a 1.90 g/t cut-off for underground as at December 31, 2009. These reserves are a subset of the resources tabulated in Table 1.1.

Table 1.2: Macraes Mineral Reserve Inventory as at December 31, 2009

Reserve Cut-Off Grade	Reserve Area	Proven		Probable		Total Reserve (Proven and Probable)			Resource Model
		Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au Moz	
0.5 g/t	Coronation	.	.	0.81	1.35	0.81	1.35	0.03	CO01A
0.5 g/t	Frasers Pit	10.32	1.58	14.98	1.00	25.30	1.24	1.01	FR07A
0.5 g/t	Southern Pit[1]	.	.	5.93	1.27	5.93	1.27	0.24	RHFR09
1.90 g/t	Frasers Under-ground P1 ,P2 & P2D	0.40	3.23	1.18	2.78	1.58	2.89	0.15	P1_0806, P2_0912 & P2D_0906
0.5 g/t	Stockpiles	5.69	0.64	.	.	5.69	0.64	0.12	
	Macraes Total	**16.42**	**1.30**	**22.89**	**1.17**	**39.31**	**1.23**	**1.55**	

Based on a gold price of US$800/ounce (NZ$1,333/ounce)

[1] Note: The Southern Pit reserve is a subset of Innes Mills, Southern Pit and Round Hill resources as tabulated in Table 1.1

1.7 Conclusions and Recommendations

1.7.1 Geology

The Macraes area is a mature exploration province and much of the near-surface, along-strike exploration potential has been tested. Good exploration potential remains at the Coronation prospect (located approximately 8km north of the current infrastructure) and areas immediately to the north. The areas to the south of Golden Bar also require further attention. Significant resource potential exists down dip/plunge of known open pits and drilling has been planned to test this.

The quality control database is incomplete so assessment of the data quality has been based on the available quality control database and reconciliation data. Based on this, the drilling data is considered to meet acceptable industry standards, subject to the qualifications below.

Reverse circulation (RC) percussion samples collected under wet drilling conditions remain in the exploration/resource database and represent a degree of risk. The factoring approach applied by Oceana to reduce the impact of the remaining wet RC percussion drilling is reasonable although it cannot account for local variability and down-hole contamination i.e. artificially extended ore zone widths.

The grade estimates have been constrained within suitable geological frameworks which are well established. Furthermore, available reconciliation data indicates the resource models represent robust estimates of metal and are generally acceptable estimators of tonnage and grade. Two deposits, Deepdell and Golden Bar, would require reinterpretation of the oxidation surfaces and extra allowances for dilution and ore loss, should cut-backs be considered. No cut-backs are planned for these deposits at this stage.

Oceana's mineral resource modelling process for the Frasers Open Pit is readily reproducible. In December 2006, Hellman and Schofield recommended that Oceana review the block support adjustments for Frasers open pit stockwork mineralization. The reconciliations have been reviewed on an ongoing basis since then.

Further infill drilling is planned from underground development to better define the local geological controls and the grade distribution at the FRUG.

1.7.2 Mining

Macraes is mined by a combination of conventional open pit and underground retreat uphole stope methods along the line of strike.

The open pit mining operation utilises hydraulic excavators and rear dump diesel trucks to extract both overburden and ore. Blasting requires relatively light powder factors compared with some other operations due to the comparatively weak and fractured rock mass. Ore is blasted in 7.5m high benches and excavated in three, nominally 2.5m high flitches. Waste is blasted in 15m benches and excavated in four flitches.

The underground retreat uphole stope mining operation utilises electro-hydraulic development jumbos, diesel load-haul-dump units, diesel haul trucks and a production drill rig to extract both waste and ore. The uphole retreat stope voids are not backfilled. Instead the mine design utilises yielding pillars between adjacent extracted stopes to gradually deform over a timeframe that permits ore extraction.

The LOMP09 schedule has factors applied to account for poor weather, public holidays, equipment availability, equipment utilisation, historically justified limitations on mine production and historically justified limitations on mill throughput.

The Frasers deposit provides the bulk of future reserves under LOMP09 for the immediate future so the operation is benefiting from fewer equipment moves, fewer haul roads to maintain and more homogeneous feed to the mill. Oceana plans to develop the Coronation deposit to the north, contingent on an access agreement with the land owners. Oceana plans to restart mining of the Southern Pit deposit prior to completion of the Frasers Open Pit Stage 6.

The open pit operation is owner-operated by Oceana whilst the underground operation is mined under an alliance agreement with Oceana providing management and technical guidance to the mining contractor who performs the physical mining tasks. A range of other contracts support the mining operations.

Oceana's performance at Macraes has shown that the mining equipment and mining methods are suited to the required mining rates and deposit geometry. Open pit and underground mine design procedures are appropriate and have been conducted in accordance with established industry standards and with input from appropriately qualified geotechnical specialists, hydrological specialists and consultants. Historical productivity and safety records are generally in line with or better than industry standards. The LOMP09 open pit and underground life of mine plan schedule has been prepared to 2015. The schedules were subsequently amended to include mining from Southern Pit which extends mine life out to 2017. The schedules rely only on reserves, and are considered appropriate and reasonable.

1.7.3 Processing

Over the last eighteen years Oceana has developed considerable experience in development and operation of the complex ore processing technology required to optimise gold recovery from the Macraes refractory ores.

Emphasis is placed on the control of costs. The relatively high tonnage processed, the simple flotation reagent regime and economies resulting from concentration of the gold into a flotation product comprising between 1.5% and 3% of the ore mass treated reduce operating cost. Labour costs are also lower than in most comparable developed countries. The operating cost of the core sulphide process is due to low comminution costs (contributed to by the coarse grind, and relatively soft ore).

Plant utilisation has been maintained at about 95% which is at the high end of typical industry benchmarks. Gold recovery on open pit ore and underground combined, for 2009 averaged 79.6%. Overall, recoveries are considered reasonable given the refractory nature of the ores.

It should be noted that the treatment of sulphide concentrate from the Reefton operation at Macraes utilises spare capacity in the autoclave circuit. In some years excess concentrate will be produced.

Oceana will bypass some low-preg robbing Macraes concentrate around the autoclave, feeding it directly to the CIL plant to enable all the highly refractory Reefton material to be oxidised.

1.7.4 Infrastructure, Environment and Tenement Status

Oceana continues to maintain appropriate infrastructure at Macraes, including road access, power, water supplies and administration facilities.

Environmental management and mitigation measures are maintained at Macraes, including ongoing monitoring to ensure compliance with resource consent conditions and permit requirements. These consents and permits are issued by the Ministry of Economic Development (MOED), the Otago Regional Council (ORC) and the Waitaki District Council (WDC). Tailings and waste rock disposal facilities are maintained and managed on an ongoing basis. Progressive rehabilitation is ongoing.

Consents are in place for additional uplifts to be constructed on tailings storage facilities (Mixed Tailings Impoundment and Southern Pit 11). There is sufficient tailings storage capacity in the present facilities to store tailings until 2014. Permitting of an additional storage facility (Back-road) is scheduled for the first half of 2010.

The project reserves, plant site, tailings dams and waste dumps are located on land that is covered by mining permits, and which Oceana owns or has access to mine. The sole exception is the Coronation deposit which is covered by an exploration permit and to which access is under negotiation. All material tenements and landholder agreements are in good standing and have been independently reviewed.

There are no material compliance issues relating to the principal mining and processing operations. Oceana has consents for the creation of a Heritage and Art Park at the Macraes site as part of its mine closure and restoration strategy. Implementation of this Heritage and Art Park is now well underway, with various artworks completed or under consideration. Oceana is in partnership with Otago Fish and Game, a semi-government organisation, to manage a Trout Hatchery on the Macraes mine site.

A draft closure and rehabilitation plan has been prepared and is being reviewed. Oceana intends to incorporate the closure plan into future LOM plans. Estimated costs for final closure may require review. As more of the Heritage and Art Park is developed a better knowledge of costs involved will be available.

1.7.5 Production

Oceana has prepared LOMP09 production plans from reserves only which cover 2010-2015 for Macraes. LOMP09 has subsequently had Southern Pit reserves included in the schedule to extend mine life to 2017. The production rates forecast are consistent with recent performance and the anticipated grades. The mine production plans are considered reasonable for the purpose of long term scheduling.

During the 2010-2017 peak production years the open pit excavator fleet is planned to comprise two Caterpillar 5130's (being replaced with similar units in 2011), one Caterpillar 5230 (replaced in 2011) and one Hitachi EX3600, to load six Caterpillar 785 haul trucks and eleven (increasing to thirteen in late 2010) Caterpillar 789 haul trucks. Oceana is satisfied that there are sufficient working areas for the excavators to operate and there is reasonable opportunity to reassess the requirements.

During the 2010-2012 peak production years the underground operation in accordance with LOMP09 is planned to provide approximately ~15% of the Macraes ore using a fleet of three Tamrock H205D electric-hydraulic jumbos, one Caterpillar 2900, one Caterpillar 1700 and two Tamrock 1400 LHD's in conjunction with five Tamrock 50D haul trucks. The underground ore is dumped at an in-pit stockpile for periodic rehandling by the open pit fleet to the process plant's run of mine stockpile. Planned production for 2010 to late 2011 is primarily stope ore with additional development ore when encountered within the mine design. LOMP09 has production during 2012, being solely derived from stope extraction. Oceana is satisfied with the accuracy of the dilution factors, ore loss factors and constraints placed upon the LOMP09 schedules and the 2009 underground life of mine schedule is considered reasonable for the purpose of long term scheduling.

The projected plant throughput fluctuates between 5.4Mt and 5.6Mt for 2010 to 2015.

1.7.6 Management

The owner operator open pit mine and the alliance agreement underground mine are performing to expectation.

Oceana management has aggressively sought out new opportunities for cost reduction and increased efficiency. The mining and processing operations have concentrated on minimising production costs to maintain profitability. Oceana continues to pursue cost reduction innovations.

The general management approach is strongly safety-oriented and the safety performance statistics reflect that attention.

1.7.7 Capital and Operating Costs

Capital cost estimation and forecasting are considered reasonable and consistent with proposed development programmes and ongoing requirements. In practice, capital expenditures over the period LOMP09 may be more variable than forecast due to unforeseen problems, modifications, upgrades and introduction of new technology.

Capital expenditure provisions (2010 to 2017) include expenditures for capitalised mining costs totalling NZ$236M and sustaining capital of NZ$66M (excluding exploration) and are considered accurate to within ±15%.

Plant operating cost estimates for Macraes are generally considered reasonable and consistent with recent experience and trends, and are regarded as accurate to ±15%.

1.7.8 Environment

The Macraes gold mine is fully consented for environmental purposes, with actual and potential environmental effects regularly monitored and reported to the relevant agencies (except for consenting of the back road tailings impoundment, which is not required until 2014 and is planned to be permited in 2010).

The site is achieving environmental compliance, with good internal reporting of environmental issues and performance. The site environmental documentation is appropriate and follows Environment Management Strategy (EMS) principles, although a full EMS is not in place. Documentation is reviewed and updated regularly.

Overall, no material environmental issues have been identified to limit the ongoing operation of the mine within the LOMP09.

2 INTRODUCTION

2.1 Report Preparation

This report has been prepared at the request of OceanaGold Corporation and Oceana Gold (New Zealand) Limited (Oceana).

OceanaGold Corporation is the ultimate holding company in which Oceana (New Zealand) Limited is a subsidiary. OceanaGold Corporation is the reporting issuer in Canada.

References in this report to "Oceana" include Oceana Gold (New Zealand) Limited, OceanaGold Corporation and their subsidiaries.

2.1.1 Purpose of the Report

This report was prepared as a Canadian National Instrument 43-101 Technical Report for Oceana by internal qualified persons employed by Oceana to provide updated technical information relating to the upgrading of the Round Hill Resource, the reinstatement of part of the Southern Pit resource, the inclusion of the Southern pit Reserve and the expansion of the Frasers pit reserve. The quality of information, conclusions and estimates contained in this report is based upon:

i) information available internally at the time of preparation;

ii) data obtained from outside sources; and

iii) the assumptions, conditions, and qualifications set forth in this report.

This report is intended to be used by Oceana and to be filed as a Technical Report with Canadian Securities regulatory authorities pursuant to Canadian provincial securities legislation. Except for the purposes legislated under Canadian provincial securities laws, any other use of this report by any third party is at that party's sole risk.

2.1.2 Reporting Standards

The report has been prepared in accordance with Canadian National Instrument 43-101 for the 'Standards of Disclosure for Mineral Projects' of December 2005 (the Instrument) and the resource and reserve classifications adopted by CIM Council. This report complies with disclosure and reporting requirements set forth in the Instrument, Companion Policy 43-101CP, and Form 43-101F1.

This report has also been prepared in accordance with the 'Code for the Technical Assessment and Valuation of Mineral and Petroleum Assets and Securities for Independent Expert Reports' of 2005 (the "Valmin Code") as adopted by the Australasian Institute of Mining and Metallurgy (AusIMM), and is consistent with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2004 (the "JORC Code"), as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC). The satisfaction of requirements under both the JORC and Valmin Codes is binding upon the authors as Members of the AusIMM.

2.1.3 Currency

All monetary amounts expressed in this report are in New Zealand dollars (NZ$) unless otherwise stated.

2.2 Authors of the Report

This technical report has been prepared by or under the supervision of the following authors:

- Mr Rodney Redden, who is employed by Oceana as Exploration and Development Manager; and

- Mr Jonathan Moore, who is employed by Oceana as Principal Resource Geologist.

The authors are the Qualified Persons, as defined by the Instrument, and have the qualifications and experience set out below.

The authors are both members of AusIMM.

2.3 Qualifications and Experience of the Qualified Persons

2.3.1 Mr Rod Redden

Mr Redden holds a Bachelor of Engineering (Mining) with Honours from the University of Wollongong, graduating in 1994 and a Masters of Business Administration (2005) from the University of Wollongong.

Mr Redden has experience in many aspects of mining, ranging from exploration, open pit and underground mining, to metallurgy and processing. He has worked in Australia, Tanzania, Kazakhstan, New Zealand and the Philippines in Coal, Iron Ore, Lead-Zinc, Chromite, Gold and Copper.

He has been Project Manager for a major new mining project in Kazakhstan.

Mr Redden has been with Oceana for over three years in a variety of roles including Principal Underground Engineer, Development Manager, Mining Manager (Underground and Open Pit). He is currently Exploration and Development Manager.

Mr Redden is the author of the following sections of this report: 1.6, 1.7.2 to 1.7.8, 4.7, 4.8, 5, 16, 17.13, 17.14, 17.16, 19.2 to 19.8, 20.2, 20.3 and 21.1 to 21.10

Mr Redden was not an author of the previously filed technical reports, and has undertaken the necessary investigations in order to satisfy himself that those parts of this report that are unchanged from the previously filed reports can reasonably be relied upon.

2.3.2 Mr Jonathan Moore

Mr Moore holds a BSc (Hons) in Geology, a GradDip in Physics and has 20 years experience in exploration, open pit and underground mining and resource geology. He has worked in epithermal gold, porphyry copper and gold, mesothermal gold and lead-zinc deposits within Australia, New Zealand and the Philippines.

Mr Moore has been employed with OceanaGold since 1996 in a variety of project, mine geology, resource geology roles. He is currently the Principal Resource Geologist.

Mr Jonathan Moore is the author of the following sections of this report: 1.1 to 1.5, 1.7.1, 2, 3, 4.1 to 4.6, 6 to 15, 17.1 to 17.12.7, 17.15, 18, 19.1, 20.1, 21.11, 22, 23, 24 and 25.

3 SOURCES OF INFORMATION

The authors of this technical report have not relied upon other experts in its preparation, other than obtaining input from persons employed within Oceana who have provided information concerning legal, environmental or other matters relevant to this report.

The information used to prepare all sections relating to Mineral Resources and Reserves was furnished by Oceana, including (specifically in relation to Reserves) input from: Knowell Madambi (Principal Development Engineer), Adrian Winchester (Mining Engineer – Projects), Matthew Mengel (Technical Services Superintendent, Frasers Underground Operation), Anthony Jones (Geotechnical Engineer, Frasers Underground Operation), Alex Zuhoski (Technical Services Superintendent, Frasers Open Pit Operation), Andrew Winneke (Geotechnical Engineer, Frasers Open Pit Operation).

Oceana furnished all data, modelling, testwork and financial analysis to verify the information relating to Mineral Resources and Reserves and the conclusions regarding the resource and reserve estimates.

In so far as other persons have had input into the preparation of this report, the authors have conducted appropriate due diligence and consider such reliance to be reasonable.

A list of the publications and internal reports that were used in the preparation of this report, and to which specific reference is made in the body of this report, appears in section 23.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Area of Property

The Macraes Project has a total area of 27,492 hectares.

4.2 Location

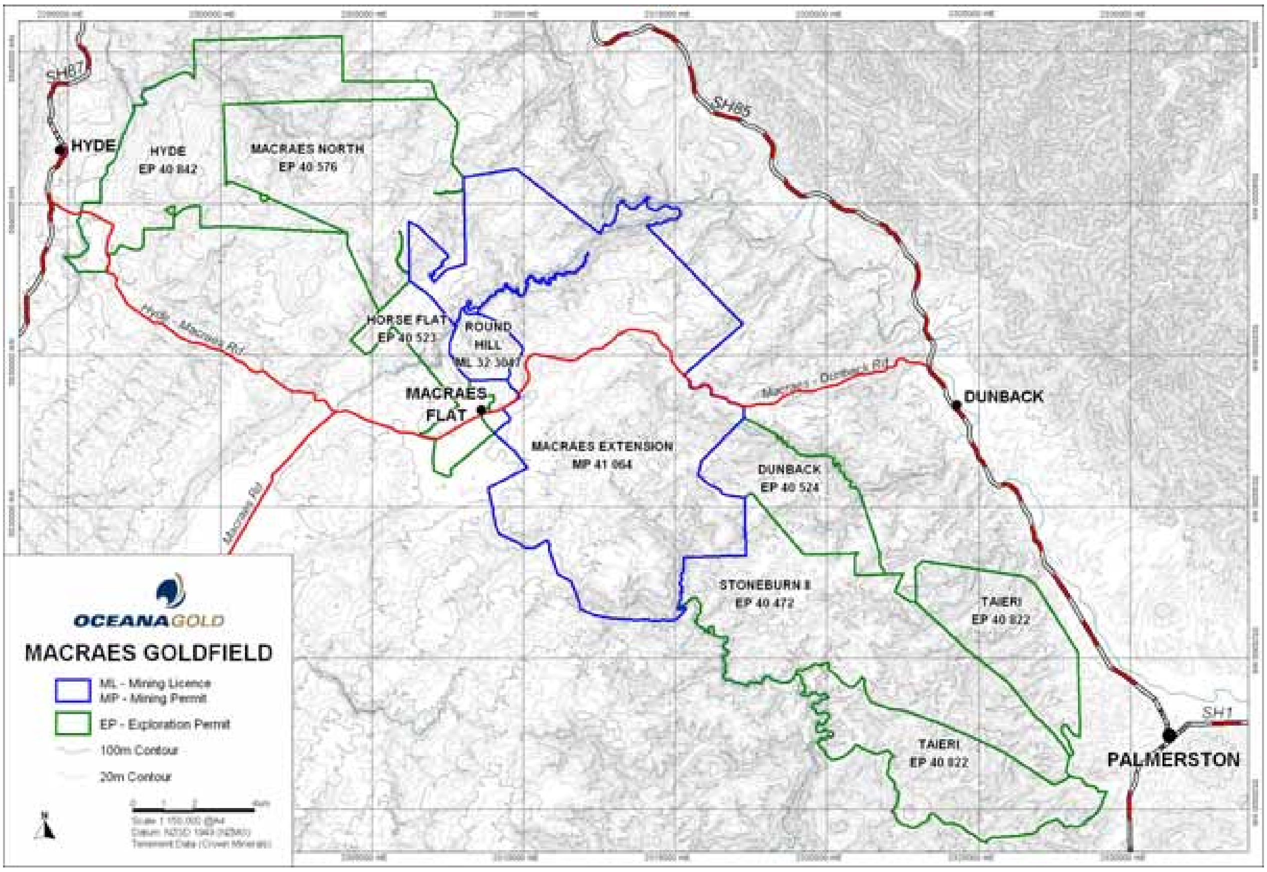
The Macraes Project is located approximately 60km north of Dunedin in eastern Otago (Figure 4.1) and is situated about 2km northeast of the small township of Macraes Flat (Figure 4.2).

The central activity is mining from the Frasers Open Pit and the FRUG mine within Mining Permit (MP) 41 064. The process plant, several waste rock stacks and tailings impoundments are located within Mining Licence (ML) 32 3047.

The Project is located at, -45.36°S, 170.43°E (Latitude/Longitude – World Geodetic System 1984) or at 5,535,600mN, 2,308,500mE New Zealand Map Grid (New Zealand Geodetic Datum 1949).

A local grid has also been established for the Macraes Project. This grid is rotated 45^{o} west of true north, parallel with the local trend of the mineralized structures.

Figure 4.1: Macraes Project Location Map



Macraes is predominantly surrounded by farmland (tussock and grassland for high country grazing) as shown in Figure 4.2.

Figure 4.2: Macraes Project

4.3 Tenure

Oceana holds a contiguous group of tenements to the north-west and south-east of Round Hill, covering approximately 35km of strike along the mineralized Hyde Macraes Shear Zone (HMSZ) as shown in Figure 4.1 and detailed in Table 4.1.

The tenements comprise a pipeline and a Mining Licence, a Mining Permit (MP) and Exploration Permits (EP) granted or applied for under the Mining Act 1971 or the Crown Minerals Act 1991.

An exploration permit, can have an initial term of five years with the right of extension of term, over 50% of the area (in one contiguous piece), for a further term of up to five years but not exceeding 10 years. An exploration permit can be converted into an appraisal permit for further terms exceeding the initial 10 years. The Crown Minerals Act 1991 allows for extensions of the permit areas subject to certain conditions for compliance.

Table 4.1: Macraes Project Tenements

Tenement No	Licensee	Location Name	Date Commenced	Term/Expires	Area (Hectares approx.)	Interest In Permit
ML 32 3047	Oceana	Round Hill	31.10.1989	21yrs Oct 30, 2010	400	100%
PLL 32 3047-5	Oceana	Pipeline	31.10.1989	21yrs Oct 30, 2010	23	100%
MP 41 064	Oceana	Macraes Extension	01.02.1994	21yrs Jan 31, 2015	9,610	100%
EP 40 472	Oceana	Stoneburn II	18.05.2001	2nd term May 17, 2010	4,296	100%
EP 40 523	Oceana	Horseflat	11.09.2001	2nd term Sep 10, 2011	941	100%
EP 40 524	Oceana	Dunback	18.05.2001	Seeking extension	1,449	100%
EP 40 576	Oceana	Macraes North	28.10.2001	2nd term Oct 27, 2011	3,434	100%
EP 40 822	Oceana	Taieri	07.12.2006	1st term Dec 06, 2011	3,970	100%
EP 40 842	Oceana	Hyde	30.01.2007	1st term Jan 29, 2012	3,369	100%

4.4 Nature and Extent of Title

Land in the immediate vicinity of the Oceana mining operations, and most of the land in tenements ML 32 3047 and MP 41 064, is owned by Oceana. Land not used for active mining activities is leased at a market rental to local farmers. Oceana land ownership also extends beyond those two tenements as shown in Figure 4.3. Land outside the mining tenements is currently owned by a variety of landowners. However, a number of properties are under option to purchase by Oceana.

The granting of a mineral permit does not confer a right of access to land subject to the permit. A permit holder must arrange land access with the owner and occupier of the land before beginning any prospecting, exploration or mining for minerals on or in land (other than minimum impact activity as defined in the Crown Minerals Act 1991). Access arrangements are binding on successors in title provided they are registered against affected land titles where the term is longer than six months.

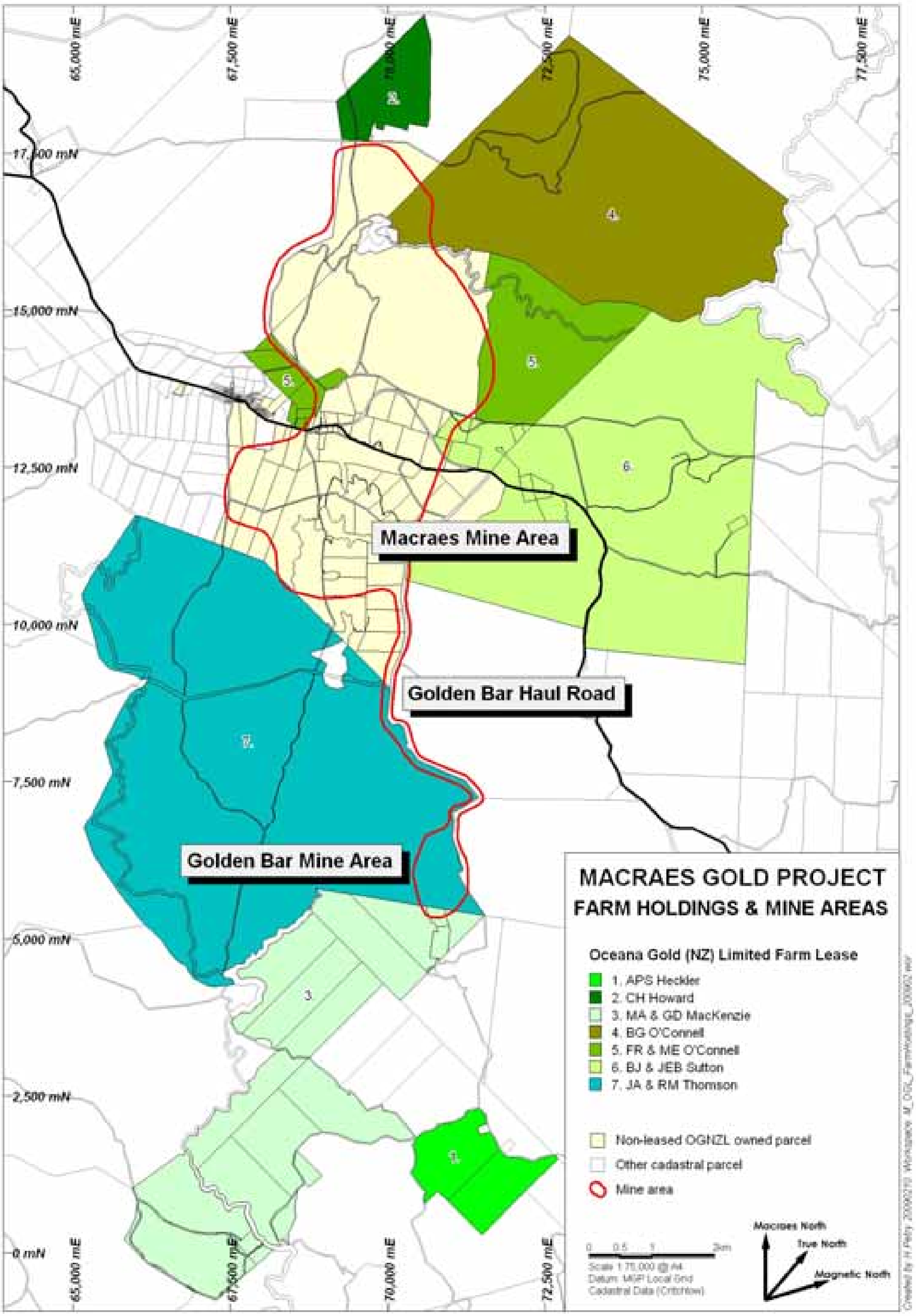
Any activity carried out below the surface of any land subject to a permit will not be considered, for the purposes of the Crown Minerals Act, to be prospecting, exploration or mining on or in the land and consequently not require an access arrangement, if the activity will not or is not likely to:

a) cause any damage to the surface of the land or any loss or damage to the owner and/or occupier of the land; or

b) have any prejudicial effect regarding the use and enjoyment of the land by the owner and/or occupier; or

c) have any prejudicial effect regarding any possible future use of the surface of the land.

Figure 4.3: Macraes Project Farm Holdings and Mine Areas



4.5 Property Boundaries

In general Oceana property boundaries follow existing cadastral boundaries. Where Oceana boundaries have departed from these, the boundaries have been surveyed by registered surveyors.

4.6 Location of Mineral Resources

Mineralized zones at Macraes are located along the surface trace of the HMSZ, a major northwest-southeast trending structure (see section 7.3). All previous mining production and current resources are located along this zone.

Figure 4.4 shows the location of mineral resources within Oceana's Macraes tenements. Local grid coordinates for the limits of the resource areas at Macraes are given in Table 4.2.

Figure 4.4: Macraes Project Mineral Resource Locations

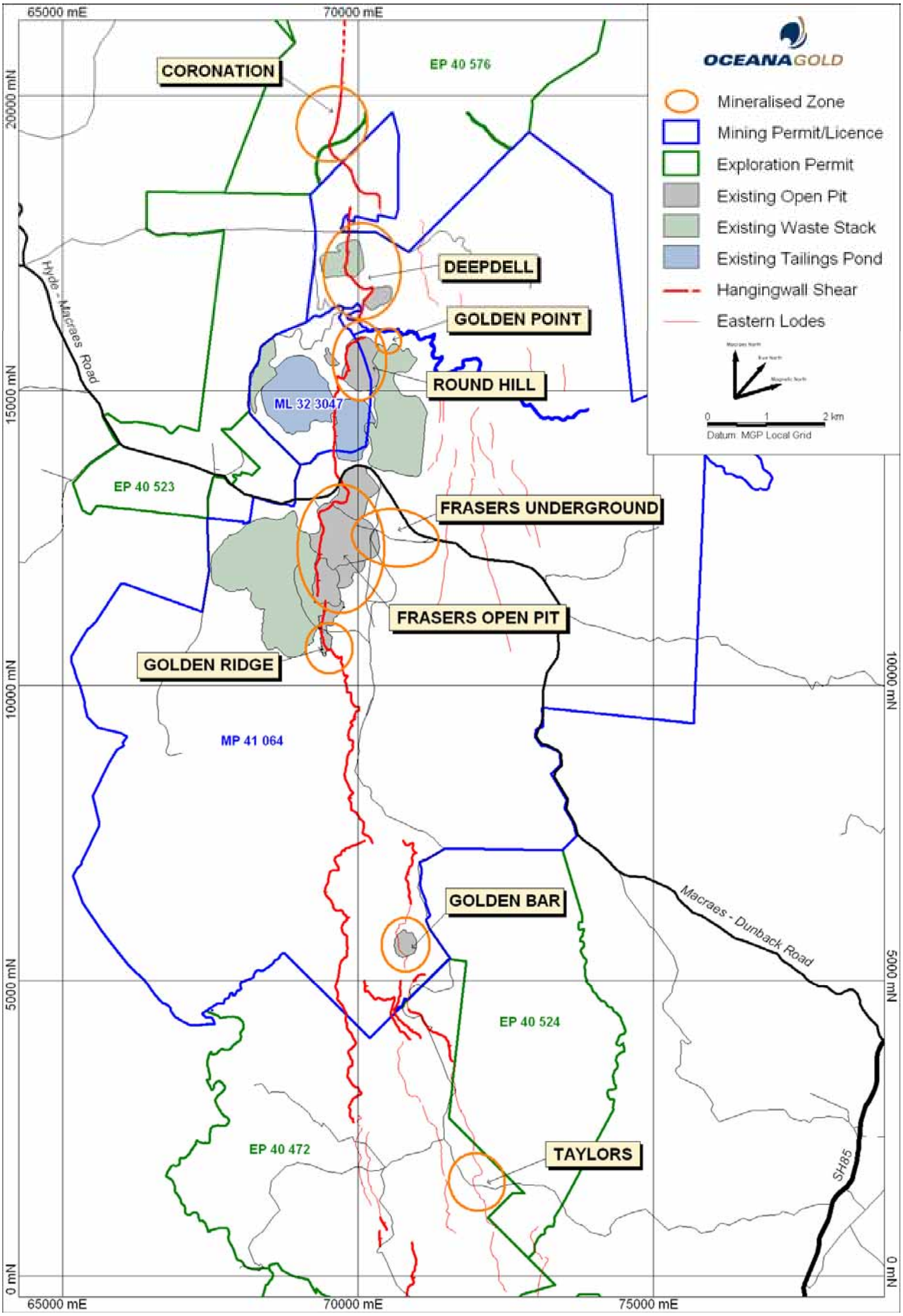


Table 4.2: Macraes Resource Area Boundaries – Southern Pit/Innes Mills

Resource Area	Northing (local grid)		Easting (local grid)	
	From	To	From	To
Coronation	18,250	20,500	69,200	70,500
Deepdell	16,060	18,250	69,200	70,650
Golden Point	15,660	16,060	70,410	70,650
Round Hill	14,000	16,000	69,200	71,100
Innes Mills	13,850	14,000	69,540	70,050
Frasers Open Pit	11,425	13,000	69,000	70,550
Frasers Underground	12,090	12,820	69,925	71,460
Golden Bar	5,300	6,500	70,500	71,400
Taylors	1,175	1,650	71,825	72,350

4.7 Royalties

Under the Mining Act 1971 no royalty on gold is payable to the Crown on ML 32 3047. A royalty of 2% of the gross proceeds from scheelite concentrate sold (if any) is payable to the Crown. The ML is covered also under a Royalty Agreement between OW Hopgood and Oceana, where Oceana pays Hopgood a royalty 5% of revenue if recovered by open pit mining and 3% if recovered by underground mining) on any gold, scheelite or other minerals recovered from the area which was formerly PL 31 595.

Under the Crown Minerals Act 1991 which applies to (MP 41 064) royalties are payable to the Crown annually in respect of all gold, silver and scheelite that are taken from the land pursuant to the mining permit. Royalties are calculated based on net sales revenue or accounting profits whichever is the greater. Royalties are generally calculated and payable at the following rates:

a) no royalty is payable if net sales revenue from a permit is less than NZ$100,000 for an annual reporting period or averages less than $8,333 per month if the annual reporting period for the permit is less than 12 months. Where the permit is part of a production unit, the thresholds will apply to net sales revenues from all permits in the production unit;

b) a royalty of 1% Ad Valorem is payable if net sales revenue from a permit is between NZ$100,000 and NZ$1,000,000; or

c) a royalty of either 1% Ad Valorem or 5% of the accounting profits, whichever is greater, if the net sales revenue from a permit is more than NZ$1,000,000.

Under the Minerals Programme for Minerals (excluding petroleum) dated February 1, 2008 the Coronation deposit once converted from Macraes North EP 40-576 to a mining permit will attract a 2% of spot price royalty payable to the Crown annually in respect of all gold, silver and scheelite.

4.8 Environmental Liabilities

4.8.1 Overview

This report provides an overview of the principal environmental statutes that Oceana operates under in order to understand the extent of Oceana's environmental liabilities and how these liabilities arise.

There are two principal agencies that oversee Oceana's mining activities together with a number of secondary agencies. The two principal agencies are:

- Otago Regional Council; and

- Waitaki District Council

In order to undertake mining of Crown owned minerals (such as gold) there are three key consents and permits required:

- Resource consents to use land, water, and air;
- Access arrangements with the owner of the land; and
- A permit under the Crown Minerals Act 1991.

The nature of the mining industry with ongoing exploration and mine site planning does mean that from time to time it is necessary to seek new approvals or variations to existing approvals. This report only considers the current known situation regarding present day (at the time of preparing the report) and forecast mining activities (known at the time of preparing the report).

4.8.1.1 Resource Consents

Territorial authorities and regional councils have primary responsibility for administering the Resource Management Act 1991 (RMA). Their functions are defined within the RMA (sections 30 and 31 RMA) but in simple terms, relevant to Oceana's activities, territorial authorities manage the effects of land use change and noise, whilst regional councils manage effects associated with:

- water quality (surface, ground and coastal water);
- taking, damming, diversion of water;
- discharges of contaminants into or onto land, air, or water, and discharges of water into water; and
- the bed of any water body, and the planting of any plant in, on, or under that land.

In managing the effects of activities on the matters above, both territorial authorities and regional councils seek to give effect to the purpose of the RMA (section 5 RMA), which is "to promote the sustainable management of natural and physical resources". Sustainable management is defined by the RMA to mean managing the use, development, and protection of natural and physical resources in a way, or at a rate, which enable people and communities to provide for their social, economic, and cultural wellbeing and for their health and safety while:

- sustaining the potential of natural and physical resources (excluding minerals) to meet the reasonably foreseeable needs of future generations;
- safeguarding the life-supporting capacity of air, water, soil, and ecosystems; and
- avoiding, remedying, or mitigating any adverse effects of activities on the environment.

Supporting the purpose of the RMA are a number of principles that persons exercising functions and powers under the RMA, in relation to managing the use, development, and protection of natural and physical resources, shall recognise and provide for (section 6 RMA), have particular regard to (section 7 RMA), and take into account (section 8 RMA).

The term "effect" includes (section 3 RMA):

- any positive or adverse effect;
- any temporary or permanent effect;
- any past, present, or future effect;
- any cumulative effect which arises over time or in combination with other effects – regardless of the scale, intensity, duration, or frequency of the effect, and also includes-
- any potential effect of high probability; and

- any potential effect of low probability which has a high potential impact.

The RMA places restrictions on the use of land (section 9 RMA), the subdivision of land (section 11 RMA), the use of the coastal marine area (Section 12 RMA), on certain uses of beds of lakes and rivers (section 13 RMA), water (section 14 RMA), and the discharge of contaminants into the environment (section 15 RMA). Activities that 'use' land, water, and air cannot legally occur unless they are permitted by a rule in a district or regional plan, or have a resource consent granted.

A resource consent is therefore permission from a territorial authority or regional council to undertake an activity that would otherwise contravene a statutory plan prepared under the RMA (or sections 9, 11, 12, 13, 14, or 15 RMA).

Applications for resource consents are typically processed in one of two ways. Non-notified applications (no general public submissions allowed) may occur when the environmental effects of the activity to be consented are considered to be no greater than minor and written approvals have been obtained from any deemed affected parties. Notified applications occur when the environmental effects of the activity to be consented may be greater than minor, and provide an opportunity for any person in New Zealand to make a submission supporting or opposing the application.

Consents are granted subject to conditions such as the requirement for an environmental bond to be paid by the consent holder, conditions to avoid, remedy, or mitigate significant adverse effects on the environment and provide for the monitoring of these effects. Failure to meet the conditions of consent may lead to prosecution, payment of fines, and in severe circumstances the cancellation of the consent. The maximum penalties available under the RMA are imprisonment for a term not exceeding 2 years, or a fine not exceeding $600,000. If the offence is a continuing one, an additional fine may be imposed not exceeding $10,000 for every day or part of a day during which the offence continues.

Oceana has been deemed, in obtaining the consents to license activities with environmental effects for this project, to have met the purpose and requirements of the RMA, which establishes a not insignificant threshold for the granting of such consents.

Oceana holds all required resource consents for the activities it undertakes. Compliance with the conditions of resource consents is discussed below.

4.8.1.2 Crown Minerals Act 1991

The allocation of rights to prospect, explore or mine for minerals owned by the Crown is carried out by the issuing of permits under the Crown Minerals Act 1991 (CMA). "Crown owned" minerals include all naturally occurring gold and silver and some coal and other metallic and non-metallic minerals and aggregates. The CMA contains transitional provisions that allow mining licenses granted under the Mining Act 1971 (such as ML 32 3047) to remain in force.

4.8.2 Resource Consents

Oceana holds a range of consents issued by the ORC and the WDC.

Consents have conditions placed on them detailing performance standards and monitoring frequency. Oceana reports on consents on a quarterly or annual basis, with these reports sent to the ORC and WDC, as appropriate.

Internal monthly reports are also prepared by Oceana. The quarterly and monthly reports identify general compliance with conditions of consent.

Seepage control in Maori Tommy Gully, below the Mixed Tailings Impoundment dam, is a previously identified issue reported by GHD in November 2005 (Report for Oceana Due Diligence). Seepage in Maori Tommy Gully that bypasses the preferential collection systems encounters a grout curtain across the gully, a line of detection bores, and finally a line of compliance bores. Seepage has been detected in the compliance bores. This was predicted to occur in early water management modelling.

The purpose of the grout curtain is to disperse seepage when it hits the curtain. This curtain is working to design. The plume is also behaving as expected, moving along the base of the gully. The seepage plume may be caused by leakage from the drains that go from the tails dam to sump B or by ground water seepage from the tails dam moving from the base of the dam to the sump. There are options to address either source. During mining operations Oceana can intercept if necessary. During restoration Oceana will have to intercept and manage for a period of 5-10 years post mining.

The seepage at this time is not considered material as the preferential pathway of conservatives into the Maori Tommy Gully is monitored and a management strategy included in the site closure plan.

The monthly reports note several incidents relating to minor floods, spills, overflows and dust levels. The monthly reports indicate some form of mitigation management has been implemented for each of these incidents. Improvements are noted throughout the year.

Total Suspended Particulate (TSP) monitoring of air occurs across the site for monitoring dust issues and compliance. On occasions there have been elevated readings but this is typically able to be explained by the proximity of the compliance site to an activity. There is no ongoing significant compliance issue associated with TSP levels.

The Annual Work and Rehabilitation Programme (AWRP) gives a detailed review of the proposed operation over the next year. This includes:

- an explanation of any departures from planned mining activities during the previous year;
- description of operations over the next year including detailed plans of proposed operations;
- description of any adverse effect that has arisen over the past year;
- description and evaluation of mitigation measures;
- a report of rehabilitation during the exercise of the consent and results;
- rehabilitation plan for the next year;
- plans for the next year showing actual contours at 5m intervals;
- calculation of costs to deal with any adverse affects on the environment;
- detailed calculation of costs of complying with all rehabilitation conditions of the consent;
- monitoring data; and
- bonding assessment and costs for rehabilitation and monitoring.

The AWRP is provided to compliance authorities as a tool for ensuring a ‘no-surprises’ approach in relation to Oceana’s forward work plan, environmental compliance, and rehabilitation.

Rehabilitation activities are documented, and predominantly involve topsoiling, fertilizing and seeding completed waste rock stack areas, drill pads, and tailings dam faces. Plant pest spraying also occurs, and company owned land not required for mining is leased to local farmers. No material rehabilitation issues have been identified.

Heritage issues are managed under a Heritage Management Plan and to date this has focused on interpretation panels and walkways. No material heritage management issues have been identified.

In conclusion, the site is monitored and has a history of general compliance. Importantly, no significant recurring non-compliance issues are identifiable. Compliance limits for TSP, as noted above, have on occasion been exceeded but there has been no enforcement/abatement action by the ORC.

Environmental Bonds for Macraes Operation are discussed in section 21.5.

4.8.3 Access Arrangements

Oceana is the owner of the majority of land in the immediate vicinity of the Macraes Operation, and most of the land within tenements ML 32 3047 and MP 41 064. A number of properties outside the mining tenements are under an option to purchase by Oceana.

The Coronation reserves for LOMP09 are located on land for which access to mine is under negotiation.

4.8.4 Mining Permits

Oceana has in place the necessary mining licenses and permits issued under the Mining Act 1971 and the Crown Minerals Act 1991 for life of mine mining requirements, and no material environmental liabilities emerged.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation and Vegetation

The project area is situated on an elevated (approximately 490m above sea level) plateau drained by a trellis pattern of north-westerly and north-easterly trending streams. Parts of the plateau are deeply dissected.

Vegetation is comprised of a combination of improved pasture and tussock grassland, while streams and gullies are choked by matagouri, gorse, thistles and wild rose. The predominant land use is stock grazing, with small areas covered by pine plantations.

5.2 Access to the Property

Access to the mine is by sealed highway from Dunedin, and then via sealed or metal roads from Middlemarch and Palmerston. There is good access along metal roads and farm tracks throughout the project area.

5.3 Proximity to Population Centres

The Macraes mine is within short driving distance to a number of populated centres:

- The Macraes plant site is approximately 6km from the Macraes Flat village. The village and surrounding farming community comprises approximately 40 families.
- Dunedin, a university city with a population of 119,000, is 102km away by road.
- Oamaru with a population of 11,000 is 105km by road.
- Palmerston, with a population of 800, is 37km by road.

Transport to the site is typically by vehicle. A domestic and international airport is located in Dunedin, which also has an operating seaport. A national trunk railway line from Christchurch to Dunedin passes through Palmerston.

5.4 Climate and Operating Season

As well as being elevated, the project area is also exposed, windy and dry, with high evaporation in the warmer part of the year. High temperatures are experienced during summer and occasional falls of snow during winter.

Rainfall averages about 550 millimetres per year, but can vary by about 80 millimetres per year depending on topography. There is little seasonal variation in rainfall, but monthly totals can be quite variable and the area is susceptible to long dry periods. Droughts, which last two or three years, have been recorded in the east Otago region every 10 to 20 years.

Climatic influence translates to a potential for 3-5 days lost mining time per year after occasional heavy rains or snowfall.

5.5 Infrastructure

5.5.1 Sufficiency of Surface Rights

Oceana has all necessary rights and permits for current and proposed mining operations at the Macraes Project, except for the landowner access to mine part of the Coronation reserve. Future discoveries may require new consents and conversion of ground currently held as an exploration permit to a subsequent mining permit prior to the commencement of mining.

5.5.2 Power

Macraes is connected to the local power grid, which provides a reliable electrical power supply. The power line has adequate capacity to supply the mine at full operating limits.

5.5.3 Water

Water is drawn from the Taieri River and pumped to the site. Through storage and active recycling, an adequate reservoir of process and potable water is maintained to enable continuous operation, even in times of drought conditions.

5.5.4 Mining Personnel

Mining, processing and support staff are drawn from the local region, with many living in the nearby towns or commuting from Dunedin. Recruitment of suitably skilled and experienced employees for all areas of the operation has been achieved and maintained.

5.5.5 Communications

Macraes is connected to the New Zealand Telecom system, providing both voice and internet access. The mine site utilises a local area network for computer connections.

A multi-channel radio network is utilised for operations communication in the mine and process plant.

5.5.6 Mining Infrastructure

The Macraes Project area is sufficient to contain the current open pit mines and underground, process plant, haulage roads, tailings storage areas and waste rock storage areas. Furthermore, sufficient surface area is available within Macraes project area for the construction of any infrastructure necessary for the potential development and mining of other deposits under consideration.

6 HISTORY

6.1 Mining History

The earliest alluvial mining in the district commenced at Murphy's Flat in 1862, with Macraes Flat, Deepdell and some parts of Horse Flat being worked soon after (Hamel, 1992). Murphy's Creek was the major early alluvial workings and there is evidence that all of its tributaries were being worked in the 1860's. The Murphy's Creek alluvial workings are reasonably well preserved and are considered to be of historic significance (Hamel, 1992).

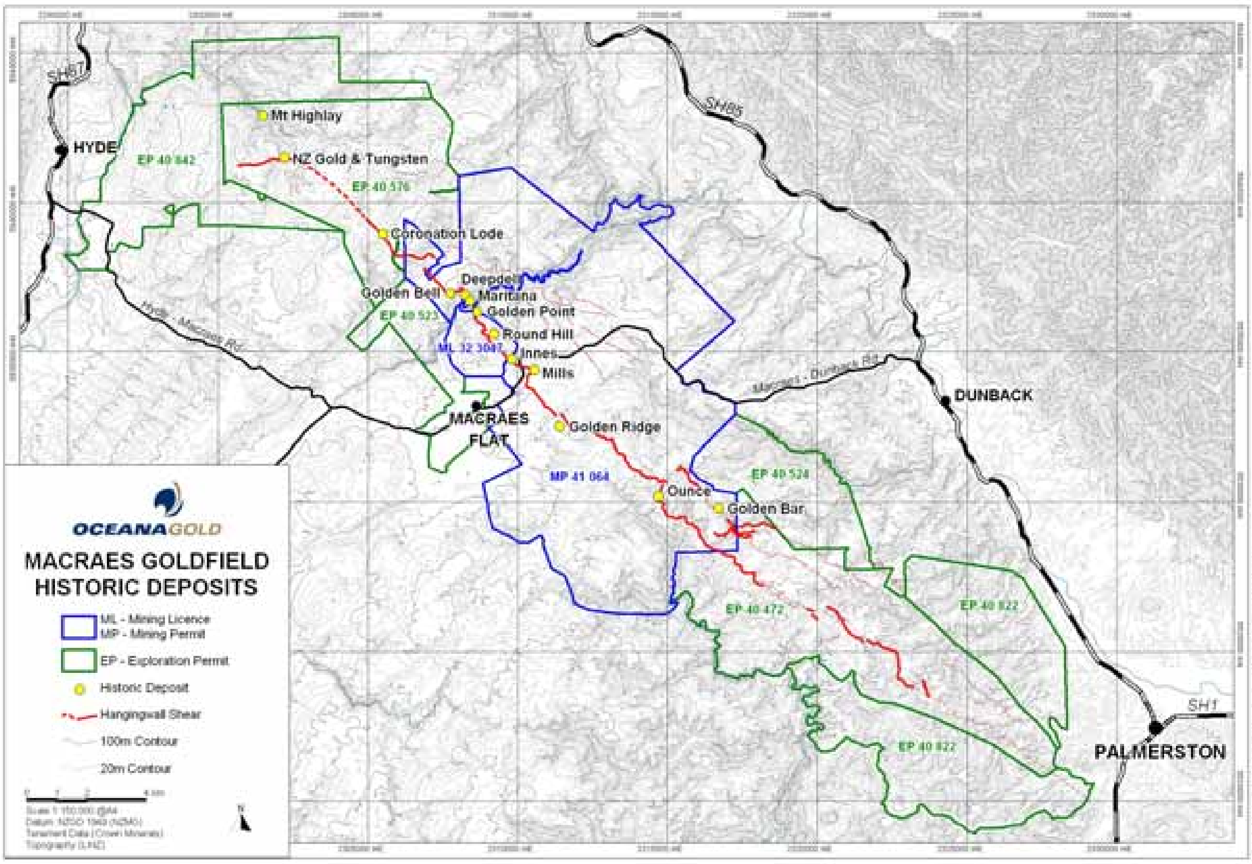
Lode quartz mining commenced in the 1860's, but the scale of operations was very small. The Golden Point/Round Hill lode system was not discovered until 1889. Development of Golden Point commenced in 1889 and it became established as a significant scheelite and gold producer. From 1890 to 1933, it produced an estimated 13,000 ounces of gold and 800 tons of scheelite (Williamson, 1939). Other areas mined included Maritana, Golden Bell and Deepdell but quantities were small with a total reported of 8,463 tons of crushed ore for 1,630 ounces of gold and 50 tons scheelite (Williamson, 1939 as quoted in Ballentyne 1971). Lodes were worked for either scheelite or gold depending on the price at the time. This was due to the fact that the fine grinding required to liberate the gold resulted in poor recovery of scheelite.

Areas continued to be mined after 1939 as tungsten was in demand during the Second World War but gold prices were sharply reduced during this time. The scale of operations at this time was small and work was discontinuous, as a result records of production of ore at this time are poor. Local miners suggest that less than 100 tonnes of scheelite was mined since 1939 but estimates are widely varied (Petrie, 1970). It was a question of economics (due to preferential recoverability of gold or tungsten) not ore availability that controlled the scheelite industry at Macraes Flat.

The first lode worked in the Macraes field was probably the Duke of Edinburgh, described by Ulrich (1875). He also mentions the Golden Bar Reef and the Moonlight Reef, at the head of Macraes Flat, but gives no detail about them. In 1888, the Highlay Reef was discovered on the Mareburn, and the lode was soon traced to Golden Point, where it was opened out in 1889. Further prospecting soon resulted in the opening of other mines along the lode, some of them, however, being little more than surface workings.

The mines that have been worked, given in order eastward, are Mount Highlay, New Zealand Gold and Tungsten, Coronation, Golden Bell, Maritana, Deepdell, Golden Point, Round Hill, Innes, Mills', Griffins, Golden Ridge, Ounce and Golden Bar (Williamson, 1939).

Figure 6.1: Macraes Historical Mining Areas



6.2 Prior Ownership

The original tenements at Macraes were owned by Golden Point Mining Limited and BHP Gold Mines (New Zealand) Limited, owned by BHP Gold Mines Limited. During December 1989, Macraes Mining Company Limited (MMCL) obtained 100% ownership of these tenements. On May 14, 1999, Macraes Mining Company Limited changed its name to Gold and Resource Developments (New Zealand) Limited and again to GRD Macraes Limited on June 30, 2000. Finally on May 18, 2004, the name was changed to Oceana Gold (New Zealand) Limited.

6.3 Previous Work

This section details exploration activities conducted in the Macraes region prior to 1990 when MMCL acquired the Macraes permits.

6.3.1 Geochemistry

6.3.1.1 Stream Sediment Sampling

During 1987, an orientation stream sediment sampling survey was conducted by BHP Gold Mines (New Zealand) Limited (BHP), in the Round Hill Area. The results from a total of 64 samples taken showed total sediment fine fraction samples (-20# and -80#), gave the best results.

Although the bulk cyanide leach method returned lower-level results, this method was adopted for use on a regional basis due to ease of sample collection.

6.3.2 Geophysics

The first geophysical survey carried out over the HMSZ was by Homestake New Zealand Exploration Limited (HNZEL), in April 1985 (Robinson, 1986). It comprised an orientation induced potential (IP)/Resistivity survey totalling 8.35 line kilometres over Round Hill and Southern Pit.

The objective of the survey was to test the ability of IP to discriminate between ore grade Au-scheelite-sulphide mineralization at Round Hill (intersected by diamond drilling) from weakly mineralized parts of the lode shear system south of Round Hill employing dipole-dipole and gradient array IP surveys. The survey lines were orientated both grid east, across the line of lode, and grid north, parallel with the strike of the lode system but across the trend of the Round Hill shoot. A dipole spacing of 50m was used.

Dipole-dipole traverses revealed chargeability responses more or less associated with outcrop of the main lode, however the anomaly was stronger than what would be expected from the sulphide content of the lode system (generally less than 1% total sulphide with maximum of 2-5% in sulfidic zones) and may be related to graphite associated with the shear system. A chargeable source near the centre of line 14900mN was associated with very weak mineralization intercepted in diamond drill hole (DDH) 5.

The surveys across the Round Hill Shoot failed to clearly discriminate between the shoot and weakly mineralized lode to the south. The gradient array surveys on these lines revealed anomalies in the vicinity of Ferguson’s workings (Southern Pit - 14200-14400mN) in which graphitic rocks are exposed. In summary, IP chargeability anomalies may define a shear system of the Macraes type, especially if sufficient graphite is present, but the variability of sulphide content within the lode system is too low to discriminate between high grade mineralized shoots and low grade or barren parts of the lode system (Robinson, 1986).

In 1986, BP Oil New Zealand Limited (Minerals Division), (BP Oil), carried out a total of 32 line km of dipole-dipole IP/Resistivity surveying at Nunn’s-New Zealand Gold and Tungsten, Frasers (south of the alluvial flats along Macraes Road), Golden Ridge, Golden Bar and Frasers East (Coochey, 1986; Moore, 1986). The bulk of this survey, 19 line km, was over Frasers and Golden Ridge. A comparative analysis of the IP survey results with subsequent drilling was not completed, however it appears that the results were similar to those of HNZEL.

On November 17, 1987, BP Oil undertook a down-hole geophysical survey on drill hole GRRC 14 (Moore, 1987). BPB Instrument Limited carried out the demonstration log recording dip-meter analysis, density logs, focused electric and resistivity logs, neutron-neutron and gamma logs. Moore reported that the logs which provided the most information and which correlated with the down-hole geology were resistivity, focused electric, density, and dip-meter analysis.

During 1987, the Ministry of Works and Development Central Laboratories used portable "OYO" equipment to log 13 holes on the eastern high wall side of the (then proposed) Round Hill pit (Brown, 1988). BPB Instruments Limited also logged one of these holes which enabled a comparison between the two contractors. The surveys were reasonably successful with a similarity of results between the two contractors. The results of the survey became very useful allowing for the interpretation of structures required for slope stability analysis.

6.3.3 Drilling

During 1970, Helpet Mining Company Limited drilled 28 holes in the Macraes Flat area exploring for tungsten mineralization. Core recovery was poor and mineralization was found to be sporadic and discontinuous. Kennecott Exploration (Australia) Pty Ltd also undertook exploration in the area in 1970-71, but their reconnaissance work did not include drilling.

In 1984, Homestake New Zealand Exploration Limited commenced exploration at Round Hill and by the end of 1986 had drilled over 5.5km of strike on the Deepdell, Round Hill and Frasers systems at 100 to 200m drill hole spacings. This drilling defined the Round Hill shoot which was amenable to open cast mining (Lee et al, 1989).

Following HNZEL's success in the Macraes Flat region, BP Oil obtained licences to the north-west and south-east of Macraes along the HMSZ. Between 1986 and 1988, BP Oil carried out drilling at Nunn’s, Golden Ridge, Ounce, Golden Bar and Frasers East.

Drilling has continued at Round Hill and adjacent prospects since the purchase of HNZEL by BHP in 1987 and subsequently by MMCL in 1990 (see section 11).

6.4 Historical Estimates

There are no relevant historical resource estimates for the Macraes Operation compliant with NI 43-101 rules or CIM guidelines. The mine has been in production for approximately 19 years and resource estimates for the deposits have been routinely updated and refined over time. These estimates have been prepared in accordance with the JORC code (JORC, 2004) and its predecessors. The current CIM compliant resource estimates (as of December 31, 2009) are presented in section 17.

6.5 Previous Production

Historical production from the Macraes Goldfield is poorly recorded. The Golden Point mine produced an estimated 13,000 ounces of gold and 800 tons of scheelite from 1890 to 1933 (Williamson, 1939).

Since the commencement of open pit mining in 1990, the combined Macraes open pits have produced more than 2.5Moz. Since 2000, annual gold production has ranged between 162koz and 184koz.

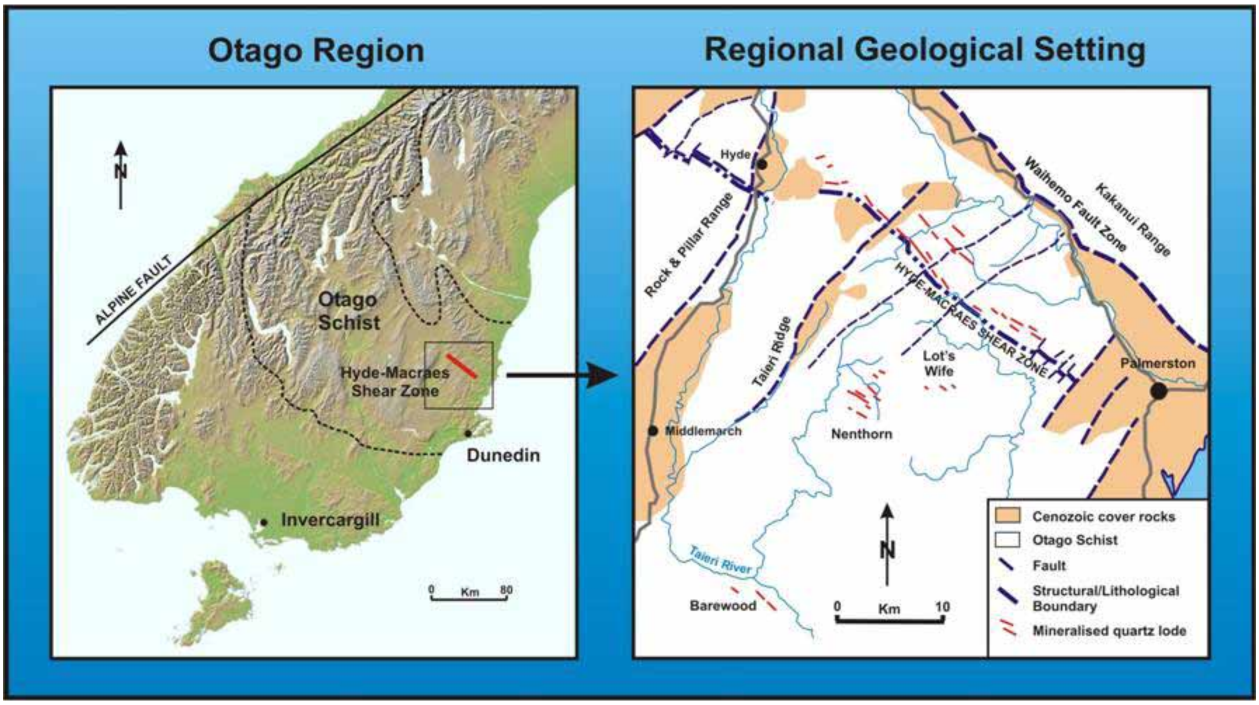
7 GEOLOGICAL SETTING

7.1 General

The Macraes gold deposits are located in a major, low-angle (~15-20°) structure known as the Hyde Macraes Shear Zone (HMSZ). This regionally continuous, late metamorphic deformation zone cuts greenschist facies metasedimentary rocks of the Otago Schist. The Otago Schist is a moderately high pressure metamorphic belt (Yardley, 1982; Mortimer, 2000) that formed by collisional amalgamation ("Rangitata I" Orogeny) of the Caples and Torlesse terranes in the Early-Middle Jurassic (Coombs et al., 1976; Bishop et al., 1985; Little et al., 1999).

The Otago Schist forms part of the more extensive Haast Schist that underlies about 10% of the New Zealand microcontinent (Mortimer, 1993a, b). To the south and west, the Otago Schist grades into mainly volcaniclastic, non-schistose greywacke and argillite of the Caples Terrane (Bishop et al., 1976). Caples Terrane greywackes comprise probable Late Paleozoic-Early Mesozoic, dominantly intraoceanic, magmatic arc lithologies, with lesser proportions of continental derived material (MacKinnon, 1983; Roser et al., 1993). To the north and east, the Otago Schist grades into non-schistose greywacke and argillite of quartzofeldspathic composition from the Torlesse Terrane (Coombs et al., 1976). Torlesse metasediments were derived from an active continental magmatic arc and granitic gneiss basement in the Permian-Late Triassic. Both terranes contain minor greenschist (metabasite) and quartzite (metachert) layers (MacKinnon, 1983).

Figure 7.1: Regional Geological Setting



7.2 Regional Geology

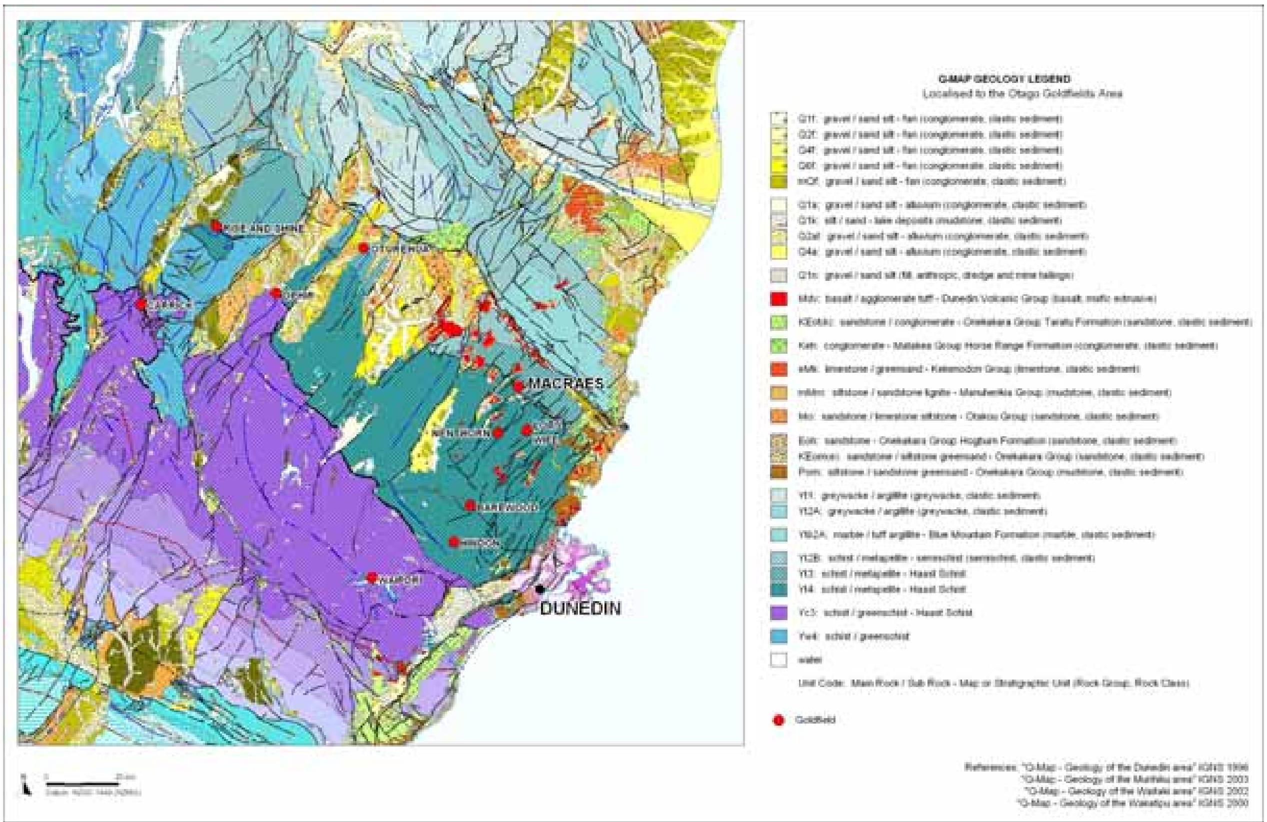
The Otago Schist straddles the Caples and Torlesse terranes, forming an approximately 150km wide northwest to southeast trending, broadly antiformal structure. It is composed predominantly of psammitic and pelitic greyschists (metamorphosed greywacke and argillite) with lesser amounts of greenschist (metabasite) (Craw, 1984; Mortimer, 1993a). In general, the schist is lithologically monotonous, without regionally extensive marker bands and lacks a recognisable lithostratigraphy. Metamorphic grade in the schist belt ranges from prehnite-pumpellyite facies in the northeast and southwest flanking non-schistose metagreywackes of the Torlesse and Caples terranes respectively, through pumpellyite-actinolite facies semischists to chlorite greenschist facies with local preservation of garnet-biotite-albite zone along the schist axis (Mortimer, 2000 and references therein). Texturally, the schist belt follows similar patterns to the metamorphic zones, grading from unfoliated metasedimentary rocks (greywackes and argillites) at the margins, through slates and phyllites (semi-schists) to strongly segregated and laminated schists and locally gneisses in the schist centre (Turnbull et al., 2001 and references therein).

At least four phases of deformation (D_1-D_4) can be attributed to the progressive fabric development of the Otago Schist (Craw, 1985 and references therein). The first phase of deformation (D_1) was characterised by ductile, isoclinal folding (F_1) of bedding (S_0) and development of a penetrative axial planar foliation (S_1). This stage was accompanied by intense transposition of lithologic layering (S_0) with metamorphic/quartz-albite segregation veins (V_1) developed parallel to S_1 (Turnbull, 1981; Craw, 1985). The second phase of deformation (D_2) folded S_1 with generally tight-similar or isoclinal style folds (e.g., Turnbull, 1981; Craw, 1985). A penetrative axial planar foliation (S_2) is commonly developed and in most places lies subparallel to S_1. F_2 folding of metamorphic segregation veins has produced rootless fold hinges that outline S_2 (intrafolial folds). These detached fold hinges form a prominent quartz rodding lineation (L_2) (Craw, 1985). Mineral elongation lineations and foliation intersection lineations (e.g., S_2 and S_1-S_0 fabric) are typically parallel to L_2 (Mortimer, 1993b). Nappe structures with S_1 as the form surface are commonly developed (e.g., Wood, 1963, 1978; Means, 1963, 1966; Bishop, 1974; Turnbull, 1981; Craw, 1985; Cox, 1991). The first two stages of ductile deformation (D_1, D_2) accompanied regional metamorphism (Craw, 1985).

Phase 3 deformation (D_3) produced localised folding that postdated the metamorphic peak. F_3 folds show thickened and rounded hinge zones with attenuated limbs. F_3 crenulations developed in the fold hinges define an L_3 lineation. Fold axial surfaces (S_3) form at high angle to the penetrative foliation (S_2-S_1) (Wood, 1963; Means, 1963, 1966; Norris, 1977; Turnbull, 1981; Craw, 1985). The final phase of deformation (D_4) is related to postmetamorphic mesoscopic angular kink folds and macroscopic warping (km-scale) of the penetrative foliation (Craw, 1985; Mortimer 1993a; Turnbull, 2000).

Not all the deformation phases are observed throughout the schist. In terms of geographic distribution, the flanking semischists of the schist belt have been through at least one transposition stage with only a single penetrative foliation recognised. The majority of primary structures such as bedding have been obliterated as they were flattened and transposed by early-generation folding (D_1). The central schists have been overprinted by multiple deformation phases (D_2, D_3 etc.) with foliation transposed two or more times, producing mesoscopic and macroscopic folds together with possible overprinted metamorphic facies (Mortimer, 1993b, 2000).

Figure 7.2: Otago Geology Map



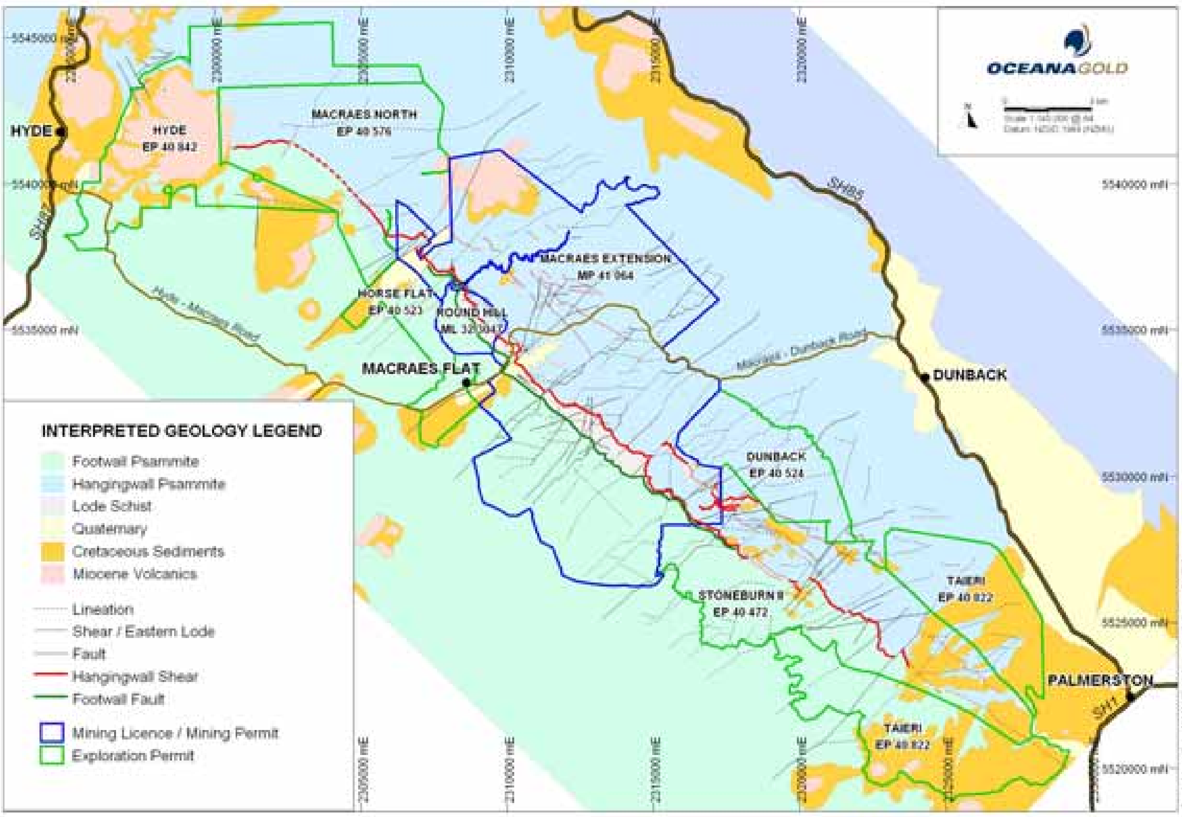
The age of peak Otago Schist metamorphism is around Early to Middle Jurassic (200-170Ma; Adams et al., 1985; Little et al., 1999), soon after Triassic accretion. The schist is then thought to have been held at mid to lower crustal depths until the Early Cretaceous (135±5Ma) after which it was rapidly unroofed at 0.6 - 1mm/y (Little et al., 1999). Exhumation of the schist was accompanied by extensional faulting (northwest and northeast oriented orthogonal fault pattern) and associated mineralization (Craw and Norris, 1991). Cover sediments unconformably lying on various schist textural zones demonstrate that deep structural levels of the Otago Schist belt were exposed to erosion by the mid Cretaceous (e.g. Adams and Raine, 1988).

Regional extension occurred in the Middle Eocene through to the late Oligocene (Turnbull et al., 1975; Norris and Turnbull, 1993). The Early Miocene marked the inception of the Pacific-Australian Plate boundary and the initiation of the Alpine Fault as a throughgoing transform (Norris et al., 1990 and references therein). Reverse faults and upright folds deforming the otherwise gently dipping foliation of the Otago Schist are mostly related to Miocene-Pliocene crustal shortening in response to an increase in the component of convergence on the plate boundary (Norris et al., 1990). Many if not all of the reverse faults represent reactivated late Cretaceous extensional faults (Turnbull et al., 1975; Norris and Turnbull, 1993). Miocene-Recent right lateral strike slip movement on the Alpine fault displaced part of the Haast Schist (Marlborough Schist) c. 480km to the northeast, with displacement becoming more oblique over the last 10Ma (Molnar et al., 1975; De Mets et al., 1990; Sutherland, 1995). At present the Otago region is being deformed in response to oblique shortening with continued development of active reverse faults and folds (Jackson et al., 1996; Markley and Norris, 1999).

7.3 Local Geology

The HMSZ, which hosts gold mineralization is one of the largest Mesozoic structures mapped in the Otago Schist, traceable for at least 30km along strike. The HMSZ developed during uplift of the host schist through the brittle/ductile transition in the Late Jurassic (Craw et al., 1999; Craw, 2002). During this period, the region was undergoing late metamorphic compression with the HMSZ initiating as a thrust system. The sense of movement on the thrust was top-to-the-west (Teagle et al., 1990). The shear zone developed sub-parallel to the foliation, and the foliation was deformed into a duplex system, which resulted in the stacking and thickening of the Intrashear Schist. Shallow dipping fault-veins (shear-parallel quartz veins) formed sub-parallel to the principal shears and the enclosing foliation during the thrusting. These veins fill local extensional sites (metre scale) in the duplex thrust system. Stockwork veins occurred syn-kinematically with thrusting, as demonstrated by mutual cross-cutting relationships (Begbie & Craw, 2006). Formation of stockwork veins post-dated the main phase of ductile deformation of the Intrashear Schist.

Figure 7.3: Macraes Geology Map



The HMSZ consists of variably altered, deformed, and mineralized schist up to 150m thick, known as the Intrashear Schist (Mitchell et al., 2006 and references therein). The thickest part of the shear zone consists of several mineralized zones stacked on metre-thick shears. These shears have ductile deformation textures overprinted by cataclasis (Craw et al., 1999). The HMSZ is hosted in lower greenschist facies (chlorite zone) schist and has been juxtaposed against upper greenschist facies schist along a normal fault, the Footwall Fault (Angus et al., 1997; Craw, 2002; Craw et al., 2004; Mortimer, 2000). This fault is younger than the HMSZ and truncates its base.

The boundary between mineralized HMSZ schist (Intrashear Schist) and unmineralized lower greenschist facies schist is commonly a well defined structure, the Hangingwall Shear. This shear ranges up to 25m thick and is typically black due to the presence of fine grained graphite and sheared sulphide minerals (McKeag et al., 1989). The Hangingwall Shear can be traced through the mined pits in the main mining area. Schist above the Hangingwall Shear is dominated by micaceous rock types which are fissile in outcrop (Petrie et al., 2005). Pods (1-10 m scale) of more massive feldspathic schist (psammitic) occur within this fissile schist (pelitic). The Intrashear Schist also has pods of massive schist surrounded by fissile schist. The Intrashear Schist is distinguished from the hangingwall schist units by subtle but pervasive alteration including the addition of graphite and sulphides, replacement of titanite by rutile, and decomposition of epidote (Craw et al., 1999).

7.4 Deposit Geology

7.4.1 Overview

At present, mining is concentrated in two areas: Frasers Open Pit and FRUG. FRUG is the down-dip extension of the open pit mine and comprises two zones of mineralization: Panel 1 and Panel 2.

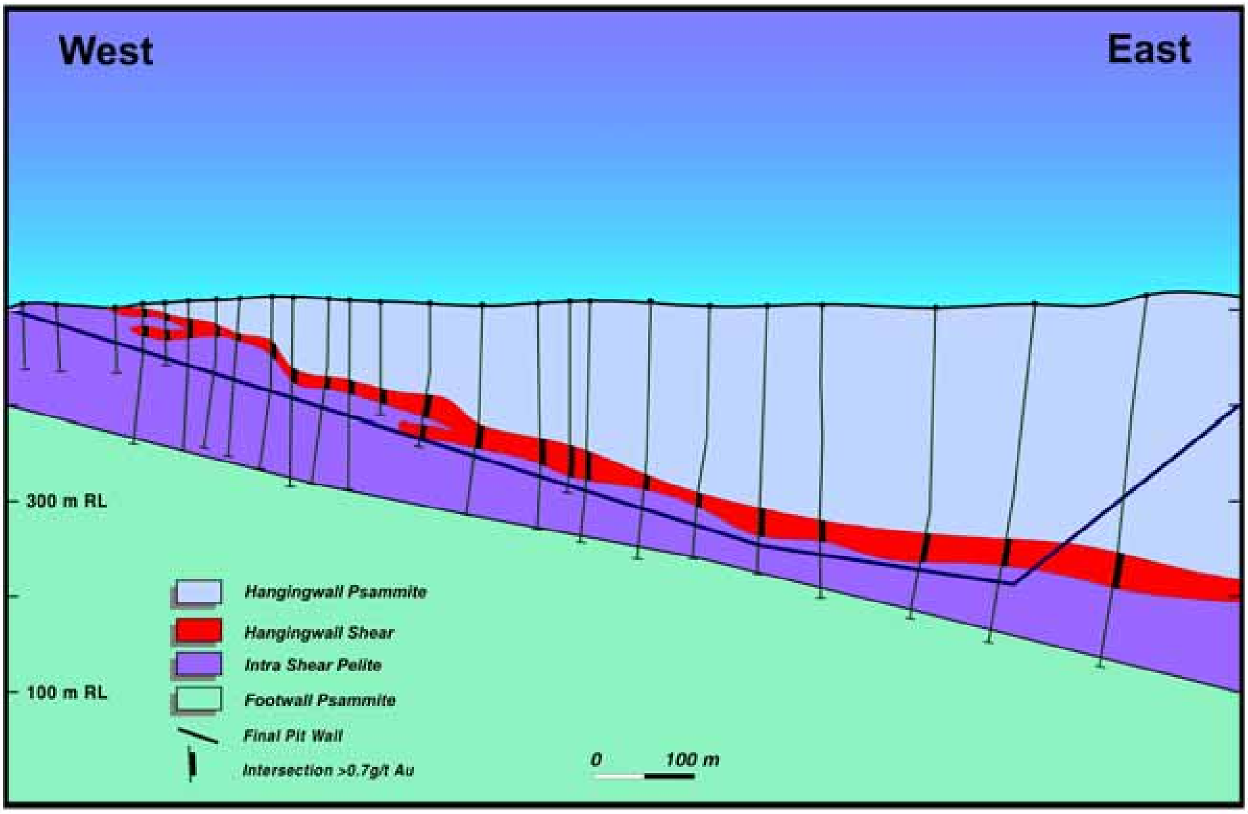
7.4.2 Frasers Open Pit

The Frasers open pit deposit is defined by the Hangingwall shear. In outcrop, the shear dips 15° to 20° to the east and is ~5 m thick. At depth, the dip of the shear flattens to around 5° to 10° and develops into a ~20m to 30m thick high grade zone of quartz cataclasite and lode schist. This interpreted ramp-flat geometry is relatively common at Frasers.

Within the Frasers pit, gold mineralization comprises a combination of Hangingwall, shear-parallel quartz veins, and stockwork veins. Hangingwall shear and stockwork veins account for the majority of mineralization within the Frasers pit, although there are a number of shear-parallel quartz veins. These veins typically splay off the Hangingwall and dip at between 5° and 10° to the east. A large amount of irregular mineralization occurs between the base of the Hangingwall and the Footwall Fault. This is stockwork mineralization and generally appears in the drilling as clusters of elevated gold grades. Stockwork mineralization refers to mixtures of quartz veins and concordant lodes, which appear discontinuous at the resource drilling scale. The Footwall Fault lies between 80m and 120m below the Hangingwall Shear and is easily identified in drill holes as a 10m wide zone of shearing. To date, no economic mineralization has been located below the Footwall Fault.

Gold-scheelite-pyrite-arsenopyrite mineralization is associated with replacement and fissure quartz veins within D4 post-metamorphic shear zones (Lee et al. 1989). Within the Frasers pit scheelite mineralization is predominantly found in proximity to the hangingwall shear. It is associated with gold mineralization, associated quartz veining, and displays complex crosscutting relationships.

Figure 7.4: Frasers Open Pit Schematic Cross Section

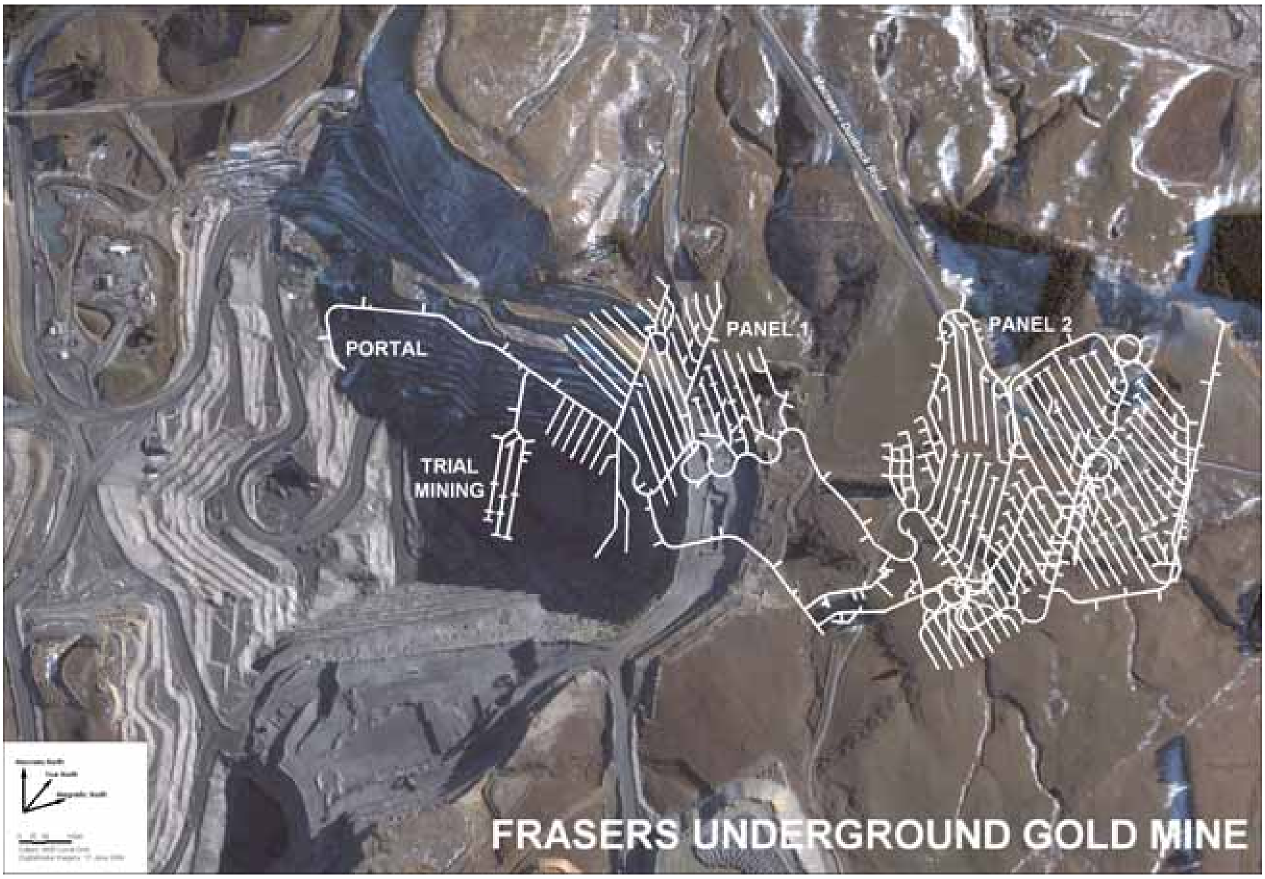


7.4.3 Frasers Underground

7.4.3.1 Panel 1

FRUG Panel 1 encompasses the down-dip continuation of the Hangingwall shear mined in the Frasers open pit, which is known to extend approximately 250m beyond the limit of the FR6 pit design. The thickest, most mineralized part of Panel 1 trends approximately northeast (050°) and tapers in width from approximately 350m at its western end (around 70,050mE) to approximately 150m width at the eastern limit of drilling (70,500mE), where it abuts the Macraes Fault Zone (MFZ).

Figure 7.5: Frasers Underground Mine Design, December 2009

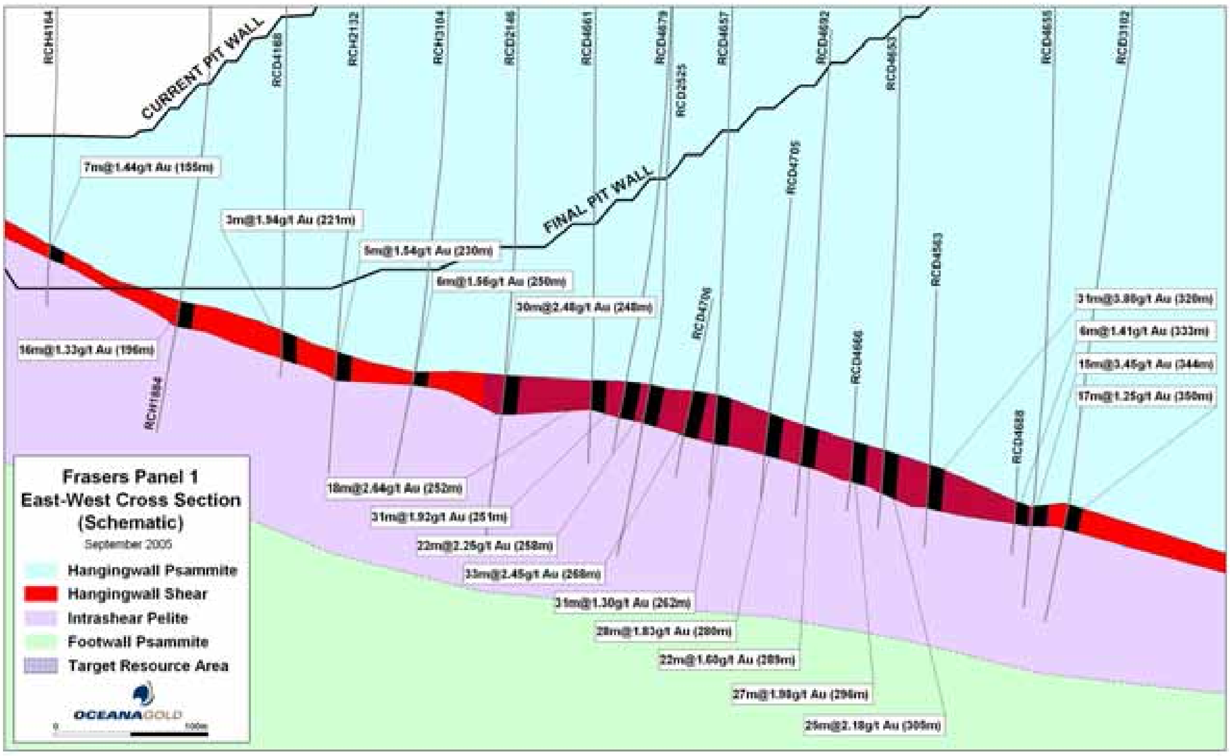


Mineralization is contained within the Intrashear Schist which is generally 80m to 100m thick, with the higher gold grades confined to the upper part, which is dominated by cataclasite, lode schist and local stockwork pelite lithologies. Numerous drill holes have penetrated through the Intrashear Schist into the Footwall Psammite, particularly at the western end of Panel 1 where the Footwall Fault is at a relatively shallow depth (<500m). Drill holes at the eastern end of Panel 1 have generally intersected the Hangingwall at around 320m to 340m depth (190mRL to 170mRL) and have been terminated within the Intrashear Schist, but beyond the limit of significant mineralization.

Mineralization within Panel 1 is consistent with the ore delineated in the Frasers Open Pit. The highest gold grades (up to 56.6 g/t Au) are contained within the strongly developed and visually distinguishable zone within the upper Hangingwall characterised by quartz cataclasite and silicified breccias. This typically forms a well mineralized, continuous zone approximately 10m to 15m thick, with a grade of approximately 3 g/t Au. Less intensely mineralized lode schist is typically developed lower in the Hangingwall package.

The package of mineralized rock is observed to thin abruptly at the margins of Panel 1, where the intensity of the Hangingwall shear weakens markedly. The exact location, nature and geological controls on this transition are not well constrained by drilling. Stockwork mineralization beneath the Hangingwall is not well developed below Panel 1 and is mainly limited to patchy development of quartz veining with comparatively low grades (from 0.1 to 5.0 g/t Au) with low apparent continuity at the scale of the current drilling. The stockwork mineralization density decreases rapidly towards the peripheries of Panel 1 and is generally absent where the Hangingwall is weakly developed. The density and grade of stockwork mineralization at Macraes is generally observed to increase where there is a change in Hangingwall orientation suggesting an overall structural control. The northern limit of Panel 1 (approximately 12,650mN at surface) is defined by the east striking MFZ. Ground conditions deteriorate significantly proximal to and within this fault zone, with a marked decrease in logged rock quality designation (RQD) and corresponding increase in fracture frequency rates. When the northern fringe of Panel 1, which came within 250m of the MFZ, difficult mining conditions were experienced with excessive dilution from stope backs and stopes self mining. Away from the MFZ stoping and development preformed reasonably well.

Figure 7.6: Frasers Underground Panel 1 Schematic Cross Section

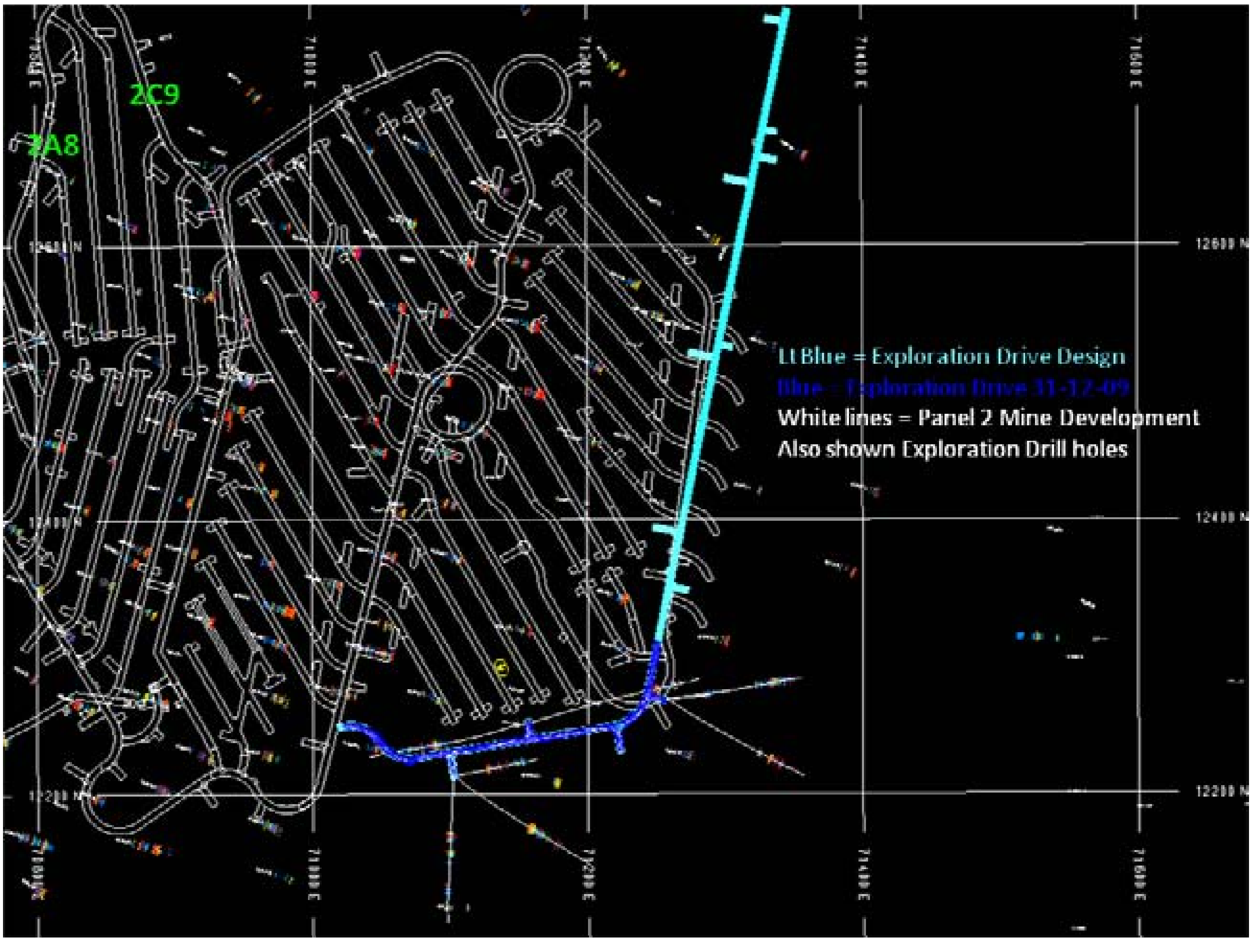


7.4.3.2 Panel 2

Panel 2 is located 300m to the southeast of Panel 1, separated by a zone of weakly mineralized Hangingwall approximately 300m wide centred on 70,500mE. Drilling intersections to date indicate that Panel 2 has a plan dimension of approximately 450m wide by 700m long down-dip, extending from about 70,600E. The Panel 2 mineralization also trends northeast (050°), similar to the trend of mineralization within Panel 1. This mineralization trend has also been observed at Innes Mills, Southern Pit and Deepdell and is considered to reflect the macroscopic structure of the HMSZ.

The southern margin of Panel 2 is defined by about twenty drill holes. The weak development of the Hangingwall in these holes suggests that the southern boundary of the panel area lies at around 12,100mN. In July 2009 an exploration drive on the southern margin of Panel 2 was commenced as shown on Figure 7.7 in light blue. The exploration drive gets 90m above the Hangingwall ore zone and has provided drilling platforms to test southern and more importantly in future, the 050 down dip extension of Panel 2. As at December 31st, 8 holes, as shown on Figure 7.7 had been completed from the exploration drive to better define the southern boundary of Panel 2. The exploration drive is expected to be completed in the 3rd quarter of 2010.

Figure 7.7: Frasers Underground Panel 2 Exploration Drive Development



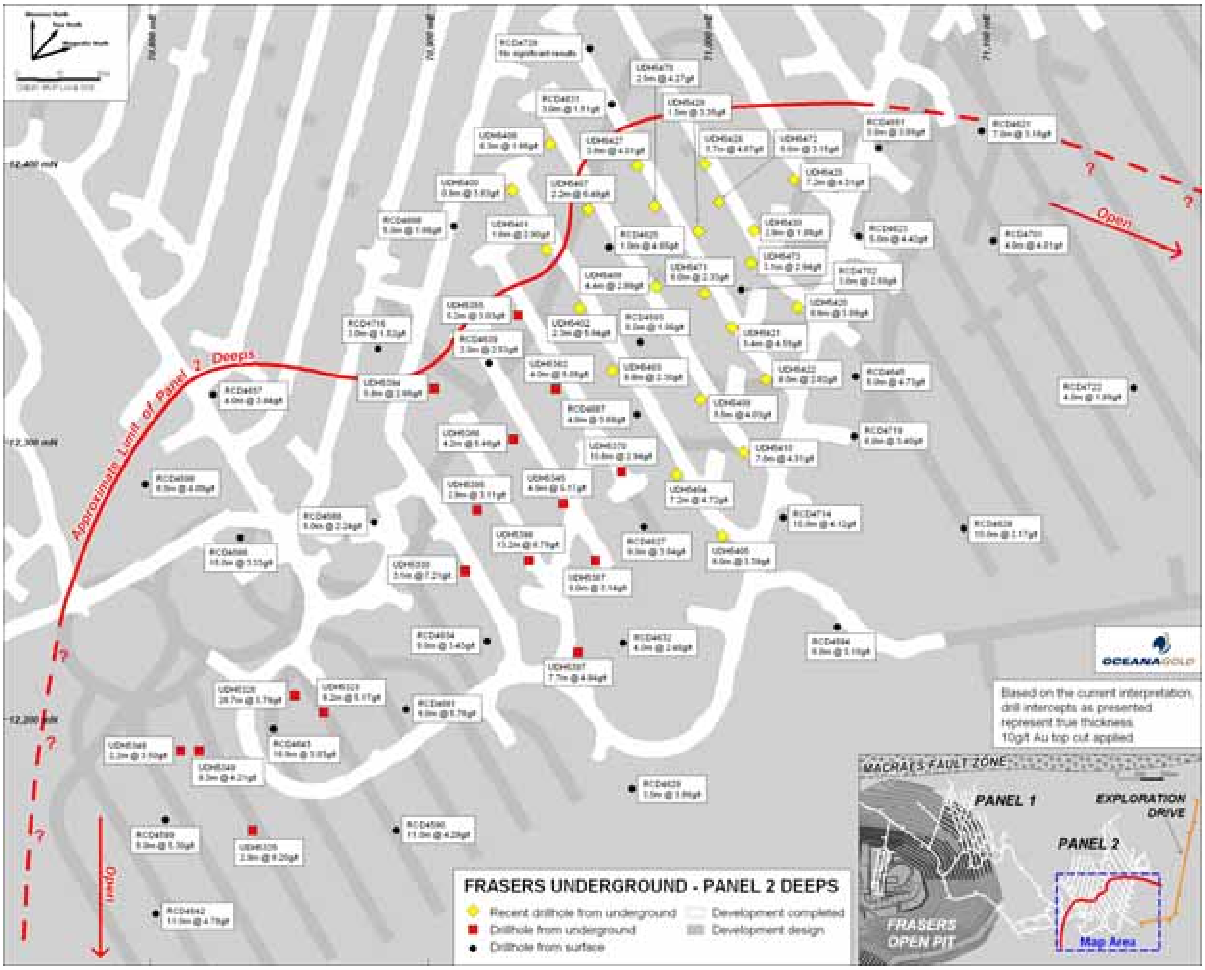
Currently the northern boundary, nor the area between Panel 1, the MFZ and Panel 2 have been adequately constrained by drilling and further work is required to determine if there is potentially economic mineralization in these areas. FRUG mine development (2C9 and 2A8 Drives shown in green on Figure 7.7) along the currently known northern Panel 2 boundary has encountered economic Au mineralisation. In December 2009 investigations commenced on the feasibility using 20m rises from existing mine development to allow Kempie drilling or something similar to allow exploration to the north of Panel 2 and between Panels 1 and 2.

The HMSZ in the Panel 2 area occurs at a depth of 370m (150mRL) at its western end, while the Hangingwall shear was located at 670m depth (-120mRL) at its most easterly intersection. The structure consistently shows a 15° to 20° east dip through the Panel 2 area. No significant flattening of the Hangingwall, as observed within Panel 1 or the Frasers pit, occurs within the Panel. This dip consistency may explain some of the observed differences in mineralization style between the Panel 1 and Panel 2 areas.

Hangingwall mineralization within Panel 2 is characterised by quartz cataclasite and zones of mineralized lode schist material. The Hangingwall is generally around 10 - 12m in thickness but locally ranges from 5m (on northern boundary) up to 20m thickness. Mineralized intersections vary in grade between 1.0 and 4.1 g/t Au and are typically higher average grade than in Panel 1.

A second mineralized structure on the south eastern margin of Panel 2 has been intersected in 7 exploration drill holes, below the Hangingwall. This mineralization was been named Panel 2 Deeps and is interpreted as potentially a low angle concordant lode in association with a zone of strongly developed stockwork. This style of mineralization is relatively common at Macraes and has been mined in the Frasers Stage1 pit and Innes Mills Stage 4 pit. In mid 2008 the FRUG mine main access decline passed through the Panel 2 Deeps area and intersected 45m @ 3.40 g/t which loosely confirmed the low angle concordant lode interpretation. As the FRUG mine development into the Panel 2C and 2D stoping blocks was completed a further 41 diamond drill holes for 1,572.5m were drilled as shown on Figure 7.8.

Figure 7.8: Frasers Underground Panel 2 Deeps Drill Hole Location



On the basis of the drilling a resource estimate was completed in July 2009 and preliminary mine design work commenced.

Panel 2 Deeps remains open to the south and south east. The diamond drilling of potential extensions will be completed once the Panel 2E stoping block is completed.

The Footwall Fault has been intersected in only a few exploration drill holes beneath Panel 2 due to the depth of this structure. In 2009 a diamond hole was drilled from 2C4 drive to the Footwall. This hole showed the Footwall to be 110m below the Hangingwall which is the expected location. As the FRUG mine develops further holes will be drilled to continue to confirm the location of the Footwall Fault.

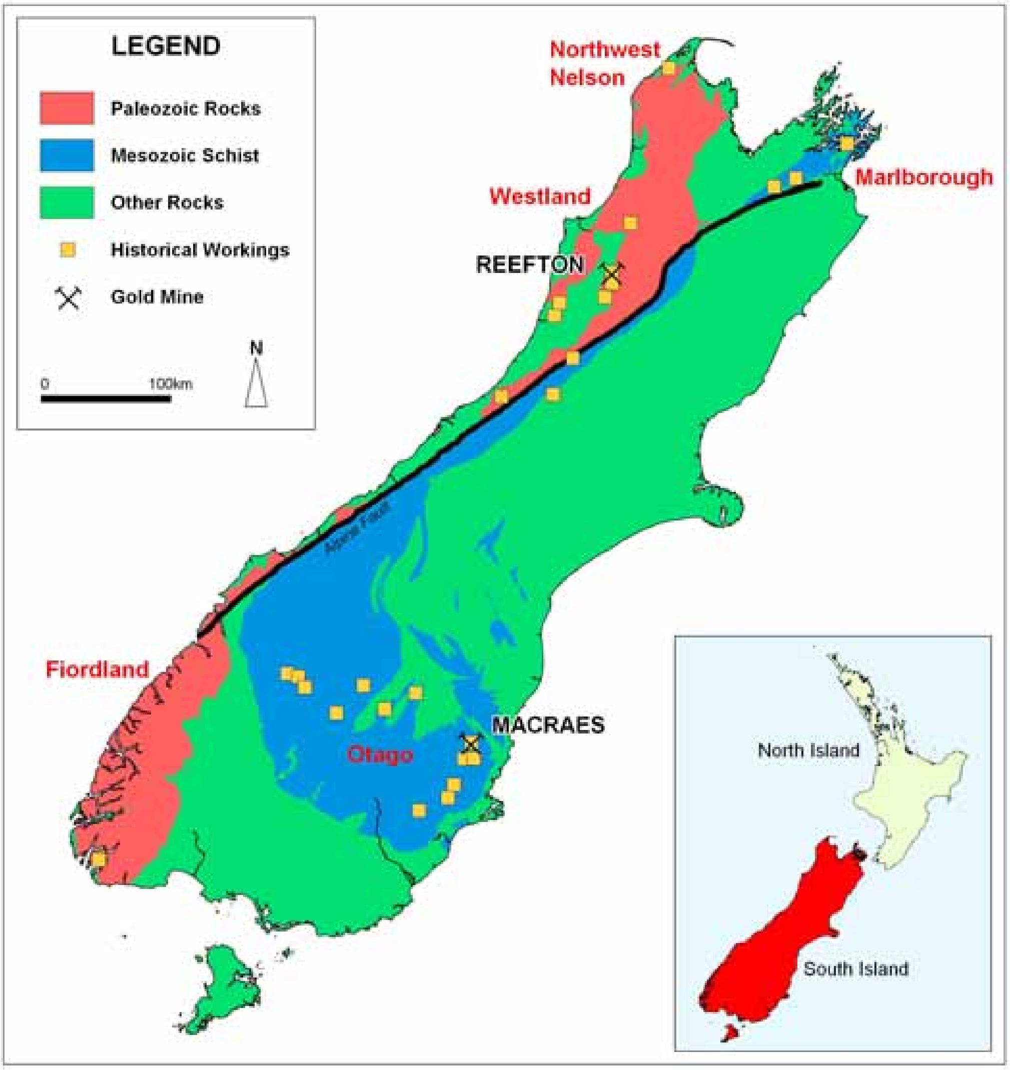
8 DEPOSIT TYPES

8.1 Orogenic Gold Deposits

The Macraes deposit is an example of an orogenic style gold deposit. This style of deposit is recognized to be broadly synchronous with deformation, metamorphism, and magmatism during lithospheric-scale continental-margin orogeny (Groves et al., 1998). Most orogenic gold deposits like Macraes occur in greenschist facies rocks. Orogenic deposits typically formed on retrograde portions of pressure-temperature time paths during the last increments of crustal shortening and thus postdate regional metamorphism of the host rocks (Powell et al., 1991 and references therein). Orogenic deposits can be subdivided into epizonal, mesozonal, and hypozonal based on pressure-temperature conditions of ore formation. The Macraes deposit falls into the mesozonal category with mineralization having occurred near to the brittle-ductile transition at about 300°C.

In orogenic deposits the association between gold and greenschist grade rocks is commonly thought to be related to: 1) the large fluid volume created during the amphibolite and/or greenschist transition and released into the greenschist zone; 2) the structurally favourable brittle-ductile zone that lies just above this transition; 3) fluid focusing and phase separation that are most likely to occur as fluids ascend into the greenschist pressure-temperature regime and/or gold solubility shows a sharp drop under greenschist facies temperatures (Phillips, 1991). Fluid migration along fault-fracture networks was likely to be driven by episodes of major pressure fluctuations during seismic events.

Figure 8.1: Orogenic Gold Deposits in New Zealand

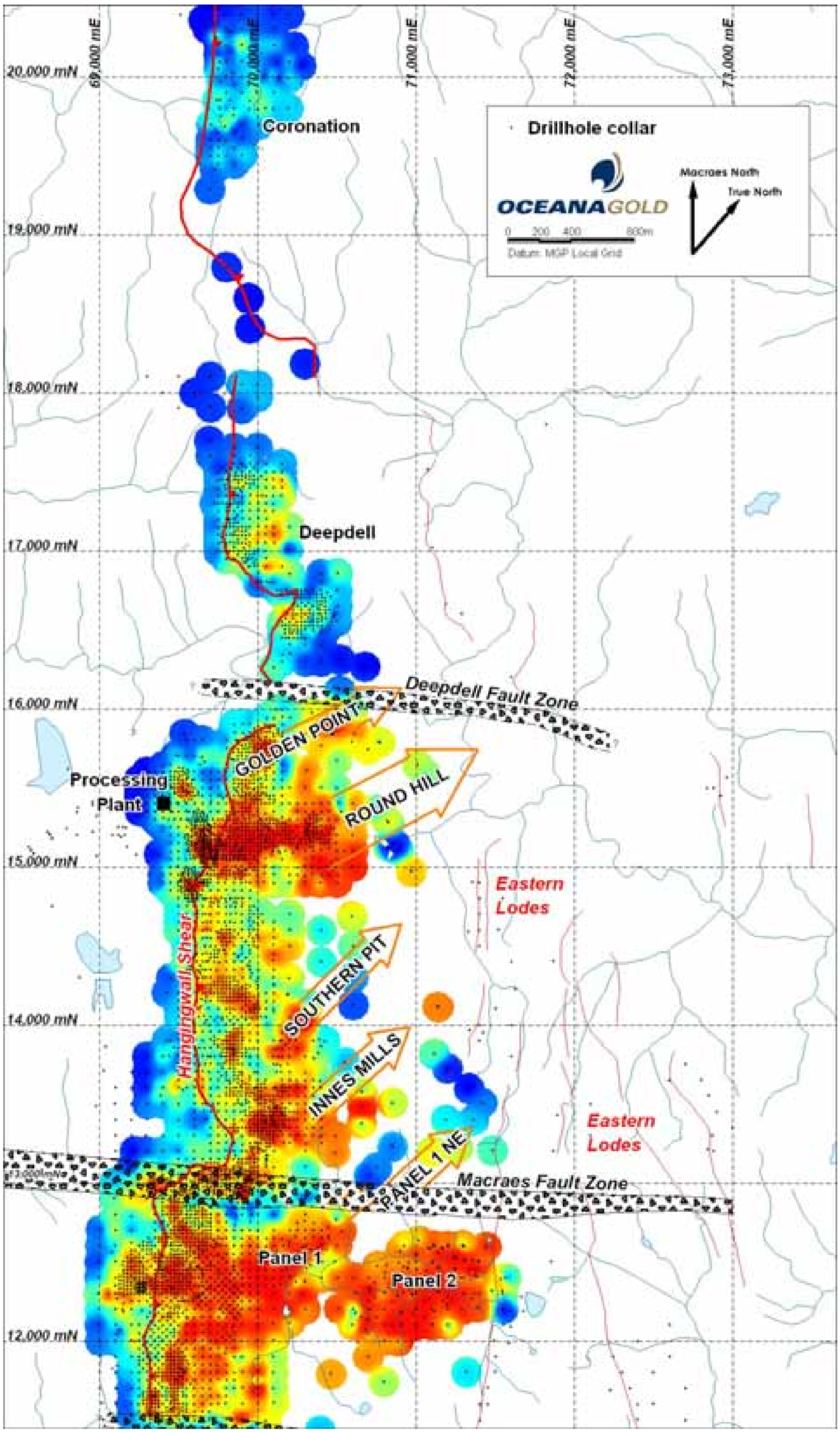


9 MINERALIZATION

9.1 Mineralized Zones

The mineralization at Macraes is principally developed within the gently northeast dipping HMSZ, though anomalous grades are also recorded in narrow, steeply dipping quartz veins locally occurring in the hangingwall schists, collectively known as the Eastern Lodes (Figure 9.1). Mining to date has occurred along a continuous strike length of 6km in numerous staged open pits, three smaller discrete satellite pits immediately to the north, and at Golden Bar, approximately 7km to the south.

Figure 9.1: Grade Distribution along HMSZ (gram*metre)



Within the shear zone, mineralization is constrained between the Hangingwall Shear and the Footwall Fault. Schists above the Hangingwall Shear and below the Footwall Fault are barren. Economic mineralization is typically restricted to the upper part of the HMSZ. The Hangingwall Shear, which varies from 1m to >30m in thickness contains the most continuous and consistent mineralization. This zone is locally underlain by extensive but low grade stockwork zones which may be developed over a width of up to 100m.

Higher grade zones of mineralization within the shear zone form tabular shoots that may have strike lengths of >300m and extend up to 800m down-dip (i.e. Frasers and Round Hill). These zones are observed to trend towards the north, oblique to the shear zone dip direction. This orientation is interpreted to be due to the interaction of the HMSZ with folds within the host schist units, creating a preferred lineation direction for mineralization.

Mineralization distribution is broadly consistent along the HMSZ but shows considerable variability in grade, width, continuity and geometry at mine-scale. This variability is attributed to the local development of the HMSZ structure during mineralization and the influence of host rock lithology, particularly with respect to competency contrasts.

9.2 Mineralization Types

The following four types of mineralization occur within the HMSZ at Macraes (Mitchell et al., 2006):

- Mineralized schist. This style of mineralization involved hydrothermal replacement of schist minerals with disseminated sulphides and microcrystalline quartz. Mineralization was accompanied by only minor deformation.

- Black sheared schist. This type of schist is pervaded by cm to mm scale anastamosing fine graphite and sulphide bearing microshears. This type of mineralization is typically proximal to the Hangingwall Shear. Scheelite mineralization occurs in the silicified cataclastic shears.

- Shear-parallel quartz veins. These veins lie within and/or adjacent to the black sheared schist, and have generally been deformed with the associated shears. The veins locally cross-cut the foliation in the host schist at low to moderate angles. Veins are mainly massive quartz, with some internal lamination and localized brecciation. Sulphide minerals are scattered through the quartz, aligned along laminae and stylolitic seams. These veins range from 1cm to >2m. . Scheelite mineralization is associated with quartz veining in some areas.

- Stockworks. These veins occur in localized swarms that are confined to the Intrashear Schist. Individual swarms range from c. 100 to 2000 m2 in area and consist of numerous (10 – 100) subparallel veins. Most of these veins formed subperpendicular to the shallow east dipping shear fabric of the Intrashear Schist. Stockwork veins are typically traceable for 1-5m vertically with most filling fractures that are 5 – 10cm thick, but can be up to 1m thick. Swarms of stockwork veins within the Intrashear Schist were lithologically controlled by the dimensions and locations of more competent pods of Intrashear Schist.

Gold is closely associated with pyrite and arsenopyrite in all of the above styles of mineralization. Rarely free gold up to 300µm occurs in quartz veins, but most gold occurs as 1-10µm scale blebs hosted in and near sulphide grains (Angus, 1993).

Scheelite mineralization is associated with gold mineralization and quartz veining and displays complex crosscutting relationships. McKeag (1987) documented its occurrence in at least three veining generations.

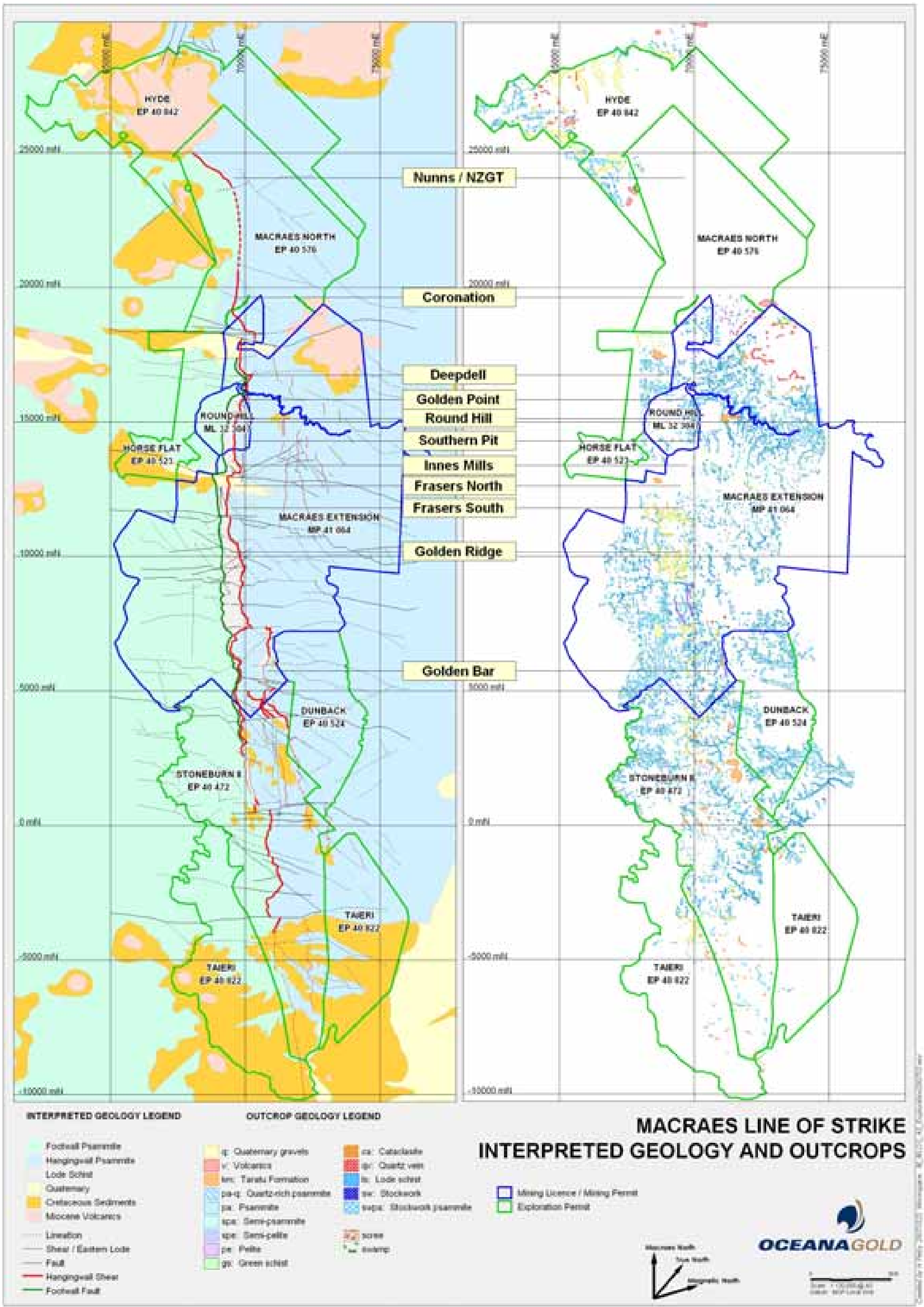
10 EXPLORATION

10.1 Geology

10.1.1 Geological Mapping

Detailed geological mapping has been completed along the strike of the HMSZ. Fact and interpreted geology is shown in Figure 10.1.

Figure 10.1: Macraes Interpreted and Outcrop Geology



10.2 Geophysics

10.2.1 Seismic Surveys

In 1991, a seismic refraction analysis was carried out at Frasers under Brown et al. (1991). Four lines, approximately 500m apart, were placed across the alluvial flat adjacent to the Macraes road with another two lines along the valley. These enabled the location of major fault zones, and provided data for modelling surfaces on the base of the alluvium and the base of the weathered rock.

During 1994, the Institute of Geological and Nuclear Sciences (IGNS) undertook a trial seismic survey of three lines across the mineralized shear zone at Southern Pit and Innes Mills to indicate whether the seismic method could successfully delineate the main shear zone between the upper psammite and the pelite which hosts strong concordant shearing (Woodward and Ravens, 1994). While reflections at depth identify the position of the Northern Gully Fault (Woodward and Ravens, 1994), the results indicated that the seismic method was not appropriate and boundaries and faults defined by the interpretation of seismic data do not appear convincing.

A further seismic study was completed in late 2004 by the University of Otago, in collaboration with the IGNS and Oceana (Leslie et al., 2005). A 2.6km seismic reflection survey composed of two parallel lines, offset by 200 m and overlapping by 400 m, was carried out at the FRUG area. The lines were oriented parallel to the HMSZ dip-direction, crossing from the confirmed underground gold resource at Frasers (Panel 2) to a lesser known region of the Eastern Lodes. Numerous drill-holes and the Frasers mine exposures define the geology beneath Line 1, allowing for reliable comparisons between seismic and geological datasets, and a well-constrained interpretation. Further east, beneath Line 2, the geology was less well known.

Bands of well-defined reflections up to 90m wide and more subtle, discontinuous reflections were “mapped” throughout the seismic sections. Dipping discontinuities and truncations of reflections were also observed. A distinct upper seismic unit, containing mostly east-dipping reflections, was resolved from a lower unit with variable east and west-dipping reflections. Reflections appear to mimic the orientation of schist fabric in the Hangingwall and Footwall of the HMSZ, and the upper and lower seismic units correlate with the general structure of the HMSZ. The transition zone from upper to lower seismic units dips gently east but its position is subjective and internal details are variable. Structures within the HMSZ could not be positioned definitively without prior knowledge.

10.2.2 Electromagnetic Survey

The Institute of Geological and Nuclear Sciences, during 1994, also carried out an electromagnetic (EM) modelling exercise at Innes-Mills and Frasers North (Caldwell, 1994). Down-hole resistivity logging was carried out in several holes at Innes-Mills and Frasers North so that four EM exploration techniques, listed as follows, could be modelled: LOTEM (long offset time domain electromagnetic soundings); CSAMT (controlled source audio frequency magneto-tellurics); TEM (time domain EM soundings); and coplanar loop-loop such as is used in helicopter electromagnetics (HEM) surveys (Funnell and Caldwell, 1994).

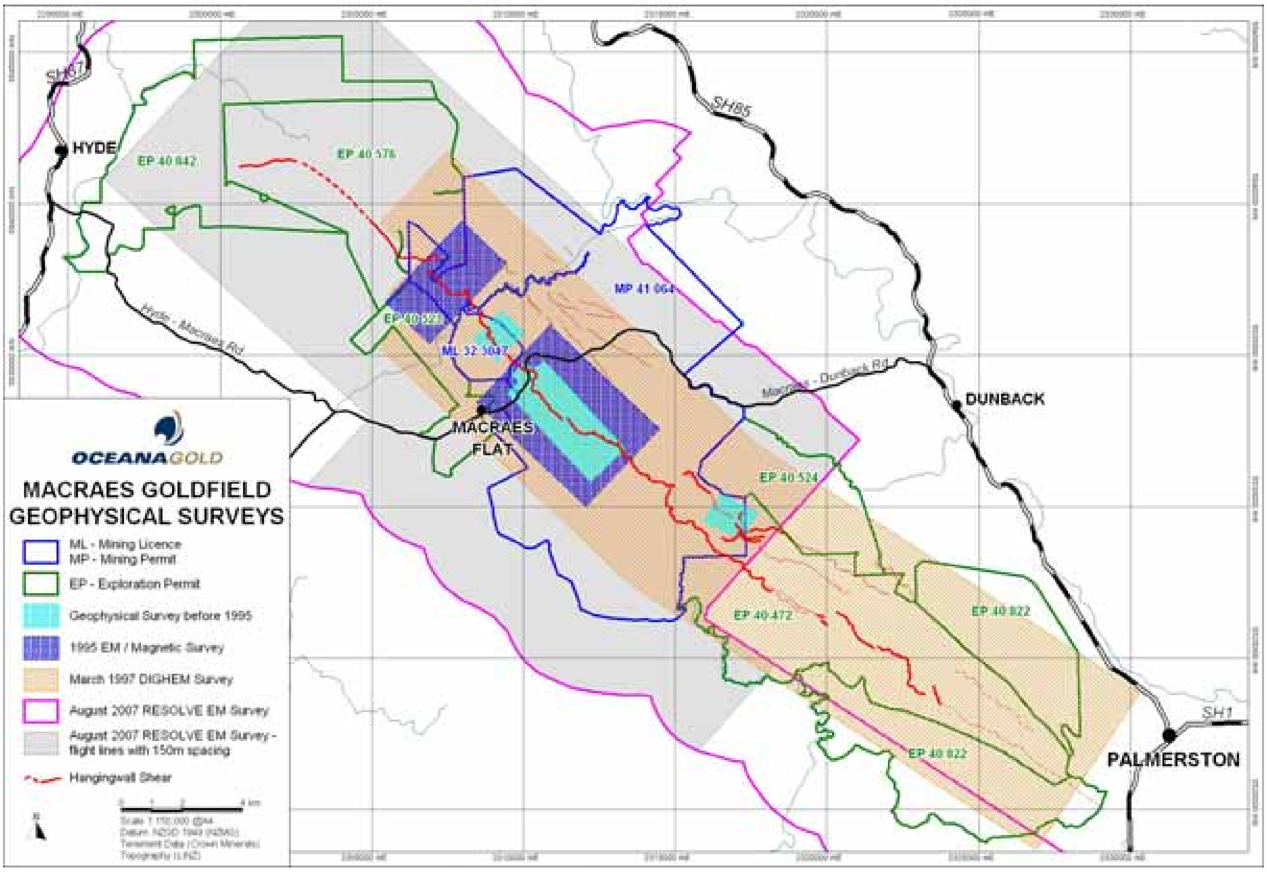
The results show that apart from the TEM method, where limitations of the 3D modelling programme prevented a realistic assessment, all the methods appear to be able to resolve a body with a resistivity contrast of 2 or greater. The theoretical results suggest that either a CSAMT or helicopter EM survey is the best option for exploration at Macraes Flat (Caldwell, 1994).

10.2.3 Magnetics and DIGHEM

A digital helicopter electromagnetic (DIGHEM) airborne geophysical survey was flown for MMCL in December 1995 by Geoterrex Limited. Two blocks were flown; one on the north of Round Hill, in the Deepdell area (151 line km), and one to the south of Innes Mills (212 line km). The DIGHEM survey areas flown in December 1995 are shown on Figure 10.2

East-west (Macraes Grid), flight lines were flown at 50m spacing, with a terrain clearance of 40m for the magnetometer sensor and 30m for the electromagnetic sensor. A Sercel real-time differential global positioning system (GPS) provided in-flight navigation control and a video camera recorded flight path terrain passing beneath the helicopter. A DIGHEM electromagnetic system was used.

Figure 10.2: Macraes Geophysical Survey Locations



Electromagnetic data was acquired every 3m along survey lines, testing the 450, 900, 5500, 7,200 and 56,000Hz frequencies. Magnetic data was captured utilising a caesium split-beam total field magnetic sensor with sample intervals of 0.1 seconds.

The DIGHEM survey was interpreted as having the ability to map the HMSZ larger cross-structures and stratigraphic units in the hangingwall succession. However, the system was likely to be limited to about 50 to 75m depth. Aeromagnetic data collected with the DIGHEM survey contained subtle magnetic characters capable of identifying structures and lithological variation not evident in existing surface mapping.

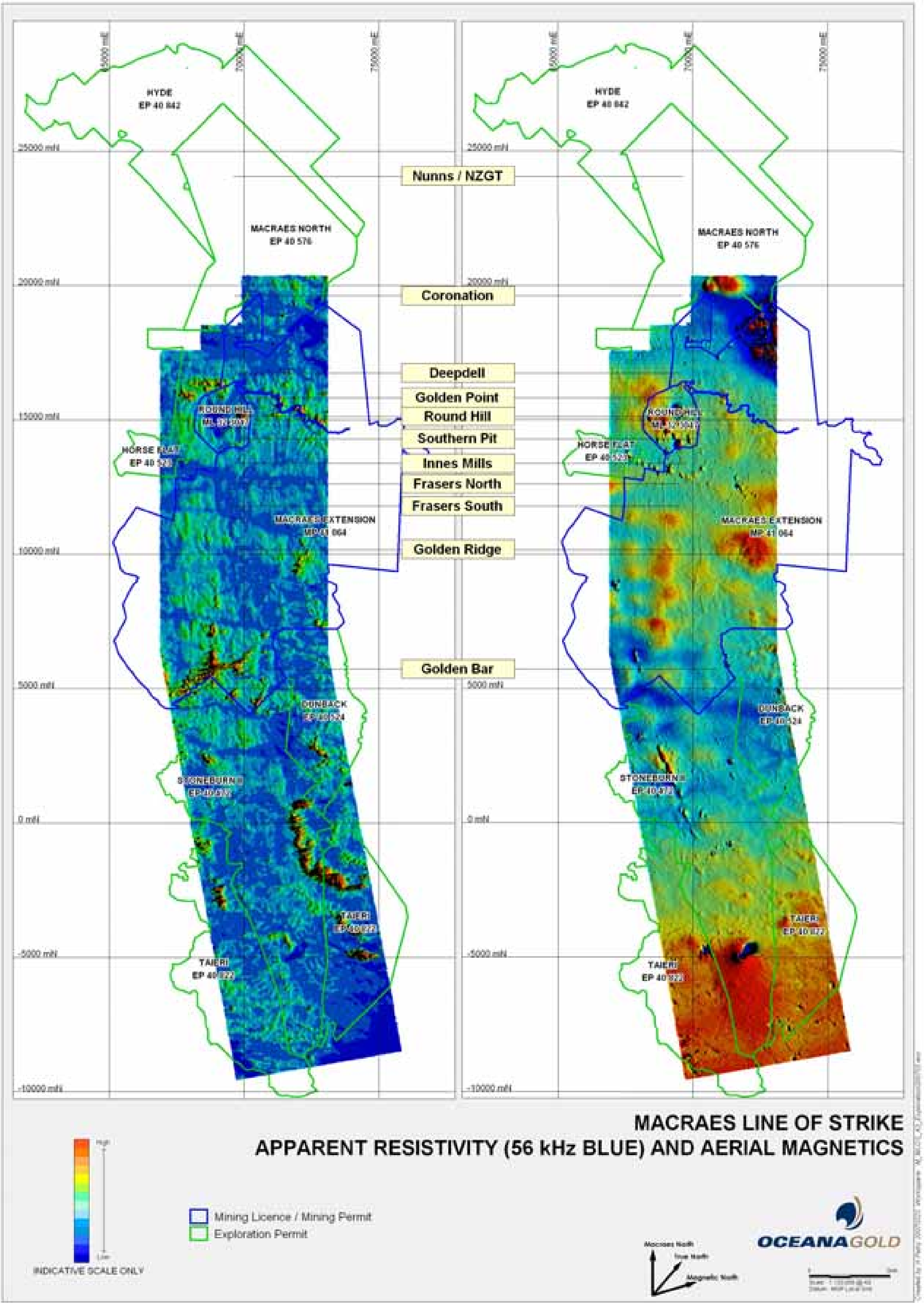
A follow-up DIGHEM airborne geophysical survey was flown for MMCL, by Geoterrex during February and March 1997. Two blocks were flown, each 6km wide. One block, called the Macraes Block, covered from the north of Round Hill, in the Deepdell area, south to the Golden Bar area (1,320 line km), The other block, called the Stoneburn Block, covered from the south of Golden Bar to the southern end of Macraes tenement coverage; the southern boundary of the Stoneburn 3 Extension (1,920 line km). The survey areas flown are shown on Figure 10.2 and processed images in Figure 10.3.

Survey specifications were as follows. East-west (Macraes Grid), flight lines were flown at 50m spacing, with a terrain clearance of 40m for the magnetometer sensor and 30m for the electromagnetic sensor. A Figuro Starfix real-time differential GPS system provided in-flight navigation control and a video camera recorded flight path terrain passing beneath the helicopter. A DIGHEM electromagnetic system was used. The instrument array was towed using an Aerospatiale AS350B Ecureuil (Squirrel) turbine helicopter.

Electromagnetic data was acquired every 3m along survey lines, testing the 450, 900, 5500, 7,200 and 56,000Hz frequencies. Magnetic data was captured utilising a caesium split-beam total field magnetic sensor with sample intervals of 0.1 seconds.

Magnetic and DIGHEM data was interpreted by Craven (1998) of Southern Geoscience Consultants Pty Ltd.

Figure 10.3: Macraes DIGHEM Images



10.2.4 RESOLVE EM and Magnetics

In August 2007 Fugro completed an extensive airborne electromagnetic (EM) and magnetic geophysical survey over Otago using a helicopter-borne "RESOLVE$_{TM}$" EM system combined with a magnetometer for Glass Earth Gold Limited. The survey included coverage over most of the Oceana tenement package at Macraes, for which Glass Earth agreed to supply all data collected within Oceana tenements and a 2.5km buffer zone in every direction.

The survey was flown along east-west (Macraes Grid) flight lines with 300m spacings, infilled to 150m spacing in a central zone (Figure 10.2). Terrain clearance averaged 30m. Electromagnetic data were acquired every 3m and testing the 400, 1800, 8,200, 40,000, 140,000 Hz frequencies.

The raw data plus resistivity, calculated vertical gradient and total field magnetic maps were finally provided to Oceana by Glass Earth September 2008. However, deliverable items not provided include vertical differential conductivity sections for 10% of the lines, multi-channel stacked profiles, logistics report and flight path videos.

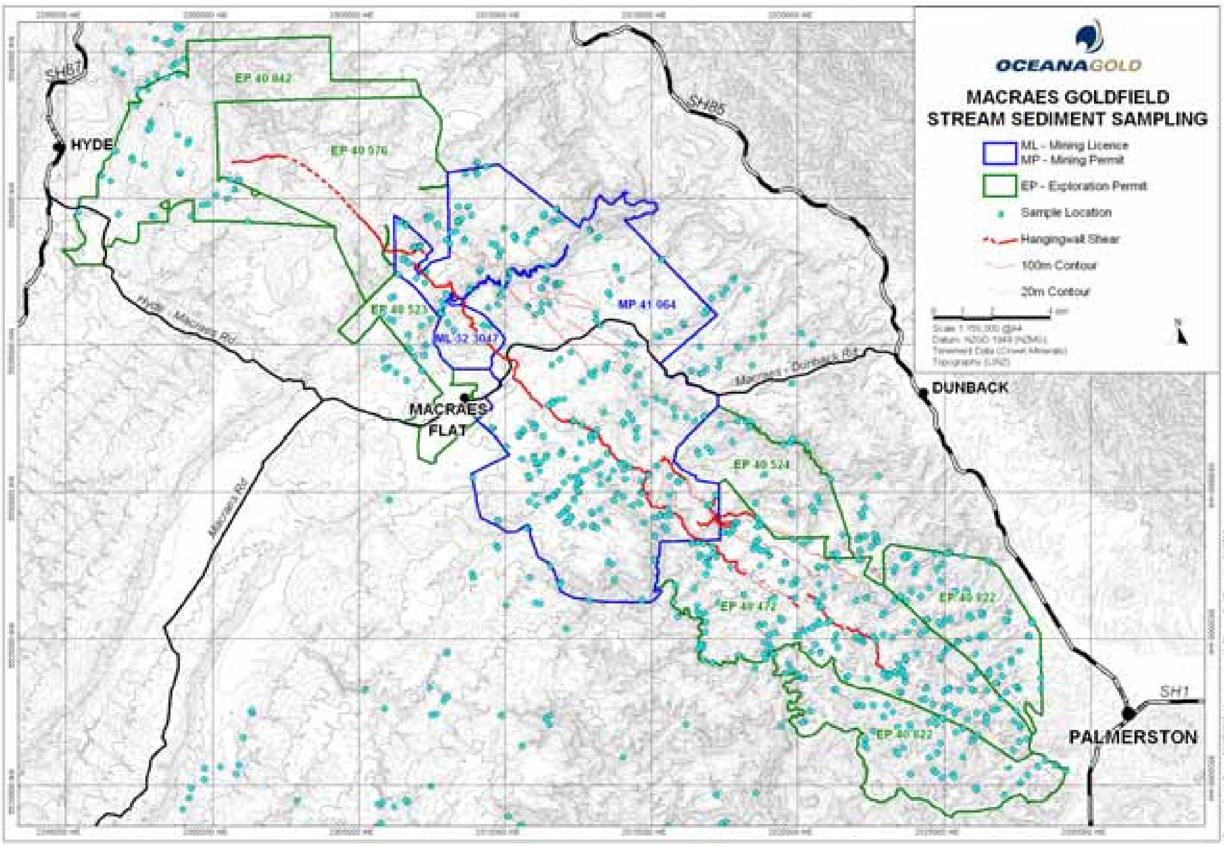
During October 2008 Southern Geoscience Consultants Pty Ltd converted the 1997 DIGHEM data from Macraes Grid to NZMG to directly compare with the Glass Earth EM survey. In the areas covered with both surveys there was good correspondence in spatial location and anomalies but the 1997 data resolved more detail.

10.3 Geochemistry

10.3.1 Stream Sediment Sampling

Further stream sediment sampling was undertaken in 1991 (Grieve, 1991), in 1994 (Bleakley, 1994) and during 1995. As of June 30, 1997, 803 BLEG (bulk leach extractable gold) stream sediment samples had been collected on the Macraes Project area to complete first pass sampling and infill anomalous catchments. 241 total sediment fine fraction (TSFF) stream sediment samples were also collected. The location of all stream sediment samples collected on the project is shown as Figure 10.4.

Figure 10.4: Macraes Stream Sediment Sampling Locations



Bulk leach extractable gold (BLEG) samples consisted of approximately 2 to 3kg (dry weight), of -2mm sediment, collected from multiple points ranging from trap sites in active creek channels to over bank fines. Many samples were collected from creeks with low water flow and small active sediment content. Sediment from these creeks consisted of organic-rich fine silts and clays trapped by vegetation. Recent orientation sampling from creeks draining known mineralization (i.e., the Frasers and Golden Ridge Prospects), produced assays from 78.7 to 3,353ppb gold and 40 to 170ppm arsenic.

Total Sediment Fine Fraction (TSFF) samples were also collected for the first time during early 1997. The samples consisted of 1 to 2kg of -1mm sediment collected from the same trap site as the BLEG samples. These samples were then analysed for a multi-element suite using the Inductively Coupled Plasma (ICP) analytical technique.

10.3.2 Soil Sampling

Soil sampling of B horizon soils using a hand or motorised hand auger has been carried out over a large part of the Macraes permit areas. Samples are routinely analysed for arsenic, with some samples also analysed for gold, tungsten and antimony. Arsenic is interpreted as the most reliable path finder element.

In total, approximately 17,000 soil samples have been collected across the Macraes permit areas. The location of all soil samples collected on the project is shown as Figure 10.5.

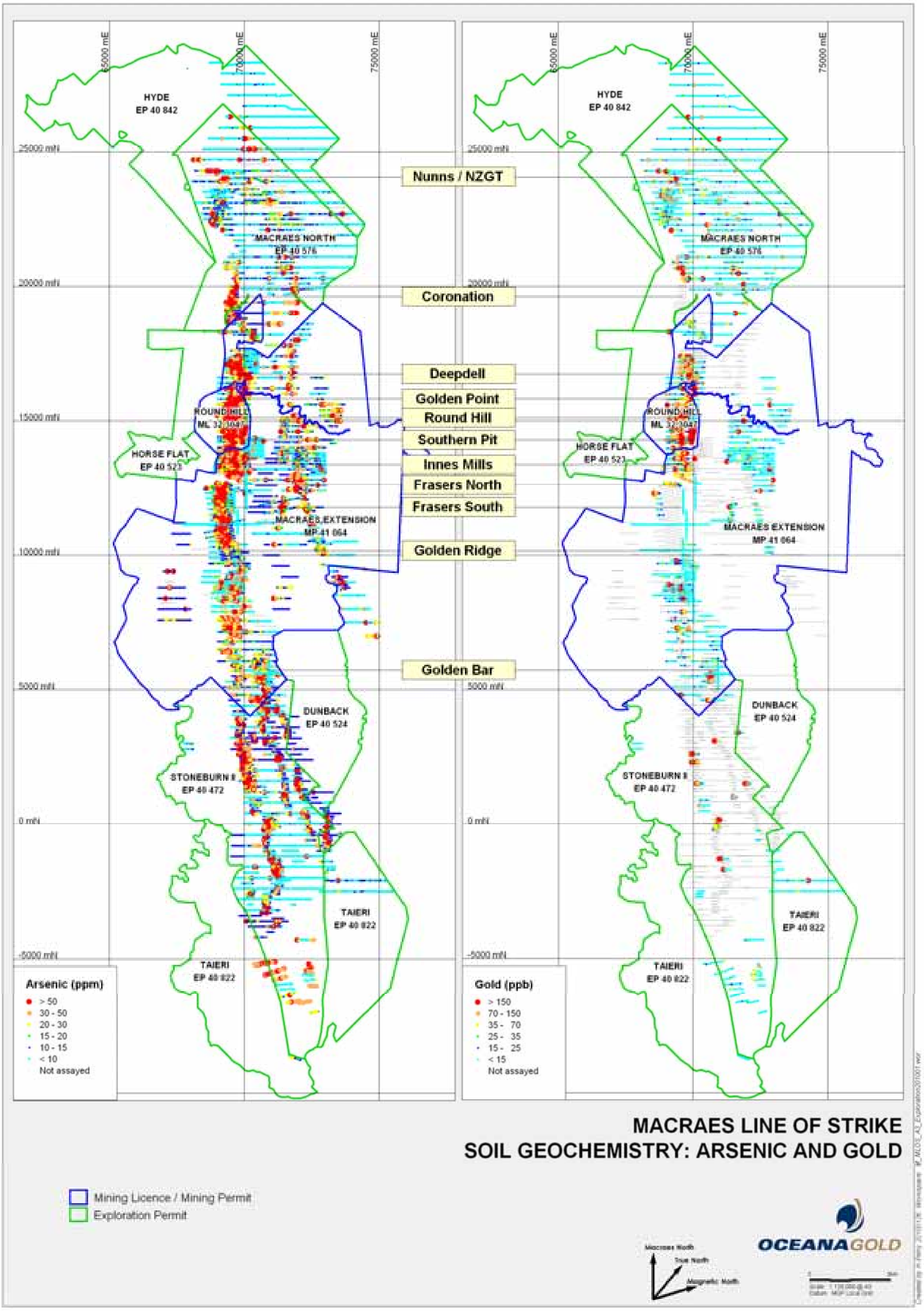
For conventional sampling, a 2kg unsieved sample is collected from 0.25 to 1m depth using an auger at each station. Samples usually reached the soil/bedrock interface and consisted of B and C horizon material.

During 1997, two new soil geochemistry techniques were trialed. A two-phase orientation survey testing the Mobile Metal Ion (MMI) technique was conducted, with a total of 604 samples collected. The technique is based on the location of ‘blind’ mineralization through the detection of highly mobile ionic species, including gold, which is shed from mineralization at depth and moves up through the substrate to become weakly bound to soil particles. A very weak solute is used followed by ICP-MS analysis. The results of the orientation were inconclusive and the programme was suspended.

In addition, considerable work has been conducted on determining whether ICP-OES multi-element suites are more effective at discriminating lithological variations and highlighting mineralization at the Macraes Project. Work included a 607 sample orientation survey, and an 848 sample follow up survey taken over various areas of the line of strike.

Since November 2008, all soil samples have been analysed by ICP-MS at SGS Waihi for Au, As, Sb and W. This has included an extensive soil program over the eastern parts of the Macraes North and Hyde exploration permits.

Figure 10.5: Macraes Soil Sample Locations



10.4 Trenching

It is estimated that approximately 16,500m of trenches has been excavated at Macraes, with approximately 5,200 trench rock samples collected.

Trenches are mapped and rock chip sampled, with samples traditionally analysed for gold ± arsenic, ± tungsten. In general, the soil profile is shallow in the Macraes area allowing trenching to be undertaken by light (12 tonnes), excavators in most areas. Although stream beds and areas of extensive alluvial cover present some difficulties, trenching has proven to be an excellent exploration tool for geological mapping and geochemical sampling.

Trenches are mapped at 1:100 scale with horizontal channel samples collected over geological intervals from 0.5 to 6m. Samples were submitted to the AMDEL laboratory on site for gold, arsenic and tungsten analysis.

10.5 Remote Sensing

In 1994, MMCL purchased a 10m resolution, monochrome 1990 Spot image of the eastern Otago region.

Digital satellite imagery over the Macraes Operation was purchased from Digital Globe Limited in July 2005, March 2006, March 2007, January 2008 and June 2009. The Quickbird satellite imagery is in the visible spectrum, with a resolution of 5m.

10.6 Aerial Photography

Colour aerial photography was flown by New Zealand Aerial Mapping Limited during January 1996. Photography was captured at a nominal scale of 1:30,000. 1:5,000 colour enlargements were produced as an aid to programme planning, geological mapping and interpretation. 1:5,000 black and white orthophotographs have been rectified differentially to the Macraes local grid.

Updated colour aerial photography was flown over the Macraes area in March 2005 by Terralink International Limited. Images were supplied as 0.5m resolution digital orthophotographs on the Macraes local grid.

10.7 Exploration Statement

Exploration surveys and investigations of the Macraes area detailed above have been carried out by Oceana, except where a contractor or consultant

11 DRILLING

11.1 Summary

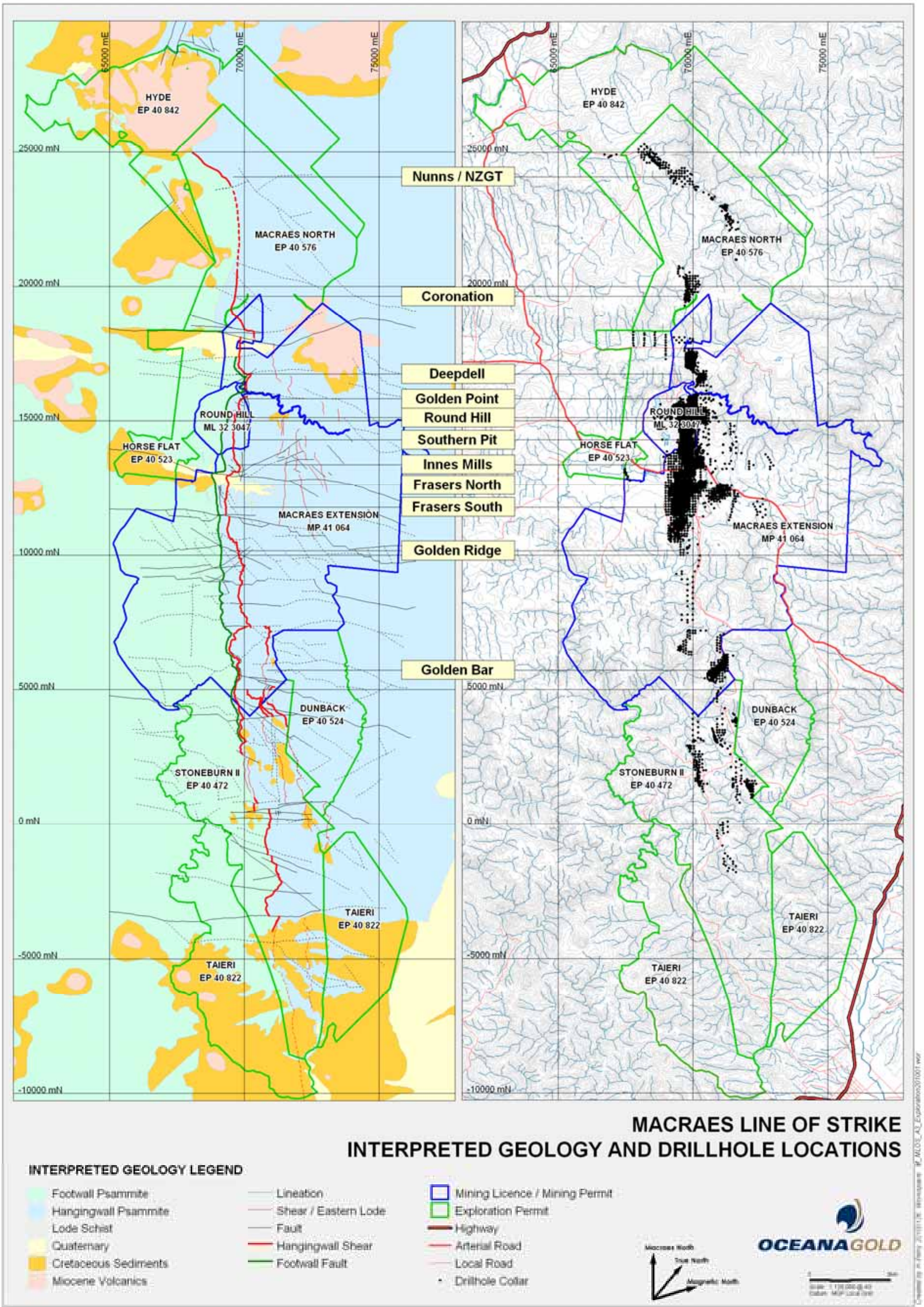
At December 2009 approximately 729,000m in about 5,650 holes have been drilled in the Macraes Goldfield (Figure 1.1). The majority of holes (>90%) completed to delineate open pit resources are RC percussion drilling, with limited diamond drilling confirmation. Diamond drilling tails have also been completed where groundwater inflows degraded RC percussion sample quality. Drilling on FRUG is dominated by diamond drilling due to the depth of mineralization. Forty-nine aircore holes sample the tailings dams to assess the scheelite content.

The Mineral Resource inventory is based on the results of 663,793 metres of drilling in 4,949 holes used in nine resource estimates areas. Four companies, BP Minerals, Homestake, BHP and Oceana have drilled the holes with over 90% of the drilling being completed by Oceana. A breakdown of drilling by resource area as at Dec 2009 are summarised in Table 11.1.

Table 11.1: Drilling Summary by Resource Area

Resource Area	Holes used in the Current Resource Estimates	
	Holes	Metres
Coronation	99	8143
Deepdell	330	29,048
Golden Point	13	2,971
Round Hill	1,456	163,800
Innes Mills	971	121,829
Frasers	1,092	168,754
Frasers Underground	629	126,722
Golden Bar	277	39,047
Taylors	82	3,479
Total	**4,949**	**663,793**

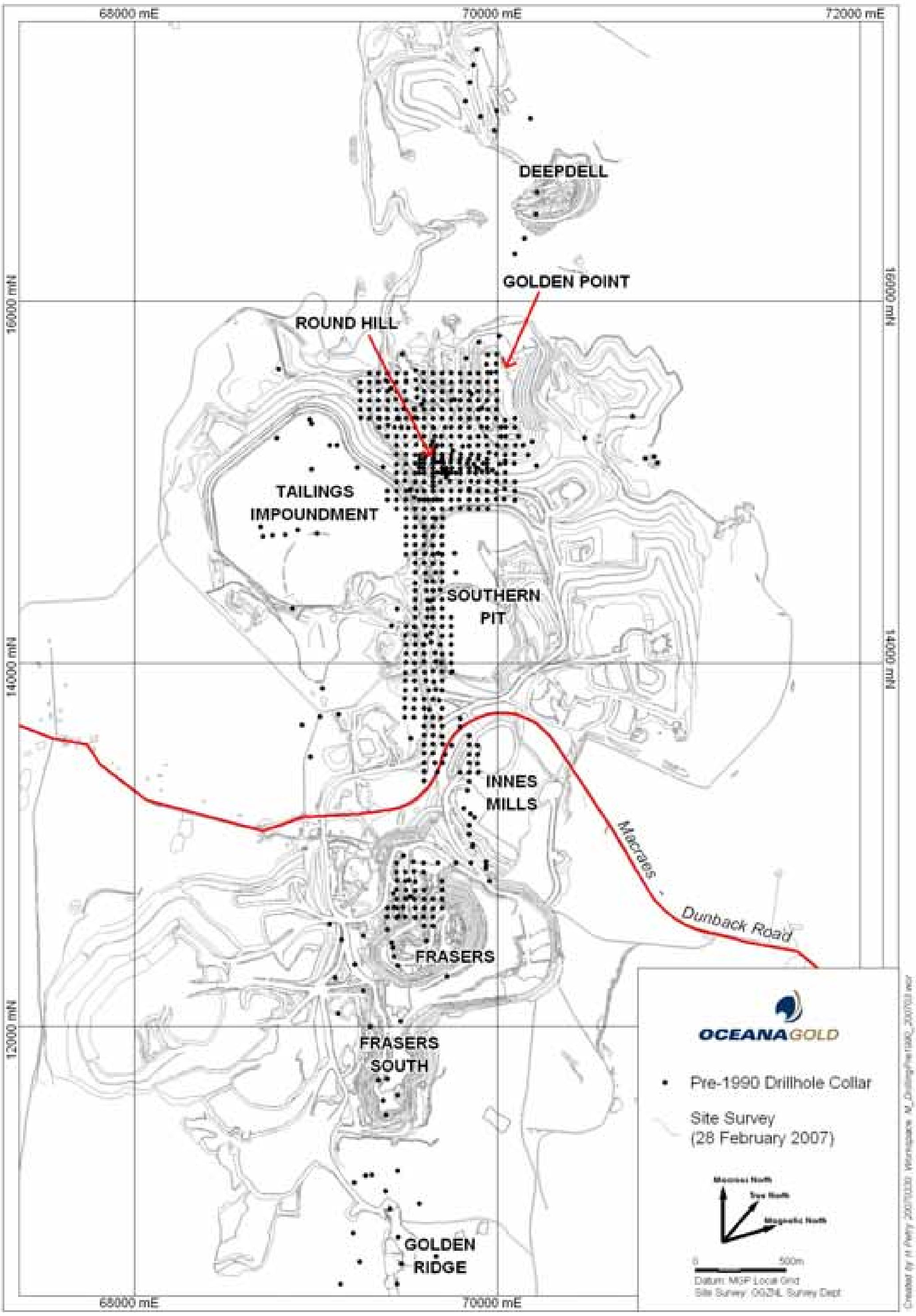
Figure 11.1: Macraes Drill Hole Locations



11.2 Historical Drilling

Limited information is available regarding the specific details of drilling prior to 1990. Drilling was principally completed on the near surface parts of Golden Point, Round Hill, Southern Pit, Innes Mills and Frasers (Figure 11.2). All resources associated with this drilling, with the exception of Round Hill, have been mined.

Figure 11.2: Drill Hole Locations Prior to 1990



11.3 Oceana

Details of the drilling completed by Oceana post 1990 is shown in Table 11.2. Where known, the details of the drilling method, contractors used to drill holes and the type of drill rigs has been supplied. Historical drilling (pre-1990) completed by Homestake and BHP minerals has been included where available.

Table 11.2: Macraes Drilling Summary

Year	Hole Type	No. Holes	No. Metres	Contractor	Drill Rig	Prospects
1984	DD	15	2,163	unknown	unknown	Round Hill
1985	DD	52	6,687	unknown	unknown	Round Hill, Fergussons, Golden Point, Deep Dell, Frasers, Maritana, Ounce
	RAB	63	3,343	unknown	unknown	Round Hill, Fergussons, Deep Dell, Innes Mills
1986	DD	9	1,192	unknown	unknown	Round Hill, Maritana, Frasers
	RC	36	2,610	unknown	unknown	Round Hill, Golden Ridge, Frasers, Innes Mills
	RAB	2	152	unknown	unknown	Tates
1987	DD	5	240	unknown	unknown	Round Hill
	RC/DD	7	1023	unknown	unknown	Round Hill
	RC	272	19,263	unknown	unknown	Round Hill, Golden Bar, Ounce, Macraes Nth, Frasers, Southern Pit, Golden Point,
1988	DD	3	870	unknown	unknown	Round Hill
	RC/DD	22	3,201	unknown	unknown	Round Hill, Golden Point
	RC	167	13,469	unknown	unknown	Round Hill, Frasers, Southern Pit, Golden Point, Innes Mills
	RAB	119	1,643	unknown	unknown	Stoneburn, Round Hill, Innes Mills
1989	DD	6	205	unknown	unknown	Round Hill
1990 - 1991	RC	378	19,884	unknown	unknown	Round Hill, Southern Pit, Innes Mills, Frasers
	DD	11	225	unknown	unknown	Southern Pit, Innes Mills, Frasers
	Met	4	52	unknown	unknown	Round Hill
	Open hole	4	64	unknown	unknown	Innes Mills, Frasers, Golden Ridge
1992	Open hole	69	1,625	unknown	unknown	Coronation, Macraes North
	RC	247	28,499	unknown	unknown	Round Hill, Southern Pit, Golden Point, Deepdell, Macraes North
1993	DD	2	412	unknown	unknown	Round Hill
	RC/DD	1	436	unknown	T685W	Frasers
	RC	40	7,152	unknown	unknown	Round Hill, Frasers
	Open Hole	1	24	unknown	unknown	Golden Ridge
1994 - 1997	DD	56	6,634	Ausdrill	unknown	Deepdell, Innes Mills, Frasers, Ounce, Golden Bar
	RC/DD	185	47,273	Ausdrill	Schramm25M, T685W, IR-T4, UDR650, UDR1000, HYDRILL	various
	RC	2,225	329,603	Ausdrill	Schramm25M, T685W, IR-T4, UDR1000	various
	RAB	18	589	unknown	unknown	Macraes North
1998 - 1999	DD	6	370	unknown	unknown	Southern Pit
	RC/DD	21	3,720	Ausdrill	T685W	Innes Mills, Frasers
	RC	542	42,163	unknown	T685W	various
2000 - 2001	RC/DD	53	12,377	unknown	T685W	Innes Mills, Frasers
	RC	69	6,663	unknown	T685W	Deepdell, Golden Point, Innes Mills, Golden Ridge, Macraes North
	RC	3	200	McNeil Drilling	UDR650	Southern Pit
	DD	1	40	unknown	unknown	Round Hill
	RC	82	7,460	unknown	T685W, UDR650	Round Hill, Innes Mills, Frasers, SP18, SP22, Coronation, Deepdell

Year	Hole Type	No. Holes	No. Metres	Contractor	Drill Rig	Prospects
	RC/DD	7	2,747	Ausdrill	T685W	Frasers
2002 - 2003	RC/DD	7	805	Major Pontil	T685W	Golden Bar
	RC	160	8,808	Major Pontil	T685W	Golden Ridge, Golden Bar, Frasers
	RC/DD	29	11,583	Major Pontil	T685W, UDR650, Schramm25M	Frasers
	RC	178	10,524	Major Pontil	Schramm25M	Stoneburn, Eastern Lodes, Macraes North, Golden Ridge
2004	RC/DD	116	52,890	Boart Longyear	UDR650, UDR1000, CS1000	Frasers
	RC	23	1,592	Washingtons	SchrammT660H	Deepdell, Frasers
2005	DD	21	5,629	Boart Longyear	LF90, UDR650, UDR1000	Round Hill, Golden Point, Frasers
	RC/DD	61	24,757	Boart Longyear, Washingtons	LF90, UDR650, UDR1000, CS1000, FOREMOST	Innes Mills, Frasers
	RC	18	930	Washingtons	CP650	Southern Pit
2006	RC	45	1,787	Washingtons	SchrammT660H	Golden Ridge, Frasers
	DD	2	402	Boart Longyear	UDR650	Frasers Geotechnical
2007	RC/DD	19	9,197	Boart Longyear	LF90, UDR650	Golden Point Extension, FRUG Panel 2
2008	AC	30	1,296	McNeill Drilling	Edson	Tailings Dams: MTD and SP11
	RC	44	3,785	Boart Longyear	UDR650	Golden Ridge, Coronation
	RC/DD	26	11,718	Boart Longyear	LF90, UDR650	Frasers Underground, Golden Point/Round Hill Extension, Trig 569, Coronation
	DD	10	601	Boart Longyear	Underground (LM45)	Panel 2 Deeps
2009	AC	47	1,342	McNeill Drilling	Edson	Tailings Dam (MTD) and Horse Flat
	RC/DD	21	7,510	Washingtons, Boart Longyear	Schramm T660H, UDR650, LF70	Round Hill, Back Road
	DD	21	3,071	Boart Longyear	Underground (LM45, LM75)	FRUG: Panel 2 Deeps, Panels E + F, Panel 2 Infill

11.4 Surveys

All drill hole collars were surveyed using the Macraes local grid to ± 10mm accuracy in easting, northing and elevation.

Prior to March 1994, down-hole deviation surveys were not completed on any of the RC percussion or percussion drill holes. For holes drilled since March 1994, down-hole deviation surveys have been attempted on all RC percussion holes that exceeded 50m in depth, using an Eastman single shot or multi-shot camera. Holes are generally surveyed at 50m intervals to the end of the hole.

Diamond holes are routinely surveyed every 25m to 50m. Current survey equipment is typically an electronic single shot or multi-shot camera. Survey information is routinely recorded in an *acQuire* geological database.

Aircore holes do not have down-hole surveys.

11.5 Logging Procedures

RC percussion and Horse Flat aircore program drill holes are geologically logged at one-metre intervals, with each metre being classified into one of thirteen summary rock codes listed in Table 11.3. Rock code classification is based on a combination of textural and mineralogical properties.

Diamond drill core is photographed and then geologically logged at one metre intervals using the same thirteen summary rock codes and additional detailed pre, post and syn-mineralization structure and mineralogy are recorded. The summary rock codes are plotted on cross sections and are used in combination with the assays to develop a geological interpretation, which usually consists of three elements.

These elements are the Hangingwall Shear, concordant lodes and stockwork. The Hangingwall and concordant lodes consist of a combination of Cataclasite, Quartz Cataclasite, Silicified Breccia and Lode Schist. In general the Hangingwall has greater proportion of cataclasite lithologies logged than the concordant lodes, which typically consist of more Lode Schist lithologies. The stockwork mineralization is identified on cross sections by a combination of Stockwork and Quartz vein lithologies.

Drill hole information is stored as hard copy drill logs and in an *acQuire* database. For holes prior to 1994 only collar, interval and assay information has been entered into the database, while for all holes from 1994 onward the database contains all logged information.

Aircore drilling holes on the tailings dams are geologically logged using two codes only: 'C' records the schist boulders and gravel used to build mattresses, causeways and embankment lifts; 'T' is used to record tailings material of fine-medium grained sand. The distinction is easily recognized by field technicians and the contacts are typically defined to within a decimetre by the drilling crew. The colour the tailings material is usually a monotonous grey although thin (<2m) horizons of yellow-brown oxidation staining are noted and can be correlated between holes.

Table 11.3: Summary of Rock Code Descriptions

Cataclasite	quartz poor (< 15%) dark-grey/black fine grained cataclasite.
Quartz Cataclasite	quartz rich (>15%) < 50% dark grey/black fine grained cataclasite.
Lode Schist	weakly sheared schist with minor cataclasite/brecciated quartz.
Silicified breccia	> 50% brecciated quartz veins. Generally associated with cataclasite.
Quartz vein	> 50% banded or milky quartz veins with no associated brecciation or cataclasis.
Stockwork	from trace to 50% banded or milky quartz veins with no associated brecciation or cataclasis and hosted by either pelitic or psammitic schist.
Alluvial	transported cover.
Fault	light to medium grey gauge or pug without sulphide and mineralized quartz, i.e., not associated with mineralization.
Pelite	massive to laminated med to dark grey mica-quartz-chlorite & graphitic schist
Semi-pelite	thinly laminated pelite and psammite with more than 50% pelitic layers.
Semi-psammite	inter-layered psammite & pelite with more than 50% psammitic layers > 1 cm thick.
Psammite	massive light grey-green quartz-felspar-mica-schist. 90% psammite.
Greenschist	light green/brown massive quartz-mica schist

11.6 Drilling Orientation

Drill holes at Macraes have typically been collared vertically, although most diamond drill holes targeting potential underground resources are started with an inclination of ca. -75º oriented towards the northwest. Down-hole survey information indicates that within a shallow depth (~100m) the holes can significantly deviate, generally veering perpendicular to the schist foliation and to the HMSZ orientation. Exceptions to this trend may occur where the foliation orientation has been disrupted, or where the schists are cut by later fault zones.

12 SAMPLING METHOD AND APPROACH

12.1 Introduction

The diamond drilling sampling approach has remained relatively constant over the life of the project while the sampling of the percussion drilling has changed dependant on the drilling method. A discussion of the sampling methods applied is provided below.

Drilling has typically been conducted on a regularly-spaced grid. Measured deviation of drill holes indicates that holes quickly trend sub-perpendicular to the host schist foliation direction and consequently drilling intersections typically represent the true width of the mineralized shear zone.

12.2 RC Percussion Sampling

The percussion drilling methods have varied substantially over the life of the project. Early drilling was open hole percussion where the drill cuttings are returned outside the drill rod and captured in a stuffing box on the drill collar prior to being sampled via a cyclone. This drilling method is historically of a lesser quality than face sampling RC due to down-hole sample contamination and loss of sample.

Subsequent to the open hole percussion programmes, RC percussion drilling was completed using a crossover sampling sub. This method of RC percussion drilling collects the drill cuttings via a sampling tool (the crossover sub) which was positioned in the drill string above the RC hammer. The sample quality of this form of RC percussion drilling is superior to that of the open hole percussion, however down-hole contamination is still more prevalent than samples collected with a face sampling RC hammer.

Programmes of RC percussion drilling since 1990 were completed with a face sampling RC hammer. This technology is considered to provide the most representative sample.

Sampling of the RC percussion drilling has been completed by trained Oceana employees and is supervised by Oceana technical staff. Definition of sampling intervals for RC percussion drilling has generally been based on 1m intervals, over the full depth of the drill hole.

Sampling of RC percussion drill holes is completed using the methods detailed below:

- RC cuttings from the drill hole are blown into a trailer-mounted or rig-mounted cyclone, then pass through a tiered riffle splitter. At the completion of each metre, the overall sample is split into a smaller analytical “A” split and larger “B” split. Both samples are collected in uniquely numbered polythene bags;

- Where the drilling sample is considered to be mineralized, the full A split is sent for analysis. Where geology is less well constrained, all A split samples are analysed. The B split is taken to a storage area, to be kept for any further possible test work that may be required; and

- Where the drilling sample is collected from rocks considered to be unmineralized (i.e. schist sequence overlying the HMSZ) then composite samples may be collected. In this case, either four or six sub-samples are collected from the B samples, transferred to a new bag, and submitted for analysis. Anomalous assay results from composite samples are verified by analysis of the original A splits.

Sample tickets were used in the sampling process with one half (identical halves) of each ticket placed in the sample bag.

Once the entire metre had been sampled and placed in the polythene bag, along with the sample ticket, the bag was closed and sealed. Certified standards and blanks were also regularly inserted into the sample sequence as part of the quality control protocols. Samples were transported directly to the on-site laboratory for preparation and subsequent analyses, along with a dispatch sheet. Bags were transported by Oceana personnel.

Prior to 1998, samples were collected from wet percussion drilling. The wet RC percussion drilling is further discussed later in the text and remains a material data quality issue. The sampling of wet RC percussion/percussion drilling is considered fundamentally flawed and has been discontinued since 1998. The recent (Oceana) RC percussion drilling sampling protocols were assessed by external consultants in 2007 and were considered acceptable and consistent with industry standards. Historical drilling completed by Homestake and BHP had defined sampling protocols, which included the logging of moisture content and some twin drilling. Where holes were not wet, a good correlation was observed. These historical drilling practises are considered to be acceptable to the company. All resources associated with this drilling, with the exception of Round Hill (see section 17.6.3), have been mined.

12.3 Diamond Core Sampling

After drill core has been geologically logged and photographed, the sections of core considered to be mineralized, or proximal to mineralized zones are cut in half using a core saw. The drill core was sampled in 1m intervals by trained and supervised technicians and geologists. Each metre was sampled by taking the same half of each piece of core for that metre (i.e. leaving the half with the orientation line and / or metre marks in the tray) and placing them into the appropriate sample bag.

Definition of sampling intervals for diamond drilling are based on geological intervals or 1m intervals, within and beyond the margins of mineralized zones identified during logging. Higher grade intervals within a lower grade intersection are characterised by more abundant sulphide mineralization and generally can be detected visually during core logging. The 1m sampling interval established by Oceana is considered to be sufficient to define these higher grade intervals.

Sample tickets were also used in the sampling process with one half (identical halves) of each ticket placed in the sample bag.

Once the entire metre had been sampled and placed in the polythene bag, along with the sample ticket, the bag was closed and sealed. Certified standards and blanks were also regularly inserted into the sample sequence as part of the quality control protocols. Samples were transported directly to the on-site laboratory for preparation and subsequent analyses, along with a dispatch sheet. Bags were transported by Oceana personnel.

The diamond drilling and sampling is consistent with industry standard practice.

12.4 Aircore Sampling

An Edson aircore rig was used in September 2008 and January 2009 to sample the tailings dams (both the Mixed Tails and SP11) as part of a project to assess the contained scheelite and gold resource. This technique is a fast and convenient method to sample the tailings although excessive torque on the rod string limited final depths to ~90m.

Water injection is used during drilling to maintain recovery of the unconsolidated tailings and consequently the samples are saturated. Therefore, a sample from each 1m interval down-hole is contained in a pre-numbered calico bag, fastened directly beneath the cyclone. The bag is securely tied with as much water and suspended fines contained as possible. Inevitably, some water along with suspended fine material is lost through spillage and overflow.

The calico bags are left on the ground in the field to de-water for a day, and then are transported directly to the on-site laboratory for preparation and subsequent analyses, along with a despatch sheet. Bags were transported by Oceana personnel. Certified standards (both gold and tungsten) and blanks are regularly inserted into the sample sequence as part of the quality control protocols.

12.5 Sample Quality

12.5.1 Summary

The sample quality for diamond drilling is considered to be high. Sample quality for RC percussion drilling is lower than for diamond drilling but generally sufficient to define the position and grade of mineralization. Where RC sample quality issues have caused a grade bias, this bias has been addressed (section 12.2).

12.5.2 Sample Recovery

Sample recovery from RC percussion drilling and diamond drill core is routinely recorded in geological logs and recovery data is stored in an acQuire database. Recovery is generally high and there is no observed correlation between recovery and grade.

12.5.3 RC Percussion Wet Sampling Bias

The potential for wet sampling bias for RC percussion drilling was first recognised at Frasers in June 1997. Since that time, biases have also been identified at Golden Bar, Innes Mills and Round Hill. Areas affected by biases have been mitigated by factoring and subsequently mined at Frasers, Innes Mills (removed from the resource inventory after completing the open pit) and Golden Bar. The approach appears to have been successful. Golden Bar and Round Hill biases are discussed in their respective resource estimate chapters. Frasers wet sampling biases are discussed below.

At Frasers, 23 twin drill holes have been completed to determine the degree of wet sample bias. A statistical analysis can be found in Moore (2001). In summary, the report concludes that wet RC percussion bias and the degree of down-hole contamination varies from hole to hole and is a result of a combination of factors.

Wet sampling bias has been addressed in two ways:

- By replacing wet sampled RC percussion drill holes with their corresponding diamond or dry RC twins; or
- In cases where no twin drill hole exists, globally determined wet sample bias correction factors have been used to factor gold grades for wet RC percussion drill hole samples.

A series of grade dependent factors were obtained by discretising both wet and dry twin sample populations and comparing the respective class means[1]. By this method, the set of factors were derived and applied to the remaining wet samples. The factoring however, takes no account of local variation, or down-hole contamination.

Table 12.1: Wet Bias Factor

Percentile (%)	Class Means (g/t Au)		Wet Threshold	Ratio	Applied Factor
	Twin	Wet			
10	0.07	0.20	0.32	2.85	2.85
20	0.32	0.50	0.63	1.57	1.57
30	0.61	0.82	1.01	1.34	1.28
40	0.96	1.21	1.37	1.27	1.28
50	1.23	1.52	1.63	1.24	1.28
60	1.54	1.90	2.18	1.23	1.28
70	1.89	2.44	2.65	1.29	1.28
80	2.32	2.97	3.28	1.28	1.28
90	2.90	3.73	4.26	1.29	1.28
95	3.72	5.25	6.16	1.41	1.41
97.5	5.06	7.61	8.22	1.50	1.50
99	6.22	10.37	12.50	1.67	1.67
Top	6.88	19.90		2.89	1.67

A substantial issue remains with the wet RC percussion drilling. The above comparison has been limited to the Hangingwall zones, as Oceana believe the chaotic nature of the stockwork precludes meaningful comparison of twin holes. The approach is limited by the nature of the biases; no local variability is modelled, and it is not possible to distinguish between grade bias versus down-hole contamination. No appropriate method exists to adjust the wet RC percussion for bias, however the factors are a reasonable global correction for open pit grade estimation when replacement drilling is not available.

[1] The factors were obtained for Hangingwall mineralization only, although applied to all wet samples.

At Frasers, the wet RC percussion drilling impacts the resource estimates most significantly at depth (Stage 4 and 5 open pits). Despite grade factoring and drilling of twin holes, a moderate risk still remains at the bottom of the Frasers Stage 5 pit. The FR05 resource model predictions will be regularly monitored to gauge the success of the factoring. Further discussion on the wet RC percussion drilling, where applicable, is provided on a deposit by deposit basis.

12.6 Definition of Sample Intervals

Definition of sampling intervals for RC percussion drilling has generally been based on 1m intervals through mineralized zones, or more recently, over the full depth of the drill hole.

Definition of sampling intervals for diamond drilling are based on geological intervals or 1m intervals, within and beyond the margins of mineralized zones identified during logging.

Higher grade intervals within a lower grade intersection are characterised by more abundant sulphide mineralization and generally can be detected visually during core logging. The 1m sampling interval established by Oceana is considered to be sufficient to define these higher grade intervals.

Sampling intervals in aircore holes on the tailings dams include all intersections of tailings material. If the hole is collared in on the embankment, then sampling is not started until the first tailings material is recovered (typically ca. 7m depth).

Drilling has typically been conducted on a regularly-spaced grid. Measured deviation of drill holes indicates that holes quickly trend sub-perpendicular to the host schist foliation direction and consequently drilling intersections typically represent the true width of the mineralized shear zone.

12.7 Summary of Mineralized Widths

The majority of mineralized intersections have been accounted for in the resource estimates for the Macraes Project (see section 17).

13 SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 Sample Preparation Statement

Half cut core samples (diamond drill core) and drill cuttings (RC percussion drilling) samples from the Oceana drilling programmes at Macraes were collected from the source drill samples by employees of Oceana.

Subsequent sample preparation and assay was not conducted by any employee, officer, director or associate of Oceana.

13.2 Sample Preparation, Assay and Analytical Procedures

13.2.1 AMDEL Limited

From 1990 to 2009, RC percussion drill chips and diamond drill core samples from the Oceana drilling programmes at Macraes have typically undergone sample preparation and assay for Au, As and S by a New Zealand based subsidiary of AMDEL Limited (AMDEL) at the Macraes Flat Laboratory, New Zealand.

Sample preparation of geological samples by AMDEL routinely includes drying, crushing (to 4mm), splitting (if required) to a maximum of 1kg and pulverising to obtain an analytical sample of 250g with >95% passing 75µm. A sample preparation flowsheet taken from the AMDEL procedures (NZ-MSPM-1046) is shown in Figure 13.1.

Assay sub-sample size, digest and analysis the prepared samples is completed as per the techniques shown in Table 13.1.

Table 13.1: Assay Techniques

Element	Sub - Sample Size (g)	Digest	Analysis	Detection Limit (ppm)
Gold	50	Fire Assay/Aqua Regia	AAS	0.01
Arsenic	0.2 - 1	Perchloric/Mixed Acid	AAS	10
Sulphur	0.25 - 0.5	N/A	Leco	100
Tungsten (WO_3)	0.2	Sodium perchloride	ICP-OES	0.001%

More recent WO_3 Analysis undertaken by Oceana for the aircore drilling between September 2008 and January 2009 (see section 12.4) has been performed by AMDEL in Auckland, New Zealand. Sample preparation is undertaken on site and pulps sent to the Auckland Laboratory for analysis. The analytical method for tungsten (reported as WO_3) is preparation of a fusion bead from a 0.2g sample followed by ICP-OES.

Note: No WO_3 resources are included in the resource inventory.

13.2.2 SGS New Zealand Limited

From June 2009 all exploration samples have been prepared and analysed off site. The majority are prepared at the SGS New Zealand Limited (SGS) Laboratory at Ngakawau, Westport, and analysed there for arsenic, tungsten (by pressed pellet XRF) and sulphur (Leco). Pulp splits are sent on to the SGS New Zealand Waihi Laboratory for gold analysis by Fire Assay.

Table 13.2: Assay Techniques at SGS New Zealand Limited

Element	Sub - Sample Size (g)	Digest	Analysis	Detection Limit (ppm)
Gold	50	Fire Assay	AAS	0.02
Arsenic	20	N/A	XRF	2
Sulphur	0.5	N/A	Leco	<0.03%
Tungsten	20	N/A	XRF	10

13.2.3 ALS Minerals Laboratory, Australia

During 2009, three diamond drill holes were sent to ALS Laboratory Group Minerals Laboratory, Brisbane, Australia for sample preparation and analyses for gold (Fire Assay), sulphur (Leco) and arsenic and tungsten (pressed pellet XRF). Samples returning relatively high grades of tungsten (>1000ppm) or arsenic (>5000ppm) are re-analysed by fused bead XRF.

Table 13.3: Assay Techniques at ALS Minerals Laboratory, Australia

Element	Sub - Sample Size (g)	Digest	Analysis	Detection Limit (ppm)
Gold	50	Fire Assay	AAS	0.02
Arsenic	20	N/A	XRF	5
Sulphur	1	N/A	Leco	0.01%
Tungsten	20	N/A	XRF	10

13.2.4 Historical Analysis

From commencement of the project to when Macraes mining took over in 1988 (i.e. under Homestake and BHP), various laboratories and analytical methods have been used for gold and tungsten analysis. The majority of these methods are documented and appear to be the valid methods of the day. Assay methods and detection limits are shown in Table 13.4.

With the exception of some drill holes at Round Hill, all the resources associated with areas drilled, sampled and assayed by Homestake and BHP have now been mined out.

Table 13.4: Historical Laboratories and Assay Techniques

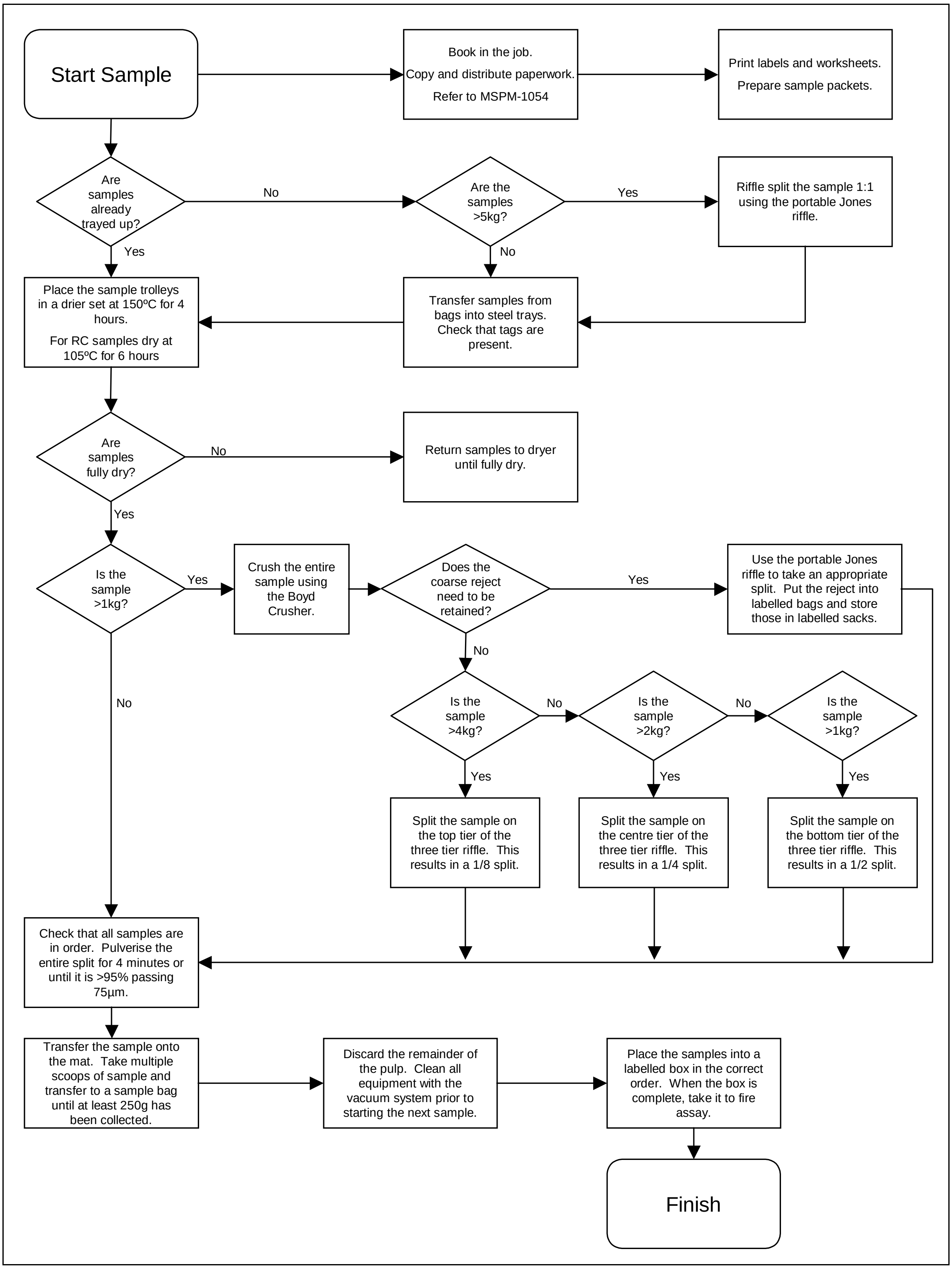
Year	Company	Element	Laboratory	Analysis	Detection Limit
Pre 87	Homestake	Gold	Analabs (Auckland)	FA-AAS	0.001 ppm
		Tungsten (WO_3)	Analabs (Perth)	XRF	0.002 %
1987	BHP	Gold	Analabs (Auckland)	CFA	
		Tungsten (WO_3)	Analabs (Cairns)	PP XRF	
1988	BHP	Gold	Graysons (Auckland or Palmerston)	CFA	
		Tungsten (WO_3)	Southland Co-operative Phosphate Company Ltd	PP-XRF	
1989	Macraes	Gold	Graysons (Macraes)	FA-AAS	0.01 ppm
		Tungsten (WO_3)	Environment (Sydney)	ICP	0.002%

13.3 Sample Security

Oceana managed drilling has been sampled and submitted to the AMDEL laboratory by trained Oceana staff. Once the samples have been submitted to the laboratory, AMDEL staff process the samples and have completed all aspects of the assaying independent of the Oceana personnel.

No measures are in place to ensure the samples’ security however the substantial reconciliation data supports the veracity of the data.

Figure 13.1: Geological Sample Preparation Flowsheet



13.4 Statement of Sample and Assaying Adequacy

The adoption of the analytical methods, including fire assay for gold, is entirely appropriate. Sufficient quality control data exists to allow review of the analytical performance of assay laboratories for the recent drilling only.

The sampling methods, sample preparation procedures and analytical techniques are all considered appropriate when supported with the production and reconciliation data.

14 DATA VERIFICATION

14.1 Introduction

In early 2007, external consultants reviewed the data collection protocols and quality control procedures. The current practises have not changed since this review.

The Macraes Project has a long history of exploration and mining. Data collection protocols and quality control procedure have varied substantially over this period. The analytical quality is monitored by the submission of certified standards, blanks, laboratory duplicates and field duplicates. In addition to the quality control data, a substantial amount of reconciliation data is available and has been used as the final measure of data quality.

14.2 Drill Hole Database

14.2.1 Historical Data

Homestake and subsequently BHP data was stored digitally and transferred to Macraes Mining when BHP left the project. Original Au assay data was recorded in parts per million and grams per tonne format. Tungsten was recorded in parts per million or percentage WO_3 format to 3 significant figures. This data was entered into the Macraes Mining Techbase Database with all tungsten data recorded as percentage WO_3. The percentage values were rounded to 2 decimal places. Repeat analyses were combined and the average result recorded in Techbase.

Digital data and metadata for all drilling post 1994 was captured in the Techbase database.

In 2002 the acQuire geoscientific database was installed and Techbase assay data transferred to acQuire. Tungsten assays in acQuire are denoted as W but represent WO_3 values (checks against historical digital files and original reports confirm this).

14.2.2 Recent Data

The drill hole database is stored in *acQuire* geoscientific database software with the assay data directly loaded from digital data supplied by AMDEL. A review of the drill hole database and data flow processes was completed by external consultants in 2005, including random checks of the drill hole database against laboratory assay data during the site visit with no material errors identified. While no exhaustive review of the data has been completed, the mining and reconciliation data can be used as a check of the data robustness.

Oceana consider the drill hole database management is appropriate and the final database to be robust.

14.3 Comparison of Wet RC Percussion Drilling

The Macraes Project database contains surface diamond and RC percussion drill holes and trench samples, although the assaying from the trench samples has been excluded from resource estimates. Refer to section 12.5.3 for more detail on work undertaken on wet samples.

Further discussion on the wet RC percussion drilling, where applicable, is also provided on a deposit by deposit basis.

14.4 Analysis of Assay Quality Control Data

14.4.1 Summary

The Company has undertaken detailed statistical analysis of the available exploration assay quality control data for the Macraes Project on drilling completed since the 2007 Report. The 2007 analysis, relating to data prior to this, is detailed in the appendix (section 24). Oceana concur with the comments in the 2007 report.

The statistical analysis resulted in a number of interpretive plots from which the analytical accuracy and precision over specific grade ranges has been assessed. The types of plots produced are briefly described below:

- Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines.

- Rank % Absolute Mean Paired Relative Difference (AMPRD) which ranks all assay pairs in terms of precision levels measured as the absolute relative difference from the mean of the assay pairs), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level. This is double the Half Absolute Relative Difference (HARD) value.

- Mean Paired Relative Difference (MPRD) used as another way of illustrating relative precision levels by showing the range of mean paired relative difference over the grade range with the sign retained, thus allowing negative or positive differences to be computed. This plot gives an overall impression of precision and also shows whether or not there is significant bias between the assay pairs by illustrating the MPRD between the assay pairs. This is double the Half Relative Difference (HRD) value.

- Scatter Plot is a simple correlation plot of the value of assay 1 against assay 2 (check assay). This plot allows an overall visualization of precision and bias over selected grade ranges. Correlation coefficients are also used.

- Quantile-Quantile (Q-Q) Plot is a means where the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased.

14.4.2 Exploration Drill Data

Oceana has reviewed a series of quality control data from the AcQuire database which relates to AMDEL, SGS and ALS Chemex assaying. The data available for review comprised certified standards, blanks, duplicates and field duplicates.

A brief discussion of the investigation is provided below.

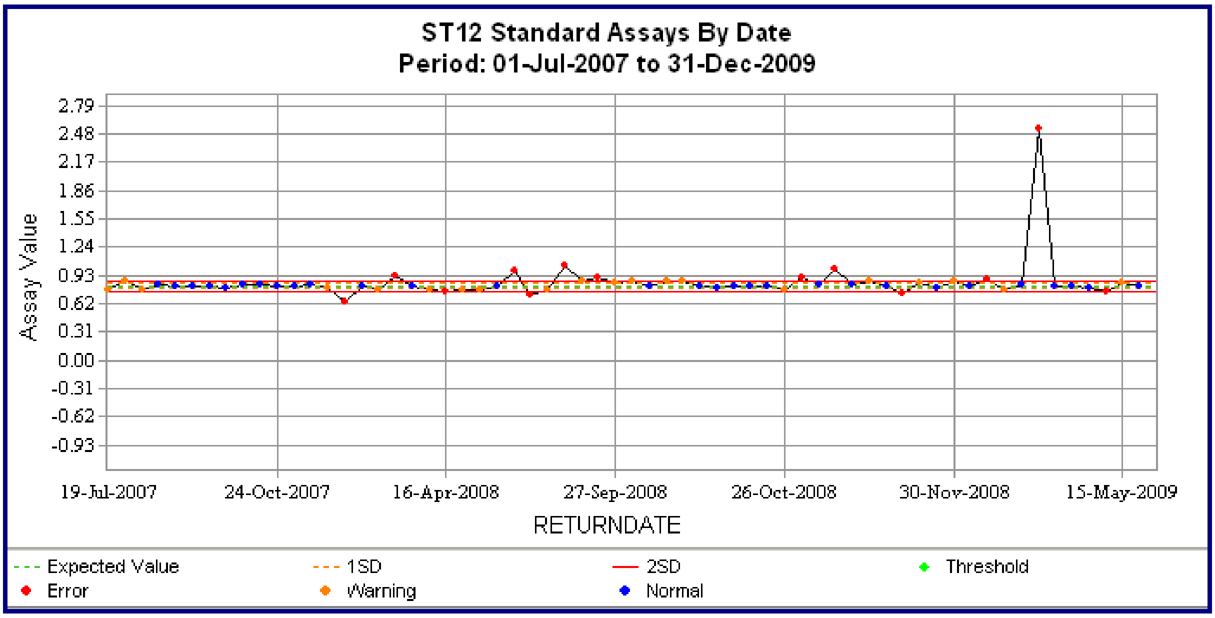
14.4.2.1 Standards for Gold

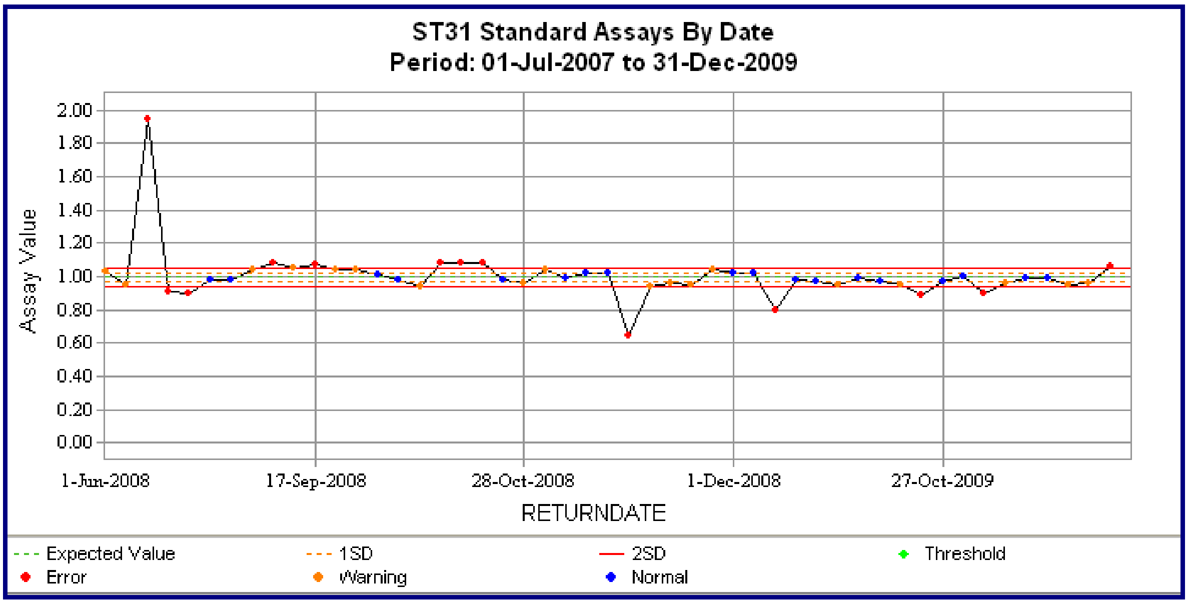
The standards database available for review comprises 239 Rocklabs Au Standards assays. The source and details of these data is well documented. Summary statistics of the standards data is presented in Table 14.1. The plots for the gold standards are shown in Figure 14.1.

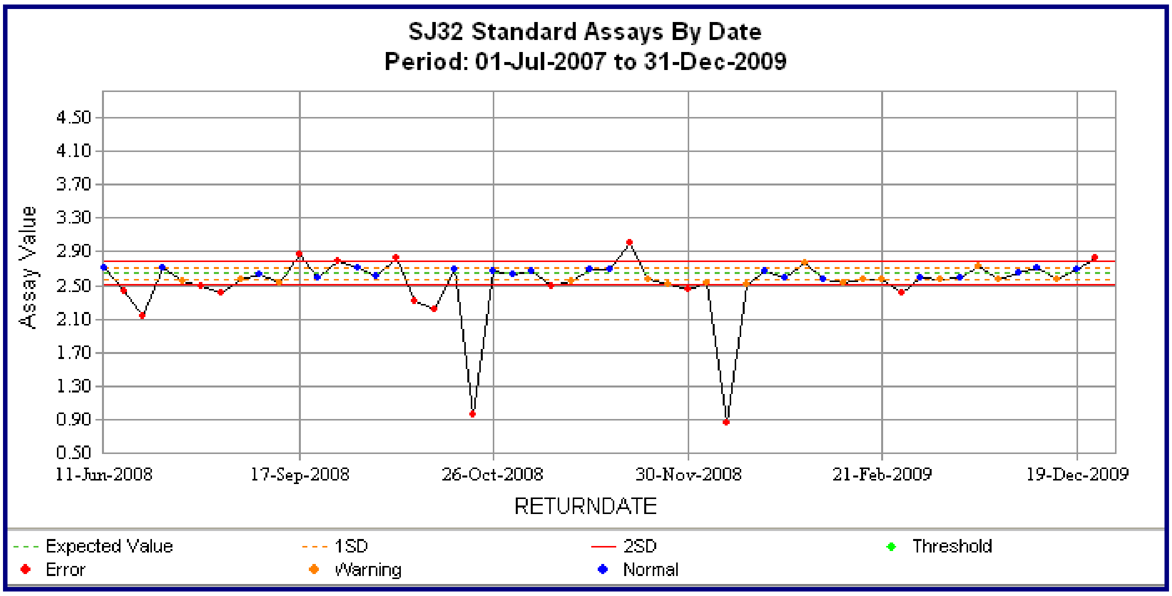
Table 14.1: Macraes Exploration – Summary of Certified Gold Standards

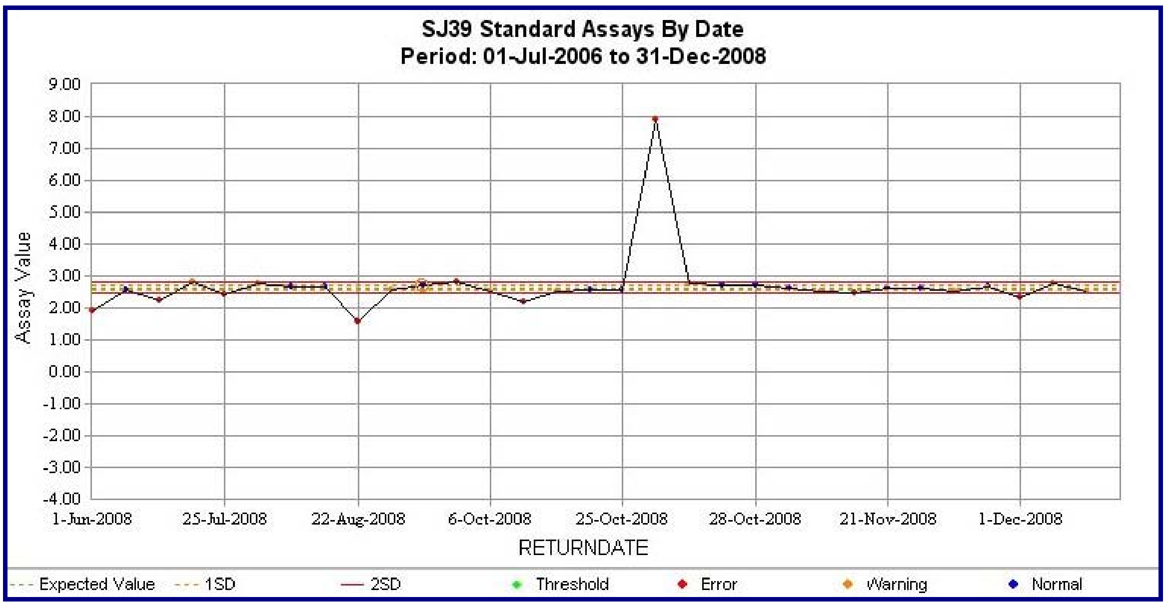
Standard	ST12 (Au)	ST31 (Au)	SJ32 (Au)	SJ32 mixed stds removed <1.1 g/t	SN38 (Au)	SN38 mixed stds removed <2.8 g/t	SJ39 (Au)
Expected Value	0.819	0.996	2.645	2.645	8.573	8.573	2.64
Expected Value Range	0.76-0.88	0.94-1.05	2.51-2.78	2.51-2.78	8.26-8.89	8.26-8.89	2.48-2.81
Count	62	49	52	50	27	23	49
Mean	0.85	1.001	2.53	2.60	6.64	7.63	2.60
Median	0.82	0.98	2.59	2.59	7.61	8.06	2.59
Minimum	0.65	0.64	0.86	2.13	0.01	4.01	0.01
Maximum	2.53	1.95	3.01	3.01	8.96	8.96	7.91
Std Deviation	0.22	0.16	0.36	0.15	2.64	1.16	0.88
% in Tolerance	79.03	73.47	69.23	72	25.93	30.43	69.39
Std Error	0.028	0.022	0.050	0.021	0.508	0.242	0.125
%RSD	26.16	15.57	14.12	5.79	39.80	15.18	33.70
Total Bias	0.042	0.005	-0.042	-0.017	-0.226	-0.110	-0.015

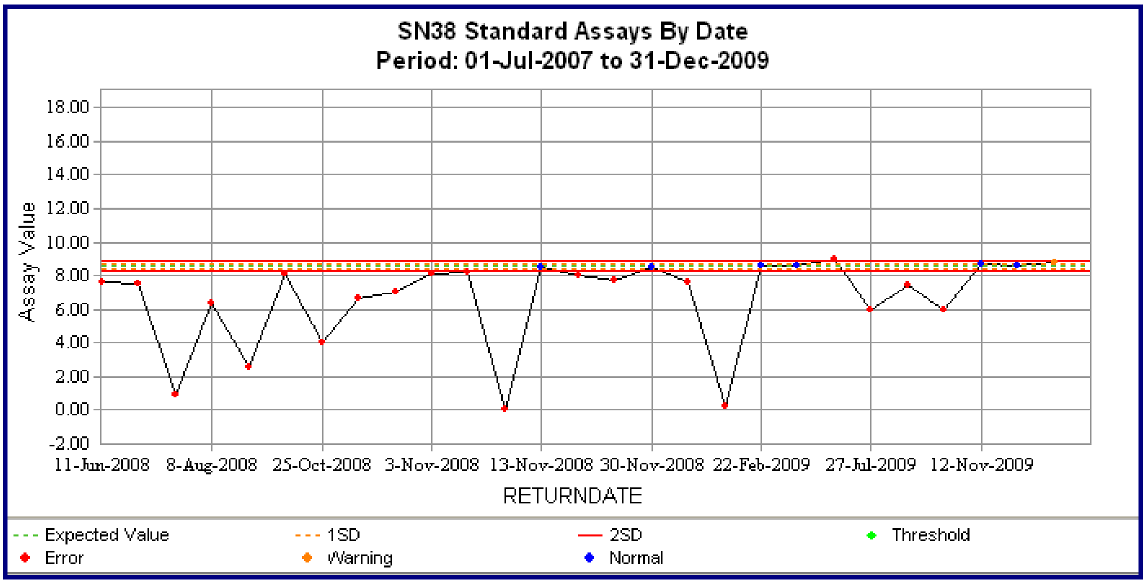
Figure 14.1: Various Standards for Gold




ST31 Standard Assays By Date
Period: 01-Jul-2007 to 31-Dec-2009
Assay Value
2.00
1.80
1.60
1.40
1.20
1.00
0.80
0.60
0.40
0.20
0.00
1-Jun-2008
17-Sep-2008
28-Oct-2008
1-Dec-2008
27-Oct-2009
RETURNDATE
Expected Value
1SD
2SD
Threshold
Error
Warning
Normal


SJ32 Standard Assays By Date
Period: 01-Jul-2007 to 31-Dec-2009
Assay Value
4.50
4.10
3.70
3.30
2.90
2.50
2.10
1.70
1.30
0.90
0.50
11-Jun-2008
17-Sep-2008
26-Oct-2008
30-Nov-2008
21-Feb-2009
19-Dec-2009
RETURNDATE
Expected Value
1SD
2SD
Threshold
Error
Warning
Normal


SJ39 Standard Assays By Date
Period: 01-Jul-2006 to 31-Dec-2008
Assay Value
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
-1.00
-2.00
-3.00
-4.00
1-Jun-2008
25-Jul-2008
22-Aug-2008
6-Oct-2008
25-Oct-2008
28-Oct-2008
21-Nov-2008
1-Dec-2008
RETURNDATE
Expected Value
1SD
2SD
Threshold
Error
Warning
Normal



Substantial mixing of standards is noted as displayed in Figure 14.1. Incorrectly assigned standards have been excluded from the analysis where appropriate.

When considering all the standards data (including the mixed data), 162 (67.8%) of the 239 standards are within a ±2SD accuracy range. When mixed standards are excluded, 69.5% of the standards are within a ±2SD accuracy range.

With the exception of SN38, the standards reveal a relative bias measure of less than 5%, which is considered to be acceptable. Standard SN38 has performed poorly, although it should be noted that in no case has the laboratory overstated the grade for this standard.

14.4.2.2 Standards for WO3

This report includes a preliminary review of WO_3 analyses. Work is ongoing. Note that no WO_3 resources are included in the Oceana resource inventory.

The WO_3 standards database available for review presents only 61 WO_3 Standards assays. These are summarized in Table 14.2.

Table 14.2: Macraes Exploration – Summary of Certified WO_3 Standards

Standard	CT-1 (WO_3)	CT-1 mixed stds removed <0.13%	TLG-1 (WO_3)	TLG-1 mixed stds removed >1.3%	MP-2 (WO_3)
Expected Value	1.31	1.31	0.11	0.11	0.82
Expected Value Range	1.29-1.33	1.29-1.33	0.10-0.11	0.10-0.11	0.79-0.84
Count	31	29	29	28	30
Mean	1.28	1.36	0.16	0.12	0.87
Median	1.38	1.39	0.11	0.11	0.87
Minimum	0.11	1.19	0.07	0.07	0.65
Maximum	1.46	1.46	1.35	0.46	0.95
Std Deviation	0.31	0.08	0.23	0.07	0.06
% in Tolerance	16.13	17.24	62.07	64.29	20.00
Std Error	0.06	0.01	0.04	0.01	0.01
%RSD	24.54	5.65	144.44	10.61	6.54
Total Bias	-0.03	0.03	0.55	0.14	0.05

The total number of analyses for each standard is low, making detection of potential biases difficult.

When considering all the WO_3 standards data (including the mixed data), 29 (47.5%) of the standards are within a ±2SD accuracy range.

Some follow-up is required to look further into the reproducibility for WO_3 standard, TLG-1.

For the other two standards (CT-1 and MP-2), it appears that minor biases may be present, although given the low numbers of data, it difficult to be sure.

14.4.2.3 Laboratory Repeats

Laboratory repeats (duplicate 50g samples of the sample pulp collected after pulverization) have been assessed. The following data is presented as quality control statistics:

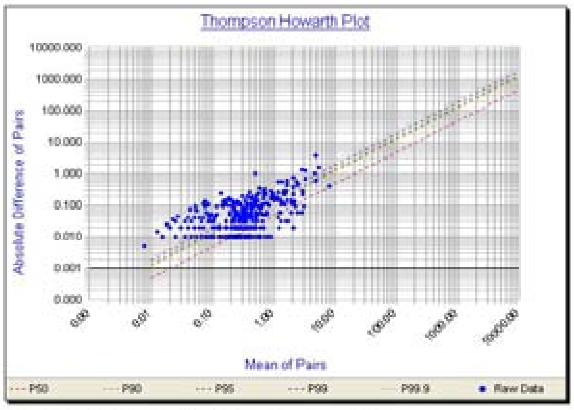
- Figure 14.2: Quality Control Statistics – All Gold Data – Laboratory Repeats;
- Figure 14.3: Quality Control Statistics – Diamond Drilling – Gold Assay (Au g/t) Laboratory Repeats;
- Figure 14.4: Quality Control Statistics – Percussion Drilling – Gold Assay (Au g/t) Laboratory Repeats;
- Figure 14.5: Quality Control Statistics – All Sulphur Data (S %) – Laboratory Repeats;
- Figure 14.6: Quality Control Statistics – All Arsenic Data (As ppm) – Laboratory Repeats; and
- Figure 14.7: Quality Control Statistics – All Tungsten Data (WO3 %) – Laboratory Repeats.

Very good correlation is noted for the gold data with the linear correlation generally >0.96. No apparent bias is evident. No significant difference is noted for the laboratory repeats for the data grouped by sample type (diamond and percussion drilling).

The sulphur and arsenic repeat data set also show strong correlation between the original and repeat assay (r = 0.98 for sulphur and r = 0.98 for arsenic). As with gold, no bias is noted in these data sets.

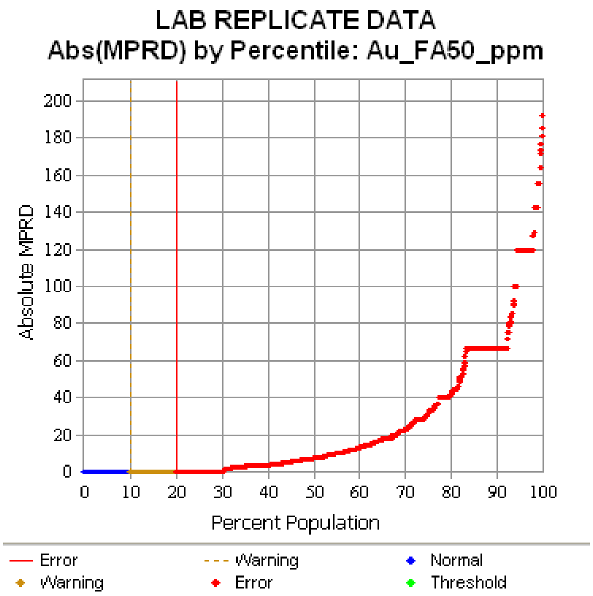
Figure 14.2: Quality Control Statistics – All Gold Data – Laboratory Repeats

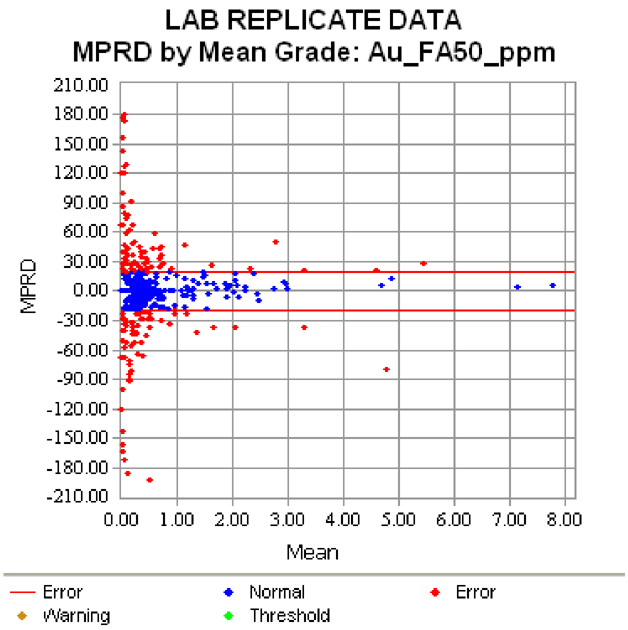
Au_FA50_ppm			
	Value	Check Value	units
No. Pairs	1157	1157	
Minimum	0.01	0.01	g/t
Maximum	8.01	7.59	g/t
Mean	0.32	0.32	g/t
Median	0.16	0.17	g/t
Std Deviation	0.63	0.62	
Coefficient of Variation	1.99	1.95	
Correlation Coefficient	0.96		
Bias	-0.01		

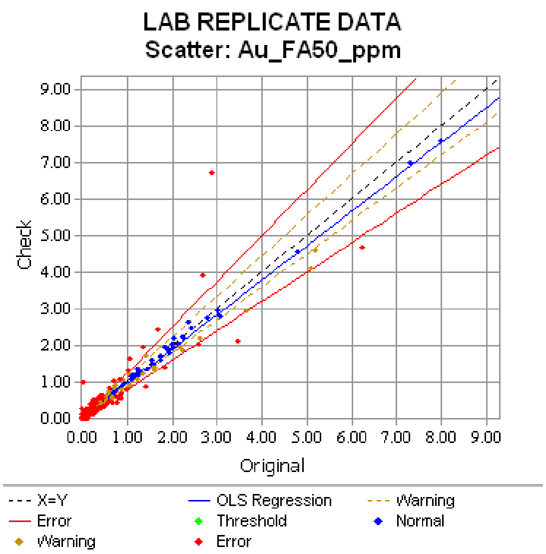


Precision Control Lines based on a Precision of 10%

Note: A point is not charted where the Absolute paired difference is zero







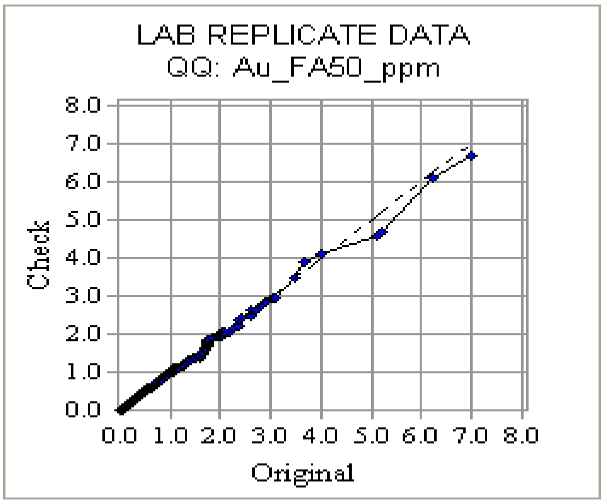
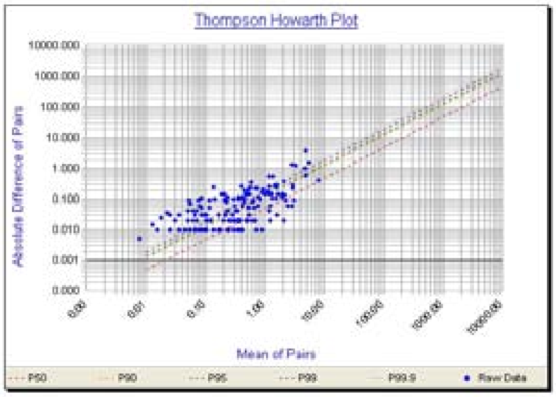


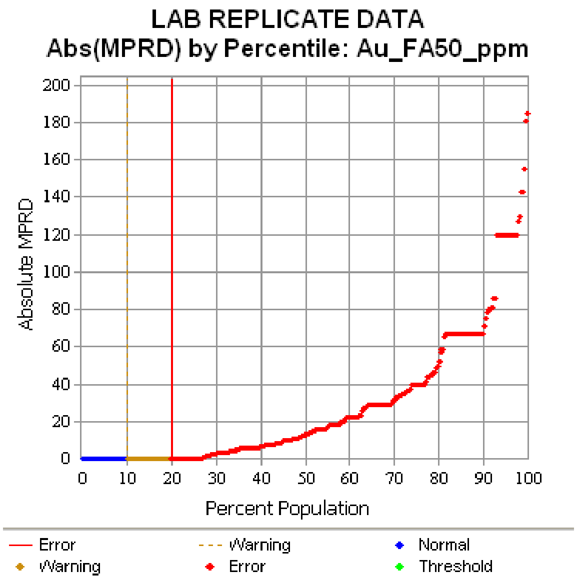
Figure 14.3: Quality Control Statistics – Diamond Drilling – Gold Assay (Au g/t) Laboratory Repeats

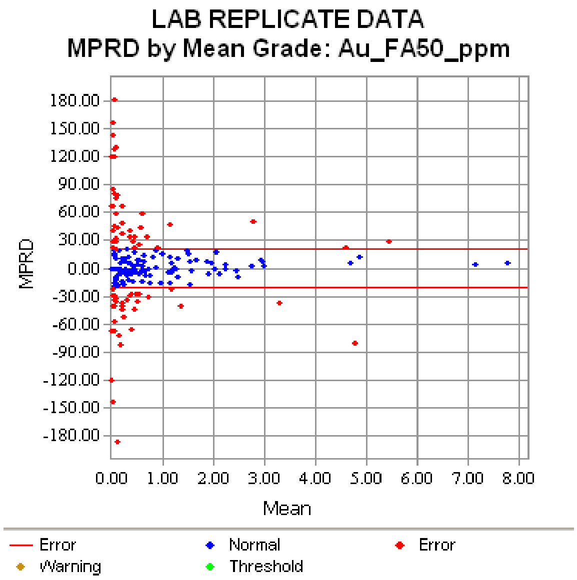
Au_FA50_ppm			
	Value	**Check Value**	**units**
No. Pairs	302	302	
Minimum	0.01	0.01	g/t
Maximum	8.01	7.59	g/t
Mean	0.84	0.48	g/t
Median	0.08	0.07	g/t
Std Deviation	1.04	1.02	
Coefficient of Variation	2.15	2.14	
Correlation Coefficient	0.96		
Bias	-0.02		

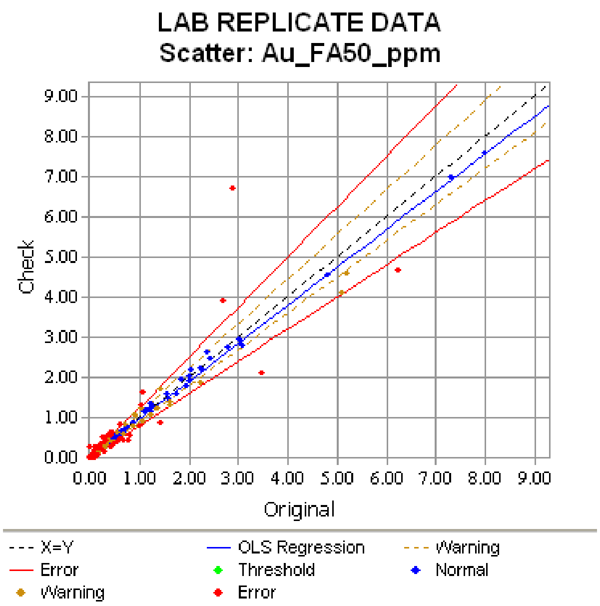


Precision Control Lines based on a Precision of 10%

Note. A point is not charted where the Absolute paired difference is zero







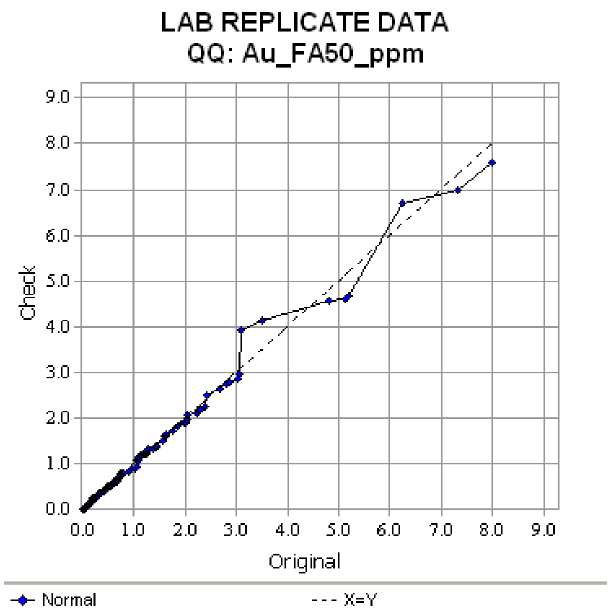
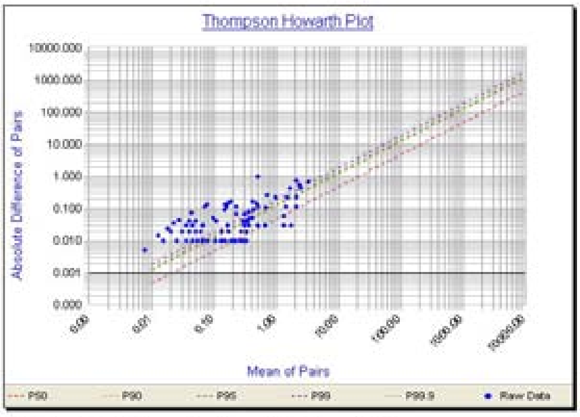


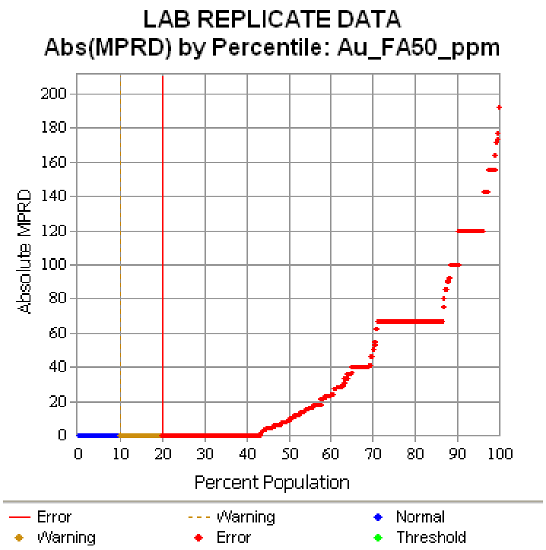
Figure 14.4: Quality Control Statistics – Percussion Drilling – Gold Assay (Au g/t) Laboratory Repeats

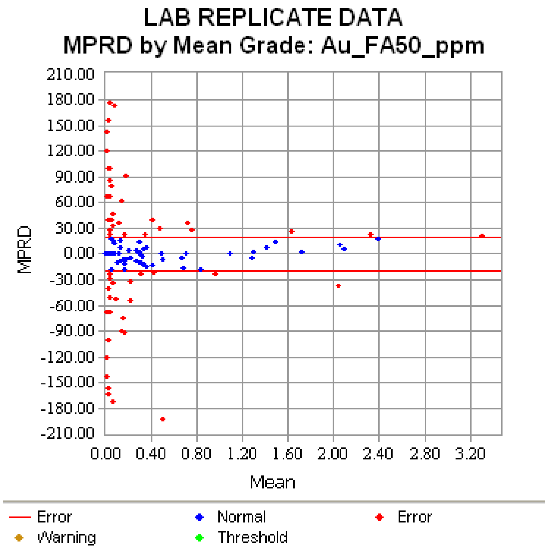
Au_FA50_ppm	Value	Check Value	units
No. Pairs	416	416	
Minimum	0.01	0.01	g/t
Maximum	3.66	2.96	g/t
Mean	0.19	0.18	g/t
Median	0.02	0.02	g/t
Std Deviation	0.49	0.45	
Coefficient of Variation	2.65	2.53	
Correlation Coefficient	0.98		
Bias	-0.03		

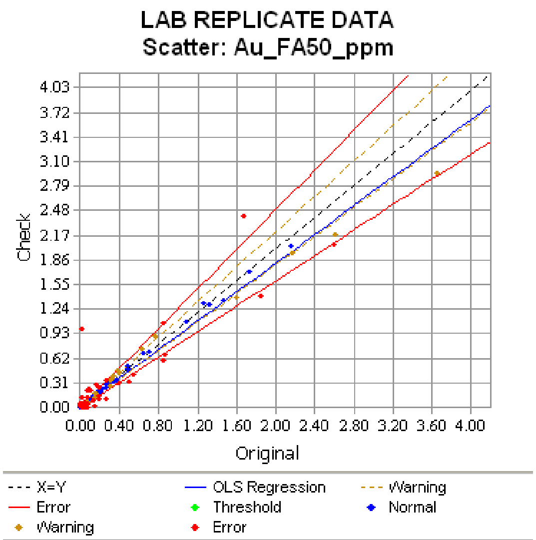


Precision Control Lines based on a Precision of 10%

Note: A point is not charted where the Absolute paired difference is zero







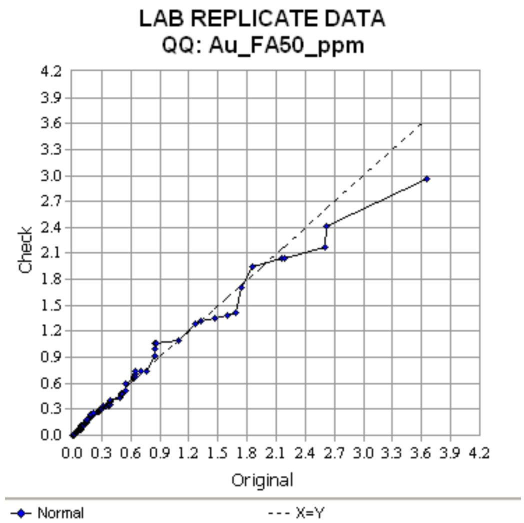
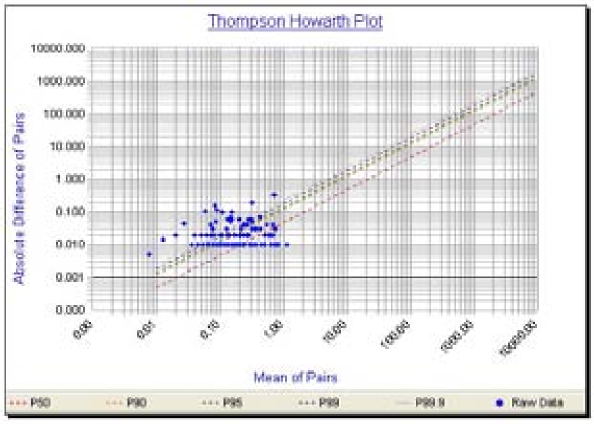


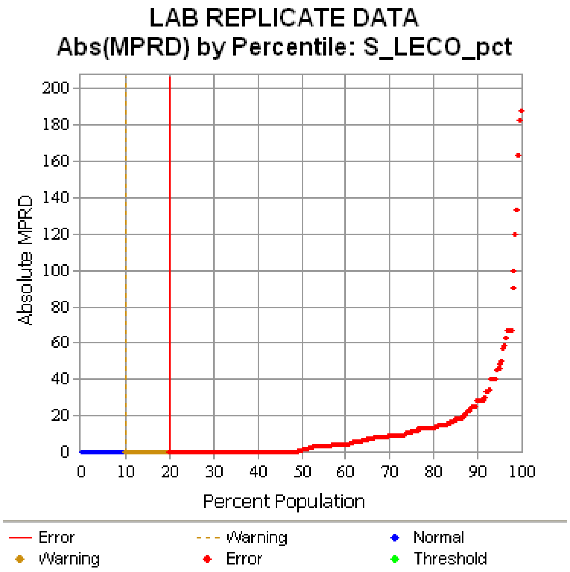
Figure 14.5: Quality Control Statistics – All Sulphur Data (S %) – Laboratory Repeats

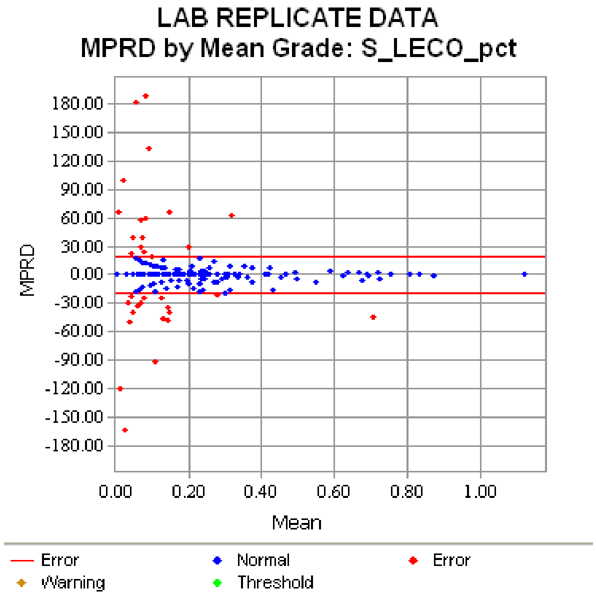
S_LECO_pct			
	Value	Check Value	units
No. Pairs	319	319	
Minimum	0.01	0.01	%
Maximum	1.13	1.12	%
Mean	0.20	0.20	%
Median	0.15	0.16	%
Std Deviation	0.17	0.17	
Coefficient of Variation	0.84	0.85	
Correlation Coefficient	0.98		
Bias	0.01		

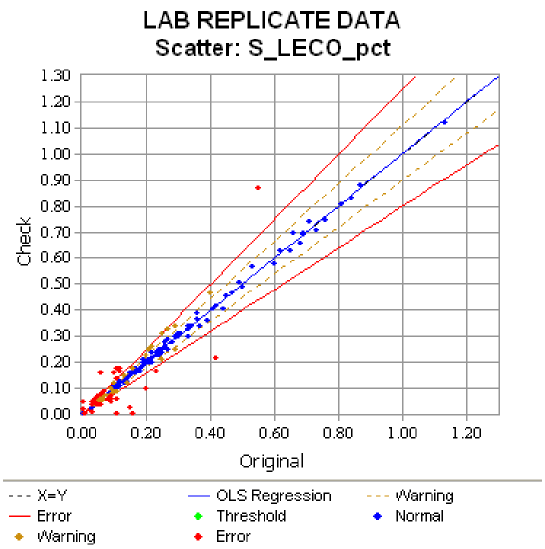


Precision Control Lines based on a Precision of 10%

Note: A point is not charted where the Absolute paired difference is zero







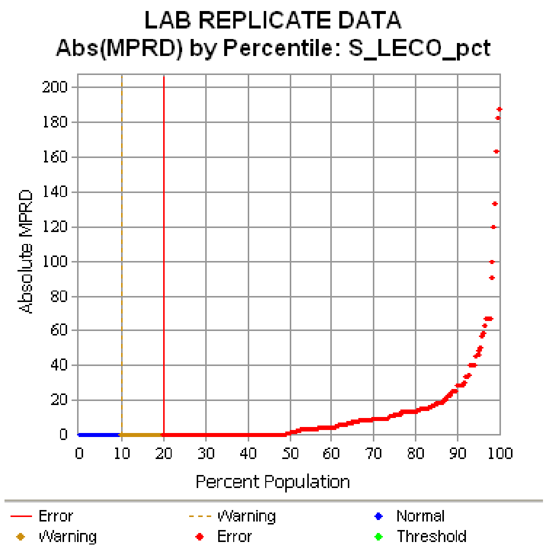
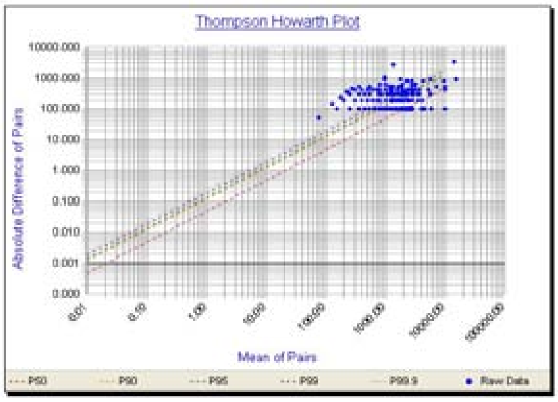


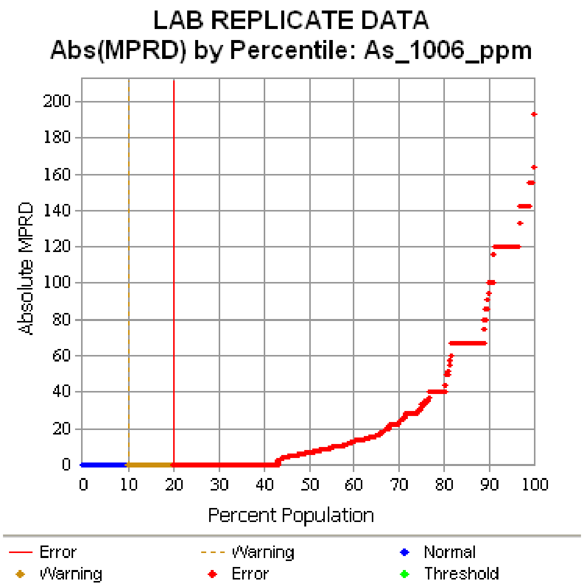
Figure 14.6: Quality Control Statistics – All Arsenic Data (As ppm) – Laboratory Repeats

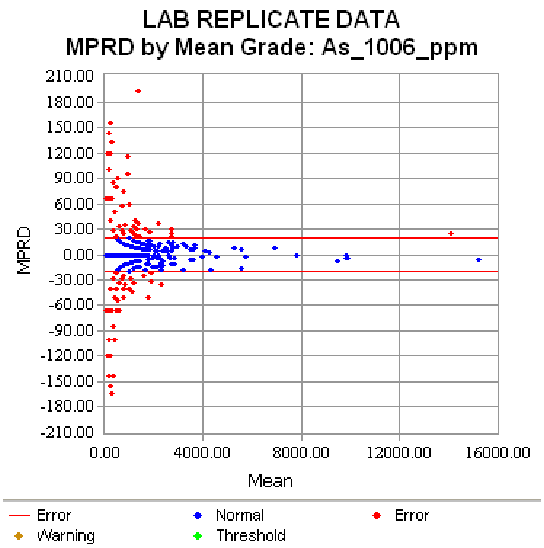
As_1006_ppm			
	Value	**Check Value**	**units**
No. Pairs	981	981	
Minimum	50	50	ppm
Maximum	15800	15700	ppm
Mean	870	854	ppm
Median	300	400	ppm
Std Deviation	1313.08	1291.97	
Coefficient of Variation	1.51	1.51	
Correlation Coefficient	0.98		
Bias	-0.02		

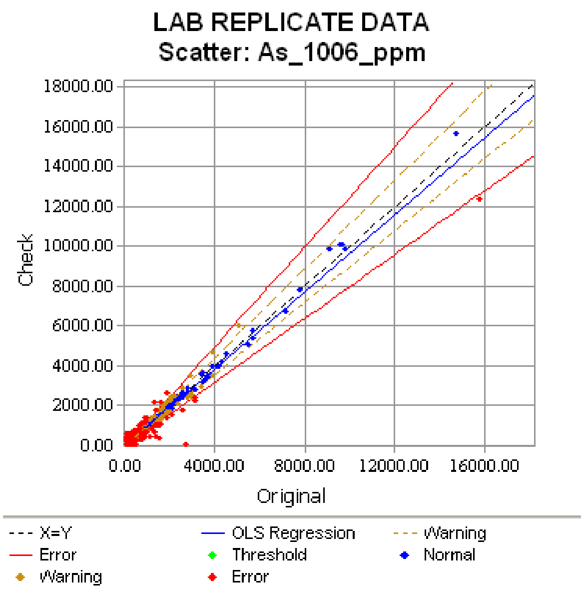


Precision Control Lines based on a Precision of 10%

Note: A point is not charted where the Absolute paired difference is zero







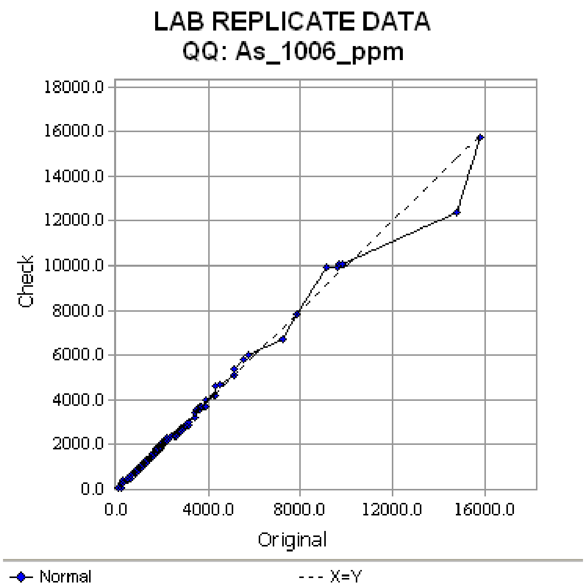
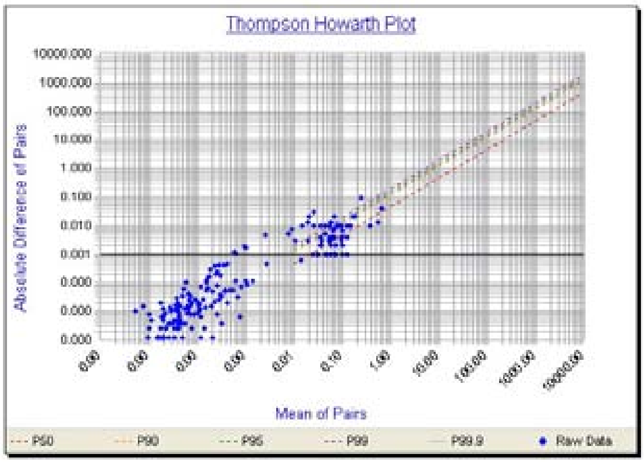


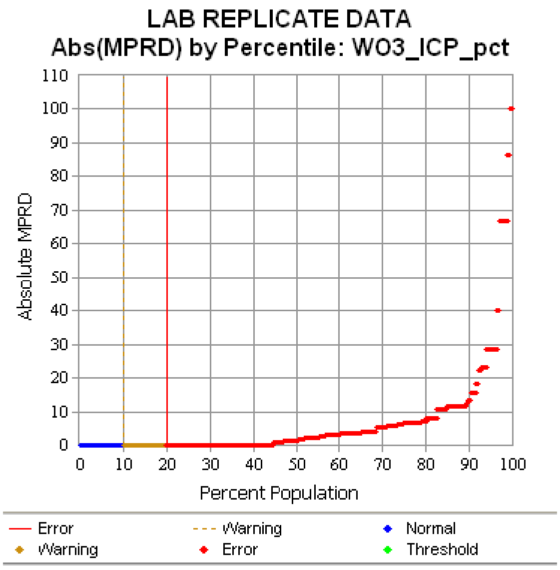
Figure 14.7: Quality Control Statistics – All Tungsten Data (WO_3 %) – Laboratory Repeats

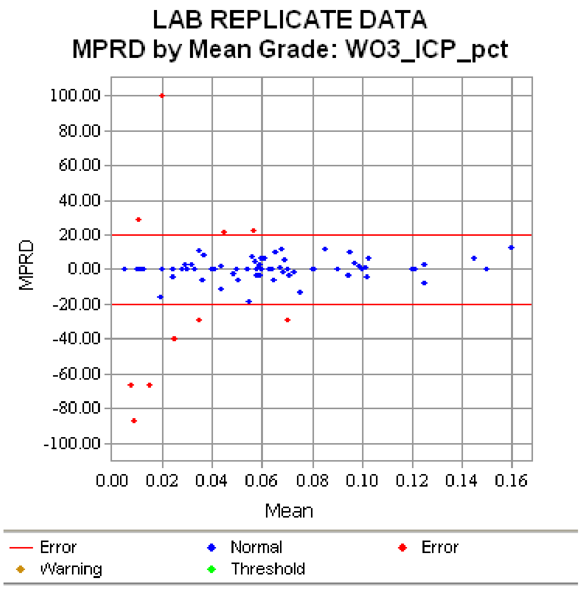
WO_3_ICP_pct			
	Value	Check Value	units
No. Pairs	332	332	
Minimum	0.01	0.01	%
Maximum	0.17	0.15	%
Mean	0.06	0.05	%
Median	0.05	0.05	%
Std Deviation	0.03	0.03	
Coefficient of Variation	0.63	0.62	
Correlation Coefficient	0.99		
Bias	-0.01		

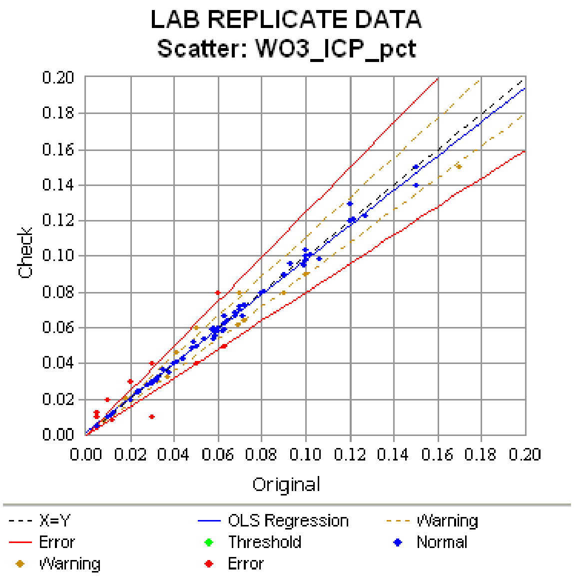


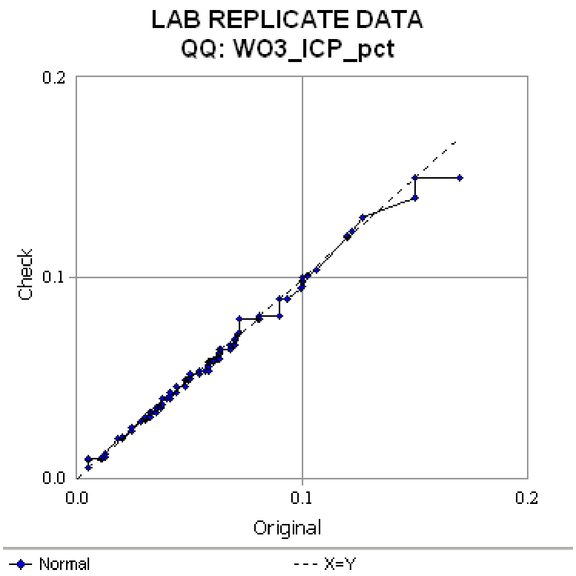
Precision Control Lines based on a Precision of 10%

Note. A point is not charted where the Absolute paired difference is zero









14.4.2.4 Field Duplicates

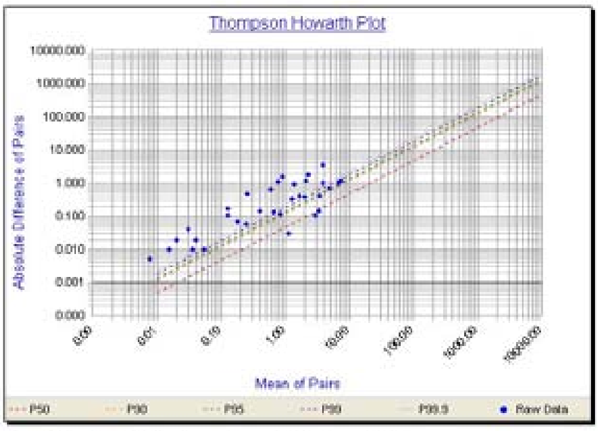
Field duplicates represent a second sample which is collected from the initial sample and then submitted for assay using the same analytical approach as the original sample and provide a measure of the total error including sampling error.

Field duplicates collected at the RC percussion drill rig are collected at the same time as the initial sample. For diamond core field duplicates represent a quarter core sample taken from the remaining half core after initial half core sampling is undertaken.

A limited field duplicate data set is available for gold (Figure 14.8). This data, which admittedly is limited, shows that the duplicate samples have comparatively reproduced the original assays with the linear correlation coefficient calculated at 0.81. 15% of the data is within ±10% precision (AMPRD) while approximately 31% of the data are ±20%. A revised duplication allocation has been implemented since the previous report. This ensures one sample targets mineralization in the hangingwall (HW). The second duplicate is selected 30m below or at the bottom of the drill hole, whichever is less. This procedure results in 50% of the duplicates representing the hangingwall (the primary source of ore), while at the same time selecting duplicates from mineralization below the hangingwall irrespective of sample grade.

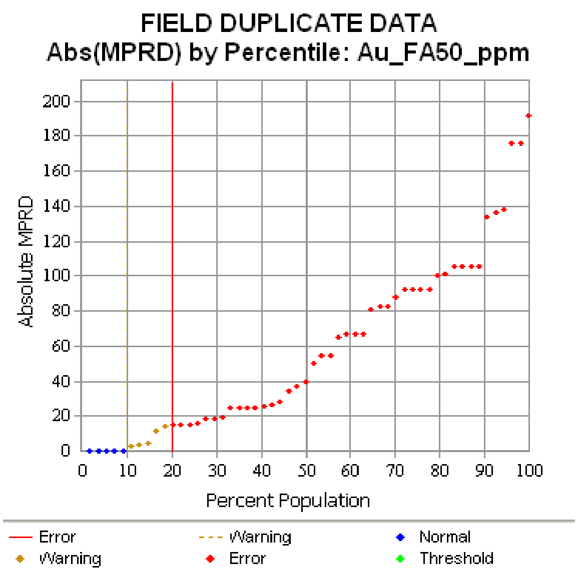
Figure 14.8: Quality Control Statistics – All Gold Data – Field Duplicates

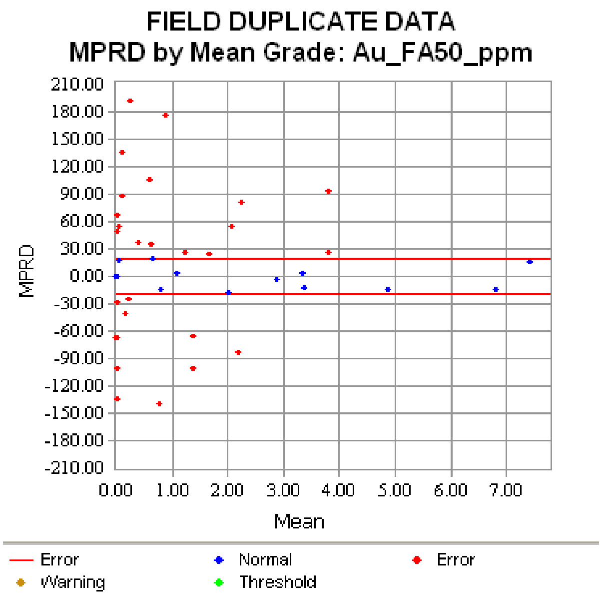
Au_FA50_ppm			
	Value	**Check Value**	**units**
No. Pairs	54	54	
Minimum	0.01	0.01	g/t
Maximum	8.01	7.33	g/t
Mean	1.61	1.28	g/t
Median	0.91	0.57	g/t
Std Deviation	1.96	1.66	
Coefficient of Variation	1.21	1.29	
Correlation Coefficient	0.81		
Bias	-0.20		

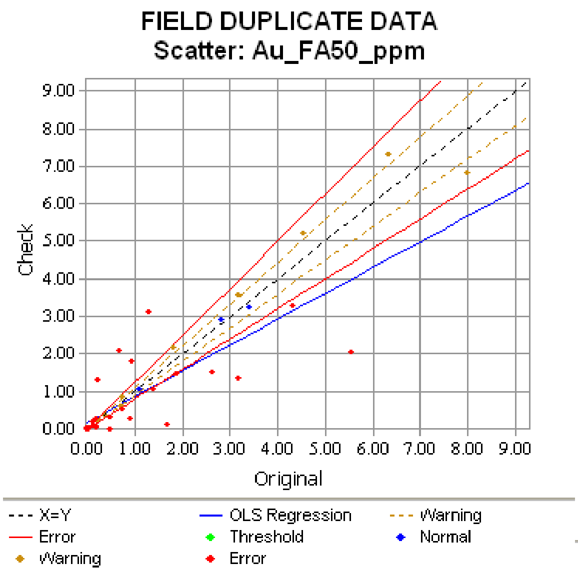


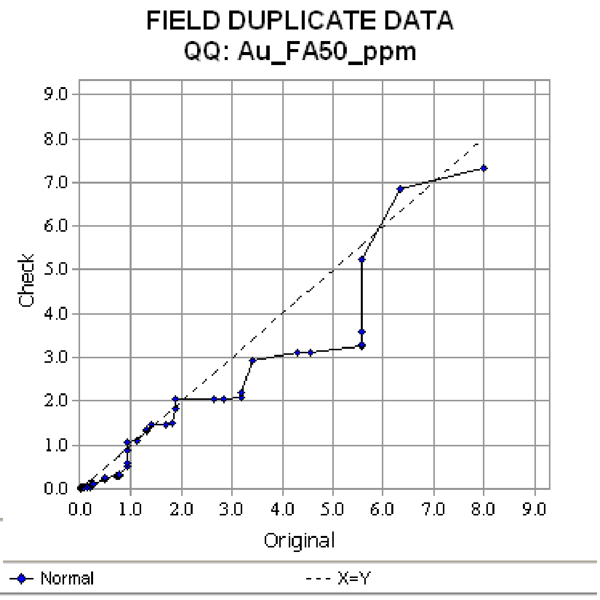
Precision Control Lines based on a Precision of 10%

Note: A point is not charted where the Absolute paired difference is zero





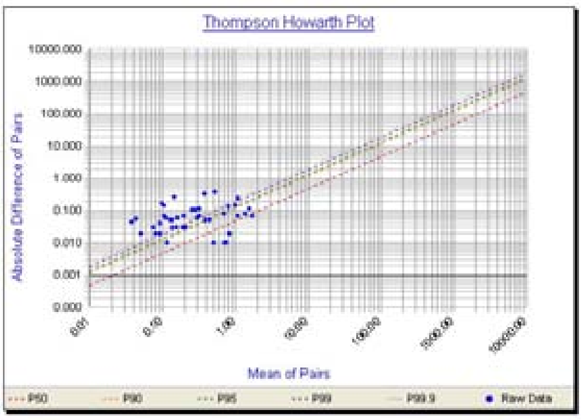




Relatively few sulphur data are available for analysis. Based on the available data, the field duplicates indicate an acceptable level of precision is being achieved in sampling when only sulphur is considered. Sulphur shows the duplicate samples have reasonably reproduced the original assays with the linear correlation coefficient calculated at 0.81. Only 24 of the data are within ±10% precision (AMPRD) while approximately 48% of the data are ±20%.

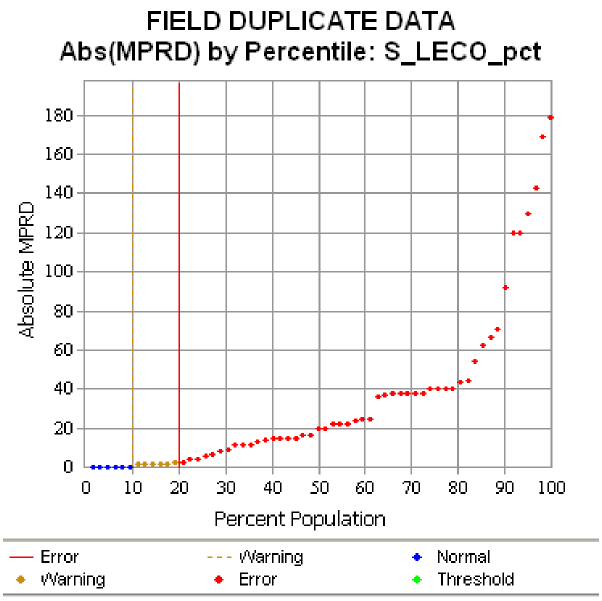
Figure 14.9: Quality Control Statistics – All Sulphur Data – Field Duplicates

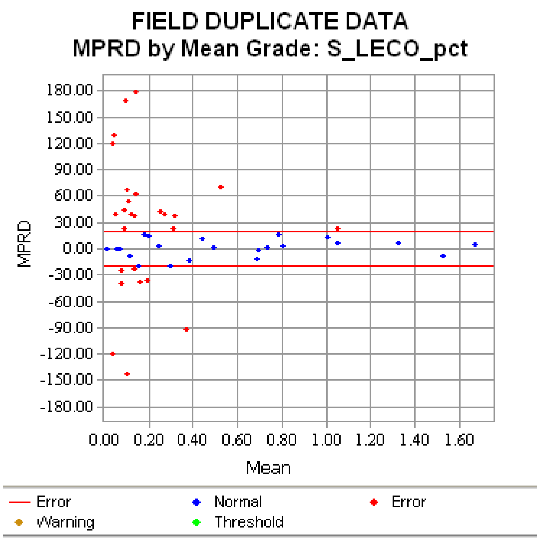
S_LECO_pct			
	Value	Check Value	units
No. Pairs	62	62	%
Minimum	0.01	0.01	%
Maximum	1.71	1.64	%
Mean	0.36	0.34	%
Median	0.20	0.19	%
Std Deviation	0.39	0.37	
Coefficient of Variation	1.08	1.11	
Correlation Coefficient	0.97		
Bias	-0.07		

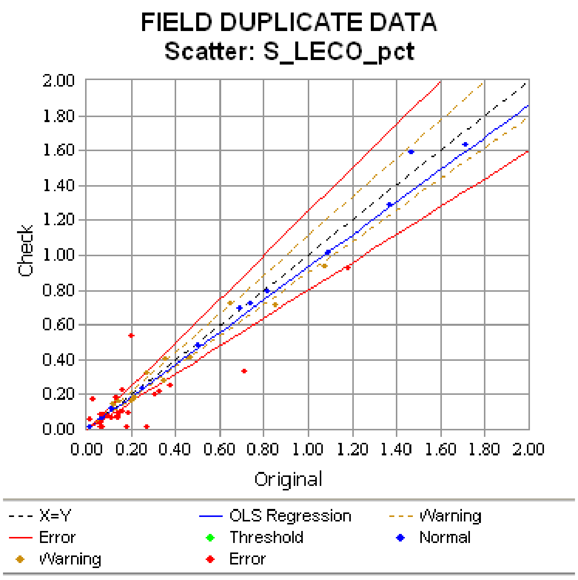


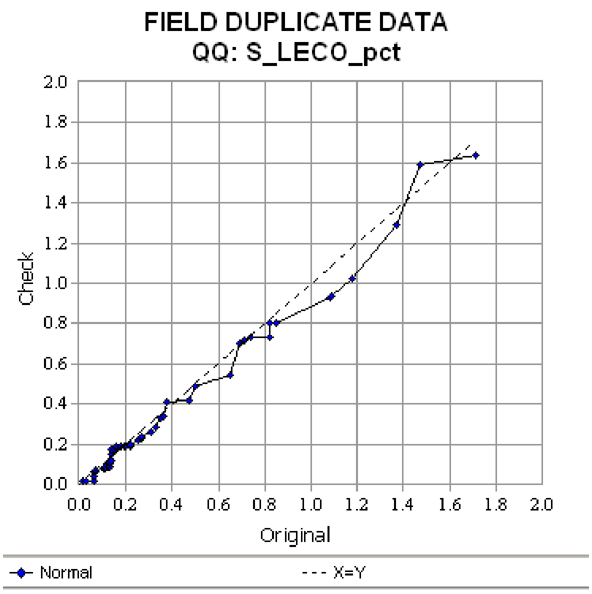
Precision Control Lines based on a Precision of 10%

Note. A point is not charted where the Absolute paired difference is zero









14.4.3 Quality Control Investigation Summary

Based on the quality control database assessed by Oceana, acceptable levels of assay precision and accuracy are generally being achieved by AMDEL, ALS and SGS. The conclusion is supported by the available reconciliation data.

14.5 Summary

Due to the long exploration and mining history of the project, the quality control database is incomplete for the Macraes Project making complete and thorough investigation impossible. The risk associated with the incomplete quality control data set is offset by the available mining and reconciliation data which supports the quality of the data.

Notwithstanding the limitations in the data set, the available recovery and QAQC data indicates the assay data meets acceptable limits of accuracy and precision and is therefore suitable for the purposes of grade estimation. The bias associated with the wet RC percussion drilling remains a material item and while Oceana have taken steps to mitigate the risks associated with this data set, ultimately only removal of this data can ensure no negative effects in the grade estimates. A risk remains with these wet samples.

In addition to the assay data, the survey data both collar and down-the-hole survey, is considered to be robust and present little risk.

15 ADJACENT PROPERTIES

There are no adjacent properties that impact on the potential merit of the Macraes Project. The mineral permits contain all known significant gold mineralization in the area.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Introduction

Macraes flotation flowsheet underwent a number of modifications in 2008. These included the replacement of previous columns with new OK-330 tank cells and reconfiguration of the cleaner section, incorporating previous rougher scavengers as new cleaners. Changes in the configuration of the carbon-in-leach (CIL) section also occurred in early 2008, to facilitate direct leaching of Macraes concentrate at various times.

These changes led to flotation and CIL recoveries being determined empirically from recent plant performance rather than being based on previous or new laboratory test work.

16.2 Throughput

Tonnes treated each month were calculated using throughput models for each of the two SAG mills. The throughput model is based on feed gold grade and throughput for period January 2009 to July 2008.

The main SAG mill (processing approximately 80% feed) ML-01 model predicted decreased throughput as feed gold grade increased. ML-500 throughput was not affected by feed gold grade. Same feed gold grade used for both models, as mill feed grades were not sampled or back calculated.

16.3 Mass Pull

As a proportion of feed tonnes are recovered to the concentrate stream, the tails tonnes are less than 100% of feed tonnes. This proportion needs to be known to correctly calculate recovery.

The mass pull to the concentrate stream was calculated from a model based on feed sulphur grade, which was generated from daily data from the period January to August 2009.

16.4 Flotation Tails Gold Grade

Tails gold grade was calculated from a model based on recent plant performance.

16.5 Flotation Recovery

Flotation recovery was calculated using the mass pull and flotation tails gold grade models. For each month a different flotation recovery was expected.

16.6 CIL Recoveries

Macraes and Reefton CIL recoveries were based on actual CIL performance over the period from January 2009 to July 2009. Macraes' CIL recovery was 93.8% and Reefton's CIL recovery was 92.6%.

All Reefton concentrate was scheduled to be treated through the autoclave and be recovered at the Reefton CIL recovery. Macraes concentrate was treated through the autoclave during the remaining time available each month. Any excess Macraes concentrate was scheduled for direct leach, where it was scheduled to be sent to the CIL section without being treated through the autoclave. Macraes concentrate treated through the autoclave had the Macraes CIL recovery applied. Macraes concentrate treated by direct leach was recovered at 7% less than the Macraes CIL rate. The 7% reduction for direct leach was based on the plant performance of Macraes concentrate over May, June and July 2008.

16.7 Overall Recovery

The same flotation and CIL recoveries were used for the open pit and underground mines at Macraes. Overall recoveries for Macraes and FRUG are the product of the Flotation and CIL recoveries. Yearly forecast recoveries for Macraes open pit and underground mines are presented in Table 16.1. They do not include Reefton nor oxide ore treated in 2015.

Table 16.1: Forecast recoveries used in LOMP09 for Macraes Open Pit and FRUG Mines

Year	Flotation Recovery (%)	CIL Recovery (%)	Overall Recovery (%)
2010	87.4	94.0	82.2
2011	88.2	93.2	82.2
2012	88.3	93.7	82.8
2013	86.9	93.8	81.5
2014	86.4	95.0	82.1
2015	88.5	95.0	83.9

16.8 Future Ore

The purpose of metallurgical testing of the future ore samples is to confirm metallurgical recovery and understand any variations in ore hardness or recovery that may affect process plant throughput. At this stage, metallurgical testing on future ore samples from diamond core samples has encompassed:

- Ore competency (grindability) data is derived from SAG power index (SPI) and bond work index (Wi) testing completed by Minnovex Technologies in USA. Sample preparation work is conducted at the Macraes site. This test work provides data on ore hardness in relationship to SAG and ball mill throughput.

- Kinetic flotation testing. This is a float that produces 4 concentrates and 1 tail for assay. Concentrates are floated off over a 1, 4, 8 and 13 minute periods using the standard Oceana laboratory float procedure and reagent doses. The primary function of this process is to provide information on how quickly the gold floats. This test indicates the expected rougher-scavenger flotation performance. It does not however give an expected total flotation recovery figure. Release analysis must be used for this.

- Release analysis flotation testing. This is a 2-staged cleaner float that produces a primary, cleaner and re-cleaner concentrate as well as a primary, cleaner and re-cleaner tail. Three concentrates are floated off over 26 minutes for the primary float. The times are at 3, 8 and 15 minutes. For the cleaner and re-cleaner, three concentrates are floated off over 3, 8 and 10 minutes. This test provides a grade recovery curve from which cleaner and recleaner recovery estimates can be made. Release analysis flotation products are analysed and provide information on Au, S, As, Fe and total organic carbon (TOC) recoveries. This is the primary test used to determine the optimum final float recovery for the Macraes flotation circuit.

- A standard PRF leach is conducted. This test assesses the preg-robbing characteristics of the ore.

The results of previous ore source testing is summarised in Table 16.2 by ore type. The ore source added to reserves recently is the Southern Pit. An analysis of plant performance (flotation) over the three previous cutbacks of Southern Pit in 1994/95, 1997/98 and 2000/01 were 88.4, 84.9 and 87.1 respectively, and targeting lower concentrate grades than current practice requires. The current reserve utilises an assumed flotation recovery therefore of 85%. Further metallurgical samples are being collected in 2010 to confirm this.

Only the 2000/01 campaign involved autoclave operation with a 93.9% CIL recovery achieved, many operational developments treating high PRF ores in both the flotation and CIL circuits have since been implemented to offset the impact of a higher concentrate grade target.

Table 16.2: Flotation Recoveries from Future Ore Testing Program

Ore Source	Hanging Wall (%)	Concordant Load (%)	Stockwork (%)
Frasers North 4	90.3	87.1	87
Frasers North 5	89.2	87	87
FRUG	92.1	-	-
Golden Bar	89.8	89.8	89.8
Southern Pit	85.0	93.0	79.0

It is believed the process for calculating flotation and CIL recoveries used are appropriate for predicting expected performance when treating future ores over the remaining LOM.

16.9 Issues

Allocation of gold between Macraes open pit and underground mines has been an issue. Higher underground gold grades could reasonably be expected to return higher recoveries and produce concentrates with higher gold grades. In practice, measuring actual flotation recovery and concentrate grades produced individually by FRUG or Macraes open pit ores is not possible. These differences, although not affecting total gold recovered, do impact on the financial performance of both mines. Investigations to more accurately measure and attribute gold recovered from Macraes open pit versus FRUG ore streams will continue during 2010.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES FOR GOLD

17.1 Mineral Resource Inventory

Mineral Resource estimates compliant with CIM standards for the Macraes Project as at December 31, 2009 by resource category and deposit are shown in Table 17.1. The Measured and Indicated Mineral Resource totals 84.25Mt at an average grade of 1.13 g/t Au for a total of 3.05Moz of gold.

The Round Hill, Southern Pit and Innes Mills resources have been reintroduced to the inventory and did not appear on the previous, December 31, 2008 inventory. For a discussion of these resource reinstatements, refer to sections 17.6 and 17.8.

Table 17.1: Macraes Resource Inventory as at December 31, 2009

Resource Cut-off	Resource Area	Measured		Indicated		Measured & Indicated			Inferred Resource		
		Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
0.5 g/t	Coronation	.	.	1.23	1.18	1.23	1.18	0.05	2.98	1.1	0.11
0.5 g/t	Deepdell	0.23	1.67	.	.	0.23	1.67	0.01	0.32	1.0	0.01
0.5 g/t	Golden Point	.	.	.	.	.	.	.	1.48	2.6	0.12
0.4 g/t	Round Hill	4.06	0.98	20.18	0.93	24.24	0.94	0.73	18.38	1.2	0.72
0.5 g/t	Southern Pit	0.52	0.90	2.90	0.85	3.41	0.85	0.09	.	.	.
0.5 g/t	Innes Mills	0.04	1.94	0.74	1.08	0.79	1.13	0.03	0.23	0.7	0.01
0.5 g/t	Frasers Pit	12.14	1.53	28.36	0.91	40.50	1.10	1.43	9.41	0.7	0.21
No cut-off	Frasers Underground P1 & P2	1.38	3.10	4.88	2.21	6.26	2.41	0.48	4.62	2.0	0.30
No cut-off	Frasers Underground Panel2 Deeps	.	.	0.35	3.56	0.35	3.56	0.04	0.54	3.7	0.06
0.5 g/t	Golden Bar	0.09	1.56	1.18	1.40	1.27	1.42	0.06	4.96	1.4	0.22
0.5 g/t	Taylors	.	.	0.28	1.50	0.28	1.50	0.01	0.41	1.1	0.01
0.5 g/t	Stockpiles	5.69	0.64	.	.	5.69	0.64	0.12	.	.	.
	Macraes Total	**24.15**	**1.30**	**60.09**	**1.06**	**84.25**	**1.13**	**3.05**	**43.34**	**1.3**	**1.77**

All Mineral Reserves reported are fully included in the Mineral Resources reported for the same deposit.

17.2 Qualified Persons Responsible for Resource Estimates

Mr Jonathan Moore, Resource Geologist, Oceana is the Qualified Person responsible for the Macraes Project resource estimates. This section summarises the methodology used by Oceana and its consultants to prepare and classify the Mineral Resource estimates for the Macraes Project.

17.3 Coronation

17.3.1 Introduction

The Coronation resource area extends from the northern boundary of the Deepdell resource area (local grid coordinates 18,250mN to 20,500mN; 69,200mE to 70,500mE).

The resource estimate presented in this report was completed in June 2009, subsequent to recently completed infill drilling. Since the previous Technical Report for the Macraes Project, November 9, 2009 (the 2009 Report), this estimate has been integrated into the Macraes reserves.

The revised resource estimate adds 1koz of indicated resource and 96koz of inferred resource.

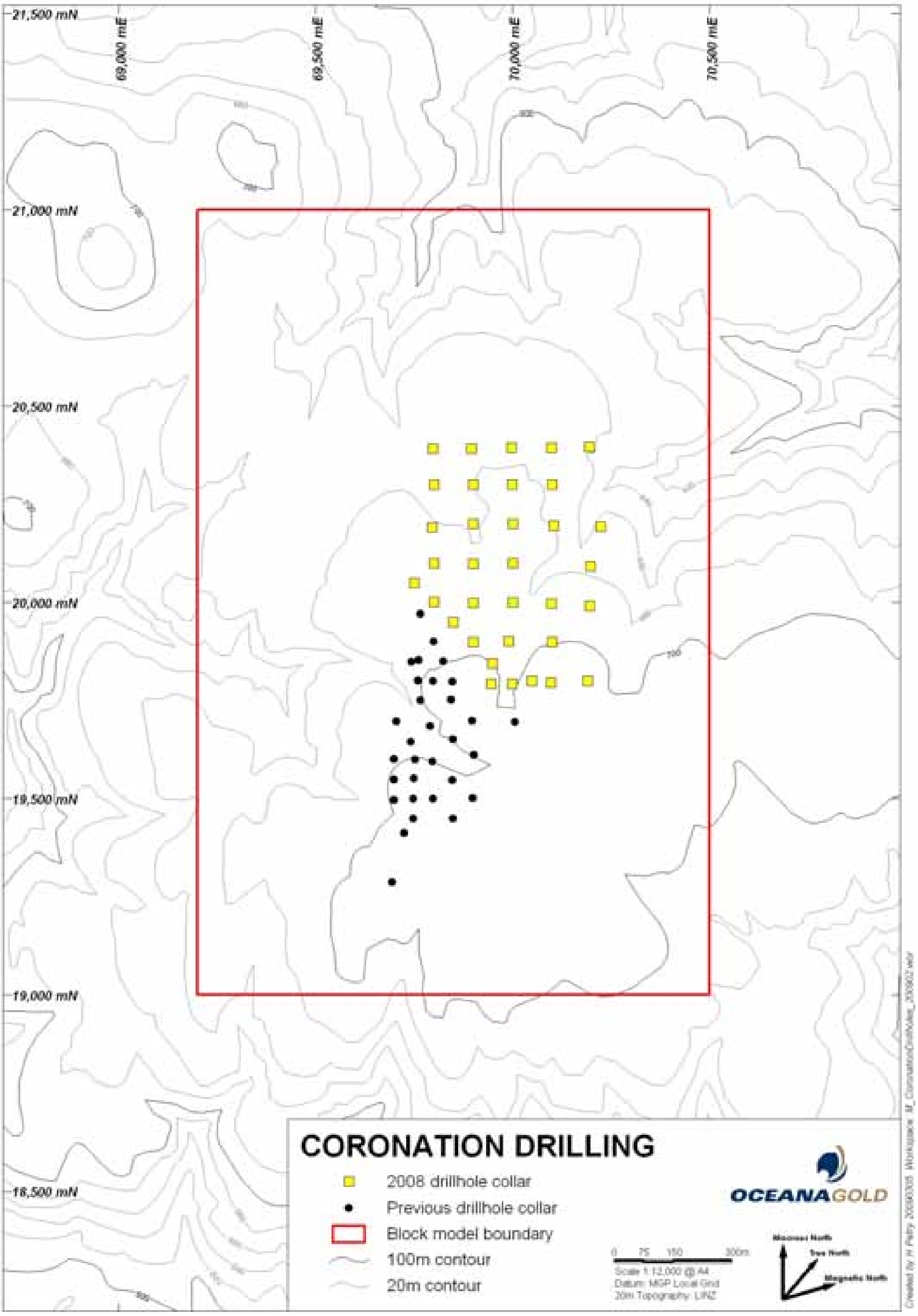
17.3.2 Database

The Coronation deposit is defined by 65 drill holes completed in three phases of drilling. The first phase was completed in 1998, when 13 RC percussion holes for 600m were completed as part of exploration drilling to test the strike extension of the HMSZ. A second phase of RC percussion drilling was completed in 2001 when 18 holes for 1,284m were drilled. The third phase was completed in late 2008 and comprised 33 RC drill holes for 2,966m as well as a single diamond drill hole for 163m. This has brought the drill hole density to between 50 x 50m and 50 x 100m for much of the resource area. The location of the drill hole collars is shown on Figure 17.1: Coronation Deposit - Drill Hole Collar Plan, with the drilling statistics provided in Table 17.2. Minimal diamond drilling has been completed on the deposit and only one RC percussion drill hole has drilled through the Footwall Fault.

Table 17.2: Coronation Deposit - Drilling Summary

Hole Type	CO09 Resource Estimate		
	Number	**Metres**	**Percentage**
Percussion	-	-	-
Reverse Circulation	64	4,850	97
Diamond* (DDH & RCD)	1	163	3
Total	**65**	**5,013**	**100**

Figure 17.1: Coronation Deposit - Drill Hole Collar Plan



17.3.3 Geological Model

The HMSZ at Coronation manifests as a predominately pelitic package of schist up to 90m thick. The package is constrained above by the Hangingwall Shear and below by the Footwall Fault as shown on Figure 17.2. The geology of the Coronation deposit is comparatively simple. It comprises the Hangingwall Shear which has a generally planar geometry and dips 15° to 20° to the east. A second, less extensive shear has been interpreted immediately below the Hangingwall Shear. A north-south subvertical fault has been interpreted to offset both mineralised shears.

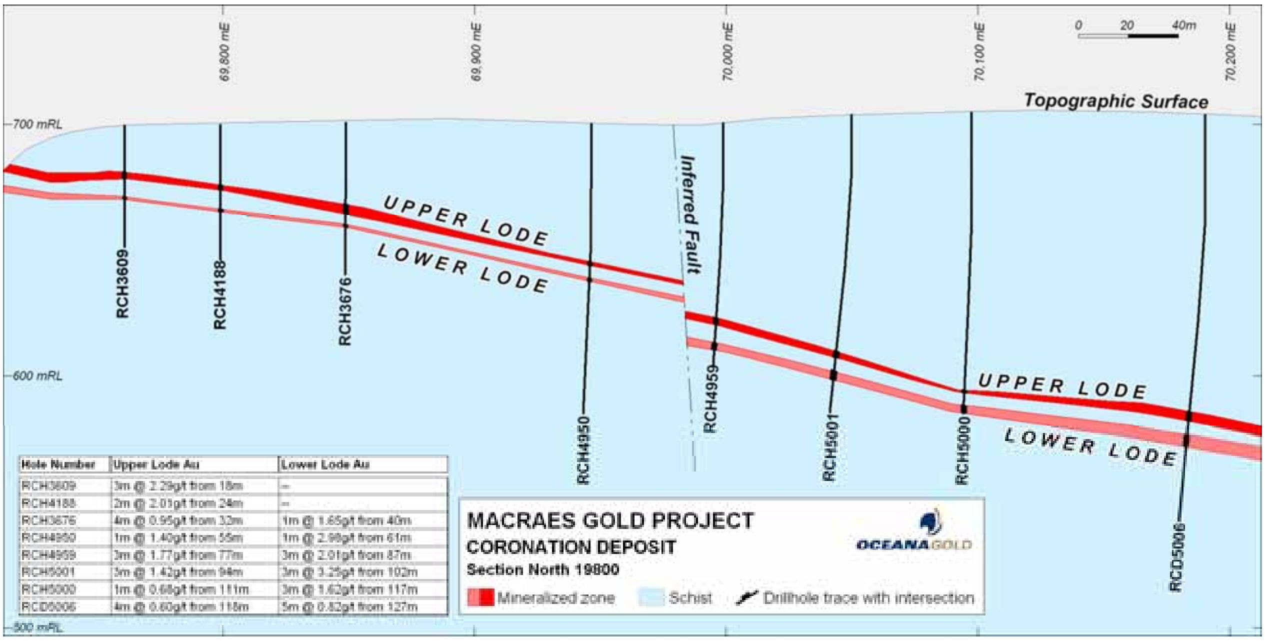
All geological wireframes were constructed in *MINESIGHT.* Due to the overwhelming proportion of RC drilling (RC chip logging is considerably less definitive than diamond core logging), gold grades were used as a proxy for geology. This approach provides a good basis for the upper shear surface. The lower surface however is less defined although can be reasonably approximated by a 0.25 g/t Au to 0.50 g/t Au lower grade cut-off.

The bulk mining (overburden) surface was built approximately 10m above the Hangingwall. Similarly, the footwall plane was approximated by stepping down 50m below the lower lode footwall surface. All samples between the footwall and the bulk mining surface, not captured by the upper or lower lode wireframe, were deemed unconstrained mineralization and comprise the grade control zone.

The Coronation resource estimate uses large block MIK recoverable resource estimates, which is the approach used to estimate most of the resources at Macraes.

An oxide surface was constructed on the basis of logged colour, perceived degree of weathering and sulphur grade.

Figure 17.2: Coronation Deposit - Cross Section



17.3.4 Historic Mining

The Coronation lode was probably discovered during the late 1880's when the Macraes Flat area was the subject of intensive prospecting for reef gold (Petchy, 1998). Two main areas of gold workings can still be distinguished, the Coronation workings and water races related to alluvial workings.

From the limited evidence available, it would appear that the Coronation area was first worked during 1888 with a second period of activity in 1911/1912. The historic workings do not impinge on the resource reported.

17.3.5 Statistical Analysis

The drill hole data was coded with the mineralization interpretation. Based on this coding, 1m composites (down-the-hole composites) were generated and applied to subsequent statistical and geostatistical evaluations.

A total of 2,709 one metre composites were available for investigation. Table 17.3 presents the statistical summary for the composites grouped by domain. The upper lode (Domain 1) captures the majority of the anomalous mineralization and returns a mean grade of 1.16 g/t Au and a coefficient of variation (CV) of 0.85. The spatially limited lower lode (Domain 2) contains 51 composites and has a mean grade of 1.34 g/t Au and CV of 0.98. The grade control zone (Domain 3) captures 2,472 data with the mean grade and CV calculated at 0.07 and 3.6 respectively.

The histogram in Figure 17.3 and Figure 17.4 show positively skewed distributions for the Domains 1 and 2 composites respectively. No histogram is shown for Domain 3.

Table 17.3: Coronation Deposit - Summary Statistics 1m Gold (g/t Au) Composites

Domain	Number	Mean	Median	Minimum	Maximum	CV
1	186	1.16	0.94	0.00	5.92	0.85
2	51	1.34	1.10	0.00	5.45	0.88
3	2,472	0.07	0.02	0.00	9.75	3.6

Figure 17.3: Coronation Deposit - Domain 1 Histogram Plot 1m gold (g/t Au) Composites

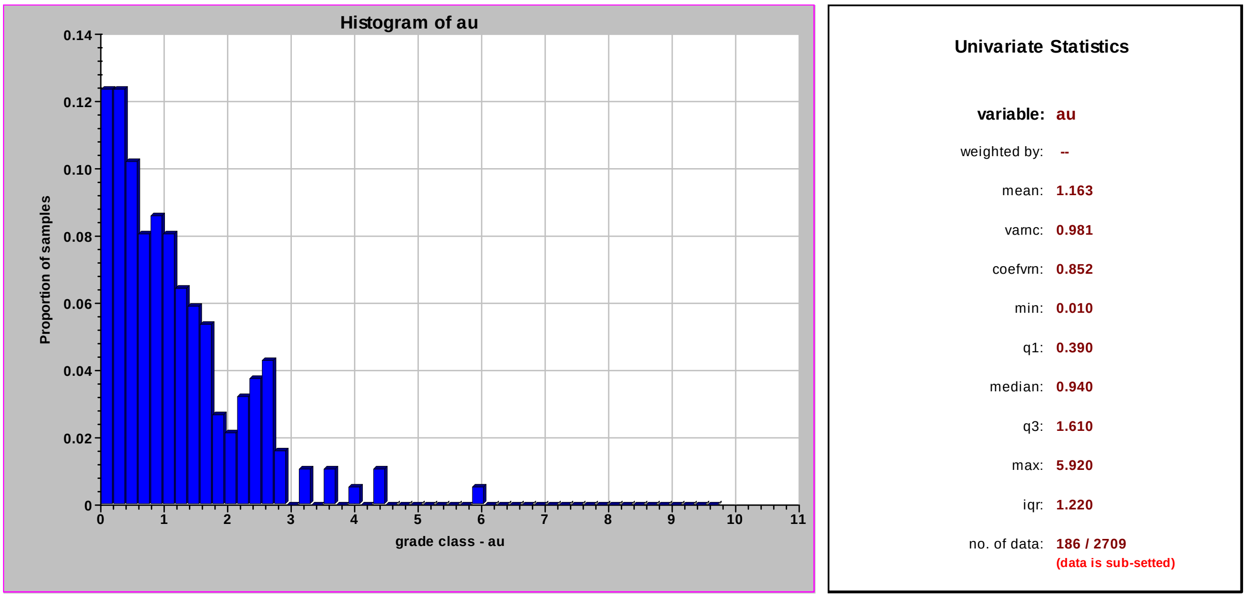
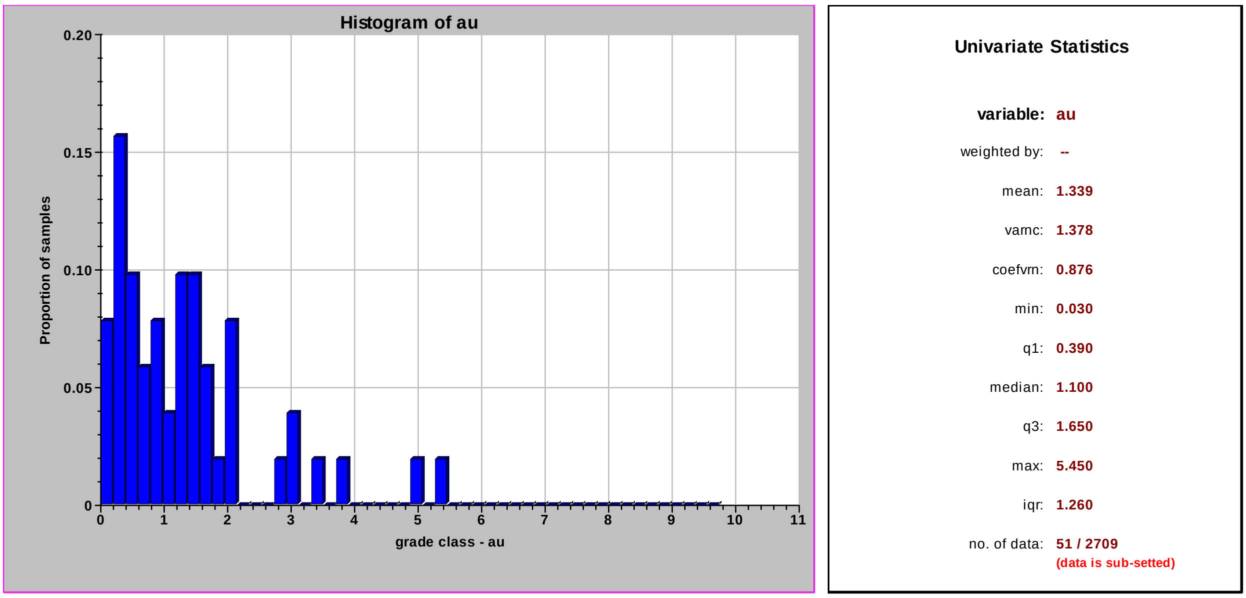


Figure 17.4: Coronation Deposit - Domain 2 Histogram Plot 1m gold (g/t Au) Composites



17.3.6 Variography

Despite the low number of samples, multiple indicator kriging was used.

Experimental variograms were generated for the Coronation deposit based on only 0.2, 0.4, 0.6, 0.8, 0.9 and 0.95 probability indicators. As a conservative measure, the Domain 1, 95% indicator, top class mean was replaced with the median for this class.

17.3.7 Block Model

A regular block model was constructed for the purposes of grade estimation with extents from 69,200mE to 70,500mE, 19,000mN to 21,000mN, and 400 to 730mRL. A 25mE x 25mN x 2.5mRL parent cell size was used. A bulk density was applied based on weathering. Oxidised material was assigned a bulk density of 2.5 t/m^3, while fresh material was assigned a 2.6 t/m^3 bulk density.

Table 17.4 summarises the Coronation block model parameters.

Table 17.4: Coronation Deposit - Block Model Parameters

	Resource Estimate Limits	
	Limits	**Block Size**
Easting	69,200 – 70,500	25.0 m
Northing	19,000 – 21,000	25.0 m
RL	400 – 730	2.5 m

17.3.8 Grade Estimation

Grade estimation was completed in Hellman and Schofield proprietary *GS3* software.

The sample search parameters are presented below in Table 17.5.

Table 17.5: Coronation Deposit - Sample Search Parameters

Domain	X,Y,Z Search	Rotations	Minimum Number	Maximum Number	Octant #
1 Primary Search	60, 60, 10	Y 12	16	48	4
1 Secondary Search	78, 78, 13	Y 12	16	48	4
1 Tertiary Search	78, 78, 13	Y 12	8	48	4
2 Primary Search	60, 60, 10	Y 12	16	48	4
2 Secondary Search	78, 78, 13	Y 12	16	48	4
2 Tertiary Search	78, 78, 13	Y 12	8	48	4
3 Primary Search	25, 25, 10	Y 12	16	48	4
3 Secondary Search	32.5, 32.5, 13	Y 12	16	48	4
3 Tertiary Search	32.5, 32.5, 13	Y 12	8	48	4

Both visual and statistical comparisons of the grade estimate versus the input composites have been completed. Table 17.6 below shows the mean estimated grades are significantly lower than the original composite means as a result of clustering.

Table 17.6: Coronation Deposit - Comparison of Model to Composite Mean Grades

	Au Means (g/t)		
Domain	1	2	3
Composite Mean	1.16	1.34	NA
Model Mean	1.02	1.19	NA

17.3.9 Resource Reporting

Table 17.7 below presents the resource classification scheme that was used for Coronation by Oceana. The scheme uses a combination of geological confidence and drilling density.

Table 17.7: Coronation Deposit - Resource Classification Methodology

	Measured	Indicated	Inferred
1 and 2	N/A	Approx 75 x 75 metres	100 x 100 metres
3	N/A	none	32.5 x 32.5 metres

Areas of the Hangingwall resource drilled to 75 by 75m have been classified as indicated. Areas drilled up to 100 by 100m have been classified as inferred. This classification is supported by the substantial mining and reconciliation history at Macraes from which an understanding of the geological controls and local variability of mineralization has been gained. Infill drilling is intended to bring the target volume to at least 37 by 37m prior to mining.

The Coronation deposit Mineral Resource, by material type, cut-off grade and classification is presented in Table 17.8

Table 17.8: Coronation Deposit - Mineral Resource Grouped by Resource Category

	0.50 g/t Cut-off						
	Sulphide		Oxide		Total		
Category	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (koz)
Indicated	1.14	1.18	0.09	1.14	1.23	1.18	47
Inferred	2.88	1.12	0.10	1.15	2.98	1.12	107

Note: Mt = million tonnes, koz = 000's contained ounces

17.4 Deepdell

17.4.1 Introduction

The Deepdell deposit is located between the Coronation and Golden Point deposits. Oceana and its consultants have estimated the resources at Deepdell using MIK grade interpolation. Significant production has been reported from Deepdell, representing 2.41Mt at a grade of 1.53 g/t Au for some 118koz when applying a 0.5 g/t Au lower cut-off grade.

17.4.2 Resource Database

The resource estimation database comprises both drilling data and trench data. In total 330 drill holes for 29,048m were applied to resource estimation. Drilling statistics are provided as Table 17.9 with Figure 17.5 displaying the drill hole collars.

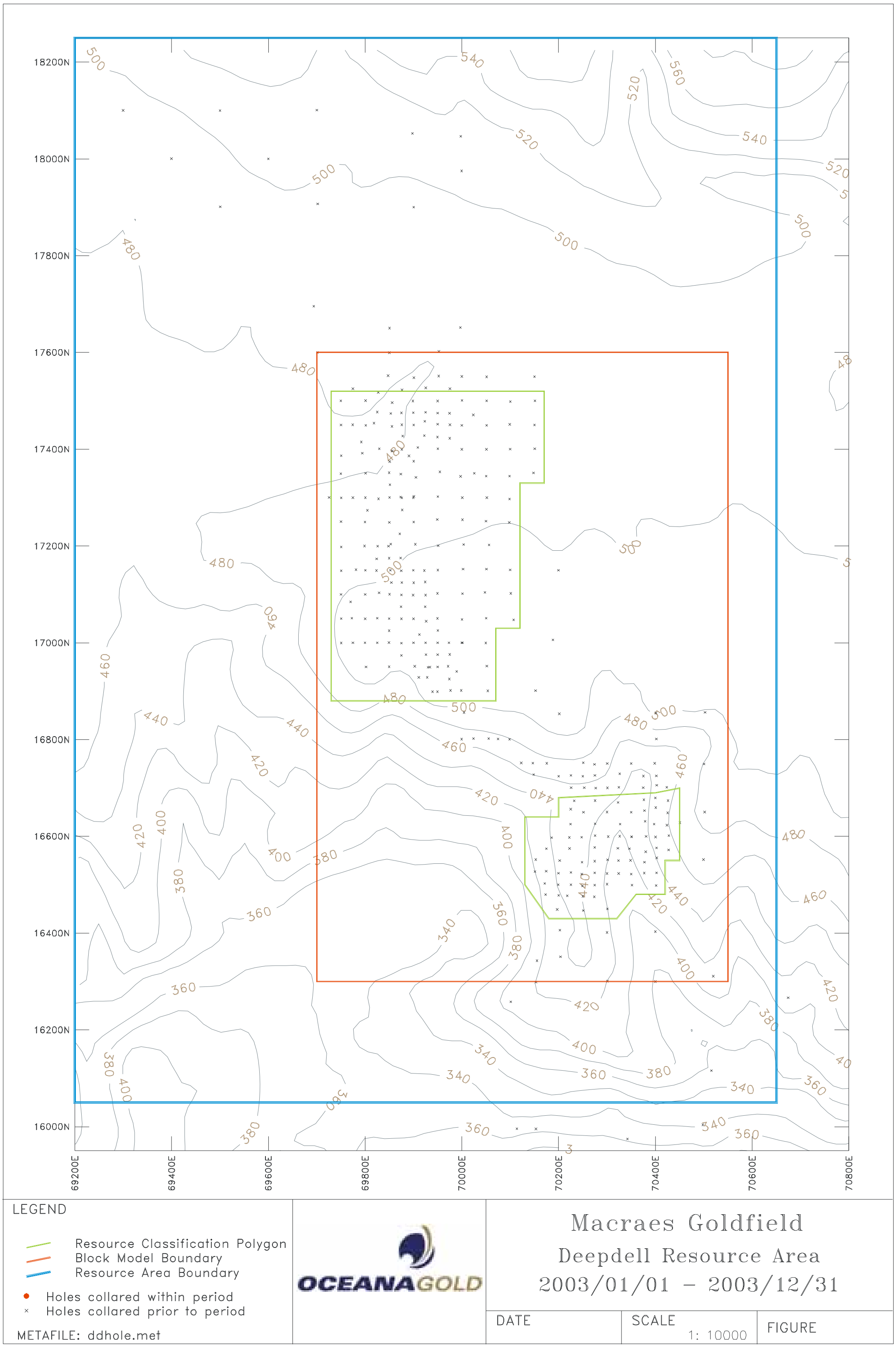
Table 17.9: Deepdell Deposit - Drilling Summary

	DD01a Resource Estimate			Prospect Total		
Hole Type	Number	Metres	Percentage	Number	Metres	Percentage
Percussion	5	310	1.1	5	310	1.1
Reverse Circulation	309	27,260	93.8	313	27,998	94.0
Diamond* (DDH & RCD)	16	1,478	5.1	16	1,478	5.1
Total	**330**	**29,048**	**100.0**	**334**	**29,786**	**100.0**

*Diamond figures include holes drilled with an RC percussion pre-collar and tailed with diamond core.

In addition to the drilling, 5 trenches totalling 990m were excavated across the surface expression of the Hangingwall and associated soil arsenic anomalies. The mapping and sampling information for each trench was converted into a sub-horizontal drill hole and was used in the geological interpretation of the resource. The trench assay data was not used to interpolate resource estimate grades.

Figure 17.5: Deepdell Deposit - Drill Hole Collar Plan

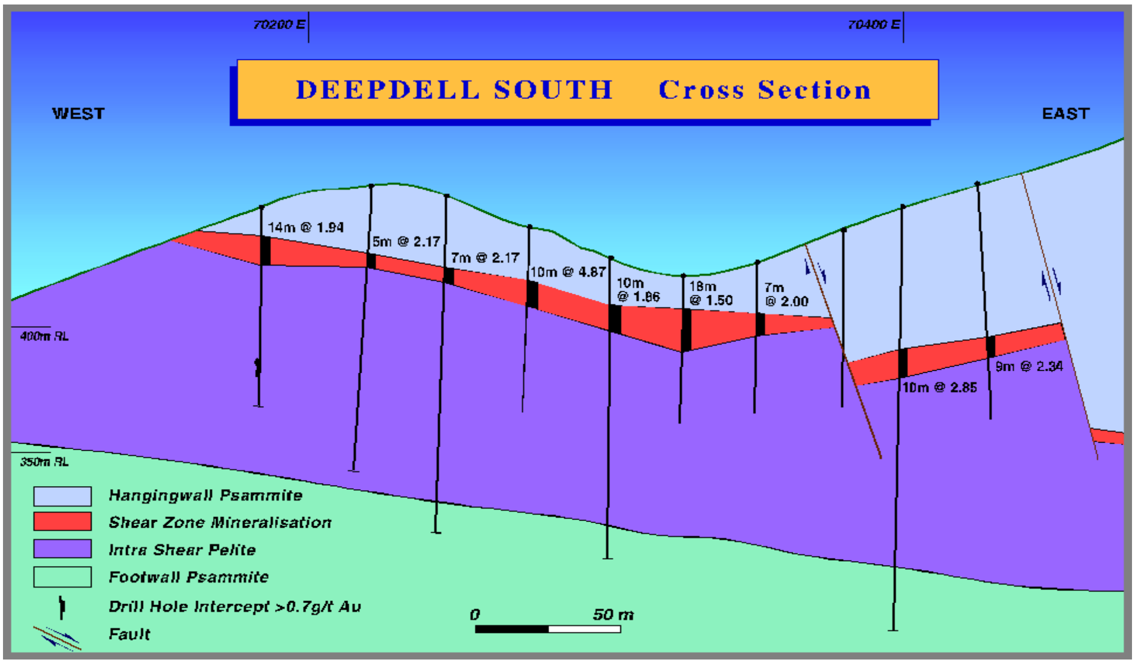


17.4.3 Geological Model

The HMSZ at Deepdell consists of a 50 to 60m thick pelite, constrained by the hangingwall and footwall shears. The geology of Deepdell North is comparatively simple. It comprises the hangingwall shear, which has a planar geometry and dips 15° to 20° to the east. Beneath the hangingwall shear, 3 sub parallel shears have been identified. These shears are generally thin (less than 3m thick), weakly mineralized, do not have the continuity of the hangingwall and are not economically significant.

At Deepdell South the hangingwall shear geometry is a little more complex (Figure 17.6). The hangingwall shear has been rotated into a south to south-east orientation and is cut by a northeast-southwest striking fault. The western portion of the hangingwall dips at 20° to 25° to the south-east while the eastern section dips at 35° to 40° to the south-west. The hangingwall shear is well developed to approximately 70,400mE where it is either offset by a north-south trending fault or is pinched out against a fault. At both Deepdell North and Deepdell South stockwork development is relatively poor.

Figure 17.6: Deepdell Deposit - Cross Section



A complex fault zone separates Deepdell South from Deepdell North. Four east-west trending faults, which terminate against a northeast – southwest trending fault, have been interpreted. From Deepdell South to Deepdell North the effect of these faults is to uplift the hangingwall and progressively displace the hangingwall outcrop position to the west.

Three lodes have been historically mined in the Deepdell area, namely: Golden Bell, Maritana and Deepdell. The Golden Bell and Maritana lodes are located on the hangingwall psammite-pelite contact (hangingwall shear) and were mined for gold. The Deepdell lode is located in the hangingwall psammite and was historically mined for scheelite. No resource estimate has been compiled for the Deepdell lode.

At Deepdell three styles of mineralization have been interpreted, the hangingwall shear, east dipping concordant lodes and unconstrained stockwork. The same interpretation approach that has been used for Frasers, Innes Mills, Southern Pit, Round Hill and Golden Point has been used at Deepdell. This involves the collective use of geological logging and 3D geometric correlation to interpret the upper contact of the shears. Where possible, the same approach has been used for the lower contacts. Where this is not possible, an approximate 0.4 g/t cut-off has been used to estimate the position of the lower shear contacts. Mine reconciliation since 1991 has shown this to be a reasonable approach.

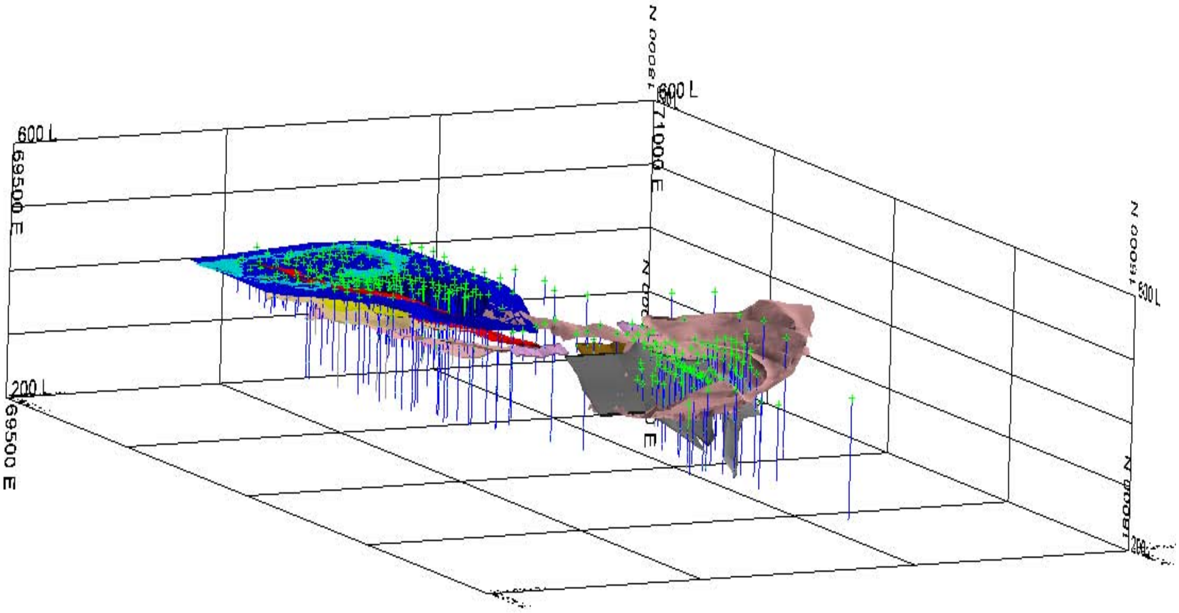
The mineralization styles and the respective codes identified at Deepdell are shown in Table 17.10.

Table 17.10: Deepdell Deposit - Mineralization Domain Codes

Mineralization Style	Domain Codes
Hangingwall Shear Zone	102, 103, 104, 111 to 115
East Dipping Concordant Lodes	301 to 303
Unconstrained Stockwork	501

The stockwork at Deepdell has been modelled as unconstrained. In the resource estimate the stockwork interpolation is constrained by the bulk mining surface, topographic surface and the footwall shear.

Figure 17.7: Deepdell Deposit - Drilling and Interpreted Domains



17.4.4 Statistical and Geostatistical Analysis

Statistical analysis has been based on 1m composites coded with the mineralization/geological interpretation. The composite data has generated as 1m run length composites.

Summary statistics for the gold data, grouped by interpreted domain, is shown in Table 17.11. Similar statistics are noted for the domain groupings (i.e.: hangingwall zones etc) with the highest mean grade reported for Domain 301. No high grade cutting has been applied during estimation although the highest bin grade used in the MIK estimation has been modelled with the median bin grade as opposed the mean bin grade. This effectively reduces the influence of high grade data and replicates the benefit of high grade cutting.

Table 17.11: Deepdell Deposit - Summary of Gold (g/t) 1m Composite Statistics

Domain	Number	Mean	Minimum	Maximum	Std Dev	CV
103	728	1.52	0.01	25.0	1.97	1.30
104	31	1.02	0.06	9.50	1.68	1.65
111	142	1.82	0.01	8.05	1.42	0.78
112	356	1.82	0.01	8.13	1.18	0.65
113	42	1.45	0.20	4.62	1.05	0.72
114	31	1.25	0.52	2.64	0.53	0.42
115	25	1.63	0.19	7.31	1.57	0.96
301	100	1.25	0.37	4.83	0.85	0.68
501	17,859	0.10	0.01	25.75	0.66	6.60

Indicator variograms were generated by Snowden Mining Industry Consultants Pty Ltd using VISOR software for the 103/104, 111, 112, 301 Domains and a combined Domain of 102, 302, 303, 501. Variogram models were fitted using nested spherical models for the first and second structures. Only the 103/104 Domain showed interpretable structure with all other domains having insufficient composite data to produce robust variography. Therefore, Oceana based the strike and dip planes on the observed trends in the data. Indicator cut-offs used for the multiple indicator kriging are tabulated in Table 17.12 while indicator variogram parameters are tabulated in Table 17.13 and Table 17.14.

Table 17.12: Deepdell Deposit - Indicator Cut-offs

Percent	103/104	111	112	113	114	115	301	501
5.0	0.40	0.40	0.10	0.42			0.10	0.10
10.0	0.58	0.64	0.57	0.56	0.78	0.42	0.56	0.11
20.0	0.73	0.86	0.71	0.77	0.81	0.69	0.64	0.12
30.0	0.88	0.99	0.90	0.86	0.94	0.89	0.74	0.14
40.0	1.05	1.19	1.09	0.98	0.97	0.92	0.87	0.17
50.0	1.28	1.32	1.42	1.15	1.11	1.20	1.01	0.21
60.0	1.57	1.83	1.73	1.47	1.15	1.39	1.10	0.25
70.0	1.96	2.36	2.23	1.94	1.42	1.61	1.36	0.30
80.0	2.58	2.86	2.87	3.25	1.68	1.93	1.73	0.39
90.0	3.52	3.46	3.92	4.21	1.98	2.86	2.06	0.52
95.0	4.59	4.88	4.52	4.48	2.11		2.47	0.83
97.5		5.36	5.03				4.13	1.30
Median Above Last	7.91	5.85	5.58	4.59	2.47	4.25	4.71	2.82

Table 17.13: Deepdell Deposit - Indicator Variogram Parameters Domains 103/104

								Rotation Parameters		
Ind Cut-	Class	Percentile	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)	New N	Dip N	Dip E
0.40	0.20	5	0.20	0.60	0.20	60x40x3.5	75x120x5	0	0	-15
0.58	0.49	10	0.20	0.60	0.20	60x40x3.5	75x120x5	0	0	-15
0.73	0.66	20	0.20	0.60	0.20	60x40x3.5	75x120x5	0	0	-15
0.88	0.81	30	0.20	0.65	0.15	50x40x1.5	50x80x4	0	0	-15
1.05	0.97	40	0.20	0.60	0.20	50x40x1.5	50x80x4	0	0	-15
1.28	1.17	50	0.20	0.80		55x55x2.5		0	0	-15
1.57	1.43	60	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15
1.96	1.77	70	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15
2.58	2.27	80	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15
3.52	3.05	90	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15
4.59	4.06	95	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15
5.99	5.29	100	0.70	0.16	0.14	20x25x4.0	55x55x4	0	0	-15

Rotation parameters above are for domain 103/104 only. For Domains 111 to 115, 301 and 501 the median variogram was applied to all the indicators shown in Table 17.12. The variogram parameters used are shown in Table 17.14.

Table 17.14: Deepdell Deposit - Indicator Variogram Parameters Domains 111 to 115, 301 and 501

						Rotation Parameters		
Domain	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)	New N	Dip N	Dip E
111	0.32	0.68		45x45x3		-50	0	30
112	0.59	0.29	0.12	30x39x4.5	30x65x4.5	0	20	-10
113	0.59	0.29	0.12	30x39x4.5	30x65x4.5	0	0	-12.5
114	0.59	0.29	0.12	30x39x4.5	30x65x4.5	0	0	-20
115	0.59	0.29	0.12	30x39x4.5	30x65x4.5	0	0	0
301	0.20	0.80		50x60x2.0		0	0	0
501	0.65	0.23	0.12	25x25x3.5	50x80x17	0	0	-10

17.4.5 Block Model

A regular block model was constructed for the purposes of grade estimation, as summarised in Table 17.15. A 10m x 5m x 2.5m parent cell size was used. Given the relative data spacing the selected block size is inappropriate and will result in an over smoothed grade estimate.

Table 17.15: Deepdell Deposit - Resource Estimate Limits and Block Model Dimensions

	DD01a	
	Limits	**Block Size**
Easting	69,700 - 70,550	10.0 m
Northing	16,300 - 17,600	5.0 m
RL	250 - 525	2.5 m

A bulk density of 2.6 t/m^3 has been applied to all blocks for resource reporting.

17.4.6 Grade Estimation

Multiple Indicator Kriging and median Indicator Kriging (medIK) have been used to estimate the various domains at Deepdell. Domains 103 and 104 have been estimated by MIK and all other domains have been estimated by medIK. All estimation has been completed in *MINESIGHT*.

The estimation sample search parameters applied in estimation are summarised in Table 17.16. The grade estimate reports only whole block grades which have been estimated by multiplying the mean grade of each indicator class (median grade above the highest indicator threshold) by the estimated intraclass probability estimated in the MIK and medIK runs. The indicator thresholds and mean/median grades are presented in Table 17.6.

As displayed below in Table 17.16, the minimum number of composites used for kriging was 2 and the maximum number was 50. Block discretization of 4 x 2 x 1 was used. Domain control was used in estimation wherein only data coded as that domain was used in the estimation of that domain.

Table 17.16: Deepdell Deposit - Sample Search Parameters

Domain	Minimum Number of Samples	Maximum Number of Samples	Octant Constraint	Sample Search Distances X,Y,Z (m)	X,Y,Z Discretization	Maximum Samples per Drill Hole
103	2	50	none	50, 50, 10	4, 2, 1	5
104	2	50	none	50, 50, 10	4, 2, 1	5
111	2	50	none	50, 50, 10	4, 2, 1	5
112	2	50	none	50, 50, 10	4, 2, 1	5
113	2	50	none	50, 50, 10	4, 2, 1	5
114	2	50	none	50, 50, 10	4, 2, 1	5
115	2	50	none	50, 50, 10	4, 2, 1	5
301	2	50	none	50, 50, 10	4, 2, 1	5
501	2	50	none	50, 50, 25	4, 2, 1	5

17.4.7 Validation and Reconciliation

Oceana validated the resource estimate visually and statistically.

The resource estimate was plotted on screen in section and plan and viewed in 3D in *MINESIGHT*. Aspects checked included geological coding, classification coding, weathering state, and grade interpolation. The block grades were visually compared to surrounding composite grades and considered to reasonably reflect the input data. The mean composite grade versus average model grade, calculated for each domain (Table 17.17), shows adequate reproduction of the input data. P. Blackney of Snowden

Mining Industry Consultants Pty Ltd (Snowden, 1999) completed a review of the kriging parameters and the resultant resource estimates and considered the estimates acceptable.

Table 17.17: Deepdell Deposit - Comparison of Input Composites versus Block Model Grades (g/t Au)

Domain	Sample Mean	Model Mean
103	1.52	1.60
104	1.02	1.05
111	1.82	1.69
112	1.82	1.53
113	1.45	1.32
114	1.25	1.07
115	1.63	1.47
301	1.25	1.21
501	0.10	0.04

Oceana have completed a detailed assessment of the mining reconciliation. Mining at Deepdell commenced with Deepdell North Stage 1 in March 2001. Mining of Deepdell North Stage 1 was completed in April of 2002, Deepdell North Stage 2 in January 2003, and Deepdell South in October 2003.

The reconciliation of the resource estimate for oxide and sulphide ore types at two mining cut-offs is shown below in Table 17.18 to Table 17.20.

Table 17.18: Deepdell Deposit - Reconciliation at 0.5 g/t for Resource Estimate versus Mined Oxide

		Survey Adjusted Grade Control								Variance		
		Unfactored		Factored			Resource Estimate			Factored GC / Estimate		
Year	Estimate	Tonnes	g/t	Tonnes	g/t	oz	Tonnes	g/t	oz	Tonnes	g/t	oz
2001	DD99a	259,331	1.62	259,331	1.62	13,507	242,251	1.60	12,462	1.07	1.01	1.08
2002	DD01a	5,256	1.21	5,256	1.21	204	5,844	0.82	155	0.90	1.46	1.31
2003[2]	DD01a	262,108	1.37	262,108	1.43	12,051	384,955	1.78	22,030	0.68	0.80	0.55
Total		**526,695**	**1.49**	**526,695**	**1.52**	**25,761**	**633,050**	**1.70**	**34,647**	**0.83**	**0.89**	**0.74**

Table 17.19: Deepdell Deposit - Reconciliation at 0.7 g/t for Resource Estimate versus Mined Sulphide

		Survey Adjusted Grade Control					Resource Estimate			Variance		
		Unfactored		Factored						Factored GC / Estimate		
Year	Estimate	Tonnes	Tonnes	Tonnes	Tonnes	oz	Tonnes	g/t	Oz	Tonnes	g/t	oz
2001	DD99a	579,822	1.69	579,822	1.52	28,386	502,320	1.66	26,809	1.15	0.92	1.06
2002	DD01a	483,381	1.42	483,381	1.42	22,065	387,443	1.59	19,836	1.25	0.89	1.11
2003	DD01a	820,212	1.50	820,212	1.60	42,193	650,136	1.86	38,878	1.26	0.86	1.09
Total		**1,883,415**	**1.54**	**1,883,415**	**1.53**	**92,644**	**1,539,899**	**1.73**	**85,524**	**1.22**	**0.89**	**1.08**

[2] Note factoring was only applied to oxide in 2003

Table 17.20: Deepdell Deposit - Reconciliation Resource Estimate versus Mined Oxide at 0.5 g/t and Sulphide at 0.7 g/t

		Survey Adjusted Grade Control					Resource Estimate			Variance		
		Unfactored		Factored						Factored GC / Estimate		
Year	**Estimate**	**Tonnes**	**g/t**	**Tonnes**	**g/t**	**oz**	**Tonnes**	**g/t**	**Oz**	**Tonnes**	**g/t**	**oz**
2001	DD99a	839,153	1.67	839,153	1.55	41,893	744,571	1.64	39,271	1.13	0.95	1.07
2002	DD01a	488,637	1.42	488,637	1.42	22,269	393,287	1.58	19,991	1.24	0.90	1.11
2003	DD01a	1,082,320	1.47	1,082,320	1.56	54,243	1,035,091	1.83	60,909	1.05	0.85	0.89
Total		**2,410,110**	**1.53**	**2,410,110**	**1.53**	**118,406**	**2,172,949**	**1.72**	**120,171**	**1.11**	**0.89**	**0.99**

Over the period 2001 to 2003, the resource model under-predicted tonnes by 11%, over-predicted grade by 11%, and over-predicted contained gold by 1%. The comparison between grade control predicted tonnes and survey-adjusted tonnes for the same period (2,346kt versus 2,410kt) suggests mining dilution in the order of 3%, which is insufficient to account for the tonnage disparity seen in the resource estimate reconciliation. The difference is interpreted to represent the selectivity assumed in the resource modelling constraint interpretation. The small block modelling methodology was superseded at Oceana from 2002 onwards, although because no further production was anticipated in the short term at Deepdell, the model was not rebuilt.

In 2003 the ore from Deepdell was sourced from the Deepdell South Stage pit. Unlike Deepdell North Stages 1 and 2 the oxide/sulphide boundary interpretation was significantly deeper than was encountered during mining. This is shown in the poor oxide/sulphide tonnage reconciliation for 2003 in Table 17.18 and Table 17.19. Improved definition of the oxide/sulphide surface is required to improve the resource confidence for oxide zones.

Based on the above reported mining reconciliation, the estimates are considered robust if sufficient ore loss and dilution modifiers are applied as part of the mine planning process used in reserve conversion.

17.4.8 Resource Reporting

Resource classification polygons were drawn around the area of 25m x 25m drilling as shown on Figure 17.5 and were used to classify the resource estimate according to the criteria shown in Table 17.21.

All the area between the Deepdell North resource classification polygon and the Deepdell South resource classification polygon is regarded as Inferred due to the complexity of the geology. With consideration of the long mine history at Macraes and the relatively high confidence in the geological interpretation of the hangingwall mineralization zones, Oceana believes the resource classification scheme is reasonable. Despite this, the confidence in the grade estimates for the small volume of mineralization based on 50m x 50m spaced drilling could be increased with infill drilling.

Table 17.21: Deepdell Deposit - Resource Classification Methodology

	Drilling Density (metres)		
Mineralization Style	**25 x 25**	**50 x 50**	**100 x 100**
Hangingwall	Measured	Measured	Indicated
Concordant Lodes	Measured	Indicated	Inferred
Unconstrained Stockwork	Inferred	Inferred	-

The Mineral Resource for Deepdell, at the block cut-off of 0.5 g/t, is shown in Table 17.22. Based on the reconciliation data, the mining modifiers showed be reviewed if mining is considered.

Table 17.22: Deepdell Deposit - Resource Estimate as at December 31, 2009

Category	0.5 g/t Cut-off						
	Sulphide		Oxide		Total		
	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (koz)
Measured	0.19	1.71	0.05	1.55	0.23	1.67	13
Indicated	-	-	-	-	-	-	-
Inferred	0.29	1.0	0.04	1.7	0.32	1.0	11

Note: Mt = million tonnes, koz = 000's contained ounces

17.5 Golden Point

17.5.1 Introduction

Golden Point is located approximately 4km north of the Frasers Open Pit and has been considered a valid higher grade underground target. The estimated resource sits approximately 200m to the east of the eastern edge of the Round Hill resource. Oceana therefore drilled the down-dip extensions of Hangingwall mineralization at Golden Point as well as Round Hill and Innes Mills.

Four holes were drilled down-dip of the Golden Point pit toe (now back-filled) in 2007/2008, targeting higher grade Hangingwall mineralization previously intersected in drill holes DDH4772 and DDW4775. In total 1512.4m was completed with stratapac or PQ collars followed by HQ diamond coring. All holes intersected the Hangingwall Shear, although the width of mineralization thinned down-dip to 5m at DDH4919. In total, the Golden Point deposit database comprises 524 holes for 88,892.8m of drilling.

17.5.2 Historic Workings

Three lodes at Golden Point were mined historically. The lodes were named the Home Reef (equivalent to the Hangingwall Shear), the Dip Reef and the Low Reef. The mine workings were digitised from old mine plans and turned into 3-D wireframe solids which were used to assist with geological interpretation. When building the geological interpretation on paper and in *MINESIGHT* the areas of known historical workings were excluded from the wireframe. On this basis it was considered that the volume of ore removed by previous mining has also been removed from the resource estimate.

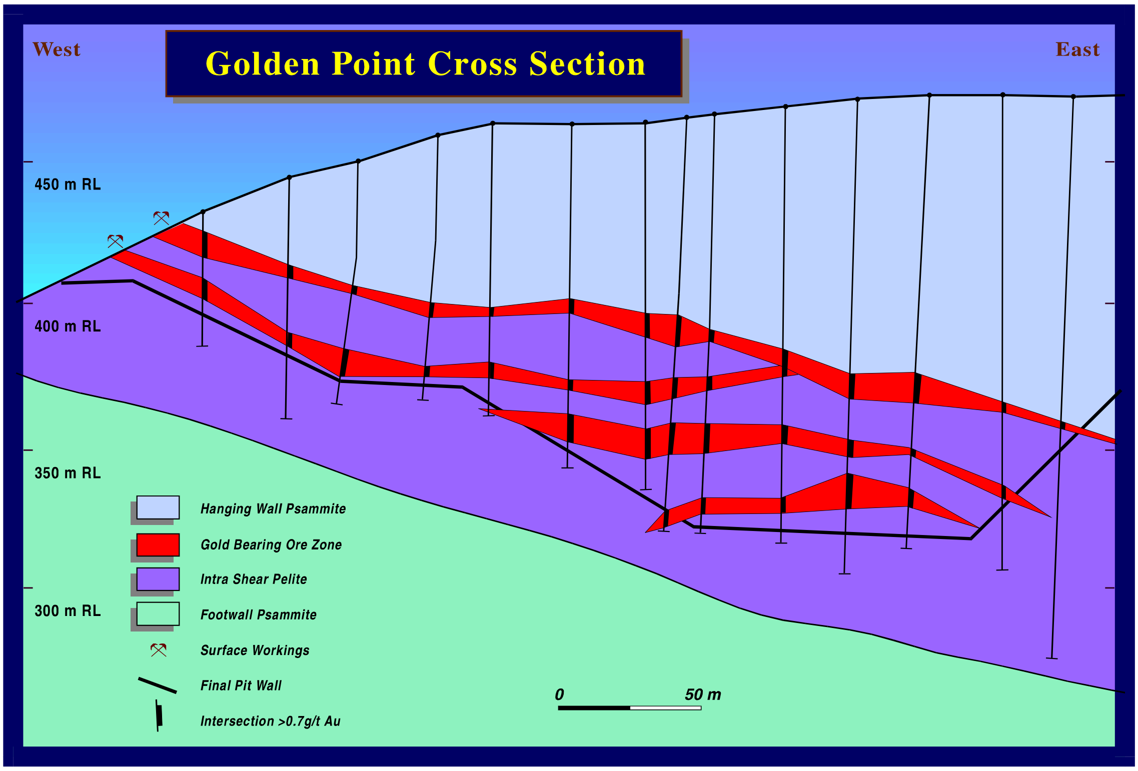
The old workings were predominantly at the northern end of Golden Point. The Golden Point Stage II cut back was centred on the northern end of Golden Point and was designed to recover a higher grade part of the resource. During 2000, Golden Point Stage II was mined to completion and in the process mined out all the known historical workings.

17.5.3 Geological Modelling

Gold mineralization around the Golden Point workings is contained within four shear structures which vary in thickness from 2 to 10m (Figure 17.8). The lodes dip gently to the east (10º to 15º) and are predominantly quartz veins and quartz breccia.

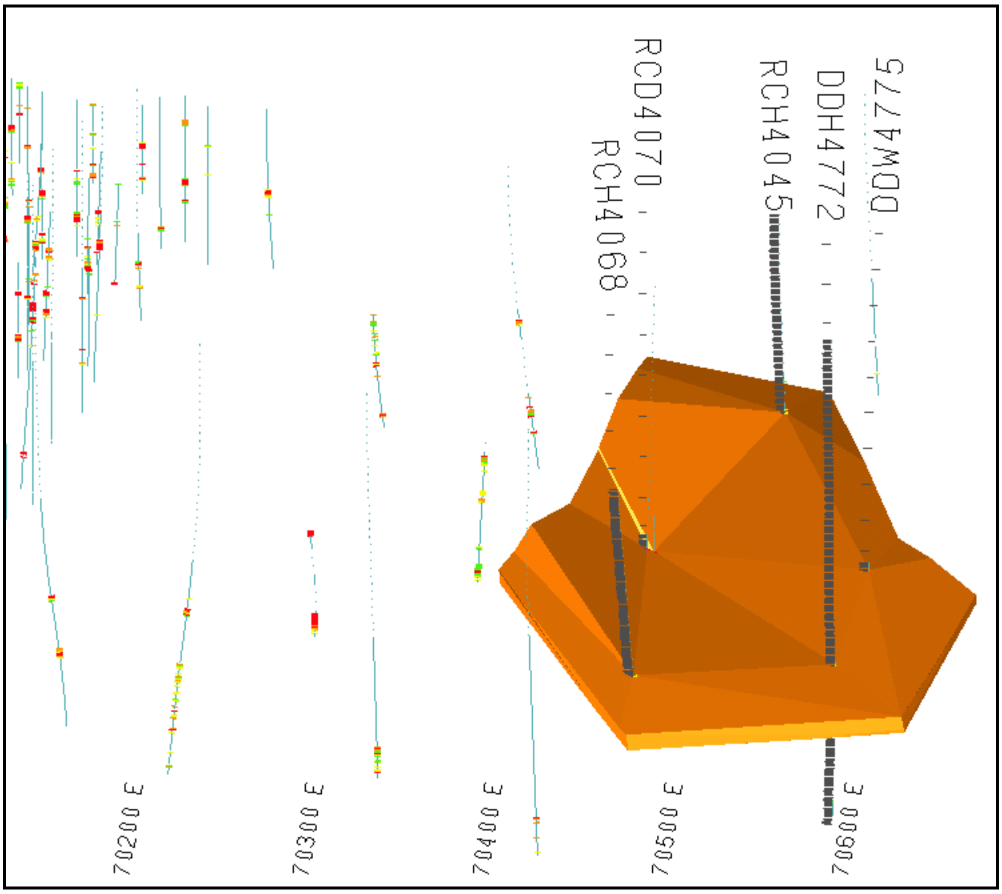
To the south of the Golden Point workings, infill drilling has allowed improved definition of the concordant lodes occurring below the Hangingwall shear. In this area the concordant lodes exhibit an anastomosing geometry with a number of bifurcations occurring in both the east-west and north-south orientations. Lodes typically thicken in the regions of convergence. The stockwork mineralization that comprises a large proportion of the adjacent Round Hill mineralization is noticeably absent at Golden Point.

Figure 17.8: Golden Point - EW Geological Cross Section



A geological interpretation was generated by extending the existing (defined in the pit region) Hangingwall interpretation into the area of new drilling. The upper and lower hangingwall contacts were interpreted using a combination of logged geology and gold assays. These were entered as points in *MINESIGHT* 3D software and were used to construct a 3D wireframe solid which has been extrapolated 50m down-dip beyond intersections.

Figure 17.9: Golden Point - Oblique View (Looking Down to NNW) of Underground Resource and Drill Holes



17.5.4 Sample Statistics

The Golden Point resource estimate is based on eight (8) drill holes. Two of these, DDH4772 and DDW4775, were drilled during 2005, three, RCH4068, RCD4069, and RCD4070, were drilled prior to 2005, and three, RCD4892, RCD4893 and RCD4894 were drilled during 2007.

Summary statistics of the 1m composite data captured within the interpretation is presented in Table 17.24. The significant intercepts captured in the interpretation are summarised below in Table 17.23.

Table 17.23: Golden Point - Hangingwall Drill Hole Intersection Summary

Hole Type	From (m)	To (m)	Interval (m)	Grade (g/t Au)
RCH4068	258	265	7	1.04
RCD4069	220	227	7	6.63
RCD4070	219	225	6	4.35
DDH4772	274	281	7	2.24
DDW4775	247	257	10	2.17
RCD4893	289	296	7	2.19
RCD4894	293	304	11	2.07
RCD4895	258	265	7	1.04

Table 17.24: Golden Point - Hangingwall Sample Statistics

Number	62
Mean (g/t Au)	2.75
Median (g/t Au)	1.72
Mean of 75m x 75m cells	2.55

17.5.5 Resource Estimate

A polygonal grade estimate was generated by Oceana within the area 70,410mE to 70,650mE, 15,660mN to 16,060mN, and 145mRL to 260mRL.

The estimate was generated for the hangingwall mineralization only and is constrained within the wireframe constructed from the drill hole intersections listed in Table 17.23. Based on the wireframe volume and a bulk density of 2.60 t/m^3, a mineralization tonnage of 1.48Mt has been calculated. The grade was assigned based on a 75m x 75m cell declustered mean grade of the composite data.

Table 17.25 the resource is quoted at a 0.0 g/t Au cut-off.

Table 17.25: Golden Point Deposit - Grade Tonnage Report

	Inferred		
	Tonnes (Mt)	Grade (Au g/t)	Contained Gold (koz)
Golden Point	1.48	2.55	120

17.6 Round Hill Resource Estimate

The Round Hill resource estimate extends across the former Southern Pit, Round Hill and Golden Point areas. Southern Pit, Round Hill and Golden Point resources had previously been removed from the resource inventory in 2003, 1998 and 2003 respectively.

This section documents interim resource estimates for the Round Hill resource area. A revised geological interpretation and resource estimate is expected to be completed by late March 2010.

Mining from Round Hill pit commenced in 1990, was mined to designed completion in July 1998 and subsequently removed from the Macraes resource inventory. The pit was then back-filled, partly to provide a short haul waste dump for the adjacent Golden Point mining and partly in response to movement of the west wall of the pit.

Similarly, the resource area immediately to the south of Round Hill, known as Southern Pit, was mined to what in August 2001 was considered the economic open pitable limit. The area was subsequently removed from the resource inventory and part of it developed as a tailings impoundment.

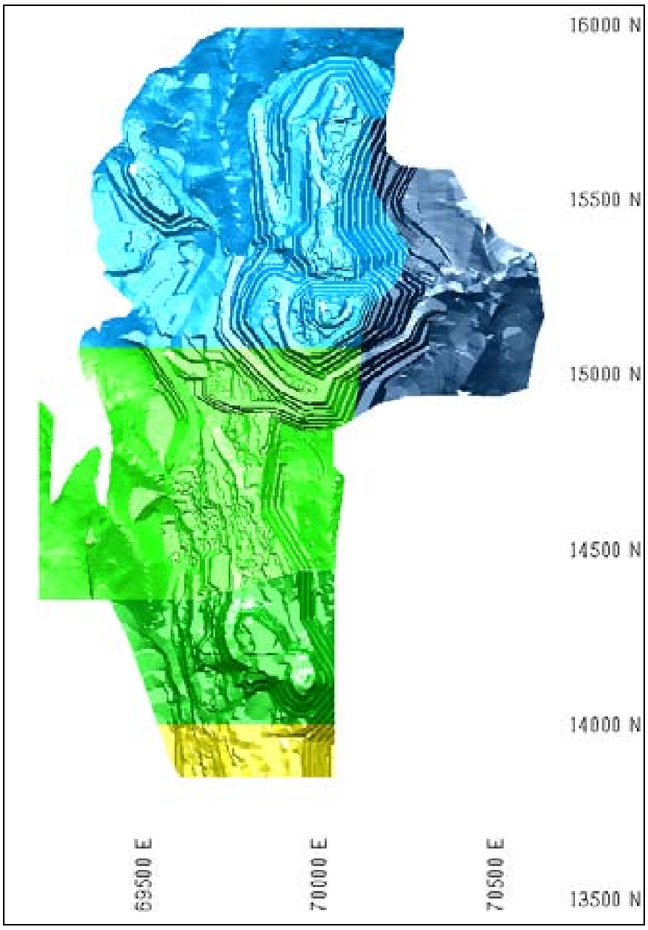
Golden Point was mined to completion in June 2002 and subsequently removed from the resource inventory. Golden Point has been partially back-filled.

Pit optimizations based on current gold prices have demonstrated the potential for a significant open pit cut back which extends from Golden Point southward, across Round Hill and Southern Pit, and into the northern part of Innes Mills. These optimizations do not consider additional revenues that potentially could be realized by processing scheelite, associated with the gold mineralization.

Because the Round Hill resource was retired in 1998, and the corporate focus subsequently shifted to other projects, a considerable effort has gone into restoring the project, both in terms of database integrity and also geological knowledge.

In November 2009, a NI 43-101 technical report (the 2009 Report) for the Macraes Project was released. In it, a preliminary resource for the Round Hill resource was included. The resource area used in the 2009 Report is shown in Figure 17.10 as that area including the light green, medium blue and dark blue regions (i.e. approximately north of 14,350mN).

Figure 17.10: Boundaries of Round Hill Resource Areas



Since the 2009 Report, further geological review has allowed the area to the south to be added to the Oceana resource inventory. This is the area shown in darker green. The yellow area further to the south is known as Innes Mills and is discussed in section 17.8.

This sequencing of the Round Hill geological review has led to a composite approach to the resource estimates. This is temporary and with time the areas will be consolidated. For the purposes of this report however, the Round Hill resource has been modeled as two resource estimates; northern Round Hill (blue regions in Figure 17.10) and southern Round Hill (green regions in Figure 17.10). The northern Round Hill estimate was completed in November 2009 while the southern Round Hill estimate was completed in December 2009.

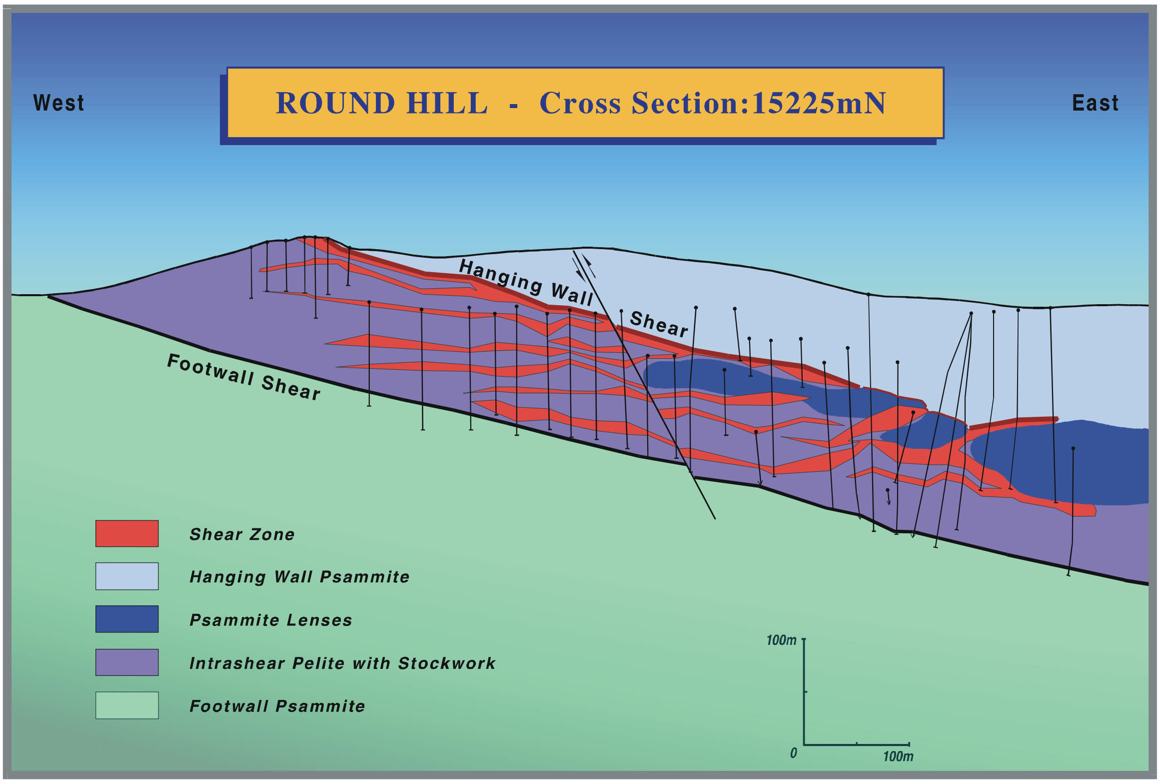
The southern Round Hill resource has been further subdivided for reporting purposes; the northern portion in light green has been reported at a 0.4 g/t Au cut-off and the southern portion in dark green (see Figure 17.10) has been reported at a 0.5 g/t Au cut-off. The 0.4 g/t Au cut-off reflects proximity to the processing plant. This is discussed in more detail in section 17.6.13.

17.6.1 Geological Interpretation

The 2009 Round Hill resource estimates, which include the adjacent Golden Point and Southern Pit areas, are based on geological interpretations for Golden Point completed in 1998, Round Hill completed in 1997, and Southern Pit completed in 2000[3]. It is believed that these interpretations provide a reasonable basis upon which to proceed; bench reconciliations of the resource estimate compare reasonably against grade control estimates (see Figure 17.18). A reinterpretation of the geology is underway, having been completed for southern Round Hill. For northern Round Hill, the reinterpretation is expected to be completed during the first quarter, 2010.

From Golden Point, through Round Hill to Southern Pit, the HMSZ is approximately 100m thick and is bounded by a well developed hangingwall shear of up to 10m thick and the Footwall shear which is up to 5m thick (Figure 17.11). The shears typically manifest as dark-grey, fine grained, micaceous, graphitic schists with local development of cataclasite, particularly towards the top of the hanging wall shear. Shearing intensity is highly variable, as are associated silicification, quartz veining and brecciation.

Figure 17.11: Round Hill - Cross Section: 15225mN



A network of anastomosing, shallowly east-dipping concordant shears lie between the Hangingwall and Footwall shears. Locally these structures can be up to 15m thick but show variable thickness and erratic geometries.

Zones of sheeted veins, in which individual veins are discontinuous and steeply dipping, are a common feature at Round Hill. These veins commonly strike north-east and dip predominantly to the north. Vein thickness varies from one millimetre to one metre. Large veins can be traced for up to 100m along strike and 10m down dip, but are typically less than 10m long. Vein textures range from massive milky quartz to finely laminated milky quartz and dark-grey quartz, with rare scheelite mineralization.

[3] The wireframe for the conformable shear below the Hangingwall for the 2000 Southern Pit interpretation has been removed

The geometry of the mineralized shears is strongly controlled by large lenticular bodies of weakly to un-mineralised psammite, the shears being most thickly developed along the lower margins of the psammite lenses. Psammite lenses typically have dimensions in the order of >250m north-south, 100m east-west and 30-40m vertically. Thick zones of strongly developed mineralization are also developed around the western terminations of the lenses, in what are interpreted as zones of pressure shadow. Low angle extensional faulting has contributed to the present disposition of ore but these structures elude interpretation at the resource drilling scale.

Mapping and dating studies completed in the Round Hill Pit during production have identified a number of important structural features that elucidate the mineralization history of the HMSZ. The north dipping stockwork veins appeared to have been the first mineralised structures developed. The stockwork veins are cross cut by later concordant shears and ramp veins and are approximately 145 million years (Ma) old. A similar age for quartz veins in the Hanging wall shear indicated that through-going movement had begun to occur at this time, although a proto-Hanging wall may have developed earlier than this. As displacement continued and became more localised, flat or concordant shears formed within the Intrashear package. The concordant and ramp veins have been dated at 135 Ma.

Over most of its extent, the Hangingwall shear can be delineated and wireframed via a combination of logged geology and gold proxy (approximately 0.4 g/t Au cut-off). Below the Hangingwall however, the shear contacts tend to be more difficult to delineate, particularly with reverse circulation chip logging, so in many cases the wireframe interpretations are grade based. The intensity and continuity of these shears below the Hangingwall decreases to the south of 15,000mN (southern Round Hill) to the point where, south of 14,800mN, the wireframe interpretation, based on resource drilling becomes impractical.

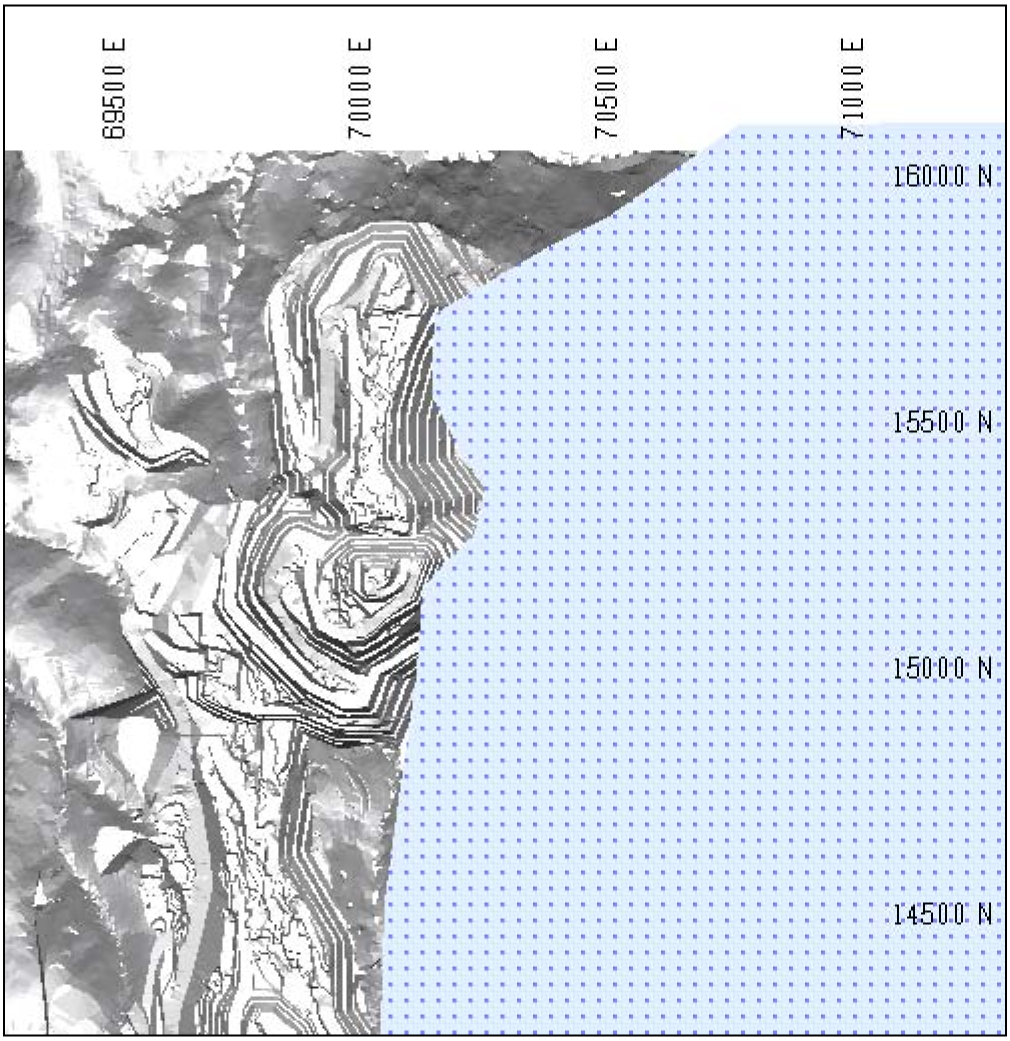
17.6.2 Wet Sampling Bias

A significant number of RC drill holes, drilled in the 1980's and early 1990's within Round Hill, were drilled under wet conditions. Mining experience in other pits suggests that sampling grade biases are likely to have resulted.

Historically, wet sampling bias has been encountered in Frasers, Innes Mills and Golden Bar pits. In each case the bias was mitigated by drilling a number of diamond drill holes to twin known wet sampled RC drill holes. In these cases diamond twins were only drilled for a subset of all wet sampled RC drill holes. So where there was no diamond twin to replace adjacent RC drill holes, a set of globally determined, grade dependent factors was applied. Having now mined some of the areas affected by wet RC sampling, it appears that the modelling approach used to mitigate the biases has been reasonably successful.

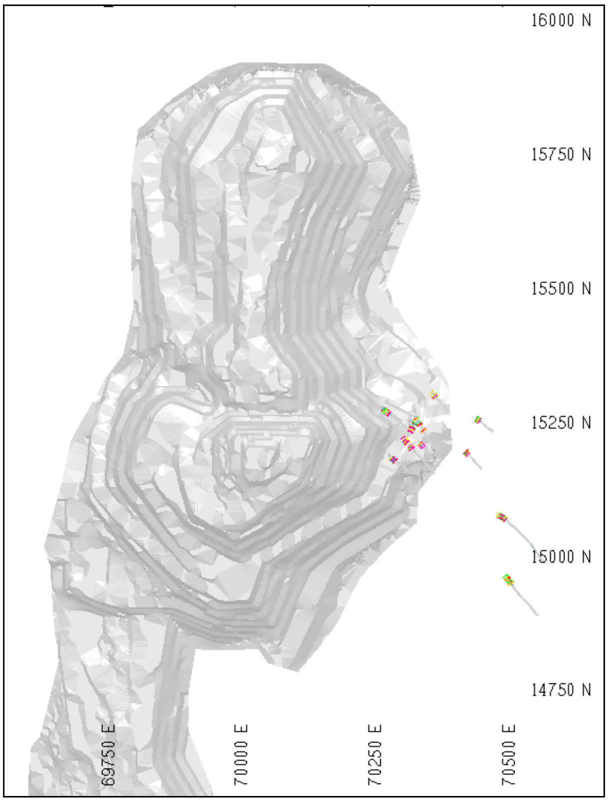
At Round Hill, the majority of wet sampled reverse circulation drill holes occur to the east of 70,100mE. The geological interpretation of this area will be reviewed, particularly in regard to the how bias factoring impacts the gold proxy criterion for the geological interpretation of shear contacts. Until this work is completed (by late March, 2010), the portion of Round Hill resource potentially affected by wet sampling biases (stipled blue in Figure 17.12, which also shows the June 30, 2009 as-mined surface) will be classified as inferred.

Figure 17.12: Plan View of Area Potentially Affected by Sampling Bias



A ten hole dedicated diamond twin drill holes has recently been completed at Round Hill (see Figure 17.13).

Figure 17.13: Location of Round Hill Diamond Twins Relative to Mined-Out Pits

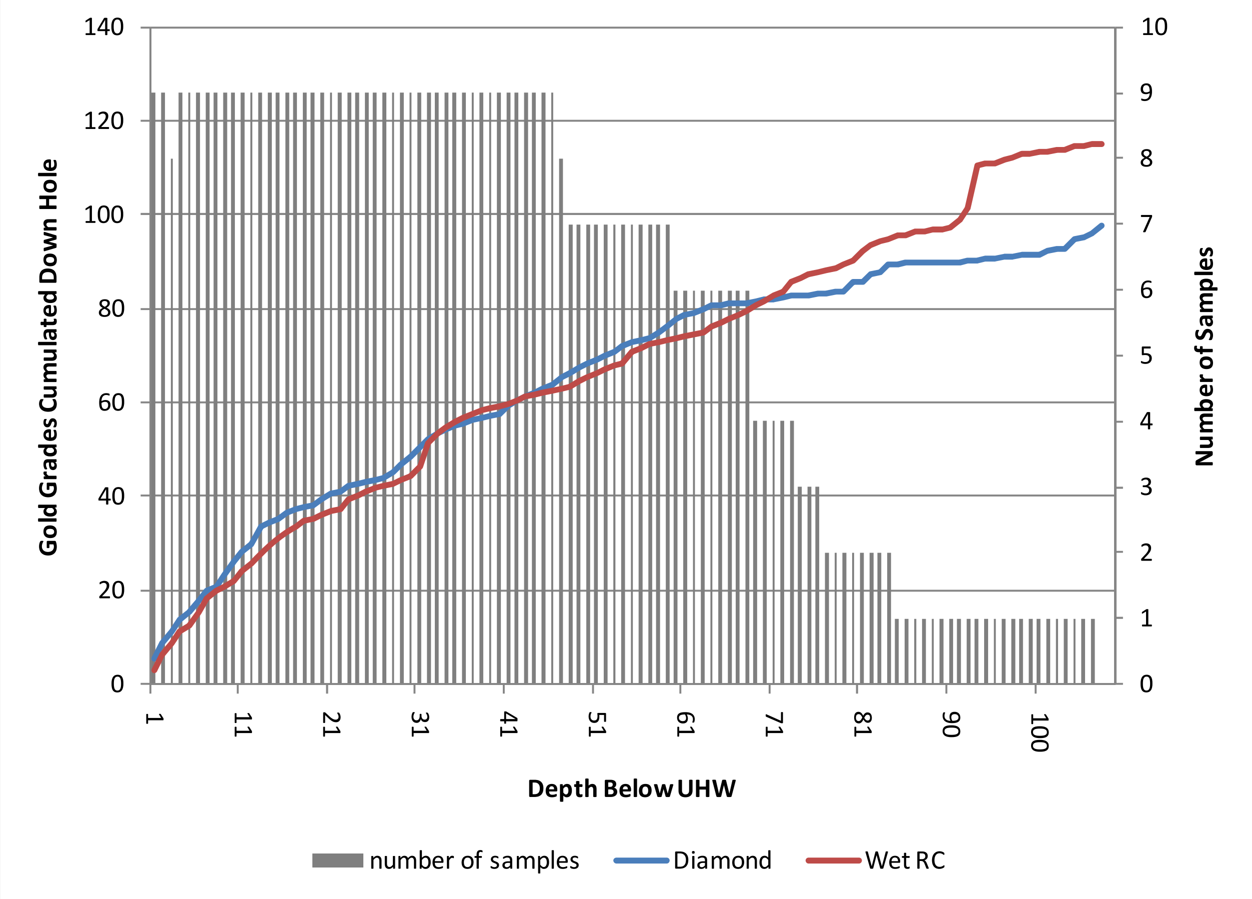


As a first pass, diamond cored sample grades were compared against adjacent wet sampled reverse circulation chip sample grades, limiting the comparison to pairs with less than 20 metres separation distance. This restricted the population for comparison to 631 sample pairs.

Figure 17.14 compares the gold grades for diamond core versus reverse circulation samples. In order to reduce the effects of local grade variation, the samples were grouped and averaged as follows; the upper surface of the Hangingwall shear (UHW) was chosen as the reference surface (little grade occurs above this) from which to relate sample pairs. All samples, one metre in length, were grouped by their depth below the UHW and the grades averaged to allow a broad comparison of the diamond and reverse circulation drill hole sample grades by depth.

The comparison provides little evidence for significant global grade bias, although does not preclude the possibility of biases occurring locally. Comparisons for pairs more than 70 metres below the UHW are based on too few pairs to be meaningful.

Figure 17.14: Comparison of Down-Hole Accumulated Gold for Diamond versus RC Sample Grades



The wet sample bias study was based on eleven diamond twins (ten drilled for the 2009 program plus one diamond twin that existed prior to the drilling program). Given the relatively low number of drill holes, and that there is considerable geological and grade variation within the mineralised shears (and therefore between drill holes), a cautious approach to the quantification of potential biases was taken:

- twins with more than 12.5 metres separation were excluded to minimize the effects of local grade variability; and
- where mineralisation within the diamond twin was stronger than in the respective reverse circulation drill hole, the diamond / reverse circulation drill holes were excluded.

These selection criteria reduced the number of diamond / reverse circulation twins to three, for a total of 196 sample pairs.

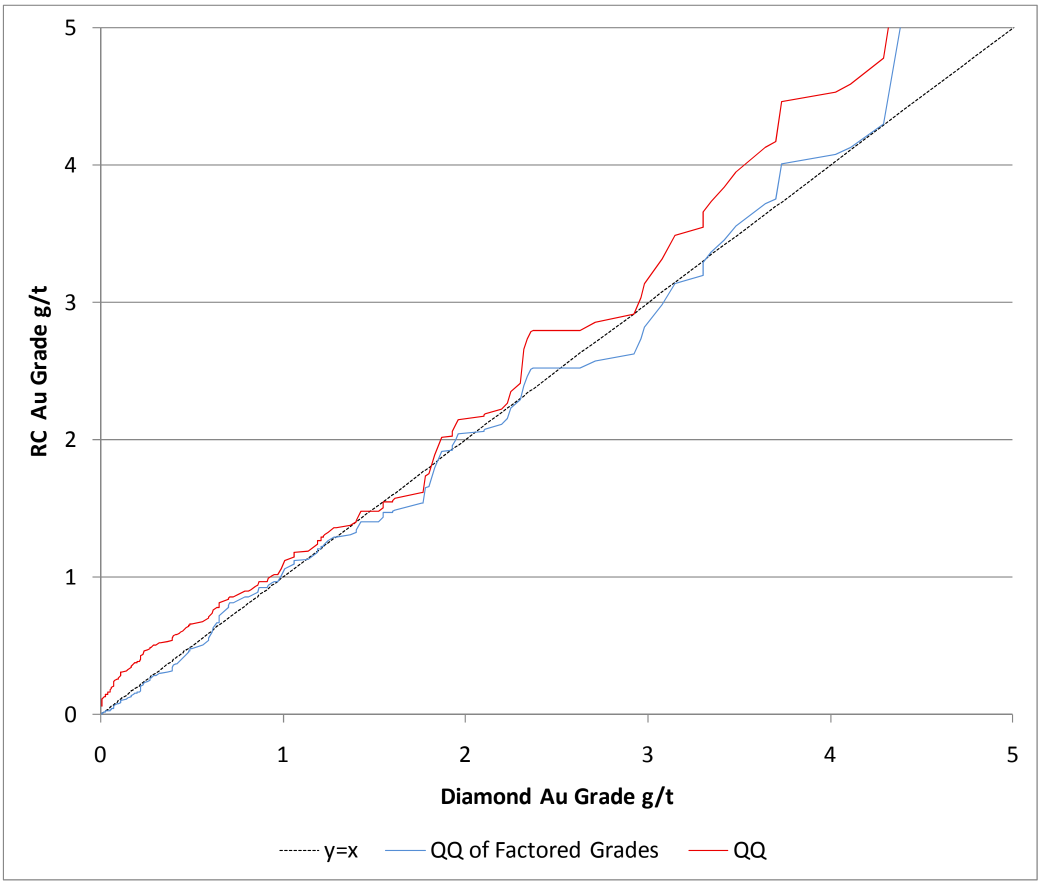
Figure 17.15 presents a quantile / quantile (QQ) plot of wet sampled reverse circulation drill hole gold grades versus diamond core sample grades.

The grade dependent factoring used for wet RC samples only was:

For gold grades less than 0.9 g/t Au,	factored grade = 1.1 x gold grade x gold grade (g/t Au)
For gold grades 0.9 to 2.5 g/t Au and below,	factored grade = 0.95 x gold grade (g/t Au)
For gold grades greater than 2.5 g/t Au,	factored grade = 0.90 x gold grade (g/t Au)

The QQ plot shows that the most severe bias occurs below 0.9 g/t Au.

Figure 17.15: QQ Plot of Round Hill Wet Reverse Circulation versus Diamond Sample Pairs

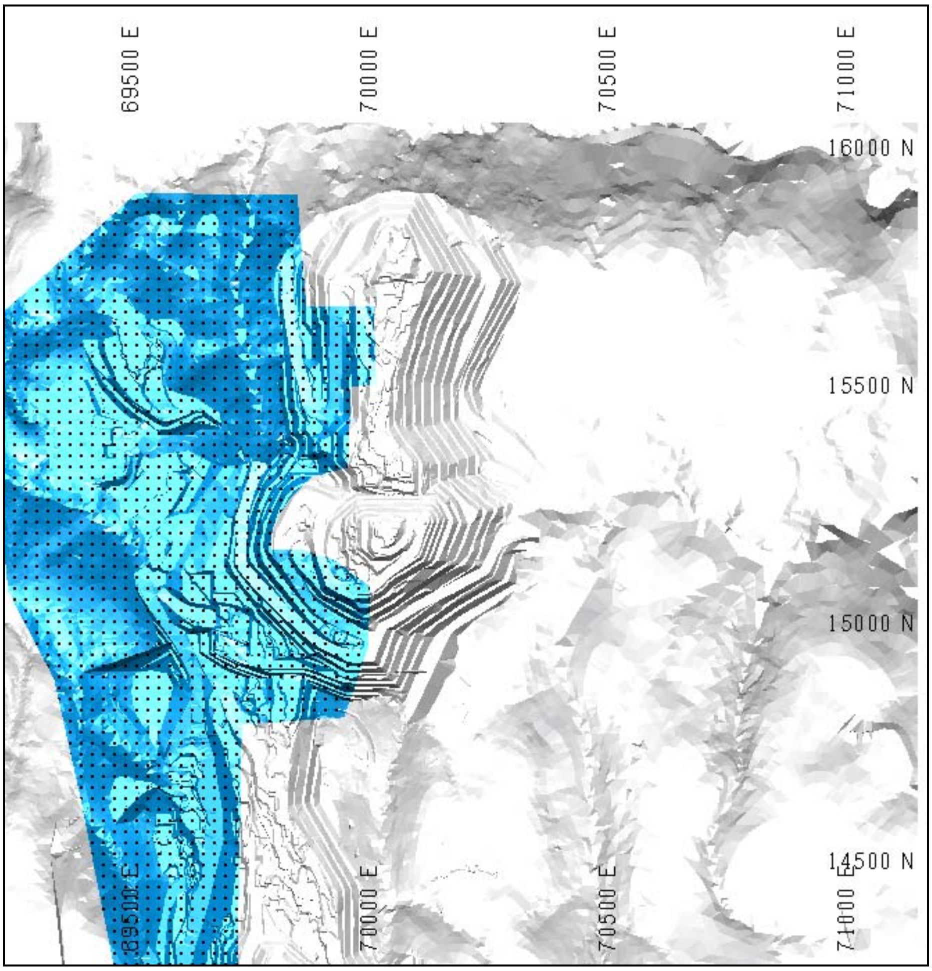


The factored sample grades have been used for all statistical analysis and modelling presented in the remainder of this chapter.

17.6.3 Pre-Oceana Drill Holes

415 drill holes were drilled prior to Oceana (then Macraes Mining Ltd), acquiring the project. Of these, 29 are conventional percussion drill holes, 292 are reverse circulation drill holes and the remainder are diamond core. Figure 17.16 highlights the area with a proportion of pre-Oceana drill holes in blue. The June 30, 2009 as mined surface is also shown; much of the volume drilled at this time has since been mined out. At the time of writing the 2009 Report, Oceana had not located documentation describing this drilling.

Figure 17.16: Round Hill – Plan View of Area Containing Pre-Oceana Drilling



Oceana have now located hard copy documentation of sampling methodology, assay and repeat assay data. This is not yet in electronic form, but visual inspection has increased Ocean's confidence in this assay data. Furthermore the northern Round Hill resource estimate incorporated these drill holes for estimation, and in terms of the mine to model reconciliation (see Figure 17.18) there is little reason to suspect significant underlying biasing of grade or contamination.

17.6.4 Reporting Area for Northern Round Hill

As discussed earlier in this chapter, as an interim measure, the Round Hill resource has been subdivided into a northern and southern area for modelling and reporting purposes (see Figure 17.10).

17.6.5 Geological – Geostatistical Interpretation Northern Round Hill

Statistical analysis has been based on 1m wet sample bias-factored composites coded with the mineralization/geological interpretation. The composite data has been generated as 1m run length composites.

Summary statistics for the gold data, grouped by interpreted domain, is shown in Table 17.26 No high grade cutting has been applied during estimation. The highest class means used in the MIK estimation however, were replaced by medians.

Table 17.26: Northern Round Hill - Summary of Gold (g/t) 1m Composite Statistics

Domain	Number	Mean	Minimum	Maximum	Median	CV
10	4,951	1.99	0.01	62.5	1.29	1.31
20	399	1.61	0.01	23.8	1.01	1.29
30	11,808	1.77	0.01	93.7	1.05	1.55
50	268	2.03	0.01	38.7	1.14	1.65
60	583	2.22	001	16.0	1.55	1.00
70	93,833	0.31	0.01	55.8	0.09	2.92

Indicator varograms were generated for the combined Domains 10 through to 60. Domain 70 variograms were generated separately. The indicator variogram parameters are tabulated in Table 17.27 and Table 17.29 whilst the rotational parameters can be found in Table 17.28.

Note for all domains, the top class means were replaced by their medians.

Table 17.27: Northern Round Hill - Indicator Variogram Parameters Domains 10, 20, 30, 50 and 60

Percentile	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)
10^{th}	0.35	0.56	0.09	25x26x4.5	36x33x34
20^{th}	0.35	0.48	0.17	25x26x3.0	36x33x14
30^{th}	0.35	0.41	0.24	25x26x3.0	36x33x9
40^{th}	0.32	0.41	0.27	25x26x2.5	36x33x9
50^{th}	0.32	0.41	0.27	27x28x2.5	36x33x9
60^{th}	0.32	0.41	0.27	25x26x2.5	36x33x9
70^{th}	0.27	0.49	0.24	27x27x2.5	36x33x9
75^{th}	0.23	0.54	0.23	27x26x2.5	36x33x9
80^{th}	0.21	0.59	0.20	28x26x2.5	36x33x8.5
85^{th}	0.21	0.59	0.20	28x26x2.0	36x33x8.5
90^{th}	0.21	0.59	0.20	28x27x2.0	36x33x7.5
95^{th}	0.21	0.59	0.20	28x24x2.0	32x30x5.5
97.5^{th}	0.21	0.59	0.20	27x24x2.0	30x27x4.5
99^{th}	0.21	0.59	0.20	27x24x2.0	20x27x2.5

Table 17.28: Northern Round Hill - Variogram Rotation Parameters Domains 10, 20, 30, 50, 60 and 70

Domain	Rotation Parameters		
	x	y	z
10	0	12	0
20	0	-7	0
30	0	12	0
50	0	8	35
60	0	-14	-20
70	0	-12	0

Table 17.29: Northern Round Hill - Indicator Variogram Parameters Domain 70

Percentile	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)	Rotation Parameters		
						x	y	z
10th	0.36	0.35	0.29	20x30x10.2	135x330x60	0	-12	0
20th	0.31	0.38	0.32	23x36x10.2	160x345x64	0	-12	0
30th	0.28	0.40	0.32	18x28x7.2	140x285x54	0	-12	0
40th	0.24	0.41	0.35	17x21x5.2	105x170x40	0	-12	0
50th	0.24	0.50	0.26	17x24x5.7	175x215x58	0	-12	0
60th	0.26	0.48	0.26	17x24x4.7	135x150x54	0	-12	0
70th	0.26	0.50	0.24	17x24x4.5	145x135x50	0	-12	0
75th	0.26	0.51	0.23	17x24x4.0	135x120x62	0	-12	0
80th	0.30	0.47	0.23	17x24x4.0	110x105x44	0	-12	0
85th	0.30	0.53	0.17	17x24x4.5	145x125x68	0	-12	0
90th	0.30	0.56	0.14	17x24x3.5	120x105x135	0	-12	0
95th	0.30	0.60	0.10	17x24x3.0	95x91x165	0	-12	0
97.5th	0.30	0.60	0.10	17x24x2.5	65x51x59	0	-12	0
99th	0.30	0.60	0.10	16x20x2.0	43x30x10	0	-12	0

17.6.6 Northern Round Hill Block Model

A regular block model was constructed for the purposes of grade estimation, as summarised in Table 17.15. A 25m x 25m x 2.5m parent cell size was used.

Table 17.30: Northern Round Hill - Resource Estimate Limits and Block Model Dimensions

	Northern Round Hill, November 2009 Estimate	
	Limits	Block Size
Easting	69,200 – 71,100	25 m
Northing	14,300 – 16,000	25 m
RL	230 - 575	2.5 m

Bulk densites of 2.5 t/m^3 and 2.6 t/m^3 have been applied respectively to all oxide/sulphide blocks for resource reporting.

Grade estimation for Round Hill mineralization used Multiple Indicator Kriging with block support adjustment as implemented in the *GS3* Modelling Software. A more detailed explanation of *GS3* can be found in section in 17.9.2. The estimation sample search parameters applied in estimation are summarised in Table 17.16 Table 17.31.

The search rotations are identical to those used for variography.

As displayed below in Table 17.31, the minimum number of composites used for kriging into blocks classified as indicated was 16 and the maximum number was 48. In some cases for inferred blocks, the minimum number reduced to 8 samples.

Table 17.31: Domain Control used in Estimation Northern Round Hill - Sample Search Parameters

Domain	Minimum Number of Samples	Maximum Number of Samples	Minimum Number of Octants	Primary Sample Search Distances X,Y,Z (m)	Secondary Sample Search Distances X,Y,Z (m)
10	16	48	6	40,40,6	80,80,12
20	16	48	4	25,25,4	50,50,8
30	16	48	6	40,40,6	80,80,12
50	16	48	4	25,25,4	50,50,8
60	16	48	4	25,25,4	50,50,8
70	16	48	4	25,25,4	50,50,8

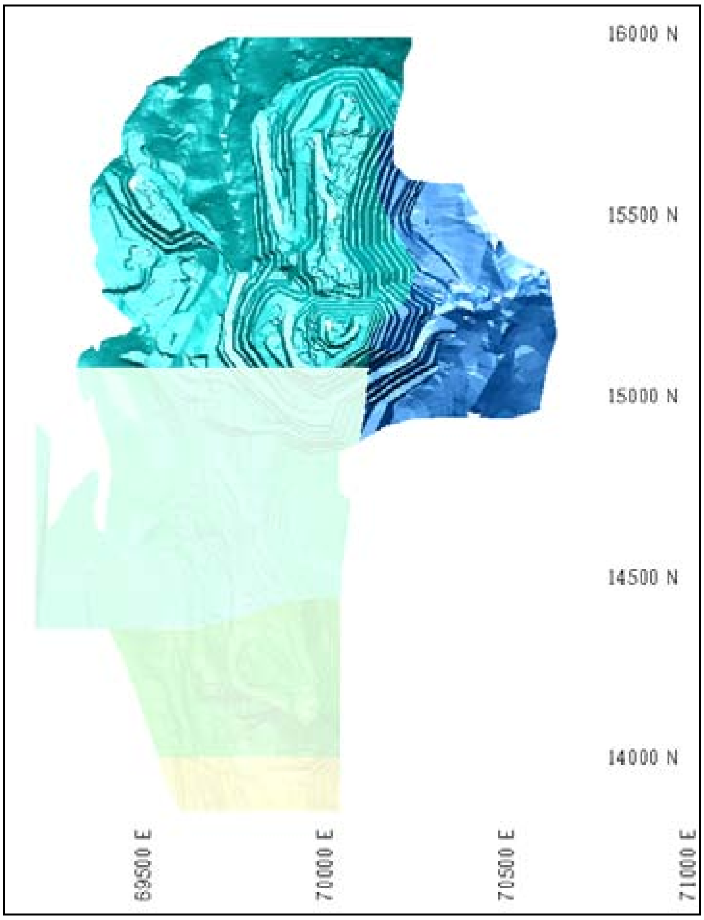
17.6.7 Resource Classification and Reporting for Northern Round Hill

The right hand most column in Table 17.31 shows the search limits used to generate resource for each domain. In all cases, model blocks beyond these limits, or blocks that captured less than 8 samples, were not estimated.

All resource within 15m of the Footwall Fault (see Figure 17.11), the structure that defines the geological base of mineralization, was not included in the resource inventory.

The quoted resource is limited to no further than 60m east of the toe of a preliminary optimized pit shell. The shell was based on a gold price of NZ$1,404 per ounce and mining costs prepared for the LOMP09. Figure 17.17 shows the actual limits in plan view, with mined topography.

Figure 17.17: Northern Round Hill – Plan View of Resource Limits



The darker blue area represents the Round Hill resource that is potentially affected by wet sampling and has been classified as Inferred due to the potential impact on the geological interpretation, particularly where a gold proxy has been used to define the upper or lower contacts for mineralised shears. The classification of this area will be reviewed once the geological re-interpretation is completed.

The medium blue area is the remaining resource area not affected by wet samples and is classified variously as Measured, Indicated and Inferred according to the following criteria;

- Measured if the block meets the primary sample search distance, minimum octant, minimum sample criteria listed in Table 17.31, its volume impinges more than 2% shear wireframe volume and the recoverable proportions at 0.4 g/t cog are greater than 80%.
- Indicated if the block meets the secondary sample search distance, minimum octant, the minimum sample criteria listed in Table 17.31, its volume impinges more than 2% shear wireframe volume and the recoverable proportions at 0.4 g/t cog are greater than 20%.
- Indicated also if all the Measured criteria are met except that the recoverable proportions at 0.4 g/t cog are less than 80%.
- Inferred if the block fails the Measured and Indicated criteria but meets the secondary sample search distance, minimum octant and half the minimum sample criteria listed in Table 17.31.

17.6.8 Validation and Reconciliation

The resource estimate has been validated both visually and statistically.

The resource estimate was plotted on screen in section and plan and being viewed in 3D via *MINESIGHT*. Aspects checked included geological coding, classification coding, and grade interpolation. The block grades were visually compared to surrounding composite grades and considered to reasonably reflect the input data.

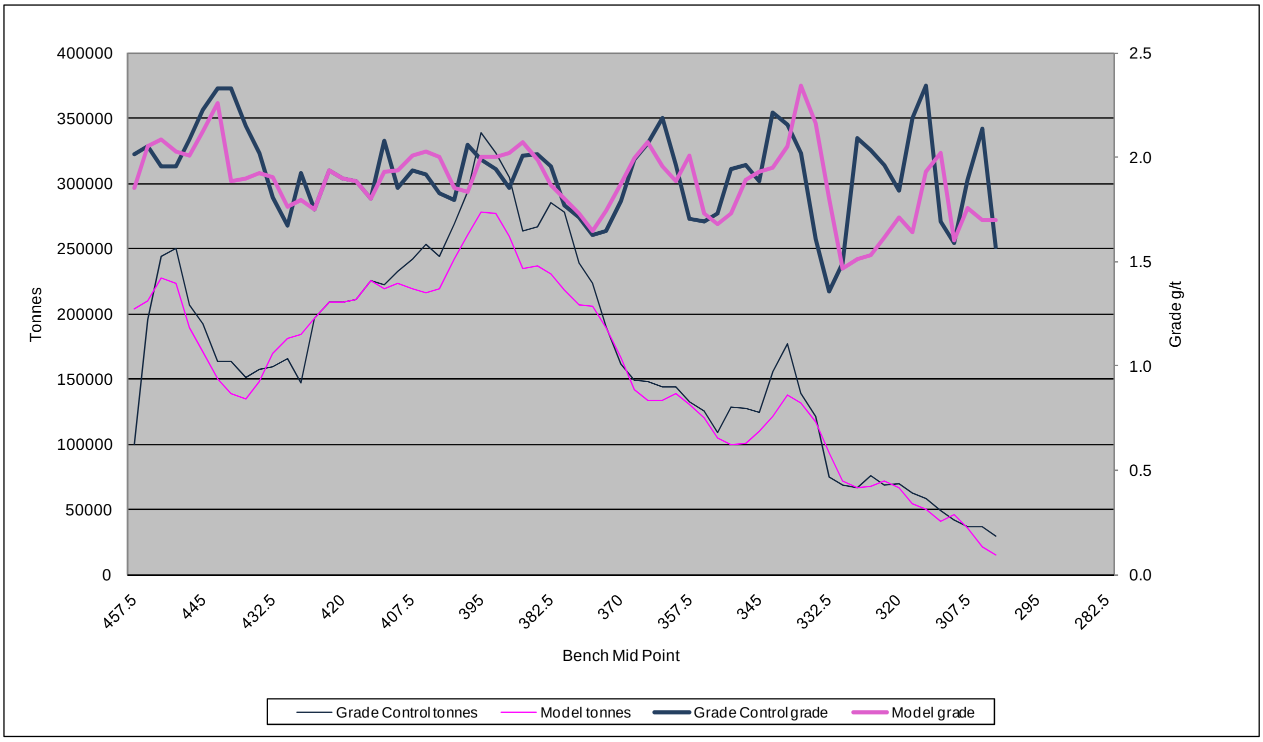
The mean composite grade versus average model grade, calculated for each domain (Table 17.32), shows reasonable reproduction of the input data.

Table 17.32: Round Hill - Comparison of Input Composites versus Block Model Grades (g/t Au)

Domain	Mt	Sample Mean Au (g/t)	Model Mean Au (g/t)
10	12.8	1.99	1.67
20	0.70	1.61	1.44
30	29.21	1.77	1.53
50	0.57	2.03	1.84
60	1.50	2.23	0.89
70	400.26	0.31	0.21

Figure 17.18 below compares the Round Hill resource estimate against a grade control, ordinary kriged block model on a bench by bench basis. The reconciliation represents approximately 10.6 Mt of mined ore. Due to incomplete archiving of data collected during the early 1990's, the earliest mined ore, in the upper reaches of the orebody (between the 515mRL and 587.5mRL) was not able to be reconciled. The reconciliation shows that the resource estimate provides reasonable predictions in terms of tonnes and grade. Overall, the estimate (Measured and Indicated) under-estimates tonnes by 8% and over-estimates grade by 1%, resulting in under-estimating contained gold by 9%. Note that the volume reconciled was not significantly affected by wet sampling.

Figure 17.18: Round Hill Reconciliation at 0.8 g/t Cut-off for Measured, Indicated and Inferred



Following the diamond twin drilling program currently in progress and the geological review later in March 2010, it is expected that an improved resource estimation will be achieved.

17.6.9 Reporting Area for Southern Round Hill

As discussed earlier in this chapter, as an interim measure, the Round Hill resource has been subdivided in a northern and southern areas for modelling and reporting purposes (see Figure 17.10).

17.6.10 Geological – Geostatistical Interpretation Southern Round Hill

Statistical analysis has been based on 1m wet sample bias-factored composites coded with the mineralization/geological interpretation. The composite data has been generated as 1m run length composites.

Summary statistics for the gold data, grouped by interpreted domain, is shown in Table 17.33. No high grade cutting has been applied during estimation. The top class means for all domains used in the MIK estimation however, were replaced by medians.

Table 17.33: Southern Round Hill - Summary of Gold (g/t) 1m Composite Statistics

Domain	Number	Mean	Minimum	Maximum	Median	CV
10	4,958	1.99	0.01	62.5	1.29	1.31
20	399	1.61	0.01	23.8	1.01	1.29
30	10,856	1.83	0.01	93.8	1.10	1.52
50	274	1.81	0.01	32.9	1.04	1.62
60	570	2.15	001	16.0	1.52	0.96
70	94,540	0.31	0.01	55.8	0.08	2.98

Indicator varograms were generated for the combined Domains 10 through to 60. Domain 70 variograms were generated separately. Indicator variogram parameters are tabulated in Table 17.34 and Table 17.36 whilst the rotational parameters can be found in Table 17.35 and Table 17.36.

Note for all domains, the top class means were replaced by their medians.

Table 17.34: Southern Round Hill - Indicator Variogram Parameters Domains 10, 20, 30, 50 and 60

Percentile	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)
10^{th}	0.35	0.56	0.09	25x26x4.5	36x33x34
20^{th}	0.35	0.48	0.17	25x26x3.0	36x33x14
30^{th}	0.35	0.41	0.24	25x26x3.0	36x33x9
40^{th}	0.32	0.41	0.27	25x26x2.5	36x33x9
50^{th}	0.32	0.41	0.27	27x28x2.5	36x33x9
60^{th}	0.32	0.41	0.27	25x26x2.5	36x33x9
70^{th}	0.27	0.49	0.24	27x27x2.5	36x33x9
75^{th}	0.23	0.54	0.23	27x26x2.5	36x33x9
80^{th}	0.21	0.59	0.20	28x26x2.5	36x33x8.5
85^{th}	0.21	0.59	0.20	28x26x2.0	36x33x8.5
90^{th}	0.21	0.59	0.20	28x27x2.0	36x33x7.5
95^{th}	0.21	0.59	0.20	28x24x2.0	32x30x5.5
97.5^{th}	0.21	0.59	0.20	27x24x2.0	30x27x4.5
99^{th}	0.21	0.59	0.20	27x24x2.0	20x27x2.5

Table 17.35: Southern Round Hill - Variogram Rotation Parameters Domains 10, 20, 30, 50, 60 and 70

Domain	Rotation Parameters x	Rotation Parameters y	Rotation Parameters z
10	0	12	0
20	0	-7	0
30	0	12	0
50	0	8	35
60	0	-14	-20
70	0	-12	0

Table 17.36: Southern Round Hill - Indicator Variogram Parameters Domain 70

						Rotation Parameters		
Percentile	C0	C1(sph)	C2(sph)	R1(xyz)	R2(xyz)	x	y	z
10th	0.36	0.35	0.29	20x30x10.2	135x330x60	0	-12	0
20th	0.31	0.38	0.32	23x36x10.2	160x345x64	0	-12	0
30th	0.28	0.40	0.32	18x28x7.2	140x285x54	0	-12	0
40th	0.24	0.41	0.35	17x21x5.2	105x170x40	0	-12	0
50th	0.24	0.50	0.26	17x24x5.7	175x215x58	0	-12	0
60th	0.26	0.48	0.26	17x24x4.7	135x150x54	0	-12	0
70th	0.26	0.50	0.24	17x24x4.5	145x135x50	0	-12	0
75th	0.26	0.51	0.23	17x24x4.0	135x120x62	0	-12	0
80th	0.30	0.47	0.23	17x24x4.0	110x105x44	0	-12	0
85th	0.30	0.53	0.17	17x24x4.5	145x125x68	0	-12	0
90th	0.30	0.56	0.14	17x24x3.5	120x105x135	0	-12	0
95th	0.30	0.60	0.10	17x24x3.0	95x91x165	0	-12	0
97.5th	0.30	0.60	0.10	17x24x2.5	65x51x59	0	-12	0
99th	0.30	0.60	0.10	16x20x2.0	43x30x10	0	-12	0

17.6.11 Southern Round Hill Block Model

A regular block model was constructed for the purposes of grade estimation, as summarised in Table 17.15. A 25m x 25m x 2.5m parent cell size was used.

Table 17.37: Southern Round Hill - Resource Estimate Limits and Block Model Dimensions

	RH09	
	Limits	Block Size
Easting	69,200 – 71,100	25 m
Northing	14,300 – 16,000	25 m
RL	230 - 575	2.5 m

Bulk densites of 2.5 t/m^3 and 2.6 t/m^3 have been applied respectively to all oxide/sulphide blocks for resource reporting.

Grade estimation for Round Hill mineralization used Multiple Indicator Kriging with block support adjustment as implemented in the *GS3* Modelling Software. A more detailed explanation of *GS3* can be found in section in 17.9.2. The estimation sample search parameters applied in estimation are summarised in Table 17.38. The search rotations are identical to those used for variography.

17.6.12 Resource Classification and Reporting for Southern Round Hill

The right hand most column in Table 17.38 shows the search limits used to generate resource for each domain. In all cases, model blocks beyond these limits, or blocks that captured less than the minimum number of samples, were not estimated.

Table 17.38: Domain Control used in Estimation Southern Round Hill - Sample Search Parameters

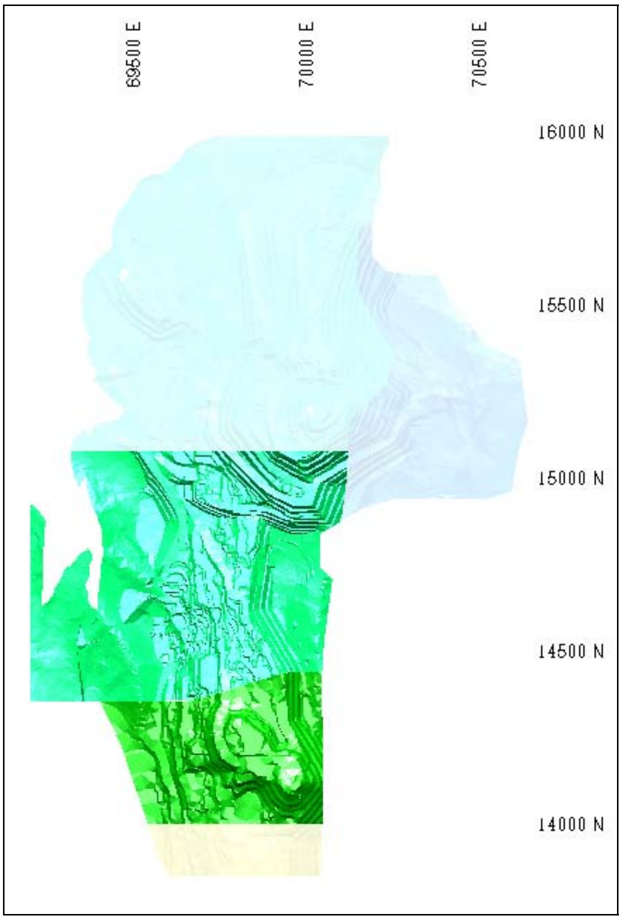
Domain	Minimum Number of Samples	Maximum Number of Samples	Minimum Number of Octants	Primary Sample Search Distances X,Y,Z (m)	Secondary Sample Search Distances X,Y,Z (m)
10	16	48	8	40,40,6	80,80,12
20	16	48	4	25,25,4	50,50,8
30	16	48	8	40,40,6	80,80,12
50	16	48	4	25,25,4	50,50,8
60	16	48	4	25,25,4	50,50,8
70	30	48	8	22,22,4	44,44,8

All resource within 15m of the Footwall Fault (see Figure 17.11), the structure that defines the geological base of mineralization, was not included in the resource inventory.

The resource, within the southern Round Hill area (shown in two tone green in Figure 17.19) is classified as Measured and Indicated according to the following criteria:

- Measured if the block meets the primary sample search distance, minimum octant, minimum sample criteria listed in Table 17.38 and the recoverable proportions at 0.4 g/t cog are greater than 80%.
- Indicated if the block meets the secondary sample search distance, minimum octant, minimum sample criteria listed in Table 17.38 and the recoverable proportions at 0.4 g/t cog are greater than 20%.
- There is no Inferred.

Figure 17.19: Southern Round Hill – Plan View of Resource Limits and Mined Topography



17.6.13 Resource Reporting for Round Hill

The total (i.e. northern and southern) Round Hill resource has been geographically subdivided as described below for reporting purposes, and is tabulated by class in Table 17.39. Due to proximity to the processing plant, the northern Round Hill resource and the northern portion of southern Round Hill shown in light green in Figure 17.19 (i.e. approximately north of 14,350mN) were reported to a 0.4 g/t Au cut-off. For the purpose of resource reporting, it is named Round Hill. This is in keeping with the November 2009 Report. See Table 17.39.

Table 17.39: Round Hill North - Open Pit Resource Estimate, 0.4 g/t Au Cut-off

Category	Total Resources as at December 31, 2009		
	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	4.06	0.98	0.13
Indicated	20.18	0.93	0.60
Inferred	18.38	1.22	0.72

The remaining portion of the Round Hill resource (approximately to the south of 14,350mN shown in darker green in Figure 17.19) was reported to a 0.5 g/t Au cut-off. For the purpose of resource reporting, it is named Southern Pit. See Table 17.40.

Table 17.40: Southern Pit - Open Pit Resource Estimate by Class, 0.5 g/t Au Cut-off

Category	Total Resources as at December 31, 2009		
	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	0.52	0.90	0.01
Indicated	2.90	0.85	0.08
Inferred	0	0	0

17.7 Southern Pit Open Pit Resource Estimate

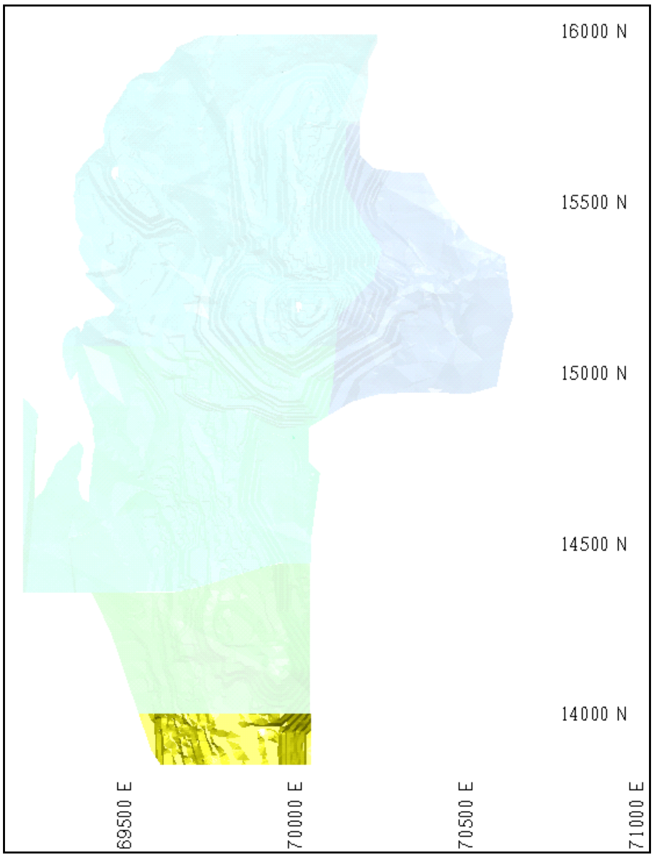
As discussed in section 17.6, the modelling of Southern Pit is included in an expanded Round Hill resource estimate which covers the former resource areas of Golden Point, Round Hill and Southern Pit. The resource reporting for Southern Pit is summarised in section 17.6.13.

17.8 Innes Mills Open Pit Resource Estimate

The Innes Mills resource was removed from the Oceana resource inventory in 2005 following the completion of open pit mining to what was then considered to be the economic limit. Back-filling commenced thereafter.

For the purposes of this report, a small increment (13,850mN to 14,000mN) at the northern end of the remaining Innes Mills resource has been resurrected to provide a contiguous resource extending from Golden Point through Round Hill, Southern Pit and into the northern end of Innes Mills. This increment of Innes Mills resource is shown in yellow in Figure 17.20 (the ghosted areas show the full extent of the Golden Point through Southern Pit contiguous resource, discussed previously in chapter 17.6).

Figure 17.20: Reporting Area for Innes Mills Resource



It is anticipated that this increment of resource will be remodelled during 2010 and integrated into a larger Round Hill / Southern Pit resource estimate. Given the small resource contained within this increment of Innes Mills, a full resource description was not felt to be warranted, although this will follow when this increment Innes Mills is integrated into a larger estimate.

The resource model, IM02a is a large block, recoverable resource estimate, built using GS3 software in 2002. Wet sample bias is not a significant issue in this restricted resource area, although areas previously mined to south were affected. The IM02a estimate was externally audited by Hellman and Schofield Pty Ltd.

Table 17.41 below shows the IM02a reconciliation. The tonnage reconciled is relatively small, and is therefore unlikely to reflect global model performance.

Table 17.41: Innes Mills, Stage 4 Resource Model Reconciliation at a 0.7 g/t Au Cut-Off

GC Survey Adjusted			Resource Model IM02a			Actual/Model		
Tonnes	g/t	ozs	Tonnes	g/t	ozs	Tonnes	g/t	ozs
768,883	1.45	35,844	922,729	1.40	41,533	0.83	1.04	0.86
52,568	1.76	2,975	48,245	1.28	1,985	1.09	1.38	1.50
821,451	1.47	38,819	970,974	1.39	43,518	0.85	1.05	0.89

The Innes Mills resource increment, called Northern Innes Mills is summarised below in Table 17.42, and represents a small proportion of the overall Macraes resource.

Table 17.42: Northern Innes Mills Resource at a 0.5 g/t Au Cut-off

Category	Innes Mills Resource as at December 31, 2009		
	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	0.04	1.94	0.002
Indicated	0.74	1.08	0.026
Inferred	0.23	0.71	0.005

17.9 Frasers Open Pit Resource Estimates

17.9.1 Geological – Geostatistical Interpretation

The gold mineralization at Frasers Open Pit comprises Hangingwall, concordant lode and “stockwork” mineralization. Hangingwall and stockwork mineralization account for the majority of the ore at Frasers, shown in red in Figure 17.21. These spatially distinct styles of mineralization provide the basis for defining the six geostatistical domains used in the resource modelling; Domains 10, 11 and 12 – Hangingwall, Domains 40,41 and 42 – stockwork mineralization. Domain 41 defines a more weakly mineralized Stockwork usually overlain by more strongly mineralized Hangingwall.

The Hangingwall generally dips at between 10 and 15 degrees to the east. A significant exception occurs at the bottom of the Frasers Stage 4 pit where the Hangingwall flattens to near horizontal over an area extending 300mE by 300mN (Domain 11). This flattening coincides with a substantial thickening and increase in grade of the mineralization.

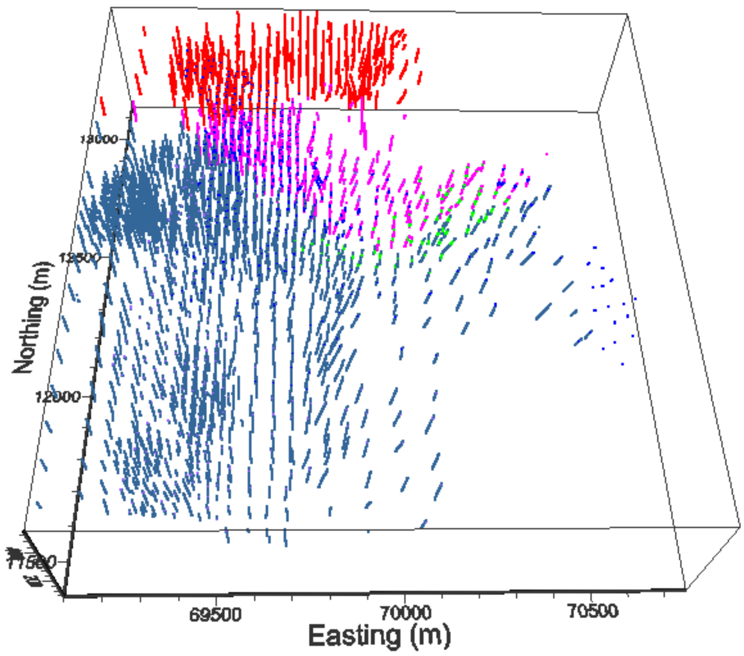
A large amount of erratic mineralization occurs between the base of the Hangingwall and the Footwall fault. This mineralization is interpreted as a Stockwork mineralization and generally appears in drilling as clusters of elevated grades. In reality, the term “Stockwork Mineralization” refers to mixtures of quartz veins and erratic lodes whose geometric complexity precludes effective wireframe interpretation at the scale of resource drill hole spacing.

There are a number of mineralized shears or concordant lodes running sub-parallel to the Hangingwall. These typically splay off the Hangingwall and dip at between 5 and 10 degrees to the east. In the current resource model, these zones have been modelled as Stockwork.

Figure 17.21 shows the overall spatial distribution of the three Stockwork domains, which have been separated on the basis of grade.

The Footwall Shear lies between 80 and 120m below the Hangingwall and is easily identified in drill holes as a 10m wide zone of shearing. No economic gold mineralization has been identified below this structure.

Figure 17.21: Drill Hole Coding of Frasers Stockwork Domains: 40 Blue, 41 Purple, 42 Red



17.9.2 Resource Estimation Process and Method

The FR05 resource model of the Frasers gold mineralization was initially constructed by the resource modelling team of Oceana under the leadership of Jonathan Moore. In September 2005, Neil Schofield of Hellman and Schofield (H&S) undertook a limited audit of the FR05 model. This audit found that the resource modelling parameters could be readily reproduced and that the differences between the Oceana model and the H&S check model were essentially due to differences in the block support correction applied in the Stockwork mineralization. H&S suggested the subdivision of the Stockwork mineralization mentioned above as well as the use of a particular type of reconciliation when comparing the estimates of the FR05 model to grade control outcomes.

The method used to generate estimates of the classified mineral resources in the Frasers mineralization was Indicator Kriging with block support adjustment as implemented in the *GS3* Modelling Software. This software is marketed and supported by Hellman and Schofield Pty Ltd. Details of the Indicator Kriging (IK) estimator can be found in Deutsch and Journel, 1998. The *GS3* software integrates a number of techniques including domain geometry modelling and block support adjustment into a single approach which allows the generation of "recoverable" resource estimates for a range of cut-off grades. The term "recoverable" refers to the resource that will be recovered as ore at the time of mining at a certain cut-off grade and at a particular scale of ore selection, for example a bench height of 2.5m and a 4m mining width.

The basic unit of a recoverable resource model is the panel (large rectangular block) with horizontal dimension approximately equal to the dominant drill hole spacing in the deposit. The vertical dimension of the panel is typically some multiple of the bench (or flitch) height. In the Frasers mineralization, a panel size of 25m east, 25m north and 2.5m vertical was adopted. Mining takes place on a 2.5m flitch with a minimum mining width of 5m.

The implementation of the IK method requires that the histogram of the sample grades for each mineralized domain (e.g. Hangingwall Domain 10) be defined by a set of indicator threshold grades. For each threshold, the conditional univariate statistics of the sample data and the indicator sample variograms and variogram models must be defined because these are used as parameters in the modelling method.

With the complete set of conditional statistics and indicator variogram models for each threshold specified, the indicator kriging method estimates the probability that the grades of samples within the panel will be greater than a particular threshold. This set of cumulative probabilities defines an estimate of the cumulative histogram of sample grades within the panel. Using block support adjustment, this cumulative histogram can be modified to reflect the cumulative histogram of the grades of mining blocks of specified dimensions.

17.9.3 Conditional Statistics of All Domains

Table 17.43 presents the conditional univariate statistics of the one metre composite data in all domains. In all domains, a top cut of 50 g/t was used to reduce the influence of very high grade samples occurring in the highest indicator class.

17.9.4 Sample Variograms

Figure 17.23 and Figure 17.24 present the directional sample variograms of the Hangingwall and Stockwork mineralization respectively. These variograms are based on closely spaced grade control data. In each case, five directions are plotted, each direction defined by an azimuth and plunge. Azimuth 0 (azm0) is due east. Azm90 is due north and Pln90 is vertical. The figures demonstrate well the much stronger continuity of gold grade in the Hangingwall mineralization.

17.9.5 Indicator Variogram Models of All Domains

Table 17.44 through to Table 17.49 present the variogram model parameters for indicator variogram models of all domains. In these tables, C0, C1 and C2 represent the nugget and successive incremental spatial variances and Ax, Ay etc represent the ranges of each nested structure of the model for the three Cartesian axes X, Y and Z. The 3D rotations use the trigonometric convention. For example, Y13 means a rotation of 13 degrees in the anticlockwise direction around the positive end of the Y axis (usually north).

All of the variogram model sets show small positive rotations of between 5 and 13 degrees around the Y axis. This has the effect of imparting a shallow easterly plunge to the X direction range, consistent with the easterly dip of the mineralized units shown in Table 17.43 above. All variogram models have been standardized to a sill of 1.0.

Table 17.43: Frasers Deposit - Conditional Univariate Statistics of 1m Samples in All Domains

Domain	10		11		12		40		41		42	
Class	T 'hold	Mean	T' hold	Mean	T' hold	Mean	T' hold	Mean	T' hold	Mean	T 'hold	Mean
10th	0.24	0.13	0.19	0.08	0.40	0.25	0.01	0.01	0.01	0.01	0.01	0.01
20th	0.37	0.31	0.33	0.26	0.64	0.53	0.02	0.02	0.01	0.01	0.02	0.01
30th	0.50	0.43	0.52	0.42	0.92	0.77	0.04	0.03	0.01	0.01	0.03	0.02
40th	0.64	0.57	0.70	0.61	1.18	1.05	0.06	0.05	0.02	0.02	0.04	0.03
50th	0.79	0.71	0.92	0.81	1.52	1.34	0.09	0.07	0.03	0.02	0.06	0.03
60th	0.97	0.89	1.23	1.07	1.94	1.73	0.15	0.11	0.04	0.04	0.09	0.07
70th	1.21	1.09	1.60	1.40	2.45	2.19	0.25	0.20	0.06	0.05	0.17	0.12
75th	1.37	1.28	1.83	1.71	2.74	2.60	0.34	0.29	0.08	0.07	0.24	0.20
80th	1.59	1.48	2.14	1.99	3.21	2.98	0.46	0.40	0.11	0.10	0.33	0.28
85th	1.89	1.72	2.60	2.35	3.84	3.50	0.64	0.54	0.17	0.13	0.50	0.40
90th	2.36	2.14	3.18	2.85	4.95	4.29	0.93	0.77	0.26	0.21	0.78	0.62
95th	3.34	2.78	4.24	3.64	6.82	5.78	1.52	1.18	0.48	0.34	1.47	1.07
97.5th	4.32	3.82	5.37	4.71	9.71	8.17	2.30	1.84	0.84	0.63	2.36	1.83
99th	5.20	4.66	7.18	6.25	15.20	12.15	3.78	2.89	1.44	1.05	4.08	3.02
100th	8.16	6.20	29.40	11.57	50.00	27.39	50.00	7.02	29.80	3.12	50.00	6.85
Number	**1,435**		**5,795**		**1,471**		**72,172**		**13,258**		**17,347**	
Mean	**1.11**		**1.419**		**2.41**		**0.37**		**0.12**		**0.32**	

Figure 17.22: Frasers Deposit – An East-West Geological Cross Section of the Frasers Deposit

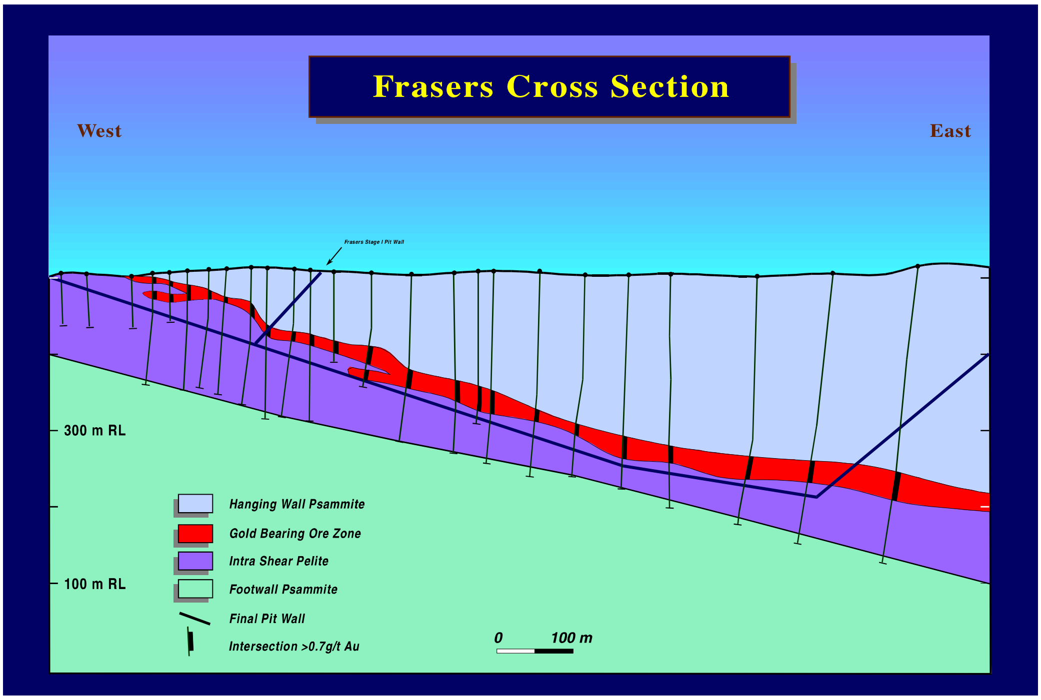


Figure 17.23: Frasers Deposit - Directional Sample Variograms of Gold, Hangingwall Mineralization

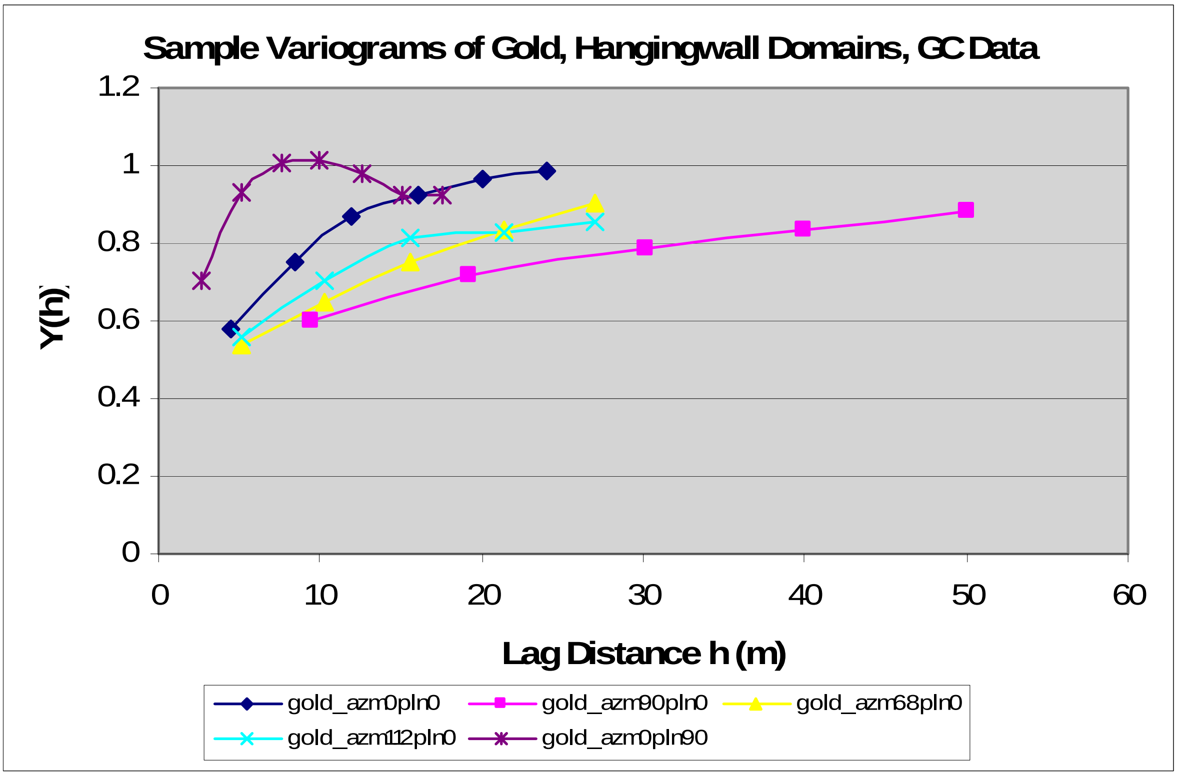


Figure 17.24: Frasers Deposit - Directional Sample Variograms of Gold, Stockwork Mineralization

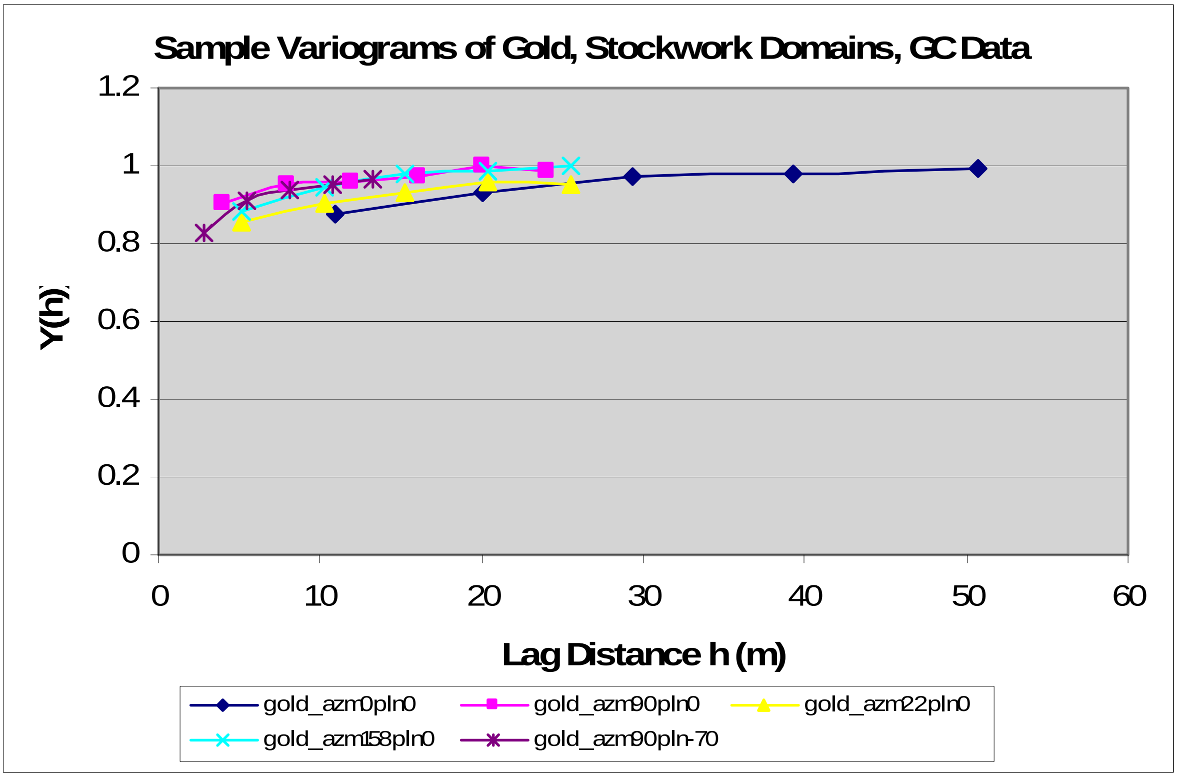


Table 17.44: Frasers Deposit - Indicator Variogram Model Parameters, Hangingwall Domain 10

Hangingwall Domain 10										
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**
0.24	0.35	0.29	37	46	8	0.36	90	205	29	Y13
0.37	0.27	0.34	37	74	11	0.39	115	295	29	Y13
0.50	0.29	0.34	41	62	8	0.37	130	295	33	Y13
0.64	0.35	0.34	31	62	7.5	0.31	280	280	37	Y13
0.79	0.35	0.37	29	54	5.5	0.28	235	245	28	Y13
0.97	0.39	0.39	29	44	5.5	0.22	235	255	24	Y13
1.21	0.38	0.47	40	44	5	0.15	235	255	19	Y13
1.37	0.35	0.56	43	44	4.2	0.09	235	280	23	Y13
1.59	0.35	0.59	44	46	4.2	0.06	250	335	23	Y13
1.89	0.35	0.62	42	42	4.2	0.03	250	335	23	Y13
2.36	0.41	0.57	35	42	3.7	0.02	250	335	26	Y13
3.34	0.45	0.5	26	42	2.7	0.05	56	335	14	Y13
4.32	0.45	0.48	30	32	2.7	0.07	56	74	14	Y13
5.20	0.45	0.48	22	28	1.8	0.07	43	40	5.5	Y13

Table 17.45: Frasers Deposit - Indicator Variogram Model Parameters, Hangingwall Domain 11

Hangingwall Domain 11										
Threshold	C0	C1	AX	AY	AZ	C2	AX	AY	AZ	Rotation 1
0.19	0.35	0.29	37	46	8	0.36	90	205	29	Y16
0.33	0.27	0.34	37	74	11	0.39	115	295	29	Y16
0.52	0.29	0.34	41	62	8	0.37	130	295	33	Y16
0.70	0.35	0.34	31	62	7.5	0.31	280	280	37	Y16
0.92	0.35	0.37	29	54	5.5	0.28	235	245	28	Y16
1.23	0.39	0.39	29	44	5.5	0.22	235	255	24	Y16
1.60	0.38	0.47	40	44	5	0.15	235	255	19	Y16
1.83	0.35	0.56	43	44	4.2	0.09	235	280	23	Y16
2.14	0.35	0.59	44	46	4.2	0.06	250	335	23	Y16
2.60	0.35	0.62	42	42	4.2	0.03	250	335	23	Y16
3.18	0.41	0.57	35	42	3.7	0.02	250	335	26	Y16
4.24	0.45	0.5	26	42	2.7	0.05	56	335	14	Y16
5.37	0.45	0.48	30	32	2.7	0.07	56	74	14	Y16
7.18	0.45	0.48	22	28	1.8	0.07	43	40	5.5	Y16

Table 17.46: Frasers Deposit - Indicator Variogram Model Parameters, Hangingwall Domain 12

Hangingwall Domain 12										
Threshold	C0	C1	AX	AY	AZ	C2	AX	AY	AZ	Rotation 1
0.40	0.35	0.29	37	46	8	0.36	90	205	29	Y11
0.64	0.27	0.34	37	74	11	0.39	115	295	29	Y11
0.92	0.29	0.34	41	62	8	0.37	130	295	33	Y11
1.18	0.35	0.34	31	62	7.5	0.31	280	280	37	Y11
1.52	0.35	0.37	29	54	5.5	0.28	235	245	28	Y11
1.94	0.39	0.39	29	44	5.5	0.22	235	255	24	Y11
2.45	0.38	0.47	40	44	5	0.15	235	255	19	Y11
2.74	0.35	0.56	43	44	4.2	0.09	235	280	23	Y11
3.21	0.35	0.59	44	46	4.2	0.06	250	335	23	Y11
3.84	0.35	0.62	42	42	4.2	0.03	250	335	23	Y11
4.95	0.41	0.57	35	42	3.7	0.02	250	335	26	Y11
0.40	0.35	0.29	37	46	8	0.36	90	205	29	Y11
0.64	0.27	0.34	37	74	11	0.39	115	295	29	Y11
0.92	0.29	0.34	41	62	8	0.37	130	295	33	Y11

Table 17.47: Frasers Deposit - Indicator Variogram Model Parameters, Stockwork Domain 40

Hangingwall Domain 40										
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**
0.01	0.36	0.51	23	25	7	0.13	93	140	100	Y5
0.02	0.49	0.43	23	25	4.6	0.08	59	51	45	Y5
0.04	0.49	0.45	23	25	3.1	0.06	41	35	32	Y5
0.06	0.49	0.45	23	22	2.8	0.06	41	32	21	Y5
0.09	0.49	0.45	21	22	2.3	0.06	35	32	15	Y5
0.15	0.49	0.45	18	19	2	0.06	32	28	10	Y5
0.25	0.49	0.45	19	20	2	0.06	32	28	10	Y5
0.34	0.49	0.45	19	20	2.1	0.06	32	28	12	Y5
0.46	0.6	0.4	18	20	3.4	0	0	0	0	Y5
0.64	0.6	0.4	18	20	3.3	0	0	0	0	Y5
0.93	0.6	0.4	18	20	2.8	0	0	0	0	Y5
1.52	0.6	0.4	18	20	3	0	0	0	0	Y5
2.30	0.6	0.4	18	20	3	0	0	0	0	Y5
3.78	0.6	0.4	18	20	3.1	0	0	0	0	Y5

Table 17.48: Frasers Deposit - Indicator Variogram Model Parameters, Stockwork Domain 41

Hangingwall Domain 41										
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**
0.01	0.36	0.51	23	25	7	0.13	93	140	100	Y5
0.01	0.49	0.43	23	25	4.6	0.08	59	51	45	Y5
0.01	0.49	0.45	23	25	3.1	0.06	41	35	32	Y5
0.02	0.49	0.45	23	22	2.8	0.06	41	32	21	Y5
0.03	0.49	0.45	21	22	2.3	0.06	35	32	15	Y5
0.04	0.49	0.45	18	19	2	0.06	32	28	10	Y5
0.06	0.49	0.45	19	20	2	0.06	32	28	10	Y5
0.08	0.49	0.45	19	20	2.1	0.06	32	28	12	Y5
0.11	0.6	0.4	18	20	3.4	0	0	0	0	Y5
0.17	0.6	0.4	18	20	3.3	0	0	0	0	Y5
0.26	0.6	0.4	18	20	2.8	0	0	0	0	Y5
0.48	0.6	0.4	18	20	3	0	0	0	0	Y5
0.84	0.6	0.4	18	20	3	0	0	0	0	Y5
1.44	0.6	0.4	18	20	3.1	0	0	0	0	Y5

Table 17.49: Frasers Deposit - Indicator Variogram Model Parameters, Stockwork Domain 42

Hangingwall Domain 42										
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**
0.01	0.36	0.51	23	25	7	0.13	93	140	100	Y5
0.02	0.49	0.43	23	25	4.6	0.08	59	51	45	Y5
0.03	0.49	0.45	23	25	3.1	0.06	41	35	32	Y5
0.04	0.49	0.45	23	22	2.8	0.06	41	32	21	Y5
0.06	0.49	0.45	21	22	2.3	0.06	35	32	15	Y5
0.09	0.49	0.45	18	19	2	0.06	32	28	10	Y5
0.17	0.49	0.45	19	20	2	0.06	32	28	10	Y5
0.24	0.49	0.45	19	20	2.1	0.06	32	28	12	Y5
0.33	0.6	0.4	18	20	3.4	0	0	0	0	Y5
0.50	0.6	0.4	18	20	3.3	0	0	0	0	Y5
0.78	0.6	0.4	18	20	2.8	0	0	0	0	Y5
1.47	0.6	0.4	18	20	3	0	0	0	0	Y5
2.36	0.6	0.4	18	20	3	0	0	0	0	Y5
4.08	0.6	0.4	18	20	3.1	0	0	0	0	Y5

17.9.6 Dilution of Resource Estimates

Historically at Frasers, there has been a tendency to over-estimate ore grade in areas of weakly developed gold mineralization. A small amount of dilution was applied to both the Hangingwall and Stockwork mineralization primarily in areas of weakly developed gold mineralization.

Twenty centimetres of barren dilution has been applied to the Hangingwall and this has improved reconciliations in areas where the style of mineralization thins to less than two metres. In the Stockwork mineralization, panels estimated to contain less than 30 percent ore at 0.5 g/t cut-off incur eight percent dilution at 0 g/t cut-off. Panels estimated to contain between 30 and 80 percent ore at 0.5 g/t cut-off incur three percent dilution.

17.9.7 Resource Reporting and Classification

The Frasers Open Pit resource estimate is reported within the limits shown in Table 17.50. The underground resource volume called “Panel 1” which extends up dip to the toe of the ultimate pit has been excised from the open pit resource area.

The Frasers resource area is effectively drilled on a 50 by 50m pattern with areas of 25 by 25m infill and some areas of 100 by 100m step out drilling. The classification of the estimated resources in the Frasers Open Pit is essentially based on the drill hole spacing and is shown in Table 17.51. For the Stockwork mineralization, an additional constraint is placed on the definition of Measured and Indicated resources. For the resources in a panel to remain Measured given that the drill hole spacing is 25 by 25m, the percentage of the panel estimated to be ore at the 0.5 g/t cut-off must be at least 80 percent. In the case of Indicated estimates, in addition to the drill hole spacing, of 50 by 50m, the estimated percentage or ore in the panel at 0.5 g/t cut-off must be at least 30 percent.

Table 17.50: Frasers Deposit - Reporting Limits of the Open Pit Resource Estimates

Direction	From (m)	To (m)
Easting	69,000	70,550
Northing	11,425	13,000
RL	130	560

Table 17.51: Frasers Deposit - Resource Classification Methodology for the FR05 Resource Model

Mineralization Style	Classification Criteria		
	Measured	Indicated	Inferred
Hangingwall	<50 x 50 metres	<100 x 100 metres	>100 x 100 metres
Stockwork	<25 x 25 metres, and >80% Block above 0.5 g/t Au	<50 x 50 metres, and >30% Block above 0.5 g/t Au	>50 x 50 metres

17.9.8 Frasers Open Pit Resource

The global recoverable resource estimates for the Frasers Open Pit mineralization as of December 31, 2009 are presented in Table 17.52 below. The cut-off grade used is 0.5 g/t. The resource contains less than one percent of oxide material.

Table 17.52: Frasers Deposit - Open Pit Resource Estimates, 0.5 g/t Au Cut-off

Category	Total Resources as at December 31, 2009		
	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Measured	12.14	1.53	0.60
Indicated	28.36	0.91	0.83
Measured & Indicated	40.50	1.10	1.43
Inferred	9.41	0.7	0.21

17.9.9 Validation of Resource Estimates

The Frasers resource model has been checked with a number of processes to identify shortcomings in the model:

- The panel estimates were plotted on screen in section and plan and viewed in 3D. Estimates of panel grades were compared to neighbourhood sample grades and the distribution of high grades particularly noted. Reasonable comparison of the average model grades and sample grades for the three Hangingwall domains were achieved.

- A limited independent audit of the modelling process and results was undertaken by H&S. This audit found the Oceana data analysis and modelling process to be readily reproducible and that the differences between the Oceana and the H&S estimates in the Stockwork mineralization could be resolved by changes to the block support adjustment based on the variogram of the grade control sample data.

- In addition, cumulative reconciliation for the period January 01, 2006 to December 31, 2009 shows a reasonable comparison.

Table 17.53: Frasers Open Pit Reconciliation (0.5 g/t Au Cut-off) from January 01, 2006 to December 31, 2009

GC Survey Adjusted			Resource Model FR05			Actual/Model (FR05)		
tonnes	g/t	ozs	tonnes	g/t	ozs	tonnes	g/t	ozs
15,601,583	1.28	644,423	15,474,190	1.27	630,551	1.01	1.01	1.02

17.9.10 Estimation of Recoverable Sulphur Grade

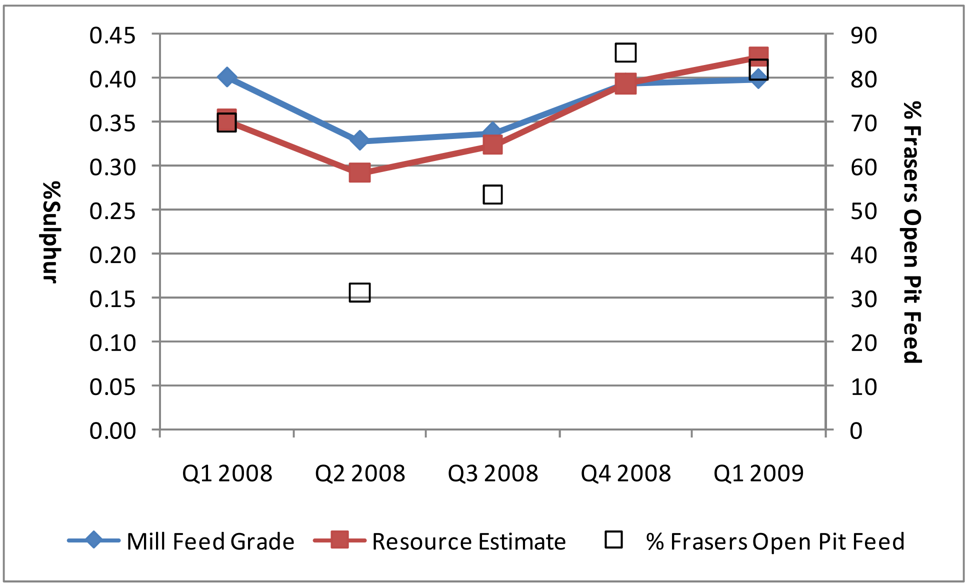
In July 2005, a model of the recoverable sulphur grade for the Frasers Open Pit was completed on behalf of Oceana by H&S. There are two aspects to the problem of sulphur estimation at Frasers; first there is a significant under-sampling of sulphur in the resource data and second, gold and sulphur are geologically and statistically related so the sulphur grade of the ore is influenced by mining to a gold cut-off grade.

Within the 2004 Frasers ultimate pit, some 13,800 out of a possible 72,500 samples have been assayed for sulphur and these data are spatially clustered. Because the recoverable resource estimates of sulphur in a panel rely on the variability of the local grades of both sulphur and gold, the problem of incomplete sulphur data has been tackled by simulating the sulphur grades at all sample locations where no sulphur grade exists. This simulation seeks to honour the statistical properties of the existing sulphur data as well as the gold–sulphur relationship. This process is an approximation because of the paucity of data, but it is better than using a simple regression between gold and sulphur to model the sulphur grade.

With gold and sulphur data at all sample locations, it is possible to generate the local conditional distributions of gold and sulphur for each panel in the model with a regular application of indicator kriging. The estimates of recoverable sulphur grade for a range of gold cut-off grades are then achieved by assuming that the ratio of sulphur grade to gold grade observed in the existing sulphur – gold sample data will be the same as the ratio of recoverable sulphur to gold grades for mining blocks with the panels of the model at the same cut-off grades.

Figure 17.25 shows a comparison of the predicted sulphur grade of the ore to the back calculated sulphur grade from the mill for the period January 01, 2008 to March 31, 2009. The second quarter, 2008 has a large proportion of stock pile feed for which the sulphur grade has been approximated. The results suggest the prediction of sulphur head grade is performing reasonably given data limitations.

Figure 17.25: Frasers Stage 4, Model to Mill Sulphur Reconciliation



17.10 Frasers Underground

The re-optimisation, and expansion of the Frasers Open Pit in late 2009 has resulted in the moving of the open pit / underground resource boundary to the east.

17.10.1 Resource Data

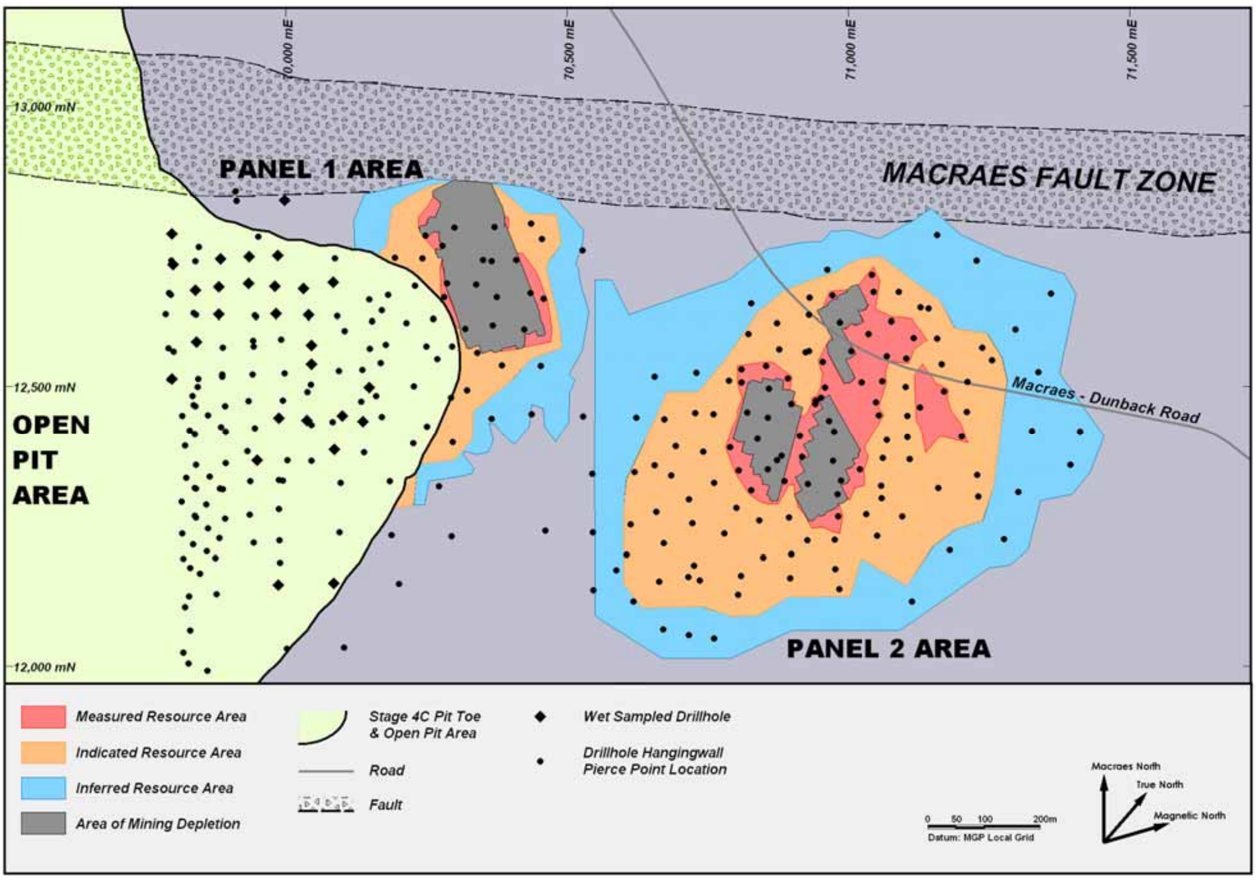
The underground Resource estimates are based on a combination of diamond and RC percussion drilling. Diamond drilling dominates the database representing 603 of the 629 drill holes or 95% of the drilling metres. The RC percussion drilling is limited to Panel 1 and represents 26 of the 101 drill holes or 23% of the drill metres that intersect the Panel 1 Hangingwall.

Table 17.54: Frasers Underground - Drill Hole Database

Prospect	Diamond			Reverse Circulation		
	(No)	(m)	(%)	(No)	(m)	(%)
Panel 1 Area	75	25,134	76.6	26	7,662	23.4
Panel 2 Area	528	93,926	100.0	0	0	0.0
Total	**603**	**119,060**	**95.1**	**26**	**7,662**	**4.9**

Sampling bias has been identified in the wet RC percussion drilling (Figure 17.26). Due to the shifting of the open pit / underground resource boundary to the east, wet sampling no longer impacts the underground resources.

Figure 17.26: Frasers Underground - Location of Wet RC Percussion Drill Holes



17.10.2 Geology and Mineralization

The mineralization targeted in the FRUG is the down-dip extension of the hangingwall shear currently exploited in the Frasers open cut mine. Two main regions have been reasonably drill tested: Panel 1 and Panel 2. Panel 1 is located approximately 600m beyond the planned Frasers Stage 5 pit while Panel 2 is located a further 300m to the southeast.

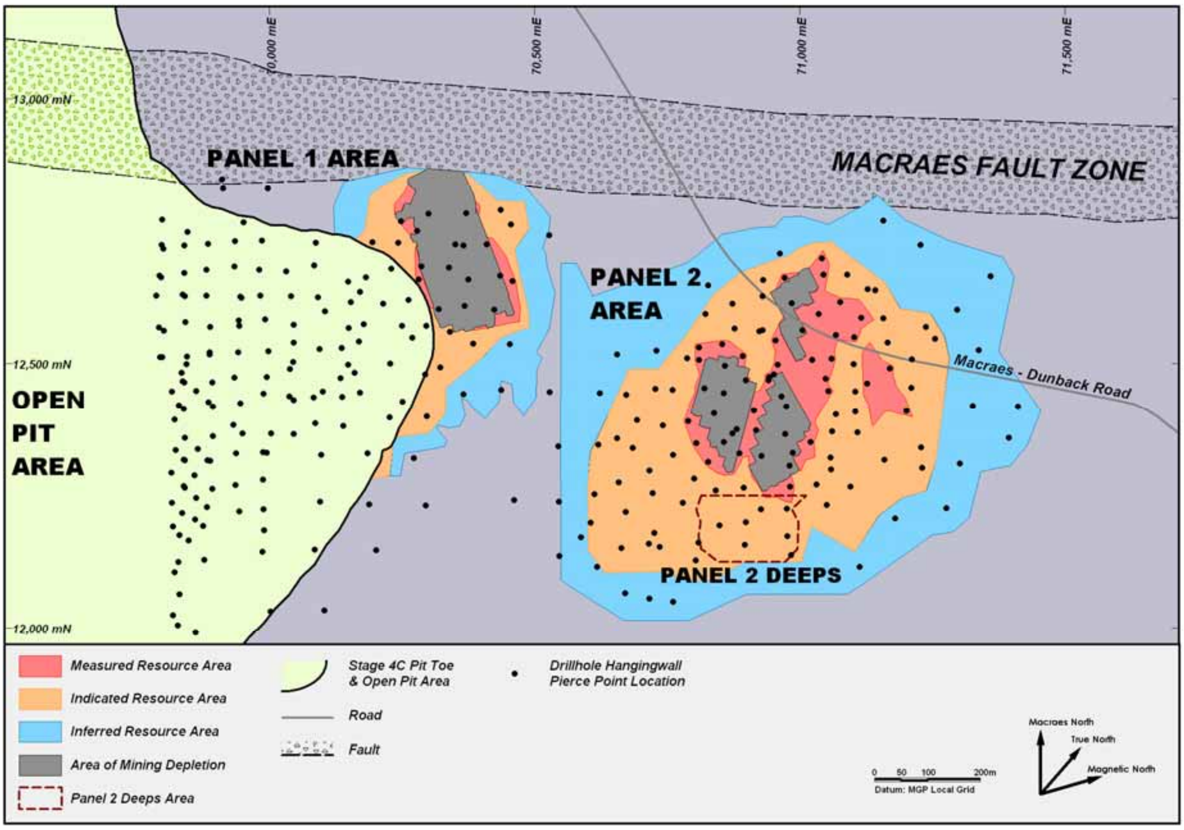
The geological controls for the FRUG are consistent with those described in the Frasers Open Cut. The mineralization is contained within the 80m to 100m thick intra-shear pelite, bounded by hangingwall and footwall psammites. Cataclasite, lode schist (concordant zones) and stockwork gold mineralization have been identified with the highest-grade mineralization located proximal to the hanging wall contact. Further description of geology is provided in Sections above.

Geological constraint applied in grade estimation is limited to the construction of Hangingwall and footwall psammite surfaces. These surfaces were used to capture and code drill hole data and composites.

17.10.3 Grade Estimation Approach

The grade estimates for Panel 1 and Panel 2 have been generated using Ordinary Kriging (OK) in geostatistical software PANGEOS. Multiple Indicator Kriging (MIK) E-type estimates have also been generated using GS3 software for Panel 2 Deeps. Figure 17.27 presents the relative location of the different zones.

Figure 17.27: Frasers Underground - Location Map of Zones



The OK approach is predicated on unfolded composite data. Oceana unfolded the composite data using the upper hangingwall (UHW) contact as a reference surface.

A summary of the steps taken in the OK model construction are provided as the following points:

- the drill hole data is coded with the geological interpretation (UHW etc), and composited;
- the data is unfolded relative to the UHW pelite/psammite contact;

- ordinary kriging is implemented; and
- the model is restored (refolded) back into the original UHW plane.

Sulphur grades for the Panel 1 and Panel 2 estimates have been generated via a regression based on the global data set. This is considered a lower confidence estimate than the gold estimate as significant dispersion exists between gold and sulphur. In addition, the relationship between gold and sulphur is not well defined above 5 g/t Au.

17.10.4 Panel 1 and 2 Estimate

As discussed above, the Panel 1 and 2 estimates have been generated with OK. The estimates have been based on 6,532 and 4,201 composites captured within the panel regions 1 and 2 respectively. The statistics are presented below in Table 17.55.

Table 17.55: Frasers Underground – Gold (g/t Au) Sample Statistics for Panel 1 and 2, 1m Composite Data

	Panel 1	Panel 2
Number of Samples	6,532	4201
Minimum Grade (g/t)	0.0	0.01
Maximum Grade (g/t)	65.8	15.0 (top cut to 15)
Mean (g/t)	2.13	3.28
Median (g/t)	1.27	2.74
Coefficient of Variation	1.65	0.77

The Panel 1 estimate is based on a final 5m x 5m x 1m block size as presented in Table 17.56.

Table 17.56: Frasers Underground - Model Dimensions Panel 1 and Panel 2

Panel 1			
	Minimum (m)	**Maximum (m)**	**Block Size (m)**
Easting (m)	69,735	70,560	5
Northing (m)	12,150	12,870	5
RL (m)	100	400	1
Panel 2			
Easting (m)	70,500	71,520	5
Northing (m)	12,000	12,850	5
RL (m)	-170	210	1

Bulk density and specific gravity test work is in progress (20 determinations to date) and currently indicates bulk densities around $2.70 gcm^{-3}$. Historically $2.60 gcm^{-3}$ has been used and was applied to the tonnage reporting at FRUG.

Global comparisons of the resource model and the input model versus the composite data have been completed by Oceana staff. The model reproduces the input composites adequately, as shown by the statistical summary provided in Table 17.57.

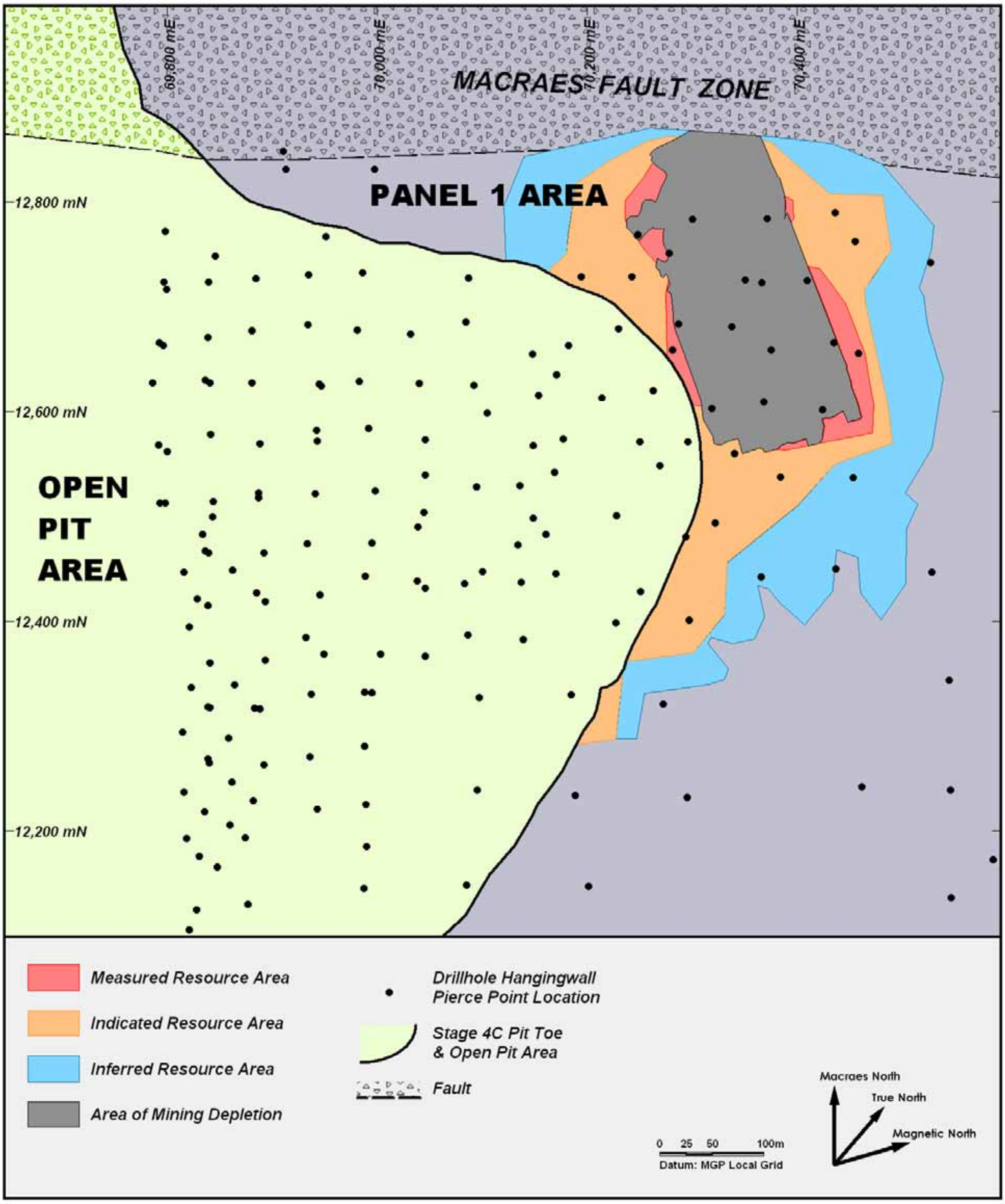
Table 17.57: Fraser Underground – Comparison Composite and Block Mean Grades

	Panel 1	Panel 2
Composite Data Mean (g/t Au)	2.13	3.28
Estimate Mean (g/t Au)	1.90	2.78

The Frasers Panel 1 estimate has been classified by Oceana on the basis of mining method, geological confidence and drilling density. Panel 1 is comprised almost entirely of hangingwall mineralization which generally shows reasonable continuity. On average the drill hole spacing of the central region is 50m by 50m. Peripheral to the core of the panel, the drill density is approximately 100m by 100m (see Figure 17.28).

Panel 1 Resource has decreased since the 2009 Report due to the inclusion of Frasers Open Pit Stage 6 which consumes a portion of the Panel 1 underground resource. A 20 meter vertical and horizontal exclusion zone around the pit design has also been included in the depletion of Panel 1. Mineral Resources have been transferred to the Frasers Open Pit Resource and are reported accordingly (Refer to Section 17.9.8).

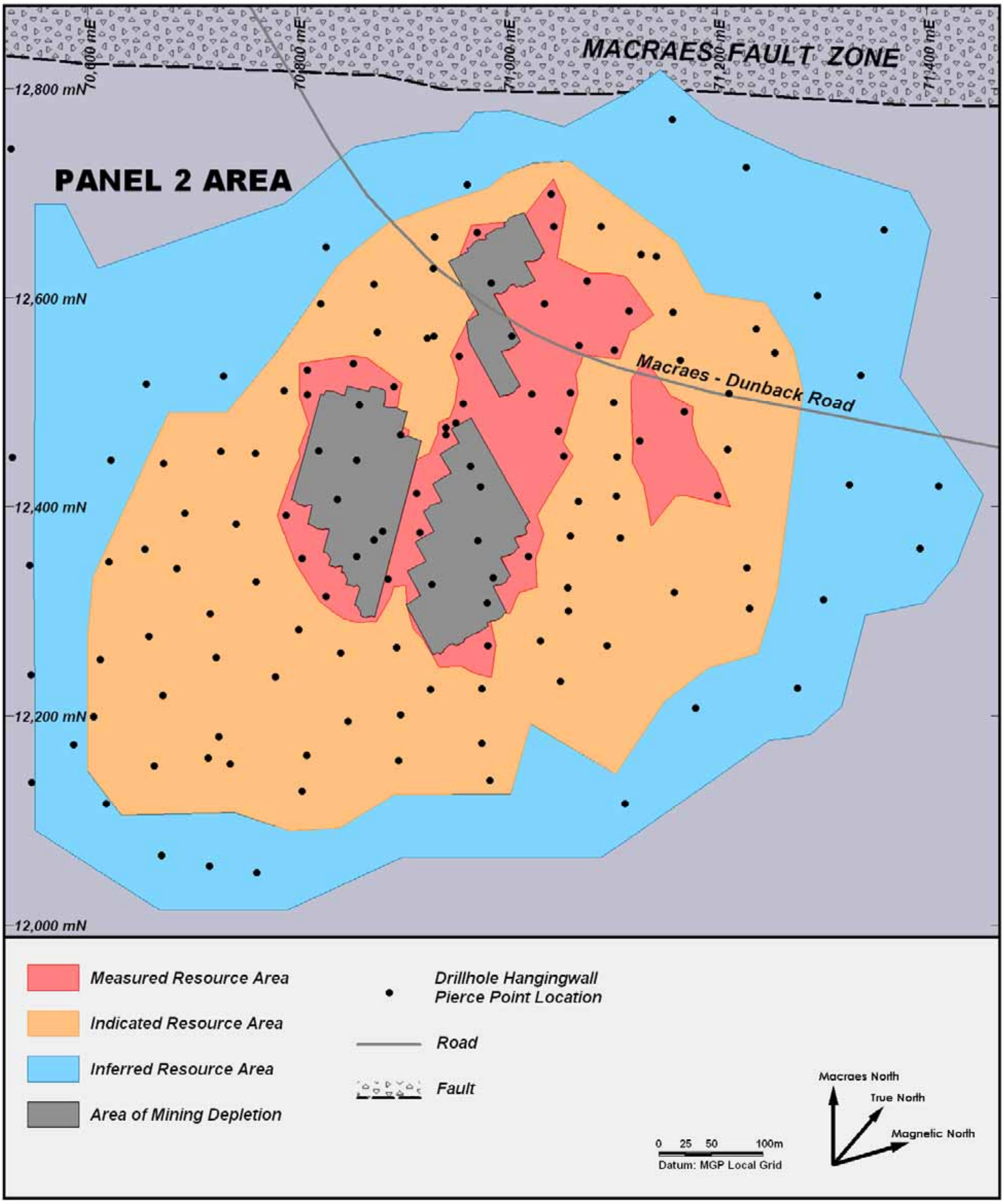
Figure 17.28: Frasers Underground - Resource Classification for Panel 1 Hangingwall Mineralization



The FRUG Panel 2 estimate has been classified by Oceana on the basis of mining method, geological confidence and drilling density. Panel 2 is comprised of both hanging wall and stockwork mineralization, although no stockwork mineralization was modelled. The hanging wall mineralization generally shows reasonable geometric continuity, while stockwork mineralization tends to be quite erratic.

On average the drill hole spacing within the core of Panel 2 is 50m by 50m with approximately 100m 100m spacing beyond (see Figure 17.29). Panel 2 resource now contains mineralisation previously modelled and labelled as Panel 2 extension.

Figure 17.29: Frasers Underground - Resource Classification of Panel 2 Hangingwall Mineralization



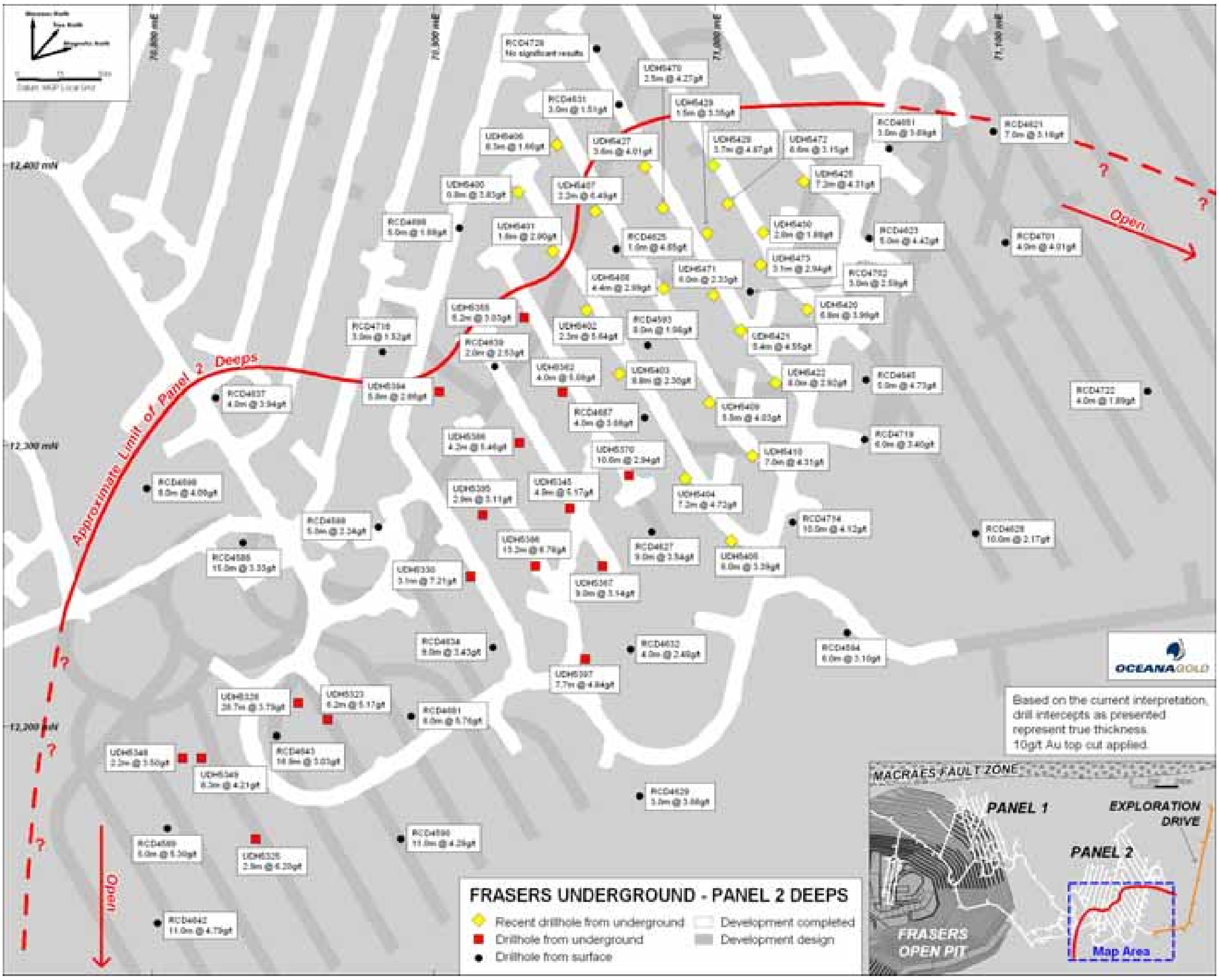
17.10.5 Panel 2 Extension

Previously, Oceana had estimated a resource for Panel 2 Extension, which extended approximately 300m north-east of the existing Panel 2 resource. This has now been incorporated into the Panel 2 Resource model and is reported within the Panel 2 Resource.

17.10.6 Panel 2 Deeps Estimate

Infill drilling has been undertaken from underground (see Figure 17.30) to supplement drilling collared from surface. A resource estimate based on drill holes completed up to July 31, 2009 (up to and including UDH5430) has been completed and is detailed below.

Figure 17.30: Panel 2 Deeps Surface and Underground Drill Hole Intercepts



The estimate uses ordinary kriging within a geologically constrained wireframe. A 14 g/t top cut (97.5 percentile) was applied prior to compositing to 1m intervals. The summary 1m composite statistics are in Table 17.58 below.

Table 17.58: Frasers Underground, Panel 2 Deeps - Summary of uncut 1m Gold (g/t) Composite Statistics

Domain	Number	Mean	Minimum	Maximum	Std Dev	CV
2	192	4.01	0.19	26.17	3.05	0.76
3	149	3.90	0.02	39.50	4.47	1.15

Panel 2 Deeps mineralization is located approximately 20m beneath the Panel 2 Hangingwall shear, dipping shallowly to the east and averaging 5m thick. The mineralization comprises a zone of quartz cataclasite and siliceous breccias which appears to have a broad negative correlation to the overlying

Hangingwall as shown on Figure 17.31. The Panel 2 Deeps zone marks a structural boundary between steeply dipping foliation below and moderately dipping foliation above.

The zone is largely closed off to the west and northwest but remains open to the east-northeast and southwest.

Figure 17.31: Frasers Underground - Section 12,260mE of Panel 2 Deeps

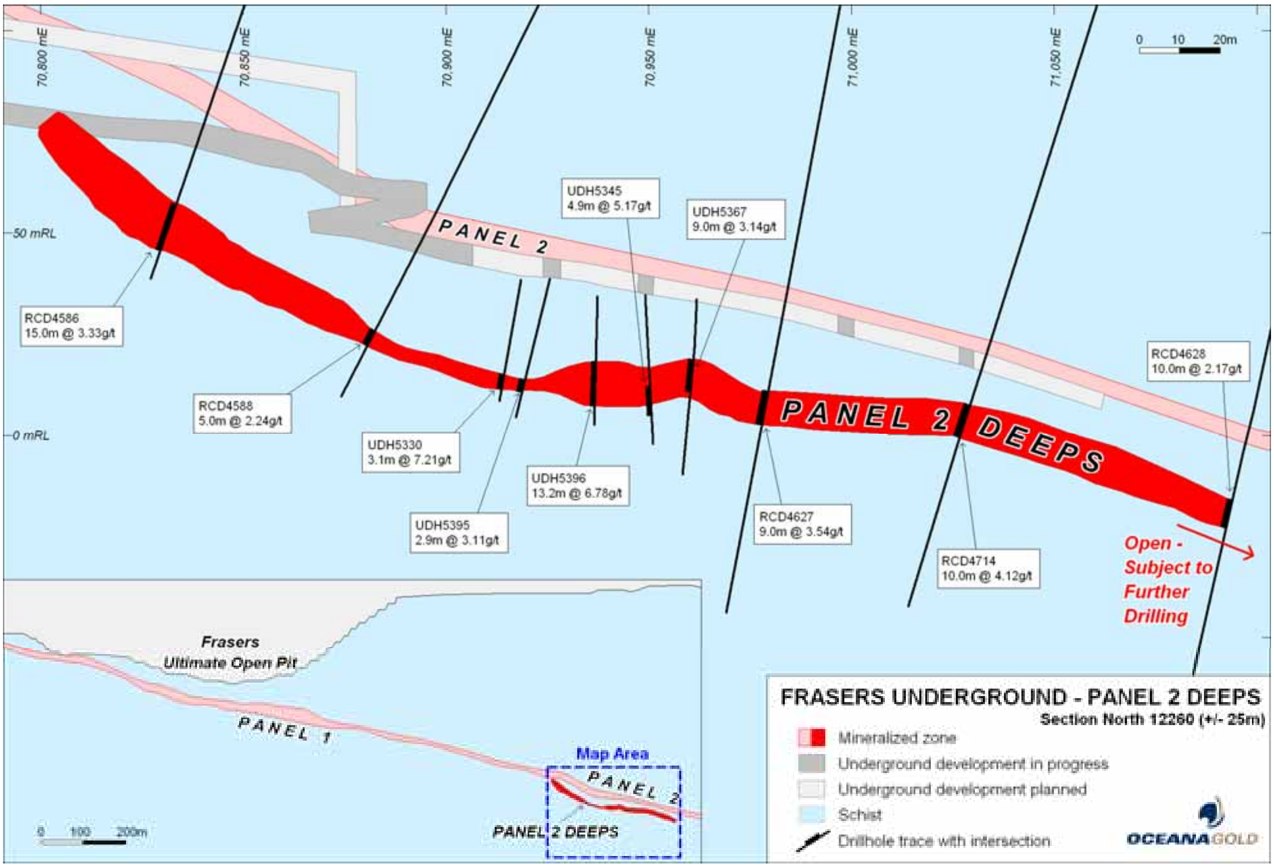


Table 17.59: Frasers Underground, Panel 2 Deeps - Variogram Model Parameters

Hangingwall Domain 42										
Domain	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**
2 and 3	0.26	0.49	26	25	2	0.25	29	27	4	Y 15

Note the first and second structures used exponential and spherical models respectively

Domain 2 defines a core to the mineralisation that has been drilled to approximately 25m by 25m, while Domain 3 is more sparsely drilled. Given this, Domain 2 has been classified as Indicated, whilst domain 3 has been classified as Inferred. The resource is reported in Table 17.62.

Table 17.60: Frasers Underground Panel 2 Deeps – Search Parameters

Domain	Minimum Number of Samples	Maximum Number of Samples	Maximum Distance to Sample	X Y Z Search	Maximum Number of Samples per Drill Hole
2	4	32	35	35m x 35m x 10m	4
3	4	32	70	65m x 65m x 20m	4

A bulk density of 2.60 t/m^3 was applied for tonnage reporting, consistent with Panel 2 bulk density.

The model dimensions are shown below in Table 17.61.

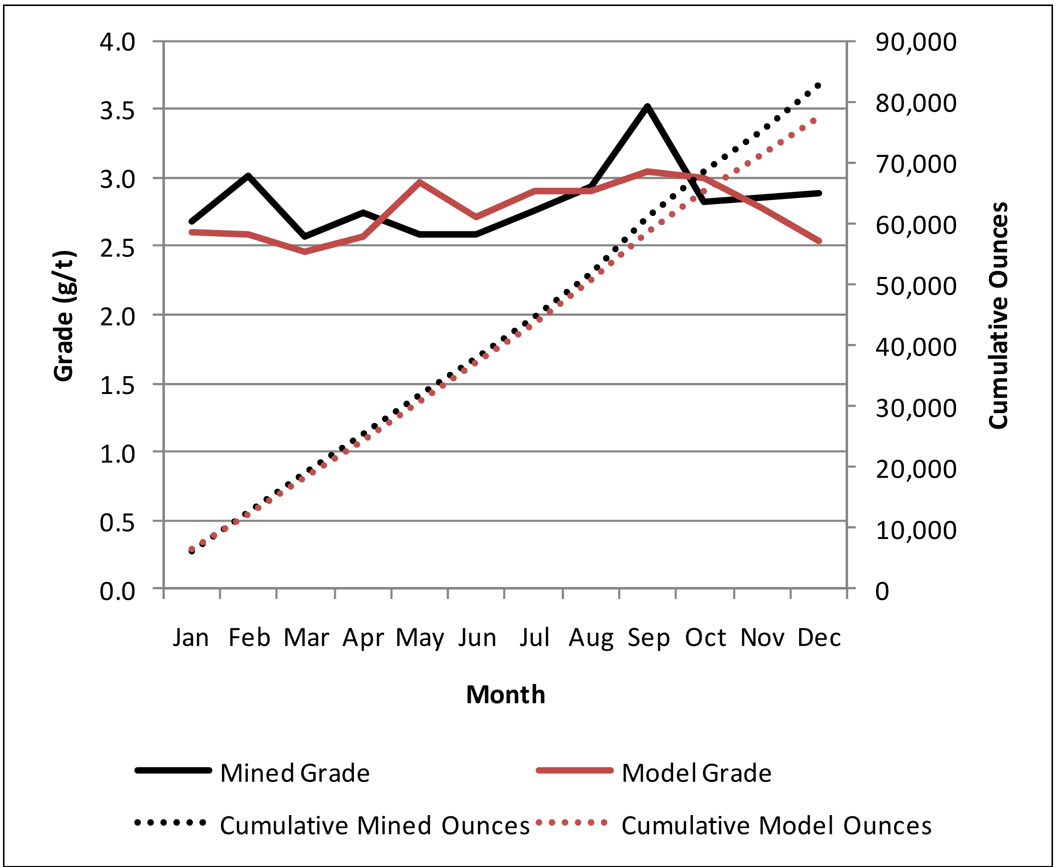
Table 17.61: Frasers Underground, Panel 2 Deeps - Model Dimensions

	Minimum (m)	Maximum (m)	Block Size (m)
Easting (m)	70,750	71,200	5
Northing (m)	12,075	12,450	5
RL (m)	-75	125	1

17.10.7 Reconciliation

Figure 17.32 summarises the reconciliation of the 2009 monthly Panel 2 model estimates against the grade control estimates. The reconciliation suggests the Panel 2 estimates have been acceptable, both on a monthly and an annual basis. It is difficult however to meaningfully reconcile the grade control estimate to the mill estimates because the Macraes open pit feed and underground feed are blended for processing.

Figure 17.32: Panel 2 Model to Grade Control Reconcilation for 2009



17.10.8 Combined Resource Reporting

The combined grade tonnage reporting for the FRUG grouped by NI 43-101 resource category is presented below as Table 17.62.

Table 17.62: Frasers Underground - Frasers Underground Resource by Category as at December 31, 2009

Resource Area	Measured		Indicated		Measured & Indicated			Inferred Resource		
	Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Panel 1	0.22	1.81	1.13	1.46	1.36	1.52	0.07	1.03	1.40	0.04
Panel 2	1.16	3.34	3.75	2.44	4.91	2.65	0.42	3.59	2.20	0.25
Panel 2 Deeps	.	.	0.35	3.56	0.35	3.56	0.04	0.54	3.69	0.06
Total	**1.38**	**3.10**	**5.23**	**2.30**	**6.61**	**2.47**	**0.52**	**5.17**	**2.17**	**0.36**

17.11 Golden Bar

17.11.1 Introduction

The Golden Bar resource (GB02a) was estimated by Oceana in November 2002. This 2002 estimate represented a resource update and was completed primarily to remove and / or mitigate the impact of wet sampling bias associated with wet RC percussion drilling. During August and September 2002, 981m of RC percussion and 116m of diamond drilling were completed at Golden Bar. This drilling twinned existing RC percussion drill holes that were known to be wet. A positive wet sampling bias of approximately 34 percent was determined to exist. The twinned wet RC percussion drill holes were removed from the resource database and replaced by their dry RC percussion / diamond counterparts. All the remaining wet samples were adjusted with globally determined grade dependent factors that were derived from the twin drill hole sample pairs. The treatment of the wet sample bias was audited by independent consultants Hellman and Schofield.

17.11.2 Database

The resource estimate at Golden Bar is based on a total of 277 drill holes for 39,047m. There have been five phases of exploration drilling at Golden Bar.

In July 1985, BP Minerals drilled four diamond holes totalling 441.2m (GBDDH001 to GBDDH004). This programme was followed by a reverse circulation (RC percussion) drilling programme of six holes totalling 303m (GBRC001-GBRC006).

In December 1994, Oceana conducted a diamond drilling programme consisting of 5 holes for 496.1m.

During December 1995, a RC percussion drilling programme consisting of five holes (RCH2084 - RCH2088) for 606m was completed on the Golden Bar Prospect.

The main phase of drilling occurred between June 1996 and October 1997. This drilling programme located and delineated much of the current resource using 25 x 25m drill spacing. The drilling also tested the strike and dip extensions to the currently known resource.

During August and September 2002, 981m of RC percussion and 116m of diamond drilling were completed at Golden Bar. This drilling twinned existing wet RC percussion drill holes.

A breakdown of drilling by sample type is shown in Table 17.63. Figure 17.33 shows drill hole collar locations.

Table 17.63: Golden Bar Deposit - Drilling Summary

	GB02 Resource Estimate			Prospect Total		
Hole Type	**Number**	**Metres**	**Percentage**	**Number**	**Metres**	**Percentage**
Percussion	0	0	0	0	0	0
Reverse Circulation	243	35,106	89	243	35,106	89
Diamond* (DDH & RCD)	34	3,941	11	34	3,941	11
Total	**277**	**39,047**	**100**	**277**	**39,047**	**100**

*Diamond figures include holes drilled with an RC percussion pre-collar and tailed with diamond core.

In addition to the drill hole data, BP Minerals excavated four trenches (totalling 250m) across the Golden Bar structure. The trenching was completed in 1985. In November/December 1994 a further five trenches totalling 452m were excavated in the Golden Bar area to test soil anomalies.

A total of 224m of drill access track with exposed mineralization was mapped and sampled (113 samples) at Golden Bar. An additional 19 selective rock chip samples were collected from exposures during mapping to assist geological understanding of the mineralization present. This work tested known near surface mineralization and assisted in geological interpretation and modelling. The highest-grade samples were returned from the sigmoidal quartz veins, which is consistent with the drilling results.

The trench and chip assay data has been excluded from the grade estimates.

17.11.3 Wet Sample Bias

Sampling bias associated with wet RC percussion drill hole samples had been identified elsewhere at Macraes with a positive grade bias of approximately 30% common. Based on these observations, and the fact that a significant number of wet samples were present at Golden Bar, 3 RC percussion and 7 diamond-tailed drill holes (for a total of 98m of RC percussion and 116m of diamond drilling) were drilled to twin wet RC percussion drill holes. These twin drill holes indicated a positive wet sampling bias of approximately 34% was present and required to be addressed. In Figure 17.34, both the Q-Q and scatter plots show the sample bias.

Figure 17.33: Golden Bar Deposit - Drill Hole Collar Plan

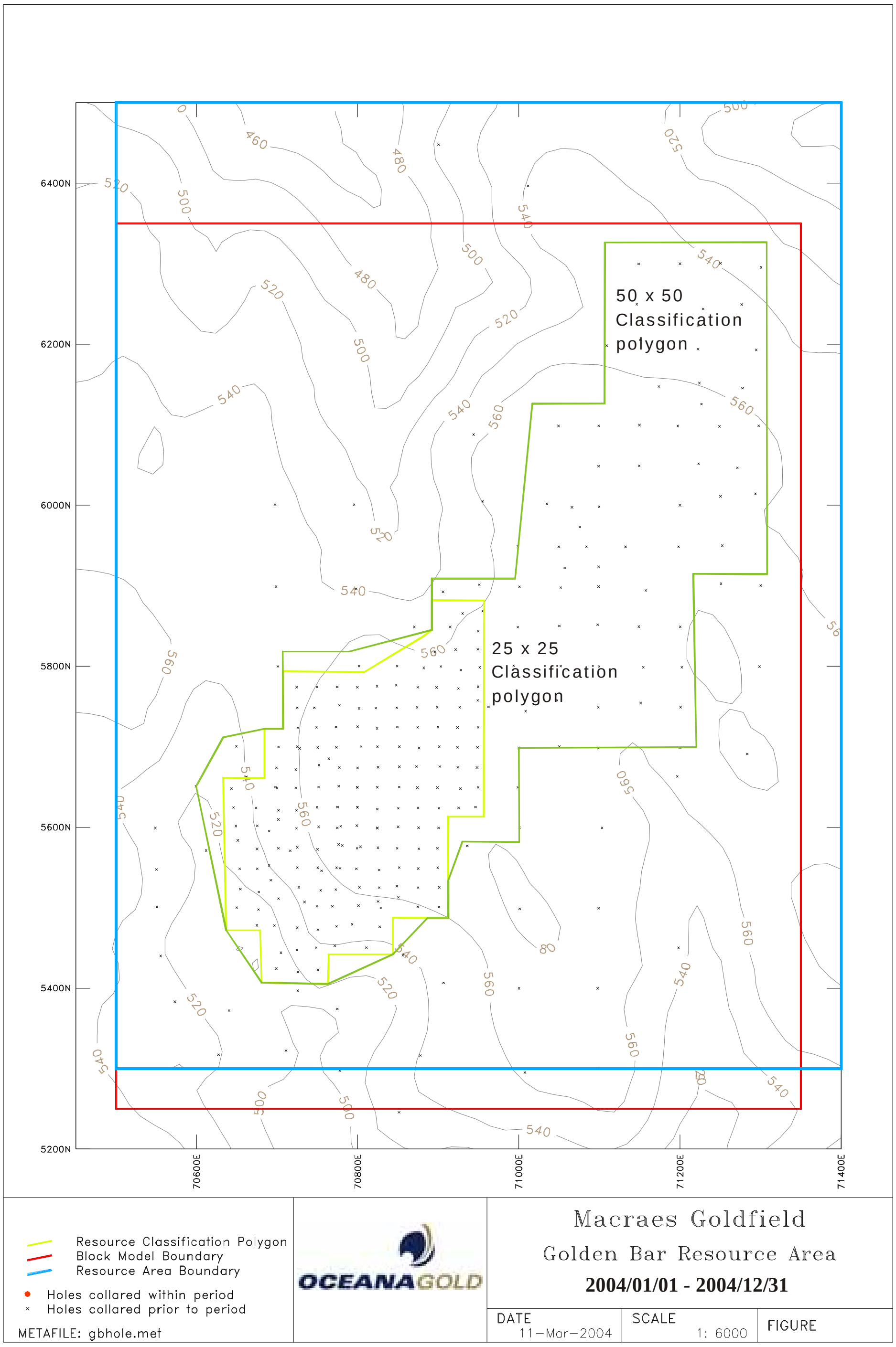
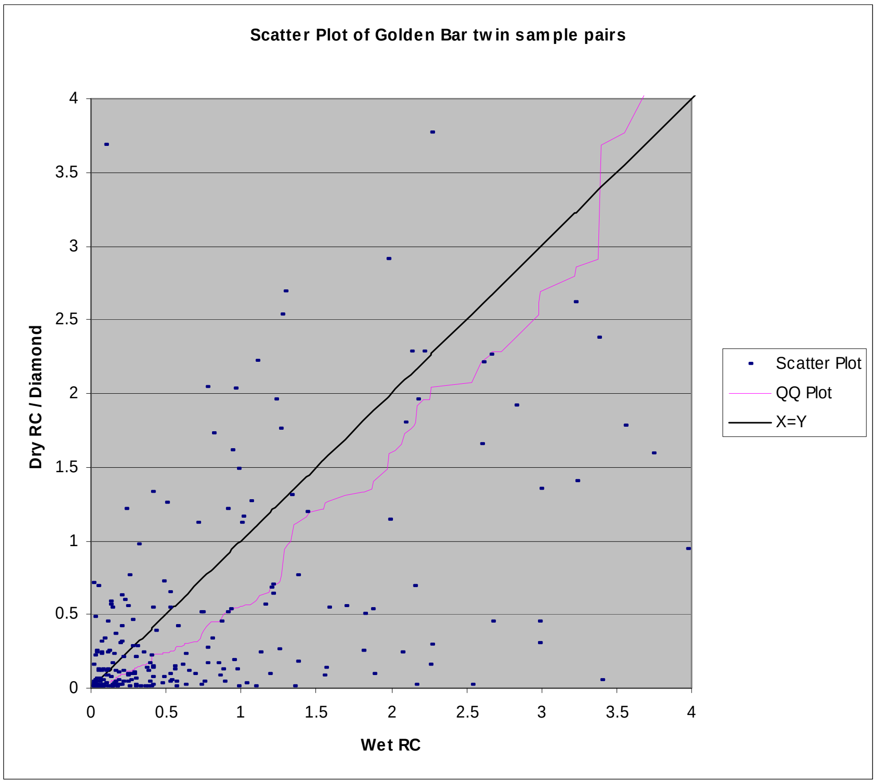


Figure 17.34: Golden Bar Deposit - Scatter and Q-Q Plot of Wet versus Dry Samples



Using the twin sampling pairs to compare class means a set of grade dependent, global bias factors have been determined. All twinned wet sampled RC percussion drill holes were removed from the resource database and replaced by their dry RC percussion / diamond twins. All the remaining wet sample grades were factored with the set of factors shown in Table 17.64.

Table 17.64: Golden Bar Deposit - Wet Sample Bias Factors

Wet Threshold (g/t)	Class Means (g/t)		Ratio
	Original	Twin	
0.30	0.11	0.04	2.51
0.71	0.47	0.23	2.11
1.26	0.97	0.53	1.82
3.40	2.10	1.75	1.20
45.00	9.78	6.16	1.59

Oceana believe that the removal/factoring approach is reasonable, however if further mining is undertaken, then further drilling would be required to replace some of the remaining wet RC percussion drilling and thereby mitigate risks associated with the wet RC percussion drilling. There are no plans in the short term to mine Golden Bar, but Oceana would review the replacement drilling requirements if mining were to be considered.

17.11.4 Geology Model

The Golden Bar prospect lies some 400m vertically above the interpreted position of the HMSZ Footwall Shear and is located within the hangingwall psammites. Golden Bar is grouped with the Eastern Lodes, which lie 2-3km east of the outcrop position main shear zone as shown on Figure 17.35. The main shear zone thins to the south of the Ounce deposit, which is coincident with the start of the Golden Bar shear zone.

Two distinctive structural styles have been identified at Golden Bar. Concordant lodes which anastomose and are generally thinly developed, and sigmoidal vein structures. The sigmoidal veins are strongly, mineralized, dominated by quartz veining. These structures link between the upper and lower concordant lodes.

The concordant lodes vary in style from thin (<1m) discrete cataclastic shears to thick (15m) quartz rich lode schist. South or south-easterly dipping shears are generally thin, highly sheared, while flat or northerly dipping shears are thick, strongly mineralized and show evidence of extension.

Two major shears are present as illustrated in Figure 17.36. These structures are 40m apart at surface but converge into a single structure at depth with the line of separation between these structures trending north-east. The lower shear west of this splitting is thickly developed and strongly mineralized. The rock between the shears contains a number of sigmoidal extension veins. Although no gross lithological differences could be clearly identified from logging, it is likely that this rock is more competent than the surrounding rock mass and has accommodated deformation by brittle extension, thus creating sites for development of the sigmoidal veins.

Figure 17.35: Golden Bar Deposit - Geology Plan

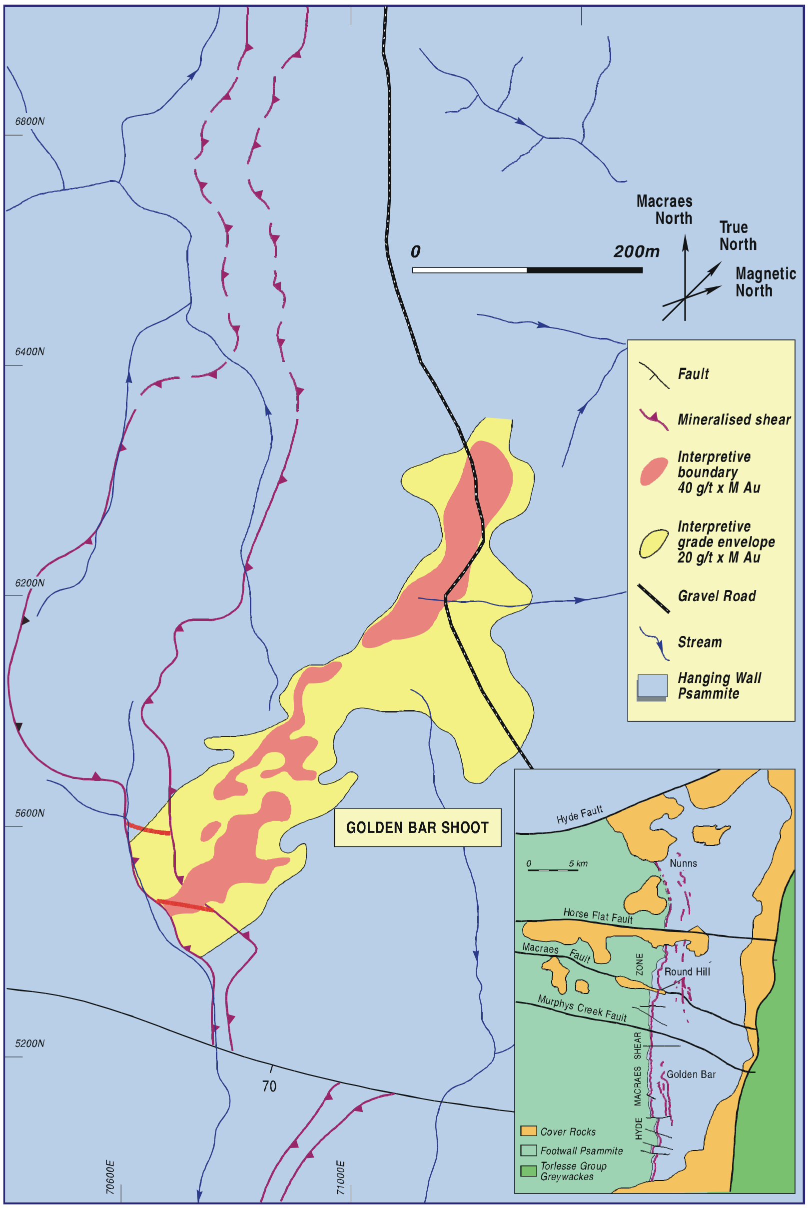
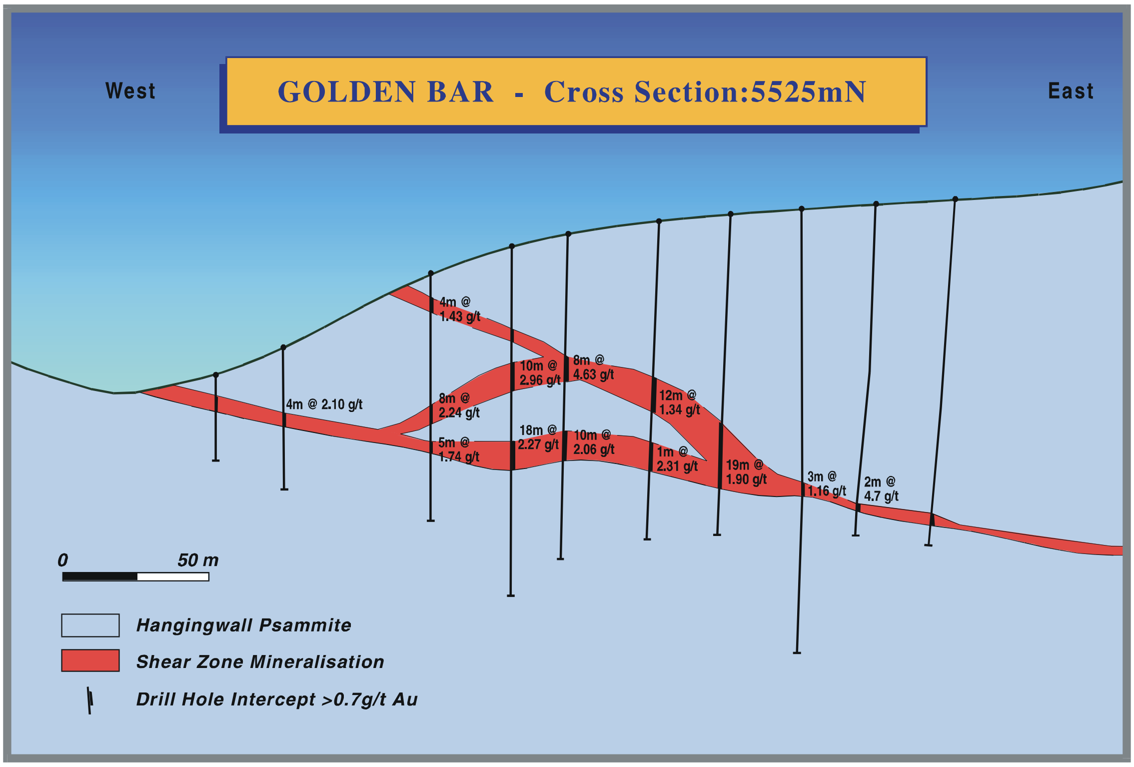


Figure 17.36: Golden Bar Deposit – Schematic Cross Section



The sigmoidal vein packages have a curved tabular geometry, striking to the north-east and dipping to the north-west at around 25°. The vein dip is steepest (and most dilatational) where the intra-shear distance between upper and lower concordant structures is high. In areas where these structures converge, the sigmoidal veins are more concordant.

The sigmoidal veins were the target of historic underground mining. All accessible mine workings have been mapped in detail and the observations included in interpretation of the geological wire frame.

The GB02a geological interpretation defines the following:

The upper shear structure, which is interpreted loosely as a hangingwall style feature, has been used to constrain the top of the grade interpolation. It is typically thin (2 to 4m) and curvi-planar. The lower shear is a large continuous feature which is relatively predictable and contains approximately 75% of the mineralization. Overall the structure dips towards 050° at between 0° and 15° and is typically 6 to 12m thick.

Domain 1 accommodates a considerable range of orientations ±20° strike and ±15° dip and range of grades. The structures are generally narrow relative to the block height. The sample search has been restricted to within the wireframe (i.e. no sharing of the data during estimation) which results in low numbers of composites being available for estimation.

Between the upper and lower shears of Domain 1 are two sigmoidal shear structures which are characterised by 1 to 2m thick laminated quartz veins. Both of these sigmoidal vein structures have been combined into a single zone (Domain 2). These features dip to the north-west at 30° and are typically associated with high assays (>5 g/t).

Two zones of unconstrained stockwork were previously recognised in GB97b. Both have been combined to form Domain 3. Within Domain 3 it is possible to recognise sigmoidal vein style intercepts outside the constrained structures but these cannot be interpreted into a constrained structure at the current drill spacing.

The styles of mineralization recognised at Golden Bar are summarised in Table 17.65 .

Table 17.65: Golden Bar Deposit - Mineralization Styles

Mineralization Style	GB02a Resource Estimate
East Dipping Concordant Shears	1
Sigmoidal Vein structures	2
Unconstrained Stockwork	3

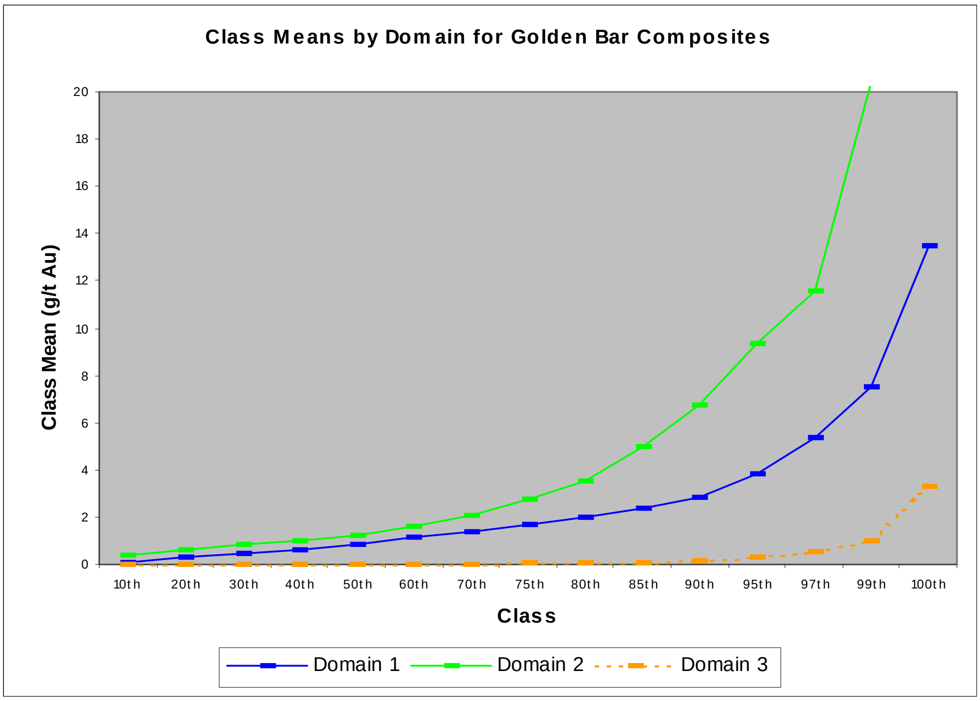
17.11.5 Statistical and Geostatistical Modelling

Statistical analysis and variography were based on the coded one metre composites. A summary of gold statistics by domain code is shown in Table 17.66 and Figure 17.37.

Table 17.66: Golden Bar Deposit - Summary of 1m Composite Statistics (Au g/t) for Domains

Domain :	1		2		3	
Rank	Threshold	Mean	Threshold	Mean	Threshold	Mean
10th	0.22	0.10	0.55	0.39	0.01	0.00
20th	0.41	0.30	0.72	0.64	0.01	0.01
30th	0.55	0.49	0.90	0.82	0.01	0.01
40th	0.68	0.62	1.14	1.02	0.02	0.01
50th	0.99	0.82	1.36	1.26	0.02	0.02
60th	1.27	1.13	1.84	1.64	0.03	0.02
70th	1.59	1.42	2.38	2.08	0.05	0.04
75th	1.86	1.71	2.98	2.72	0.06	0.06
80th	2.17	2.01	4.21	3.51	0.09	0.08
85th	2.55	2.35	5.50	5.00	0.14	0.11
90th	3.10	2.80	7.99	6.73	0.21	0.17
95th	4.72	3.85	10.30	9.33	0.40	0.29
97th	6.22	5.34	14.90	11.58	0.66	0.50
99th	9.48	7.51	21.55	20.38	1.44	1.00
100th	21.90	13.51	56.60	39.15	20.30	3.27
Number	**1,945**		**151**		**16,864**	
Mean	**1.51**		**3.25**		**0.11**	

Figure 17.37: Golden Bar Deposit - Graphical Summary of Class Statistics by Domain



Most of the populations are positively skewed (close to being log-normal), however, there is a significant truncation of the distribution at 0.5 g/t due to the imposition of wireframes.

Correlograms were modelled for both Domains 1 and 3. There was insufficient data to attempt Domain 2 variography. Consequently Domain 1 variography was applied to Domain 2 but rotated into the plane of Domain 2.

For Domains 1 and 3, correlograms were modelled for the 10th, 20th, 30th, 40th, 50th, 60th, 70th, 75th, 80th, 85th, 90th, 95th, 97.5th and 99th percentiles. Nested, exponential models were used. Indicator variogram parameters are shown in Table 17.67 to Table 17.69.

Table 17.67: Golden Bar Deposit - Domain 1, Modeled Variogram Parameters

Hangingwall Domain 1											
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**	**Rotation 2**
0.22	0.30	0.48	41	29	6.1	0.22	48	50	17.0	y 14	none
0.41	0.30	0.48	44	34	3.1	0.22	48	50	17.0	y 14	none
0.55	0.30	0.48	44	29	2.2	0.22	48	50	23.0	y 14	none
0.68	0.40	0.48	50	35	4.4	0.12	52	50	9.0	y 14	none
0.99	0.35	0.48	43	27	3.4	0.17	52	50	4.0	y 14	none
1.27	0.35	0.48	35	23	3.9	0.17	52	25	4.0	y 14	none
1.59	0.35	0.48	32	16	3.9	0.17	38	17	4.0	y 14	none
1.86	0.35	0.48	27	16	3.4	0.17	29	17	3.5	y 14	none
2.17	0.35	0.48	28	16	3.0	0.17	29	17	3.5	y 14	none
2.55	0.35	0.48	21	19	3.0	0.17	24	20	3.5	y 14	none
3.10	0.35	0.48	18	22	2.5	0.17	22	26	3.5	y 14	none
4.72	0.35	0.48	18	29	1.8	0.17	22	32	3.5	y 14	none
6.22	0.35	0.48	16	24	1.7	0.17	18	32	2.0	y 14	none
9.48	0.40	0.48	14	19	0.7	0.12	17	23	1.0	y 14	none

Table 17.68: Golden Bar Deposit - Domain 2, Applied Variogram Parameters

NW Dipping Lodes											
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**	**Rotation 2**
0.55	0.30	0.48	41	29	6.1	0.22	48	50	17.0	y -20	z 30
0.72	0.30	0.48	44	34	3.1	0.22	48	50	17.0	y -20	z 30
0.90	0.30	0.48	44	29	2.2	0.22	48	50	23.0	y -20	z 30
1.14	0.40	0.48	50	35	4.4	0.12	52	50	9.0	y -20	z 30
1.36	0.35	0.48	43	27	3.4	0.17	52	50	4.0	y -20	z 30
1.84	0.35	0.48	35	23	3.9	0.17	52	25	4.0	y -20	z 30
2.38	0.35	0.48	32	16	3.9	0.17	38	17	4.0	y -20	z 30
2.98	0.35	0.48	27	16	3.4	0.17	29	17	3.5	y -20	z 30
4.21	0.35	0.48	28	16	3.0	0.17	29	17	3.5	y -20	z 30
5.50	0.35	0.48	21	19	3.0	0.17	24	20	3.5	y -20	z 30
7.99	0.35	0.48	18	22	2.5	0.17	22	26	3.5	y -20	z 30
10.30	0.35	0.48	18	29	1.8	0.17	22	32	3.5	y -20	z 30
14.90	0.35	0.48	16	24	1.7	0.17	18	32	2.0	y -20	z 30
21.55	0.40	0.48	14	19	0.7	0.12	17	23	1.0	y -20	z 30

Table 17.69: Golden Bar Deposit - Domain 3, Modelled Variogram Parameters

Stockwork											
Threshold	**C0**	**C1**	**AX**	**AY**	**AZ**	**C2**	**AX**	**AY**	**AZ**	**Rotation 1**	**Rotation 2**
0.01	0.27	0.46	42	38	8.7	0.27	44	41	98.0	y 14	none
0.01	0.45	0.15	36	30	2.2	0.40	38	36	26.0	y 14	none
0.01	0.45	0.15	36	35	1.7	0.40	42	40	22.0	y 14	none
0.02	0.39	0.23	46	43	5.2	0.38	56	52	39.0	y 14	none
0.02	0.30	0.23	42	43	4.7	0.47	52	52	32.0	y 14	none
0.03	0.28	0.25	45	47	4.7	0.47	52	52	24.0	y 14	none
0.05	0.28	0.21	44	47	4.2	0.51	52	52	17.0	y 14	none
0.06	0.28	0.16	38	47	2.7	0.56	48	50	14.0	y 14	none
0.09	0.33	0.16	38	47	3.2	0.51	48	50	13.0	y 14	none
0.14	0.37	0.16	35	44	3.2	0.47	43	46	12.0	y 14	none
0.21	0.42	0.16	34	44	2.2	0.42	38	46	11.0	y 14	none
0.40	0.45	0.16	29	34	2.2	0.39	30	36	8.5	y 14	none
0.66	0.50	0.16	22	27	2.7	0.34	24	29	9.5	y 14	none
1.44	0.46	0.16	19	16	3.7	0.38	21	18	8.0	y 14	none

17.11.6 Block Model Limits

A resource estimate was built using *GS3* software with the block model parameters shown in Table 17.70. Bulk density was assigned the resource model based on oxidation. Oxide material was assigned a bulk density of 2.50 t/m^3 and fresh material was assigned a bulk density of 2.60 t/m^3.

Table 17.70: Golden Bar Deposit - Block Model Construction Parameters

	GB02a (November 2002)	
	Limits	**Block Size**
Easting	70,500 - 71,350	25 m
Northing	5,250 - 6,350	25 m
RL	310 - 565	2.5 m

17.11.7 Grade Estimation

Multiple Indicator Kriging has been used to produce the grade estimate at Golden Bar. The estimate has been completed in geostatistical software *GS3*.

The estimation sample search parameters applied are summarised in Table 17.71. The minimum number of 1m composites for kriging was set at 16 and the maximum number was 48. The search strategy was a simple elliptical search with no modifications. Domain control was used in estimation wherein only data coded as that domain was used in the estimation of that domain.

Table 17.71: Golden Bar Deposit - Sample Search Parameters

Domain	Minimum	Maximum	X Y Z Discretisation	X Y Z Search	Octant Constraint
1	16	48	5 x 5 x 2	50m x 50m x 8m	4 required
2	16	48	5 x 5 x 2	30m x 30m x 5m	4 required
3	16	48	5 x 5 x 2	37.5m x 37.5m x 6m	4 required

A block support adjustment was applied to the MIK estimate, as implemented in the *GS3* Modelling Software. The block support adjustment at Golden Bar was via a mixed normal log normal change of support to replicate a selective mining unit of 5mE x 5mN x 2.5mRL. Table 17.72 summarises the block adjustment factors.

Table 17.72: Golden Bar Deposit - Change of Support Variance Ratios (Gold)

Domain	Block Support	Information	Block Distribution
1	0.186	0.651	Norm + Logn
2	0.196	0.692	Norm + Logn
3	0.288	0.696	Norm + Logn

No detailed description of the change of support is provided, however it is assumed that a test on the conditional cumulative distribution function (ccdf) is completed to determine the presence of positive skew (or not) for the estimated node. If positive skew exists then an indirect lognormal correction is applied otherwise a normal correction is applied. The general approach is commonly used in mining projects elsewhere around the world and is considered acceptable.

17.11.8 Validation and Reconciliation

Oceana have validated the resource estimate in the following ways:

- The resource estimate was viewed on screen in MINESIGHT to check geological coding, classification coding, weathering state and rock densities. Block grades were compared to surrounding composite grades and the distribution of high grades particularly noted;

- 1:750 scale EW sections were plotted with factored composite gold grades, recoverable grades and proportions (at 0.5 cut-off), sliced GB02a interpretation and the current pit design;
- An external audit has been completed by Hellman and Schofield Pty Ltd; and
- Resource model to Mill reconciliation, as summarised below.

Mining commenced at Golden Bar in February 2004 and was completed in October 2005.

The reconciliation for the period February 2004 to December 31, 2005 is presented in Table 17.73 to Table 17.75 for oxide at a 0.5 g/t cut-off and sulphide at 0.7 g/t cut-off.

Table 17.73: Golden Bar Deposit - Reconciliation at a 0.5 g/t for Resource Estimate versus Mined Oxide

		Survey Adjusted Grade Control					Resource Estimate			Variance		
		Unfactored		Factored						Factored GC / Estimate		
Year	Estimate	Tonnes	g/t	Tonnes	g/t	oz	Tonnes	g/t	oz	Tonnes	g/t	oz
2004	GB02a	82,025	1.55	82,025	1.48	3,914	72,009	1.16	2,697	1.14	1.28	1.45
2005	GB02a	0	0.00	0	0.00	0	0	0.00	0	-	-	-
Total		**82,025**	**1.55**	**82,025**	**1.48**	**3,914**	**72,009**	**1.16**	**2,697**	**1.14**	**1.28**	**1.45**

Table 17.74: Golden Bar Deposit - Reconciliation at a 0.7 g/t for Resource Estimate versus Mined Sulphide

		Survey Adjusted Grade Control					Resource Estimate			Variance		
		Unfactored		Factored						Factored GC / Estimate		
Year	Estimate	Tonnes	g/t	Tonnes	g/t	oz	Tonnes	g/t	oz	Tonnes	g/t	oz
2004	GB02a	1,232,204	1.73	1,232,204	1.72	68,140	1,009,615	1.88	61,025	1.22	0.91	1.12
2005	GB02a	361,522	1.87	361,522	1.87	21,734	350,980	1.74	19,634	1.03	1.07	1.11
Total		**1,593,726**	**1.76**	**1,593,726**	**1.76**	**89,874**	**1,360,595**	**1.84**	**80,659**	**1.17**	**0.96**	**1.11**

Table 17.75: Golden Bar Deposit - Reconciliation Resource Estimate versus Mined Oxide at 0.5 g/t and Sulphide at 0.7 g/t

		Survey Adjusted Grade Control					Resource Estimate			Variance		
		Unfactored		Factored						Factored GC / Estimate		
Year	Estimate	Tonnes	g/t	Tonnes	g/t	oz	Tonnes	g/t	oz	Tonnes	g/t	oz
2004	GB02a	1,314,229	1.72	1,314,229	1.71	72,043	1,081,624	1.83	63,710	1.22	0.93	1.13
	GB02a	361,522	1.87	361,522	1.87	21,734	350,980	1.74	19,634	1.03	1.07	1.11
Total		**1,675,751**	**1.75**	**1,675,751**	**1.75**	**93,777**	**1,432,604**	**1.81**	**83,344**	**1.17**	**0.97**	**1.12**

The resource estimate under calls both the tonnage (17%) and metal (12%) reported in grade control and to the Mill. Overall, the grade is over called by approximately 3%. RSG Global reviewed this in the 2007 Report and considered this difference in grade and tonnages to be due to the resource model targeting a level of mining selectivity not achieved in practice. They recommended additional dilution and ore loss be included in the grade estimate through change of support and a review of the change of support correction applied to the MIK estimate. They also recommended further drilling and refinement to the model to increase drill data coverage in the stockwork zone such that an improved estimation of the resource tonnage can be made. There are no plans in the short term to mine Golden Bar, but Oceana would review the block support correction and drilling requirements if mining were to be considered.

17.11.9 Resource Reporting

The Golden Bar grade estimate has been categorised based on a combination of geological confidence, drill data density and a mining reconciliation. Less than 5% of the resource has been classified as Measured, and 36% as Indicated. The criteria applied to the resource classification is summarised below and at Table 17.66.

- At 25 x 25m drill centers, all constrained shear structures have been classified as indicated. Areas 50 x 50m drilling are classified as inferred;
- All sigmoidal vein structures have been classified as indicated. These are all drilled to approximately 25 x 25m; and
- For unconstrained stockwork to be classified as measured, it must be drilled to 25 x 25m drill spacing and have greater than 80% of the block above 0.5 g/t. To be classified as indicated, it must be drilled to 37.5 x 37.5m spacing and have greater than 30% of the block above 0.5 g/t. At drill spacing greater than 37.5 x 37.5m unconstrained stockwork is classified as inferred.

To achieve the resource classification for the constrained structures outlined above, two polygons were drawn around the areas of the 25 x 25m drilling and the areas of 50 x 50m drilling.

Table 17.76: Golden Bar Deposit - Resource Classification Methodology

Mineralization Style	Domain Code	Classification Criteria		
		Measured	Indicated	Inferred
Shears	1	none	<25 x 25 metre	>25 x 25 metre
Sigmoidal Veins	2	none	<25 x 25 metre	none
Unconstrained Stockwork	3	<25 x 25 metres, and >80% Block above 0.5 g/t Au	<37.5 x 37.5 metre, and >30% Block above 0.5 g/t Au	>37.5 x 37.5 metre

The reported Mineral Resource grouped by category, and reported above a lower cut-off grade of 0.5 g/t Au is shown in Table 17.77. After reviewing the estimate, input data, and reconciliation data, Oceana considers the reported resource to be reasonable.

Table 17.77: Golden Bar Deposit - Resource as at December 31, 2009

	0.5 g/t Cut-off						
	Fresh		Oxide		Total		
Category	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Contained Au (koz)
Measured	0.09	1.56	-	-	0.09	1.56	5
Indicated	1.18	1.4	-	-	1.18	1.40	53
Inferred	4.96	1.1	-	-	4.96	1.4	217

Note: Mt = million tonnes, koz = 000's contained ounces

17.12 Taylors

17.12.1 Introduction

The Taylors prospect is located approximately 12km south of the Macraes Gold Mine. Oceana generated a resource estimate for Taylors in February 2003 based on the available drilling data.

17.12.2 Database

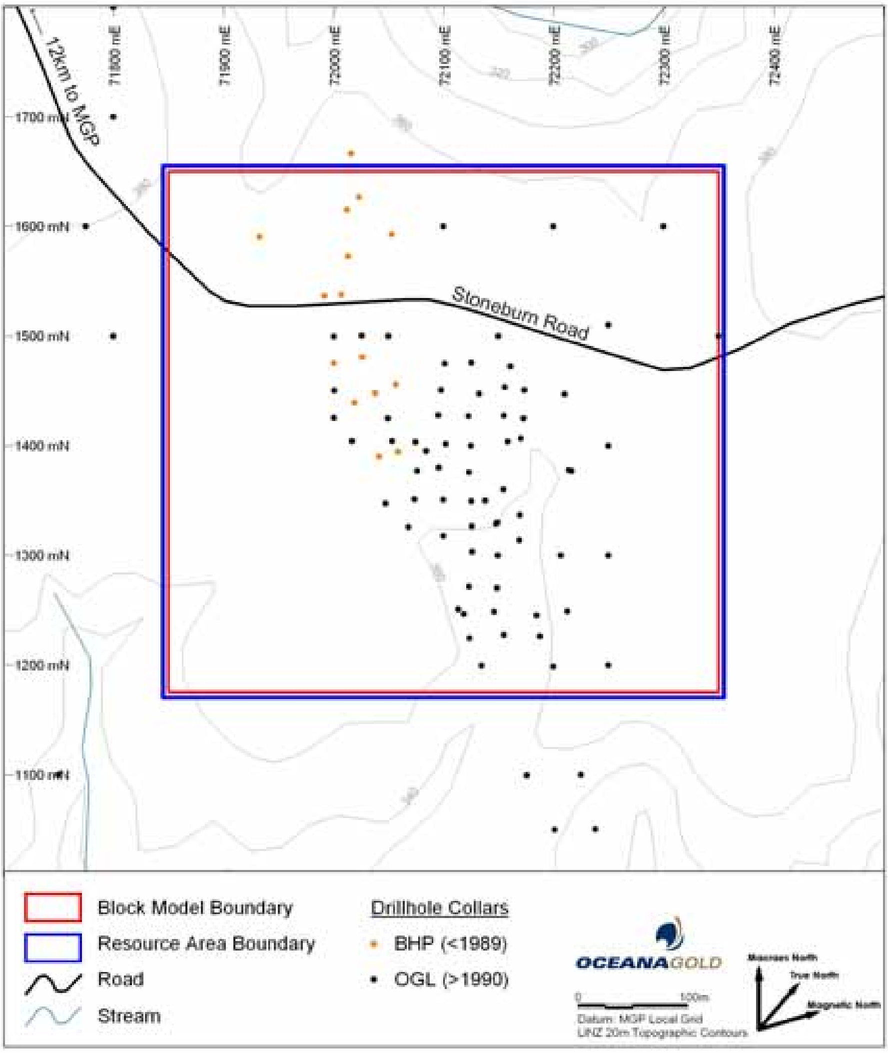
BHP (NZ) Ltd and Oceana have completed the drilling at Taylors. BHP (NZ) Ltd drilled 15 open hole percussion drill holes to test what was known as Home Reef. Oceana then followed up the BHP (NZ) Ltd drilling in 1996 with 79 RC percussion holes. The location of the drill holes is shown in Figure 17.38.

Table 17.78 gives a breakdown of the drilling used in the current resource estimate.

Table 17.78: Taylors Deposit - Resource Drilling Summary

Hole Type	TL03a Resource Estimate		
	Number	**Metres**	**Percentage**
Percussion	14	216	6
Reverse Circulation	68	3,263	94
Total	**82**	**3,479**	**100**

Figure 17.38: Taylors Deposit - Drill Hole Collar Plan



17.12.3 Geological Modelling

The HMSZ in the Stoneburn area consists of three shallow easterly dipping shears. The Taylors area occurs at the southern extension of the eastern most shear which known as the Home Reef. The Home Reef structure is interpreted to continue from Golden Bar.

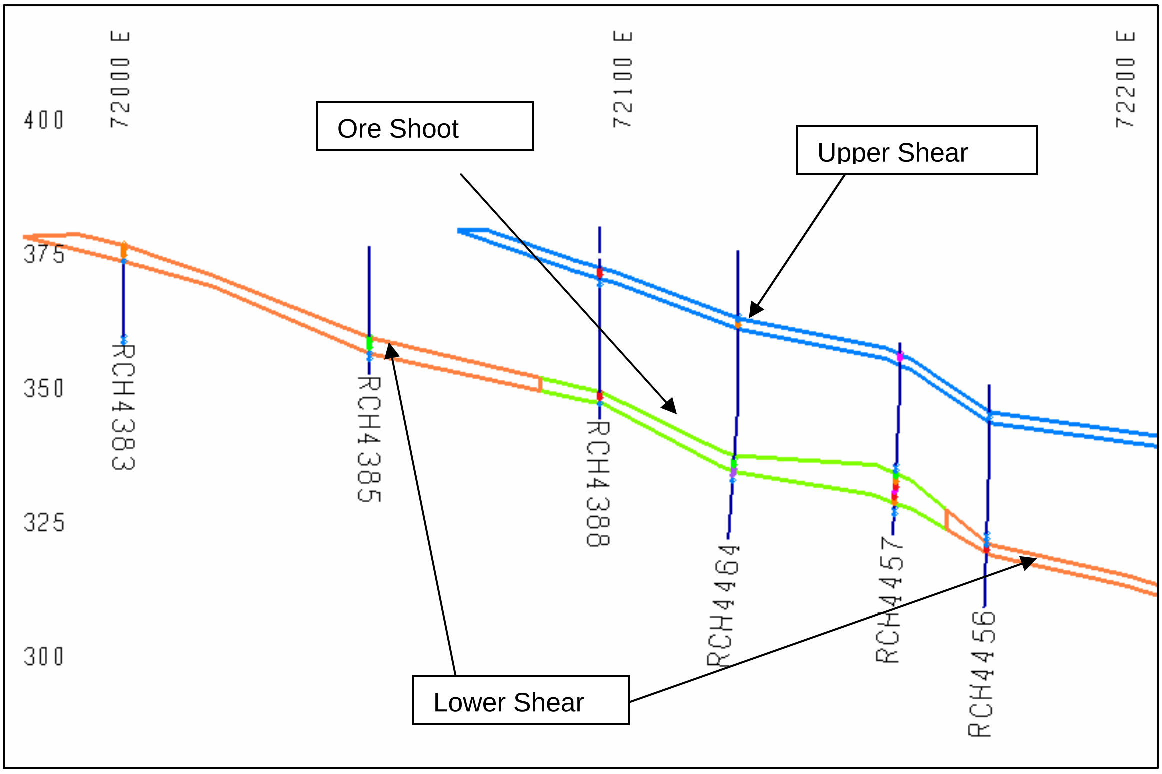
The Taylors deposit lies some 450m above the interpreted position of the HMSZ footwall shear. Between the Taylors deposit and footwall shear is a 450m thick zone of mineralized intra-shear pelite. Above the Taylors deposit, the schist is psammitic to semi psammitic.

No major faulting has been identified in the Taylors area. However, major E-W faulting occurs 2km north and 1km south of the Taylors area. There is some minor NW faulting in the area which appears to bound the anomalous zones of mineralization to the north and south.

Mineralization in the Taylors deposit is confined to two near-concordant mineralized structures (Figure 17.39). The upper shear is approximately 1m thick and is located 25-30m above the lower shear. There are no indications that these two shears are linked as at Golden Bar. The shear zones strike 350^{o} and dip $12\text{-}20^{o}$ to the east. Drilling has identified a mineralized ore shoot within the lower shear that plunges obliquely along this structure in a NNE direction.

The Taylors area contains a number of historic workings that were developed on the mineralization. To date, the extent and the production history of these workings remains unknown.

Figure 17.39: Taylors Deposit - East-West Geological Cross Section



Mineralization in the shear zone is dominated by near-concordant quart veining with minor pyrite and arsenopyrite. No stockwork has been identified below the lower shear zone. To the south, only the lower shear is evident which is typically <1m thick and weakly mineralized.

To date two mineralized shears have been interpreted at Taylors. The first is a poorly developed upper shear, which is approximately 2m thick. The second, a more strongly mineralized lower shear, has an average thickness of approximately 3m. Within the shear, a 200m long by 100m wide, north trending high grade shoot has been defined.

Both shears have been interpreted to dip at approximately 20° to the east and have been defined based on a 0.3 g/t cut-off, and 2m minimum mining width.

17.12.4 Statistical and Geostatistical Modelling

The drill hole samples were composited to 1m and geologically coded using the interpreted geological/mineralization solids. Due to few composite data being available, variography was not generated. The Frasers South variogram models were used in estimation as Oceana believed Frasers South to be a good geological analogue for the Taylors prospect.

Figure 17.40 and Table 17.79 present the sample and conditional statistics grouped by domain. Domain 2 captures the highest tenor mineralization with a 2.17 g/t Au mean grade calculated, significantly higher than 0.97 g/t Au and the 0.87 g/t Au returned for Domains 1 and 3 respectively. As with other deposits in the project area, the captured data distributions are positively skewed although no outlier composites requiring adjustment have been identified.

Figure 17.40: Taylors Deposit - Class Means by Domain

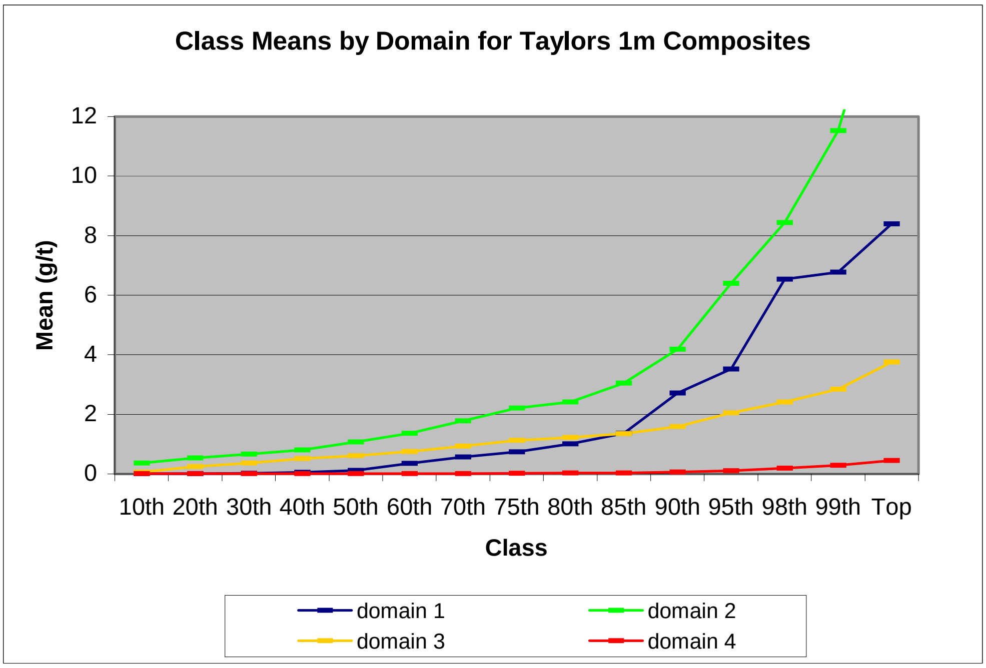


Table 17.79: Taylors Deposit – 1m Sample Statistics (g/t Au) by Domain

	Domain 1		Domain 2		Domain 3		Domain 4	
Class	**Threshold**	**Mean**	**Threshold**	**Mean**	**Threshold**	**Mean**	**Threshold**	**Mean**
10^{th}	0.09	0.01	0.09	0.36	0.10	0.06	0.10	0.01
20^{th}	0.19	0.01	0.19	0.53	0.20	0.25	0.20	0.01
30^{th}	0.29	0.02	0.30	0.66	0.29	0.36	0.30	0.01
40^{th}	0.40	0.06	0.40	0.80	0.40	0.51	0.40	0.01
50^{th}	0.50	0.12	0.50	1.07	0.50	0.61	0.50	0.01
60^{th}	0.59	0.36	0.59	1.36	0.60	0.75	0.60	0.01
70^{th}	0.69	0.57	0.69	1.79	0.70	0.94	0.70	0.02
75^{th}	0.74	0.74	0.75	2.21	0.75	1.12	0.75	0.02
80^{th}	0.79	1.01	0.80	2.41	0.80	1.23	0.80	0.03
85^{th}	0.85	1.37	0.85	3.05	0.85	1.36	0.85	0.04
90^{th}	0.90	2.72	0.90	4.18	0.90	1.59	0.90	0.06
95^{th}	0.95	3.52	0.95	6.40	0.95	2.05	0.95	0.11
98^{th}	0.97	6.54	0.97	8.44	0.97	2.42	0.98	0.20
99^{th}	0.98	6.77	0.99	11.53	0.99	2.84	0.99	0.29
Top	1.00	8.39	1.00	17.60	1.00	3.76	1.00	0.45
Number	**58**		**98**		**163**		**2752**	
Mean	**0.97**		**2.17**		**0.87**		**0.03**	

17.12.5 Block Model Limits

A resource estimate was built using *GS3* software with the block model parameters shown in Table 17.80. Bulk density was assigned to the resource model based on oxidation. Oxide material was assigned a bulk density of 2.50 t/m^3 and fresh material was assigned a bulk density of 2.60 t/m^3.

Table 17.80: Taylors Deposit – Block Model Limits

	Limits	Block Size
Easting	71,850 – 72,350	25 metres
Northing	1,175 – 1,650	25 metres
RL	235 – 390	2.5 metres

17.12.6 Grade Estimation

GS3 software has been used to produce the MIK grade estimate at Taylors.

The estimation sample search parameters applied are summarised in Table 17.71. A 50mE x 50mN x 4m search was used, rotated 18° down-dip, and 12° to the west of north. Domain control was used in estimation wherein only data coded as that domain was used in the estimation of that domain.

Table 17.81: Taylors Deposit - Sample Search Parameters

Domain	Minimum Number of Samples	Maximum Number of Samples	X Y Z Descretisation	X Y Z Search	Octant Constraint
1	8	48	5 x 5 x 2	50m x 50m x 4m	4 required
2	8	48	5 x 5 x 2	50m x 50m x 4m	4 required
3	8	48	5 x 5 x 2	50m x 50m x 4m	4 required
4	8	48	5 x 5 x 2	50m x 50m x 4m	4 required

A block support adjustment was applied to the MIK estimate, as implemented in the *GS3* Modelling Software. The block support adjustment at Taylors was via a mixed normal log normal change of support to replicate a selective mining unit of 5mE x 5mN x 2.5 mRL. Table 17.82 summarises the block adjustment factors.

Table 17.82: Taylors Deposit - Change of Support Variance Ratios (Gold)

Domain	Block Support Variance Ratio	Information Effect Ratio	Block Distribution Shape Model
1	**0.278**	**0.791**	**Norm + Logn**
2	**0.278**	**0.791**	**Norm + Logn**
3	**0.278**	**0.791**	**Norm + Logn**
4	**0.278**	**0.791**	**Norm + Logn**

No detailed description of the change of support is provided, however it is assumed that a test on the conditional cumulative distribution function (ccdf) is completed to determine the presence of positive skew (or not) for the estimated node. If positive skew exists then an indirect lognormal correction is applied otherwise a normal correction is applied. The general approach is commonly used in mining projects elsewhere around the world and is considered acceptable.

Visual and statistical validation has been completed by Oceana. Statistical checks have also been completed. Oceana believes the model to acceptable.

17.12.7 Classification

The Taylors deposit has been classified as a combination of Indicated and Inferred Mineral Resource by Oceana based on a combination of drill density and geological confidence. Mineralization hosted within the lower shear and with a minimum of 8 samples within the 50mN x 50mE x 4m search is classified as Indicated Mineral Resource. All shear hosted mineralization with a minimum of 4 samples within the 50mN x 50mE x 4m search is classified as Inferred Mineral Resource. The classification scheme applied is considered to be reasonable.

The Taylors Mineral Resource is tabulated below in Table 17.83.

Table 17.83: Taylors Deposit –Mineral Resource

	0.5 g/t Cut-off						
	Fresh		Oxide		Total		
Category	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Tonnes (Mt)	Grade (g/t)	Contained Au (koz)
Indicated	0.23	1.61	0.05	0.95	0.28	1.50	14
Inferred	0.31	1.05	0.09	1.07	0.41	1.05	14

The MIK grade estimates based on 25mE x 25mN x 2.5mRL panel and reporting a 5mE x 5mN x 2.5mRL Selective Mining Unit.

Table 17.84: Taylors Deposit - Mineral Resource as at December 31, 2009

	0.7 g/t Cut-off							0.5 g/t Cut-off						
	Sulphide		Oxide		Total			Sulphide		Oxide		Total		
Category	Mt	g/t	Mt	g/t	Mt	g/t	Koz	Mt	g/t	Mt	g/t	Mt	g/t	koz
Indicated	0.20	1.78	0.03	1.08	0.24	1/67	13	0.23	1.61	0.05	0.95	0.28	1.50	14
Inferred	0.24	1.2	0.07	1.2	0.31	1.2	12	0.31	1.1	0.09	1.1	0.41	1.1	14

17.13 Mineral Reserve Inventory

The Macraes reserve estimate represents that part of the Measured and Indicated resource which can be economically mined and for which the necessary design work and mine planning have been carried out. Proved and Probable reserve blocks are based on Measured and Indicated resource blocks respectively. Stockwork blocks categorised as Indicated, and falling within the pit outline are not necessarily included within the Probable reserve. To be included, the blocks have to show reasonable continuity of mineralization. Inferred blocks are considered to be inadequately defined and therefore are not included in reported reserves, although if they fall within the pit outlines they do represent potential additions to ore mined if confirmed by grade control drilling. The reserves are included within the overall resource figures.

Macraes open pit reserve tonnages and grades are based on Whittle 4X pit optimizations, using projected costs, slope angles based on geotechnical studies, plant recoveries and future gold prices. Progressively staged cutbacks are based on a NZ$1,333/oz (US$800/oz) gold price and an exchange rate of US$0.60/NZ$. An ad valorem royalty of 1% is payable to the New Zealand government and refining and handling charges are included at NZ$6/oz.

Reserve tonnages and grades are reported in accordance with CIM criteria to include any anticipated mining losses and mining dilution.

For open pit inventory, the resource block model estimation methodology incorporates adequate dilution and additionally depletes for the as-built FRUG openings and provides a reasonable estimate of mined tonnage and grades (see Table 17.88). No additional dilution or mining losses are applied during the Whittle 4X optimizations to the open pit inventory by Oceana to the block model inventory. Oceana considers that the block model estimation methodology already incorporates adequate dilution. Since 2007 the continuing reconciliation of data although variable, (see Table 17.88) confirms that overall the open pit reserves provide reasonable estimates of the mined tonnages and grades.

The following justification of dilution and ore loss factors for FRUG stope inventory were specified in the September 2005 Frasers Underground Technical Study. During 2009 a comparison of Cavity Measuring System (CMS) survey results verses blast designs was undertaken and a number of stope dilution and ore loss factors have been amended to reflect average actual performance. Where factors are not measureable the September 2005 Frasers Underground Technical Study factors still apply.

17.13.1 Frasers Underground Dilution

Dilution is the unwanted material that cannot avoid being mixed with the targeted ore and unless removed by some means reports to the processing facility thereby reducing the mined grade realised. Normally expressed as a percentage. In all discussions and calculations the formula should be taken to be:

- Dilution = unwanted material/planned stope material.

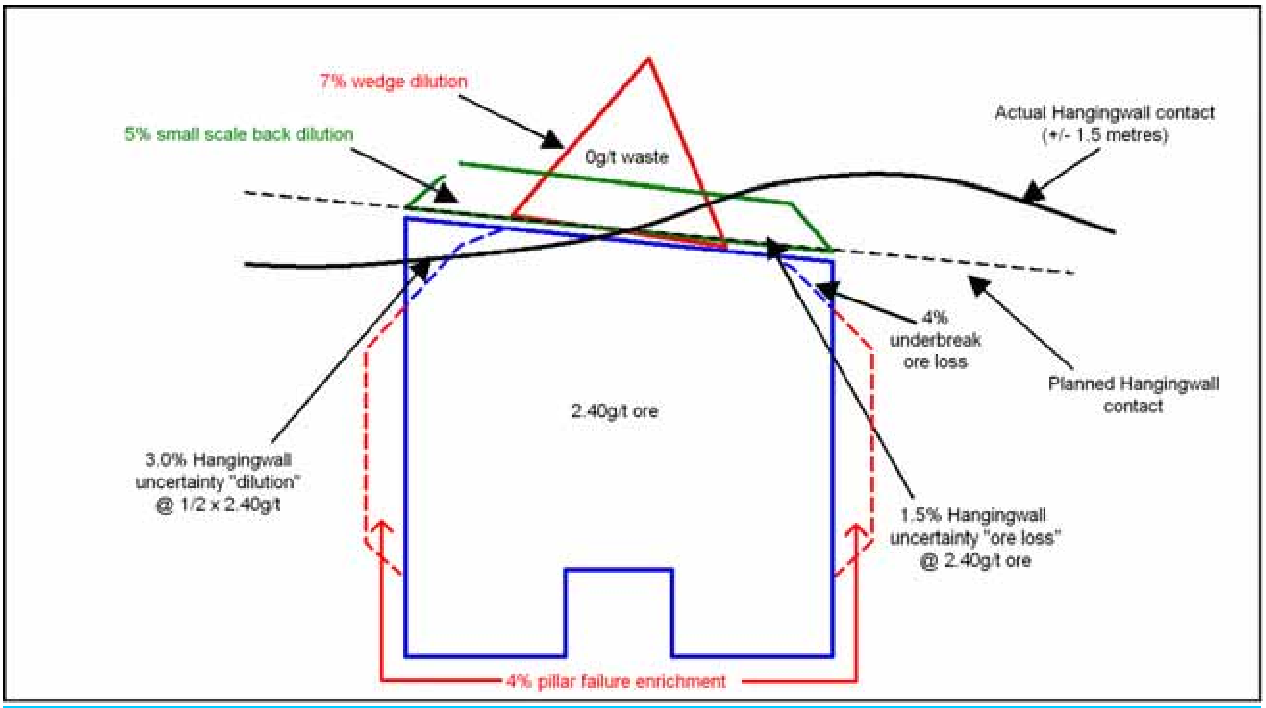
Dilution almost always carries some grade material and as such here will be expressed as a percentage with a grade. Coffey Mining Consultant's were requested to provide their estimates on stoping dilution given the rock mass conditions and mining methods planned. The following modes of dilution depicted in Figure 17.41 were assessed:

- Waste material inadvertently mined due to remaining Hangingwall position uncertainty;
- Blocky dilution from poor drill and blast practices;
- Wedge controlled dilution from the stope backs; and
- Ore enrichment or dilution from over-break on the yielding pillars.

Dilution factors amended since the 2009 Report as a result of the reconciliation work conducted during 2009 include:

- Wedge failure dilution from stope backs decreasing from 10% @ 0 g/t to 7% @ 0g/t.
- Addition of pillar enrichment not previously considered is set at 4% @ stope grade.

Figure 17.41: Schematic Depiction of Ore Loss and Dilution



17.13.2 Frasers Underground Ore Loss

Ore loss is the planned ore that is not recovered as a result of any number of means. Targeted stoping material is rarely fully realised over any medium to long term. Ore loss occurs from many sources, these are broadly divided into in-situ and broken ore losses.

Due to the low grade nature of the orebody, ore covered in excessive waste following extreme dilution events is considered to be lost rather than take the heavily diluted material. Ore loss is estimated to occur from the four following sources:

- From upper hangingwall position uncertainty;
- Broken ore lost through excessive waste cover;
- Sterilised ore stocks tied up in rib pillars required when re-slotting after major dilution, pillar failure events; and
- Broken ore not able to be effectively or efficiently cleaned up by the remote load-haul-dump unit (LHD).

Ore loss factors amended since the 2009 Report as a result of the reconciliation work conducted during 2009 include:

- Sterilised ore stocks through re-slotting decreased from 6.6% @ stope grade to 5.5% at stope grade;
- Addition of ore loss by under-break of stope back not previously considered is set at 4% at stope grade; and
- Broken ore not able to be cleaned up on stope edges, rough floors decreased from 5% @ diluted stope grade to 4.3% at diluted stope grade.

Currently 32% of the factored stope inventory will be accessed through long term infrastructure development, e.g. return air tunnels and decline access tunnels, and therefore those stopes extraction will be delayed until life of mine is drawing to a close or alternate infrastructure development is created. For those stopes identified as infrastructure delayed an additional 40% sterilised ore loss factor is initially applied to account for:

- Potential infrastructure tunnel damage due to the time delay between development and production that may prevent safe re-entry for economic stope production activities; and
- A potential requirement for more frequent re-slotting due to major dilution caused by a weakened rock state brought about by the regional stress redistribution associated with the extraction of the adjacent primary stopes and aggravated by the time delay.

Table 17.85: Stoping Ore – Dilution and Ore Loss First Principles Calculation

Mode	Ore Loss (%)	Dilution
Dilution		
Hangingwall uncertainty (from Oceana investigations) *3% Dilution (half of this @ average stope grade of 2.40 g/t)* *** Based on 25m x 25m drill density*		3% @ 1.20 g/t
Wedge failures (from Oceana investigations) *** Based on 2009 CMS verses design reconciliation task* *(Coffey's 2005 unwedge analysis recommendation initially 10%)*		7% @ 0.0 g/t
Blocky Dilution (from Oceana investigations) *** Based on 2009 CMS verses design reconciliation task* *(Coffey's 2005 of 5% @ 0 g/t recommendation of 0.5 to 1m back skin, over-break & blocky ground)*		5% @ 0.0 g/t
Pillar Failure Enrichment (from Oceana investigations) ***Based on 2009 CMS verses design reconciliation task*		4% @ 2.40 g/t
Total Dilution		**19% @ 0.69 g/t**
Ore Loss		
Hangingwall uncertainty (from Oceana investigations) ***The other 1.5% of ore loss is treated as grade dilution above*	*1.50*	

Mode	Ore Loss (%)	Dilution
Ore lost through excessive waste cover ***2,700 tonnes/100metres of stope drive*	*3.70*	
Sterilised tonnes through re-slotting (from Oceana investigations) *** Based on 2009 CMS verses design reconciliation task* *(Coffey's 2005 recommendation was initially 6.6%, based on 6.6m rib pillar left every 100m)*	*5.50*	
Under-break of back / roof (from Oceana investigations) *** Based on 2009 CMS verses design reconciliation task*	4.00	
Broken ore not able to be cleaned up on stope edges, rough floors (from Oceana investigations) *** Based on 2009 CMS verses design reconciliation task* ***(Coffey's 2005 recommendation was initially 5.00%)* ***For LOMP08, 150 tonne per firing, LOMP09, 130 tonne per firing*	4.30	
Total Ore loss (Stoping only)	**19.00**	
Global Calculations		
Starting with 100% of tonnes @ 2.40 g/t *In-situ ore-loss* = 11.0% (1.50% + 5.50% + 4.00%) = 89.0% @ 2.40 g/t *Adding dilution* = 19% @ 0.69 g/t = 108.0% @ 2.13 g/t *Finally broken ore-loss* = 8.0% (3.70% + 4.30%) = 97.5% @ 2.13 g/t		

- 19% dilution has been applied to all stoping tonnes from both the up-hole retreat mining method depicted in Figure 17.41 and the side wall only retreat mining method, e.g. Chevron Retreat method. The ability to better support the mining backs in the Chevron Retreat method, should allow for better control of the roof wedges and may lead to better conditions than with the retreat long hole open stope (RLHOS).

As is evident in Table 17.85., 19% of ore is considered lost through major dilution events and the need to re-slot stopes periodically. Considerations include broken ore stocks that may be covered by large volumes of waste material from back failures. This material can be uneconomic to continue extraction because:

- The unstable nature of the stope means that the load-haul-dump unit is exposed to a high risk of serious damage or loss;
- The waste material is too large to be moved; and
- The waste is of too fine a size to be separated from the ore and by its volume dilutes the stope material below economic cut-offs for haulage to surface and treatment.

The re-slotting consideration accounts for the need to re-establish stope stability after either a major stope back or yielding pillar collapse.

17.13.3 Macraes Reserves Inventory

A breakdown of open pit, stockpile and underground reserves is shown in Table 17.86. The reserves are reported by category using a 0.5 g/t open pit and stockpile cut-off and a 1.9 g/t cut-off for underground as at December 31, 2009. These reserves are a subset of the resources tabulated in Table 1.1.

Table 17.86: Macraes Mineral Reserve Inventory as at December 31, 2009

Reserve Cut-Off Grade	Reserve Area	Proven		Probable		Total Reserve (Proven and Probable)			Resource Model
		Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au Moz	
0.5 g/t	Coronation	.	.	0.81	1.35	0.81	1.35	0.03	CO01A
0.5 g/t	Frasers Pit	10.32	1.58	14.98	1.00	25.30	1.24	1.01	FR07A
0.5 g/t	Southern Pit[1]	.	.	5.93	1.27	5.93	1.27	0.24	RHFR09
1.90 g/t	Frasers Under-ground P1 ,P2 & P2D	0.40	3.23	1.18	2.78	1.58	2.89	0.15	P1_0806, P2_0912 & P2D_0906
0.5 g/t	Stockpiles	5.69	0.64	.	.	5.69	0.64	0.12	
	Macraes Total	**16.42**	**1.30**	**22.89**	**1.17**	**39.31**	**1.23**	**1.55**	

Based on a gold price of US$800/ounce (NZ$1,333/ounce)

[1] Note: The Southern Pit reserve is a subset of Innes Mills, Southern Pit and Round Hill resources as tabulated in Table 1.1

The reserves for the open pits include the ore identified in the Frasers, Southern Pit and Coronation deposits, giving a total of 32.04Mt at an average grade of 1.25 g/t Au. These are shown in Table 17.87.

Table 17.87: Macraes Open Pit Mineral Reserves Inventory, All Sources, 0.5 g/t Cut-off

Category	Total Reserves at at December 31, 2009		
	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)
Proven - Frasers	10.32	1.58	0.53
Total Proven	**10.32**	**1.58**	**0.53**
Probable - Frasers	14.98	1.00	0.48
Probable - Southern Pit	5.93	1.27	0.24
Probable - Coronation	0.81	1.35	0.03
Total Probable	**21.71**	**1.09**	**0.76**
Total Proven and Probable	**32.04**	**1.25**	**1.28**

The Frasers Open Pit is currently planned to extend to a depth of 330m, with an average remaining stripping ratio of around 9.5:1 over the life of mine to 2017.

The Southern Pit open pit is currently planned to extend to a depth of 160m, with an average remaining stripping ratio of around 13.6:1 over the life of mine to 2017.

The Coronation open pit is currently planned to extend to a depth of 50m, with an average remaining stripping ratio of around 11.9:1 over the life of mine to 2017.

The mineralization down dip of the Frasers Open Pit, which is included in the underground reserves, has not been included in the Macraes open pit reserves above. As shown in Table 17.86, the total FRUG reserves are separately stated as 1.58Mt at an average of 2.89 g/t Au.

17.14 Qualified Persons Responsible for Reserve Estimates

Comments are expressed by Mr Rod Redden regarding the updating of explanations and estimates in section 17.12.

17.15 Reconciliation

Reconciliation refers to the historical comparison of ore mined and processed with reserves depleted, to provide a level of confidence in the reserve estimates. Annual reconciliations of sulphide ore at a 0.7 g/t cut off at Macraes from 1996 to 2006 are shown in Table 17.88. From 2007 onwards, the reconciliations are shown in Table 17.89 at a 0.5 g/t cut-off to reflect the lowered in-pit cut-off grade.

Table 17.88: Macraes Reserve Reconciliations by Deposit and Year, 1996 to 2006 at a 0.7 g/t Cut-off

Year	Grade Control			Model			Reconciliation%		
0.7 g/t Cut-off	Mt	Au g/t	Kozs	Mt	Au g/t	Kozs	Tonnes	Grade	Ozs
1996	3.172	2.13	209	3.065	1.95	194	103	109	108
1997	3.048	1.65	162	3.238	1.64	170	094	101	95
1998	3.277	1.30	137	3.450	1.49	166	095	87	83
1999	2.350	1.59	120	2.265	1.69	123	104	94	97
2000	2.989	1.88	181	2.617	1.85	159	114	102	114
2001	3.768	1.48	179	3.341	1.51	162	113	98	111
2002	3.875	1.35	168	3.590	1.46	169	108	92	100
2003	3.713	1.46	174	3.423	1.53	169	108	95	103
2004	4.571	1.46	215	4.535	1.46	211	101	100	102
2005	4.705	1.29	195	4.640	1.24	186	101	104	105
2006	4.439	1.34	191	4.106	1.31	172	108	102	111
Total	**39.907**	**1.51**	**1931**	**38.270**	**1.53**	**1881**	**104**	**0.99**	**103**

Table 17.89: Macraes Reserve Reconciliations by Deposit and Year, 2007 to December 2009 at a 0.5 g/t Cut-off

Year	Grade Control			Model			Reconciliation%		
0.5 g/t Cut-off	Mt	Au g/t	Kozs	Mt	Au g/t	Kozs	Tonnes	Grade	Ozs
2007	3.16	1.15	117	2.82	1.18	107	1.12	0.97	1.09
2008	3.61	1.36	157	3.51	1.34	151	1.03	1.01	1.04
2009	4.06	1.41	185	3.59	1.56	180	1.13	0.90	1.02
Total	**10.83**	**1.32**	**459**	**9.92**	**1.37**	**438**	**1.09**	**0.96**	**1.05**

On average, over the last 14 years, the annualized tonnes, grade and ounces mined, based on survey adjusted tonnages and mill reconciled grades, have been mostly within ±5% of the block model reserve estimates. This is a satisfactory result. Results from individual deposits and individual years have been more variable, but are still within acceptable limits. Only rarely on an annual basis have the mined tonnes or grade been more than 10% below the reserve estimates. During the first half of 2009, the Frasers open pit realized grades from Frasers Stage 4B, 10 to 15% lower than predicted by the resource model. As mining activity moved from Stage 4B to 4C during the second half of 2009, the grade disparity disappeared. Ore mining is based on strict grade control procedures based on 2.5m bench samples collected from a 4.5m blast hole grid. Ore mining is mostly carried out on day shift under geological supervision and with ore spotters to minimize dilution.

In many areas, grade control drilling has identified “stockwork” mineralization. The term applies to a variety of mineralization styles (including sheeted quartz veining, stockwork and erratic shears) all of which show poor continuity at the resource drilling scale. In 2002 Oceana reviewed the resource data and estimation procedures, particularly in relation to “stockwork” and decided to adopt the H&S *GS3* modelling. The reconciliation results since then have improved with block model grade estimates in line with grade control results and overall contained gold giving a consistently favourable reconciliation.

The bulk of the poorer reconciliation results have related to the upper 50m of the pit and to relatively small quantities of ore. Below this depth reconciliation results were generally good. The shallower drilling generally relates to earlier drilling campaigns where quality control standards were lower, and estimates based on this data were typically found to be less reliable.

17.16 Environmental and Permitting Constraints to Mineral Estimates

The Macraes Project has been operational for a number of years now and is a key employer within the district. The mine site has a history of environmental compliance with no infringement notices issued.

The suite of consents and permits to operate the mine within the current mine plan are in place. No constraints are identified in terms of environmental or permitting matters for the current mine plan other than the operational constraints imposed by the requirement to comply with resource consent conditions.

18 OTHER RELEVANT DATA AND INFORMATION

18.1 Topography

The surface topography used for all the resource estimates with the exception of Coronation was a combination of 2.5m contour information derived from a detailed aerial survey completed in early 1994 by the Department of Survey and Land Information (DOSLI) on behalf of Oceana, surveyed drill hole collars and the 31 December 2005 end of month mine survey.

At Coronation the surface topography was derived from the 20m DOSLI contour data and drill hole collars. This data was modeled into 20m x 20m cells, using *TECHBASE* programme MINQ, then imported into *MINESIGHT*. This topography is very coarse and needs to be resurveyed at 2.5m contours prior to any mining.

18.2 Bulk Density

A bulk density of 2.5 t/m³ is assigned to oxide blocks and 2.6 t/m³ to sulphide (fresh) blocks. These are the accepted standard values for the Macraes Goldfield and have been applied everywhere to ensure consistency between resource estimation, grade control and mine planning. They are slightly lower than the experimentally determined SG's but are thought to more accurately reflect the bulk density of the overall rock mass. The experimental measurements are determined on small pieces of core, which do not include the joints, fractures, and faults present in the overall rock mass. Long-term reconciliation of truck volumes against milled tonnes has confirmed the validity of these bulk density values.

The specific gravity data for the various prospects is summarised in Table 18.1.

Table 18.1: Summary of Specific Gravity Data

	Oxide Ore		Oxide Waste		Sulphide Ore		Sulphide Waste	
Prospect	**No.**	**Mean**	**No.**	**Mean**	**No.**	**Mean**	**No.**	**Mean**
Deepdell	4	2.55	7	2.49	9	2.64	18	2.68
Golden Point	-	-	-	-	-	-	-	-
Round Hill	6	2.61	2	2.58	54	2.68	64	2.68
Southern Pit	-	-	-	-	4	2.67	3	2.66
Innes Mills	-	-	6	2.45	32	2.71	37	2.70
Frasers	2	2.32	10	2.47	62	2.69	73	2.67
Golden Ridge	-	-	-	-	-	-	-	-
Golden Bar	-	-	-	-	3	2.63	3	2.57
Total	**12**	**2.54**	**25**	**2.48**	**164**	**2.69**	**198**	**2.68**

19 INTERPRETATIONS AND CONCLUSIONS

19.1 Geology

The Macraes area is a mature exploration province and has been well tested by exploration over all but the northernmost end of the HMSZ. The best strike extension target is the Coronation prospect which is located approximately 5km north of the current process plant infrastructure. To the northwest of Coronation, unmineralized cover rocks that locally overlie the HMSZ geochemically obscure any exploration target. While this provides an opportunity, it will be challenging to explore. Significant resource potential exists down dip/plunge of known open pits. The 2010 Macraes exploration budget totals NZ$5.9M excluding the cost of the Underground exploration drive excavation.

The Oceana sampling procedures adopted for the drilling activities are considered appropriate and the programmes are well supervised by suitably qualified technical personnel.

After review of the database and available data quality, and considering the reconciliation data which is also available, the drilling data is considered to meet accepted industry standards. However, the quality control database is incomplete so complete assessment of all data quality is impossible. The later Oceana managed dry RC percussion drilling and diamond drilling is considered to represent the highest quality drilling as quality control records are available for review. Earlier open hole percussion drilling and cross over RC percussion drilling is considered to be of lower confidence given the opportunity for down-hole contamination which is inherent in these drill methods. The open hole percussion drilling methods are particularly at risk for this contamination. While this is the case, the majority of the drill hole database is considered to represent high quality data.

The wet RC percussion drilling that remains in the exploration/resource database represents a significant risk. The factoring approach applied by Oceana to reduce the impact of the remaining wet RC percussion drilling is reasonable although it does not account for local variability and down-hole contamination (i.e. artificially extended ore zone widths.) At Frasers, the wet RC percussion drilling impacts the resource estimates most significantly at depth (Stage 4 and 4C of the open pits).

Available reconciliation data indicates the resource models represent robust estimates of metal and are generally acceptable estimators of tonnage and grade. In the case of Deepdell, the resource model has been overly selective resulting in an undercall of tonnage and an overcall of grade. Note that this area is not included in the mining plan.

The Mineral Resource statement determined as at December 31, 2009 has been prepared and reported in accordance with Canadian National Instrument 43-101, ‘Standards of Disclosure for Mineral Projects’ of December 2005 (the Instrument) and the classifications adopted by CIM Council in August 2000. Furthermore, estimation and classification is consistent with the Australasian Code for the ‘Reporting of Identified Mineral Resources and Ore Reserves’ of December 2004 (the Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

19.2 Mining

Oceana’s performance at Macraes has shown that the mining equipment and mining methods are suited to the required mining rates and deposit geometry. Open pit and underground mine design procedures are appropriate and have been conducted in accordance with established industry standards and with input from appropriately qualified geotechnical specialists, hydrological specialists and external consultants. Historical productivity and safety records are generally in line with industry standards. The LOMP09 has been prepared from 2010-2015 and subsequently had Southern Pit reserves included; the 2010-2017 schedules rely only on reserves, and are considered appropriate and reasonable.

The LOMP09 schedule has factors applied to account for poor weather, public holidays, equipment availability, equipment utilization, historically justified limitations on mine production and historically justified limitations on mill throughput.

19.3 Processing

Over the last nineteen years Oceana has developed considerable experience in development and operation of the complex ore processing technology required to optimise gold recovery from the Macraes refractory ores.

Emphasis is placed on the control of costs. The relatively high tonnage processed, the simple flotation reagent regime and economies resulting from concentration of the gold into a flotation product comprising between 1.5% and 3% of the ore mass treated reduce operating cost. Labour costs are also lower than in most comparable developed countries. The lower operating cost of the core sulphide process is also due to low comminution costs (contributed to by the coarse grind, and relatively soft ore).

Plant utilisation has been maintained at about 95% which is at the high end of typical industry benchmarks. Gold recovery on open pit ore and underground combined, in 2009 averaged 79.6% a slight decrease on the previous 12 months. Overall, recoveries are considered reasonable given the refractory nature of the ores.

It should be noted that the treatment of sulphide concentrate from the Reefton operation at Macraes utilises spare capacity in the autoclave circuit. In some years excess concentrate will be produced. Oceana will bypass some low-preg robbing Macraes concentrate around the autoclave, feeding it directly to the CIL plant to enable all the highly refractory Reefton material to be oxidised.

19.4 Infrastructure, Environment and Tenement Status

Oceana continues to maintain appropriate infrastructure at Macraes, including road access, power, water supplies and administration facilities.

Environmental management and mitigation measures are maintained at Macraes, including ongoing monitoring to ensure compliance with resource consent conditions and permit requirements. These consents and permits are issued by the MOED, the ORC and the WDC. Tailings and waste rock disposal facilities are maintained and managed on an ongoing basis. Progressive rehabilitation is ongoing.

Consents are in place for additional uplifts to be constructed on tailings storage facilities (Mixed Tailings Impoundment and Southern Pit 11). There is sufficient tailings storage capacity in the present facilities to store tailings until 2014. Permitting of an additional storage facility (Back-road) is scheduled for the first half of 2010.

The project reserves, plant site, tailings dams and waste dumps are located on land that is covered by mining permits, and which Oceana owns or has access to mine with the sole exception of the Coronation deposit, where land access is under negotiation. All material tenements and landholder agreements are in good standing and have been independently reviewed. There is sufficient consented tailings capacity to hold all but circa 8Mt of LOMP09 tailings in the current tailings impoundments. Further tailings storage is needed in June 2014. A new tailings facility is currently being permitted. This permitting is 80% complete. Significantly increased pumping requirements to and from a new tailings facility may find the spare capacity of electrical power supply a limiting factor.

There are no material compliance issues relating to the principal mining and processing operations. Oceana has consents for the creation of a Heritage and Art Park at the Macraes site as part of its mine closure and restoration strategy. Implementation of this Heritage and Art Park is now well underway, with various artworks completed or under consideration. Oceana is in partnership with Fish and Game, a semi-government organisation, to manage a Trout Hatchery on the Macraes mine site.

A draft closure and rehabilitation plan has been prepared and is being reviewed. Oceana intends to incorporate the closure plan into future LOM plans. Estimated costs for final closure may require review. As more of the Heritage and Art Park is developed a better knowledge of costs involved will be available.

19.5 Production

Oceana has prepared production plans based on LOMP09 with subsequent inclusion of Southern Pit from reserves only which cover 2010-2017 for Macraes. The production rates forecast are consistent with recent performance and the anticipated grades. The mine production plans are considered reasonable for the purpose of long term scheduling.

During the 2010-2017 peak production years the open pit excavator fleet is planned to comprise two Caterpillar 5130's (being replaced with similar units in 2011), one Caterpillar 5230 (replaced in 2011) and one Hitachi EX3600, to load six Caterpillar 785 haul trucks and eleven (increasing to thirteen in late 2010) Caterpillar 789 haul trucks. Oceana is satisfied that there are sufficient working areas for the excavators to operate and there is reasonable opportunity to reassess the requirements.

During the 2010-2012 peak production years the underground operation in accordance with LOMP09 is planned to provide approximately ~15% of the Macraes ore using a fleet of three Tamrock H205D electric-hydraulic jumbos, one Caterpillar 2900, one Caterpillar 1700 and two Tamrock 1400 LHD's in conjunction with five Tamrock 50D haul trucks. The underground ore is dumped at an in-pit stockpile for periodic rehandling by the open pit fleet to the process plant's run of mine stockpile. Planned production for 2010 to late 2011 is primarily stope ore with additional development ore when encountered within the mine design. LOMP09 has production during 2012, being solely derived from stope extraction. Oceana is satisfied with the accuracy of the dilution factors, ore loss factors and constraints placed upon the LOMP09 schedules and the 2009 underground life of mine schedule is considered reasonable for the purpose of long term scheduling.

The projected plant throughput fluctuates between 5.4Mt and 5.6Mt for 2010 to 2016. 2017 sees the final Southern pit material being treated in the first half of the year.

19.6 Management

The owner operator open pit mine and the alliance agreement underground mine are, performing to expectation under Oceana's management.

Oceana management has sought out new opportunities for cost reduction and increased efficiency. The mining and processing operations have concentrated on minimising production costs to maintain profitability. Oceana continues to pursue cost reduction innovations.

The general management approach is strongly safety oriented and the safety performance statistics generally reflect that attention, with performances in line with industry averages.

19.7 Capital and Operating Costs

Capital cost estimation and forecasting appear reasonable and consistent with proposed development program and ongoing requirements. In practice, capital expenditures over the LOM period may be more variable than forecast due to unforeseen problems, modifications, upgrades and introduction of new technology.

Capital expenditure provisions (2010 to 2017) include expenditures for capitalised mining costs totalling NZ$236M and sustaining capital of NZ$66M (excluding exploration) and are considered accurate to within ±15%.

Mine capital expenditure (excluding pre-strip) has been based on known expenditures required for 2009 and includes replacement of some equipment (Open Pit mining fleet) in 2010 and 2011.

The process plant capital expenditure consists of known expenditure for replacement mill parts and modifications. Included are the tailings embankment expenditures on both the mixed tails dam and SP11, along with the capital construction cost for the new facility in 2013.

Plant operating cost estimates for Macraes are considered reasonable and consistent with recent experience and trends, and are regarded as accurate to ±15%. However, these costs are considered to be subject to volatility in commodities prices particularly power, reagents and steel (grinding media).

With the mine operating cost estimates Oceana is very focused on cost efficiency in all phases of the operation. Business improvements strategies are implemented to help control costs and strive for continued improvements. The mine operating cost estimates for Macraes are also considered reasonable and consistent with recent experience and trends, and are regarded as accurate to ±15%.

19.8 Environment

The Macraes Project is consented for environmental purposes, with actual and potential environmental effects regularly monitored and reported to the relevant agencies.

The site is achieving environmental compliance with good internal and external reporting of environmental issues and performance. The site environmental documentation is appropriate and follows EMS principles although it does not constitute a full EMS.

Overall, no material environmental issues have been identified which might limit the ongoing operation of the mine within LOMP09.

20 RECOMMENDATIONS

20.1 Geology

Two deposits, Deepdell and Golden Bar, would require reinterpretation of the oxidation surfaces and extra allowances for dilution and ore loss, should cut-backs be considered. No cut-backs are planned for these deposits at this stage.

20.2 Mining

Oceana reconciled stope CMS void survey data against design during 2009 and resultant dilution and ore loss factors have been amended where possible to answer the BDA recommendation in the 2007 report relating to clarification of FRUG ore loss and dilution factors. Where factors have been unable to be measured the stated factors within the September 2005 Frasers Underground Technical Study still apply.

20.3 Infrastructure, Environment and Tenement Status

In May 2007, GHD recommended that Oceana:

- Determine the cause of tailings seepage to Sump B within Maori Tommy Gully, and monitor the preferential pathway and contaminant levels. In response to this Oceana reviewed the role and operation of Sump B. Sump B replaced a smaller stand pipe collector system in 1994. The drains to the original collector system were laid in gravelled channel. This channel carried a small volume of seepage which bypassed Sump B. This volume is not considered significant by Oceana; and

- Finalise an EMS. Oceana is progressing with this but the report is not yet complete. Documentation for the key components is in place.

Oceana intends to:

- Continue the current monitoring and reporting regime, and expand to cover additions to the project;

- Refine the mine closure plan and cost estimates; and

- Continue to refine documentation along the principles of an EMS.

21 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

This section is an amended summary of the 2009 Report.

21.1 Mining Operations

Macraes is mined by a combination of conventional open pit and underground retreat long-hole open stope (RLHOS) methods along the line of strike.

21.1.1 Open Pit

The Macraes open pit mining operation is centred on the Macraes line of lode. Since open pit mining commenced at the Macraes project, a series of conventional open pits have been developed along the northwest trending (mine grid north) HMSZ. Gold mineralisation is variably distributed along the hangingwall shear as well as along a number of erratic, concordant shears, located below the hangingwall shear. Zones of sheeted, and variously orientated, quartz veins within the intrashear schist also carry some mineralisation. The footwall schists are typically barren.

The hangingwall shear dips at 15-20° to the east (mine grid east) with a mineralized width of 5-30m. The structure has been mined over a strike length of 6km and, in the Frasers area, is known to be mineralized to depths in excess of 500m.

The open pit operation is owner-operated. Ore mining at Macraes has come from ten pits, comprising, from north to south, Deepdell North, Deepdell South, Northwest Pit, Golden Point, Round Hill, Southern Pit, Innes Mills, Frasers North and South, Golden Ridge and Golden Bar. Current operations are in Frasers North Stage 4C (FN4C) and Frasers North Stage 5. Southern Pit (SP), Coronation and Frasers North Stage 6 (FN6) are still to be mined.

The conventional open pit mining operation at Macraes, utilises two Montabert drills for ore grade control and blast holes, one Drilltech drill for overburden blast holes and various ancillary units in addition to the four hydraulic excavators and seventeen rear-dump diesel trucks to remove both overburden and ore. Contracts for fuel and other commodities and services support the open pit operations. Blasting requires relatively light powder factors compared with other operations due to the relatively weak and fractured rock mass. Ore is blasted in 7.5m high benches and excavated in three, nominally 2.5m high flitches. Waste is blasted in 15m benches and excavated in four flitches.

The LOMP09 has factors applied to account for poor weather, public holidays, equipment availability, equipment utilisation, historically justified limitations on mine production and historically justified limitations on mill throughput.

The Frasers deposit provides the bulk of future reserves under the LOMP09. Oceana plans to develop the Coronation deposit to the north once an access agreement with the land owners has been signed. In December 2009, FN6 and SP were added respectively to the Macraes reserves after the pits were optimised at NZ$1,333/oz. SP was last mined in 2001 and was converted to a Tailings Storage Facility (TSF) in 2003. SP would be mined by conventional Macraes mining equipment with the bulk tails being mined by a combination of dozer, truck and excavator fleets. SP tails would be hauled to the Mixed Tails Dam and rock waste will be hauled to the Backroad waste dump.

21.1.2 Pit Optimisation

Whittle Four-X was used to perform pit optimisations using the Lersch-Grossman algorithm. Macraes pits were optimised at NZ$1333/oz in April 2009 for Frasers pits and December 2009 for Southern Pit. All Inferred material was excluded from processing during optimisation and was treated as waste.

Optimisations were based on the following inputs that were provided by Ocean Corporate office and Oceana Exploration and Development department:

- gold price;
- mining costs;

- processing costs;
- discount rates and royalties;
- general and administration costs;
- mining recovery and dilution;
- mining and process throughput rates; and
- pit slope angles.

The parameters are peer reviewed as part of the optimisation process.

21.1.2.1 Frasers Pit Optimisation

21.1.2.1.1 Resource Models

Two resource models, Frasers (FR05), and Coronation (CO01A) cover the planned future mining areas. Of these, Frasers contains the vast majority of both ore and total material.

Resource models are then exported and modified to create Four-X optimisation models, which contain relevant mining information.

Only Measured and Indicated Resource blocks are included in the optimisation models. Lower confidence Inferred blocks are treated as waste. In areas of active mining the Resource models were cut to 31 March 2009 topographical surface.

21.1.2.1.2 Mining Costs

The mining costs used for pit optimisations are derived from the mining 2008 LOM budget cost model (LOMM08rev16_Scenario_Oct24), which was completed in November 2008. All cost areas are updated to reflect the current costs at the time of optimisation. The haulage strategy is uniform over the LOM plan and is not modeled using stage specific costs.

Hauling cost is the major variable cost area, increasing with pit depth due to longer cycles and inclined hauls. Haul cost were modeled with a combination of actual and theoretical data. Drill and blast costs are modeled with costs for bulk waste blasting and ore zone blasting. Drill and blast costs are excluded from fill material.

21.1.2.1.3 Processing

Two types of ore are processed at Macraes. The bulk is sulphide ore (99%) and some oxide ore (1%). Given the insignificant amount of oxide ore, this is assumed to have the same processing costs as sulphide ore.

The simplified process plant configuration is:

Crushing → Milling → Flotation→ Regrinding → Pressure Oxidation (autoclave) → (Carbon in) Leaching

The process plant throughput assumed in optimisations is 5.4Mtpa. This is the current throughput capacity of the processing plant.

The current base milling costs (includes milling and feeding the crusher) at the nominated throughput rates is summarised in Table 21.1. This cost is derived from the LOMP08 model.

Table 21.1: Milling Costs used in 2009 Optimisations

Activity	Total (NZ$/t)
Processing Cost	10.81
Feed Crusher	0.34
Milling Cost per Tonne	**11.15**

21.1.2.1.4 Overhead Costs

The overhead costs used in optimisation are summarised in Table 21.2. Most of the overhead costs were from the current 2009 Forecast costs at the time.

Table 21.2: Overhead Costs used in Optimisation

Mill Feed Rate	Total Mining Rate	Overhead Rate per Tonne Milled	Overhead Rate per Tonne Mined
5.4Mtpa	50Mtpa	NZ$5.39	NZ$0.20

Overhead costs are detailed in Table 21.3 below.

Table 21.3: Derivation of Time Cost used for Pit Optimisation

TIME COST SUMMARY		Proportion of Cost to every tonne			
Overhead Classification	**Cost estimate pa**	**milled**	**mined**		
Mining Department	$14,383,657	76%	24%	$2.02/ t milled	$0.07/t mined
General Manager	$328,100	50%	50%	$0.03/ t milled	$0.00/t mined
Administration	$10,611,000	80%	20%	$1.57/ t milled	$0.04/t mined
Commercial & Royalties	$1,995,250	100%		$0.37/ t milled	
HR + Safety + Environment	$2,343,500	80%	20%	$0.35/ t milled	$0.01/t mined
Sustainable Development	$1,769,000	80%	20%	$0.26/ t milled	$0.01/t mined
Macraes Cat & ANZ Finance Lease Interest	$2,539,667	10%	90%	$0.05/ t milled	$0.05/t mined
Sustaining Capital	$4,897,855	81%	19%	$0.74/ t milled	$0.02/t mined
TOTAL TIME COSTS	**$38,868,029**	75%	25%	**$5.39/ t milled**	**$0.20/t mined**
Expected yearly mill throughput	5,400,000 tpa				
Expected yearly mining capacity	50,000,000 tpa				
Time costs per tonnes milled	**$5.39/t milled**			$29,101,088	
Time costs per tonne mined	**$0.20/t mined**			$9,766,941	
				$38,868,029	
PROCESSING COSTS					
Mill				$10.81 / t milled	
Feeding Crusher				$0.34/ t milled	
Total Mill				$11.15 t milled	
FT Dam Construction				$0.57/ t milled	
Mill cost per tonne				$11.72 t milled	

Derivation of Time Cost used for Whittle Four-X analysis

21.1.2.1.5 Metallurgical Recovery

Process plant recoveries were split into basic geology type and geographical area as per Table 21.4. The Recoveries were based on historical results.

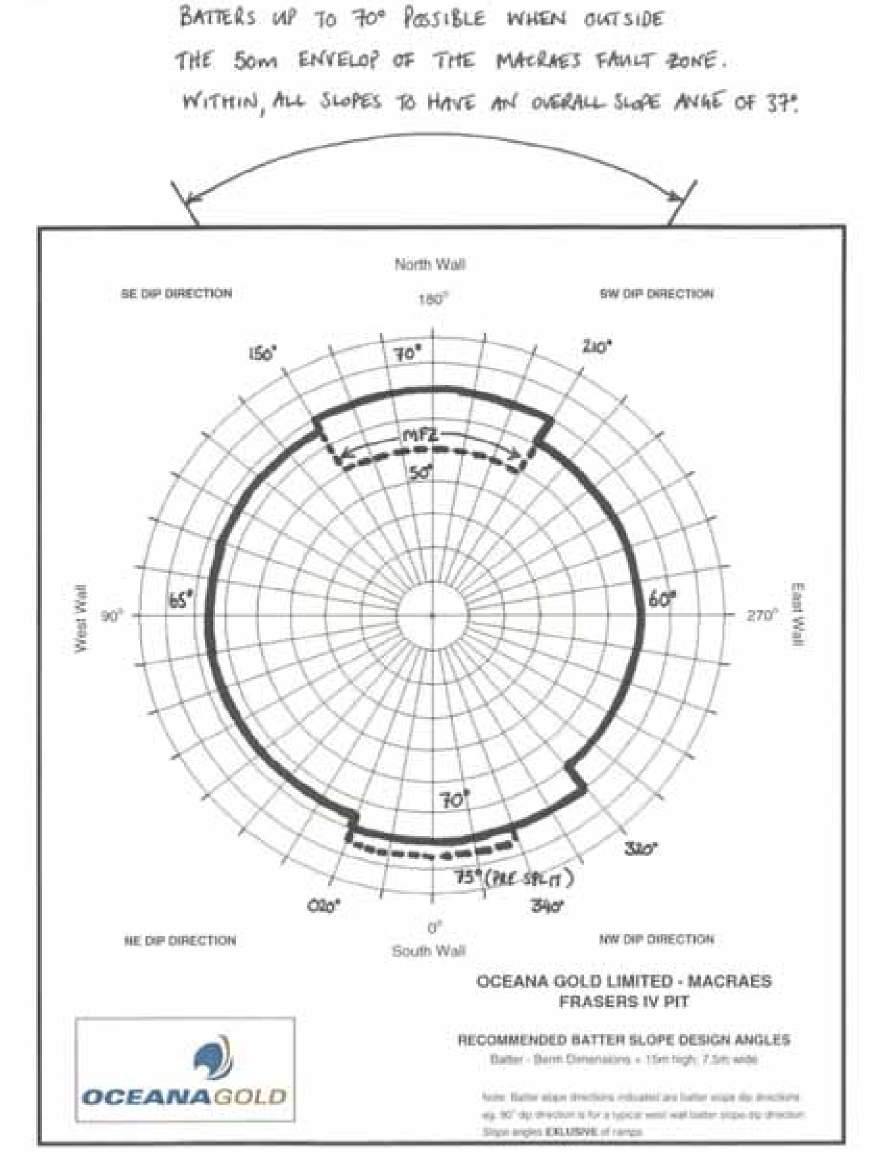
Table 21.4: Gold Metal Recoveries used in Optimisation

Rock Type	Combined Flot/CIL Recovery (%)
Oxide (default)	75.0
Sulphide (default)	82.0
Frasers North Hanging Wall	82.0
Frasers North Stockwork	82.0

21.1.2.1.6 Slope Design

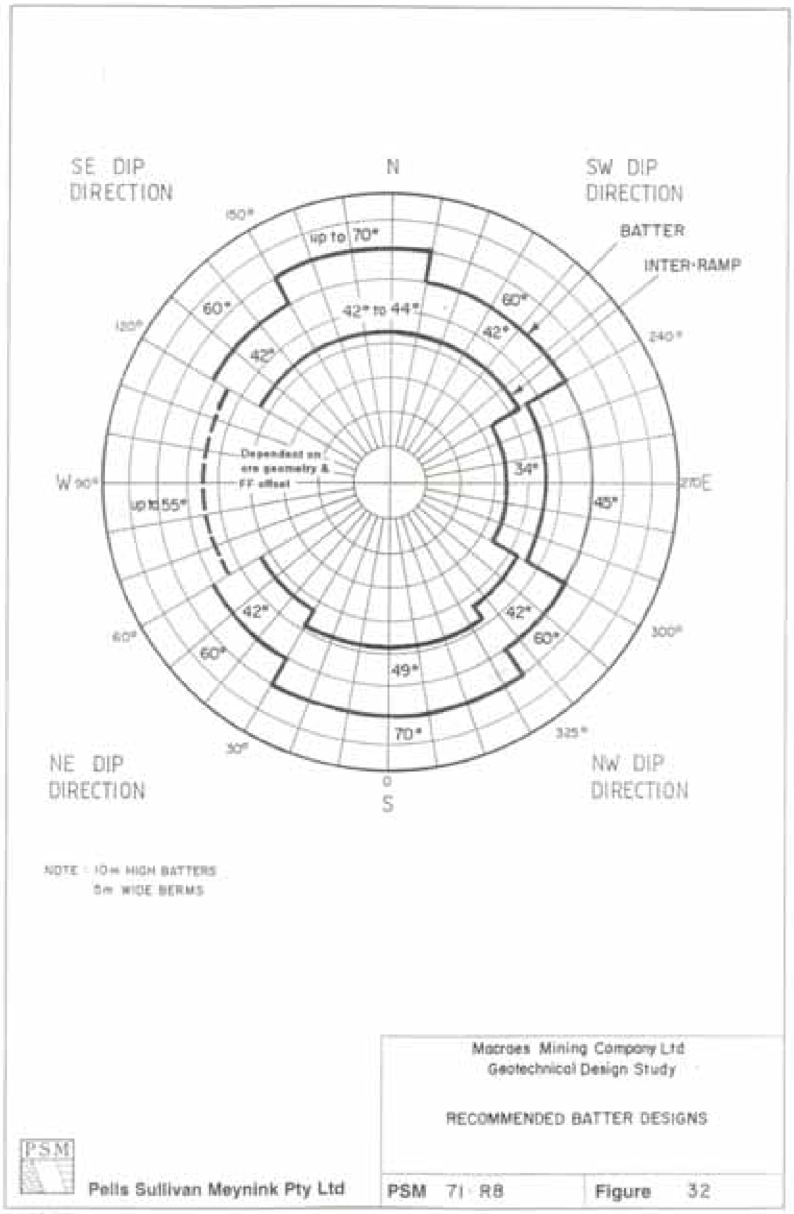
The rosette shown in Figure 21.1 was used for the Frasers deposit. Lower slope angles (37deg) were used within the Macraes Fault Zone. Areas where ramps are expected have lower slope angles to accommodate ramp positions.

Figure 21.1: Slope Rosette - Frasers Pit



The rosette shown in Figure 21.2 was used where there was no previous mining activity. Once further geotechnical investigations have been undertaken, these slopes generally would be steepened.

Figure 21.2: Default Slope Rosette



21.1.2.1.7 Mining Dilution

Selective ore mining procedures are used. This is done to maximise ore recovery and minimise mining dilution. During the ore definition process, grade control blast hole assays are used as the input data to a conditional simulation grade control process. The results of bench grade estimates are then used in conjunction with detailed geological mapping to produce mining blocks. Ore mining is supervised by geologists and ore spotters. Definition of the ore waste contacts is often done by trenching with small excavators. Dilution is accounted for in the resource model calculations by adding a waste veneer to the hanging wall contact, and using dilution estimation during the kriging process. The result is a dilution/recovery factor of close to 2%, which realistic taking into account the control techniques outlined above.

21.1.2.1.8 Gold Price

Optimisations shells were generated for a base gold price of NZ$1,333 per ounce. Gold price was analysed for price variance within a range of +/- 10%.

Sensitivity analyses were performed to investigate the effects of changes in mining cost, processing cost, gold price, recovery, and mining rate. Results confirmed previous work at Frasers (and indeed for all of the Macraes deposits) showing that the project is most sensitive to factors affecting income, namely: gold price, gold grade and recoveries. The southeast section of the Frasers deposit was found to have a very firm boundary as price increases indicating that future cutbacks towards the south are unlikely even at very high gold prices.

21.1.2.1.9 Pit Shell Selection

The selection of a contour (pit shell) around which an ultimate pit design will be made is an important step in the Ore Reserve estimation process. The process is based on the optimisation of resource models followed by sensitivity analysis on key inputs that impact on the "value" of a block in a model.

The final Frasers pit shell was selected by simulating a life of mine schedule within Whittle 4X for all of the existing pits. This was done using the 'Milawa' scheduling options within 4X. The key assumptions were:

- mill throughput at 5.4Mtpa;
- maximum mining rate of 50Mtpa (excluding stockpile movements);
- final Frasers shells examined were 30, 33, 34, and 36. The undiscounted cashflow becomes very flat over the range of shells 31-41;
- average processing cost of NZ$11.15/t;
- total general overheads of NZ$38.9M/yr;
- sustaining capex of NZ$4.9M/yr; and
- embankments and tails pumps/pipelines capex at NZ$3.1M/yr.

The basis for selecting the optimal shell was to choose the shell which maximised the DCF in the Best Case schedule. Shell 36 was selected as the optimal shell as it had the highest DCF of $263M for best case scenario. Interest, tax, depreciation and capital were not included in the DCF calculation. Shell 36 contained 34.4Mt ore at 1.16 g/t and 286.8Mt of waste (strip ratio of 8.34).

Figure 21.3: Pit Optimisation Results

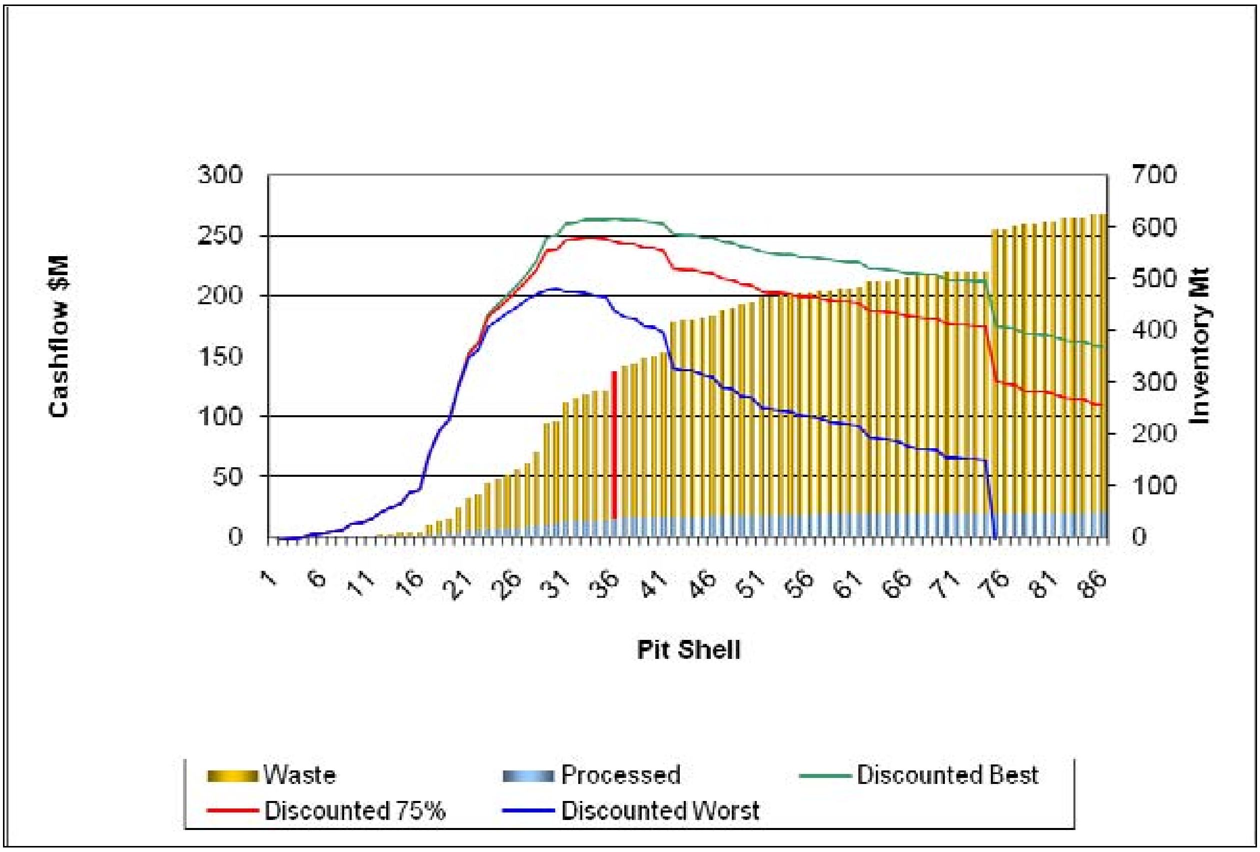


Figure 21.4 shows the cumulative undiscounted operating cashflow versus pit size.

Figure 21.4: Ore Quantity versus Discounted Cashflow

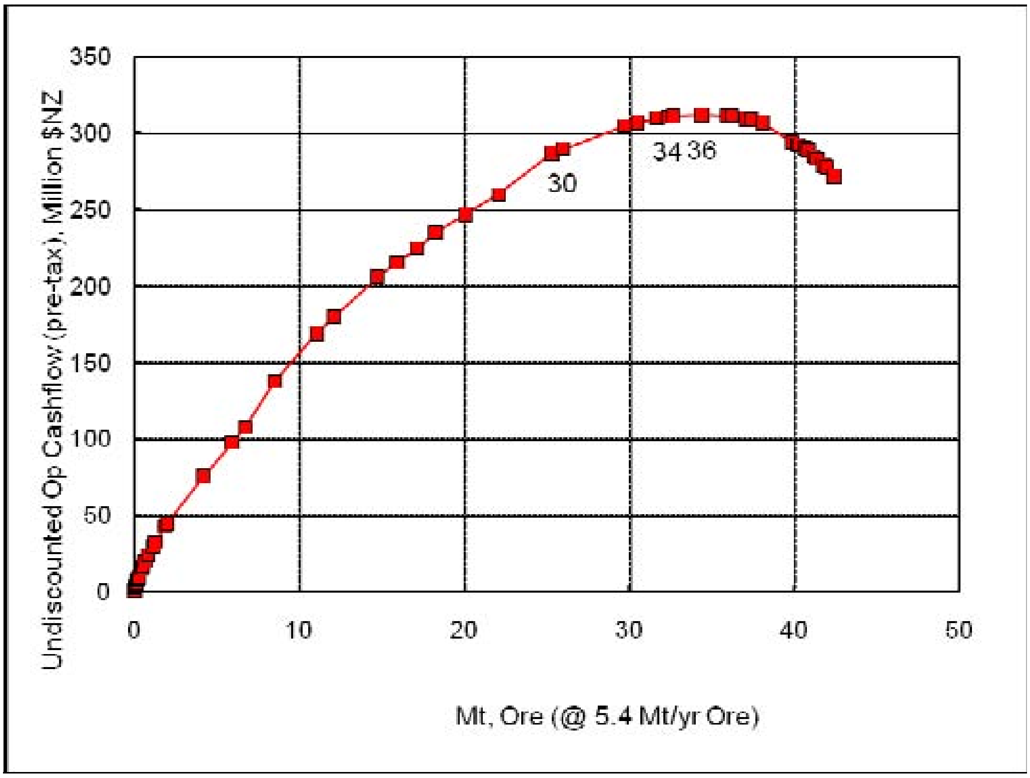
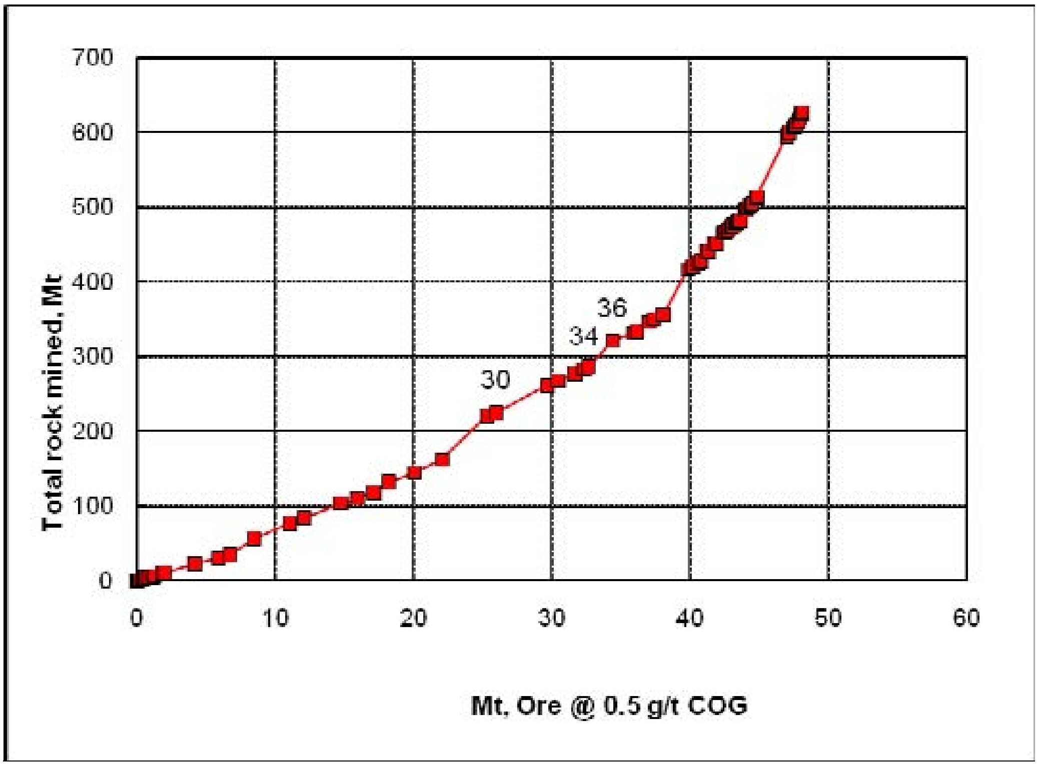


Figure 21.5 below shows the relationship between ore quantity and total rock quantity. The pit shell numbers are shown above the curve and tonnages for the shells can be read from the graph.

Figure 21.5: Ore Quantity versus Total Movement



The graph above shows that the amount of material that would be mined in order to expose significant amounts of ore rises slowly reflecting a consistently increasing stripping ratio. This is due to the increase in overburden as pit expansions move further down-dip in the Frasers ore body. By plotting the ore quantity versus stripping ratio, Figure 21.6, it becomes apparent that the cumulative stripping ratio increases steadily with the size of cutback. The high incremental stripping ratio points correspond to high grade points shown in the ore quantity versus ore grade in Figure 21.7. These high stripping ratio points are a result of pushing back the south east and east walls of the Frasers Deposit.

Figure 21.6: Ore Quantity versus Stripping Ratio

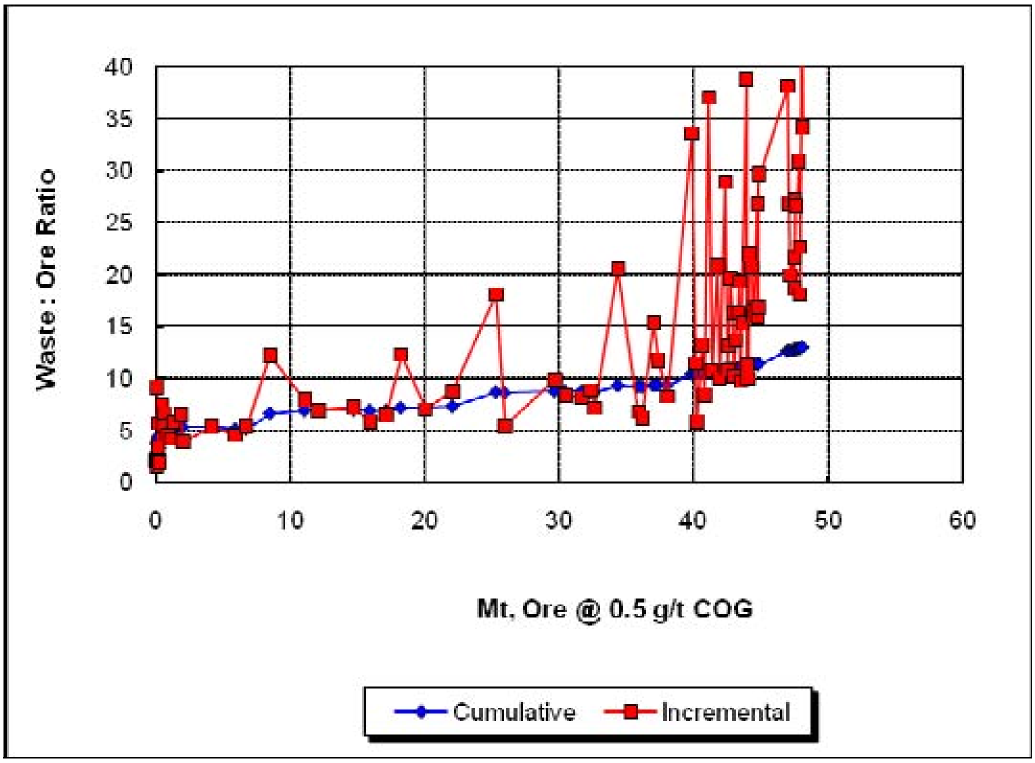
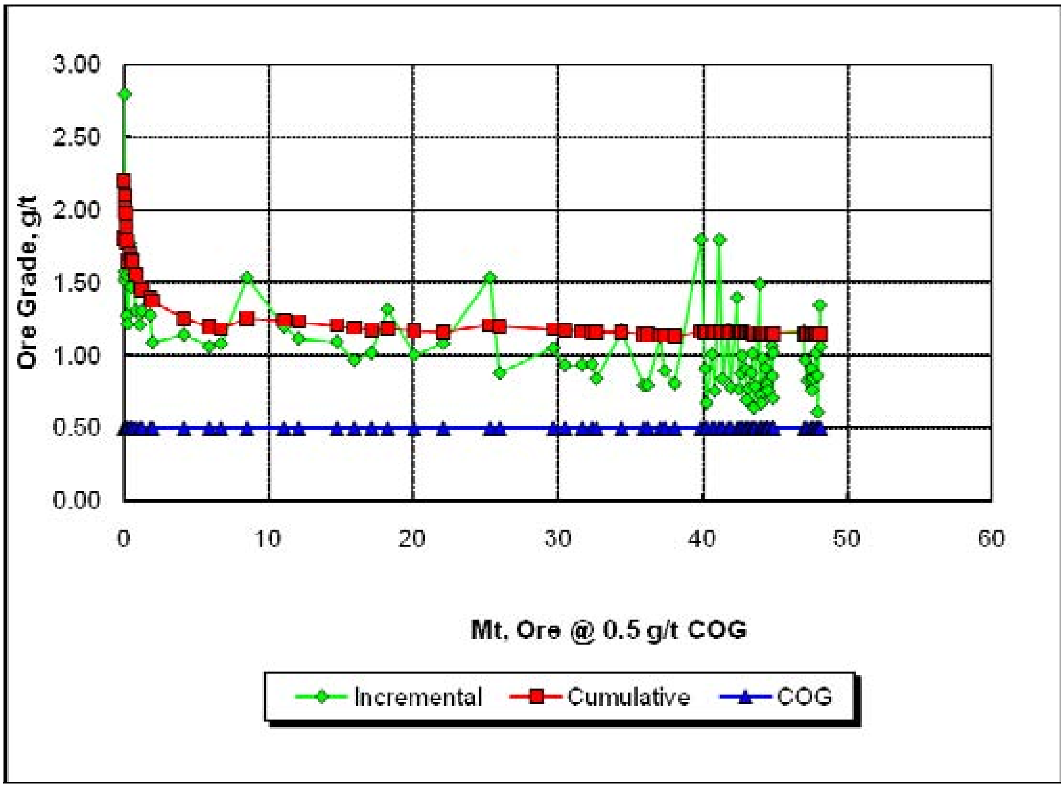


Figure 21.7: Ore Quantity versus Ore Grade



21.1.2.1.10 Generic Pit Design

Pit Stages are designed using MineSight 3-D software. Generic design parameters used in the pit design are detailed in Table 21.5.

Table 21.5: Generic Design Parameters

Parameter	Value
Minimum mining width of lowest 5m cut	30m
Minimum mining width of cutbacks	60m
Ramp width (including windrows)	30m
Inside turning radius on switchbacks	15.0m
Maximum ramp gradient	10%
Maximum bench height	15m
Minimum berm width	7.5m

The slope design philosophy is one of accepting and managing minor localised slope instability rather than incurring the additional costs of designing conservative slopes to guarantee a zero failure rate. It is accepted that on average 20% of any wall may experience some minor bench scale failures, however these will largely be contained on berms and will not adversely effect production. However, in order to optimise pits and reduce costs, slope angles are designed, specifically for each pit, based on kinematic analysis and interpretation of existing geotechnical data. For new pit excavations data is collected from air photo interpretation; surface trenching and diamond drill holes, whilst wall performance and in-pit mapping is used to further refine and optimise staged and final pit walls. This practice has proven to be very successful.

21.1.2.1.11 Frasers Pit Design

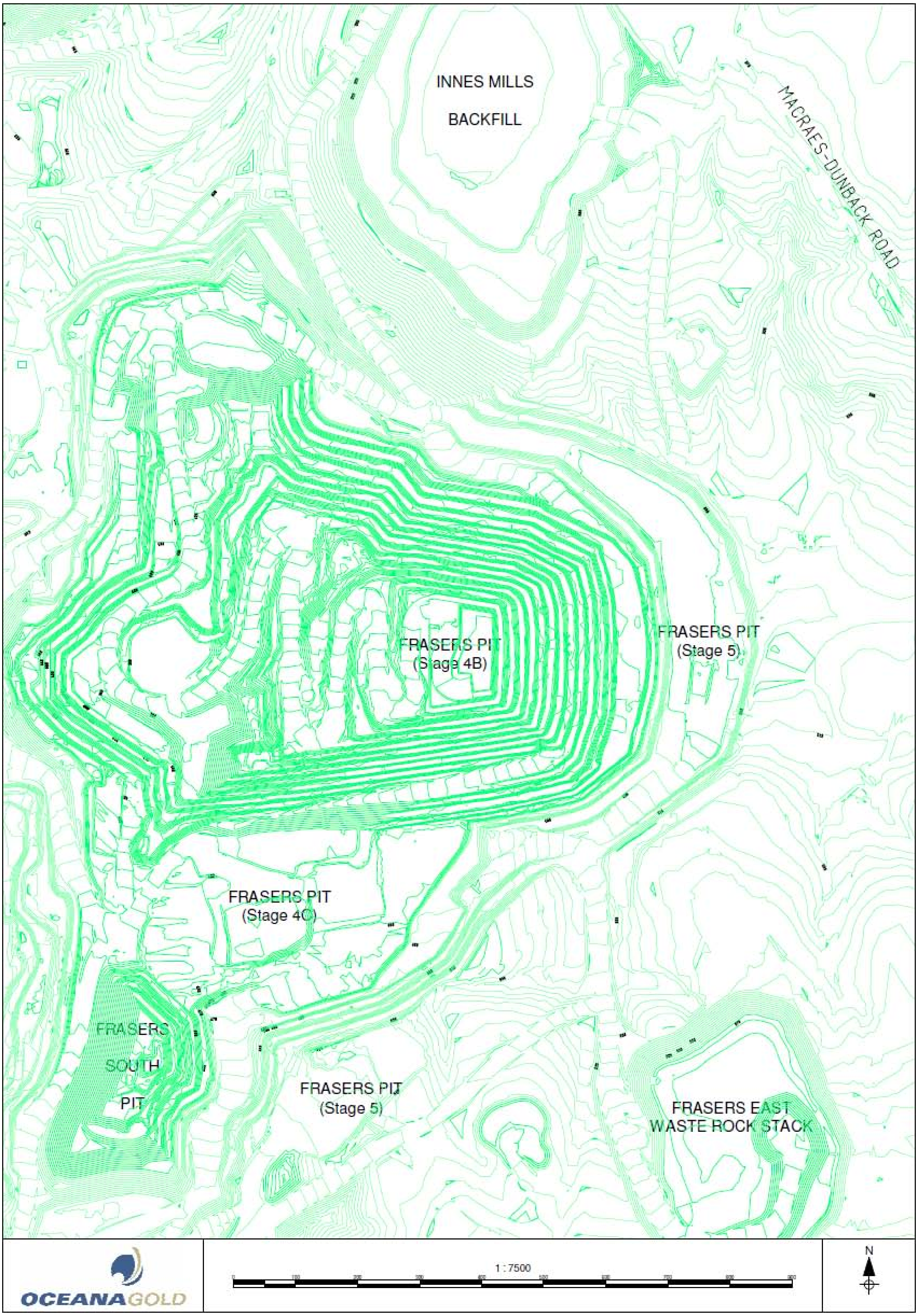
PSM inter-ramp slope recommendations for the Frasers Pit have been adopted (Pells Sullivan Meynink Pty Ltd, 2000). These are summarized in Table 21.6.

Table 21.6: Frasers Pit Slope Parameters

Overall Slope	49^{o}	42^{o}	34-42^{o}	37^{o}
General Region	South Wall	East Walls and North Walls	West Walls	Fault Zone (North)
Batter Angle	70.0-75.0	60.0	50.0-60.0	50.0
Berm Interval (Vertical m)	15.0	15.0	15.0	15.0
Berm Width	7.5	7.5	7.5	7.5

The majority of overall slopes are designed at 42^{o}. Slopes in the Macraes Fault Zone have been laid back to 37^{o} overall. Current topography of the Frasers area is shown in Figure 21.8. The main focus of operations is FN4C, and FN5. FN1, FN2, FN3, FN4A, and FN4B have been completed, along with FS1, FS2, and Golden Ridge. Frasers-Innes Mills (FRIM) and Innes Mills have been mined, but still hold potential for future cutbacks.

Figure 21.8: Frasers Area Current Topography



The FN4C design is shown in Figure 21.9. FN4C is the current active ore source and will be completed in January 2011.

FN4C is a cutback from the completed FN4B pit. Upper level waste was hauled directly to the southern extents of the Frasers West Waste Dump (FWWD). The remaining waste is being hauled through the Frasers pit main ramp to FWWD.

Figure 21.9: Frasers North 4C Pit Design

The FN5 design is shown in Figure 21.10 . The split between Frasers 4C and Frasers 5 is based on trying to maintain a consistent stream of HG ore to the mill.

FN5 upper level waste is hauled via a south wall ramp to the Frasers East Waste Dump (FEWD). The bottom 30m vertical of this ramp is subsequently removed once the main Frasers South ramp is reached at 465mRL. Haulage then switches to the Frasers main pit ramp with material being hauled to the Frasers West Dump (FWWD).

Figure 21.10: Frasers North Stage 5 Pit Design

The FN6 design is shown in Figure 21.11. The design is based on shell 33, and is the final pit stage. The cutback averages 140m wide from the previous FN5 pit stage. The final eastern wall is 330m high with an overall pit slope of 42.9 degrees.

Upper level waste is hauled via a north wall ramp to FEWD. A permanent ramp is designed through the Innes Mills backfill down to 450mRL. Below 450mRL ore and waste are hauled out through the Frasers main pit ramp, with waste material going to both the FWWD and the FEWD using the permanent north wall ramp. Material is mined using a succession of temporary ramps extending off of the existing Frasers 5 ramp network.

Figure 21.11: Frasers North Stage 6 Pit Design

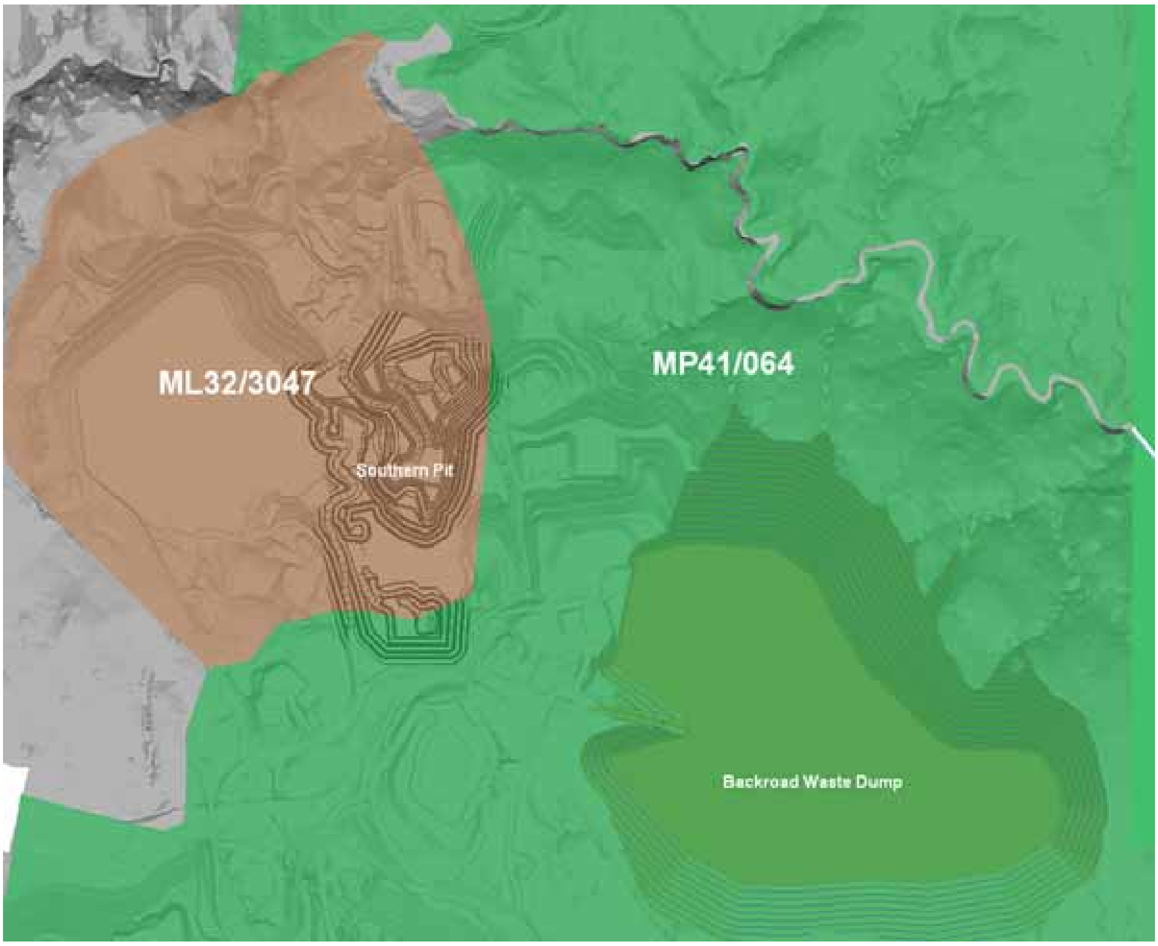
21.1.2.2 Southern Pit Optimisation

Oceana estimated and optimised the resource encompassing Southern Pit and the southern end of Round Hill in December 2009 with a view to reopening a pit in the near future. SP was last mined in 2001 and has since been filled with processing plant tails.

Up to end of October 2009, SP was scheduled to have another lift of plant tails, however, due to the current optimisation work and potential of this resource, alternative sites for a TSF are being assessed and optimisations used a current tailings surface.

This report presents the completed mining findings of the project and includes final figures and all items of the mining study. The mining evaluation has been undertaken on the ML32/3047 mining licence and MP41/064 mining permit as shown in Figure 21.12.

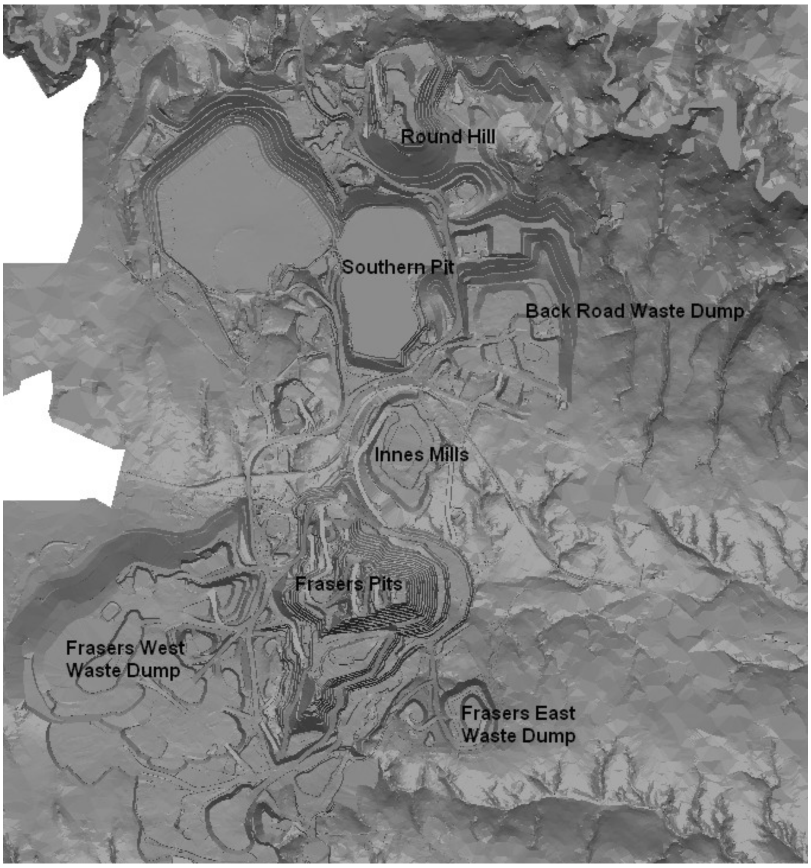
Figure 21.12: Oceana Mining Licence and Permit at Macraes Mine



21.1.2.2.1 Topography

End of October 2009 survey face positions were used for this mining study. The survey file was used to code the Topo field in the block model. Figure 21.13 shows the topography used.

Figure 21.13: Macraes End of October 2009 Topography



21.1.2.2.2 Resource Block Models

Two resource models were supplied for use during the optimisation study. The resource models were for Southern Pit and Round Hill (RH0915.m4) and Frasers and Innes Mills (FRIM15.09) models. The RH0915.M4 resource model was last updated on 21 December 2009. FRIM resource model was created by merging the Frasers resource model last updated in 2005 with Innes Mills model last updated in 2002. Model parameters are shown in Table 21.7.

Table 21.7: Resource Model Parameters

Model	Southern Pit (including Round Hill)	Frasers and Innes Mills
File 10 name	Rh0910.dat	Frim10.dat
File 15 name	Rh0915.m4	Frim15.09
Origin coordinates		
Minimum Easting	14000	69000
Maximum Easting	16000	70800
Minimum Northing	69200	10450
Maximum Northing	71100	13990
Minimum Elevation	0	200
Maximum Elevation	575	620
Block sizes		
X	25	25
Y	25	25
Z	2.5	2.5
SG		
Ore	2.6	2.6
Classification		
Measured	1	1
Indicated	2	2
Inferred	3	3
Weathering		
Oxide	1	1
Sulphide	2	2
Grade fields		
Au grade (g/t)	Au04 to Au10	Au05 to Au10
Percent (%)	Pr04 to Pr10	Pr05 to Pr10

21.1.2.2.3 Pit Slope Angles

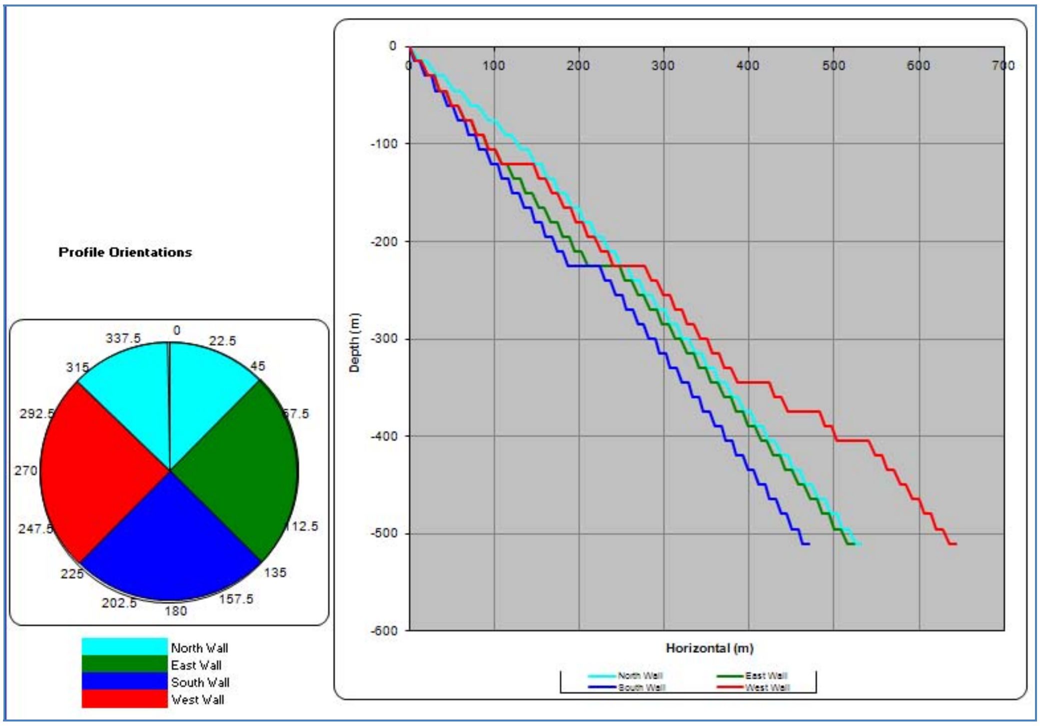
Pit slope angles were derived from mined out pits and current recommended pit slope angles from Pells Sullivan Meynink Engineering Consultants (PSM). The pit slope angles used are shown in Table 21.8.

Table 21.8: Slope Design Parameters

Zone/Bearing	Overall Slope Angle crest to toe (°)	Comments
1	42	Rest of Macraes insitu rock zones
North - 0°	43.5	
East - 045°	44.5	
South - 135°	47.5	
West - 225°	37.6	
North - 315°	43.5	
2	42	Frasers East and North Walls
3	37	Macraes Fault Zone
4	28	Fill and Tails

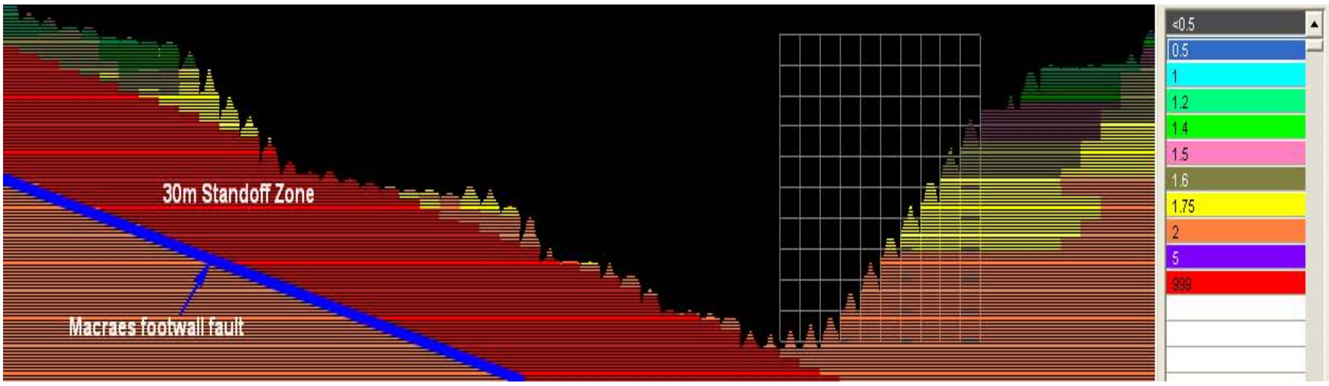
Figure 21.14 shows a plot of slope profiles and profile orientations.

Figure 21.14: Insitu Rock Slope Profiles Comparison



The Macraes shear zone overlies the Footwall fault. The Footwall fault has caused movements in the Frasers pits' west walls and the Processing Plant site when the Round Hill and Golden Point pits were mined. Exposure of the footwall fault on pit floors during mining cause accelerated movement. A standoff distance of 30 metre from the footwall fault was recommended by PSM as a broad rule. The standoff zone was coded in the block model and a high cost was assigned to the blocks in the zone to prevent Whittle 4X from selecting these blocks. The 30 metre standoff zone is shown in Figure 21.15.

Figure 21.15: Macraes Footwall Fault and 30m Stand-Off Zone



21.1.2.2.4 Mining Block Model

A mining block model was created by combining FRIM and SP models to allow optimizations of the Frasers South to Golden point portion of the Macraes strike. The combined block model parameters are shown in

Table 21.9.

The block model was cut to the end of October 2009 survey face shown in Figure 21.13. The block model was coded with the SP10, SP11 and MTD tails solids. The mined out survey topographic solid, excluding fill, was also used to code the model. In-situ, fill and tails regions were assigned different mining costs and slope angles.

The mining block model was coded using several Minesight Meds scripts before exporting to Whittle 4X for optimisation.

Table 21.9: Mining Block Model Parameters

File 10 name File 15 name	rhfr10.dat rhfr15.tls
Origin Coordinates	
Minimum Easting	69000
Maximum Easting	71200
Minimum Northing	10400
Maximum Northing	16200
Minimum Elevation	150
Maximum Elevation	620
Block Sizes	
X	25
Y	25
Z	2.5
SG	2.6
Ore	2.6
Waste	2.6
Fill	2.1
Tails	1.3
Classification	
Measured	1
Indicated	2
Inferred	3
Weathering	
Oxide	1
Sulphide	2
Grade Fields	
Au grade (g/t)	Au05D to Au10D

Percent (%)	Pr05D to Pr10D

21.1.2.2.5 Major Project Assumptions

The following assumptions were made during the study:

- tailings would be rehandled by a fleet of dozers, trucks and an excavator;
- SP10 and SP11 plant tails would have sufficient time to dry before proposed mining;
- rehandled SP10 and SP11 plant tails would be placed on a decommissioned Mixed Tails Dam (MTD);
- waste from SP will be hauled to the Backroad waste dump however an opportunity to lower hauling costs exists when waste is used to backfill RH;
- controlled mining of SP would be south of Northing 15250 to mitigate plant site geotechnical movement;
- the Macraes-Dunback road would not be relocated; and
- additional mining equipment purchases would not be required as this is a one year increment to the current reserve and can be handled by the current fleet with its existing replacement schedule.

21.1.2.2.6 Gold Price and Royalties

The gold price used for this project was based on the Oceana recommended long term gold price.

Using a long term currency conversion of NZ$1 = US$0.60 and US$800 gold price as supplied by Oceana corporate, the calculated gold price used in the optimisation was NZ$1,333/oz. The price and exchange rates are shown in Table 21.10. Parameters used for optimisation are shown in the Long Term (LT column). Diesel price and exchange rates were update during the year and $1.00/l was used as the LT diesel price.

Table 21.10: Oceana Major Economic Assumptions

OCEANAGOLD

MAJOR ECONOMIC ASSUMPTIONS

As at March 2009

		1H 2009	2H 2009	2010	2011	2012	2013	LT
USD Gold Price	USD/oz	890	860	880	850	800	800	800
NZD Gold Price	NZ$/ozs	1,712	1,593	1,544	1,466	1,379	1,379	1,379
FX rate	NZD/USD	0.52	0.54	0.57	0.58	0.58	0.58	0.58
FX rate	AUD/USD	0.65	0.65	0.69	0.69	0.69	0.71	0.71
FX rate	AUD/NZD	1.25	1.20	1.21	1.19	1.19	1.22	1.22
FX rate	USD/PHP	50	50	49	49	48	47	47
FX rate	AUD/PHP	33	33	34	34	33	33	33
Copper Price	USD/lb	1.90	1.90	2.10	2.40	2.60	2.10	2.10
Tungsten Price	USD/mtu	140	140	140	140	140	140	140
West Texas Crude	USD/barrel	45	56	80	90	95	100	100
MOPS diesel	USD/barrel	65	76	100	110	115	120	120
NZ diesel	NZD/litre	0.83	0.98	1.23	1.32	1.37	1.43	1.43

Crown royalties were based on 1% of gold price on positive cashflow. Crown royalty payments were incorporated in the administration cost as determined by Oceana Accountants. Normal practice would be to apply the royalty cost as a selling cost in the optimisation but due to the way New Zealand Government treats the royalty payment, it has been calculated over the current life of mine and the average has been incorporated into the Administration cost.

A discount rate of 8% was applied to the optimisations to produce discounted cashflow values (DCF).

21.1.2.2.7 Time Costs

Fixed operating costs were derived from the LOMP09 and averaged over the current life of mine. The fixed costs were apportioned to processing and mining cost. A summary of time costs is shown in Table 21.11.

Table 21.11: Time Costs

TIME COST SUMMARY		**Proportion of Cost to every tonne**			
Overhead Classification	**Cost estimate pa**	**Milled**	**mined**		
Mining department	$13,778,590		100%		$0.23/t mined
Administration	$8,539,174	100%		$1.58/ t milled	
Refining & Royalties	$3,887,542	100%		$0.72/ t milled	
Human Resources, Safety, Environment	$1,874,348	100%		$0.35/ t milled	
Sustainable Development	$1,249,367	100%		$0.23/ t milled	
Sustaining capital	$3,927,208	79%	21%	$0.58/ t milled	$0.01/t mined
TOTAL TIME COSTS	**$33,256,229**	56%	44%	**$3.46/ t milled**	**$0.24/t mined**
Expected yearly mill throughput	5,400,000 tpa				
Expected yearly mining capacity	60,000,000 tpa				
Time costs per tonne milled	**$3.46/t milled**			$18,657,032	
Time costs per tonne mined	**$0.24/t mined**			$14,599,197	
				$33,256,229	

21.1.2.2.8 Mining Cost

Mining costs were derived from the current LOMP09 document. LOMP09 represent the actual Macraes operating costs and is the basis for 2010 operating budget.

Hauling costs were based on hauling distance and were varied by bench and by pit of origin. The total mining cost was coded into the block model as MCAF on a block by block basis. Table 21.12 shows the elements that make up the total mining cost.

Table 21.12: Mining Costs

Mining Parameters	Units	Value
Drill Ore Zone	($/t)	$0.07
Blast Ore Zone	($/t)	$0.23
Drill Waste Zone	($/t)	$0.05
Blast Waste Zone	($/t)	$0.25
Load Ore Zone	($/t)	$0.32
Load Waste Zone	($/t)	$0.24
Ancillary (All)	($/t)	$0.18
Fuel	($/l)	$1.00
785C Unit Cost (excluding Fuel)	($/hr)	$159.56
789C Unit Cost (excluding Fuel)	($/hr)	$177.10

21.1.2.2.9 Processing Cost

Processing costs were based on the LOMP09 average costs. Processing costs for SP and RH were adjusted to include extra tailings pumping costs to a new TSF that would replace the current MTD. Ore overhaul and grade control costs were calculated for each bench and added to the block processing cost. Total block processing cost including time costs were coded as a processing cost adjustment factor (PCAF) in each block. Table 21.13 summarises the processing costs.

Table 21.13: Optimisation Processing Cost

Variable	Unit	Cost
Processing Cost	$/t ore	10.63
Feed Crusher Cost	$/t ore	0.76
Grade Control Cost	$/t ore	0.07
Time Cost	$/t ore	3.46
Tailings Dam Construction	$/t ore	1.31
Ore Over/(under) Haul	$/t ore	(0.06)
Total Processing Cost	$/t ore	16.16

21.1.2.2.10 Gold Recovery

Gold recovery factors were supplied by Oceana Metallurgists. An average of 81.8% was applied to Frasers Innes Mills and Round Hill pits and 79% was applied to the SP resource. The lower recovery for SP was based on historical performance and is expected to be lower than the Frasers pits due to pregnant solution robbing by carbonaceous material in the hanging wall. The effect of the carbonaceous material would be managed through blending with Frasers ore. Detailed modelling of the different structures in the SP geological resource would enable mill recoveries to be associated to either hangingwall, concordant or storkwork ore. Historically, ore affected by carbonaceous material has been approximately 10% of the hangingwall ore. By applying the low recovery to the 10% of hangingwall material it is expected that the overall recoveries would be higher than 79%.

21.1.2.2.11 Mining Dilution and Recovery

Selective ore mining procedures are used. This is done to maximise ore recovery and minimise mining dilution. During the ore definition process, grade control blast hole assays are used as the input data to a conditional simulation grade control process. The results of bench grade estimates are then used in conjunction with detailed geological mapping to produce mining blocks. Ore mining is supervised by geologists and ore spotters. Mining of the ore waste contacts is done by backhoe excavator.

Dilution is accounted for in the resource model calculations by adding a waste veneer to the hanging wall contact, and using dilution estimation during the kriging process. The result is a dilution/recovery factor of close to 2%. However SP has a mining dilution factor of 5% applied in the optimisation. A mining recovery factor of 97% was also applied.

Macraes mining department has strict control on ore zone mining and it is expected that dilution and mining losses will be minimal.

21.1.2.2.12 Parameters Summary

Table 21.14 summarises the parameters used in the pit optimisations.

Table 21.14: Optimisation Parameters

Parameter	Units	Value
Dilution	%	5
Mining Recovery	%	97
Average Mining Cost	$/t	1.67
SP and RH Processing Cost	$/t	16.16
US$ Gold Price	US$/oz	800
NZ$ Gold Price	$/oz	1,333
Discount Rate	%	8
Southern Pit Au Recovery	%	79
Other Pits Au Recovery	%	81.8

21.1.2.2.13 Capital Costs

Capital costs were not included in the optimisation scenarios, but must be considered in the overall financial evaluation of the study.

21.1.2.2.14 Optimal Pit Selection

Whittle 4X software uses the Lerchs-Grossmann algorithm to generate a series of nested pit shells by incrementally changing the revenue using a revenue factor (RF) and keeping all other input parameters constant. Shells are individually scheduled and evaluated against common mining and processing limits, discount rate, time and capital costs. The results are utilised to determine the best or "optimal" shell from which to base a pit design on.

The basis for selecting the optimal shell was to choose the shell which maximised the DCF in the Best Case schedule.

21.1.2.2.15 Optimisation Results

Figure 21.17 shows optimisation results plot for tonnes and cashflows against pit shell number. Shell 36 was selected as the optimal shell as it had the highest DCF of $249 M for best case scenario. Interest, tax, depreciation and capital were not included in the DCF calculation. Shell 36 contained 35.1 Mt ore at 1.17 g/t and 295.1 Mt of waste (strip ratio of 8.41). Figure 21.16 shows shell 36 in relation to the Macraes topography.

This optimisation was for the Macraes Line-of-Strike and was used to identify potential pushbacks in the SP area.

It can be seen from the flatness of the DCF curves from shell 29 to shell 38 that there is very little difference between any of these shells. This indicates that any shell from 29 to 38 could be suitably selected as the optimal shell, however the risk increases rapidly beyond shell 36 due to the amount of waste stripping required.

Figure 21.16: FRIM, SP and RH Optimisation Shell 36

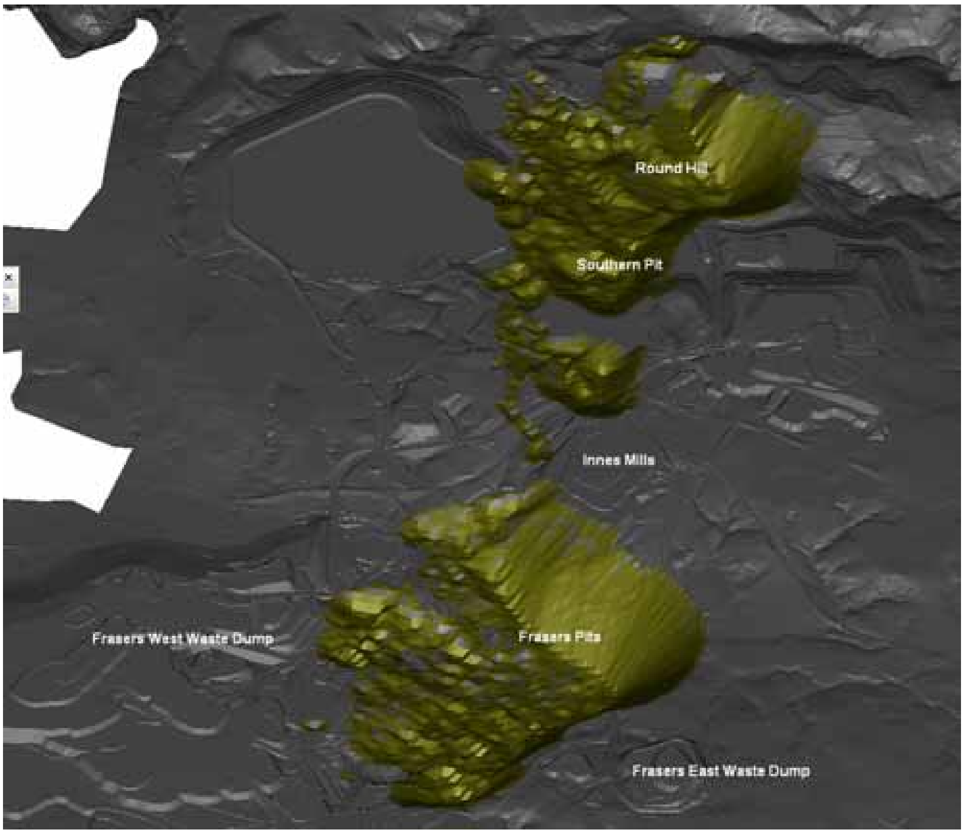


Figure 21.17: Pit Optimisation Results

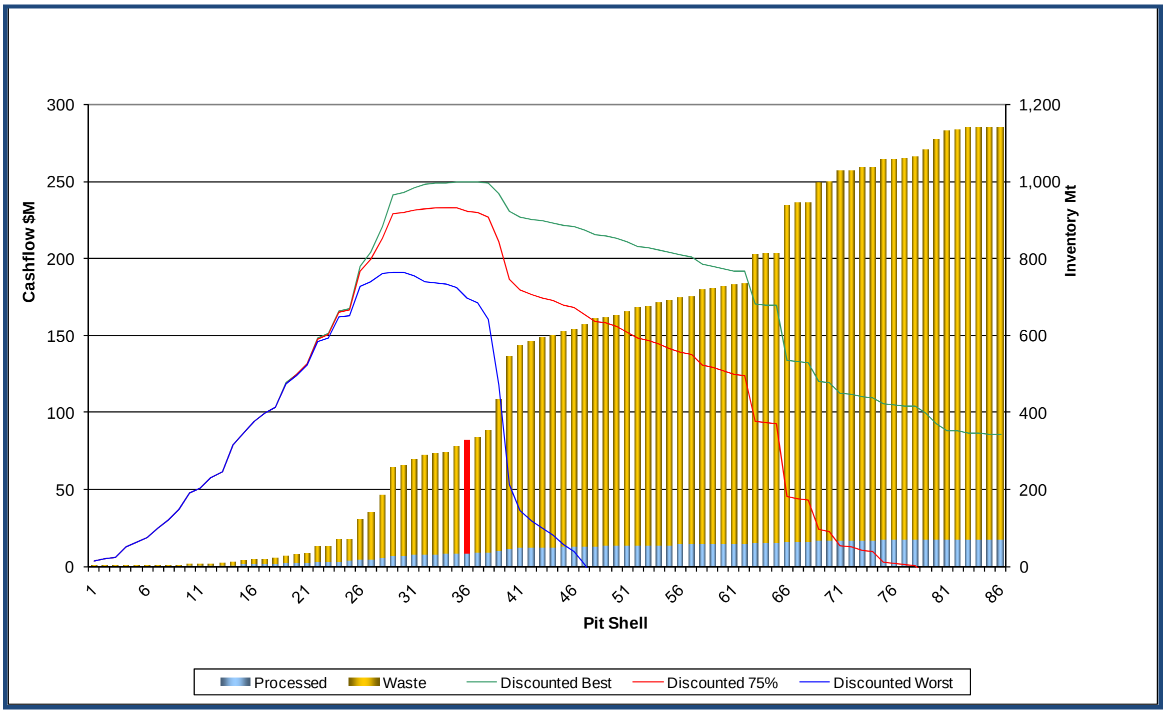


Figure 21.18 shows undiscounted operating cashflow against ore quantity. Shell 36 is at the pick of the curve with 35.16 Mt of ore. Shells less than 36 and greater than shell 36 have lower cashflow values. Pit shells greater than shell 36 carry a high risk of losing value due to high incremental striping ratios (Figure 21.20) and high required annual mining rates (Figure 21.21) to maintain required process plant throughput. Minor changes in gold price, gold recovery and operating costs would make these shells uneconomic.

Figure 21.18: Undiscounted Operating Cashflow versus Ore Quantity

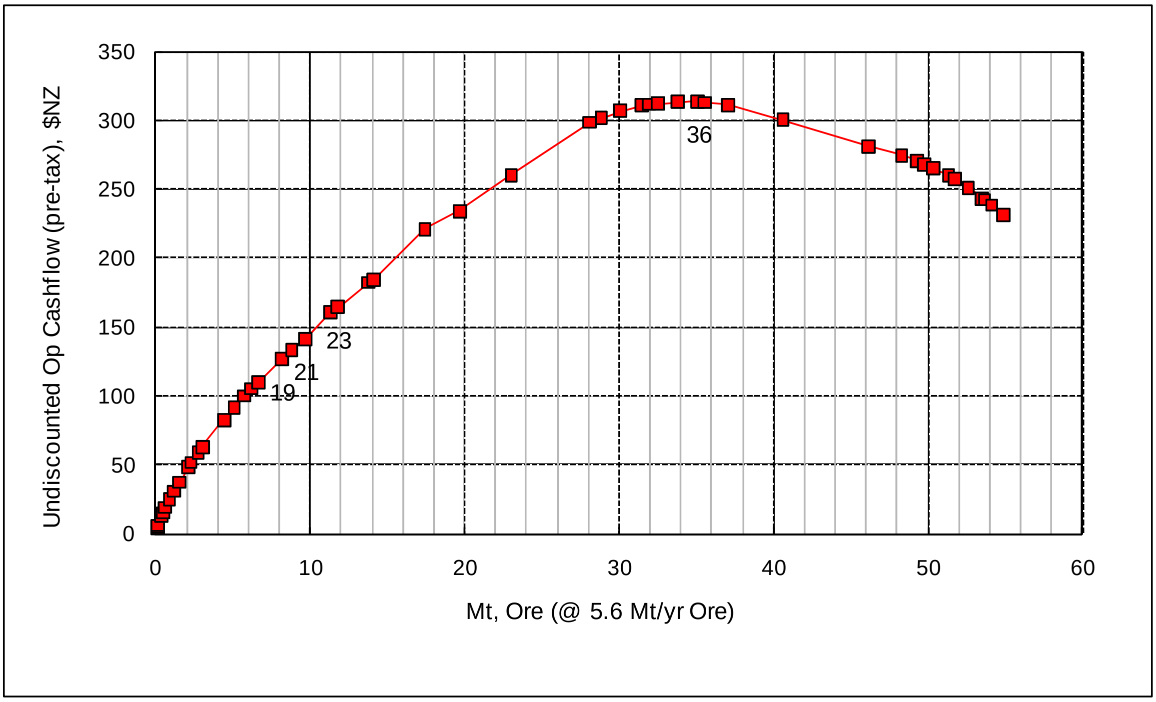


Figure 21.19 shows ore grade versus ore quantity. For pit shells greater than shell 36 the incremental grade rises to 1.4 Au g/t and drops immediately back to 1.2 Au g/t. This is in the Innes Mills area where the resource model still needs to be corrected and audited. Shells in the Innes Mills region are high risk shells as they depend mainly on grade estimation accuracy, which still needs to be confirmed.

Figure 21.19: Ore Grade versus Ore Quantity

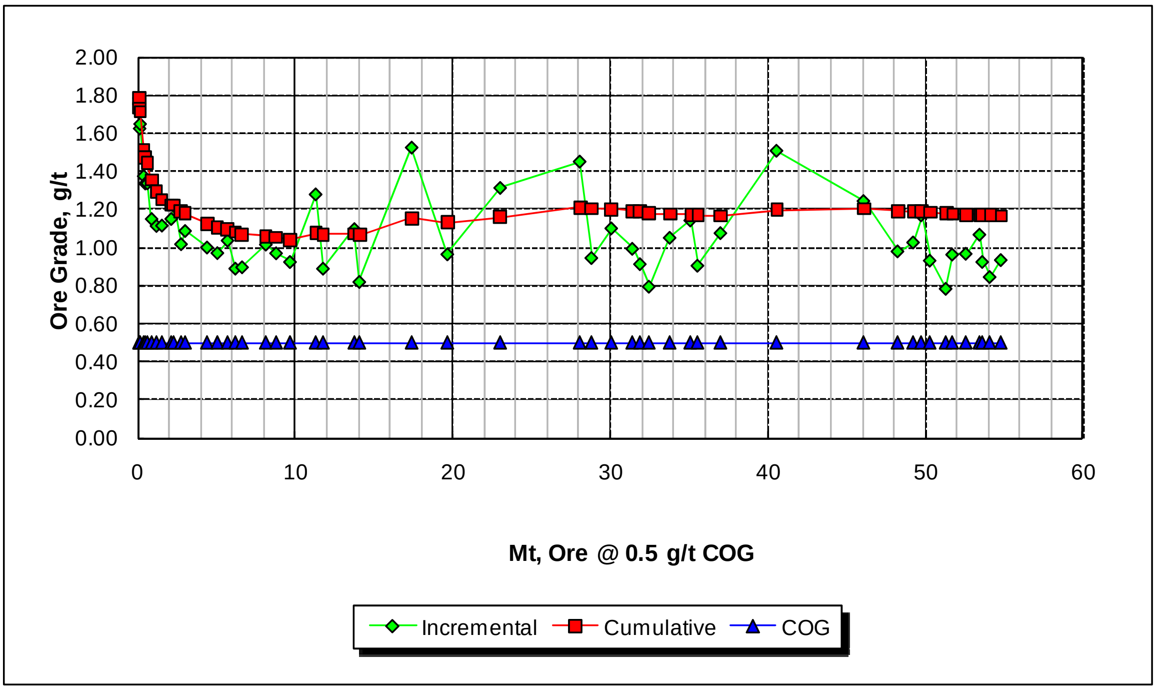


Figure 21.20: Strip Ratio versus Ore Quantity

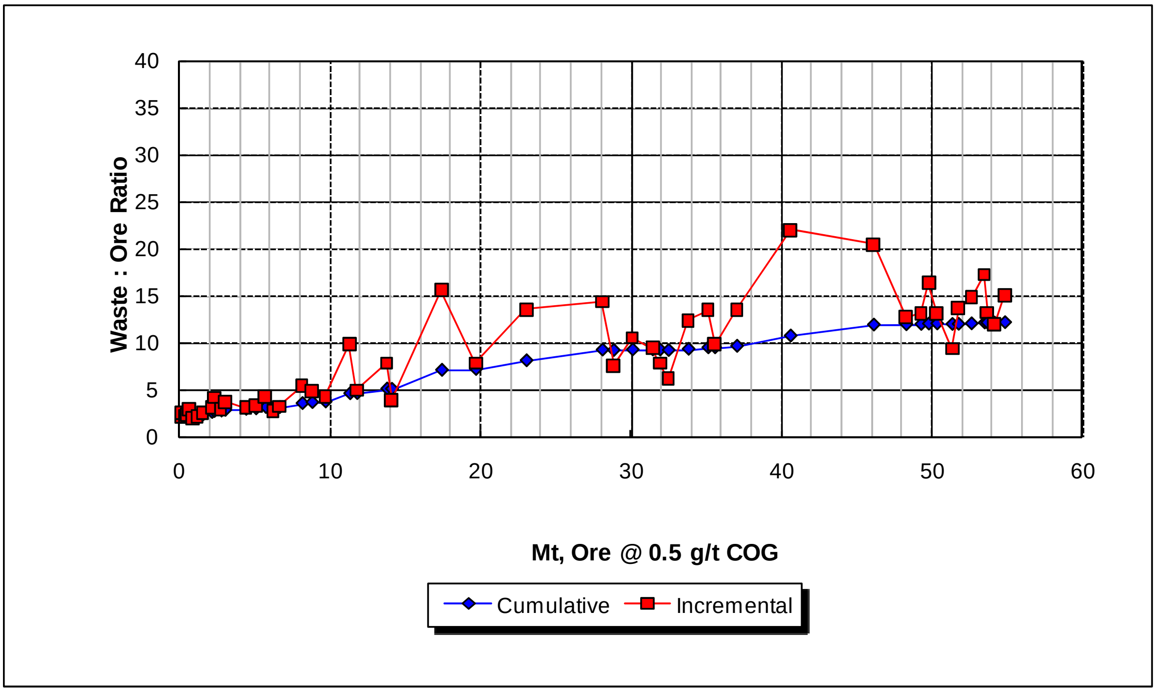
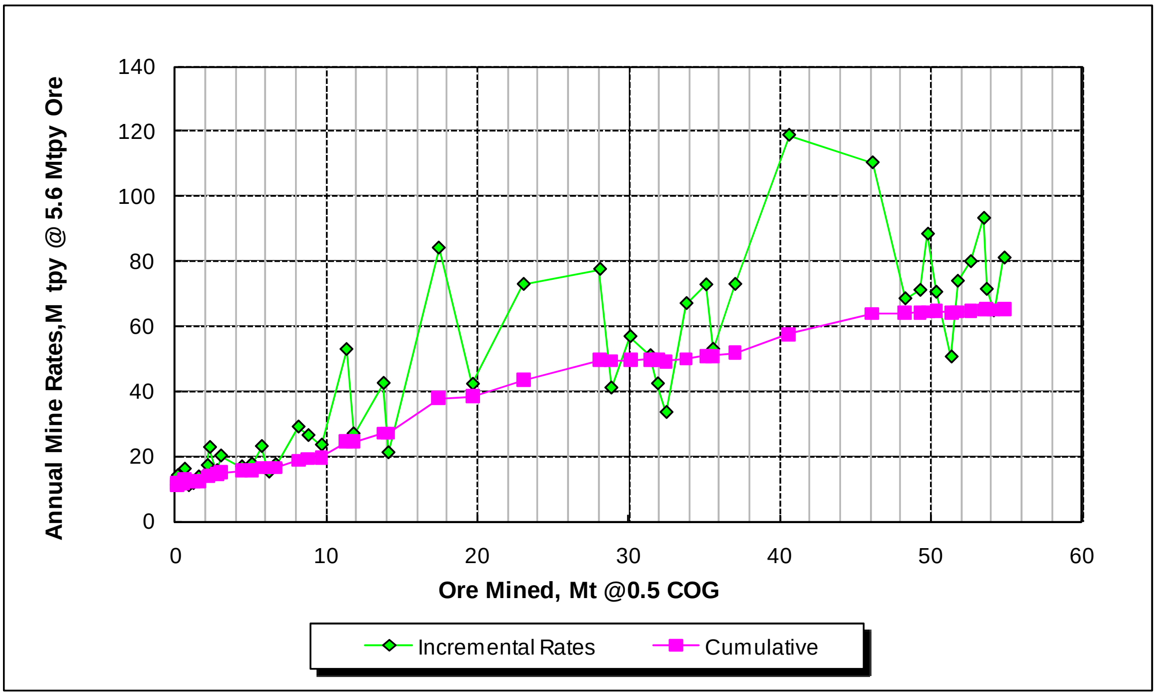


Figure 21.21: Required Annual Mining Rate versus Ore Quantity



21.1.2.2.16 Sensitivity Analysis

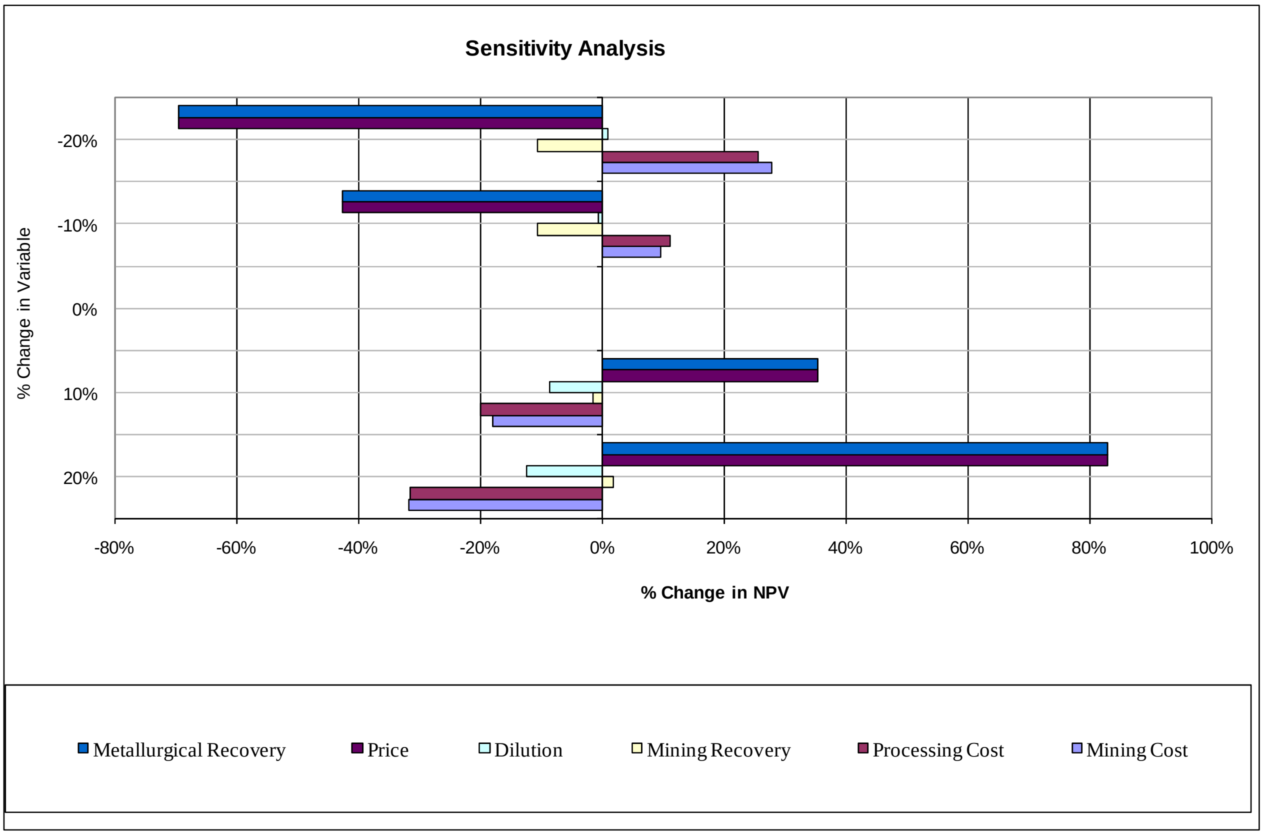
A sensitivity analysis was completed on a number of the inputs and assumptions. The purpose of this analysis was to identify the critical input variables that affect the value of the project.

The following variables were varied in the sensitivity analysis:

- processing cost ±20%;
- mining cost ±20%;
- mining dilution ±4%;
- mining recovery ±3%;
- metallurgical recovery ±20%; and
- metal price ±20%.

Figure 21.22 shows the results of the sensitivity analysis. The results indicated that the project NPV was most sensitive to gold price and metallurgical recovery. This was followed by the processing cost and mining cost, while it was least sensitive to mining dilution, and mining recovery.

Figure 21.22: Sensitivity Analysis Graph



21.1.2.2.17 Pit Designs

The pit design parameters used are shown in Table 21.15. The flitch height, minimum mining width and ramp widths were based on current mining practice at Macraes open pit mine.

Table 21.15: Pit Design Parameters

Parameter	Units	Value
Bench Height	m	15
Flitch Height	m	2.5
Batter Angle –Insitu Rock	°	65
Batter Angle –Fill/Tails	°	50
Berm Width	m	7.5
Berm Interval	m	15
Ramp Width (two-way)	m	30
Ramp Gradient	%	10
Overall Slope Angles	**Units**	**Value**
Fill	°	28
Tails	°	28
East	°	44.5
West	°	37.6
North	°	43.5
South	°	47.5
Minimum Mining Width	m	20

Overall pit wall angles comply with the geotechnical recommendations that are summarised in Section 21.1.2.2.3.

The ultimate pit design was based on pit shell 36 of the optimisation.

Figure 21.23 shows the ultimate pit design and associated haul roads. Figure 21.24 shows the ultimate pit design and shell 36 in the Southern pit area. The current AR3 road to the ROM pad would be closed and the old AR2, which is shorter, reopened. Sections of the bottom benches were designed without ramp access and mining would be based on top loading this material with excavators.

Figure 21.23: Southern Pit Design

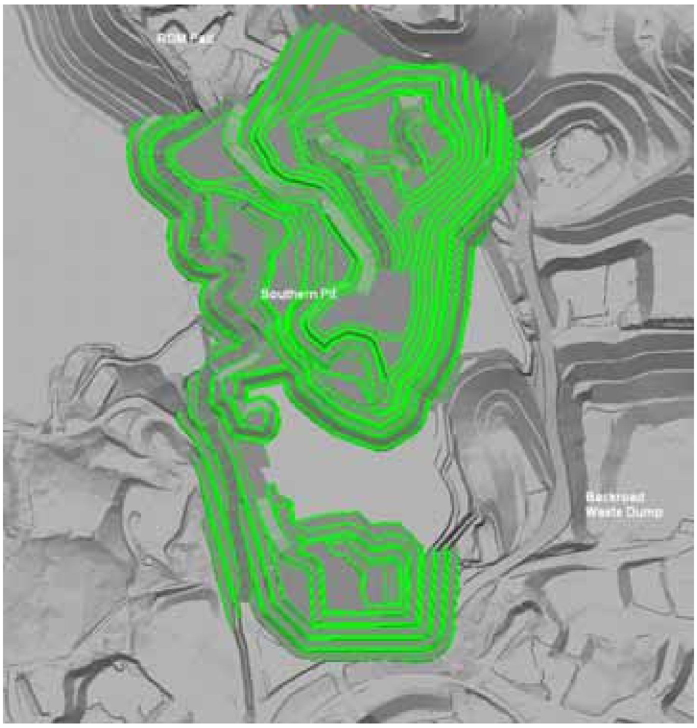


Figure 21.24: Southern Pit Design and Shell 36

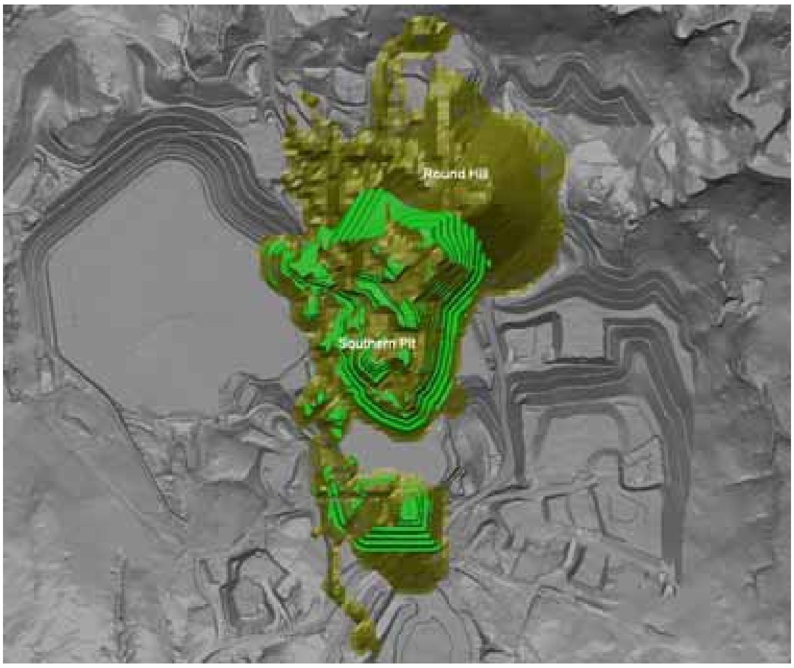


Figure 21.24 shows the ultimate pit design (green surface) against the Whittle pit shell 36 (golden surface). The main difference between the two surfaces was the additional waste required to accommodate the ramps without reducing the quantity of ore in the ultimate pit.

Table 21.16 shows the Ore Reserve and waste inside the ultimate pit design using a 0.5 Au g/t in-situ cut-off grade. All inferred material is excluded from the total Ore Reserve.

Table 21.16: Design Inventory

	Ore tonnes (Mt)	Au (g/t)	Insitu Waste tonnes (Mt)	Fill tonnes (Mt)	Tails tonnes (Mt)	Stripping ratio	Total Tonnes (Mt)
SP Design	5.93	1.27	28.80	36.13	15.79	13.62	86.65

21.1.2.2.18 Optimisation Changes

New tailings rehandling costs were provided after the main optimisation and pit design was completed. Optimisation runs were performed to check if the results would be different to the design that had been done. The results were similar and no further changes were made to the pit design.

New pit slope angles were also supplied after the study was completed. The angles used in the optimisation are shallower compared to the new angles submitted by PSM. Since the angles used in the optimisation were derived from current pits, they were adopted as the official slope angles.

21.1.2.2.19 Waste Storage

Due to the potential increase in the size of the ultimate pit design as a result of potential increase in geological resource and gold price, all waste was assumed to be deposited in external waste dumps despite the potential for backfilling the pit.

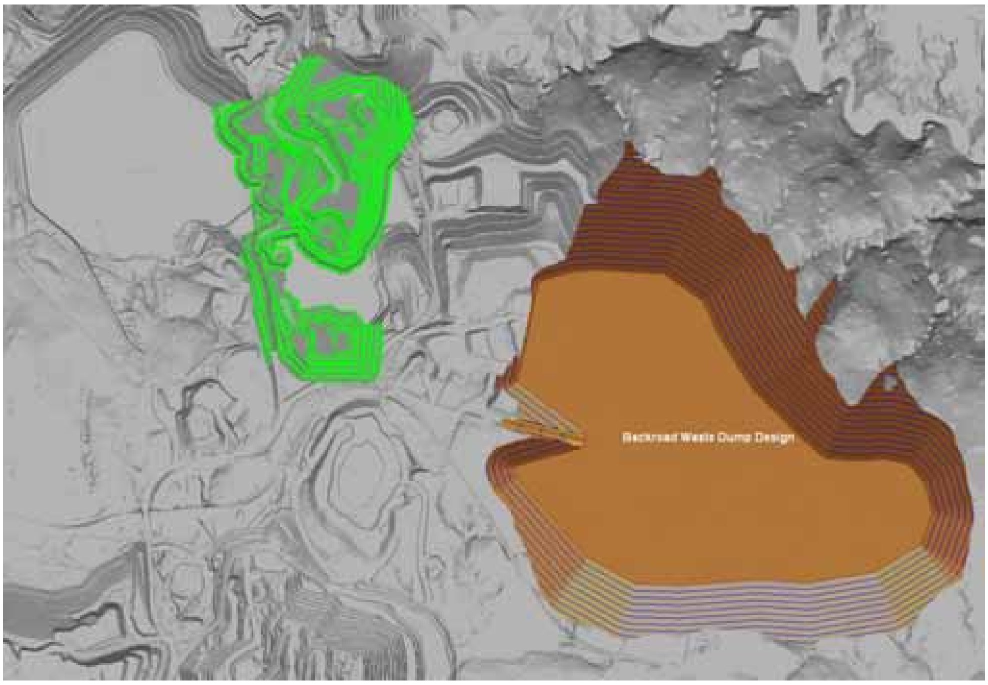
The dumps were designed to a 620 RL using a 19% swell factor (after compaction). The outside slopes were designed to 18 degrees and access ramps were designed at 1:10. Table 21.17 shows the capacity, area and source for each of the waste destinations.

Table 21.17: Waste Destinations

Destination	Capacity (Mlcm)	Source
Back Road Waste Dump	144	Southern Pit Round Hill Frasers Stage 6 Innes Mills
Total	**144**	

Figure 21.25 shows the location of the Backroad Waste Dump. The northern edge of the dump was restricted by the Deepdell creek. The dump encroaches onto the Sutton's property (owned by Oceana to the east and is limited by the Macraes-Dunback road to the south. The western toe of the dump is at 2 times revenue factor to allow for further pit expansion.

Figure 21.25: Site Plan showing Backroad Waste Dump



Top soil stripped from the dump site would be stockpiled on existing top soil stockpiles and would be used for rehabilitation work at mine closure.

21.1.2.3 Geotechnical Design and Control

Given the long duration of experience in mining the local rock mass, pit walls are designed at angles that are well optimised in light of geotechnical considerations. While this results in occasional localised batter failure, an active monitoring system has ensured that safety is not compromised and has allowed potential geotechnical problems to be anticipated. Minimal ore has been lost through wall failures to date and the approach has allowed the minimisation of overburden removal costs. Pit wall conditions remain manageable with regard to safety and the continued extraction of ore.

Specialist geotechnical expertise is utilised to ensure that any potential ground movement at the various operations is unlikely to interfere with production. The Footwall Fault, which runs beneath all the pits along strike at Macraes, has a long history of "creep" movement. Oceana has developed several operational techniques, including displacement monitoring, toe buttressing with backfill, drainage depressurisation and intermittent mining, all of which control the risk to operations. Another technique has been to split the pits into two or more sections, so that one zone is supported while mining takes place in adjacent areas. Oceana's 19 years of experience in six separate pits has evolved geotechnical provisions that are appropriate and effective to be able to continue mining without major interruption from Footwall Fault movement.

At this stage, the Frasers pits comprise the bulk of the remaining resource and has a final depth of around 330m. The Frasers pit is located to the south of the Macraes fault zone, an east-west striking shear zone approximately 100m in width. The Macraes fault zone intersects with the upper north wall of the Stage 6 Frasers Pit cut back, and accordingly the affected section of northern pit slope has been moderated to 37 degrees. Site experience with the Macraes fault zone and similar fault zones has shown that this designed slope angle allows the wall to be safely excavated without inducing large-scale failure. Additionally a set of discontinuous north-south structures, dipping moderately steeply to the west may cause batter scale failures on the east wall and a large persistent moderately angled east-dipping set of shears known locally as the Northern Gully Fault Family provides release surfaces for occasional batter-scale wedge failures.

21.1.2.4 Open Pit Scheduling

Xpac Advanced Destination Scheduler is used to schedule Macraes pits. The software was used to produce LOMP09 schedules based on set objectives and constraints. Objectives that were applied aimed to:

- maximise grade and hence ounces;
- maximise mill and load and haul equipment utilisation; ad
- minimise hauling distance from source to destination hence minimising cost.

Constraints applied during scheduling were mainly to:

- utilise available equipment capacity as a maximum;
- obey mining dependency rules which prevent uncovered blocks from being mined;
- obey set source-destination pairing as specified during scheduling; and
- obey maximum periodic mining targets for stages to achieve blending and striping requirements.

Details of the scheduling process are in LOMP09 document.

The proposed mine plan for the remaining life of the Macraes open pit operation is summarised in Table 21.18. Due to cycles of waste stripping for different stages, gold production fluctuates throughout the schedule from a minimum of 158koz in 2014 to a maximum of 291koz in 2015. Year 2017 reduced gold production is due to left over ore in 2016 flowing into 2017.

Total material movement is scheduled to remain constant around 52Mtpa from 2010 to 2016, tailing off as stripping operations reduce and the ore in the final pits is exposed. Due to pre-strip requirements and scheduling constraints, there are periods when the mining operations are unable to provide sufficient ore to the mill and the mill feed is then supplemented with medium/low grade stockpiles ore.

Table 21.18: Macraes Mining Schedule 2010 to 2017

Item	Unit	2010	2011	2012	2013	2014	2015	2016	2017	Total
Macraes Open Pit Mining										
Ore Milled	Mt	4.50	4.93	5.07	5.54	5.66	5.76	4.90	1.03	37.38
Grade – Au	g/t	1.15	1.20	1.24	0.95	0.82	1.51	1.22	1.49	1.16
Recovery – Au	%	82.2	82.2	82.5	81.5	82.1	80.8	79.1	79	81.4
Gold Produced	koz	137	156	168	138	123	226	152	39	1138

Potential exists to define additional mineable ore from material not in the current reserves, including open pit stockwork zones and down-dip extensions of current ore regions that, depending on gold price, further infill drilling and mine planning may identify some of this material as mineable.

Gold production shown in Table 21.18 is only the contribution of the Macraes open pit mining operations.

21.1.3 Frasers Underground Mine

The Frasers deposit mineralization remains open down dip to the east of the final walls of the Frasers Stage 6 open pit, which is planned to be mined to a depth of 330m. Diamond drilling has intersected mineralization along the hangingwall shear structure, extending to depths of 650m. Although the dip flattens to around 10-15°, the incremental stripping ratio down dip is high and the economics of further cut-backs appear marginal.

The FRUG operation has been operating since 2008 and is mined under an alliance agreement, with Oceana providing management and technical guidance to the mining contractor who performs the physical mining tasks.

Oceana has retained the services of a number of geotechnical and mining engineering specialists that contributed to the 2005 Technical Study to maintain a working knowledge of this evolving mining method and to provide futher design recommendations based on the actual mines performance.

The underground retreat long-hole open stope (RLHOS) mining operation utilises electric-hydraulic development jumbos, diesel load-haul-dump units, diesel haul trucks and a production drill rig to extract both waste and ore. The retreat stope voids are not backfilled, instead the mine design utilises yielding pillars between adjacent extracted stopes to gradually deform over a timeframe that permits ore extraction.

The up-dip panel (Panel 1) has had a significant amount of reserve transferred from the underground block model into the Frasers Open Pit block model as a result of the inclusion of Frasers Open Pit Stage 6 to LOMP09. The remainder of Panel 1 represents an extension of the Frasers Open Pit Stage 6 mineralization, extending from 300m to a depth of around 320m.

At around 320m, the tenor of mineralization drops, and remains patchy down dip for a distance of approximately 250m. The width, grade and continuity then improve below 370m to at least 650m, forming a second panel (Panel 2) which to date remains open down dip below 650m. Both panels have been drilled to approximately 50m patterns from surface.

The mineralized zone within Panel 1 averages 15-25m thick. Panel 2 intersections are generally thinner. A small reserve increase relating to the Panel 2 deeps has recently added to the LOMP09 and remains open particularly to the east. Oceana personnel have created practical conceptual designs and final designs for development and stopes down to 630m for Panel 1, Panel 2 and Panel 2 Deeps which have estimated ore loss and dilution factors applied. Inferred resources have influenced development placement but have not been included in the reserves or the financial analysis. Although in historical mining the extraction of inferred ounces has been adding to the mines production.

Exploration drilling of the Panel 2 Deeps and for a potential Panel 3 are being undertaken during 2010 due to the open pits extended life contributing to processing plant feed out to 2017. The extension of open pit mine life has put focus back on whether additional underground ore deposits exist to contribute processing plant feed over the same period.

Oceana's performance at FRUG has shown that the mining equipment and mining methods are suited to the required mining rates and deposit geometry. The underground mine design procedures are appropriate and have been conducted in accordance with established industry standards and with input from appropriately qualified geotechnical specialists, mining engineering specialists and external consultants. Safety records are generally in line with industry standards. The FRUG LOMP09 schedule has been prepared for 2010 to 2011 and the schedule relies on reserves and approximately 67,700 additional mined ounces from a number of inferred stopes and a conservative low grade for development through inferred resource, which are considered appropriate and reasonable for long term site planning. For this NI 43-101 document the quantities and financial figures have been back adjusted for the FRUG mine to remove the LOM's approximate 67,700 mined ounces derived from inferred resource.

21.1.3.1 Development Plan

The established infrastructure in the FRUG mine consists of the portal at 352.5mRL within the Frasers Open Pit with a decline to 570mRL, a 355m by 4mØ return air raise to surface with fans installed underground, a 350m by 2.4mØ secondary egress raise with ladder to surface, an underground crib room, explosives magazine, satellite refuelling station, four Mineark refuge stations and primary dewatering pump station.

Development waste, development ore and stope ore are hauled from underground to an in-pit stockpile area at the 367.5mRL and paddock dumped and manually screened for metal and other rubbish. Twice a month the stockpiled ore is rehandled by the open pit fleet to the process plant ROM. The waste is dumped onto the footwall of the Frasers pit where it is not forecast to need to be rehandled.

The revised underground development design will facilitate the completion of infill drilling of Panel 2 and Panel 2 Deeps from suitable locations underground, as well as further define the identified Panel 2 extension resources further down dip. All these being drilled from an exploration drill drive commenced in mid 2009.

Oceana has continued to utilise an underground mining contracting firm through an alliance contract arrangement. The option mentioned in the 2007 Technical Report to change to owner-operator for the underground mine was reviewed in 2008 through a cost/benefit analysis which resulted in the continuance of the alliance.

The underground long term plan is to produce at around 1.1Mt in 2010 and 0.6Mt in 2011 with the Macraes open pit production operating in parallel. The underground ore is being blended with the Macraes ore and Reefton concentrate at the Macraes Process Plant.

Depending on ongoing ore definition drilling, the overall mine life may be extended if the open pit contribution that ceases in 2017 can be supplemented to keep the processing costs economic.

21.1.3.2 Mining Method

The FRUG mine has evolved the RHLOS mining process from the initial trials undertaken in 2007. The experience obtained from extracting the 3 adjacent trail stopes suggested that a continuously retreating parallel set of stopes of 15m excavation width with 6m of pillar between the stope voids would permit enough time for ore extraction before yielding of the surrounding rock occurred. Reslotting frequency as a result of significant dilution was within the consultant's expectations and resulted in planning of pillar placement for the Panel 1, Panel 2 and Panel 2 Deeps mine designs.

During 2008 the Frasers Open Pit mined through and exposed the caved ground above the collapsed trial stope voids and in the process defined the degree of unravelling and permitted the geotechnical consultant's to calibrate their recommendations in predicting future rock displacement and standoff requirements for future underground infrastructure.

The RLHOS mining method has been site proven with the extraction of 492,200 tonnes of stope ore during 2008 and 544,300 tonnes of stope ore during 2009.

The Chevron stoping method, which is a hybrid form of room and pillar mining method has been recently site trialled in Panel 2 "C" block with good results. As a result of including Frasers Open Pit Stage 6 to LOMP09 the proposed Panel 1 blocks that were to utilise the Chevron stoping method will now be extracted by the open pit. There are no current reserves to be extracted by the Chevron stoping method. The conceptual mine design for Panel 2 Deeps includes a proportion of the mineralization being extracted by the Chevron stoping method but the extraction method may alter depending on the results of in-fill drilling.

The designs assume a mining recovery of around 65% of the in situ targeted mineralization. An alternate mining option described in the 2005 Technical Study considered inclusion of mining with paste fill, adopting a similar layout but increasing recovery by allowing a portion of the pillar material between the adjacent stopes to be recovered. Should additional resource be delinated down dip then the backfilling of voids option will need reassessment as part of the financial analysis component before adding to the reserve.

The Panel 1 ore geometry is flat-lying, at around 10-15°, averaging 15-20m thickness, and has been reassessed due to the inclusion of Frasers Open Pit Stage 6 which mines out most of the previous underground 1C and 1D sub-panels. Panel 1 mining is now considered final though there remains potential to scavenge some remnant infrastructure related stopes once Frasers Open Pit Stage 6 is complete and the damage to the adjacent Panel 1 infrastructure can be assessed for re-entry. This has not been factored into the LOMP09.

The Panel 2 mineralization tends to be thinner, but is also suited to the RLHOS mining method with a similar lead and lag mining front being utilised in the completed "2B" and "2C" stope blocks and the current "2D" stope block that began extraction in October 2009. Panel 2 designs are considered final for stope blocks "2B", "2C", "2D", "2E" and "2G" while stope blocks "2A" and most of "2F" are conceptually designed but are awaiting in-fill diamond drilling results before final designs are completed over the coming 12 months.

The Panel 2 Deeps mineralization tends to average 5m in thickness and extraction by the Chevron stoping method is currently proposed. Panel 2 designs are considered conceptual for the entire resource and are awaiting in-fill diamond drilling results before further conceptual and potentially final designs are completed over the coming 12 months.

21.1.3.3 Geotechnical Parameters

Over the previous 24 months of the FRUG operational life the technical services personnel have learned and adapted to the ground conditions. The use of shotcrete was widespread during the early stages of development but since the beginning of 2009 it was deemed unsuitable to the deforming ground conditions. The decision to change drive surface support to mesh continues to be a success with the mesh surface supports ability to withstand ground deformation better than the brittle aspects of shotcrete.

Structural conditions can be difficult at times with a pervasive east dipping fault set. These conditions are managed in development drives by use of a Ground Control Management Plan (GCMP). The GCMP is an easy to follow plan for what to do when poor ground conditions are encountered, and can be updated as experience is gained. The GCMP details the geotechnical engineer's recommendations for the various ground conditions and drive geometries. The recommendations include the types of bolts to be installed, the bolt spacing and location of placement, the type of surface support and the requirement for subsequent cable bolt installation and who to contact should the ground support installer encounter ground conditions outside of what is considered normal.

Stope production is carried out safely and efficiently by careful design of stope pillars and installation of long twin stranded grouted brow cables. Stope stability is managed by orientating ore drives at an angle to the pervasive fault set. This minimises structural effects on pillar and back stability.

The upper section of FRUG is situated beside the Frasers Open Pit. The open pit's wall relaxation causes ground movement in the main access decline which results in periodic rehabilitation of the declines surfaces though the installation of additional or replacement ground support. Monitoring prisms are located on the pit walls and within the access decline to monitor movement rates. Weekly inspections of the upper decline and scheduled rehabilitation days mitigate the risk to the underground workforce to the hazard of a ground support failure. Recently three 50m extensometors were installed to gain more data of the movement rates in this section of the decline.

Specialist geotechnical consultants are utilised on a regular basis to review installed ground support performance, to assess current ground support practices and predict the ground support requirements for future conceptual development drive and stope designs.

21.1.3.4 Production Forecasts

The production target of 2,200-2,400 tpd has been consistently achieved since mid 2008 after a slow start resulting from delays in interpreting the trial stope data and the resultant modifications to mine designs through consultant recommendations. The 770-880ktpa production forecasts derived through the use of Runge's Xpac scheduling software using time and rate constraints derived from the 2005 Technical Study and actual site achievements is considered reasonable given the relatively stable performance of the FRUG operation since 2008. The 2009 production was 906kt, 4% ahead of budget.

2010 is forecasting 2,900 to 3,000 tonnes per day.

21.1.3.5 Underground Mining

Gold production from the FRUG operation produces an average of 60 koz per annum until mid 2011. Total material movement detailed in LOMP09 which includes inferred material is around 1.1Mtpa in 2010 falling to 570 kt in 2011. The amended mine plan without inferred material for the remaining life of the Macraes FRUG operation is summarised in Table 21.19. Note: Full year figures are not detailed.

Table 21.19: Frasers Underground amended Mining Schedule 2010 to 2014

Item	Unit	2010	2011	2012	2013	2014	Total
Macraes FRUG Mining							
Ore Milled	Mt	1.10	0.57				1.67
Grade – Au	g/t	2.73	2.66				2.71
Recovery – Au	%	82.2	82.1				82.1
Gold Produced	koz	76	40				116

Potential exists to define additional mineable ore from material not in the current reserves, including underground stockwork zones and down-dip extensions of current ore regions that, depending on gold price, further infill drilling and mine planning may identify some of this material as mineable reserves.

Gold production shown in Table 21.19 is only the contribution of the Macraes FRUG operation to the combined output from the Macraes Process Plant, which includes ore and concentrates from other sources.

21.2 Processing

21.2.1 Ore Mineralogy

Gold is mostly present as <10 micron (µm) particles in sulphide grains, principally within pyrite and arsenopyrite. This gold is refractory and is not readily recoverable by standard cyanidation methods. It requires the sulphide grains to be broken down prior to cyanidation. This is accomplished using pressure oxidation in an autoclave. The sulphide component is concentrated into a sulphide flotation concentrate prior to pressure oxidation. Free gold content varies between deposits, comprising between 5-30% of the total gold in the ore.

The Macraes ore also contains a carbonaceous fraction. Coarse grained ores tend to contain less organic carbon, while finer grained ores contain higher levels of carbon. The carbonaceous material has a negative impact in the CIL circuit, adsorbing some of the dissolved gold from the CIL circuit liquor; this effect is not uncommon and is termed ‘preg-robbing’. The carbonaceous material is typically recovered to the flotation concentrate, although its flotation kinetics are slower than those of the sulphide minerals, so that carbon recovery is generally lower than sulphide recovery. The soft carbonaceous material also tends to smear on the gangue components of the ore, imparting some degree of hydrophobicity increasing the recovery of non-sulphides in the flotation concentrate. Experience at Macraes and at other plants worldwide indicates that the autoclave pressure oxidation under normal oxidising conditions tends to further activate the carbonaceous material. Macraes has adopted technology developed by Newmont Limited of the US that allows passivation of the carbonaceous material by introducing limestone into the feed to the autoclave. This, along with the use of kerosene in the CIL circuit and judicious management of the activated carbon in the CIL circuit has provided an effective means of controlling and mitigating the preg-robbing effect.

21.2.2 Plant Description

The Macraes Process Plant recovers gold by concentrating the metal into a relatively small fraction of flotation concentrate, oxidising the reground concentrate in a pressure oxidation autoclave, washing the oxidised residue and then utilising a carbon-in-leach process to recover gold from the residue. Figure 21.26 is a schematic diagram of the plant flow sheet following the 2007 Flotation Plant Upgrade.

In detail the plant comprises:

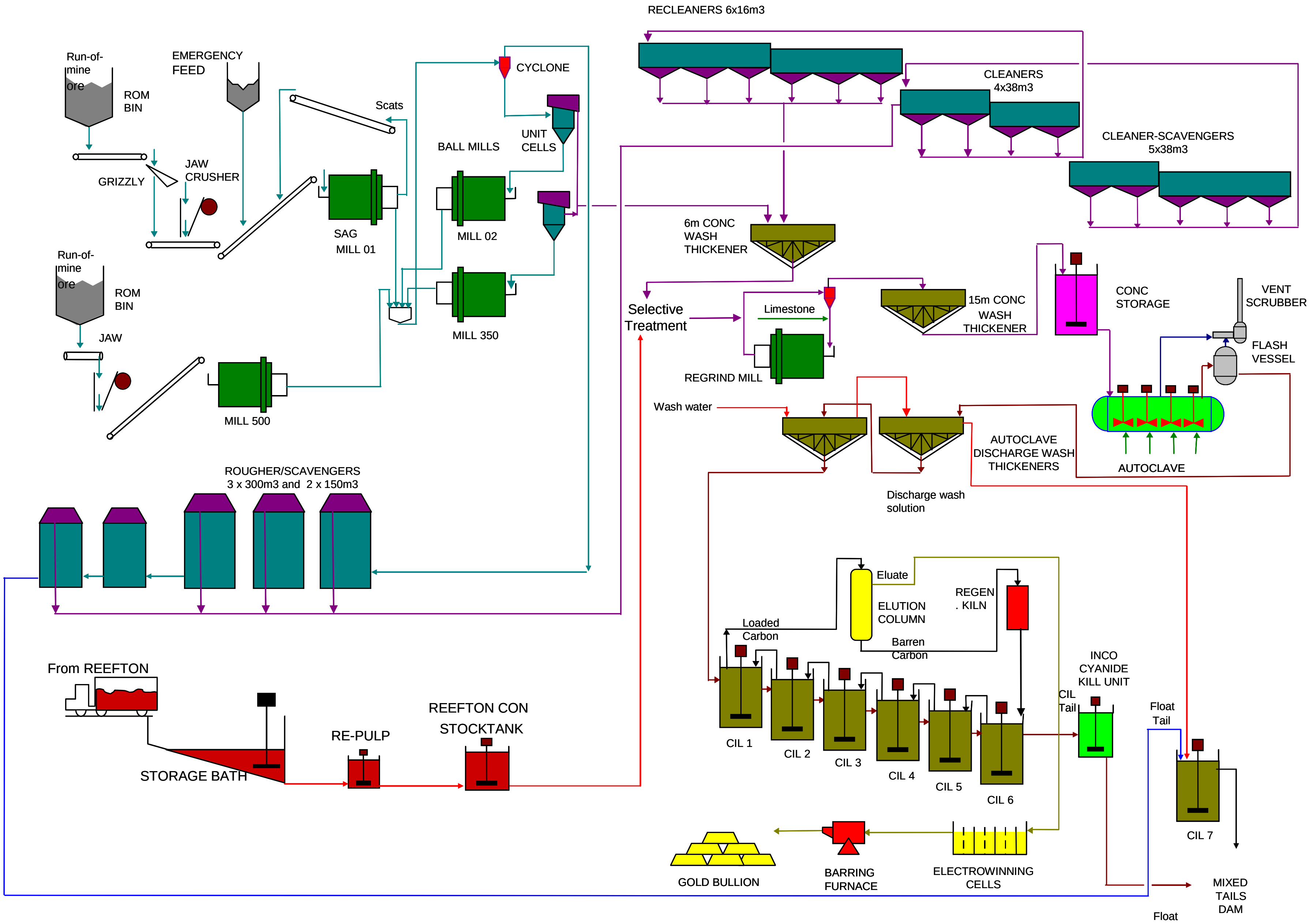
- two single stage jaw crushing circuits, which reduce the ore to a top size of approximately 200mm; the products from these two circuits are directly fed to the two SAG mills and an emergency feeder on the conveyor system feeding the higher capacity circuit provides continuity of feed to the grinding circuit if the jaw crusher feed is interrupted;

- a complex grinding circuit to reduce the particle size of the ore to 80% passing 140 µm; the original, higher capacity crushing circuit feeds a 2,300kW SAG mill and the new crushing circuit feeds a 1,500kW SAG mill; discharge from the two SAG mills are directed to two separate cyclone clusters, the underflow from which is fed to the flash flotation section or two parallel ball mills (2,300kW and 2,500kW); ball mill discharge is directed to the larger of the cyclone clusters;
- the flash flotation circuit, comprising both roughing and cleaning, on the circulating load of the grinding circuit, to recover as much of the sulphide minerals, and consequently the gold, at as coarse a size as possible;
- a main primary flotation circuit comprising tank cells and trough-style cells to maximise gold recovery to flotation concentrate; a cleaner flotation circuit controls to the sulphur grade of the concentrate to optimise the performance of the following pressure oxidation circuit;
- regrinding of the flotation concentrate in a 900kW ball mill to 80% passing 15 µm improves pressure oxidation kinetics; limestone is added to the regrind circuit feed to control net acid generation in the pressure oxidation circuit;
- washing and thickening of the reground flotation concentrate in a high rate thickener to control the level of chloride ion in the liquor fed to the pressure oxidation circuit;
- repulping and treatment of Reefton concentrate through a 450kW IsaMill, generating an 80% passing 20 µm sized product;
- pressure oxidation in a 77 m3 autoclave at 3,150kPa and 225°C to achieve greater than 75% oxidation of the sulphide component of the Macraes concentrate; oxygen is supplied to the autoclaves from a cryogenic plant operated by BOC;
- pressure oxidation through the autoclave of Reefton concentrate at the same operating conditions as for Macraes concentrate, targeting greater than 88% oxidation;
- Macraes and Reefton concentrates treated separately, excess concentrate from either source results in a proportion of Macraes concentrate being sent to CIL unoxidised as direct leach (due to the higher recoveries achievable on unoxidised Macraes concentrate compared to unoxidised Reefton concentrate);
- washing of the oxidised residue from the pressure oxidation (POX) process to separate the acid liquor generated by the oxidation;
- further neutralisation of the acid liquor using flotation tailings and lime;
- leaching of gold from the POX residue using cyanide in a CIL circuit that uses kerosene and high activated carbon concentrations to control preg-robbing by the carbonaceous component in the ore;
- destruction of the cyanide in the CIL tailings using the INCO process, with sodium metabisulphite as a source of sulphur dioxide (SO2); and
- recovery of gold from the loaded carbon using a normal elution and single pass electrowinning circuit, followed by smelting to produce gold bullion.

The POX plant uses technology that minimises formation of gold chloride complexes in the autoclave. Formation of these soluble complexes in the presence of active carbon can result in preg-robbing prior to contact of the oxidised residue with cyanide. Washing the concentrate with water in a thickener controls chloride levels in the flotation concentrate pulp. The acidity of the autoclave pulp is controlled by the addition of limestone to the regrind. The sulphur oxidation was designed to about 97% of the total sulphide present but more recently test work has indicated that reduction in the level of oxidation to greater than 75% has enabled increased gold extractions in CIL to be achieved by increased throughput through the autoclave.

Commissioning of the POX circuit in 1999 proceeded with minimal disruption to plant operations and the process has been both a technical and economic success, being a large contributor to a significant reduction in cost per ounce of production.

Figure 21.26: Macraes Process Plant Flowsheet – Post 2007 Flotation Upgrade



In 2007 Oceana replaced the primary flotation columns and the 38m^3 trough-style flotation cells with three 300m^3 tank cells simplifying the rougher/scavenger flotation circuit and enabling the large conventional cells to be used to increase the capacity of the cleaner circuit.

The underground ore is being separately crushed and primary milled before being blended with open pit ore during the secondary grinding stage at the Macraes Process Plant.

21.3 Recoverability

Details are provided in section 16.

21.4 Contracts

21.4.1 Open Pit

21.4.1.1 Concentrating, Smelting, Refining, Transportation and Sales

Concentrating of the Macraes ore to produce a concentrate for further processing is part of the operation carried out by Oceana at the Macraes site and is not contracted out.

A contract is in place with AGR Matthey Pty Ltd for the transportation and refining of the dore bullion into fine gold and silver for sale. This contract sets prices for transporting and refining the dore under conditions which generally comply with industry norms. The contract was initially for a three year period ending in July 2006 and has subsequently rolled over annually on the same terms and conditions.

21.4.1.2 Mining

Open pit mining at Macraes is carried out by Oceana personnel using mining equipment leased by Oceana under master lease agreements with ANZ National Bank Limited, Caterpillar Financial New Zealand Limited and other individual operating lease agreements. Gough, Gough and Hamer Limited maintains the mining equipment under an agreement that is in place through to January 2013 with the exception of one Hitachi 3600 excavator, which is maintained by Oceana personnel.

Tyres for rubber-tyred mobile mining equipment are sourced directly from local suppliers Tyreline Distributors Ltd (Michelin brand) and Bridgestone Firestone New Zealand Limited with a minimum number of Michelin brand tyres secured by a long-term supply agreement.

The supply and mixing of explosives for mining is provided by Orica New Zealand Limited under an expired contract which is under negotiation for a further term through to September 30, 2012. The scope of the proposed contract comprises the supply of explosive materials for both the open pit and the underground mines. Charging and blasting of blast holes is carried out by Oceana personnel.

All of the mining contracts in place and under negotiation are structured, and include terms and conditions and pricing arrangements, which comply or are expected to comply with industry norms.

21.4.1.3 Power Supply

Power is supplied under a base supply contract with Meridian Energy Limited which was entered into in October 2003 and is current until September 2013. This contract provides for supply at spot prices which have in the past fluctuated over a wide range. Oceana has also entered into a hedging agreement with Meridian which provides partial protection for Oceana against significant increases in the spot price. The terms of the hedging agreement are set out in a standard Master Agreement as published by the International Swap Dealers Association which is widely used throughout the power supply industry in New Zealand and internationally.

The supply and hedging agreements are based on standard documents used by large power consumers in New Zealand. The agreements are well documented and were freely negotiated with a substantial public utility to provide reasonable security of supply and pricing for the operation.

21.4.1.4 Water Supply

Water is supplied from the Taieri River, approximately 12km west of the site. Oceana owns water rights to pump water from the river provided its flow is above a minimum level and has entered into agreements with two local farmers for the use of the farmers' water entitlements in the event that the river flow falls below that specified. While the water supply has to be managed carefully in drought conditions, inability to pump from the river has never curtailed processing operations to date. The water rights agreement and the agreements with the farmers are in accordance with industry norms.

21.4.1.5 Engineering, Procurement and Construction

There are no significant construction or engineering works currently in progress at Macraes process plant.

Oceana has now fully commissioned and is operating the most recent projects, comprising the repulping and regrinding facility to prepare the concentrate from the Globe Progress operation for treatment in the autoclave at the Macraes Process Plant and three additional large flotation cells to improve overall recovery of gold.

21.4.2 Frasers Underground

21.4.2.1 Concentrating, Smelting, Refining, Transportation and Sales

Concentrating of the Frasers ore to produce a concentrate for further processing is part of the operation carried out by Oceana at the Macraes site and is not be contracted out.

The AGR Matthey Pty Ltd contract for the transportation and refining of the dore bullion into fine gold and silver for sale described in the Macraes section above applies to the production from Frasers ore.

21.4.2.2 Mining

Underground development and production mining at Frasers is carried out under an “Alliance” contract let to Byrnecut Mining Pty Limited. The current contract is in effect until 30 April 2012. Equipment for the mining contract is owned by Byrnecut or is being financed under equipment finance leases from the CBA Bank (NZ or Australia) to which Byrnecut is party. Oceana holds step-in rights under a tripartite deed with CBA NZ and rights to purchase major items of Byrnecut's equipment. A number of items of equipment are owned by Oceana or leased under operating leases with GE Finance and Insurance and ET Underground Solutions Pty Ltd. Maintenance of the mining equipment is included in the contract with Byrnecut. The terms of the mining contract and the finance leases are generally in accordance with current mining practice.

Orica New Zealand Limited supplies and mixes explosive materials under the arrangements described in the Macraes section above. Byrnecut performs the charging of blast holes.

21.4.2.3 Diamond Drilling

Boart Longyear (NZ) Ltd provide all underground diamond drilling services under a contract with a term that runs to 30 June 2011 with no fixed volume of work.

21.4.2.4 Power Supply

The power supply contract with Meridian described under Macraes above also applies to the power supply for the Frasers mining operation.

21.4.2.5 Water Supply

The water supply arrangements described under Macraes above also apply to the water supply for the Frasers mining operation.

21.4.2.6 Engineering, Procurement and Construction

Ore from the Frasers mine is fed to the current Macraes process plant.

21.4.3 Hedging and Forward Sales Contracts

The hedge book is split between delivery into a flat forward contract and an options collar contract with puts and calls. All hedge contracts are with three banks: CBA, ANZ and Societe Generale. Oceana will deliver into the hedges out of its total production from all operations.

The flat forward contract has an average price of NZ$773 and continues to December 2010. This contract comprises 99,840 remaining ounces (at December 31, 2009).

The options call contract is for 104,024 ounces at the end of December 2009, all of which could be called in 2010. The average price of these options is NZ$1,062 per ounce (at December 31, 2009). It is expected that these will be called by the banks over the course of 2010. There also remains under the options contract a total of 82,080 put ounces contracted for 2010 with an average price of NZ$1,000.

21.5 Environmental Considerations

21.5.1 Open Pit

Bonding can be required as a condition of the statutory approvals required to licence mining activities. The bonding levels are given below for the Macraes mines, noting the agencies to which the bond is in favour. The bonding levels are calculated on the basis of estimated costs to complete rehabilitation/remediation of mining activities. In particular, the RMA (section 108A) makes provision for a bond to be attached as a condition of resource consent "to secure the ongoing performance of conditions relating to long term effects". The bond may cover:

- structures;
- remedial, restoration, or maintenance work; and
- ongoing monitoring of long-term effects.

The bond requirement may continue beyond the term of the consent. Annual bond limit reviews may occur to account for the progressive nature of mine site development and operation with associated rehabilitation.

Bonding is assessed on an annual basis in conjunction with the next years Project Overview and Annual Work and Rehabilitation Plan. Bonds in place for the 2010 are:

- NZ$1,500,000 joint bond for site rehabilitation covering ML 32-3047. This bond is split between the Ministry of Economic Development (MED), the Otago Regional Council (ORC) and the Waitaki District Council (WDC). Expiry date October 30, 2010. This bond is a fixed quantum and is not subject to annual review.
- NZ$25,000 in favour MED covering rehabilitation of the pipeline corridor from Taieri River to ML32-3047. Expiry date October 30, 2010. This bond is a fixed quantum and is not subject to annual review.
- NZ$2,000,000 in favour of ORC for rehabilitation and monitoring within ML32-3047/MP41-064. Expiry date August 31, 2032. This bond is subject to annual review.
- NZ$2,825,000 in favour of ORC for tailings impoundment and open pit rehabilitation. Expiry date October 12, 2041. This bond is subject to annual review.
- NZ$7,125,000 in favour of WDC for land use rehabilitation. Expiry date March 31, 2011 and subject to annual review.

21.6 Taxes

21.6.1 Income taxes

The NZ company tax rate is 30%. NZ tax rules allow mining companies operating in NZ to claim an immediate tax deduction for mining related capital expenditure rather than an amount for tax depreciation. Consequently due to Oceana's high levels of capital expenditure in recent years the company has built up considerable tax losses.

21.6.2 Other Taxes

Fringe Benefit Tax (FBT) is a tax paid on non-cash benefits paid to staff such as motor vehicle and subsidized medical insurance. The rate paid by the employer on the benefit received by an employee is 61%.

See section 4.7 for information regarding royalties.

21.7 Capital and Operating Cost Estimates

21.7.1 Open Pit

21.7.1.1 Capital Expenditure Programme

A summary of forecast capital expenditure for the Macraes operation for the life of the mine is set out in Table 21.20.

Table 21.20: Macraes Capital Expenditure for Life of Mine (NZ$M)

Item	2010	2011	2012	2013	2014 - 2017	Total
Mine	43.8	27.4	60.5	37.0	56.2	224.9
Process Plant and Tailings	6.7	5.7	4.2	12.7	4.5	33.8
Other	1.0	1.3	7.1	2.0	0.7	12.1
Total	**51.5**	**34.4**	**71.8**	**51.7**	**5.2**	**270.8**

The programme covers the period from 2010 to 2017 and includes a range of expenditures which are:

- essential for the continuity of the operation;
- justified by improvement in the economics of the operation;
- required for regulatory compliance;
- required to maintain plant conditions at satisfactory levels; and
- closure costs, including site rehabilitation.

Based on historical performance, specific capital estimates are regarded as being reasonably reliable and accurate, based on Oceana experience of the equipment condition refurbishment requirements. Based on reviews of specific items in the various capital categories, the overall capital estimates for most refurbishment components are accurate within ±15% and carry a reasonable contingency provision.

21.7.1.2 Macraes Operating Cost Estimates

The projected mining operating costs for the Macraes open pit life of mining and processing operation have been derived from current costs, using known consumable and labour costs, ownership/lease costs and fleet maintenance costs. The plan includes the use of contract maintenance, tyre supply and on-site mixing and loading of bulk explosives, with performance-based incentives and/or penalties forming part of the contract terms. The picture is somewhat complicated by the fact the Macraes mill and plant will be treating ore from FRUG and concentrate from Reefton, and these costs have all been consolidated into one table later in this document to present a consolidated operating cost estimate for the three Oceana operations.

Since 1990, total mining costs have progressively reduced from NZ$1.90 per tonne of material moved to the current level for the full 2009 year of NZ$1.34 per tonne for all material moved, which is one of the lowest cost performances in Australasia. Oceana has maintained a focus on cost reductions, targeting maintenance, tyre costs, drilling, explosives and vendor guarantees. Oceana is very focused on cost efficiency in all phases of the operation and the mining costs projected for 2010 are soundly determined and achievable.

Macraes site processing costs were NZ$10.65 per tonne milled in 2009 which is higher than previous years due to the introduction of processing Reefton and FRUG ore. Oceana estimates its operating costs from first principles, using known manning levels, power, reagent and consumable usage rates and established maintenance spares costs, together with estimates of future salaries and prices for power, reagents, consumables and maintenance spares. Oceana maintains an aggressive approach to cost control with an active cost improvement programme. Power is probably the cost component which is least predictable, due to large fluctuations in the spot price which are related to storage dam levels in New Zealand's largely hydro-based electricity supply system. It is understood from previous experience at Macraes that spot power prices can vary between 4 and 14c/kWh or more, and as a consequence, Oceana hedges the price on the majority of its Macraes power supply.

Oceana process operating cost estimates for Macraes are likely to be accurate to within ±15% at the time they were completed.

Administration and environmental cost projections for 2010 are reasonably consistent with actual costs in 2009 and are considered achievable.

The Macraes operating costs, as they relate to the open pit mining and the processing operation, are presented in Table 21.21. It should be noted that the costs set out below include the processing costs of treating ore from the FRUG operation and concentrate from the Reefton mining and processing operation.

Table 21.21: Macraes Open Pit Operating Cost Schedule 2009 to 2017

Item	Unit	2009 Actual	2010 Plan	2011 Plan	2012 Plan	2013 Plan	2014 - 2017 Plan	Total 10 – 17
Unit Cash Cost Per oz Au Sold	NZ$/oz	599	730	842	622	971	899	839
	US$/oz	433	496	539	317	602	557	520

21.7.1.3 Combined Operating Costs – All Oceana Mining And Processing Operations

The operating costs are somewhat complicated by the fact that both the Macraes open pit and the FRUG ores are treated through the Macraes plant, as well as the concentrate from the Reefton operation. To put all this in perspective, Table 21.22 sets out operating cost forecasts for all of the Oceana operations for the period from 2010 to 2017, comparing these projections with the operating cost levels achieved in 2009.

Table 21.22: OGNZL Operating Cost 2009 to 2017

Item	Unit	2009 Actual	2010 Plan	2011 Plan	2012 Plan	2013 Plan	2014 - 2017 Plan	Total 10 – 17
Unit Cash Cost Per oz Au Sold-Consolidated	NZ$/oz	648	720	790	745	896	899	822
	US$/oz	468	489	505	461	555	557	510

21.7.1.4 Macraes Operating Costs

Site mining and processing costs have been estimated from first principles, using known manning levels, power, reagent and consumable usage rates and established maintenance spares costs, together with estimates of future salaries and prices for power, reagents, consumables and maintenance spares. The power cost is the least predictable, due to large fluctuations in the spot price. Oceana hedges the price on most of its Macraes power supply.

21.7.1.5 Frasers Underground Operating Costs

FRUG mining costs are discussed in detail in the later sections of this report. With respect to the processing costs, it should be noted that the process operating costs for FRUG are the costs of treating the FRUG ore through the Macraes plant. In terms of unit costs per tonne milled, these costs are very similar to the costs of processing Macraes open pit ores and have been estimated using identical methodology.

The FRUG project is integrated into the Macraes site operations and is not considered to have any administration costs which are separable from the administration costs of the open pit mining operation.

21.7.1.6 Reefton Operating Costs

Reefton mining costs are discussed in detail in the later sections of this report. Reefton process plant operating costs have been estimated in a similar manner to those for the Macraes process plant. Unit consumption of power, reagents and consumables has been estimated from test work data and from in-house knowledge and prices have been applied to the consumption rates.

21.7.2 Frasers Underground

21.7.2.1 Capital Expenditure Programme

Oceana has developed a capital expenditure programme for FRUG. Projected yearly expenditures are shown in Table 21.23.

Table 21.23: Frasers Underground Capital Expenditure Summary Schedule (NZ$M)

Item	2010	2011	2012	2013	2014	Total
Total	**22.6**	**8.2**	**0**	**0**	**0**	**30.8**

The programme covers the period from 2010 to 2011 and includes a range of expenditure which is:

- for underground capital development and initial fixed mine equipment;
- essential for the continuity of the operation;
- justified by improvement in the economics of the operation;
- required for regulatory compliance; and
- closure costs, including site rehabilitation.

The capital requirements for mine development have been estimated in great detail, and sustaining capital has also been estimated in reasonable detail but is likely to be subject to some adjustment as operating experience is gained by the operations personnel

The overall capital estimates for mine development and sustaining capital are accurate within ±15% and carry a reasonable contingency provision.

21.7.2.2 Operating Cost Estimates

The FRUG mining costs have been estimated based on the alliance contractor's budget estimates with an allowance for additional works outside contract. With this approach you would expect the cost estimates to be accurate within ±15%. Costs include the ongoing stope development but exclude capital development on permanent underground infrastructure and accesses.

The process operating costs for FRUG are the costs of treating the FRUG ore through the Macraes process plant. In terms of unit costs per tonne milled, these costs are very similar to the costs of processing Macraes open pit ores and have been estimated using identical methodology.

The FRUG project is integrated into the Macraes site operations and is not considered to have any administration costs which are separable from the administration costs of the open pit mining operation. Table 21.24 sets out operating cost forecast for the FRUG operations for the period from 2010 to 2011.

Table 21.24: Frasers Underground Operating Cost Schedule 2010 to 2014

Item	Unit	2009 Actual	2010 Plan	2011 Plan	2012 Plan	2013 Plan	2014 Plan	Total 10 - 14
Unit Cash Cost/oz Sold	NZ$/oz	701	485	534	0	0	0	**501**
	US$/oz	506	329	341	0	0	0	**331**

21.8 Economic Analyses

The projected net mine cash flows are shown net of operating costs, development capital, sustaining capital, reclamation costs, taxes, repayment of the external project debt and associated interest costs, exploration costs, and excluding any allowances for the potential to extend the mine life beyond current reserves.

21.8.1 Macraes Project

Table 21.25 below sets forth the net mine cash flows currently projected to be generated from the Macraes Project.

Table 21.25: Macraes Projected Net Cash Flow (NZ$'000)

	2010	2011	2012	2013	2014 – 2017	Total
Macraes Project	(11,052)	56,587	(30,306)	2,353	166,327	**183,909**
Gold Price in US$/oz	980	890	850	800	800	**841**

The forecast net mine cash flows for the period 2010 to 2014 from the Macraes Project (including both Macraes Open Pit and FRUG) are estimated to total NZ$184 million.

21.8.2 Open Pit

Table 21.26 below shows the projected net cash flow for the Macraes Open Pit mine. The project is expected to generate approximately NZ$136.2 million over its remaining mine life based on current reserves.

Table 21.26: Macraes Open Pit Baseline Net Cash Flow (NZ$'000)

Baseline Scenario	2010	2011	2012	2013	2014 - 2017	Total
Macraes	(32,850)	30,720	(30,306)	2,353	166,327	**136,244**

Table 21.27 below shows the sensitivity of the Macraes open pit net cash flow to variations in operating expenditure, capital expenditure and gold grade. The model demonstrates that the project is robust against a range of sensitivities applied to key operational parameters.

Table 21.27: Macraes Open Pit Sensitivity Analysis (NZ$'000)

Macraes Open Pit						
	2010	**2011**	**2012**	**2013**	**2014 -**	**Total**
Opex +10%	(45,803)	14,199	(44,316)	(13,121)	112,037	**22,342**
Opex – 10%	(19,896)	47,242	(16,295)	9,241	220,617	**250,149**
Capex +10%	(38,001)	27,279	(37,484)	(11,894)	160,189	**109,164**
Capex – 10%	(27,698)	34,162	(23,127)	(9,986)	172,465	**163,327**
Grade – 5%	(25,578)	41,355	(19,772)	(3,030)	201,087	**208,895**
Grade + 5%	(40,262)	19,636	(42,715)	(18,866)	131,500	**61,042**

21.8.3 Frasers Underground

Table 21.28 below shows the projected net cash flow for the FRUG mine. The project is expected to generate approximately NZ$47.7 million based on current reserves.

Table 21.28: Frasers Underground Baseline Net Cash Flow (NZ$'000)

Baseline Scenario	2010	2011	2012	2013	2014	Total
FRUG	21,798	25,867	0	0	0	**47,665**

Table 21.29 below shows the sensitivity of the FRUG mine's net cash flows to variations in operating expenditure, capital expenditure and gold grade.

The cash flow model sensitivities shown relate only to reserves (that portion of the resource that has been drilled to an approximate 50 by 50m pattern, which is classified as Indicated). While there are additional potentially minable Inferred resources adjoining the reserves, there is insufficient drilling at present to provide accurate grade and tonnage estimates to the standards required for reserves definition.

Table 21.29: Frasers Underground Sensitivity Analysis (NZ$'000)

Frasers Underground	2010	2011	2012	2013	2014	Total
Opex +15%	9,743	18,369	0	0	0	**28,112**
Opex -15%	33,853	33,364	0	0	0	**67,218**
Capex +15%	18,412	24,638	0	0	0	**43,050**
Capex -15%	25,184	27,095	0	0	0	**52,279**
Grade +5%	26,031	28,663	0	0	0	**54,694**
Grade -5%	17,551	23,099	0	0	0	**40,650**

21.9 Payback

The Macraes open pits have been operating for nearly two decades and are currently running at an operating surplus. The underground operation had a maximum cash draw-down in early 2008 and since then has been cash-flow positive with pay-back on spot gold occurring in late 2009.

21.10 Mine Life

The projected mine life based on defined mineral reserves for the Macraes open pit operation extends to 2017. Current work, based on gold prices of NZ$1,333 is anticipated to extend mine life.

The projected mine life (including Inferred production) of the Macraes underground operation is 2013. Additional geological drilling of the known adjacent mineralization is being undertaken and, if additional reserves are subsequently defined and the economics of operating in parallel with an open-cut ore supply to cover overheads can be justified then the underground mine life may be extended.

21.11 Exploration Potential

21.11.1 Summary

Future exploration will continue to focus along the strike extent of the HMSZ and subsidiary Hangingwall or splay structures that may contain significant mineralization. The exploration targets are direct analogues of the mineralization that has been previously exploited by open pit and underground mining methods.

The near surface part of the HMSZ has been well tested in the vicinity of the main open pit mines and consequently the greatest geological prospectively along the HMSZ is down-dip. Concurrent open pit and underground mining operations are likely to continue, with a longer term trend towards underground targets over time. Exploration for semi-concealed deposits (analogous to discovery of the Frasers deposit) and underground targets analogous to discovery of the FRUG deposit will be continuously refined as the key geological parameters that control the distribution of mineralized "shoots" at a mine scale are better understood and incorporated into the exploration model.

21.11.2 Open Pit Mineralization

The acquisition of the Macraes North tenements in 2002 has provided additional along-strike exploration potential. The northern tenement covers 7.5km of the HMSZ, which has been only sparsely explored in recent years. Mineralization is known to occur at a number of sites including Nunns, Coronation North, Mareburn and Trimbels. Oceana's exploration has outlined a small reserve at Coronation and further drilling is planned.

The discovery of potentially mineable underground mineralization down dip at Frasers opens up the possibility of extensions of mineralization down dip of some of the northern pits.

Stockwork mineralization below the Hangingwall shear is not always present, and where it does occur, it is generally discontinuous. Given however, its proximity to the Hangingwall shear, stockwork provides an opportunity to increase reserves for a small additional increment of waste stripping.

21.11.3 Underground Mineralization

A number of broadly defined exploration opportunities that may be exploitable from an underground operation have been identified. These areas will be targeted over the next few years. These include:

- down dip extension of Panel 2 which is currently open at depth;
- potential Panel 3 mineralization down dip of Panel 2;
- narrow (5m) higher grade (2.5-3.0 g/t Au) along the hangingwall of Panel 1 and Panel 2; and
- extensions of the Round Hill shoot (Round Hill East) where high grade mineralization has been intersected in drill holes.

Figure 21.27: Macraes Open Pit Target Areas

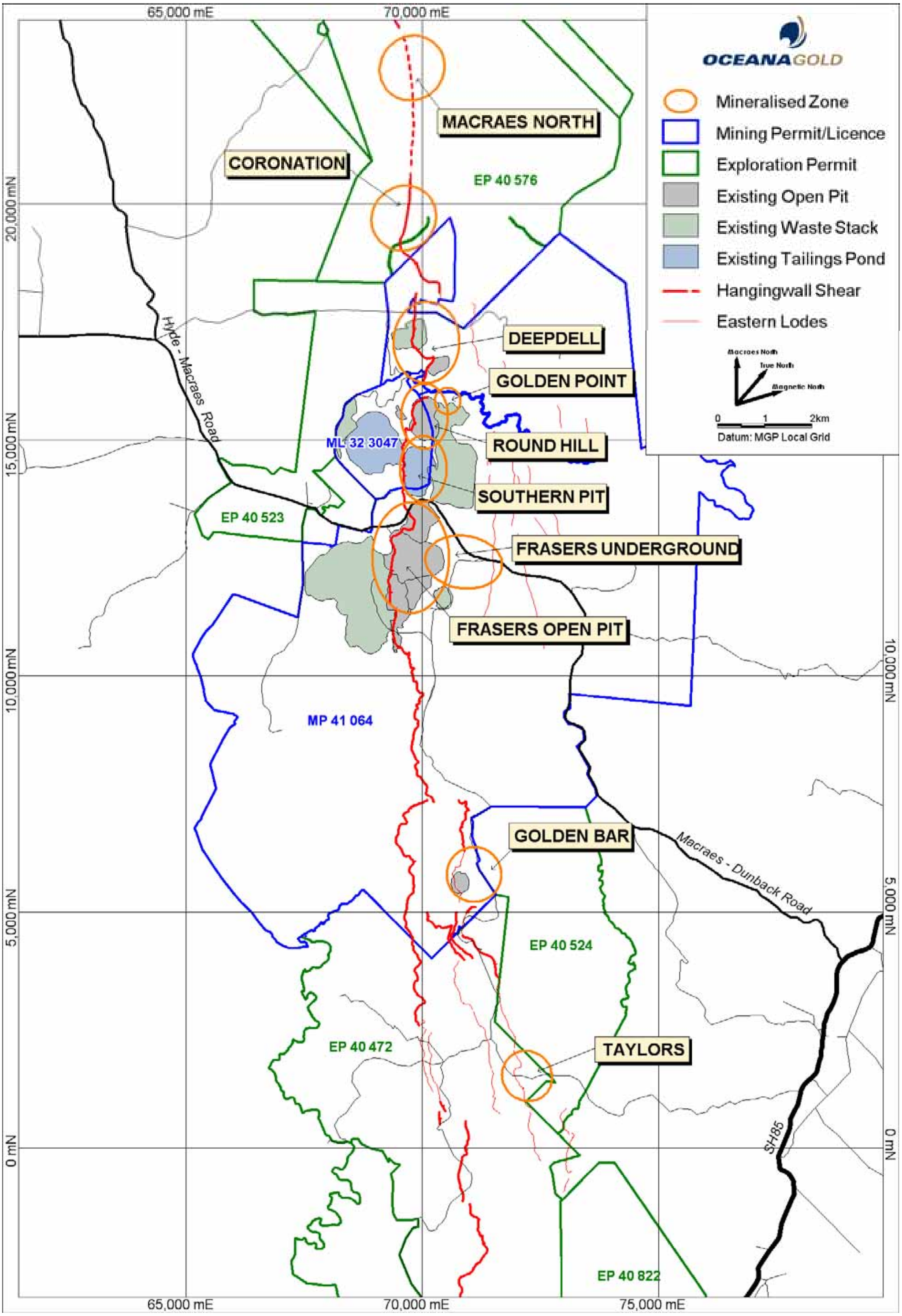
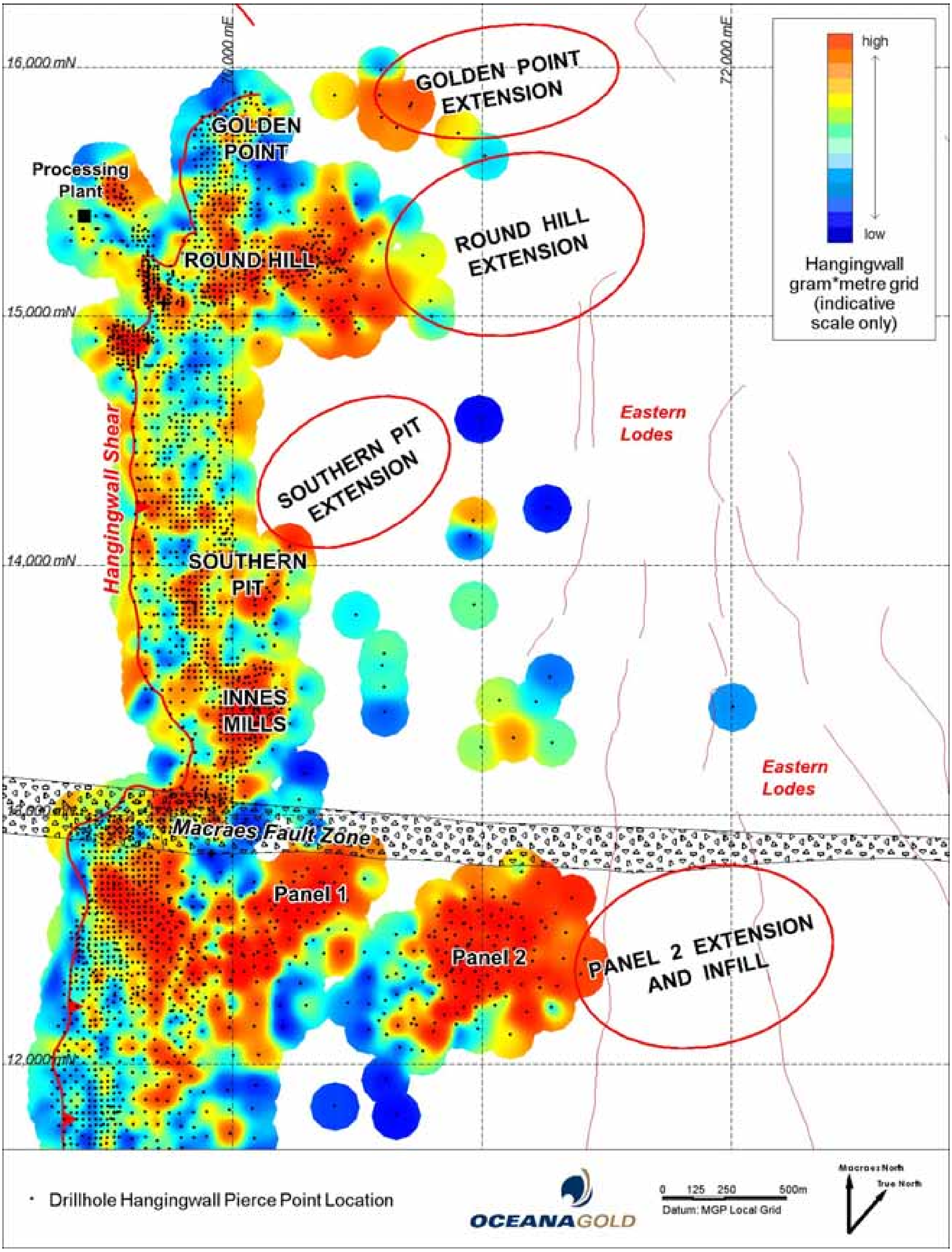


Figure 21.28: Macraes Underground Target Areas



Note: Background indicative grid summarizes mineralisation adjacent to the Hangingwall Shear only.

22 GLOSSARY

>	greater than
<	less than
=	equal
%	percent
±	plus or minus
‘	feet
#	mesh
$	dollars
^{o}	degrees
^{o}C	degrees Celsius
Ωm	ohm metres
3D	three dimensional
AMPRD	Absolute Mean Paired Relative Difference
As	arsenic
Au	gold
AusIMM	Australasian Institute of Mining and Metallurgy
AWRP	Annual Work and Rehabilitation Programme
BDA	Behre Dolbear Australia Proprietary Limited
BHP	BHP Limited
BLEG	bulk leach extractable gold
BOC	BOC Limited
ccdf	conditional cumulative distribution function
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMA	Crown Minerals Act
CMS	Cavity Measuring System, survey tool to survey underground voids
CS	conditional simulation
CSAMT	controlled source audio frequency magneto-tellurics
CV	coefficient of variation
DD	diamond drilling
DDH	diamond drill hole
Decline	An inclined tunnel that permits vehicular access to the economic portion/s of an underground mineral resource
DIGHEM	digital helicopter electromagnetics
DOSLI	Department of Survey and Land Information
Drive	Underground mines general term for a tunnel
E	east

ea	each
EM	electromagnetic
EMS	Environment Management Strategy
EP	Exploration Permit
FBT	fringe benefit tax
FRUG	Frasers Underground
GCMP	Ground Control Management Plan
GHD	GHD Limited
gcm^{-3}	grams per cubic centimetre
g/t	grams per tonne
GPS	global positioning system
GRD	Gold Resource Development
HARD	Half Absolute Relative Difference
H&S	Hellman & Schofield Pty Ltd
HEM	helicopter electromagnetics
HMSZ	Hyde Macraes Shear Zone
HNZEL	Homestake New Zealand Exploration Limited
HRD	Half Relative Difference
HW	hangingwall
Hz	hertz
ICP	inductively coupled plasma
ICP-MS	inductively coupled plasma mass spectroscopy
ICP-OES	inductively coupled plasma optical emission spectroscopy
IGNS	Institute of Geological and Nuclear Sciences
IK	Indicator Kriging
INCO	proprietary INCO metallurgical process
IP	induced potential
IsaMill	proprietary Mt Isa mill technology
JORC	Joint Ore Reserves Committee
km	kilometre
km^2	square kilometre
koz	thousand ounces
kPa	kilopascal
kt	Thousand tonnes
kW	kilowatt
l	litre
LHD	Load-Haul-Dump, specialised underground front end loader
LOM	Life of Mine
LOMP08	Life of Mine Plan completed in late 2008, relating to following years

LOMP09	Life of Mine Plan completed in late 2009 (supercedes LOM08), relating to following years
LOTEM	long offset time domain electromagnetics
ls	lode schist
m	metre
m^3	cubic metre
M	million
Ma	million years
MARC	Maintenance and Repairs Contract
MED	Ministry of Economic Development
medIK	median Indicator Kriging
MFZ	Macraes Fault Zone
MIK	multiple Indicator Kriging
ML	Mining Lease
MLOS	Macraes Line of Strike
mm	millimetre
µm	micron / micometre
MMCL	Macraes Mining Company Limited
MMI	Mobile Metal Ion
MOED	Ministry of Economic Development
Moz	million ounces
MP	Mining Permit
MPRD	Mean Paired Relative Difference
mE	metres East
mN	Metres North
mRL	metres Relative Level
Mt	million tonnes
Mtpa	million tonnes per annum
N	north
NZ	New Zealand
NZ$	New Zealand dollar
NZMG	New Zealand Map Grid
Oceana	Oceana Gold (New Zealand) Limited
OK	Ordinary Kriging
ORC	Otago Regional Council
oz	ounce
ppb	parts per billion
ppm	parts per million
POX	pressure oxidation

Q-Q	Quantile-Quantile
QAQC	quality assurance, quality control
RC	reverse circulation
RCD	diamond drill hole with percussion pre-collar
RL	relative level
RLHOS	retreat long hole open stope
RMA	Resource Management Act 1991
RQD	rock quality designation
RSG	RSG Global Consulting Pty Ltd
SAG	semi-autogenous grinding
S	sulphur
S^{2-}	sulphide sulphur
SO_2	sulphur dioxide
t	tonnes
TEM	time domain EM soundings
The 2009 Report	Technical Report for the Macraes Project, November 9th, 2009
TSP	total suspended particulate
TSFF	total sediment fine fraction
US	United States of America
US$	United States of America dollar
UHW	Upper hangingwall
W	tungsten
WO_3	tungsten oxide
WDC	Waitaki District Council

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24 APPENDIX

24.1 Exploration Drill Data

RSG Global has reviewed a series of ASCII files of quality control data provided by Oceana which relates to AMDEL assaying only. The data available for review comprised certified standards, blanks, duplicates and field duplicates.

A brief discussion of the investigation is provided below.

24.1.1 Standards

The standards database available for review comprises 439 Gannet Standards assays. The source and details of these data is not well documented which lessens the value of the subsequent review. Summary statistics of the standards data is presented in Table 14.1.

Substantial mixing of standards is noted. Where appropriate, RSG Global has excluded the clearly incorrectly assigned standards from the investigation.

When considering all the standards data (including the mixed data), 384 (87%) of the 439 standards are within a ±10% accuracy range. When mixed standards are excluded, 89% of the standards are within a ±10% accuracy range.

No systematic bias is noted in the data, with the majority of standards displaying a relative bias measure of less than 5%, although notable exceptions do occur such as ST11 and ST19.

In general, RSG Global concludes that no systematic bias exists, although it is apparent that no reasonable monitoring and follow up of the exploration assay quality is completed. Any batch which includes an out of range standard assay should be flagged and reassay completed prior to accepting the data.

Figure 24.1: Control Plot – Standard ST10

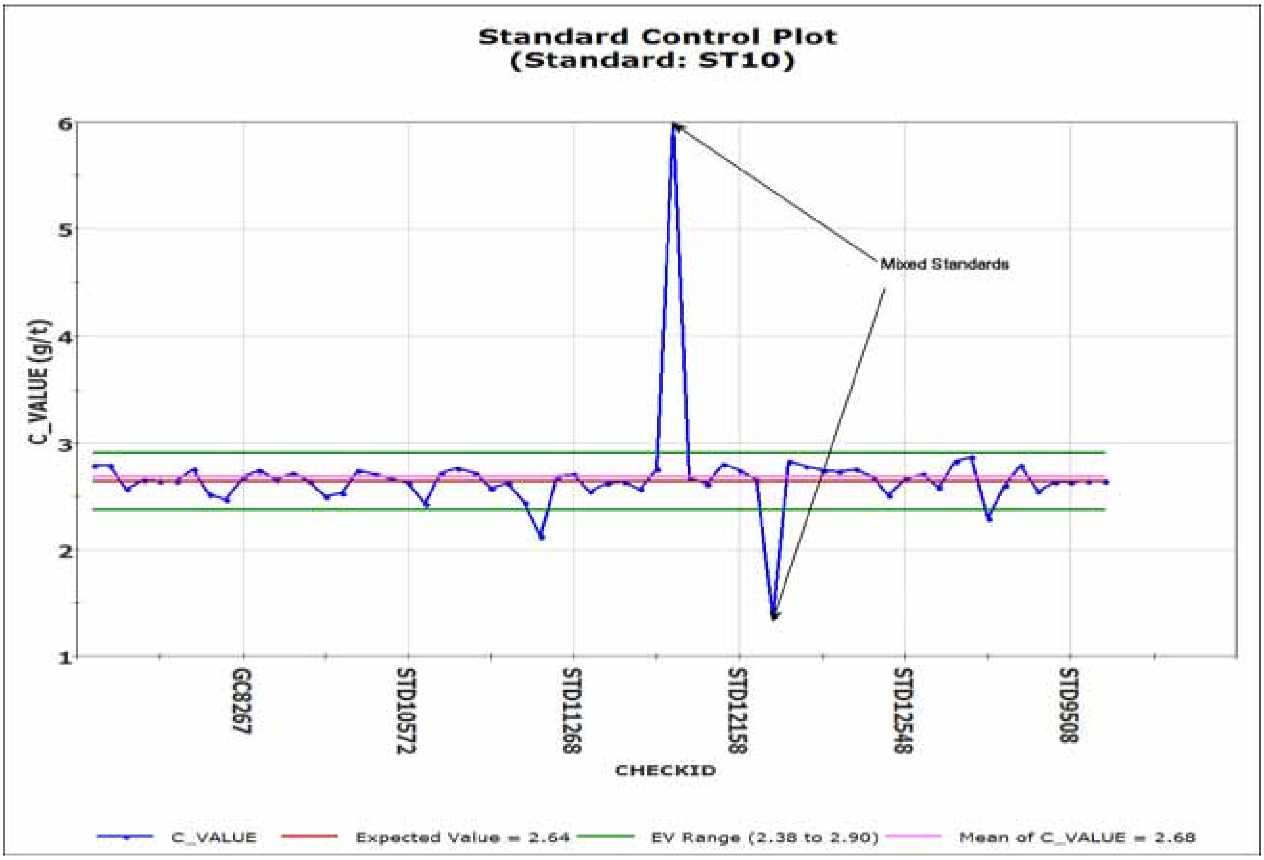

Table 24.1: Macraes Operation – Summary of Certified Standards

Standard	ST10	ST11	ST12	ST13	ST19	ST19 (Mixed Standards Removed <1.4 g/t)
Expected Value:	2.64	4.82	0.82	1.32	0.58	0.58
Expected Value Range:	2.38 to 2.90	4.34 to 5.30	0.74 to 0.90	1.18 to 1.45	0.52 to 0.64	0.52 to 0.64
Count	62	28	55	80	76	75
Minimum	1.37	0.58	0.64	0.32	0.53	0.53
Maximum	5.98	5.29	0.94	6.51	2.68	0.73
Mean	2.68	4.25	0.83	1.4	0.65	0.62
Std Deviation	0.47	1.41	0.04	0.61	0.24	0.04
% in Tolerance	93.55%	78.57%	94.55%	86.25%	76.32%	77.33%
% Bias	1.50%	-11.89%	0.80%	6.62%	10.93%	6.28%
% RSD	17.50%	33.17%	5.19%	43.57%	36.78%	6.19%
Standard	**ST20**	**ST20 (Mixed Standards Removed >3.0 g/t)**	**ST22**	**ST23**	**ST42**	**ST205**
Expected Value:	5.91	5.91	2.6	0.83	1.37	0.41
Expected Value Range:	5.32 to 6.50	5.32 to 6.50	2.34 to 2.86	0.75 to 0.91	1.23 to 1.51	0.37 to 0.45
Count	41	39	6	6	27	38
Minimum	1.26	5.6	1.92	0.76	1.27	0.35
Maximum	6.24	6.24	2.74	0.85	1.48	0.54
Mean	5.69	5.91	2.49	0.81	1.35	0.43
Std Deviation	1.01	0.14	0.27	0.03	0.05	0.03
% in Tolerance	95.12%	100.00%	83.33%	100.00%	100.00%	84.21%
% Bias	-3.78%	0.04%	-4.25%	-2.73%	-1.38%	4.17%
% RSD	17.70%	2.40%	10.64%	3.38%	3.86%	7.20%
Standard	**ST274**	**ST274 (Mixed Standards Removed >4.0 g/t)**				
Expected Value:	6.27	6.27				
Expected Value Range:	5.64 to 6.90	5.64 to 6.90				
Count	20	17				
Minimum	1.74	5.46				
Maximum	6.88	6.88				
Mean	5.66	6.22				
Std Deviation	1.37	0.34				
% in Tolerance	80.00%	94.12%				
% Bias	-9.67%	-0.87%				
% RSD	24.27%	5.40%				

24.1.2 Laboratory Repeats

Laboratory repeats (duplicate 50g samples of the sample pulp collected after pulverisation) have been assessed. The following data is presented as quality control statistics:

- Figure 24.2: Quality Control Statistics – All Gold Data – Laboratory Repeats
- Figure 24.3: Quality Control Statistics – Diamond Drilling – Gold Assay (Au g/t) Laboratory Repeats
- Figure 24.4: Quality Control Statistics – RC Percussion Drilling – Gold Assay (Au g/t) Laboratory Repeats
- Figure 24.5: Quality Control Statistics – All Sulphur Data (S %) – Laboratory Repeats
- Figure 24.6: Quality Control Statistics – All Arsenic Data (As ppm) – Laboratory Repeats

Very good correlation is noted for the gold data with the linear correlation generally >0.97. No apparent bias is evident with the mean HRD calculated as -0.13 for the entire gold data set. No difference is noted for the laboratory repeats for the data grouped by sample type (diamond and percussion drilling). Acceptable levels of precision are noted for the gold repeats as shown by the mean HARD of 7.70.

The sulphur and arsenic repeat data set also show strong correlation between the original and repeat assay (r= 0.93 for sulphur and r=0.97 for arsenic). As with gold, no bias is noted in these data sets and the majority of data is ±20%.

Figure 24.2: Quality Control Statistics – All Gold Data – Laboratory Repeats

MGM Summary - Lab Repeats (Gold Only)

	VALUE	C_VALUE	Units			Result
No. Pairs:	13,779	13,779			Pearson CC:	0.98
Minimum:	0.11	0.11	g/t		Spearman CC:	0.97
Maximum:	120.00	125.00	g/t		Mean HARD:	7.70
Mean:	0.97	0.97	g/t		Median HARD:	4.90
Median	0.43	0.44	g/t			
Std. Deviation:	2.23	2.17	g/t		Mean HRD:	-0.13
Coefficient of Variation:	2.30	2.24			Median HRD	0.00

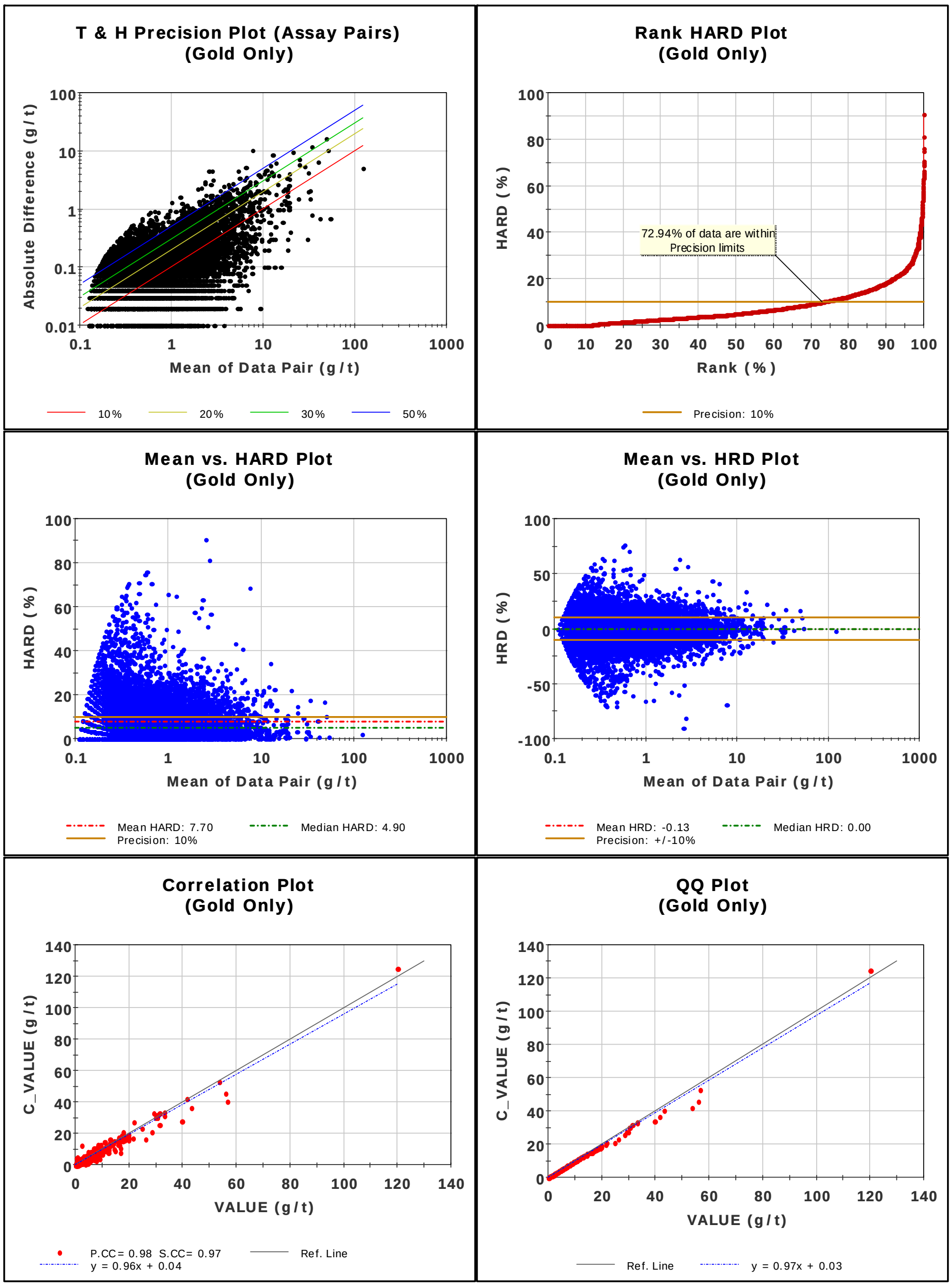

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Figure 24.3: Quality Control Statistics – Diamond Drilling – Gold Assay (Au g/t) Laboratory Repeats

MGM Summary - Lab Repeats (Diamond (Au))

	VALUE	C_VALUE	Units			Result
No. Pairs:	1,944	1,944			Pearson CC:	0.99
Minimum:	0.11	0.11	g/t		Spearman CC:	0.98
Maximum:	120.00	125.00	g/t		Mean HARD:	7.51
Mean:	1.26	1.26	g/t		Median HARD:	4.74
Median	0.52	0.52	g/t			
Std. Deviation:	3.75	3.77	g/t		Mean HRD:	0.08
Coefficient of Variation:	2.98	2.99			Median HRD	0.00

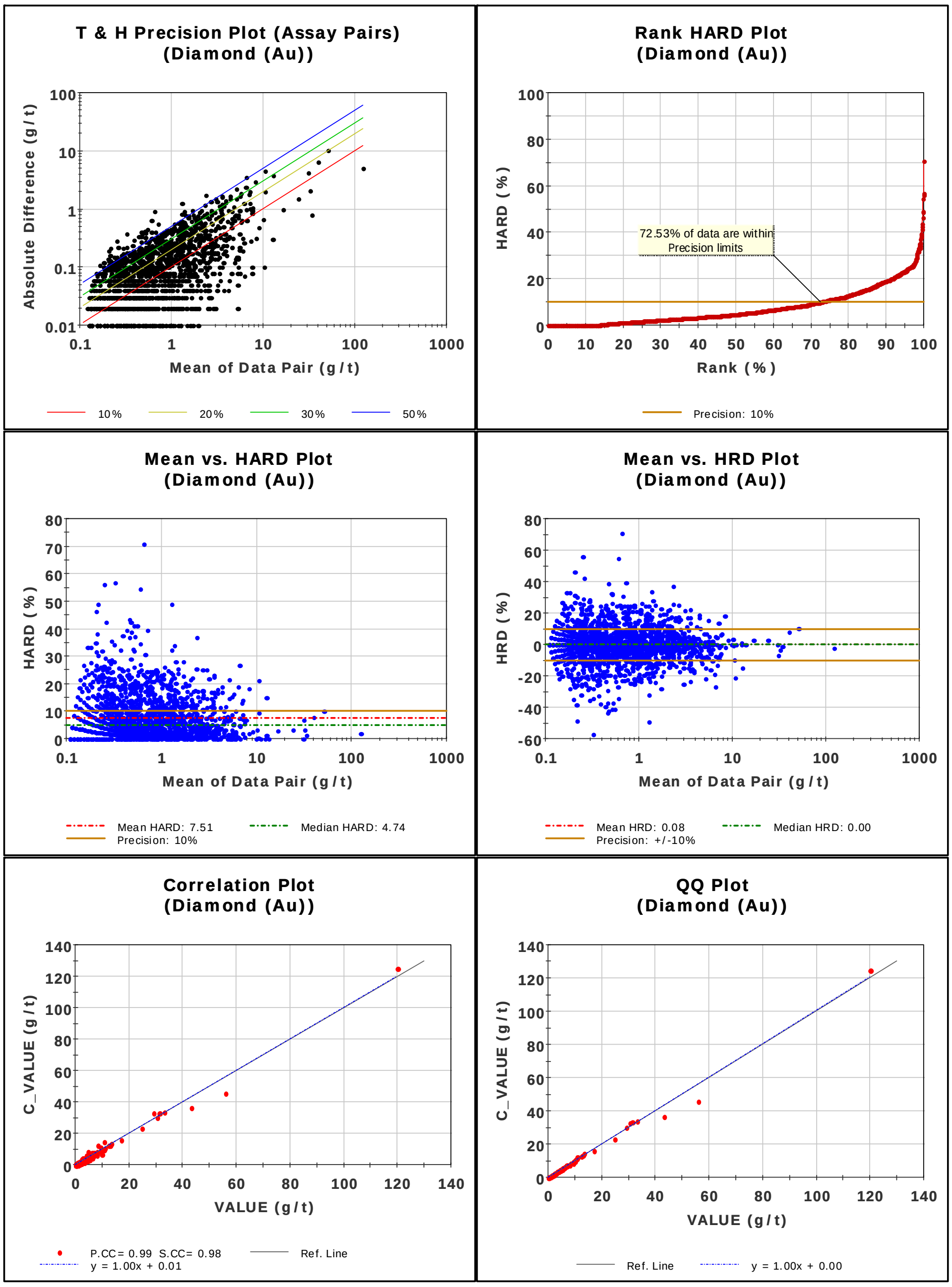

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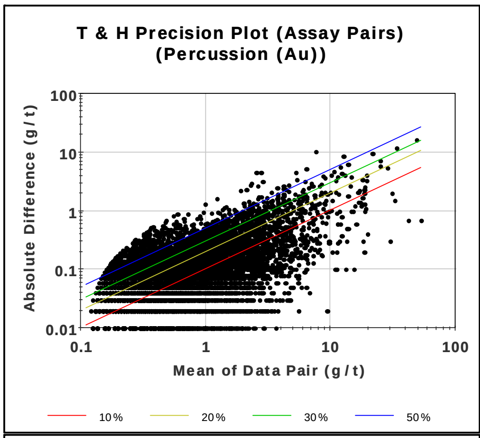
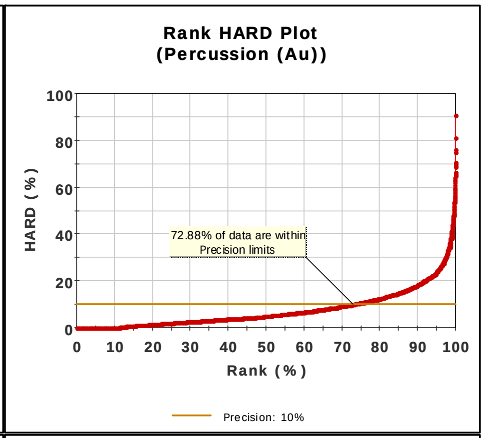
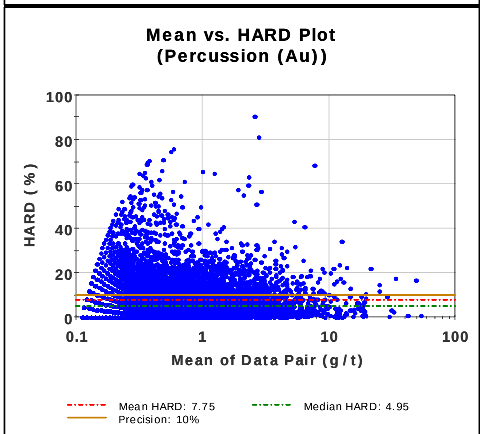
Figure 24.4: Quality Control Statistics – RC Percussion Drilling – Gold Assay (Au g/t) Laboratory Repeats

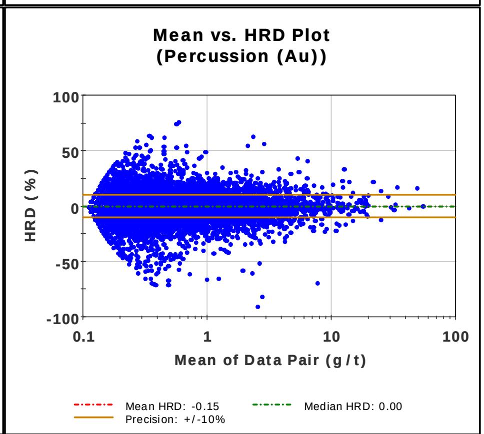
MGM Summary - Lab Repeats (Percussion (Au))

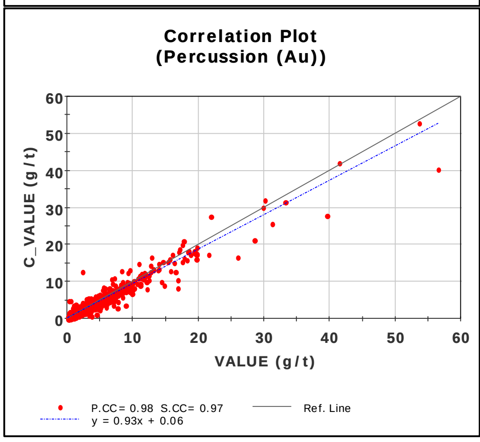
	VALUE	C_VALUE	Units			Result
No. Pairs:	11,665	11,665			Pearson CC:	0.98
Minimum:	0.11	0.11	g/t		Spearman CC:	0.97
Maximum:	56.60	53.00	g/t		Mean HARD:	7.75
Mean:	0.92	0.92	g/t		Median HARD:	4.95
Median	0.42	0.43	g/t			
Std. Deviation:	1.87	1.77	g/t		Mean HRD:	-0.15
Coefficient of Variation:	2.02	1.93			Median HRD	0.00











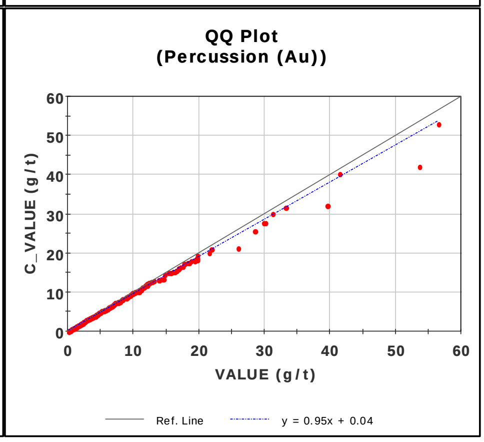


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Figure 24.5: Quality Control Statistics – All Sulphur Data (S %) – Laboratory Repeats

MGM Summary - Lab Repeats (Sulphur)

	VALUE	C_VALUE	Units			Result
No. Pairs:	1,251	1,251			Pearson CC:	0.93
Minimum:	0.11	0.11	g/t		Spearman CC:	0.97
Maximum:	1.66	2.34	g/t		Mean HARD:	3.73
Mean:	0.32	0.32	g/t		Median HARD:	2.04
Median	0.24	0.24	g/t			
Std. Deviation:	0.24	0.25	g/t		Mean HRD:	-0.35
Coefficient of Variation:	0.76	0.79			Median HRD	0.00

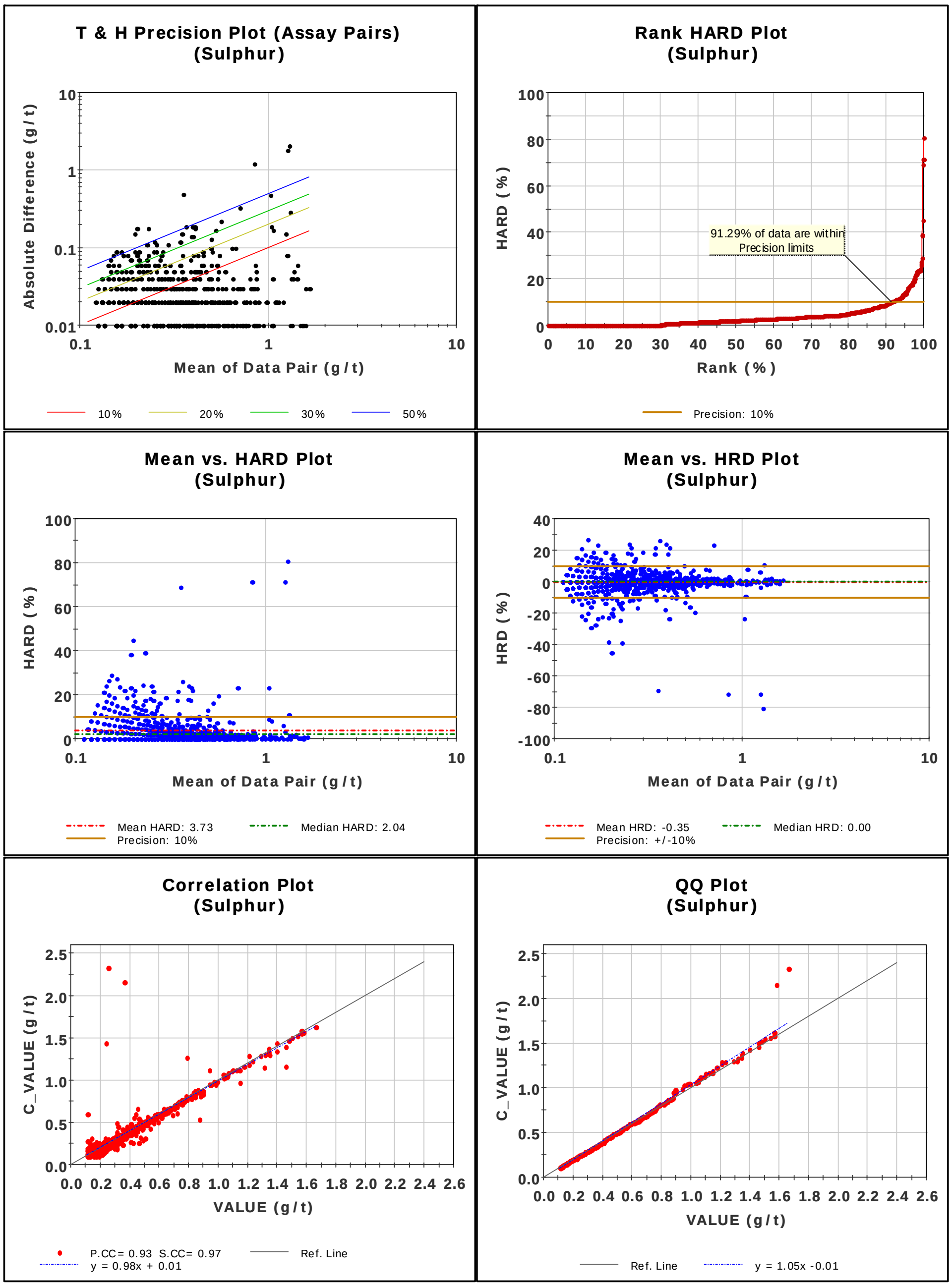

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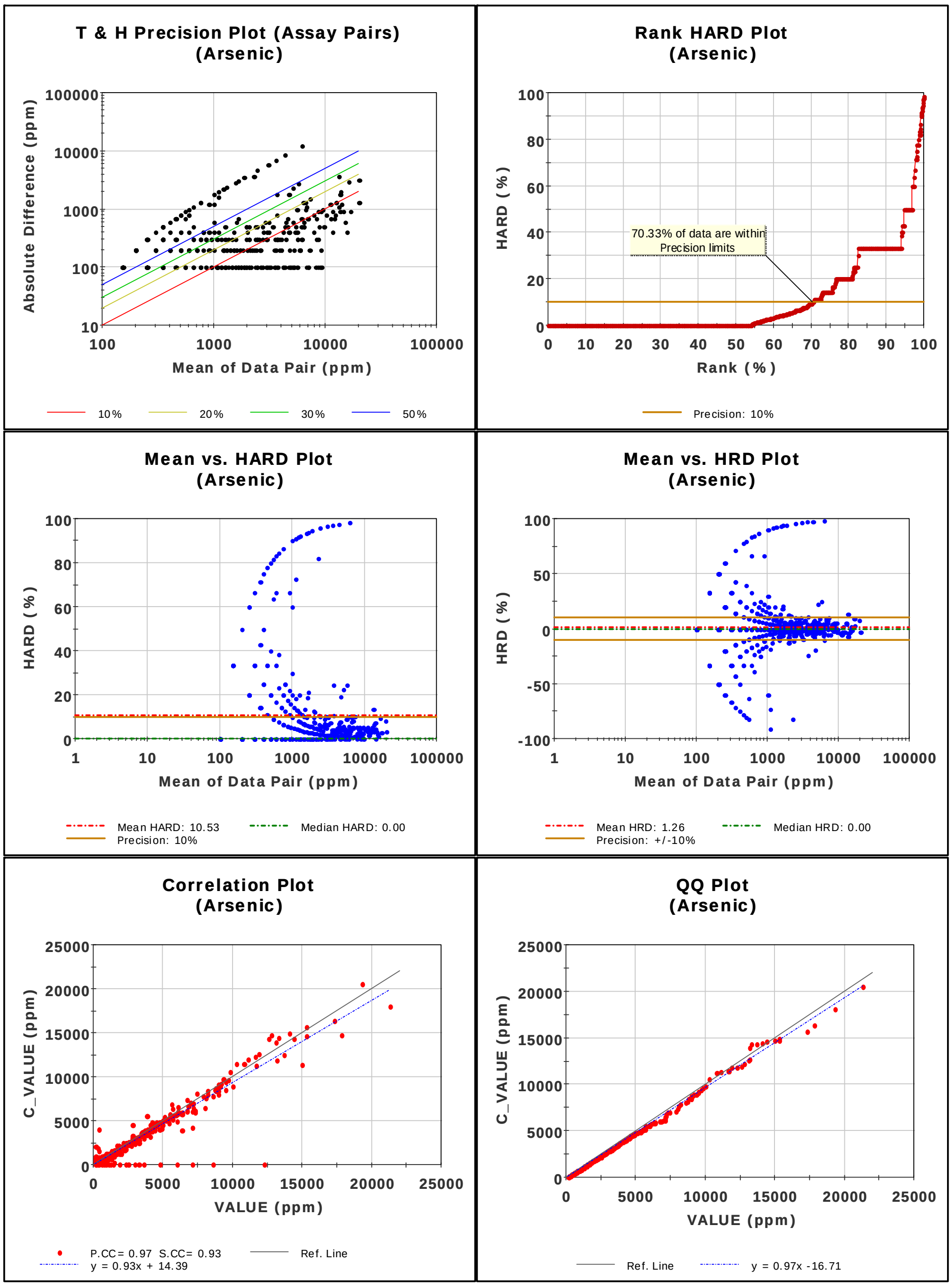
Figure 24.6: Quality Control Statistics – All Arsenic Data (As ppm) – Laboratory Repeats

MGM Summary - Lab Repeats (Arsenic)

	VALUE	C_VALUE	Units			Result
No. Pairs:	1,601	1,601			Pearson CC:	0.97
Minimum:	100.00	100.00	ppm		Spearman CC:	0.93
Maximum:	21,300.00	20,600.00	ppm		Mean HARD:	10.53
Mean:	994.25	943.54	ppm		Median HARD:	0.00
Median	200.00	200.00	ppm			
Std. Deviation:	2,248.36	2,173.87	ppm		Mean HRD:	1.26
Coefficient of Variation:	2.26	2.30			Median HRD	0.00



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24.1.3 Field Duplicates

Field duplicates represent a second sample collected at the RC percussion drill rig. These field duplicates are then submitted for assay using the same analytical approach as the original sample and provide a measure of the total error including sampling error.

A limited field duplicate data set is available. The gold data (Figure 24.7) shows the duplicate samples have reasonably reproduced the original assays with the linear correlation coefficient calculated at 0.92. Only 52% of the data is within ±10% precision (HARD) while approximately 80% of the data are ±20%. This relatively poor precision is interpreted to indicate a high sampling error component. RSG Global strongly recommends this be reviewed and appropriate strategies are put in place to improve sampling.

Relative few sulphur data are available to assess. Based on the available data, the field duplicates indicate an acceptable level of precision is being achieved in sampling when only sulphur is considered. Figure 24.8 displays the quality control charts for sulphur.

Figure 24.7: Quality Control Statistics – All Gold Data – Field Duplicates

Field Duplicates
(Au)

	VALUE	C_VALUE	Units			Result
No. Pairs:	98	98			Pearson CC:	0.92
Minimum:	0.14	0.11	g/t		Spearman CC:	0.87
Maximum:	5.44	5.61	g/t		Mean HARD:	14.24
Mean:	0.97	0.91	g/t		Median HARD:	9.70
Median	0.64	0.53	g/t			
Std. Deviation:	0.93	0.92	g/t		Mean HRD:	4.38
Coefficient of Variation:	0.95	1.01			Median HRD	2.61

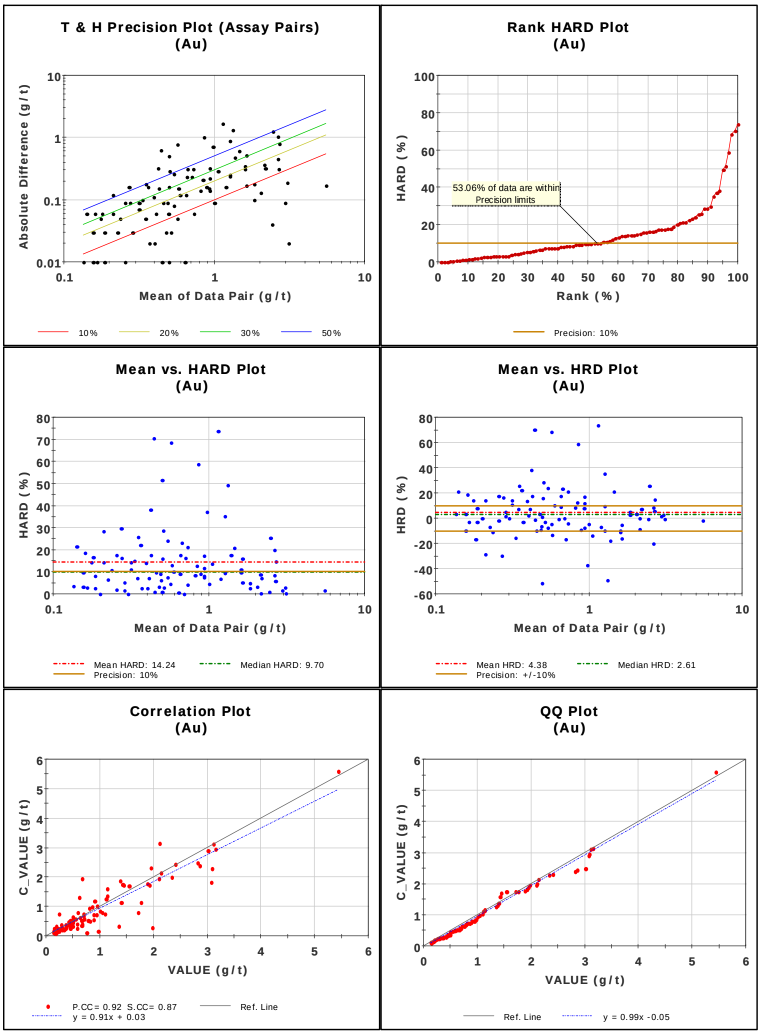

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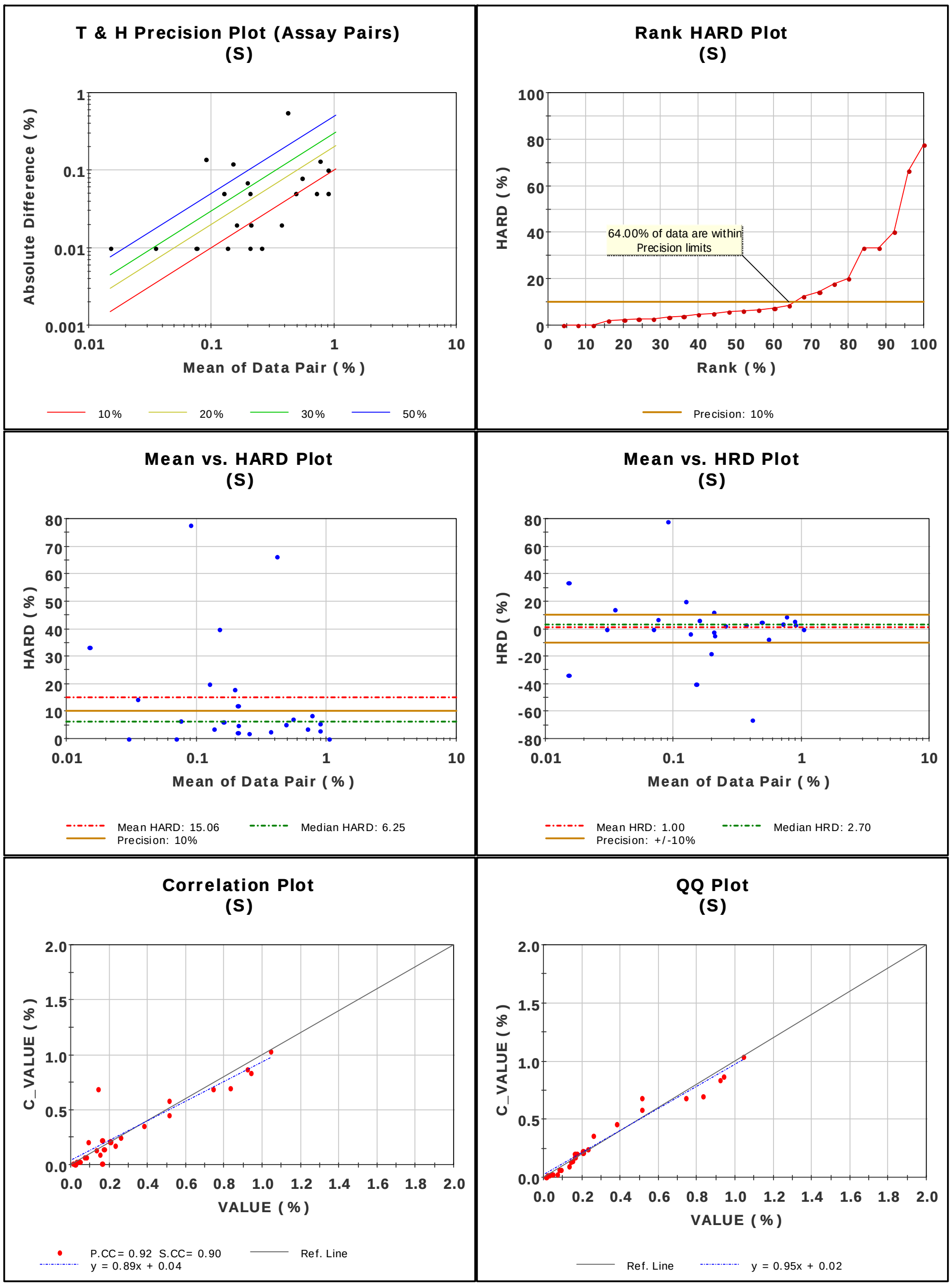
Figure 24.8: Quality Control Statistics – All Sulphur Data – Field Duplicates

Field Duplicates
(S)

	VALUE	C_VALUE	Units			Result
No. Pairs:	25	25			Pearson CC:	0.92
Minimum:	0.01	0.01	%		Spearman CC:	0.90
Maximum:	1.04	1.04	%		Mean HARD:	15.06
Mean:	0.32	0.33	%		Median HARD:	6.25
Median	0.17	0.21	%			
Std. Deviation:	0.32	0.31	%		Mean HRD:	1.00
Coefficient of Variation:	0.99	0.94			Median HRD	2.70



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24.2 Mining Department Data

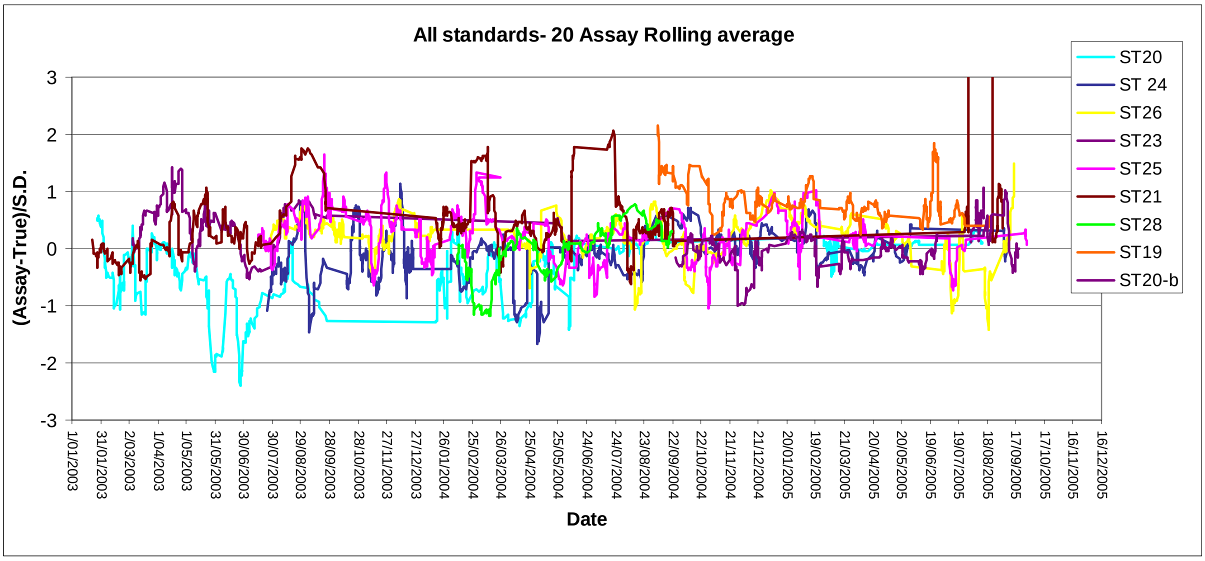
The Macraes mining department monitors the AMDEL laboratory assay quality on an ongoing basis. RSG Global therefore reviewed a series of quality control data files sourced from the Macraes Operation mining department whilst on site in November 2005. The data available for review comprised certified standards, assay blanks, duplicates and field duplicates.

A brief summary of the data and assessment is provided below.

24.2.1 Standards

Consistent with exploration data presented above, the standards data are generally well within the ±10% and 2 standard deviations accuracy limits targeted. Figure 24.9 presents the standards data as function of standard deviation from the certified value.

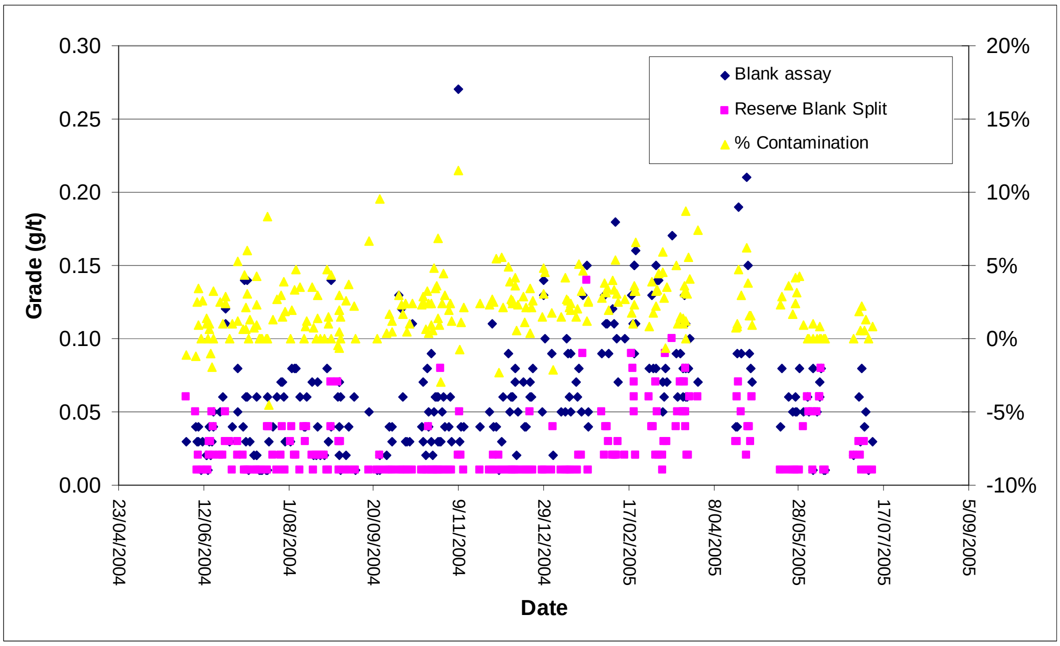
Figure 24.9: Mining Data – Control Plot of Gold Standards



24.2.2 Blanks

As highlighted in Figure 24.10, the majority of the blank assays data return less than 5% relative contamination. While little contamination is present in these data, some primary and reserve blank blanks exceed a 0.1 g/t Au grade threshold which represents 10 times the detection limit.

Figure 24.10: Mining Data – Control Plot of Blanks (Au g/t)



24.2.3 Laboratory Repeats

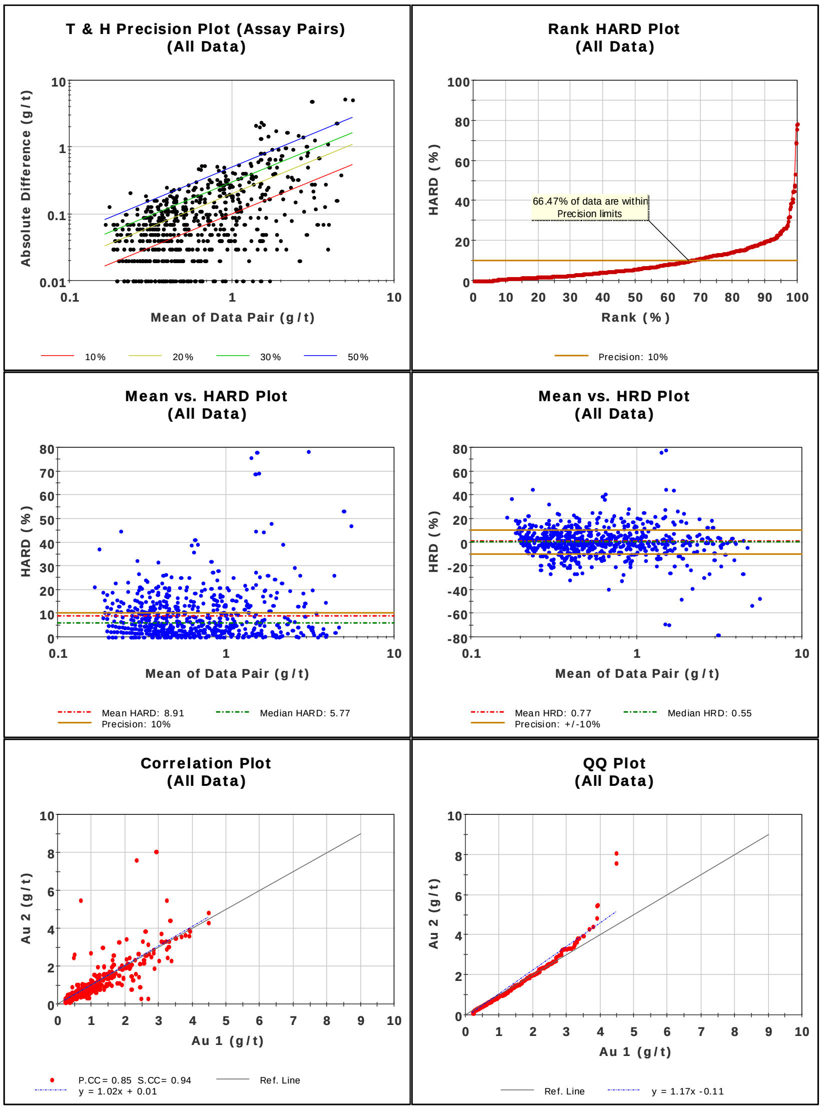
The mining department laboratory gold repeats have been assessed as presented in Figure 24.11. High levels of correlation is noted between the data pairs (r=0.85) with a mean HARD of 8.91 indicating acceptable levels of precision is returned. It is of note that the marginal distributions between the data sets are different (see QQ Plot) which is due to a small number of duplicate assays being significantly higher than the original. It is entirely reasonable to suggest that this is due to mixing of the sample numbers/data although RSG Global suggests these data warrant further investigation.

In general, this analysis supports findings of the exploration data set investigations. RSG Global concludes the laboratory repeats indicate acceptable levels of precision are being achieved in the AMDEL laboratory.

Figure 24.11: Mining Data – Laboratory Repeats (Au g/t)

Laboratory Repeats
(All Data)

	Au 1	Au 2	Units			Result
No. Pairs:	677	677			Pearson CC:	0.85
Minimum:	0.20	0.11	g/t		Spearman CC:	0.94
Maximum:	4.48	8.10	g/t		Mean HARD:	8.91
Mean:	0.79	0.81	g/t		Median HARD:	5.77
Median	0.50	0.49	g/t			
Std. Deviation:	0.73	0.88	g/t		Mean HRD:	0.77
Coefficient of Variation:	0.93	1.08			Median HRD	0.55



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24.2.4 Field Duplicates

RSG Global elected not to review the field duplicate data as the sampling approach is different to that completed for the percussion drilling (exploration and resource development). Therefore the sampling error associated with the grade control drilling (blastholes) will be vastly different from that associated with the percussion drilling.

24.3 Quality Control Investigation Summary

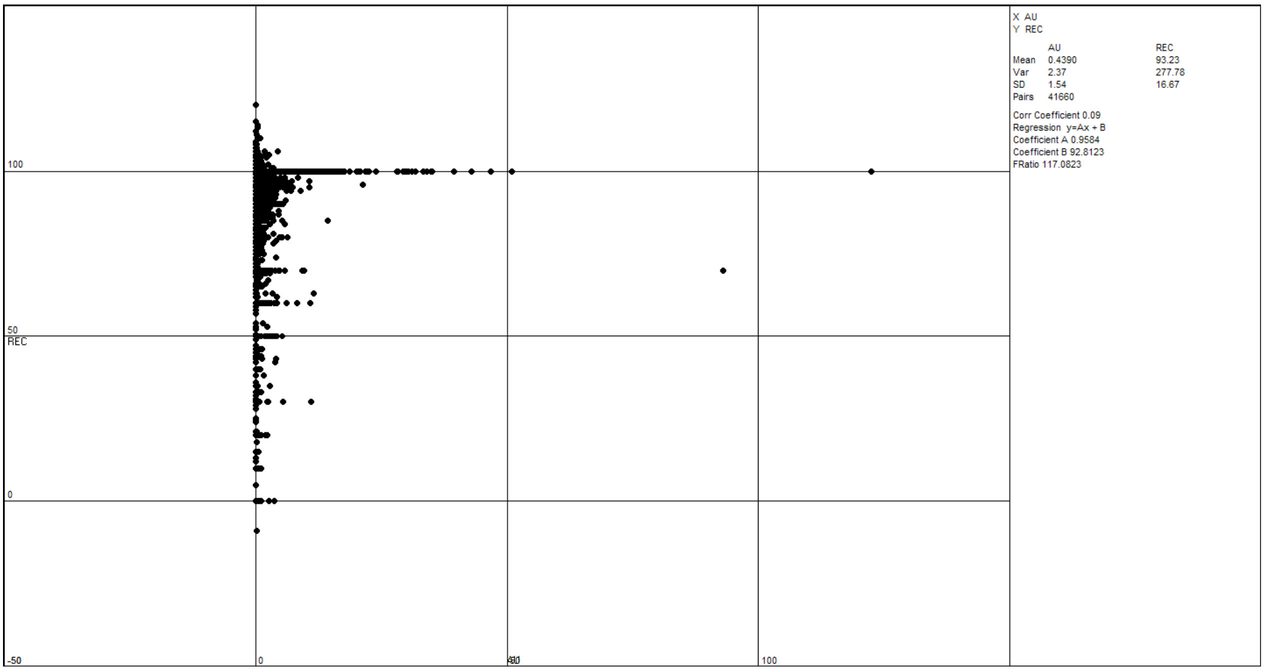
Based on the quality control database assessed by RSG Global, acceptable levels of assay precision and accuracy are generally being achieved by AMDEL. The conclusion is supported by the available reconciliation data.

24.4 Recovery

As described above, a portion of the Macraes database (pre 1994) contains insufficient data to assess the recovery achieved in either the diamond or RC percussion drilling. However enough data is available to assess the recovery trends from the Frasers deposit.

High diamond core recoveries were visually noted by RSG Global during the site visit. The available recovery data supports this observation with the average recovery for the diamond drilling calculated as above 90%. Furthermore there is no relationship between drilling recovery and gold grade, as illustrated in Figure 24.12.

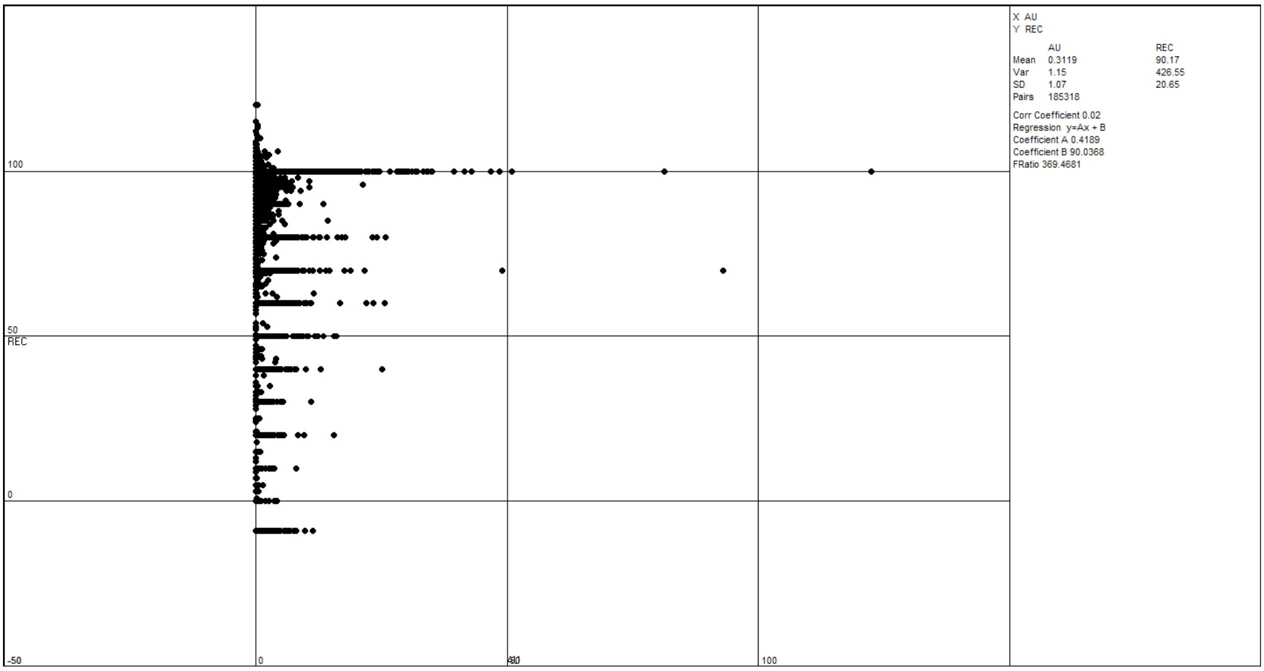
Figure 24.12: Scatter Plot Comparing Gold versus Recovery (Diamond Drilling)



The sample recovery for the percussion drilling has been visually estimated by Oceana technical personnel. This data is presented as a scatter plot of recovery against gold in Figure 24.13. Similar to the diamond drilling, no relationship is evident between gold grade and recovery. However, it is also evident that the estimate of recovery is qualitative and not always reliable, with recoveries of above 100% and below 0% common. In addition, precision steps in the recovery estimates are also noted. The average percussion recoveries are approximately 90% which also supports the low data confidence observation as percussion drilling recovery is generally unlikely to exceed 85% on an ongoing basis.

Despite shortcomings in the recovery estimates, no relationship exists between the recovery and gold grade.

Figure 24.13: Scatter Plot Comparing Gold versus Recovery (RC Percussion Drilling)



24.5 Summary

Due to the long exploration and mining history of the project, the quality control database is incomplete for the Macraes Project making complete and thorough investigation impossible. The risk associated with the incomplete quality control data set is offset by the available mining and reconciliation data which supports the quality of the data.

Notwithstanding the limitations in the data set, the available recovery and quality assurance, quality control (QAQC) data indicates the assay data meets acceptable limits of accuracy and precision and is therefore suitable for the purposes of grade estimation. The bias associated with the wet RC percussion drilling remains a material item and while Oceana have taken steps to mitigate the risks associated with this data set, ultimately only removal of this data can ensure no negative effects in the grade estimates. Additional drilling is likely to be required at the Frasers Open Pit mine where significant amounts of wet RC percussion drilling impact the depth extensions of the resource model.

In addition to the assay data, the survey data both collar and down-the-hole survey, is considered to be robust and present little risk.

25 TECHNICAL REPORT CERTIFICATION AND SIGN OFF

The effective date of this Technical Report and sign off is February 12, 2009.

Rodney Thomas REDDEN

Date of Signature: February 12, 2010

Jonathan Godfrey MOORE

Date of Signature: February 12, 2010

CERTIFICATE OF AUTHOR

As a qualified person responsible for the report titled "Technical Report for the Macraes Project" dated February 12, 2010, (the "Technical Report") to which this certificate applies, I, Rodney Thomas Redden do hereby certify that:

1. I, Rodney Thomas Redden, am the Exploration and Development Manager of Oceana Gold (New Zealand) Limited. My business address is OceanaGold, Taunton Mews, 22 Maclaggan Street, Dunedin, New Zealand.

2. I graduated with a B.E (Mining), Hons degree from the University of Wollongong, Australia in 1994. Subsequently I received a postgraduate Master of Business Administration also from the University of Wollongong in 2005.

3. I am the holder of a 1st Class Mine Managers Certificate for New Zealand and Western Australia

4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy.

5. I have worked as an open pit and underground engineer in various countries and metals in the mining industry and more recently as a manager of various aspects of mining operations continuously for a total of 16 years since my graduation.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. Since my employment with Oceana in 2004-2005 and 2007 to current I have either been site based or visit the site on a regular basis.

8. I am responsible for sections 1.6, 1.7.2 to 1.7.8, 4.7, 4.8, 5, 16, 17.13, 17.14, 17.16, 19.2 to 19.8, 20.2, 20.3 and 21.1 to 21.10 of the Technical Report.

9. I am not independent of OceanaGold Corporation applying all the tests in section 1.4 of NI 43-101 because I am an employee of Oceana Gold (New Zealand) Limited.

10. Prior to my employment with Oceana in November 2004 I had no involvement with the Macraes Project.

11. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.

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

Rodney Thomas REDDEN

Date of Signature: February 12, 2010

CERTIFICATE OF AUTHOR

As a qualified person responsible for the report titled "Technical Report for the Macraes Project" dated February 12, 2010, (the "Technical Report") to which this certificate applies, I, Jonathan Godfrey Moore do hereby certify that:

1. I, Jonathan Godfrey Moore, am the Principal Resource Geologist for Oceana. My business address is OceanaGold, Taunton Mews, 22 Maclaggan Street, Dunedin, New Zealand.
2. I graduated with a BSc (Hons) Mining degree in geology from the University of Otago in 1985 and a Graduate Diploma (Physics) in 1993 also from the University of Otago.
3. I am a member in good standing of the Australasian Institute of Mining and Metallurgy.
4. I have worked as a geologist in the mining industry for a total of 20 years since my graduation.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. My most recent personal inspection of the Macraes Project was in January 2010.
7. I am responsible for sections 1.1 to 1.5, 1.7.1, 2, 3, 4.1 to 4.6, 6 to 15, 17.1 to 17.12.7, 17.15, 18, 19.1, 20.1, 21.11, 22, 23, 24 and 25 of the Technical Report.
8. I am not independent of OceanaGold Corporation applying all the tests in section 1.4 of NI 43-101 because I am an employee of Oceana Gold (New Zealand) Limited.
9. Prior to my employment with Oceana in May 1996 I had no involvement with the Macraes Project.
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Jonathan Godfrey MOORE

Date of Signature: February 12, 2010