Preliminary Economic Assessment

of the

Blackwater Gold Project

Reefton, Westland Province, New Zealand

S0031-REP-069-0

Report Date:	October 21st, 2014
Effective Date:	October 21st, 2014

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DATE AND SIGNATURE PAGE

The effective date of this technical report is October 21st, 2014

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IMPORTANT NOTICE

The Preliminary Economic Assessment (PEA) has been completed on a production target which is based solely on an Inferred Mineral Resource. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised. The stated production target is based on the company's current expectations of future results or events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment will be realized.

The PEA enables OceanaGold to make an informed decision on the investment in an exploration decline and associated infrastructure at Blackwater. The exploration decline is required to enable resource drilling to improve geological and geotechnical confidence.

The report was prepared by OceanaGold with sections of the report contributed by Mining Plus, Golder Associates and Gekko Systems (collectively the Consulting Firms). The quality of information, conclusions, and estimates contained within the contributor-prepared sections is consistent with the level of effort involved in the contributors' various services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. The report is intended for use by OceanaGold subject to the terms and conditions of its contracts with the Consulting Firms. Those contracts permit OceanaGold to file this report as a Preliminary Economic Assessment with the Canadian Securities Regulatory Authorities pursuant to provincial and territorial securities law. Any other use of, or reliance on, the contributor-prepared sections of this report by any third party is at that party's sole risk.

The PEA includes (as Appendix 1 to this report) a report for the Blackwater Inferred Resource, dated October 21st, 2014, prepared in accordance with JORC Code, 2012 Edition ("JORC Table 1"). The JORC Table 1 has been prepared by or under the supervision of J.G.Moore, Chief Geologist, a full-time employee of the Company's subsidiary, Oceana Gold (New Zealand) Limited at the time of writing. Mr Moore is a Member and Chartered professional with the Australasian Institute of Mining and Metallurgy and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the JORC Code. As a senior employee of the Company, Mr Moore participates in the Company's management and employee incentive schemes which involve the grant of stock options and restricted share rights.

In order to meet the requirements of ASX Listing Rule 5.16.6 the PEA is released with the report titled "Independent Technical Report for Blackwater Gold Project" dated October 21st, 2014 (the "ITR"), prepared by or under the supervision of M.G.Dorricott, Principal Mining Engineer. Mr Dorricott is an employee of AMC Consultants Pty Ltd and independent of OceanaGold as that term is defined in the VALMIN Code. Mr Dorricott is a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the JORC Code. The ITR is attached as Appendix 2 to this report.

The reader is referred to Appendix 3 of this report for a number of cautionary notes and details of technical disclosure associated with the public release of this report and the accompanying ITR.

1 SUMMARY

1.1 Introduction

OceanaGold is a leading Pacific Rim gold producer, with three operating gold mines and a portfolio of assets located in New Zealand, the Philippines, El Salvador and Australia. OceanaGold is listed on the Toronto, Australian and New Zealand stock exchange under the code “OGC”.

OceanaGold has prepared this Preliminary Economic Assessment (the “PEA”) for the Blackwater Project (the “Project”) according to National Instrument 43-101 and Form 43 101F1. The Blackwater project is located south of Reefton, South Island, New Zealand and is owned by OceanaGold (New Zealand) Ltd (OceanaGold NZL), a wholly owned subsidiary of OceanaGold. The Mineral Resource is the reef continuation of an historic mine.

OceanaGold has undertaken this study to assess technical and economic potential of the Blackwater Project and to support investment decisions for an exploration decline. Due to the depth and geometry of the Birthday Reef (which hosts the resource) and surface land ownership constraints, the project is technically stalled because it cannot be progressed through to a higher confidence level of Mineral Resource by conventional exploration alone prior to deriving a production target.[1] It is not technically possible to achieve the required level of confidence in the Mineral Resource by drilling from surface alone. Thus an exploration decline from which an underground drilling platform can be established is required to progress the project.

Figure 1-1 shows gold gram-metres values (gold assay values multiplied by width of intercept) from historical workings, drill intercept locations with estimated true widths, gold assay results and the limits of the updated resource estimate.

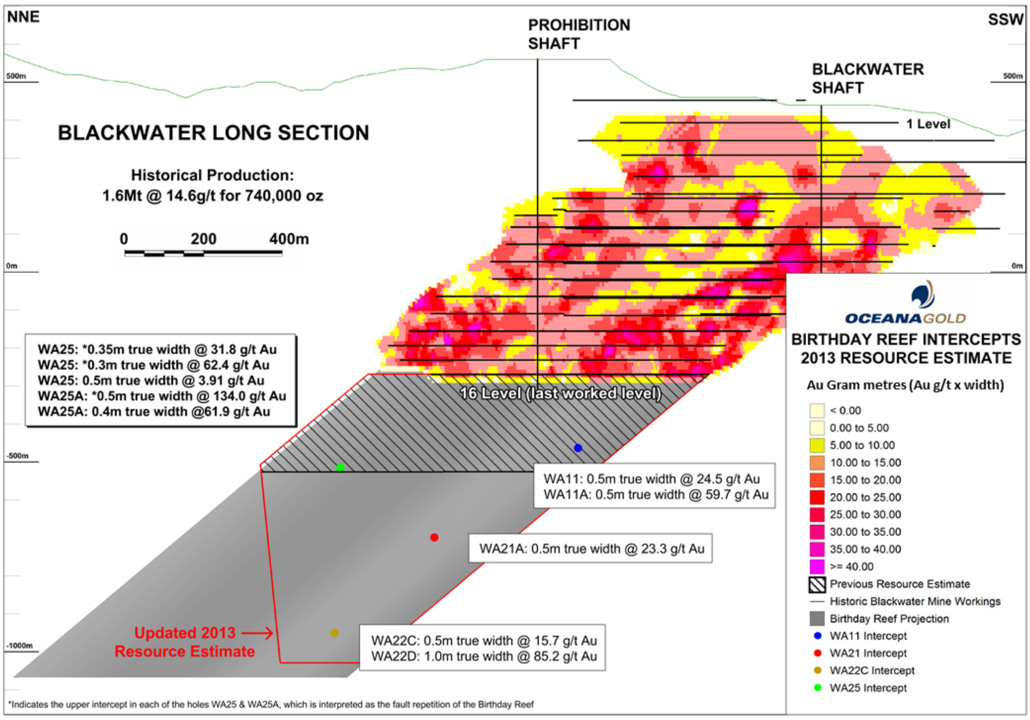

Figure 1-1: Blackwater Mine Long Section

[1] ASX Listing Rules define a production target as: “a projection or forecast of the amount of minerals to be extracted from a particular mining tenement or tenements for a period that extends past the current year and the forthcoming year.”

The Mineral Resource estimate detailed in Item 14, which is the basis for this PEA Technical Report, is an 'Inferred Mineral Resource', being that part of a Mineral Resource for which quantity and grade (or qualities) are estimated on the basis of limited geological evidence and sampling.

OceanaGold has successfully intersected the Birthday Reef with four deep diamond holes (and their daughters) collared from surface in two campaigns in 1996 and 2010 to 2013, supporting the projected extension of the Birthday Reef. The intersection results are summarized in Table 10-3. The results are consistent with the range of historically mined widths and grades and indicate that the Birthday Reef continues for at least 680m vertically below the last worked level of the Blackwater Mine.

Test work completed on material sourced from the current diamond core and surface sampling around the mine waste dumps indicate a high level of gravity recoverable gold. Processing will involve producing a low mass recovery gravity gold concentrate followed by fine gold flotation. This process should recover in excess of 97% of the gold into a high grade concentrate. Intensive leaching of the concentrates and electro-winning will allow production of gold doré bars on site with an expected overall recovery in excess of 96%.

Over 43 years, Blackwater Mines Ltd extracted gold-bearing quartz from the Birthday Reef, producing 740,000oz from 1.6Mt of ore (14.6g/t Au recovered) at a process recovery of approximately 90%.

While the Mineral Resource in this PEA is classified as an Inferred Mineral Resource, the relevant authors believe that it is a reasonable estimate of the global Mineral Resource as constrained by the deep exploration drilling. This level of confidence is based on a number of factors which are explained in detail in this PEA and confirmed by an Independent Technical Review (discussed below). They include the historic mining data at the Blackwater Mine, results from the recent drilling programs which have been used to estimate the Inferred Resource boundaries, and reasonable interpretation of the continuation of the grades and payability information from the records of historical production. On that basis and in view of the fact that the project is technically stalled, the company believes it has a reasonable basis for applying detailed modifying factors to the resource to determine the reported production target.

1.2 Independent Technical Review

An Independent Technical Review (ITR) of OceanaGold's PEA, including a site visit by key specialist reviewers, has been completed by AMC Consultants Pty Ltd. The ITR is presented as Appendix 2 to this report and for the purposes of meeting ASX listing rules.

1.3 Key Outcomes

The study has demonstrated technical and economic viability for extraction of the Birthday Reef subject to improved understanding of the geological resource, geotechnical and hydrogeological conditions. The potential returns for the selected base case scenario, together with range sensitivity support a recommendation for construction of the exploration decline.

A mining method and ore processing solution has been recommended and supported by detailed first principles operating and capital cost estimates. Environmental studies have been completed and resource consent obtained. Due to the scale of the investment relative to the size of OceanaGold's operating activities the project can be funded from OceanaGold's operational cashflow, should it so choose.

The key outcomes of the PEA are summarised in Table 1-1.

Table 1-1: PEA Key Findings

Description	Low/Base/High
Mine life	10 Yrs.
Pre-production	2.5 years
Product	Gold doré bars
Mining Method	Air-leg Resue
Ore Processing Method	Grind/Gravity/Float/RIS
Ore Processing Recovery	96%
Ore Mined (per annum)	120,000t
Gold recovered	(400koz.) 570koz. (740koz.)
Gold recovered (per annum)	(40koz.) 58koz. (75koz.)

Notwithstanding the above and OceanaGold's confidence in the project, the basis for this PEA is an Inferred Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised. The stated production target is based on the company's current expectations of future results or events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met, hence the requirement for an exploration decline.

1.4 Property Description and Ownership

The Blackwater Mine is in the Buller District of the west coast of the South Island of New Zealand, 37km south of Reefton (by road) and 60km northeast of Greymouth. The Mineral Resource is located beneath the abandoned township of Waiuta, situated on a saddle in the foothills of the Victoria Range at an elevation of about 440m above sea level. Road access to Waiuta is from State Highway 7 (SH7) by way of the Class III road from Hukarere to the hamlet of Blackwater (7km sealed) and on to Waiuta (7km unsealed).

The project is located within exploration permit EP40 542, covering an area of 4,308 hectares. Topography in the mine area is steep to moderate relief ranging from 240m elevation at the historic Snowy River battery site to over 560m at the Prohibition Shaft. The proposed underground mine will be accessed by a decline from land adjacent to the Snowy River, accessed via the Snowy Road from Hukarere (refer to Figure 4-3). OceanaGold has entered into an option agreement to acquire the land required for surface infrastructure works (Figure 4-1), which expires in April 2016.

OceanaGold has received resource consent for the construction of an exploration decline, mining and associated infrastructure works. Specific building permits will be required for site works related to the exploration decline. A variation to the resource consent is required to permit on site ore processing and co-disposal of filtered tailings.

1.5 Geological Setting and Mineralisation

The extensive and continuous production history of the Blackwater Mine, and associated data, has demonstrated strong continuity of gold mineralisation within the Birthday Reef. It is in this context that broad exploration drilling has been used to define an Inferred Mineral Resource of 0.9Mt @ 23g/t Au for 0.7 Moz below the historical mine workings, in accordance with the JORC 2012 Code and the CIM Definition Standards for Mineral Resources and Mineral Reserves. The JORC 2012 Code and CIM Standards are identical except that JORC 2012 Code requires additional disclosure around resources extrapolated beyond actual sampled locations (see Item 14.3). The resource remains open at depth, and has some strike extension potential. The extent of Inferred Resource classification is limited by the coverage of drilling.

Detailed reconstructions of the historically mined reef show that the reef pinches, swells and is disrupted by fault sets. Grade control and mapping strategies are proposed to mitigate these anticipated reef disruptions. The potential for fault offset continuations requires further investigation.

1.6 Mining

There is a limited amount of geological and geotechnical information currently available to assess appropriate mining methods. Upon completion of the access decline there will be a diamond drilling campaign to collect the data required to confirm mining methods. Data to be collected will include:

- Reef variability – width and grade;
- Geological faults and structures;
- Geotechnical data – rock mass strengths and insitu stresses; and
- Hydro-geological data.

In the absence of this detail, a single mining method has been assumed to represent a base case though it is possible that a combination of mining methods could be required. Additional mining methods will be considered once more information becomes available.

Conventional mechanised development mining is planned to access the reef via a twin access decline from surface. Hand-held (air-leg) mining techniques are proposed for stoping activities, along with resue blasting which enables segregation between ore and waste. The stope mining width is 3.0m, comprising of 0.68m of reef (resource assumption) and 2.3m of waste. Production panels will be broken into work areas of 60m in length. Once fired, the ore will be scraped to a central ore pass within each work area using a scraper. After ore scraping, the waste will be fired using flat back drilling and retained in the stope as fill for the next lift, with a 2m mining lift height excavated during each cycle. Previous study work has been completed on mechanised cut and fill and long hole benching methods both of which are considered applicable if the thickness of the reef exceeds what is expected for the base case air-leg method, and where ground conditions allow.

1.7 Ore Processing

Test work completed on material sourced from the current diamond core and surface sampling around the mine waste dumps indicates a high level of gravity recoverable gold.

Ore processing will involve producing a high mass recovery gravity gold concentrate followed by fine gold flotation. This process should recover in excess of 97% of the gold into a high grade concentrate. Intensive leaching of the concentrates and electro-winning will allow production of gold doré bars on site with anticipated overall recovery in excess of 96%.

Blackwater ore was successfully processed at two different processing plants between 1908 and 1949. Records indicate that achieved gold recovery was between 85% and 95% using a combination of gravity, flotation and cyanide leach processes.

Variability testing on deep sourced core samples is not a viable option given the depth, cost and lack of core materials. This represents a residual risk to the current understanding of gold recovery. An extensive data collection and confirmatory test programme is planned from the exploration decline.

1.8 Environment

The Blackwater Project will be consistent with regional and district plan objectives and will aim to achieve compliance with appropriate monitoring standards for any adverse effects on the receiving environment. It will bring social and economic benefits to the local, West Coast and national economies.

An Environmental and Social Impact Assessment (ESIA) prepared for the project as the basis for the issue of resource consents, and based on a range of technical reports prepared by external subject-matter experts, has been reviewed to incorporate updated reports now that the processing option and mining methods are better understood.

The main environmental consents that are required to develop and operate the Blackwater Project are the regional and district council resource consents, which have already been issued based on a preliminary mine design that remains broadly applicable. These consents will permit the construction of the exploration decline, exploration drilling and underground mining.

Prior to construction of an ore processing plant resource consents will be required to accommodate on-site processing and tailings storage facilities. The updated ESIA has not identified any reason why these additional facilities, provided they are appropriately managed, would not receive resource consents. Remaining environmental consents (comprising building consents for the waste rock dump and any other permanent structures) are routine and, while not currently held, may be assumed to be obtainable in due course on appropriate terms.

Further work is planned on management of groundwater, waste management strategies including management of tailings residue once geochemistry characteristics are known.

1.9 Capital and Operating Costs[2]

1.9.1 Capital Costs

The base case scenario identifies that two and a half years of "pre-production" are required to establish the access decline and initial underground exploration drilling platform. Capital expenditure in the first two years (the only years in which there is no mining of ore material) is estimated to be US$76M, and sustaining life of mine capital is estimated to be US$78M, including 15% contingency – refer to Table 1-3. Capital cost estimates are ±25% and assume that the mobile mining fleet is purchased rather than leased.

The US$76M cost estimate includes the cost estimate for the pre-production mining, surface infrastructure, process plant and contingency. Expenditure on resource definition diamond drilling is incurred from the third year onwards, once underground drilling platforms have been established. Total life-of-project resource definition capital expenditure has been estimated to be US$9M (plus contingency), and is included in the life-of-mine sustaining capital total of US$78.1M shown in Table 1-3.

To reflect uncertainty associated with the estimation of capital, a range of likely pre-production capital costs has been assessed. The low and high cases reported in Table 1-2 are based on +/-30% range limits.

Table 1-2: Summary Pre-Production Capex (US$M)

Item	US$ Millions		
	Base	+30%	-30%
Infrastructure	8	10	6
Processing	21	27	16
Mining	30	39	23
Management & Indirects	4	5	3
Operational Readiness	3	4	3
Contingency (15%)	10	13	8
Totals	**76**	**98**	**58**

[2] The capital and operating expenditure discussed in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

Table 1-3: Base Case Pre-production Capital Cost Summary (US$M)

Description	Year 1 ($M)	Year 2 ($M)
Processing Capex:		
Infrastructure & Power	4.1	2.1
Ore Processing		20.5
Total	**4.1**	**22.6**
Indirect:		
Engineering & Design	1.8	
Operational Readiness		3.1
Commissioning		0.3
Management & Indirects		3.7
Total	**1.8**	**7.1**
Underground Mining Capex:		
Development	10.1	10.4
Mobile Equipment	6.2	
Electrical Equipment	1.4	0.4
Infrastructure		0.5
Other	0.2	1.1
Total	**17.9**	**12.3**
Pre-production Capital Total:	**23.8**	**42.0**
Contingency Factor @ 15%	3.6	6.3
Sub Total per annum	27.4	48.3
Total Pre-Production Capital		**75.7**
LOM Sustaining Capital Total		**78.1**
Total Project Capital		**153.7**

1.9.2 Operating Costs

Operating costs have been estimated using first principles derived from supplier quotations and/or benchmark data from OceanaGold and other similar operations. The low and high cases in Table 1-4 are based on +/-30% range, to reflect the early stage of the project. The mining costs quoted for the base case scenario do not include capital costs.

Table 1-4: Operating Cost Inputs (US$/t Ore)

Item	US$/t Ore		
	Base	+30%	-30%
Mining	154	200	118
Processing	42	55	32
Site G&A	19	24	14
Selling Costs	2	3	2
Totals	**217**	**282**	**167**

An allowance of 2,500m of grade control drilling for each panel within the mine has been costed, which at a cost of US$225/m for drilling and assaying totals US$4M for the life-of-mine. This is included in the Mining operating cost in Table 1-4.

1.10 Economic Analysis[3]

The economic analysis is based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the Preliminary Economic Assessment based on these Mineral Resources will be realised.

The results of the economic analyses discussed in this item represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes:

- Mineral Resource estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected recovery rates;
- Infrastructure construction costs and schedules; and
- Assumptions that the required permits will be approved by the relevant authorities.

Having due regard to these limitations, the outcome of the economic evaluation suggests robust economics for the base case scenario reported.

The confidence in the cost estimates relating to the surface infrastructure and processing plant is higher than that of mining due to the level of detail that was required for the consenting process.

The reef width and grade is expected to vary during mining, and a single resource shape has been generated to represent the Inferred Mineral Resource. A block model has not been generated for the production target. The mining study has therefore applied a constant thickness and grade as a base case and has assessed a range of possible outcomes.

Given that the production target is based on 100% Inferred Resource an NPV range analysis has been completed to estimate the expected outcome for a range of reef thickness and reef gold grades. The selected range limits were based on ±30% which are considered indicative ranges in confidence for an Inferred Resource.

The maximum impact on contained metal modelled is approximately ±30%, whether by flexing only width, only grade, or a combination of both. This range is considered to be a reasonable maximum variability that could be encountered when undertaking extraction of the Birthday Reef. The results are presented in Table 1-5 including a lower post-tax NPV scenario of US$21M and an upper post-tax NPV scenario of US$243M. The base case scenario suggests a post-tax NPV of US$132M.

Table 1-5: NPV Array – Flexing Reef Width and Grade

NPV ARRAY			Reef Thickness {Diluted Thickness} (m)				
			-30%	-15%	Base	15%	30%
			0.48 {0.75}	0.58 {0.87}	0.68 {1.00}	0.78 {1.17}	0.88 {1.30}
Reef Grade (g/t)	-30%	16			21		
	-15%	20		46	83	105	
	Base	23	42	90	132	159	199
	15%	26		133	180	212	
	30%	30			243		

[3] The financial analysis discussed in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

Examination of Table 1-5 shows that NPV remains positive over the full range of reef width and grade scenarios modelled, and the project financials are considered robust. It should be noted that within any of the modelled scenarios the grade and width were maintained as constants. Both width and grade will vary during the course of mining but with the current level of information available it is not possible to model such scenarios.

Table 1-6 shows the results of the Preliminary Economic Assessment of the base case scenario. The entire project is intended to be internally funded by OceanaGold cashflow, and 5% is the OceanaGold internal hurdle rate of return when assessing investment opportunities.

Table 1-6: Preliminary Economic Results of the Base Case Scenario

Description		Value ($M)
Estimate of Cash Flow		
Payable Metal	Gold koz.	569
Net Smelter Return (NSR)		
	Dore Revenue	737
	Total Revenue	**737**
	Royalty (incl. Royalco)	-16
	After Royalty	**721**
Operating Costs:		
	Mining	-180
	Processing	-49
	G&A	-22
	Total Operating	**-251**
	Operating Margin (EBITDA)	**470**
	Initial Capital	-66
	Initial Capital Contingency @15%	-10
	Life-of-Mine Sustaining Capital	-68
	Sustaining Capital Contingency @15%	-10
	Income Tax	-89
	After Tax Free Cash Flow	**228**
	(Post-Tax) NPV @: 5%	**132**
	IRR	**23%**
	Pre Tax Free Cash Flow	**316**
	(Pre-Tax) NPV @: 5%	**193**
	IRR	**29%**

1.11 Conclusions and Recommendations

OceanaGold has delineated a significant gold project near the town of Reefton in the Buller District of the west coast in the South Island of New Zealand. 100% interest in the project is held through the wholly owned subsidiary OceanaGold NZL. OceanaGold has secured the required option agreements to acquire the surface land for the project and has the requisite resource consents to construct an exploration decline and the associated infrastructure.

Notwithstanding the confidence limitations associated with an Inferred Resource, there is sufficient evidence to suggest continuity of the Birthday Reef as detailed in this study. The anticipated continuity can only be confirmed by mining the exploration decline and undertaking the planned resource drilling. Each of the confirmatory drill holes drilled by OceanaGold to date has intersected the reef in predicted locations at depths of up to 1600m below surface. Modelling of the reef in the historical mine has enabled OceanaGold to statistically analyse anticipated grade and reef thickness.

The results from this study demonstrate that the project is technically and economically viable and it is recommended that the project advances to construction of the exploration decline.

Upon establishment of the initial underground exploration resource definition drill drive, a diamond drilling campaign will be required to support a Mineral Resource update, and conversion of Mineral Resources to Mineral Reserves through to full feasibility study.

The Blackwater Project has very good access and supporting infrastructure. The deposit appears to be amenable to conventional underground mining methods with estimated mining recoveries assumed of 85%.

Process test work suggests that gold recovery of 96% is achievable using the proposed treatment process.

The project has a ten year mine life after two and a half years of pre-production. The base case scenario is robust, returning positive post-tax NPV results over a range of inputs.

OceanaGold has been pro-active with regard to environmental and socioeconomic issues. Environmental monitoring, baseline studies and site investigations have been ongoing at the Blackwater Project site. Additional environmental baseline programs are expected to continue, as required through 2014. Consultation to date has included meetings with local councils and discussions with local land-owners. During the resource consent application process all notified parties withdrew their right to be heard.

Whilst the Company has recently received the required consents to construct the exploration decline and to undertake exploration drilling and underground mining, resource consents will be required to allow for ore processing and tailings co-disposal facilities.

The production target is currently based 100% on an Inferred Mineral Resource and it is technically not possible to progress the project without investment in an exploration decline. The decline is required for the purposes of data collection for resource definition, metallurgical test-work, geotechnical assessment and groundwater management without which the project cannot advance.

There is extensive information from the historical Blackwater mine located directly above the Inferred Resource. This information along with OceanaGold's recent deep drill programme provides sufficient confidence to justify investment in the exploration decline.

In the context of OceanaGold's ongoing operations in New Zealand and the Philippines the investment requirement for the exploration decline and associated works is not material.

CERTIFICATE OF QUALIFIED PERSON

As a qualified person and co-author of the report titled “Preliminary Economic Assessment – Blackwater Gold Project” (Blackwater PEA) dated October 21st 2014, to which this certificate applies, I, Simon Owain Griffiths do hereby certify that:

1. I, Simon Owain Griffiths, am the General Manager Studies for OceanaGold Corporation. My business address is OceanaGold, Taunton Mews, 22 MacLaggan Street, Dunedin, New Zealand.

2. I graduated with a B.Eng. (hons) Minerals Surveying and Resource Management degree from the University of Exeter, Camborne School of Mines in 2000 and an MSc Mining Engineering in 2003 from the University of Exeter, Camborne School of Mines and an MSc Mineral Economics in 2010 from Curtin University School of Business.

3. I am a member and Chartered Professional (Mining) in good standing with the AusIMM and SME.

4. I have worked as a mining engineer and study manager in the mining industry for a total of 14 years since my graduation.

5. I have read the definition of “qualified person” set out in the 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 fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.

6. My most recent personal inspection of the Blackwater Project was on 29th August 2014.

7. I am responsible for Items 1.1-1.4, 1.8-1.11, 2-5, 18-22, 24, 25, 26.1.3 of the “Preliminary Economic Assessment – Blackwater Gold Project” dated October 21st 2014.

8. I am not independent of OceanaGold Corporation applying all the tests in item 1.4 of NI 43-101 because I am an employee of OceanaGold (New Zealand) Limited.

9. Prior to my commencement of employment with OceanaGold in March 2013, I have had no involvement with the Blackwater Project.

10. I have read NI 43-101 and the items of the Blackwater PEA under my responsibility have 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 Blackwater PEA contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
MINING
Simon Griffiths

Simon Owain GRIFFITHS
Date of Signature: October 21st 2014

CERTIFICATE OF QUALIFIED PERSON

As a qualified person and co-author of the report titled “Blackwater PEA” dated October 21st 2014, to which this certificate applies, I, Jonathan Godfrey Moore do hereby certify that:

1. I, Jonathan Godfrey Moore, am the Chief Geologist for OceanaGold Corporation. My business address is OceanaGold, Taunton Mews, 22 MacLaggan Street, Dunedin, New Zealand.

2. I graduated with a BSc (Hons) Geology degree 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 and Chartered Professional (Geology) in good standing with the AusIMM.

4. I have worked as a geologist in the mining industry for a total of 25 years since my graduation.

5. I have read the definition of “qualified person” set out in the 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 fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.

6. My most recent personal inspection of the Blackwater Project was in June 2013.

7. I am responsible for Items 1.5, 6-12, 14, 23 and 26.1.1 of the “Preliminary Economic Assessment – Blackwater Gold Project” dated October 21st 2014.

8. I am not independent of OceanaGold Corporation applying all the tests in item 1.4 of NI 43-101 because I am an employee of OceanaGold (New Zealand) Limited.

9. Prior to employment with OceanaGold in May 1996, I have had no involvement with the Blackwater Project.

10. I have read the Blackwater PEA which 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 Blackwater PEA 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: October 21st 2014

CERTIFICATE OF QUALIFIED PERSON

As a qualified person and co-author of the report titled "Preliminary Economic Assessment – Blackwater Gold Project" (Blackwater PEA) dated October 21st 2014, to which this certificate applies, I, Murray Sydney Smith do hereby certify that:

1. I, Murray Sydney Smith, am a Principal Mining Consultant working for Mining Plus Pty Ltd. My business address is Level 27, 459 Collins St, Melbourne, Victoria 3000, Australia.

2. I graduated with a Bachelor of Engineering (Mining) degree in 1993 from the Western Australian School of Mines, a regional campus of Curtin University in Perth, Western Australia.

3. I am a member and Chartered Professional (Mining) in good standing with the AusIMM.

4. I have worked in the mining industry for a total of 20 years since my graduation, including more than ten years in gold mining operations.

5. I have read the definition of "qualified person" set out in the 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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. My most recent personal inspection of the Blackwater Project was in August 2014.

7. I am responsible for Items 1.6, 15, 16 and 26.1.2 of this Technical Report titled "Preliminary Economic Assessment – Blackwater Gold Project" dated October 21st 2014.

8. I am independent of OceanaGold Corporation and any subsidiaries, other than providing consulting services.

9. Prior to providing consultancy services to OceanaGold in 2014, I have had no involvement with the Blackwater Project.

10. I have read NI 43-101 and the items of the Blackwater PEA under my responsibility have 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 Blackwater PEA contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
MINING
Murray Smith

Murray Sydney SMITH

Date of Signature: October 21st 2014

CERTIFICATE OF QUALIFIED PERSON

As a qualified person and co-author of the report titled "Preliminary Economic Assessment – Blackwater Gold Project" (Blackwater PEA) dated October 21st 2014, to which this certificate applies, I, Timothy Raymond Hughes do hereby certify that:

1. I, Timothy Raymond Hughes am the Process Engineering Manager for Gekko Systems Pty. My business address is 323 Learmonth Road, Ballarat 3350, Victoria, AUSTRALIA.

2. I graduated with a Chemical Engineering degree from Curtin University of Technology in 1989 and a Post Graduate Diploma in Mineral Processing in 1992 from the Western Australian School of Mines, a regional campus of Curtin University in Perth, Western Australia.

3. I am a Fellow in good standing with the AusIMM and member of the IChemE.

4. I have worked as a Metallurgist in the mining industry for a total of 24 years since my graduation.

5. I have read the definition of "qualified person" set out in the 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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I have not personally inspected the Blackwater Project.

7. I am responsible for Items 1.7, 13, 17 and 26.1.4 of the Blackwater PEA dated October 21st 2014.

8. I am independent of OceanaGold Corporation applying all the tests in item 1.4 of NI 43-101 because I am an employee of Gekko Systems Pty Ltd.

9. Prior to the engagement contract between OceanaGold and Gekko Systems Pty Ltd in June 2013, I have had no involvement with the Blackwater Project.

10. I have read NI 43-101 and the items of the Blackwater PEA under my responsibility have 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 Blackwater PEA contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Timothy Raymond HUGHES

Date of Signature: October 21st 2014

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2 INTRODUCTION

2.1 Terms of Reference and Issuer for Whom the Technical Report is Prepared

OceanaGold have prepared this Preliminary Economic Assessment (the "PEA") for the Blackwater Project (the "Project") according to National Instrument 43-101 and Form 43-101F1, to support the PEA. The Blackwater Project is located south of Reefton, South Island, New Zealand and is owned by OceanaGold (New Zealand) Ltd (OceanaGold NZL), a wholly owned subsidiary of OceanaGold. OceanaGold is listed on the Toronto, Australian and New Zealand stock exchange under the code "OGC" and is the issuer of this Technical Report.

This Technical Report presents the results of the PEA for the Blackwater Project. The PEA is based on the 31 December 2013 Mineral Resource Estimate previously announced on 26 March 2014, as part of the 2013 end-of-year Resource and Reserve statement. For the purposes of this Technical Report, the Resource statement as at 31 December 2013, inclusive of Table 1 disclosures in accordance with clause 5 of the JORC (2012) Code, is appended to and forms the basis for this PEA.

The 31 December 2013 Mineral Resource Estimate is also compliant with the Canadian Securities Administrators National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101").

The objectives of the PEA are to:

- Examine the potential economic viability of mining the Blackwater deposit (Birthday Reef); and
- Propose a strategy and preliminary timetable to develop the Blackwater Project.

The PEA herein evaluates and/or provides:

- The best project design from multiple options;
- The most appropriate mining method according to the geometry, grade and rock mass conditions of the Blackwater deposit;
- The basic design for most of the required facilities and infrastructure to access, develop and mine the mineralised zones;
- An estimate of the capital and operating costs;
- A preliminary cash flow model;
- An estimate of the cost and timeframe for the pre-production and production periods;
- An analysis of the financial aspects of the Project;
- An estimate of resources potentially amenable to mining;
- Recommendations for additional work to advance the Project to the next stage; and
- A technical report compliant with Form 43-101F1.

2.2 Principal Sources of Information

The PEA was prepared by OceanaGold. Information for the Report was based on published material as well as the data, professional opinions and unpublished material obtained from work completed by OceanaGold, and materials provided by, and discussions with, third-party contractors/consultants retained by OceanaGold.

Some information was sourced from reports prepared by previous owners of the Blackwater Project and from historical archives.

Reports and documents listed in Item 27 were also used to support preparation of the Report. Additional information was sought from OceanaGold personnel where required to support preparation of this Report.

Table 2-1identifies the external specialists who provided information for various portions of the study.

Table 2-1: Specialist Consultants Who Provided Information to the Study

Consulting Company	Work Package
Mining Plus Pty Ltd	Mining and Geotechnical Engineering
Gekko Systems	Ore Processing
Golder Associates (NZ)	Hydrology and Hydro-geology, Terrestrial Ecology, Aquatic Survey
AMC Consultants Pty Ltd	Mining Method and Geotech Peer Review
BECA (NZ)	Air Discharges and Traffic Impact Assessment
OPUS (NZ)	Visual Impacts Assessment
Hegley Acoustic Consultants	Acoustic Impact Assessment
Brown, Copeland and Co	Economic Impact Assessment
Hills Laboratory (NZ)	Baseline Water Quality Testing
Engineering Geology Ltd	Engineering
GHD	Subsidence Assessment
Mamaku Archaeological Consultancy	Archaeology
TechNick Consulting	Vibration Assessment
Anderson Lloyd Lawyers	Legal Title, Land Purchase and Consenting

2.3 Qualified Persons and Inspection of the Property

The Qualified Persons (QPs) for the Report are OceanaGold employees or external consultants contracted for the preparation of the Report, as follows:

Table 2-2: Blackwater Gold Project, PEA Qualified Persons

Qualified Person (QP's)	Employer	Position	PEA Report Item(s) Contributed to or Reviewed
Simon Griffiths[1] (not independent) B.Eng.(Hons), MSc (Mining), MSc (Mineral Economics), MAusIMM (CP Mining), SME	OceanaGold	General Manager – Studies	1.1-1.4, 1.8-1.11, 2-5, 18-22, 24, 25 and 26.1.3
Jonathan Moore[1] (not independent) B.Sc.(Hons) Geology, GradDip Physics, MAusIMM (CP Geology)	OceanaGold	Chief Geologist	1.5, 6-12, 14, 23 and 26.1.1
Murray Smith (independent) B.Eng.(Mining), MAusIMM (CP Mining), CEng MIEI, FFin, RPEQ	Mining Plus Ltd	Principal Mining Consultant	1.6, 15, 16 and 26.1.2
Tim Hughes, (independent) B Eng. (Chemical), GradDipMineralProcessing, FAusIMM, MIchemE	Gekko Systems	Process Engineering Manager – Gekko Systems	1.7, 13, 17 and 26.1.4

1 Messrs Griffiths and Moore are full-time employees of the Company's subsidiary, Oceana Gold (New Zealand) Limited at the time of writing. As senior employees of the Company, Mr Griffiths and Mr Moore participate in the Company's management and employee incentive schemes which involve the grant of stock options and restricted share rights.

Mr. Griffiths visited the property on 19th and 20th of June 2013, 3rd July 2013, August 28th and 29th 2014. During the site visits Mr. Griffiths inspected drill core and assessed infrastructure and logistics requirements, property boundaries, portal location, proposed location of process facility and waste rock storage.

Mr. Moore last visited the property on the 19th and 20th of June 2013. During the site visit, Mr. Moore inspected drill core and the locations of portal and processing facilities.

Mr. Smith visited the property on 28th and 29th August 2014. During the site visit Mr. Smith inspected drill core and assessed infrastructure and logistics requirements, property boundaries, portal location, proposed location of process facility and waste rock storage.

3 RELIANCE ON OTHER EXPERTS

The authors, Qualified, Independent and Non-Independent Persons as defined by Regulation 43-101, were contracted by the Issuer to study technical documentation relevant to the Report, to contribute to or review the PEA study on the Blackwater deposit, and to recommend a work program if warranted. The authors relied on reports detailed in Item 27, and opinions as follows for information that is not within the authors' fields of expertise:

- Legal advice relating to titles and option agreements was supplied by Anderson Lloyd Lawyers. OceanaGold is not qualified to express any legal opinion with respect to property titles or current ownership and possible litigation;
- Golder Associates Ltd ("Golder") was retained by OceanaGold to provide professional services with respect to the Blackwater Project. The scope of services was to determine the hydrology and hydro-geological conditions relating to the project and to advise on terrestrial and aquatic ecology. The Golder reports were used to as inputs to this report;
- Mining Plus and Gekko Systems relied on external supplier quotations to derive capital and operating cost estimates for mining and ore processing aspects of the study respectively;
- OceanaGold's internal EPCM team generated the infrastructure designs and layouts for the project and relied on conceptual supplier specifications and quotations; and
- A range of other specialist consultants provided reports relating to environmental and social impacts, which were taken into account in the preparation of this report: BECA (NZ) (Air Discharges and Traffic Impact Assessment), OPUS (NZ) (Visual Impacts Assessment), Hegley Acoustic Consultants (Acoustic Impact Assessment), Brown, Copeland and Co (Economic Impact Assessment), Hills Laboratory (NZ) (Baseline Water Quality Testing), Engineering Geology Ltd (Engineering), GHD (Subsidence Assessment), O'Kane Consultants (NZ) Ltd, Mamaku Archaeological Consultancy (Archaeology), TechNick Consulting (Vibration Assessment) and Anderson Lloyd Lawyers (Legal Title, Land Purchase and Consenting, liability for contaminated sites).

The authors believe the information used to prepare the Report and formulate its conclusions and recommendations is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared. The authors, by virtue of their technical review of the Project's exploration potential, affirm that the work program and recommendations presented in the Report are in accordance with NI 43-101 and CIM technical standards.

4 PROPERTY DESCRIPTION AND LOCATION

The Blackwater Project is in the Buller District of the west coast of the South Island of New Zealand. The co-ordinates for the approximate centre of the Property are 42°17'30"S latitude and 171°49'30"E longitude, also expressed in New Zealand Transverse Mercator 2000 (NZTM2000) grid co-ordinates of 1,503,000mE and 5,317,000mN.

4.1 Legal Tenure and Permitting

OceanaGold holds sufficient rights in the Blackwater Project and the main mining and environmental permits required to:

- Acquire the necessary land access rights (subject to Overseas Investment Office and subdivision consents);

- Undertake exploration activities and in due course (if those activities establish a suitable Mineral Resource, ordinarily an Indicated Mineral Resource) secure a Mining Permit; and
- Construct the proposed Exploration Decline and undertake exploration drilling and mining in compliance with environmental laws.

Prior to construction of an ore processing plant, resource consents will be required to accommodate on-site processing and tailings storage. The updated ESIA has not identified any reason why these additional facilities, provided they are appropriately managed, would not receive resource consents.

Whilst there is currently no planned surface expression other than on the Surface Site, any ventilation rise or other aspect of the workings day-lighting beyond the boundaries of the Surface Site, should these become necessary at any stage, will require the relevant landowner's consent and environmental permits.

4.1.1 Forms of Tenure

The legal ability to explore and mine for gold (a Crown-owned mineral) in New Zealand depends primarily on the ability to secure:

- An appropriate minerals permit issued under the Crown Minerals Act 1991. In this case the appropriate permit is an exploration or mining permit for gold;
- Ownership of the land upon which the mining occurs, or an appropriate access arrangement authorising the mining activity; and
- Resource consents issued under the Resource Management Act 1991 for all mining-related activities which are not permitted as of right by the relevant district and regional plans.

The following features of the Project are of relevance:

- The gold being targeted for exploration and mining (the downwards continuation of the Birthday Reef) is the property of the Crown pursuant to section 10 of the Crown Minerals Act 1991. This means that exploration for and mining of the resource can only occur if a relevant minerals permit has been issued by New Zealand Petroleum and Minerals, a division of the Ministry of Business, Innovation and Employment;
- The Project is located within New Zealand's terrestrial land mass, and therefore falls within the scope of the Resource Management Act 1991;
- The Project is located within the local authority areas of the Buller District and West Coast Region. Accordingly, the Buller District Council and West Coast Regional Council are the relevant consent authorities pursuant to the provisions of the Resource Management Act 1991; and
- The Project is an underground mine. The only surface expression will be the decline portal, and all surface elements of the mine will be located adjacent to the portal. The land where the portal and surface mine components will be located (the Surface Site) is legally described as section 9 and 10, Block XIV, Mawheraiti Survey District, Nelson Land District, NL10A/347. The registered proprietor is Granville Mining Limited.

4.1.2 Rights to Land

Figure 4-1 shows the Surface Site. In legal terms it comprises part of section 10 Block XIV Mawheraiti Survey, District Nelson Land District, NL10A/347.

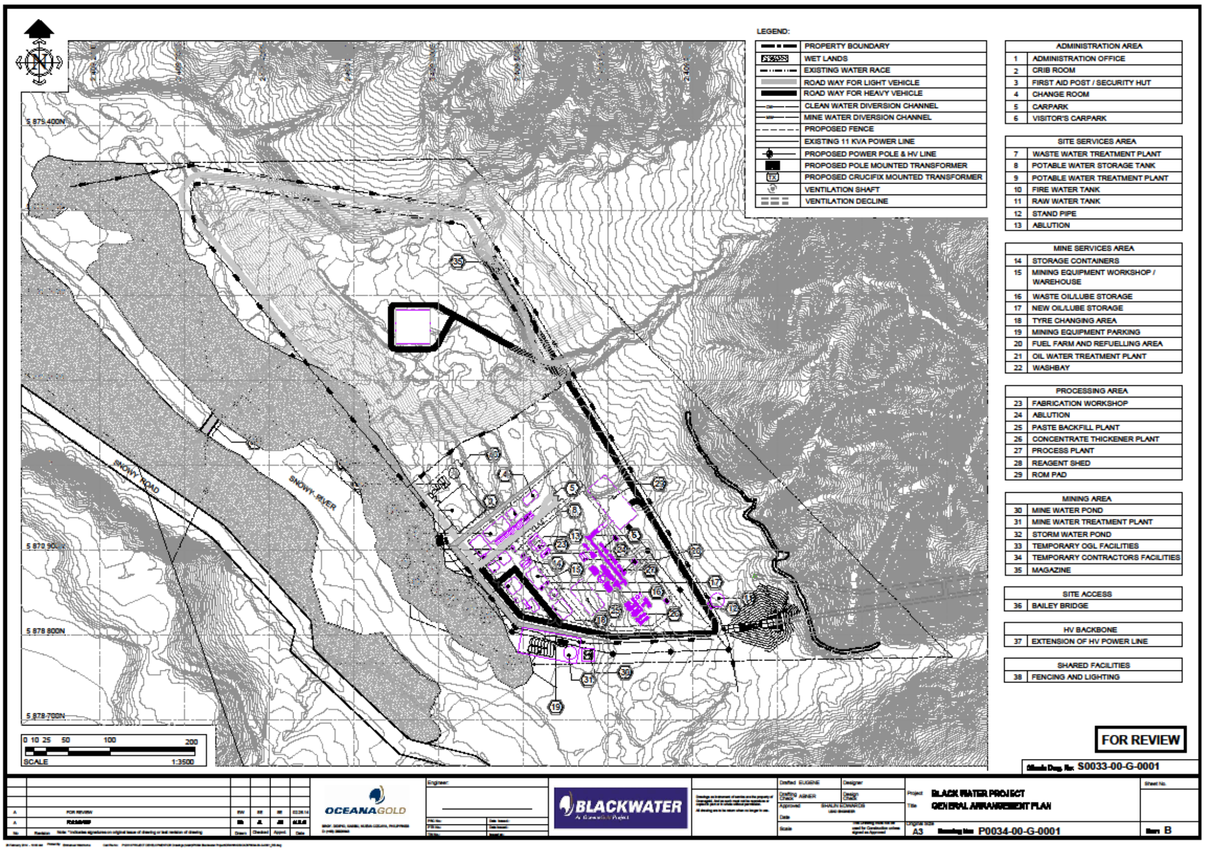

Figure 4-1: Surface Site

OceanaGold has an existing agreement which grants access for exploration activities for a term expiring on 21st April 2016. The agreement also gives OceanaGold an option to purchase some or all of the land at any time up to and including the expiry of the term (i.e. April 2016). Completion of purchase of the land upon exercise of the option to purchase is conditional upon securing subdivision consent (where part rather than all the land is purchased) and Overseas Investment Office approval, both of which are considered likely to be achieved.

Road access to the portal will need to be upgraded and a new bridge constructed over the Snowy River. Bridge construction will require an appropriate easement from the Crown as landowner of the riverbanks and bed of the River, and a licence to occupy from Buller District Council as landowner of road reserve flanking the River. Figure 4-2 illustrates the relevant land designations. OceanaGold has had preliminary discussions with the relevant Crown agency and Council about this, and anticipates no difficulty in securing appropriate authorisations.

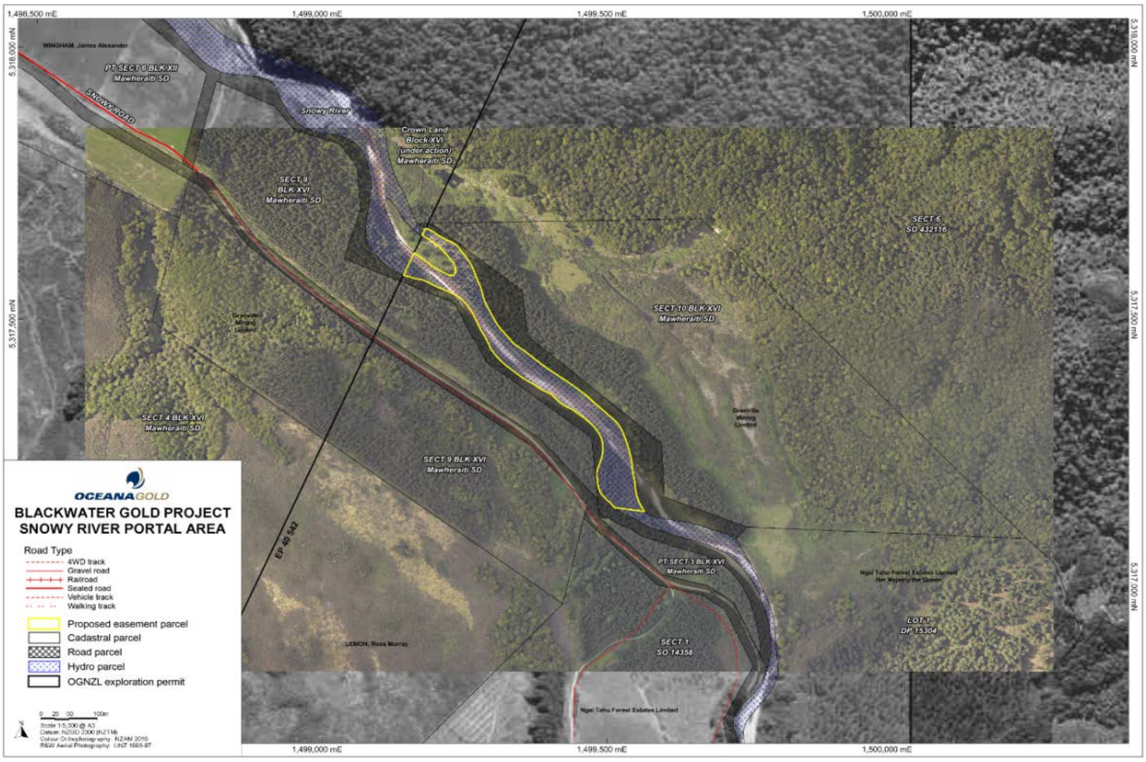

Figure 4-2: Land Designation Plan 1

4.1.2.1 Underground Workings (Including Any Surface Expression)

The underground workings of the proposed Blackwater mine will pass through land owned by various parties, including Crown land administered by the Minister of Lands on behalf of the Crown, public conservation land owned by the Minister of Conservation and land in private ownership. Refer to Figure 4-3.

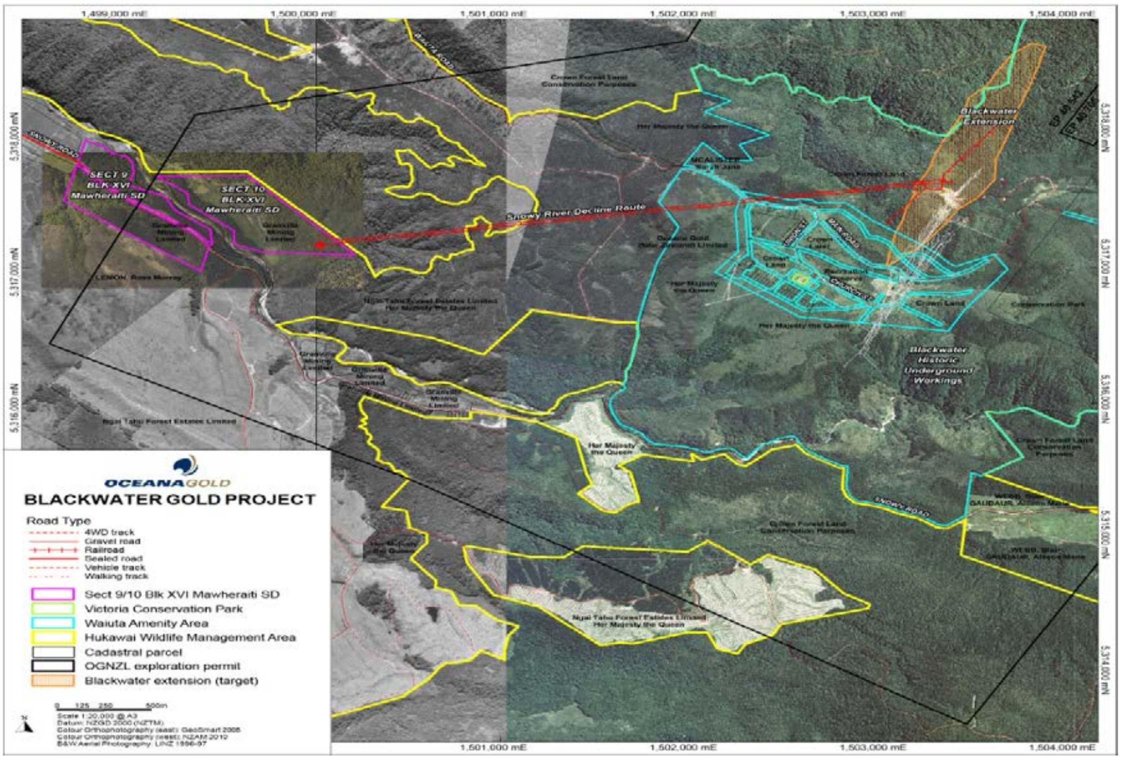

Figure 4-3: Land Designation Plan 2

The law governing the requirement, if any, for landowner consent to mine under the surface of land is found in the Crown Minerals Act 1991 (CMA). Under that Act OceanaGold will not require any access arrangements with the owners of the land through which the Blackwater Mine underground workings pass, provided that OceanaGold's activities have no surface impact on those landowners.

Notwithstanding the position under the CMA, OceanaGold has consulted with relevant landowners in the course of obtaining resource consents, in relation to which both the Department of Conservation (DOC) and the relevant Runanga, Ngati Wae Wae were held to be affected parties. In keeping with its usual practice where it is an affected party under the Resource Management Act 1991 (RMA), DOC has given affected party approval to allow the company's resource consent applications to proceed. Ngati Wae Wae were also consulted and similarly provided affected party approval to the Blackwater resource consent applications.

4.1.3 Environmental Permits

4.1.3.1 Resource Consents

New Zealand's principal environmental protection law is the RMA. District and regional councils have primary responsibility for administering the RMA. OceanaGold's use of land, water, and air in the course of its mining operations must be permitted by a rule in a district or regional plan, or sanctioned under resource consents.

Some of the activities involved in developing and operating the Project will be permitted under the district and regional plans and others require resource consents.

OceanaGold holds a suite of resource consents from the West Coast Regional Council and Buller District Council which authorise the proposed exploration and mining activities associated with the Project, based on an earlier version of the Project which did not include on-site ore processing and tailings deposition. Item 20 of this Technical Report lists the activities for which resource consents will be required and those for which resource consents have been obtained.

O'Kane Consultants (NZ) Ltd (OKC) was retained by OceanaGold to complete a preliminary review of acid and metalliferous drainage (AMD) at the Blackwater Project. OKC have concluded that, in general, assumptions made for the management of AMD, utilising the company's experience at Globe Progress Mine near Reefton to predict water quality and the effects of acid and metalliferous drainage, appear reasonable. However, further work is required to characterise the rocks (and tailings) in regards to geochemistry and forecast water quality to reduce project uncertainties. Amongst other things, geochemical data is needed on the flotation tailings and concentrate tailings including acid base accounting and leach testing to derive potential contaminant loads. This information will be required to determine the stability of the tailings and waste rock under different environmental conditions, prior to applying for resource consents covering co-disposal of waste rock and tailings and disposal of concentrate tailings in the underground workings.

Based on the work undertaken by OKC, OceanaGold has updated its ESIA to include on-site ore processing and tailings deposition. The updated technical assessments from OKC to support that ESIA, while recommending further detailed investigations in some instances, appear to confirm that ore processing and tailings deposition will not result in unacceptable risk or adverse effects in terms of fundamental matters such as geotechnical stability, water and air quality, and ecology. OceanaGold is not aware of any reason why the technical assessments would indicate any significant risks or adverse effects. On that basis, the appropriate consent variations and / or new consents are considered to be attainable.

4.1.3.2 Other Environmental Permits

As the regulatory authorities responsible for granting resource consents, the West Coast Regional Council and the Buller District Council will be the primary agencies with regulatory oversight of the environmental effects of the Blackwater mine.

Secondary agencies include Heritage New Zealand Pouhere Taonga from whom authorities will be required where mining and exploration activities threaten to affect archaeological sites, and the Environmental Protection Authority, which regulates the transport, handing and storage of hazardous goods.

The company has a track record of successfully obtaining and retaining resource consents and the other regulatory permits that it requires to conduct its operations and, as noted above, holds the requisite consents for all proposed activities with the exception of on-site processing and tailings storage activities.

In view of OceanaGold's track record for environmental management, it is reasonable to expect that all necessary environmental permits will be secured.

4.1.3.3 Environmental Bonds

The company will be unable to activate its resource consents until it has furnished the Councils with a bank guarantee or guarantees (**bonds**) for the full amount of the estimated cost of having third parties undertake rehabilitation, covering the expected environmental impact of the company's forecast activities over the first 12 months of development operations (the **bond sum**).

The bond sum will be reassessed annually and increased or decreased to take into account activities to date and forecast activities for each successive year of operations. A preliminary estimate assesses the likely bond amount for the first year of operations to be in the range of US$250,000 to US$500,000.

The company will be required to maintain bonds in place for the applicable bond sum applying from year to year.

OceanaGold is in a position to furnish all necessary bonds based on its proposed activities.

4.1.4 Mineral Rights

4.1.4.1 Exploration Permits and Mining Permits

Rights to prospect, explore or mine for minerals owned by the Crown are granted by permits issued under the Crown Minerals Act 1991. Crown-owned minerals include all naturally occurring gold and silver.

OceanaGold holds an exploration permit (EP) under the Crown Minerals Act 1991, EP40 542, over an area of 4,308 hectares, which includes the relevant area of interest for the purposes of this Technical Report. Figure 4-4 shows the permit boundaries.

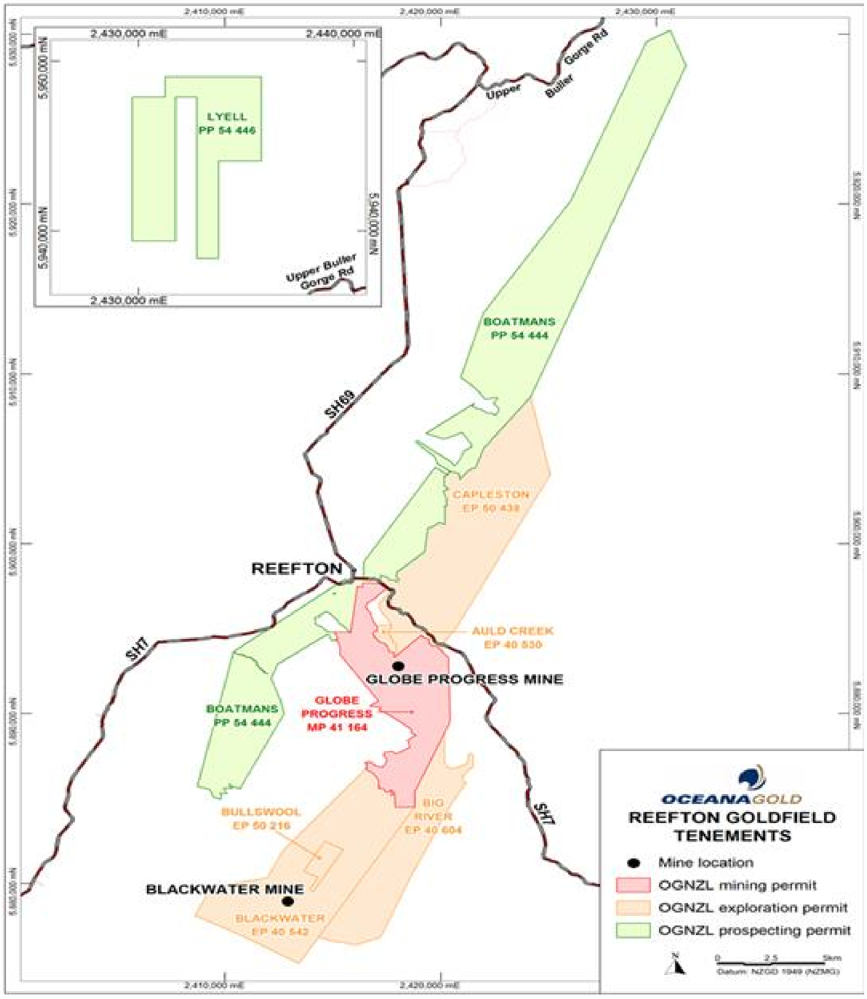

Figure 4-4: Reefton Goldfield OceanaGold NZL Permit as at 30 June 2014

EP40 542 is now in its 12 year, with a current 4 year term for appraisal purposes that runs through to 18 November 2016. In keeping with regulatory custom and practice internationally, the Crown Minerals Act imposes requirements for the reduction in area of any EP by up to 50% over the initial 10 years of the permit's life. The Blackwater EP was reduced from its original size, on first grant in November 2002, of approximately 9,000 Ha.

The Crown Minerals Act 1991 allows a single further extension of the EP of up to 4 years for appraisal purposes, if certain conditions are met.

Provided the permit remains in good standing (principally requiring the payment of annual fees and completion of work programme commitments), and assuming OceanaGold's exploration activities achieve the appropriate level of Mineral Resource (ordinarily an Indicated Mineral Resource), OceanaGold has a statutory right (section 32(3) of the Crown Minerals Act 1991), in priority and to the exclusion of all other parties, prior to the expiry of EP40 542, to surrender the permit in exchange for a mining permit.

Once a mining permit is obtained, OceanaGold will be authorised to commercially extract the gold resource, subject to the conditions attending to the mining permit.

A mining permit (MP) may be issued for a maximum period of 40 years. The Blackwater EP is currently in good standing.

4.1.4.2 Crown Royalty

The Crown Minerals Act 1991 provides for the payment of royalties to the Crown in respect of all gold and silver mined. Under Schedule 1 to the Act, those royalties are calculated in accordance with the minerals programme that applied when the initial prospecting permit (PP) or EP, giving rise to the subsequent MP, was granted.

In the case of the Blackwater EP, the initial permit was granted in November 2002, and came under the first Minerals Programme to be promulgated under the Act, being the 1996 Minerals Programme. Accordingly, the Crown royalties' payable under a subsequent MP will be calculated at the rate of 1% ad valorem or 5% of accounting profits, whichever is the greater within any given calendar year. Accounting losses can be carried forward, and at the ultimate expiry or surrender of the permit there is reconciliation, with provision for any over-payment of royalties to be refunded by the Crown. It is also permissible to amortise future expected rehabilitation costs over the expected life of the permitted operations, meaning the risk of a large correction at the end of the permit's life is reduced.

Note that Crown royalties are calculated on spot price for gold and silver, and take no account of the losses or gains associated with hedging.

4.1.4.3 Royalco Royalty

In addition to Crown royalties, the Blackwater EP is also subject to an agreement contained in a Deed of Novation between Royalco Resources Pty Ltd (Royalco) and OceanaGold, under which an annual royalty of between 1% and 3% of gold produced is payable, according to the gold price at the time the royalty is due. Where the spot gold price is NZ$900 and above the royalty is fixed at 3%.

The royalty reverts to 1.5% of annual gold production once an aggregate of 1,000,000 ounces of gold is produced from all of the Reefton tenements (including from the current Reefton mine).

The Royalco agreement grants OceanaGold an option to buy back the royalty over Blackwater EP for the sum of A$5,000,000, CPI adjusted from 14 May 1991. The option to buy back the Royalco royalty over Blackwater EP may be exercised at any time until OceanaGold makes a decision to mine EP 40 542 and applies to all gold produced from that decision (i.e. from EP 40 542).

4.1.5 Intellectual Property Rights

There is no significant Intellectual Property (IP) understood to be required to implement or operate the project, beyond that acquired in the normal course of business.

4.1.6 Taxes

Net income from the Blackwater mine will be taxed together with all other net income arising from the NZ operations under the Income Tax Act 2007. The NZ company tax rate is currently 28%.

Exploration expenditure is tax deductible in the year in which it is incurred. Development expenditure (including, in some cases, expenditure on exploration assets that subsequently continues to be used in the development and operational phases of a project) is spread for tax deduction purposes. The tax payer is given the option (exercisable once only, in the first affected tax year, and irrevocable after that) to spread the costs of developing a mine over the subsequent production years of the mine on either an estimated (and annually adjusted) life of mine basis or on a units of production basis.

4.1.7 Health and Safety

The health and safety of New Zealand workers is managed under the Health and Safety in Employment Act 1992. Pursuant to that legislation specific regulations relating to the mining industry have been promulgated. Because the Project will come within the definition of a mining operation for the purpose of the regulations OceanaGold will be required to ensure that key management positions are filled and that those managers have the appropriate mining experience and certifications as set out in the regulations.

Aspects of the mine design and management will be required to conform to prescriptive requirements for risk and hazard management.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Location and Accessibility

The Blackwater Mine is in the Buller District of the west coast of the South Island of New Zealand, some 37km south of Reefton (by road) and 60km northeast of Greymouth. The mine is located in the abandoned township of Waiuta, 15km from OceanaGold's Globe Progress Mine. A location map for the project is presented in Figure 5-1. OceanaGold's existing open pit mine, Globe Progress is located south of Reefton.

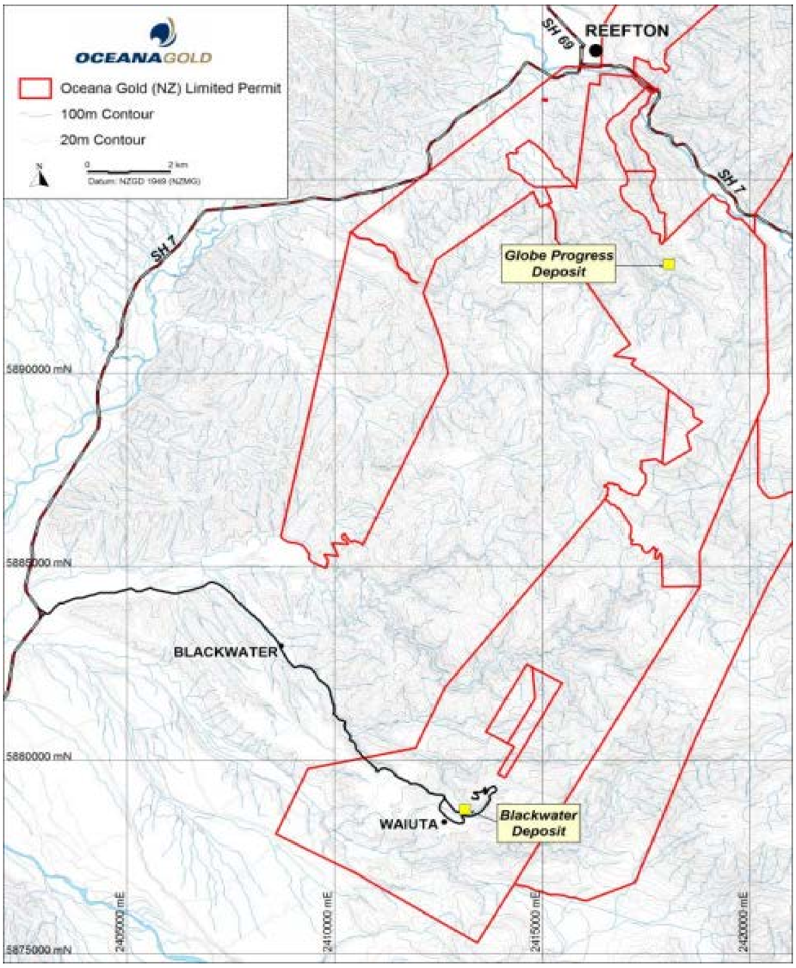

Figure 5-1: Blackwater Gold Mine Location Map

Road access is from State Highway 7 (SH7) by way of the Class III road from Hukarere to the hamlet of Blackwater (7km sealed) and on to Waiuta (7km unsealed). Waiuta is situated on a saddle in the foothills of the Victoria Range at an elevation of about 440m above sea level. Topography in the mine area is hilly with steep to moderate relief ranging from 240m elevation at the historic Snowy River battery site to over 560m at the Prohibition Shaft (Figure 5-2).

Figure 5-2: Aerial View of the Blackwater Mine Looking East

Commercial airlines provide daily services between Hokitika (89km to the south) and Christchurch. Local firms operate helicopter charter services.

5.2 Vegetation

A variety of vegetation coverage exists at the Blackwater site (Figure 5-2). Vegetation types include gorse scrub and rough exotic pasture to the sole native vegetation type of regenerating mixed beech forest, located on the eastern hill slopes. The Snowy River alluvial plain is lightly vegetated, and a pine plantation is located along the banks of the Snowy River.

The Victoria Conservation Forest Park of over 206,000 hectares is protected, and adjoins the eastern edge of the property, where the area is steep and heavily vegetated.

5.3 Climate

The Reefton area climate is moist and temperate, with an average annual rainfall of 1,988mm at the town of Reefton and an average of 2,338mm (1989-1992) at the mine site. The seasonal distribution of rainfall is relatively uniform. Rainfall and wind from the prevailing storms are moderated to some degree by the sheltering effect of the Paparoa Range to the West.

Average monthly mean temperature at Reefton ranges from 5°C in June-July to 17°C in January-February, with a mean of 11°C. Reefton averages 2 days of snowfall and 68 days of ground frost per year. A 10°C mean temperature, 10-15 days snowfall and 115 days with ground frost are predicted for the more elevated sections of the mine site.

6 HISTORY

Historical ownership of the Blackwater Mine is not clearly documented, but ownership as best can be determined is documented in Table 6-1.

Macraes Mining Company Limited, Gold and Resource Developments (New Zealand) Limited and GRD Macraes were previous names of the same entity, being OceanaGold (New Zealand) Ltd, who are the current owner of the Blackwater Mine.

The Blackwater Mine was the largest producer in the Reefton Goldfield with a production of 740,000oz of gold (23t) from 1.6Mt of ore (recovered grade 14.5g/t Au), for the period 1908 to 1951. Table 6-9 provides a breakdown of historical production by year.

Over the 105 years since the Birthday Reef was discovered and work commenced on the Blackwater Mine, there have been a succession of historical resource and reserve estimates for the gold mineralisation. During the life of the mining operation there were annual ore reserve statements, which were generally never large, usually representing one to two years ore supply (Table 6-7). The mine existed on limited development ahead of mining, and only those areas of the mine blocked out on two or more sides were included in these historical estimates.

Table 6-1: Blackwater Mine Ownership History

Year	Mine Owners
1905	PN Kingswell
1906	Consolidated Goldfields Ltd who then formed a subsidiary Company, Blackwater Mines Ltd.
1951	Blackwater Shaft failure, mine closure
1952	Receivership
1969	Blackwater Gold (Waiuta) Ltd
1975	Carpentaria Exploration Company Pty Ltd option agreement with Blackwater Gold (Waiuta) Ltd
1980's	CRA Exploration (CRAE)
1989	Golden Shamrock farm-in JV with CRAE
1991	Macraes Mining Company Limited (MMCL)
1999	Gold and Resource Developments (New Zealand) Limited
2000	GRD Macraes (GRDM)
2004	OceanaGold (New Zealand) Limited (OceanaGold NZL)

The 2003 resource statement was the first comprehensive resource estimate completed by OceanaGold at Blackwater and included both Indicated and Inferred Resources. In 2006 the entire 2003 Resource was reclassified as Inferred prior to OceanaGold listing on the Toronto Stock Exchange. The Resource was updated in 2013 following the completion of the 2011-2013 deep drilling programmes.

Since closure of the Blackwater Gold Mine in 1951 a number of resource estimates of what remains in the old Blackwater Mine and what resource lies below 16 Level have been completed. Item 6.1 summarises these various estimates and concludes with a revised Mineral Resource statement, completed in December 2013 and reported in March 2014.

Due to the changing reporting requirements over the years, many of the old reports use antiquated terminology and methods. These have been reported herein as quoted in the original texts. The December 2013 Mineral Resource estimate, reported in March 2014, and as detailed in Item 6.1.18 is reported to the JORC 2012 and CIM reporting standards. For the purposes of this Technical Report, the Resource statement as at 31st December 2013, inclusive of Table 1 disclosures in accordance with clause 5 of the JORC (2012) Code, is appended to and forms the basis for this PEA.

Figure 6-1: Underground in the Blackwater Mine

Figure 6-1 illustrates typical reef conditions and timber support when the Blackwater Mine was in operation during the first half of the twentieth century.

6.1 Previous Studies and Resource Estimates

Numerous studies of the Blackwater Mine have been completed since the mine closed in 1951. Item 6.1 details these and Table 6-2 gives a summary. Estimates prior to 1995 in Table 6-2 were not reported in accordance with the JORC Code. From 2007 onwards, the Blackwater resource was reported in accordance with the JORC 2004 Code and the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Table 6-2: Summary of Past Development Studies and Mineral Resource Estimates

Year	Resource Estimate Tonnes (Mt)	Grade (g/t)	Ounces (Moz)	Proposed Surface Drilling	Proposed Underground Development	Proposed Underground Drilling
1975	0.59	21.9	0.416			
1987	0.32	21.5	0.220	3 parent holes		
				9 intersections		
1987	1	10 - 15	0.3 - 0.5	4 parent holes		
				12 intersections		
1991	0.18	13.6	0.08	2 parent holes	200m strike driving	
	+1.3	10-15	0.4 - 0.6	6 intersections	150m H/W cross-cutting	
					3 drill cuddies	9 holes
1992	0.075	12.5	0.03	2 parent holes	recover 16 Level	
	1	10 - 15	0.3 - 0.5	6 intersections	120m h/w cross-cutting	
					3 drill cuddies	18 holes
1993	0.18	13.6	0.08	2 parent holes	1,000m strike driving	3,000m
	+1.3	10 - 15	0.4 - 0.6	8 intersections		
1994a	1.7	22 - 25	1.2			
1994b	0.31	21	0.21			
1994					1,000m reef driving	total cost
					500m in cross cutting	$2.35M
					10 drill cuddies	
1995	0.18	22	0.13	2 parent holes		
				6 intersections		
1996	0.31	21	0.21			
1997	0.46	21	0.31			
2003	0.48	22	0.3	1 parent, 1 daughter	Re-furbish shaft	25 holes
					To 17LevelDrill drives	
2005	0.94	9.8	0.28	1 parent, 1 daughter	New shaft Decline & drives	25 holes
2011	0.48	22	0.3	6 parent	Decline & Drives	9,000m

6.1.1 Carpentaria Exploration Company Pty Ltd, 1975

In a 1975 assessment, Carpentaria Exploration Company Pty Ltd (Carpentaria) considered that the 'mineralisation potential' below the 16 Level to 1,000m depth, was a total of 0.59Mt at 21.91g/t Au (in-situ grade) for 0.416Moz of gold (Murfitt, 1975). Carpentaria took the 832m depth of 16 Level as the start point for this calculation. A further 1.38Mt at 21.91g/t Au (0.97Moz) in a potential northern extension was defined as a 'target' for exploration.

This gave a total of 1.97Mt at the historic mined grade of 21.91g/t Au for almost 1.4Moz of gold in 'mineralisation potential' and 'exploration 'target'.

6.1.2 James Askew, 1987

In March 1987, James Askew of James Askew Associates Pty Ltd completed a report on the Blackwater Gold Mine for CRAE (Askew, 1987). In this report the ore reserve and mineral resource estimates were discussed. Askew took the 1950 year-end ore reserve and subtracted 1951 production (to 9 July, when the shaft collapsed) to give estimated remaining (diluted) ore reserves of 66,000t at 13.7g/t Au.

Askew then built a contoured longitudinal section of accumulations in inch-pennyweights from old stope outline plans. It was concluded that there was no perceived geological reason why the tonnes or grade of the mineralisation should diminish with depth and, that therefore, a panel of mineralisation 700m long, 250m deep and 0.7m wide was likely to exist below the old workings. This gave a 'target' for exploration of 318,500t at 21.5g/t Au for 0.22Moz.

Askew recommended drilling three diamond drill holes from the surface, each with two daughters (9 intersections), to establish continuity of the mineralisation prior to refurbishing the Prohibition Shaft. Following recovery of the shaft Askew recommended re-opening 16 Level, developing the 17 Level on ore and then completing a feasibility study. The plan was then to install a raise-bored vent rise, on-sink the shaft 250m and develop four intermediate levels and a level at the base of the 250m panel of ore. Askew envisaged a 60,000tpa shaft hoisting mining operation, with a life of 8 years.

6.1.3 CRA Exploration, 1987

In November 1987, CRAE carried out an in-house study on the Blackwater Gold Mine (Berkman & Lew 1987). Using a simple polygonal estimate of 1,000m by 400m by 1m (stoping width) and using a bulk density of 2.5t/m^3 gave a 'potential resource' of 1Mt at an assumed (historic) mill head grade in the range 10-15g/t Au (including dilution), for a contained 0.3-0.5Moz of gold.

In this study CRAE proposed four deep diamond drill holes, each with two daughters for a total of twelve intersections of the Birthday Reef. It was suggested that the twelve intersection programme was sufficient to define a 1Mt 'resource' as a precursor to refurbishing the Prohibition Shaft and commencing underground development.

6.1.4 GRD Macraes Ltd, 1991

In October 1991, GRD Macraes Ltd (GRDM) completed its first review and prepared a redevelopment proposal for the Blackwater Mine (Hazeldene, 1991). This study also repeated the assertion that 66,000t at 13.6g/t Au of proven and probable ore reserves remained above 16 Level in the Blackwater Mine. However, it also added a proven ore reserve of 115,000t at 13.6g/t Au between the 16 and 17 Levels, for a total of 180,000t at 13.6g/t Au in ore reserves (79,000oz). An additional 'potential resource' of 1.25Mt was projected for the 500m of reef below 17 Level (880-1380m depth in Prohibition Shaft), calculated as 2,500t/vertical metre by 500m depth. At the 'historic' recovered grade of 10-15g/t Au this represented over 0.5Moz of resource.

The study proposed a limited surface deep diamond-drilling programme comprising two parent holes, up to 1,500m deep, each with two daughter holes (6 intersections). This work was considered adequate to define a 1Mt 'possible resource'. Following the surface drilling, a programme of shaft recovery and underground development was proposed.

At shaft bottom (17 Level) drives and cross-cuts would be developed to test the mineralisation and provide sites for underground grid drilling of the mineralisation.

The proposed 17 Level development comprised two drives, 100m to the north and south, on the reef, and the excavation of three 50m long cross-cuts into the hanging wall, at the ends of which drill cuddies would be excavated. From each cuddy two to three holes would be drilled on section and extending up to 100m below 17 Level (6-9 intersections).

6.1.5 Emperor Gold Mining Company Ltd, 1992

In November 1992, Emperor Gold Mining Company Ltd (Emperor) carried out a project evaluation of the Blackwater Gold Mine as a precursor to entering into a farm-in agreement with GRDM (Secker, 1992). Emperor estimated 'mineable reserves' of 75,000t at 12.5g/t Au between 16 and 17 Levels and then 'geological reserves' of 1,000m by 400m by 1m (at 2.5t/m^3) for 1Mt at the usual 10-15g/t Au recovered head grade. Emperor recognised that the Prohibition Shaft passed through the mineralisation between Levels 15 and 16 and that there was some difficulty experienced in this area in the shaft during mining operations. They concluded that more extensive shaft stabilisation work might be required at this point.

Emperor planned two deep surface drill holes, each with two daughter holes (6 intersections). Emperor proposed recovering the Prohibition Shaft and then using the existing 16 Level to excavate three 40m long cross-cuts into the hanging wall, for underground diamond drilling. Six holes would then be completed from the end of each cross-cut (18 intersections). If successful these holes would define 305,500t of 'resource' below the 17 Level. Subsequent production would come from sinking a footwall decline (4mx3m) down (at 1 in 8) from 17 to 18 Level, cross-cutting 30m to the reef, 300m of reef driving (2.5m x 2.5m) to both the north and south and then rising every 30m along strike and establishing hand-held shrink stopes.

6.1.6 GRD Macraes Ltd, 1993

In June 1993, GRDM completed a remote video survey of the Prohibition Shaft that demonstrated that the shaft was blocked at a depth of 124m by a collapse of the shaft at the 1 Level plat (Hazeldene, 1993).

In September 1993, an in-house study concluded that there were ore reserves of 66,000t at 13.6g/t Au between 15 and 16 Levels in the mine. In addition there were 115,000t at 13.6g/t Au of measured (diluted) mineral resource between 16 and 17 Levels. Projecting down dip a further 500m below 17 Level gave an Inferred Mineral Resource of 1.25Mt at 10-15g/t Au for 0.4-0.6Moz of gold.

It was reasoned that (if present) a northern extension of 1km strike length by 1km vertically by 0.66m wide, would contain 1.72Mt of 'pre-resource mineralisation' at 22g/t Au for 1.22Moz of gold.

It was proposed that two drill holes up to 1,500m deep from surface, each with up to three daughter holes (eight intersections), for a total of 8,000m of drilling, be completed to define 1Mt of Indicated mineral resource. The study considered options for both shaft refurbishment and decline sinking (a 3.6km long decline, 4mx3m, at 1 in 7 grade from Hukawai) for access to the mineralisation, but recommended that shaft refurbishment was preferred due to its lower initial cost. It was proposed to carry out 1,000m of level development on 17 Level and then 3,000m of underground diamond drilling.

6.1.7 GRD Macraes Ltd, 1994a

In February 1994, GRDM completed another preliminary scoping study on underground mining at the Blackwater Mine (GRD Macraes Ltd, 1994). This study concluded that there was likely to be a 'resource' of up to 1.65Mt at 22-25g/t Au (1,000m x 1,000m x 0.66m x 2.5t/m^3) below the old workings (for 1.17-1.33Moz of gold). The existing shaft was considered the best option for ore haulage, although the option of a decline was considered. The preferred mining method was mechanised cut and fill using a resuing method to selectively mine the waste then the ore.

6.1.8 GRD Macraes Ltd, 1994b

In September 1994, GRDM completed an appraisal of the historic mining records and prepared an inventory of potential mineral resources below the Blackwater Mine (Ainscough, 1994). In this report the previous ore reserves were downgraded to mineral resources and the dilution removed to bring the reporting into line with the JORC Code reporting of in-situ tonnes and grade.

A total of 211,500oz of gold was estimated to be present within and below the Blackwater Gold Mine, occurring as outlined in the following tables:

Table 6-3: Mineral Resources in the Blackwater Gold Mine 1994 (By Location)

Resource Category	Tonnes	Grade g/t Au	Ounces Au	Comment
Measured	31,800	20.60	21,060	Historical proven ore reserves in the mine at closure
Indicated	11,600	20.60	7,680	Historical probable ore reserves in the mine at closure
Measured	400	13.40	170	16 Level3200N Block
Indicated	12,900	15.45	6,400	Prohibition Block Levels 9-15
Inferred	16,000	16.65	8,560	Prohibition Block Levels 9-15
Inferred	16,700	21.30	11,440	Prohibition Block Level16
Inferred	18,700	21.30	12,800	Prohibition Block Level17
Inferred	27,100	22.35	19,470	16 Level north and south of workings
Inferred	21,800	22.25	15,600	17 Level
Inferred	132,400	22.25	94,700	18 & 19 Levels (roughly 80m below 17 Level)
Inferred	17,600	24.00	13,580	North of Prohibition Fault Levels 7-16
Total	**307,000**	**21.43**	**211,500**	

A small tonnage (30,900t) at a very low grade (1.59g/t Au) was also ascribed to the West Reef, but this is deleted here as the grade is clearly not potentially economic for underground mining.

Table 6-4 summarises the minerals resource by category.

Table 6-4: Mineral Resources by Category in the Blackwater Gold Mine 1994

Resource Category	Tonnes	Grade g/t Au	Ounces Au	Comment
Measured	32,200	20.50	21,230	16 Level and Prohibition Block 7-10 Levels
Indicated	51,600	20.20	33,550	16 Level and Prohibition Block 7-15 Levels
Inferred	223,200	21.85	156,680	17-19 Levels and north of Prohibition Block
Total	**307,000**	**21.43**	**211,500**	

These mineral resource estimates remained the basis for most GRDM resource statements since 1994.

6.1.9 Gemell Mining Engineers Study for GRD Macraes Ltd, 1994

In November 1994, GRDM commissioned Gemell Mining Engineers to undertake a scoping study to explore the viability of mining below the old workings of the Blackwater Mine (Gemell, 1994). This report did not discuss mineral resources or ore reserves.

The study looked at the shaft refurbishment and raise-bored option, decline and raise-bored option, existing shaft and decline option and new and twin shaft (old and new) option. The study recommended the refurbishment of the Prohibition Shaft as the most cost effective approach, although the decline option examined suffered from having the portal at Waiuta and therefore much longer than necessary. A detailed underground exploration plan was devised including 1,000m of reef driving and 10 by 50m long cross-cuts (total 500m of cross-cutting) to drill cuddies, at a cost of NZ$2.35M. The details of the proposed underground drilling programme were not discussed.

6.1.10 GRD Macraes Ltd, 1995

In July 1995, GRDM completed a development plan to commence a mining operation at the Blackwater Gold Mine in conjunction with a separate Globe Progress mining and processing operation (GRD Macraes Ltd, 1995). The study recommended proceeding with initial deep drilling from surface to confirm the presence of at least 400,000t of mineralisation in the Birthday Reef. It was also recommended that shaft clearing and video inspection of the Prohibition Shaft be completed.

This study was the first to conclude that it was unlikely that any of the previous ore reserves, located in small blocks scattered throughout the old workings, would in fact be amenable to mining so long after mine closure. The report repeated the 1994 resource estimate above, but then removed the mineralisation above 16 Level to leave 181,300t at 22.28g/t Au for 129,853oz of gold. As 400,000t was required for the planned mine life (8 years at 50,000tpa) a deep drilling programme was deemed necessary to increase the 'resource'. Two parent deep holes from surface, each with two daughter holes (six intersections) were proposed to define the 0.4Mt of mineralisation.

6.1.11 Minepro Shaft Inspection for GRD Macraes Ltd, 1995

During October and November 1995, Minepro, a specialist shaft rehabilitation and shaft sinking company, provided a temporary head frame assembly over the Prohibition Shaft (Letts, 1995). Over a one month period the company cleared the shaft collar of rubbish, installed new steel dividers and guides and repaired 95m of the south compartment of the shaft down to just above the top of a blockage in the shaft at about 124m depth. This blockage comprised about 4-5m of earth and shaft timbers that had collapsed within the shaft just above the 1 Level plat at 128m depth. A hole was cleared through the blockage onto the plat and then a video camera lowered down the south compartment of the shaft below 1 Level, where the top of a second blockage was encountered at 234m down the shaft, just above 3 Level.

Following the video inspection and a review of the situation it was decided to discontinue clearing the blockage at 124m depth and wait until a more specialised headframe and clearing equipment were installed for the complete refurbishment of the Prohibition Shaft.

6.1.12 GRD Macraes Ltd, 1996

In April 1996, GRDM compiled a record of the current mineral resource inventory for the Reefton Project area, including the Blackwater Gold Mine area (Munro et al, 1996). This report reiterated the 307,000t at 21.43g/t Au for 211,625oz mineral resource of 1994.

6.1.13 GRD Macraes Ltd, 1997

In July 1997, GRDM compiled the annual update of the mineral resource inventory for the Reefton Project area, including the Blackwater Gold Mine (Silversmith et al, 1997). This report reassessed, reiterated and then added to the earlier 307,000t at 21.43g/t Au for 211,625oz estimate in the light of the completed deep drill hole WA11 and daughter intersection WA11A. Some resources were upgraded from Inferred to Indicated and then a large panel of Inferred Mineral Resource was added, extending 150m further than the previous study. This brought the base of the mineral resource down to 230m below 17 Level (and 270m below 16 Level, i.e. to –540mRL).

In summary, the 1997 mineral resource is in effect 1,000m by 270m by 0.67m by 2.5t/m3 block (0.45Mt), with a little additional material representing the downgraded ore reserves from the old mine.

Table 6-5: Mineral Resources in the Blackwater Gold Mine 1997 (by category)

Resource Category	Tonnes	Grade g/t Au	Ounces Au	Comments
Measured	32,200	20.50	21,230	16 Level and Prohibition Block 7-10 Levels
1994 Indicated	51,600	20.20	33,550	16 Level and Prohibition Block 7-15 Levels
New Indicated	143,600	21.60	99,620	Upgraded Inferred below 17 Level
New Inferred	227,500	21.30	155,800	Extends 150m deeper than previous estimate
Total	**455,000**	**21.20**	**310,200**	

6.1.14 John Dunlop and Associates for GRD Macraes Ltd, 2001

In March 2001, John Dunlop of John S. Dunlop & Associates (mining engineering consultancy) completed a scoping study on the Blackwater Mine (Dunlop, 2001). This study was encouraging and recommended a staged development of a small-scale shaft mining operation. A three-stage programme was recommended with initial shaft refurbishment and dewatering, shaft extension and production followed by further shaft extension and further production. A production rate of 75,000tpa was envisaged, with levels at 30m and using non-mechanised (or semi-mechanised) mining methods for level development and stoping.

The only reference to exploration in this study was in a note that a cross cut could be placed into the hanging wall to carry out a diamond drilling in-fill and step out programme, if considered necessary. The report quoted the 1994 GRDM's mineral resource estimate of 307,100t at 21.43g/t Au and then diluted this at 50% to give 480,000t, over a 300m depth, (with no grade specified) for 1,600t/vertical metre.

6.1.15 John Dunlop and Associates for GRD Macraes Ltd, 2002

In July 2002, John Dunlop of John S. Dunlop & Associates prepared an updated version of the 2001 Blackwater Mine scoping study (Dunlop, 2002). This update to the study evaluated the viability of a decline to act as a main access way, in conjunction with refurbishing the Prohibition Shaft as a second egress and ventilation return airway. Advantages were perceived in reducing the ore haulage distance to a processing plant at the Globe Progress Mine and avoiding carting ore from the Prohibition Shaft through the Waiuta historic area en route to Reefton. Mining in this scenario could be by more mechanised methods than the shaft only option above.

The report also stresses the need for accurate grade control practices in the production phase at the Blackwater Mine. Several instances are cited, from Australian mines, where grade control was achieved by pattern underground diamond drilling (at 20-25m x 10-15m spacing) from the footwall decline or drill cuddies off the decline. The comment was made that strike driving for grade control would be costly and could lead to over width drives, and hence excessive dilution. Also mine planning would be delayed and it would be difficult to decide whether to proceed when the face was barren.

A mineral resource of 455,000t at 21.20g/t Au was quoted over a depth of 270m (below 16 Level). When diluted (at 50%) this became 682,500t at 14.13g/t Au, for the same 310,000oz of gold, at 2,500t/vertical metre. A two-hole surface deep drilling programme was outlined to complete intersections 100-150m below the existing WA11 and WA11A intersections. The holes were thought to have the potential to add 150,000t at 21.4g/t Au of mineral resources, which was then diluted to give 300,000t at 10.7g/t Au. A total 'notional' 914,000t could then support a 10-year mine life at a production rate of 75,000-100,000tpa.

6.1.16 GRD Macraes Ltd, 2003

A scoping study was undertaken in 2003 for the development of a narrow high grade underground gold mine at Blackwater. Access would be provided by a decline from the Snowy River with secondary egress and ventilation provided via the refurbished Prohibition Shaft. The proposed mining method was mechanised cut and fill, producing 0.11Mtpa of ore over a 7 year period at an in-situ grade of 21.9 g/t Au, a mined grade of 13.7 g/t Au, metallurgical recovery of 95-96% and yielding 318-322koz of gold over the life of the mine.

A resource estimate was completed based on the historical sampling and the 2 intercepts in WA11 and WA11A (Table 6-6).

Table 6-6: Mineral Resources in the Blackwater Gold Mine 2003

Block	Resource Category	Tonnes	Grade g/t Au	Ounces Au	Comments
Block 1	Indicated	14,000	21.9	9,500	North end of 16 Level
Block 2	Indicated	8,000	21.9	5,500	South end of 16 Level
Block 3	Inferred	456,000	21.9	321,000	Includes WA11 and WA11A
Total		**478,000**	**21.9**	**336,000**	

The entire Blackwater resource was reclassified as Inferred in late 2006, in accordance with the JORC 2004 Code and the CIM Definition Standards for Mineral Resources and Mineral Reserves, when OceanaGold lodged on the TSX.

One surface diamond hole with one daughter hole was proposed to test the continuity of the reef below 16 levels. The Prohibition Shaft would be refurbished to 17 Level and ore drives developed on 17 Level to provide drill platforms. Drilling on 150 x 50m centres were designed to prove up a panel up to 200m below the old workings over the full strike length. Ore from the ore drives would be hoisted to surface to provide a bulk sample for metallurgical testing.

Refurbishment of the Prohibition Shaft was started by contractor Combined Resource Engineering in September 2004. The attempted refurbishment was abandoned in November 2004 after encountering a blockage at 53m. Drilling from surface found the blockage to extend to at least 129m and it was decided to abandon the refurbishment project.

6.1.17 OceanaGold (NZ) Ltd, 2005

Following the failure of the Prohibition Shaft refurbishment McIntosh Engineering was instructed to undertake a full and detailed study on the Blackwater mining method. Coffey Geosciences in conjunction with Combined Resource Engineering undertook a study into sinking a new shaft, 3.0m in diameter and approximately 762m deep for emergency egress, ventilation and de-watering of the old workings.

Additional dilution was factored into the 2003 resource to give an expected resource inventory of 934kt @ 9.82 g/t Au (282koz). Updating of the 2003 scoping study commenced but was never completed and many sections were left unchanged from 2003.

The additional drilling proposed in the 2003 study was included unchanged. None of the drilling was carried out and the project was put on hold.

6.1.18 OceanaGold (NZ) Ltd, 2010-2013

The project was revived in 2010. The Blackwater Deeps drilling program began drilling on the 9th of September, 2010 and continued until the 10th of January 2011 when the program was abandoned due to technical drilling issues.

Three parent holes and three daughter holes were abandoned without any success. The deepest hole (WA20), progressed to 906m before failing, 300m short of the projected target.

Drilling re-commenced under the Exploration group on the 10th of November, 2011 using a new drilling contractor and continued to the 25th of January 2013. Over the course of the program, in addition to the previous intercepts from WA11 and WA11A, 5 successful intercepts (excluding fault repetitions) were achieved by 3 holes from surface: WA21, WA22 and WA25, and their 4 daughter holes: WA21A, WA22C, WA22D, and WA25A – refer to Table 10-3. These intercepts proved that the reef had continuous potential to 680m vertically below the historic workings of the Blackwater Mine.

At the same time a technical study commenced based on a twin decline concept from the Snowy River site to provide access and ventilation. Once close to the Birthday Reef exploration drives would be developed and the 300m of reef immediately beneath the old workings drilled out on an 80 x 80m grid initially, closing to 40 x 40m for final mine design. The mine plan was based on the 2003 resource with targeted production of 45-50koz of gold per annum from a mechanised cut & fill mining operation. Optical sorting of the ore was investigated as a means to sort ore (quartz) from waste post mining and before processing to increase the mill feed grade, and initial trials were encouraging.

6.2 Historical Mine Data

6.2.1 Blackwater Gold Mine Ore Reserves

Table 6-7 lists the annual ore reserve statements for the period 1926 to 1945 for the Blackwater Mine. Note that the Years of Life figure is at the production rate at the time of reserve statement preparation and that over the life of the mine the production rate did vary.

Table 6-7: Annual Diluted Ore Reserves 1926-1945 (Graham, 1947)

Year	Tonnes	Grade g/t Au	Years of Life
1926	70,734	14.66	1.74
1927	72,004	14.81	1.71
1928	77,246	13.95	1.91
1929	74,676	14.55	1.95
1930	86,015	14.69	2.06
1931	85,391	14.86	1.92
1932	76,642	14.75	1.82
1933	99,101	14.45	2.15
1934	97,564	14.00	3.01
1935	94,257	13.86	2.03
1936	92,633	13.63	2.20
1937	86,235	14.35	2.05
1938	93,113	14.47	2.11
1939	103,987	15.07	2.07
1940	101,678	15.10	2.04
1941	93,785	15.22	2.33
1942	97,204	14.61	2.24
1943	90,718	15.01	2.43
1944	81,551	14.64	2.54
1945	79,561	14.17	3.21

When the Blackwater Gold Mine closed in July 1951, the ore reserves were estimated to be 63,000t at a grade of 13.75g/t Au, allowing for 50% mining dilution.

6.2.2 Plans and Records

Many of the records of Blackwater Mines Limited have been preserved, including records of annual production, development grades on levels, level survey plans, and a number of the old Blackwater Mine level plans. Some are held by OceanaGold (NZ) Ltd and some are in the Hocken Library in Dunedin.

The level plans show the outline of the Birthday Reef in the 'backs' and also record the width (in inches) and assay results (in pennyweight) of horizontal channel samples taken across the reef on the levels, usually at five or six foot intervals. The reef width and grade in two sets of these face sampling data were analysed in some detail in 2003. The first set of plans cover the north end of the 11 to 16 Levels, while the second set covers the 4 to 13 Levels in the centre of the Blackwater Mine. A third study was an analysis of the average width and grade of the Birthday Reef from 334 measurements from stopes on Levels 1 to 16. The stope width and grade measurements were compiled from annual ore reserve long section plots, which show the average width and grade of the reef in the active stops at the end of every year. Table 6-8 and Figure 6-2 summarise the results of statistical analysis.

Table 6-8: Analysis of Level Face Sampling and Stope Widths and Grades in all Samples

	Northern Levels 11-16 Samples Set (431 Samples)		Central Levels 4-13 Sample Set (1,083 samples)		Stopes on Levels 1-16 Sample Set (334 samples)	
	Width cm	Grade g/t Au	Width cm	Grade g/t Au	Width cm	Grade g/t Au
Average	67.0	22.5	63.5	21.6	62.5	22.9
Standard Deviation	34.6	17.3	30.3	18.1	18	8.8
Variance	1,194	300.6	919.7	327.5	331.0	77.7
Median	61.0	19.4	61.0	19.0	60.7	21.7
Mode	30.5	36.0	61.0	36.0	55.9	26.0
1st Quartile	38.1	11.0	43.2	10.3	50.8	18.0
3rd Quartile	87.6	30.0	76.2	30.0	73.7	26.0
95% Percentile	133.6	50.5	116.8	45.0	96.0	34.2
Min	5.08	0.50	10.2	0.00	21.8	6.6
Max	194.3	209.9	251.5	299.9	134.6	110
Number	431	431	1,083	1,083	334	334

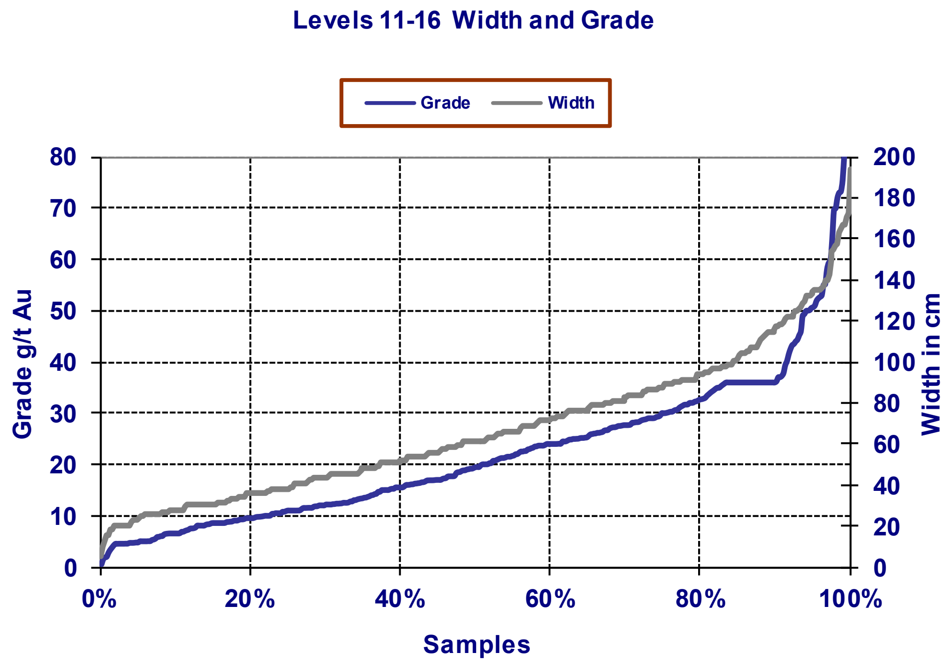

Figure 6-2: Grade and Width Distribution in 431 Samples of Reef on Levels 11-16

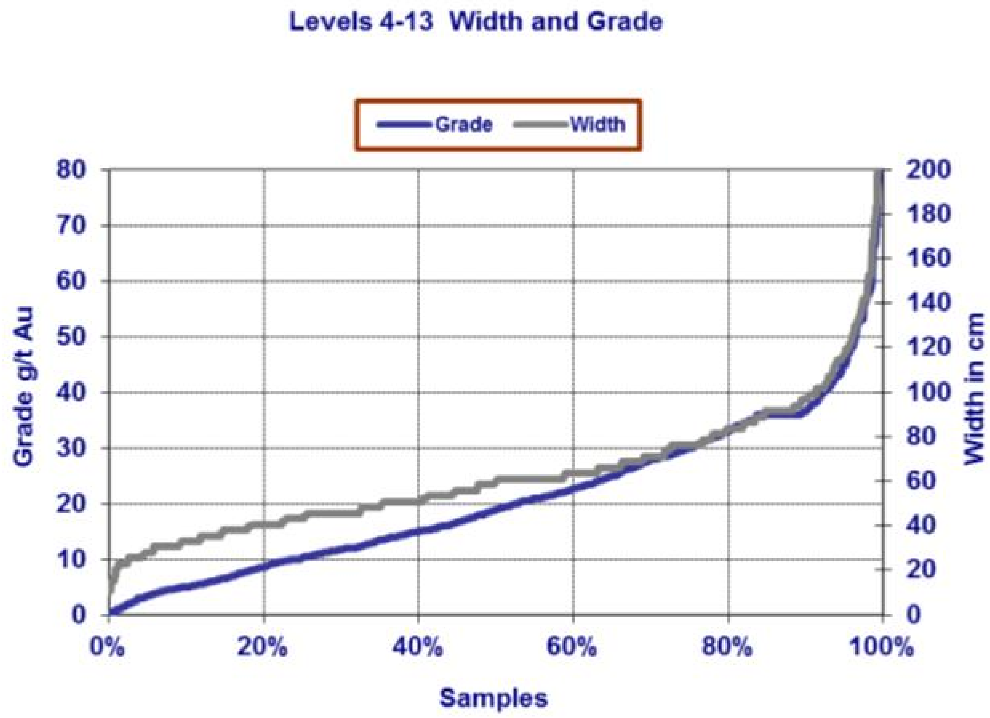

Figure 6-3: Grade and Width Distribution in 1,083 Samples of Reef on Levels 4-13

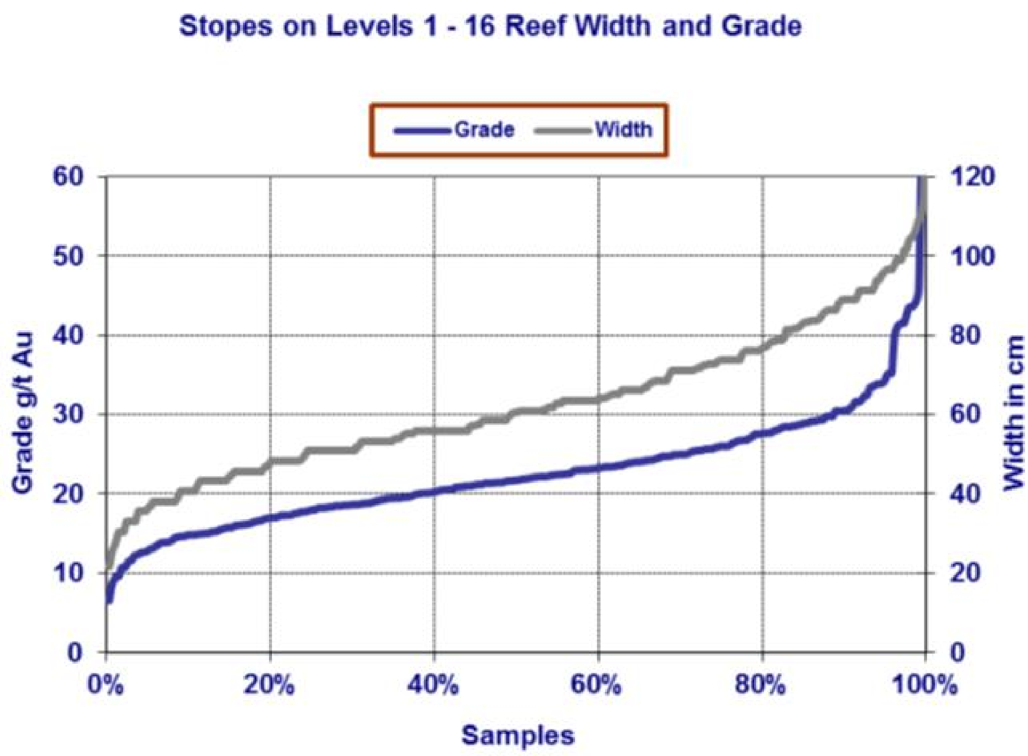

Figure 6-4: Grade and Width Distribution in 334 Samples from Stopes on Levels 1-16

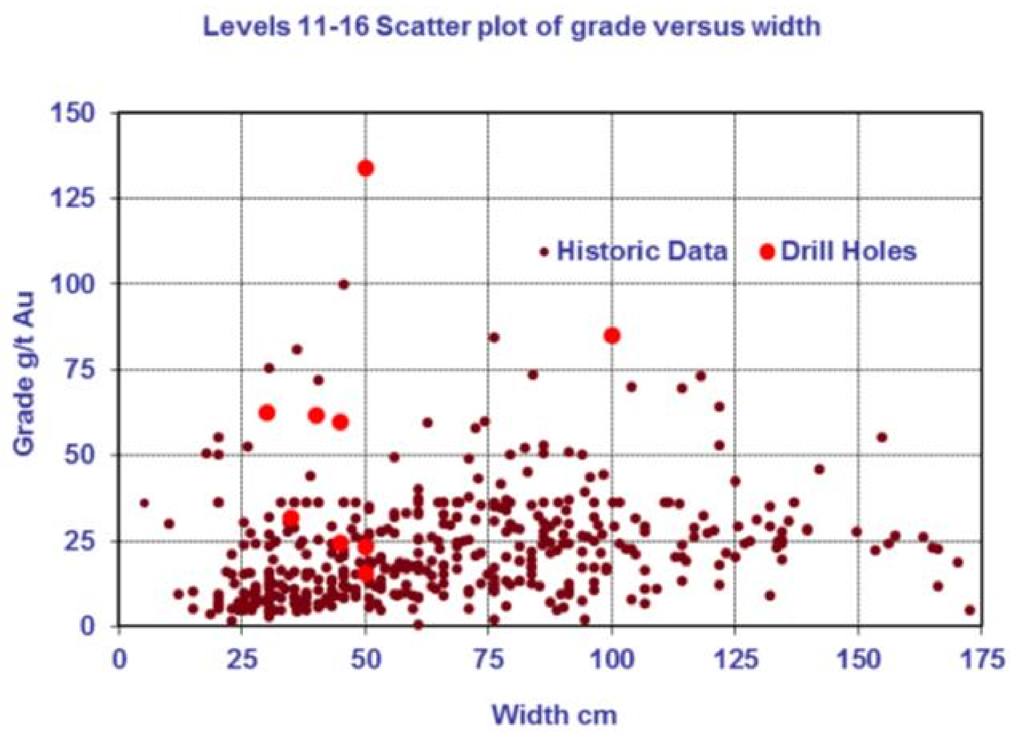

Figure 6-5: Grade versus Width Distribution of Reef Samples on Levels 11-16

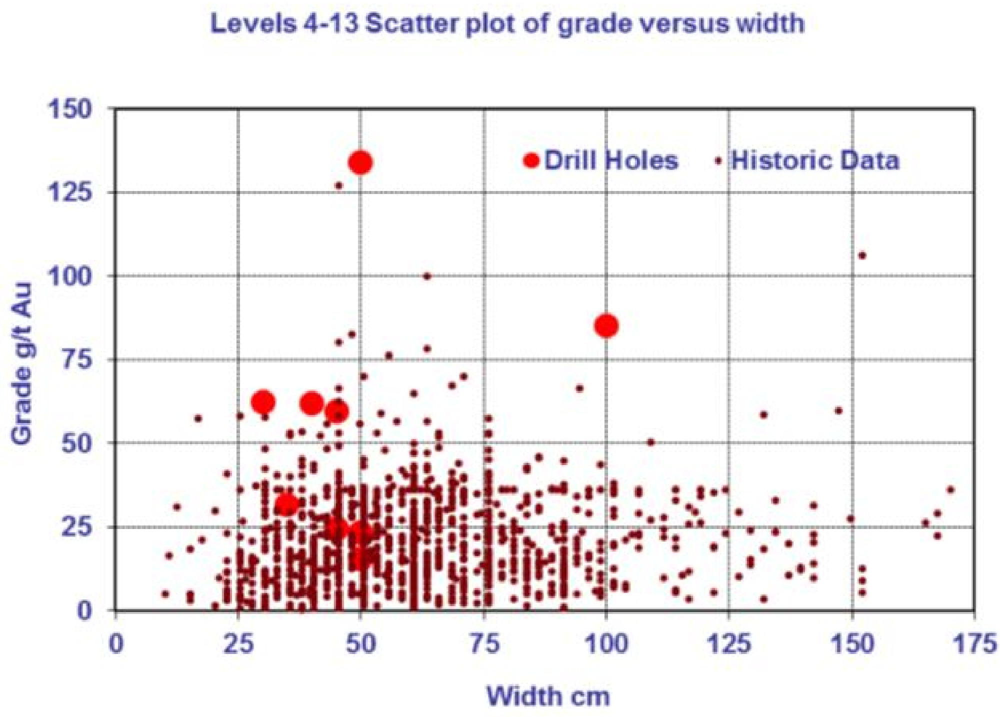

Figure 6-6: Grade versus Width Distribution of Reef Samples on Levels 4-13

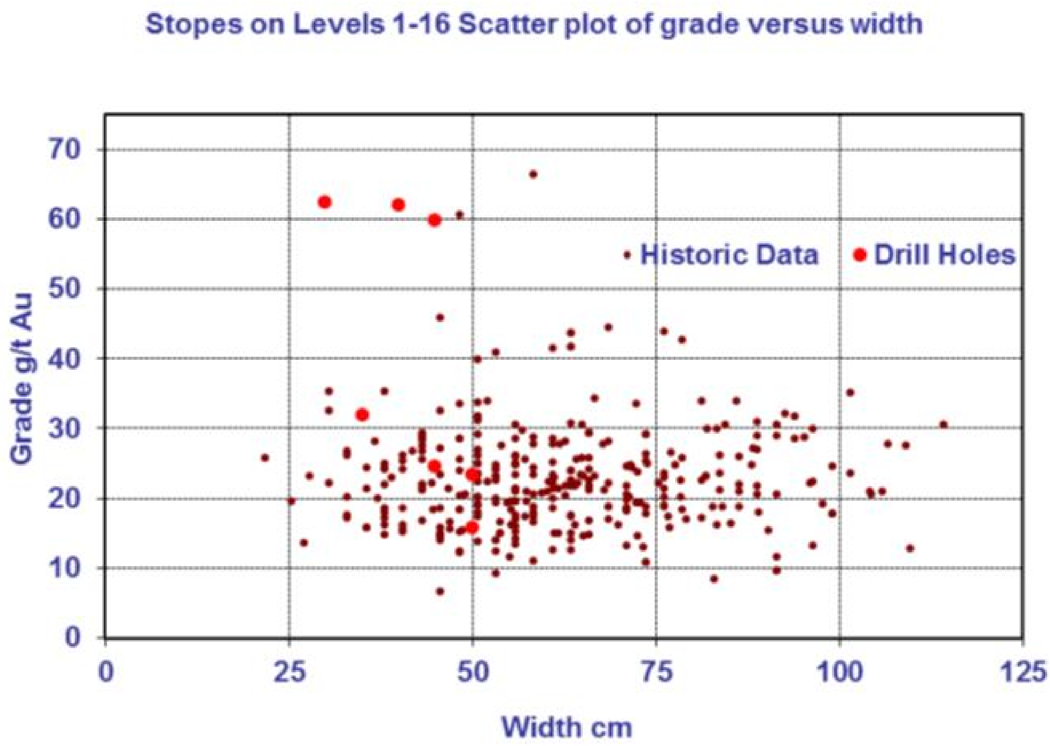

Figure 6-7: Grade versus Width Distribution of Reef Samples on Levels 1-16

Table 6-8 and Figure 6-2 to Figure 6-4 summarise face and stope sample statistics for sampled thickness and grade of the reef. The average width of the Birthday Reef in both the level sampling and the stopes varies from 0.62m to 0.67m. The widths of the Birthday Reef encountered throughout the mine are within a narrow range, and the tabulated average width is close to the claimed average of 0.65m for the life of the

Blackwater Mine. The width in the level samples shows a greater range and variance compared to the stope samples. Nonetheless, the two distributions are remarkably similar and show very similar trends.

The average grade of the 11-16 level sampling is 22.5g/t Au and for the 4-13 level sampling averages 21.6g/t Au, which are both very similar to the claimed average grade of 21.9g/t Au given the very high variance of the sample grades. The average grade of the stope sample population is slightly higher at 22.9g/t Au, but again within a very tight range and in good agreement with the claimed production figures.

The scatter plots (Figure 6-5 through to Figure 6-7) demonstrate that that there is a poor correlation between width and grade. The grade variability is seen to be consistent throughout the range of width values. The reef intercepts from the 1996 and 2011-2013 drilling programs fall within the range of values shown by the face and stope sampling.

6.3 Historical Production

Table 6-9 summarises the annual recorded production from the Blackwater Gold Mine for the period 1908 to 1951. Data has been compiled from several historical reports. The production, recovery, head grade and dilution figures used in numerous reports vary by a few percent due to different conversion factors, and other assumptions having been used in their compilation. However, none of these discrepancies are more than a few percent, and are not materially significant to the values.

Table 6-9: Historical Production from the Blackwater Project

Year	Long Tons	Tonnes	Ounces	kg Au	Recovered Grade (g/t Au)	Back Calculated In-situ Grade (g/t Au)
1908	9,169	9,316	4,681	146	15.6	23.5
1909	29,955	30,434	19,088	594	19.5	29.3
1910	39,192	39,819	23,369	727	18.3	27.4
1911	44,038	44,743	23,557	733	16.4	24.6
1912	11,538	11,723	6,844	213	18.2	27.2
1913	45,053	45,774	20,940	651	14.2	21.3
1914	50,426	51,233	23,400	728	14.2	21.3
1915	54,643	55,517	27,097	843	15.2	22.8
1916	40,247	40,891	19,520	607	14.9	22.3
1917	34,417	34,968	15,500	482	13.8	20.7
1918	31,728	32,236	15,325	477	14.8	22.2
1919	24,969	25,369	12,005	373	14.7	22.1
1920	24,468	24,859	11,065	344	13.8	20.8
1921	34,323	34,872	13,830	430	12.3	18.5
1922	40,092	40,733	19,478	606	14.9	22.3
1923	39,730	40,366	19,296	600	14.9	22.3
1924	38,140	38,750	18,550	577	14.9	22.4
1925	37,939	38,546	18,604	579	15.0	22.5
1926	40,044	40,685	18,032	561	13.8	20.7
1927	41,362	42,024	17,557	546	13.0	19.5
1928	39,907	40,546	16,609	517	12.7	19.1
1929	37,744	38,348	16,201	504	13.1	19.7
1930	41,112	41,770	17,781	553	13.2	19.9
1931	43,815	44,516	21,188	659	14.8	22.2
1932	41,402	42,064	24,474	761	18.1	27.2
1933	45,366	46,092	22,622	704	15.3	22.9
1934	31,862	32,372	16,103	501	15.5	23.2
1935	45,660	46,391	21,216	660	14.2	21.3
1936	41,990	42,662	19,024	592	13.9	20.8
1937	41,333	41,994	18,304	569	13.6	20.3
1938	43,506	44,202	19,465	605	13.7	20.6

Year	Long Tons	Tonnes	Ounces	kg Au	Recovered Grade (g/t Au)	Back Calculated In-situ Grade (g/t Au)
1939	49,482	50,274	26,442	822	16.4	24.5
1940	49,020	49,804	24,795	771	15.5	23.2
1941	39,555	40,188	20,468	637	15.8	23.8
1942	42,676	43,359	18,981	590	13.6	20.4
1943	36,721	37,309	17,246	536	14.4	21.6
1944	31,604	32,110	14,160	440	13.7	20.6
1945	24,387	24,777	11,090	345	13.9	20.9
1946	21,448	21,791	8,006	249	11.4	17.1
1947	22,915	23,282	8,167	254	10.9	16.4
1948	24,328	24,717	9,977	310	12.6	18.8
1949	22,115	22,469	9,540	297	13.2	19.8
1950	20,911	21,246	7,390	230	10.8	16.2
1951	7,128	7,242	3,416	106	14.7	22.0
Total	**1,557,460**	**1,582,379**	**740,403**	**23,029**	**14.55**	**21.9**

Note: Back calculated grades are calculated using 43% dilution and 95% recovery

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional and Local Geology

New Zealand straddles the boundary between the Australian and Pacific tectonic plates, the boundary being marked by the Alpine Fault. The northwest of the South Island of New Zealand lay adjacent to eastern Australia, as part of Gondwana, until about 80 million years ago (Ma) when, through the action of plate tectonics (rifting and faulting) the continent of Gondwana was broken up to become parts of what are now known as Australia, Antarctica, Africa, India and New Zealand.

The Reefton Goldfield lies in the area known as the West Coast Basin and Range Province, which is divided into three broad northerly-trending belts that terminate in the south and east against the Alpine Fault (Cooper, 1974). The Western Belt comprises early Palaeozoic quartz-rich rocks of the Greenland Group, within which lies the Reefton Goldfield. The Central Belt contains a mid-Cambrian to early-Ordovician volcanic island arc assemblage and the Eastern Belt consists of younger sedimentary rocks from lower Ordovician to early-Devonian in age.

7.1.1 Greenland Group

The Reefton Goldfield is hosted by late Cambrian to early Ordovician (circa 500Ma) Greenland Group sedimentary rocks which form the basement rocks in the district. These are interbedded, massive to thinly-bedded (1-20m thick), quartz-rich sediments comprising gradational psammitic (greywacke-sandstone) and pelitic (argillite-mudstone) rock types. They are interpreted as a proximal turbidite succession derived from the erosion of a mature continental landmass, which lay to the east and southeast (Nathan, 1976).

7.1.2 Deformation

The Greenland Group sediments are moderately deformed and have undergone a late Silurian to mid-Devonian (438-395Ma), low-grade metamorphic event. This event was initiated by east-west compression resulting in regional folding and metamorphism to lower-greenschist facies, with illite clay facies predominating.

Folds are close to tight, upright, with north-south trending fold axes and display a single pervasive and penetrative steeply-dipping, axial-planar cleavage.

On-going deformation has resulted in fold hinges being sheared out by high angle reverse faults and bedding-concordant slip planes. The discordant shear zones that now host the bulk of the gold mineralisation in the Reefton Goldfield are thought to have formed as a late-stage, partially strike-slip event towards the culmination of this deformation (Rattenbury & Stewart, 2000).

7.1.2.1 Igneous Rocks

Igneous activity followed the deformation and metamorphism, with the emplacement of widespread Karamea Suite S-type granitoids in the mid-Devonian (375Ma), with a second minor period of granitoid intrusion in the Carboniferous (330Ma). A third intrusive event, the Rahu Suite, comprising relatively small I and S-type granitoid plutons, occurred in the Cretaceous (120-110Ma).

The main mafic magmatic event recognised is the widespread Kirwins Intrusive dolerite, dated to the Jurassic (151-172Ma). Mafic dykes at Waiuta probably date to this event although the fact that they are metamorphosed to greenschist facies implies that they may have been emplaced prior to the Silurian/Devonian metamorphic event. Other unmetamorphosed lamprophyres have been dated to the early-Cretaceous (129Ma) and basalts to the mid-Cretaceous (98Ma).

7.1.2.2 Sedimentary Rocks

Devonian, Triassic, Cretaceous and Tertiary sedimentary sequences overlie the Greenland Group rocks in the Reefton Goldfield. These sedimentary rocks occur in a belt along the western margin of the Greenland Group and also as downthrown fault-bound basins lying on the basement rocks. These basins are Tertiary in age and formed in response to Alpine block faulting. The younger sediments are not as strongly deformed or metamorphosed as the basement rocks, although their beds commonly dip at up to 35º, probably through rotation of the down-thrown blocks.

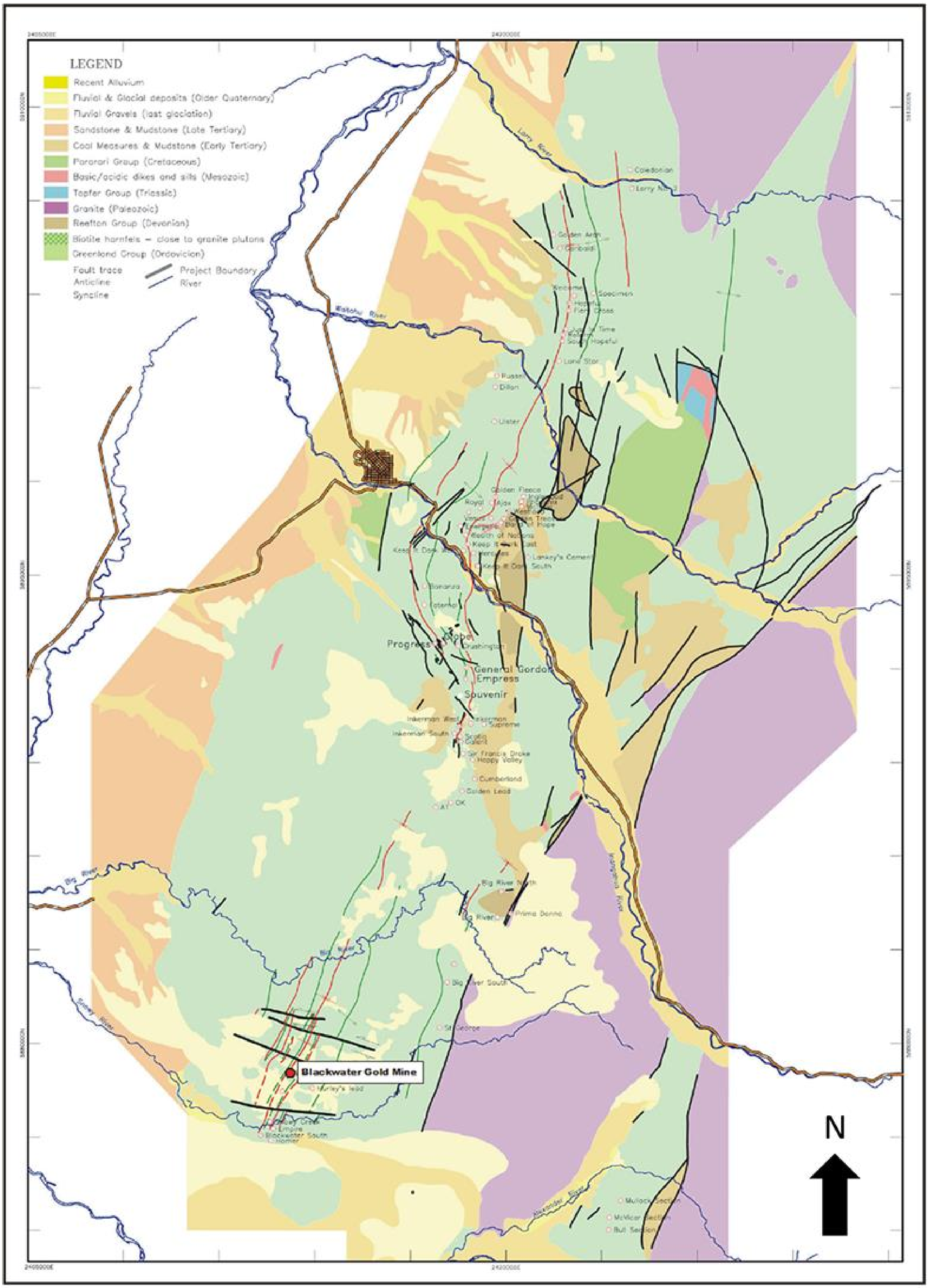

Figure 7-1: Reefton Goldfield Geology (modified from Nathan et al 2002)

A succession of Pleistocene glacial and interglacial events has resulted in the formation of extensive moraine and fluvioglacial outwash deposits. These glacial events have produced prominent terraces and mounds along the main valleys and also occur elsewhere as erosional remnants that are scattered over the Greenland Group and other rocks.

7.1.3 Structure

Outcrop of the Greenland Group sediments is limited to a 35 x 15km fault-bound block, which is bounded by uplifted Karamea granitoids to the east and the down-thrown Grey-Inangahua Depression to the west (Figure 7-1). The Reefton Goldfield therefore comprises a mid-Level (+/-500m elevation) terrain, between a Tertiary horst and graben. In the west the faulted contact is obscured by Tertiary and Pleistocene sediments, which fill the Grey-Inangahua Depression and lap onto the Greenland Group rocks.

The basement geology of the belt is commonly obscured by scattered outliers of Tertiary sediments, including coal measures, and Pleistocene fluvioglacial deposits.

Common graded bedding, flame structures and cross-bedding enable facing direction determinations, while bedding to cleavage relationships allow the mapping of structural vergence directions. The structural complexity, rapid lateral facies variation and lack of marker horizons have prevented the definition of a stratigraphic section or an overall thickness for the Greenland Group sequence.

The structural framework for the Reefton Goldfield was largely established by the 1940's (Henderson, 1917, Downey, 1928 and Gage, 1948), after detailed regional geological mapping programmes by the New Zealand Geological Survey. Detailed structural mapping of the Greenland Group rocks delineated a number of north-trending fold axes and recognised their significance in the structural setting of the major gold deposits.

The Greenland Group sediments are interpreted to be folded into a broad anticlinorium with sub-vertical, north to northeast-trending fold axes. The hinge zones, bedding planes and limbs of these folds have created loci for subsequent shearing, hydrothermal channeling and gold mineralisation, with the bulk of the mineralisation and largest gold mines occurring near the interpreted anticlinorial axis.

Gold mineralisation in the Reefton Goldfield follows a well-defined north-trending corridor through the centre of the Greenland Group rocks, which represents a zone of maximum deformation, within which fracturing and shearing allowed the creation of channelways and traps for ascending mineralised fluids.

7.1.4 Gold Mineralisation

The Reefton Goldfield mineralisation has important similarities, and is probably co-genetic and coeval to, the mineralisation at Bendigo and Ballarat in Victoria, Australia (Christie, 2003). In both Goldfields, mineralisation occurs within Ordovician sediments and is associated with the later stages of folding and thrust faulting.

Gold-bearing fluids arose from depth along highly deformed, fold-related corridors generated by high fluid pressures associated with regional metamorphism and deformation. Mineralisation occurred when these fluids precipitated gold-arsenopyrite-pyrite-stibnite, carbonate and quartz in brittle-ductile fractures.

Most of the gold-bearing quartz veins in the Reefton Goldfield are arranged along a linear belt that runs north-south through the Greenland Group sequence (Figure 7-1). This suggests the presence of a deep-seated structure that has tapped a large reservoir of mineralised fluid.

Fluid stability data from fluid inclusions in the quartz veins and the low salinity nature of the fluids, suggests that the mineralisation was probably derived from metamorphic devolatisation of the sediment pile, although the possibility of an igneous source, or component, cannot be entirely discounted.

7.2 Local and Property Geology

7.2.1 Introduction

The Blackwater Mine is situated in a hilly, dissected area of partially exposed Greenland Group rocks overlain by fluvioglacial terrace deposits and recent colluvium and alluvium.

The steep terrain, thick overburden layer and dense vegetation cover render exploration in the area very difficult. Both geophysical and geochemical methods are, at best, weakly effective due to the variable terrain, overburden and the presence of gold and sulphides in the gravels overlying the Greenland Group.

All the gold production from the Blackwater Mine came from the Birthday Reef, a single quartz vein that was mined continuously for over 1,000m in strike and over 710m in depth.

7.2.2 Lithology

Greenland Group rocks present in the mine area comprise an inter-bedded sequence of massive, jointed quartzose greywacke (a variety of sandstone) and indurated argillite (Figure 7-2). The greywacke is mainly composed of well-sorted, medium to coarse sand-sized, angular fragments of quartz and feldspar.

Minor constituents include lithic fragments, biotite, chlorite, epidote and calcite. The argillite is fine-grained, dark greenish-grey and displays a strong cleavage and relict bedding. Constituents include phyllosylicate minerals, amphibole, quartz and feldspar.

Three narrow dolerite dykes are recorded from the mine area and these are assumed to be part of the Cretaceous Kirwans Intrusive suite. The dolerite has an ophitic texture and is composed principally of augite and plagioclase (andesine-labradorite) with accessory pyrite. These dykes lie in fault zones that cut across the Birthday Reef and are therefore younger than the mineralisation.

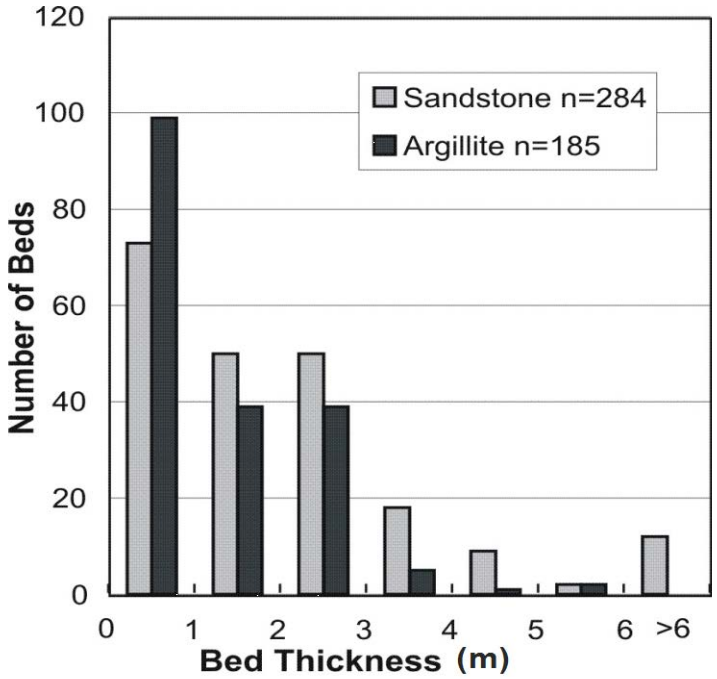

Figure 7-2: Drill Hole WA11 Bed Thickness (Cox, 2000)

Uncomfortably overlying and masking the Greenland Group basement rocks are a series of Pleistocene fluvioglacial terrace deposits. These are horizontally bedded, well-consolidated moraine outwash gravels comprising mainly greywacke, argillite and granite cobbles and boulders in a clayey-sand matrix. Locally varves and silts are present in fine-grained layers. The gravels form discontinuous remnants of flat, high level terraces that fill deep pre-Pleistocene channel structures, which are unrelated to the present topography. The presence of flat-topped terraces and herringbone-patterned ridges indicate the presence of these thick fluvioglacial deposits. The collar of the Prohibition Shaft penetrates through at least a 14m thickness of these gravels.

Minor recent alluvial deposits occur locally in streambeds and river courses, and are essentially reworked glacial and basement rocks.

7.2.3 Structure

7.2.3.1 Folding

At the surface the gold bearing Birthday Reef is hosted within the western limb of a major anticline proximal to the hinge. No detailed underground geological mapping of the Blackwater Mine was conducted and therefore, the attitude of wall rocks and structures in the Blackwater Mine can only be inferred by drill hole data and limited historical accounts. In 2000 S. Cox of the Institute of Geological and Nuclear Science (IGNS) collated the entire previous field mapping completed by GRDM and by IGNS in the Blackwater Mine area. This work showed that in general terms the Greenland Group rocks strike north to northeast and dip steeply both east and west about a near-vertical, northerly-trending fold axis (Figure 7-3). A well-developed, steeply-dipping axial planar cleavage is present, principally in the argillite rocks. Fold hinges are generally absent due to shearing, although the approximate position of the hinges can often be deduced from the fold vergences. Variation in the attitude of the intersection lineation of bedding with cleavage is wide at Waiuta; however the dominant orientation is moderately south-plunging. Significant numbers of lineations also plunge north.

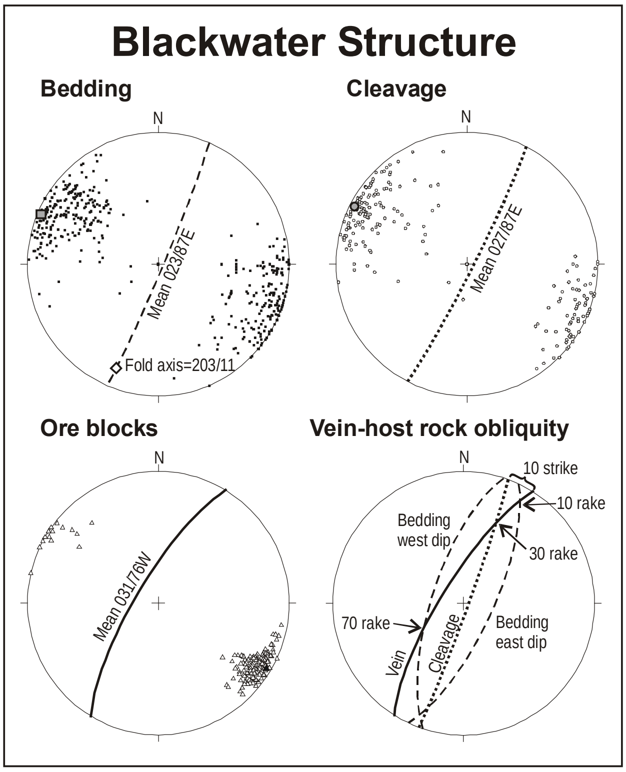

Figure 7-3: Stereo Nets of Structural Elements in the Blackwater Mine Area (Cox, 2000)

Analysis of bedding and fold vergence in drill core and outcrop by Stewart (1996) in the Snowy River, Blackwater Creek and other valleys, demonstrates that the Greenland Group rocks are tightly folded into upright, north-trending and horizontally-plunging anticlines and synclines.

Subsequent analysis of orientated drill core and surface outcrop measurements collected since 2011 expands on the inferences made by Stewart (1996) and Cox (2000). Structural measurements from the surface and from core indicate that as the reef is approached bedding gradually changes from steeply overturned east dipping to steep to moderate west dipping. Drill core evidence also suggests that the strike of bedding rotates anticlockwise from a NE-ENE strike near the surface to a northerly strike at depth adjacent to the reef. Cleavage dip changes from a steep easterly dip near surface to a steep westerly dip at depth adjacent to the reef. Cleavage strike remains consistently NE-ENE. These changes in the fold limb attitude are thought to be a significant control on mineralisation and therefore influence the geometry of the Birthday Reef. No drill holes were extended far enough past the Birthday Reef to determine the Birthday Reefs proximity to the anticline hinge at depth.

In summary, at surface the Birthday Reef has a shallower dip (circa 70º) than bedding and cleavage (near vertical), with strike remaining relatively similar (circa 030). At depth drill core data suggests that the Birthday Reef dips more steeply (circa 80º) than bedding (circa 70º) and that bedding strikes N-S, 20-30º anticlockwise of the Birthday Reef. Cleavage maintains its strike and dips steeply to the west.

7.2.3.2 Faulting

Various authors have inferred various fault plane orientations for the discontinuities affecting the Birthday Reef. In longitudinal section the reef is broken into five north plunging quartz blocks separated by five main faults that have intersection lineations that plunge to the north, and these quartz blocks are crosscut by a fault that has an intersection lineation that plunges south Fault N (Figure 7-3).

In Harold Service's thesis from 1934 he stated that there were four main faults displacing the reef, each having a general strike of 015 and dipping steeply to the east. Service stated that these faults were found to have a reverse shear sense, displacing the reef left laterally. Service also indicated that these faults were often exploited by diabase dykes.

Normal faults were also intercepted during mining but presented little difficulty whilst mining. At the southern end of the reef Service stated that the reef was bounded by a powerful fault beyond which no reef had been discovered despite much prospecting.

In Gage's 1948 report on the Blackwater Mine he mentions that the northern end of the reef was disrupted by a fault plane that was driven on in a northwest direction until the reef was intersected some 45ft away.

The fault was interpreted to be normal, downthrown to the northeast, and appears from the 'drag' exhibited by the ends of the reef and the nearby dolerite dyke to have had an important horizontal component. In the appendix of Gage's report he also discusses the presence of three subsidiary faults parallel with the Prohibition Fault (presumably the fault discussed above) that resulted in a considerable amount of dead work near the end of 15 level.

In Murfitt's 1975 report he identified three principle fault sets that disrupt the Birthday Reef. Firstly, reef-parallel shearing (030º/steep west) seen on the margins of the Birthday Reef. Murfitt suggested that the sub-parallel laminae, ribbon quartz, slickensiding on the margins of the vein, and pinch and swell structures all suggest vein-parallel deformation contemporaneous with mineralisation emplacement.

The second fault set was thought to be post-mineralisation and includes the North West Fault (Figure 7-4) that strikes almost due north and dips at 45 west. This was interpreted to be a normal fault that displaces the Birthday Reef by up to 40m to the west. It was thought that these faults are possibly associated with the Devonian-Permian orogeny that folded the Reefton Group (Devonian) sediments.

The third fault set Murfitt identified was post mineralisation thrust faults that dislocate the reef into at least five distinct blocks. These faults (including the Prohibition Fault system) were interpreted to strike 341º with a dip of 65º to the northeast. Murfitt interpreted these faults to be related to the Cretaceous, Post-Hokonui Orogeny.

Work completed by Barry (1997) indicates that the northern end of the reef is dislocated by a series of faults (known as the Prohibition Fault Zone) with left lateral displacement (<40m) that strike between 317º - 355º and dip between 40-72º northeast. Within the Prohibition Fault Zone Barry also identified a fault with right lateral displacement (<40m) striking 340º and dipping 40º northeast on 7 Level, with the fault having an easterly trend on 11 level.

When discussing the North West Fault, Barry referenced Morgan (1929). Morgan believed the North West Fault was of "no great importance: it causes a blank zone but the movement has not been great". Barry related west dipping faults in the raise plans from Morgan's report to the North West Fault concluding a strike of 015º and a dip of 55º west.

Cox and Rattenbury (2004) were contracted through IGNS to assess the geology along the proposed Blackwater Mine access decline. They reviewed some previous reports and analysed historic plans and drill core. This work is summarised in Table 7-1 and Figure 7-4.

Table 7-1: Blackwater Mine Faults (Cox & Rattenbury, 2004). Orientations dip direction / dip

Fault	Orientation	Movement	Fault rocks	Width	Comments
F	(080 / 70)	(Possibly reverse right lateral)	Unseen	(1-10 m)	Inferred from topography - linear trend of One Horse Stream & hill edge
G	Steep, 45 to core	(Reverse)	Milled breccia & gouge	1-10 m	Between 359-363m in WA15
H	155 / 70	(Right lateral)	Unseen	1-10 m	Post-mineralisation fault truncating Millerton mine
I, J	(080 / 60)	120m left lateral horizontal offset	(Clay-rich gouge)	(>10 m)	Inferred faults along the Snowy River scarp - young
K, L	115 / 55 & 100 / 55	Unknown	Milled breccia & clay-rich cataclasite	0.1 - 1m	Gage (1948)
M	Undetermined	10m	(Clay-rich gouge)	< 0.1 m	Fracturing and minor clay-rich faults between 121-185m in WA15
N	275 / 68	10m left lateral horizontal offset	Unseen	(1-10m)	Blackwater Mine
O	110 / 52	25m left lateral horizontal offset	Planar, polished fault & claycoated crush zone	> 10 m	McVicars Adit (Barry 1997) & WA1 between 116-145m
P	152 / 79	15m left lateral horizontal offset	Unseen	(0.1-1 m)	Blackwater Mine - see Cox (2000)
Q	100 / 75 to 095 / 80	Reverse. 30m right lateral horiz offset	Unseen	(0.1-1 m)	Blackwater Mine - see Cox (2000)
R	285 / 75	Reverse. 30-120m left lateral	Unseen	(0.1-1 m)	Blackwater Mine - see Cox (2000)
S	275 / 80	horizontal offset	Contains sheared dolerite dyke	(0.1-1 m)	Blackwater Mine - see Cox (2000)
T	330 / 37	Reverse. 10m left lateral horiz. offset	Unseen	(1-10 m)	Blackwater Mine - see Cox (2000)
U	065 / 60	Reverse. 30m right lateral horiz offset	Unseen	(1-10 m)	Blackwater Mine - see Cox (2000)
V	080 / 45	Reverse. 13-34m right lateral	Unseen	(0.1-1 m)	Blackwater Mine - see Cox (2000)
W	320 / 35	horizontal offset, reverse	Unseen	(1-10 m)	Blackwater Mine - see Cox (2000)
X	075 / 25-40	Reverse. 10-36m left lateral horizontal	Unseen	(1-10 m)	Prohibition fault, Blackwater Mine - see Barry (1997)
Y	075 / 57	offset	Unseen	(1-10 m)	Prohibition fault, Blackwater Mine - see Barry (1997)
Z	310 / 62	60m right lateral horizontal offset	Unseen	(>10 m)	Blackwater Mine - see Cox (2000)

Amongst all the authors the interpretations of the North West Fault (Fault N) is relatively consistent, therefore it is concluded that a strike (000-015º) and dip (45-68º W) with apparent left lateral displacement of the reef is appropriate. Cox and Rattenbury 2004, (Figure 7-4) have inferred a displacement of up to 10m on the North West Fault, whist Morgan (1929) Inferred only minor displacement.

Murfitt's (1975) reef parallel shearing and interpretation are valid, although not discussed by other authors. Recent drill holes indicate that shearing along the reef is localised to within 15cm of the hanging wall and less in the foot wall of the reef.

There is consistency between authors regarding the Prohibition Fault Zone indicating a strike from 310-340º and a dip from 40 to 65º northeast. The Prohibition Fault Zone and parallel faults are interpreted to have caused apparent left lateral displacement (up to 40 m) of the reef, although Murfitt (1975) interpreted a fault within the Prohibition Fault Zone to have apparent right lateral fault movement based on historical records from 7 Level at the northern end of the mine.

At least three other prominent faults disrupt the reef, forming north plunging intersection lineations. Service interprets these faults to strike 015, dip steeply east and cause apparent left lateral displacement. Murfitt (1975) interprets these faults to be the same orientation and relative displacement as the Prohibition Fault Zone. Cox and Rattenbury (2004) interpreted two of these faults (Fault W and Fault T, Figure 7-4) to strike northeast causing right lateral displacement and the southernmost fault (Fault U) to be the same orientation and relative displacement as the Prohibition Fault Zone.

Recent surface mapping and drilling around the periphery of the Birthday Reef has resulted in the interpretation of two NW (300º) striking faults that are thought to bound the Birthday Reef to the north and south. Known as the Birthday Reef North and Birthday Reef South faults, these faults have been interpreted to dip moderately (40-70º) northeast offsetting major fold hinges (and probably the Birthday Reef) left laterally by up to 300 m. Given the complex tectonic history, it is uncertain whether these faults were present at the time of mineralisation. It is probable that these faults are the same generation as the Prohibition Fault Zone and parallel faults interpreted throughout the workings. Recent surface mapping and drilling has also lead to the interpretation of a NNW striking fault that causes right lateral displacement of the Waiuta Anitcline just to the north of the Birthday Reef. This fault may relate to one identified by Barry (1997) on the 7 Level at the northern end of the workings.

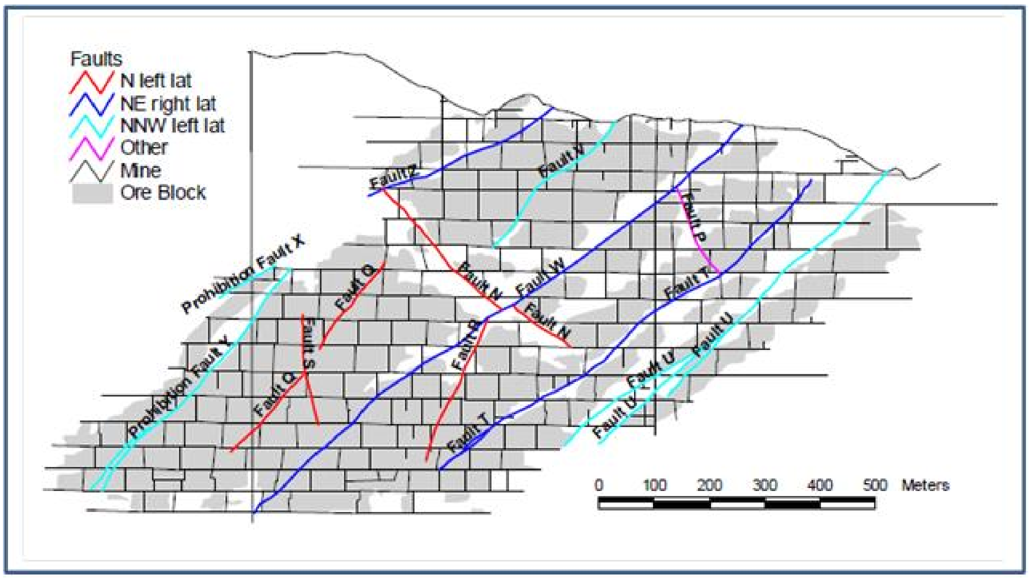

Figure 7-4: Blackwater Mine Late Stage Faults (Cox & Rattenbury, 2004)

The disruption of the Birthday Reef by the Prohibition Fault Zone and other parallel fault zones is depicted in Figure 7-5. The figure shows the historic workings at the north end of 14 levels, displaying how the reef becomes dismembered and / or pulled apart. This results in loss of reef and hence the gaps in the stopping shown on Figure 7-4 above. Disruptions of up to 25m west and 9m north have been measured within the Prohibition Fault Zone and parallel faults in the lower level of the mine. In addition to the major faults identified other faults with smaller offsets are expected throughout the line of the reef and may cause the reef to be “lost” during mining. A stereoplot of all the faults identified whilst logging the recent Blackwater Deeps drill core indicate a significant amount of bedding parallel faulting has taken place in the rocks surrounding the Birthday Reef. There is no indication of the magnitude of shearing that may have taken place on these bedding parallel faults. The Reef appears to become more dislocated as the northern extent is approached. Limited data is available for the southern extent, but a similar amount of dislocation might be expected.

The fault controls on the distribution of gold mineralisation within the Reef is difficult to determine. Grade and width data from the underground workings, when modelled, indicate that higher gram-metre areas occur where the Reef thickens. These thicker portions of the Reef appear to plunge north. It has been hypothesised that the north plunge to the “high grade shoots” are related to the intersection lineation between faults and the shear that hosts the Reef. Assuming that the shear that hosts the Reef strikes 030 and dips ~75º west then any structure that is a) <75º in dip and strikes 300º to 120º or b) >75º in dip strikes 120º to 300º will result in a north plunging intersection on the Reef. A number of the faults (Figure 7-4) discussed earlier would fall into the two options listed above. Paragenetic studies indicate that the Birthday Reef formed during late stage metamorphism syn or post S2 foliation development, given the semi-ductile nature of the environment during quartz vein formation, the attitude of both cleavage and bedding could also have influenced the variation of mineralisation seen within the Birthday Reef.

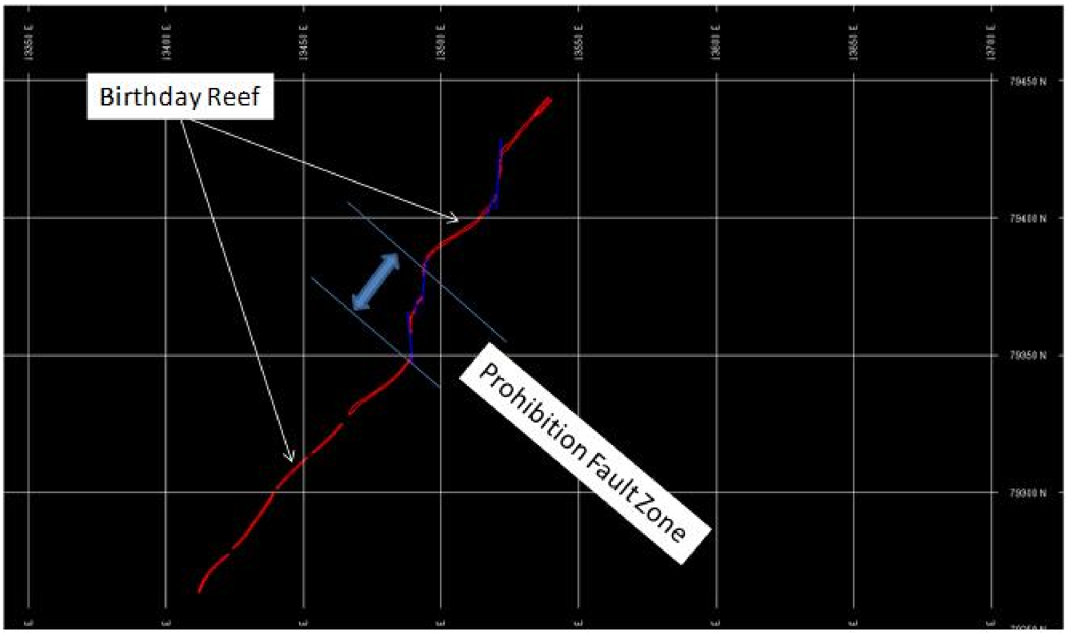

Figure 7-5: Birthday Reef and Prohibition Fault on 14 Levels

7.2.4 Gold Mineralisation

All primary gold mineralisation in the Reefton Goldfield appears to have been formed during one significant mineralisation event. The secondary fluvioglacial and alluvial deposits represent eroded residual material from this primary event.

Gold mineralisation at the Blackwater Mine is hosted within a quartz vein (reef) where about 70-80% of the gold is present as native gold, commonly occurring on the laminated host rock inclusions, with the remainder occurring as refractory gold locked in the lattice of pyrite and arsenopyrite. Sulphides comprise up to 1% by volume of the vein and besides pyrite and arsenopyrite, include minor stibnite and rare chalcopyrite and molybdenite (Morgan 1929).

Multi element analysis of the WA11 core returned a value of 0.09% sulphur confirming that only a very small proportion of gold is in sulphides. Ribbon-banded textures with bands of quartz separated by thin sericite-chlorite laminae are interpreted to demonstrate that the vein formed by incremental growth by repeated fracturing and quartz vein deposition. The sericite-chlorite laminae are the fracturing residues of slivers of wall rock peeled off during the ongoing deformation and vein growth. Minor calcite also occurs within the reef.

It is recorded that the hangingwall portion of the vein is often brecciated and fissile while the footwall is of harder, more massive quartz. This suggests ongoing movement along the hanging wall contact perhaps both contemporaneously with the mineralising event and post-mineralisation. Two types of quartz are described, a massive milky variety with occasional coarse visible gold and no sulphides, and a darker blue-grey laminated quartz with laminae and ribbons of country rock, high gold grades and abundant sulphides.

The surrounding greywacke and argillite are weakly altered and mineralised, with only a very narrow aureole of disseminated sulphide minerals occurring in the sediments. This aureole comprises weak bleaching and finely disseminated pyrite and coarser-grained arsenopyrite, within a groundmass of quartz, chlorite, sericite and carbonate, which extends for up to 1m into the hanging wall and 2m into the footwall.

In addition to the alteration, for several metres individual argillite beds may be structurally dislocated and display shearing and brecciation.

Ore at the Blackwater Mine was produced from five main ore shoots in a narrow quartz vein (the Birthday Reef), which is some 1,000m in horizontal length and extends to at least 1,350m depth below the surface (based on WA22D intersection). From south to north the five ore blocks (and their approximate horizontal lengths) were South Block (122m), No.2 Block (244m), No.3 Block (427m), North Block (92m) and

Prohibition Block (uncertain length due to a lack of data). This suggests an ore length of over 885m, but as the Southern, North and Prohibition Blocks were not always mined, an actual average horizontal ore length (based on the historical diluted cut-off grade of 8-10g/t) of about 850m per level in 1,050m of strike (81%) has been estimated.

The reef is not necessarily continuous within the main blocks identified above, but occurs as shorter segments broken and offset by small faults. A study of the available mapping on Levels 11-16 (Figure 7-6 and Figure 7-7) show continuous reef lengths up to 120m but the majority are less than 30m with an overall median length of 13m and mean of 20m. However, 76% of the tonnes are contained in the lenses with lengths greater than the average of 20m and 86% of the tonnes are contained in lenses with lengths greater than the median of 13m.

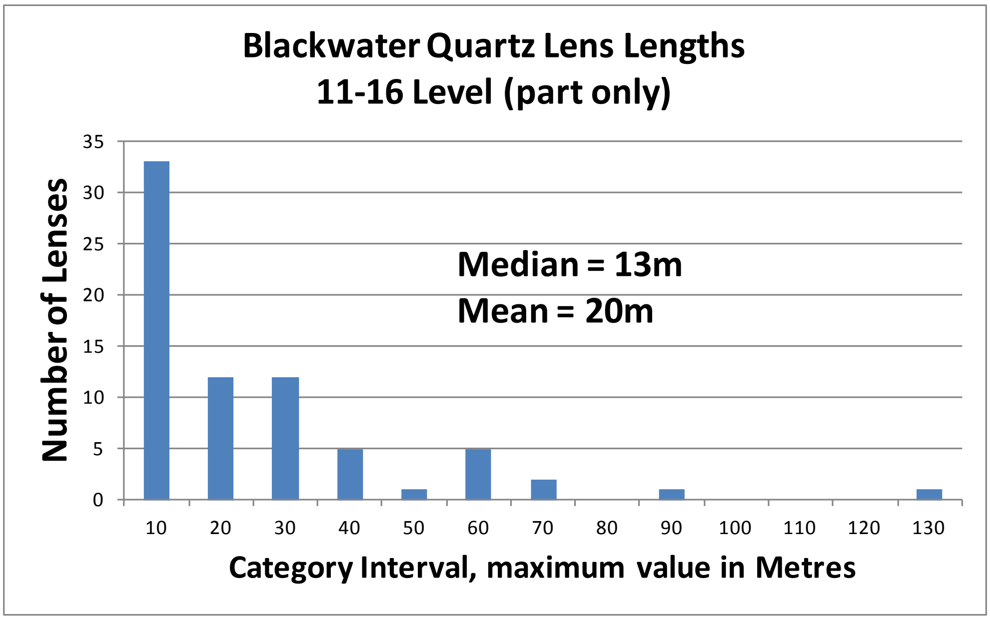

Figure 7-6: Blackwater Reef Lens Lengths - 11-26 Level(part of)

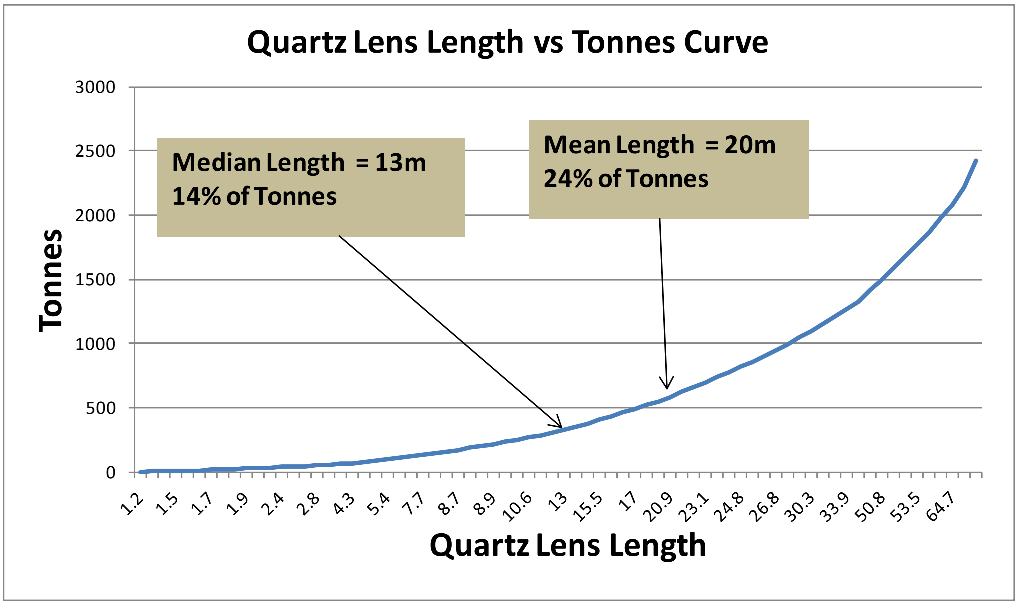

Figure 7-7: Blackwater Reef Tonnes vs Lens Length – 11-16 Level (part of)

The Birthday Reef ranged in thickness from 0.2 to 2.5m and averaged 0.65m, over the life of the mine. The reef trended at 030°, dipped at 75° west and pitched at about 40° to the north (Figure 7-11). Sixteen levels were developed in the mine with the seventeenth level just commenced at the time of mine closure.

The 16 Level is at 831m depth in the Prohibition Shaft, although as this shaft is situated on a hilltop the 16 Level is only about 710m below the surface at the Waiuta township. While the average in-situ grade was 21.9g/t Au, it was known to have varied from year to year within the range 16.9-27.0g/t Au.

The 16 Level sampling data (Figure 7-10) shows that for this level, the reef maintained its thickness (average of 0.59m) and grade (cut average grade of 24.7 g/t Au) to the lowest level of the mine. The drilling intercepts achieved below 16 levels had an average estimated true width of 0.5m and average uncut grade of 46.7 g/t Au.

Given the low number of intercepts, the grade variability, strong structural controls on reef thickness, and that WA11, WA21 and WA22 line up along the down-plunge projection of a shoot, it difficult to directly compare drilling results against averaged mined reef statistics. The range of drill hole and grades and reef widths however are consistent with those historically mined.

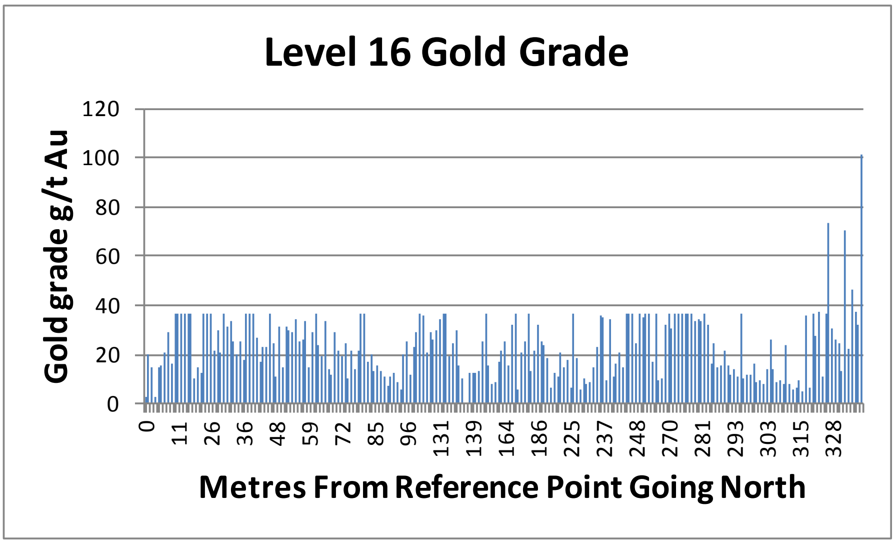

Figure 7-8: Level16 Face Sampling Data (Gold Grade)

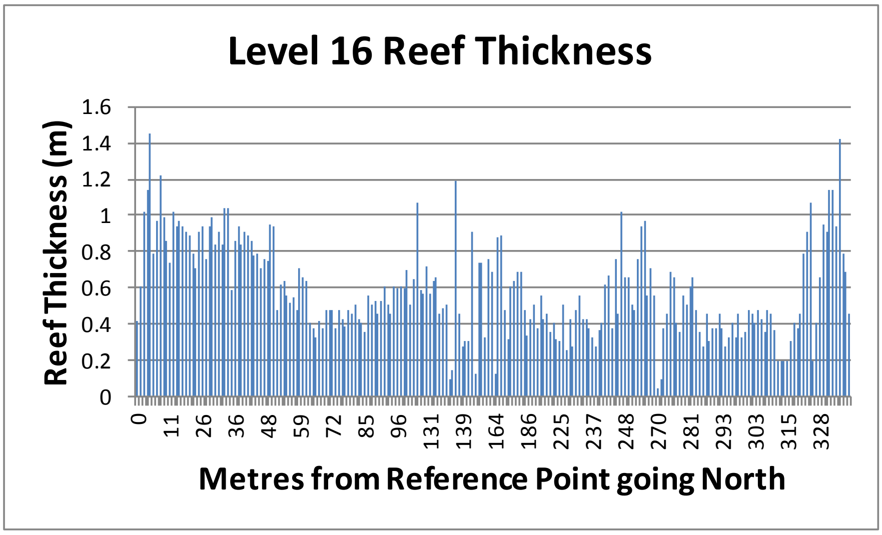

Figure 7-9: Level16 Face Sampling Data (Reef Thickness)

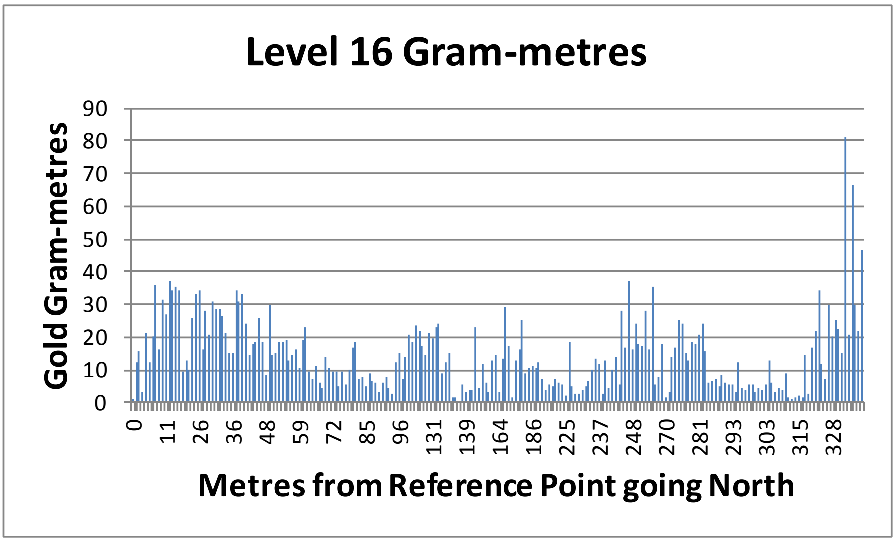

Figure 7-10: Level16 Face Sampling Data (Gram-metres)

**reference point is 13,212.63mE, 79,049.45mN,*

It is widely conjectured that a faulted offset to the Birthday Reef continues to the north of the Blackwater Gold Mine, where it is concealed beneath deep fluvioglacial deposits.

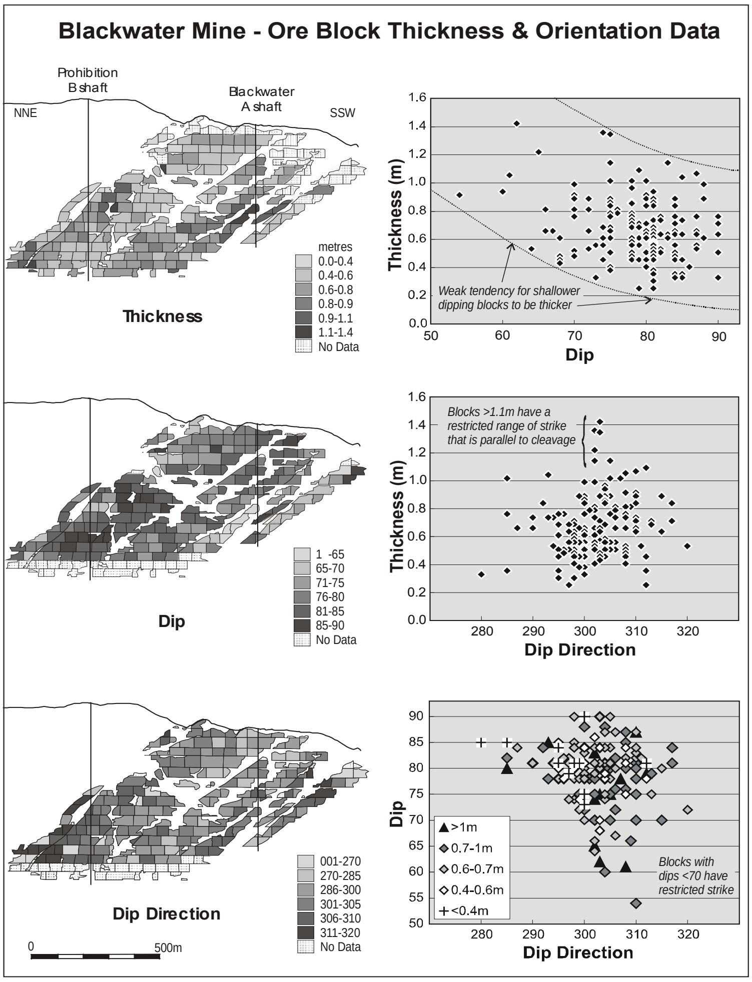

Figure 7-11: Blackwater Mine Stope Thickness and Orientation Data (Cox, 2000)

8 DEPOSIT TYPES

The Birthday Reef represents orogenic mesothermal quartz vein-hosted gold. These veins tend to strike sub-parallel with the regional geological structure; the Birthday Reef is situated in the western limb of a regionally significant anticline, proximal to the anticline hinge. Refer to Item 7 for more detail.

The focus of this Technical Report is an extension of the Birthday Reef, below the Blackwater Mine which was in operation from 1908 to 1951, as documented in Item 6. Cessation of mining in 1951 was a result of the failure of the mine shaft. Drilling completed to date (Item 9 and Item 10) has confirmed the continuation of the reef for at least 680m below the deepest level mined.

9 EXPLORATION

Exploration for significant undiscovered mineralisation in the vicinity of the Blackwater Mine has been the focus of numerous companies since the mine was established. Historically McVicars and Prohibition Adits were excavated east of the Birthday Reef attempting to intercept the Birthday Reef and/or a parallel Reef. Historic workings are also found in both the Snowy River (south circa 1km) and in Blackwater Creek (north circa 1km), indicating that historic miners searched extensively along strike to the north and south. In modern times geochemical sampling (both soils and wackering) has occurred in the area surrounding the mined lode. Extensive mullock, colluvium and glacial cover however, restricted its application. Structural mapping of the creeks and the slopes surrounding the Birthday Reef has been carried out by various geologists with the recent mapping by current OceanaGold staff being the most comprehensive. Structural assessment of geological data from both underground and surface exposure has guided exploration drill targeting.

10 DRILLING

Drilling in the last twenty years by OceanaGold and its predecessor company GRD Macraes focused on identifying extensions to the Birthday Reef, both down-dip and laterally. WA01 and WA02 targeted a possible extension of the Birthday Reef immediately northeast of the mine lode (Figure 10-1), WN0001-WN0007 targeted continuation of the mineralised reef and possible repetition up to 1.5km N-NE of the mined lode. BT001-BT005 targeted the along strike extent of the Birthday Reef up to 500m south of the mine lode. All drill holes failed to intercept significant mineralisation.

The area within 200m of the mined lode both north and south has not been completely tested for significant mineralisation. Mining records indicate that as the northern and southern extent of the mined lode is approached the reef is more dislocated. It is likely that small blocks of ore still remain beyond the northern and southern boundary of the mined lode. Whether these blocks are big enough and close enough to the surface to be economic is difficult to determine. Targets within these areas are currently being assessed by the Exploration Department.

All drill hole assay, survey and geology data is stored in an Acquire database which is maintained by the Reefton exploration team, and overseen by the Senior Database Geologist, who is based at OceanaGold's Macraes Gold Mine. The geological wireframes and drill hole locations use truncated NZMG coordinates whereby the first two digits are removed for both eastings and northings.

Drill holes WA10, WA11 and WA11A were drilled using a Vickers Keogh VK2500 rig in 1996, while WA21 to WA25 were drilled 2012 to 2013 using a UDR1200HC capable of drilling HQ core to a depth of 1,850m. The drill holes were designed to test the extent of the Birthday Reef below the historic workings. Pierce point targeting of the reef was compounded by unplanned drill hole deviation in conjunction with reasonably acute drill hole intersection angle with the Birthday Reef. Daughter holes were designed to re-evaluate zones of interest encountered in the parent, and designed to achieve a separation of 10m at the intersection of the Birthday Reef from the parent hole.

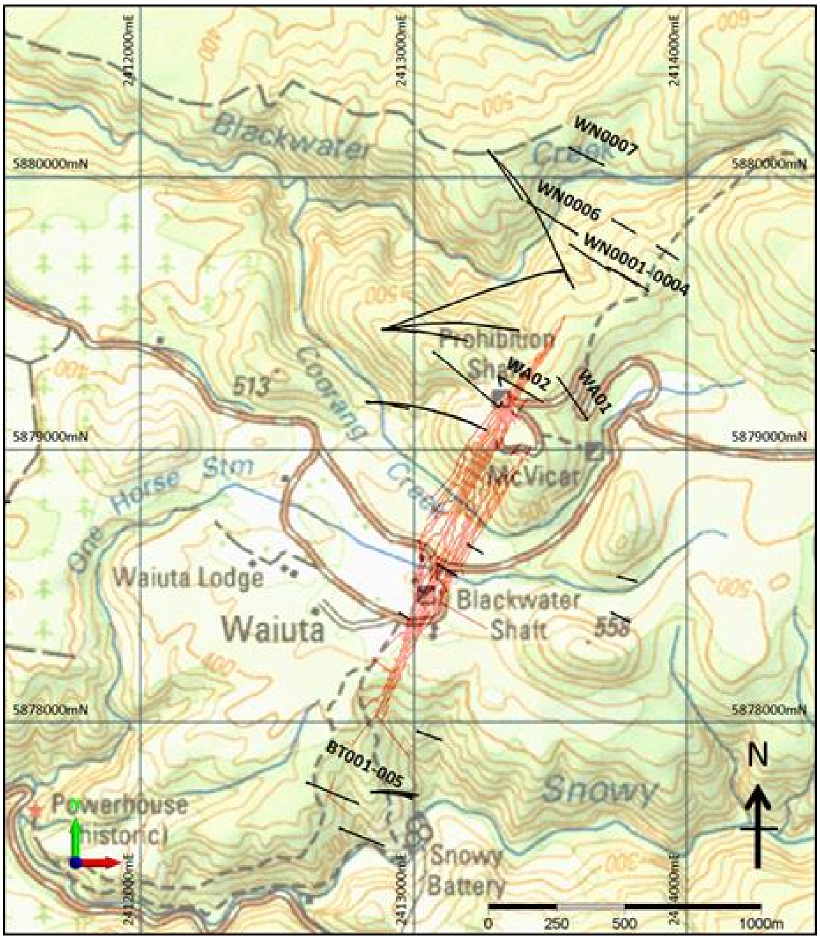

Figure 10-1: Blackwater Drilling

Holes WA10 & WA11 were diamond drilled from surface using triple tube coring equipment to optimise core recovery and reduce hole deviation. WA10 had to be abandoned when the drill string irretrievably snapped when the hole was at a depth of 687m. WA11 and WA11A successfully intersected the Birthday Reef. In total 2,869m were drilled.

Holes WA21 to WA25 were diamond drilled from surface using triple tube coring equipment to optimise core recovery and reduce hole deviation, with the exception of brief runs with a down-hole navigational drilling motor.

There were 5 holes drilled from surface and 6 daughter holes. Three of the surface holes and four of the daughter holes were successful in achieving target depth and intercepting the Birthday Reef. The unsuccessful holes were WA22, WA22A, WA22B, WA23, and WA24. Daughter holes are designated using the parent hole ID followed with a letter; e.g. WA22A is the first daughter to come from WA22. In total, 5,611.8m were drilled over the course of the program.

All drill core was orientated where the core was competent and a there was high confidence that there had been no rotation of core in the barrel.

All the Blackwater diamond drill holes were frequently down hole surveyed in order to attempt to manage hole deviation. Statistics on the down hole surveys for the holes that successfully intersected the Birthday Reef are shown in Table 10-1.

Drill holes WA11 & WA 11A were down hole surveyed using an Eastman Single shot down hole camera. The down hole survey data was then entered into a database. The developed camera discs from the down hole surveys are stored in the exploration office located at 1 Hattie Street, Reefton.

Drill holes WA21 to 25 were down hole surveyed using a REFLEX EZ-SHOT electronic single shot camera. The down hole surveys were written on the shift plods which are now stored in the exploration office located at 1 Hattie Street, Reefton. The down hole surveys were reviewed on a daily basis for consistency and then entered into the Acquire software database.

Table 10-1: Down Hole Survey Statistics

Hole ID	Number of Surveys	Minimum Survey Interval (m)	Maximum Survey Interval (m)	Average Survey Interval (m)	Median Survey Interval (m)
WA11	45	0.8	42	26.0	30
WA11A	24	0.6	60	15.5	7
WA21	144	0.8	31	9.6	9
WA22C	229	6	33	12.3	12
WA22D	126	6	33	13.0	12
WA25	121	3	30	10.6	12
WA25A	116	3	24	10.4	10

The drilling locations were situated in forested terrain approximately 16km to the south of Reefton. The holes were drilled from three pad sites (Table 10-2 and Figure 10-2). The pad sites are located approximately 0.5km to 1.5km north of the historic Waiuta mine.

The sites are accessible by travelling south from Reefton on State Highway 7 and turning East onto the Waiuta road. After 9km travel on the Waiuta road, the road turns from tar sealed to gravel. The drill sites are accessible from the gravel road.

Table 10-2: Drill Hole Co-ordinates and Drilled Depths

Hole ID	NZMG Coordinate*		Elevation (masl)	Hole Orientation		Daughter Depth Start (m)	Final Depth (m)	M Drilled
	East	North		Azimuth (Grid)	Dip			
WA10	2412835	5879174	438	90	-90		686.9	687
WA11	2412829	5879172	438	90	-65		1,171	1171
WA11A						644.4	1,011	527
WA21	2412888	5879439	528.681	83.5	-63.5		1378	1,378
WA21A						1,264.3	1324	60
WA22	2412888	5879439	528.68	65	-56		1121	1,122
WA22A						809.2	847	38
WA22B						815.2	863	47
WA22C						814.3	1675	861
WA22D						1,385.9	1641	255
WA23	2413278	5880086	540	143	-55		36	36
WA24	2413278	5880086	540	143.5	-51.5		363	364
WA25	2413278	5880086	540	140	-62		1282	1,282
WA25A						1036.2	1205	169
* GPS co-ordinates							**Total**	**7,997**

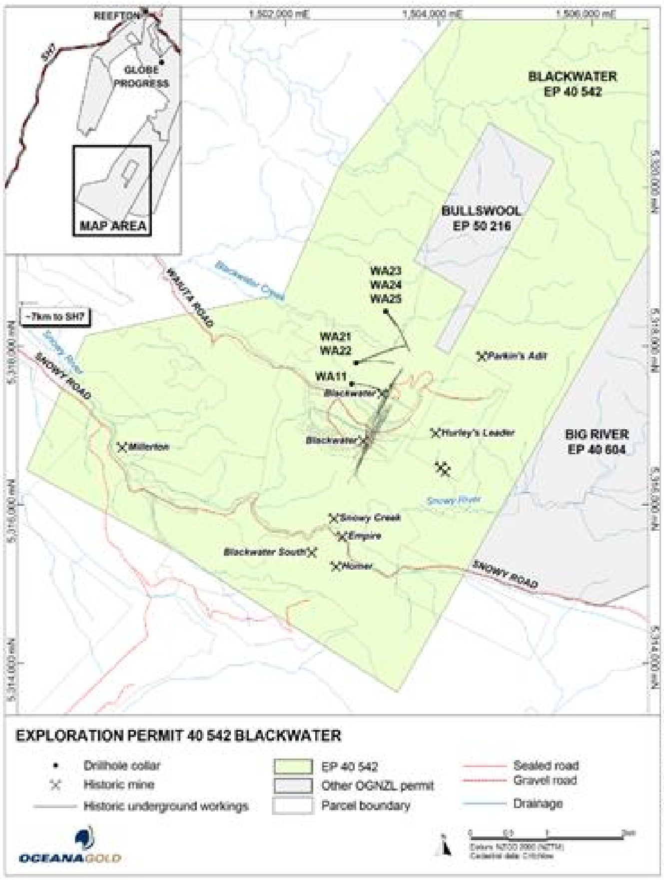

Figure 10-2: Blackwater Deeps Drill Pad Location & Access

The successful intersection of the Birthday Reef by four deep diamond holes (and their daughters) collared from surface in two campaigns in 1996 and 2010 to 2013 respectively supports the projected extension of the Birthday Reef. The intersection results are summarised in Table 10-3. The results are consistent with the range of historically mined widths and grades and indicate that the Birthday Reef continues for at least 680m vertically below the last worked level of the Blackwater Mine.

Table 10-3: Blackwater Deep Drill Hole Intercepts

Hole ID	From (m)	To (m)	Intercept (m)	True Width (m)	Grade (Au g/t)	Grade Width (g*m)	Comment
WA11	979.6	980.3	0.7	0.5	24.50	12.3	Parent Hole
WA11A	980.3	981.0	0.7	0.5	59.70	29.9	Daughter Hole
WA21A	1,315.9	1,316.8	0.9	0.5	23.30	11.7	Daughter Hole
WA22C	1,632.30	1,633.0	0.70	0.5	15.65	7.8	Parent Hole
WA22D	1,623.90	1,625.03	1.13	1.0	85.2	85.2	Daughter Hole
WA25	1,118.95	1,119.40	0.45	*0.35	31.8	11.1	Parent Hole
WA25	1,134.18	1,134.59	0.41	*0.3	62.4	18.7	Parent Hole
WA25	1,190.77	1,191.36	0.59	0.5	3.91	1.9	Parent Hole (BR)
WA25A	1,136.40	1,137.11	0.71	*0.5	134.00	67.0	Daughter Hole
WA25A	1,195.20	1,195.65	0.45	^0.4	61.90	24.7	Daughter Hole (BR)

** Indicates the upper intercept in each of the holes WA25 & WA25A interpreted as a fault repetition of the Birthday Reef. (BR) indicates the Birthday Reef intercept.*

^ Unorientated drill core. True width calculated using WA25 intercept.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

It is Mr. Moore's opinion that the procedures followed by OceanaGold for sample preparation, security and analytical measures followed accepted industry standards, and that the database is suitable for Resource Estimation. All drill core was logged to geological intervals, not fixed intervals. WA11 and WA11A were assayed over the entire length of the hole, however the sampling method and the sample length varied depending on geology and how mineralised the core appeared to be. In general, unmineralised sections of core were sampled in four metre lengths using an angle grinder to collect a grind sample.

Mineralised sections of the core were sampled predominately at 1m intervals or geologically defined intervals. Basic sampling statistics for WA11 & WA11A are shown in Table 11-1.

Table 11-1: Diamond Core Sample Length Statistics WA11 and WA11A

Statistics	Grinds	Statistics	Half Cut Core		
	All		All	≥ 0.2g/t	≥≥ 1.0g/t
Number	249	Number	280	5	2
Mean grade Au (g/t)	<0.01	Mean grade Au (g/t)	0.01	35.4	42.9
Min Length (m)	1.70	Min Length (m)	0.0	0.3	0.65
Max Length (m)	7.00	Max Length (m)	1.4	0.70	0.70
Mean Length (m)	3.77	Mean Length (m)	0.93	0.55	
Median Length (m)	4.00	Median Length (m)	1.00	0.65	
Proportion < 4m	27.7%	Proportion < 1m	11.8%		
Proportion 4m	65.1%	Proportion 1m	87.5%		
Proportion > 4m	7.2%	Proportion > 1m	0.7%		

For WA21 to WA25A only selected intervals were selected for assay. Samples were generally taken over 1m intervals in non-mineralised sections of the core, and to geologically defined lengths in the mineralised sections of the core.

Table 11-2 gives a breakdown of the sample lengths by assay and shows the bulk of the gold is in sample intervals of less than 1m. This is to be expected, as the average width of the Birthday Reef is 0.59m and the alteration halo around the Birthday Reef is 5m or less.

Diamond core recovery in the non-mineralised sections of the core was typically in the 90 to 100% range. The core recovery for the Birthday Reef for the holes that successfully intersected the Birthday Reef is shown in Table 11-3, which shows that core recovery of the Reef ore zone was in most cases 100%.

Table 11-2: Diamond Core Sample Length Statistics WA21 to WA25A

Statistics	Total	≥ 0.2g/t	≥ 1.0g/t
Number	590	32	17
Mean grade Au (g/t)	0.76	13.8	25.6
Min Length (m)	0.30	0.30	0.30
Max Length (m)	1.30	1.30	1.30
Mean Length (m)	0.97	0.84	0.75
Median Length (m)	1.00	1.00	0.71
Proportion < 1m	6.9%	46.8%	70.7%
Proportion 1m	92.0%	43.8%	17.6%
Proportion > 1m	1.1%	9.4%	11.7%

Table 11-3: Birthday Reef Core Recovery

Hole	Birthday Reef Intersection Zone				Core Recovery			
	From (m)	To (m)	Width (m)	Grade (g/t)	From (m)	To (m)	Width (m)	Recovery (%)
WA11	978.90	980.30	1.40	12.03	?	?	?	?
WA11A	217.75	218.40	0.65	63.40	217.4	218.4	1.0	100.0%
WA21	1,315.00	1,318.20	3.20	7.07	1,315.0	1,319.0	4.0	92.5%
WA22C	1,630.00	1,633.00	3.00	6.78	1,630.0	1,633.0	3.0	100.0%
WA22D	1,620.60	1,627.00	6.40	15.46	1,620.0	1,627.0	7.0	100.0%
WA25	1,117.00	1,119.40	2.40	6.18	1,117.0	1,120.0	3.0	100.0%
WA25	1,133.00	1,134.59	1.59	16.34	1,133.0	1,135.0	2.0	100.0%
WA25A	1,136.00	1,139.00	3.00	36.34	1,136.0	1139.0	3.0	100.0%

**Intersection zone defined as any interval assaying ≥0.1gt.*
#Core recovery measured on 1m basis.

11.1 Diamond Core Assay Methods for WA11 and WA11A

For WA11 and WA11A samples for assay were collected by either using an angle grinder to collect a grind sample of the core or by cutting the core in half using a diamond saw. The grind sampling method was used for sections thought to be unmineralised and half core sampling was reserved for mineralised core. The grind and half core were then sent to ALS Brisbane.

The Birthday Reef sections of WA11 and WA11A were screen fire assayed by ALS in Brisbane. Of the 529 samples submitted to ALS Brisbane for gold assay only two samples (Birthday Reef Intersections) were screen fire assayed. The core samples were assayed for gold by 50g fire assay and for Cu, Pb, Zn, As, Fe, Mn and Sb by ICP to the detection limits are shown in Table 11-4.

Table 11-4: Analysis Methods and Detection Limits of ALS Brisbane

Element	Units	Analysis Code	Detection Limit
Au (screen fire)	ppm	PM212	0.01
Au (50g fire)	ppm	PM209	0.01
Cu	ppm	IC587	5
Pb	ppm	IC587	5
Zn	ppm	IC587	5
As	ppm	IC587	5
Fe	%	IC587	0.01
Mn	ppm	IC587	5
Sb	ppm	IC587	5

11.2 Diamond Core Assay Methods for WA12 to WA25A

Samples were generally taken over 1m intervals in non-mineralised sections of the core, and to geologically defined lengths in the mineralised sections of the core. The gold particle distribution in the core appeared to be reasonably random and it is considered that the sampling of the core by cutting the core in half using a diamond saw will not introduce any sampling bias. Half the core was then sent to SGS Westport and SGS Reefton or to ALS Townsville if visible gold was observed or suspected. The half cut core was analysed for the elements listed in Table 11-5. Samples that were analysed at SGS Reefton were first sent to SGS Westport where they were prepared for analysis. The samples then returned to SGS Reefton for analysis.

32 of the 590 core samples submitted returned an assay of 0.2g/t gold or greater. Of the 32 samples 30 of the gold assays were by screen fire assay with the remaining two assays (0.24g/t & 0.61g/t) being by fire assay of 50g charge.

All SGS samples were dried, then crushed down to 2mm, if samples were >200g, they were split down to 200g. The sample was then pulverised so that 85% of material passes 75 microns. Equipment was cleaned in between processing each sample.

For FAA505 the pulverized sample was split down to a 50g charge and mixed with a fluxing agent. The flux assists in melting, helps fuse the sample at a reasonable temperature and promotes separation of the gangue material from the precious metals. In addition to the flux, lead is added as a collector. The sample was then heated in a furnace where it fuses and separates from the collector metal 'button', which contains the precious minerals.

Once the button is separated from the gangue, the precious metals are extracted from the collector through a process called cupellation. Once the button has cooled, it is separated from the slag and cupelled. When lead is used as a collector, the lead oxidizes and is absorbed into the cupel leaving a precious metal bead.

The sample is then finished by flame atomic absorption. The bead is dissolved in aqua regia and then aspirated in an acetylene flame. A beam of light at a wavelength matching that of gold is passed through the flame. The gold in the sample absorbs the light proportionately depending on the concentration of the element in the solution. The absorption is compared to standard solutions to determine gold concentration in the sample.

XRF75 involves mixing 5g of the above prepared sample into a wax binding agent and compressing the sample by 22t to create a pellet for XRF analysis.

When samples were sent to ALS Townsville for screen fire assay they were prepared by crushing to 70% less than 6mm, the sample was then pulverised down to 100 microns. For Au-SCR22AA a 1,000 g of the final prepared pulp was passed through a 100 micron (Tyler 150 mesh) stainless steel screen to separate the oversize fractions. Any +100 micron material remaining on the screen is retained and analyzed in its entirety by fire assay with gravimetric finish and reported as the Au (+) fraction result. The –100 micron fraction is homogenized and two sub-samples are analyzed by fire assay with AAS finish (Au-AA25 and Au-AA25D). The average of the two AAS results is taken and reported as the Au (-) fraction result. All three values are used in calculating the combined gold content of the plus and minus fractions. The equation used to determine the Au value is shown in Figure 11-1.

In the fire assay procedure, the sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required in order producing a lead button. The lead button, containing the precious metals, is cupelled to remove the lead and the resulting precious metal bead is parted in dilute nitric acid, annealed and weighed to determine gold content.

The gold values for both the +100 and -100 micron fractions are reported together with the weight of each fraction as well as the calculated total gold content of the sample.

$$Au - avg = \frac{Au-(1) + Au-(2)}{2}$$

$$AuTotal\,(g/t) = \frac{(Au - avg\,(g/t) \times Wt.Minus(g) \times 10^{-6}\,t/g) + (Weight\ Au\ in\ Plus(mg) \times 10^{-3}\,g/mg)}{(Wt.Minus(g) + Wt.Plus(g)) \times 10^{-6}\,t/g}$$

Figure 11-1: Equation for calculating Au Total with Screen Fire Assay (1kg)

For ME-ICP analysis a prepared sample (0.25 g) is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by inductively coupled plasma-atomic emission spectrometry. Results are corrected for spectral inter-element interferences. Elements Fe and S were only assayed for the ALS submissions.

Table 11-5: Analysis Methods and Detection Limits of ALS and SGS Laboratories

Laboratory	Element	Units	Analysis Code	Detection Limit
ALS Townsville *(screen fire)	Au	Ppm	Au-SCR22AA	0.002
ALS Townsville	Sb	Ppm	ME-ICP61	5
ALS Townsville	As	Ppm	ME-ICP61	5
ALS Townsville	Fe	%	ME-ICP61	0.01
ALS Townsville	S	%	ME-ICP61	0.01
SGS Reefton	Au	Ppm	FAA505	0.01
SGS Westport	As	Ppm	XRF75V	2
SGS Westport	Sb	ppm	XRF75V	3

11.3 Quality Control, Assurance and Results WA11 to WA11A

The QAQC data for WA11 and WA11A, drilled in 1996 was not available at the time of writing. As the WA11 / WA11A assays are not directly used for the resource estimate and that core remains, this is not believed to materially affect the estimate.

11.4 Quality Control, Assurance and Results WA21 to WA25A

Diamond core submissions included a minimum of two blanks, one standard and at least one lab duplicate taken after coarse crushing of the sample, when sent to SGS Laboratories.

Samples that were suspected to have or contained fine to coarse visible gold were sent to ALS Townsville. Submissions to ALS Townsville contained a minimum of two blanks, and one standard. Where intervals contained or were suspected to contain fine to coarse visible gold, each sample was followed with two quartz flushes.

On return of assay results, standard data was analysed and any failure of standards within a batch (i.e. standard results greater or less than two standard deviations from the certified standard value) were noted.

It would be determined on a case by case basis if re-assay was required where significant or multiple standard failure within a submission occurred. The QA/QC results for the respective laboratories are tabulated in Table 11-6 and below Table 11-7.

All assay data is imported into the Reefton project Acquire database directly from laboratory reports. All geological log, survey and assay data were imported into the Reefton project acQuire database. Logging was entered into an excel spread sheet on a HP pro book 6540b laptop and then imported into acQuire in csv format.

Table 11-6: Sample Statistics From Standards Sent to ALS Lab Townsville

Standard	Blank 4	SE44	SH41	SK52	SL51	SL61	SN50	Si54
Number submitted	24	1	1	1	1	1	4	2
Min	0.005	0.64	1.31	4.05	6.3	6.2	7.96	1.76
Max	0.06	0.64	1.31	4.05	6.3	6.2	9.13	1.79
Mean	0.014	--	--	--	--	--	8.695	1.78
Median	0.01	--	--	--	--	--	8.845	1.78
75th Percentile Value	0.01	--	--	--	--	--	9.055	1.78
Total Range	0.055	--	--	--	--	--	1.17	0.08
Standard Deviation	0.015	--	--	--	--	--	0.53	0.021
Expected Result	0	0.61	1.34	4.11	5.91	5.93	8.69	1.78
# Outside Error Limit	0	0	0	0	0	0	1	0

Table 11-7: Sample Statistics from Standards Sent to SGS Reefton

Standard	Blank 4	SH41	Si54	SG66	SF57	OxE74
Number submitted	26	1	3	1	4	1
Min	0.005	1.3	1.72	1.07	0.77	0.61
Max	0.01	1.3	1.81	1.07	0.84	0.61
Mean	0.005	--	1.773	--	0.81	--
Median	0.005	--	1.79	--	0.82	--
75th Percentile Value	0.005	--	1.8	--	0.84	--
Total Range	0.005	--	0.09	--	0.07	--
Standard Deviation	0.00098	--	0.047	--	0.034	--
Expected Result	0	1.34	1.78	1.09	0.85	0.62
# Outside Error Limit	0	0	0	0	1	0

11.5 In-Situ Density Sampling and Test Work

No in situ density, moisture content or porosity sampling test work has been completed, given the paucity of drill core. The assumptions are based on sampling of the nearby Globe deposit where equivalent rock-types occur.

11.6 Residual Sample Storage

Residual sample was held at the processing laboratory for a nominal period (generally 90 days) in case of a requirement to re-assay before being returned to OceanaGold. Residual samples for WA21 to WA25A (in the form of sample pulps and coarse reject) are now stored at the OceanaGold NZL exploration office located at 1 Hattie Street, Reefton, New Zealand.

The non-assayed half cut core diamond core is stored at the OceanaGold NZL exploration office located at 1 Hattie Street, Reefton. No core or residual assay sample material remains for drill hole WA11 or the daughter hole WA11A drilled in 1996. The core and residual assay sample material were used in 2003 for metallurgical test work and mineralogical studies as part of the 2003 Blackwater Scoping Study completed by GRD Macraes Ltd.

12 DATA VERIFICATION

The authors reviewed the existing data of all available past and recent reports. Limitations on verification undertaken on the historic mining and processing data are discussed in Item 6 of this report. This data was used to create an estimate of the historically mined Birthday Reef.

Drill holes were used to test for the presence of an extension of the mineralised reef below the historic workings, and the grade tenor of the drill holes was consistent with the range seen in the historically mined reef. The authors believe that the sample preparation, security and analytical procedures undertaken for these drill holes were correctly applied and correspond with the present standards applied in the mining industry. It should be noted that sample information from these drill holes was not directly used for grade estimation, but rather as a means to broadly compare unmined to mined resource.

The Mineral Resource estimate detailed in Item 14, which is the basis for this PEA Technical Report, is an Inferred Mineral Resource, being that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. The authors believe that the historic mining data coupled with the more recently completed exploration diamond drill holes provides sufficient confidence in the estimate to support this resource classification.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Blackwater ore was successfully processed at two different processing plants between 1908 and 1951. Records indicate achieved recovery of Gold was between 85% and 95% using a combination of gravity, flotation and cyanide leach processes.

The lack of availability of ore samples from the intended mining area has led to a test work campaign on materials sourced from waste dumps from higher mined levels of the reef. Interpretations of historical descriptions from processing flowsheets, operating data and recent test work have been the foundation sources of data.

Test work completed on material sourced from the current diamond core and surface sampling around the mine waste dumps indicate a high level of gravity recoverable gold. Processing will involve producing a high mass recovery gravity gold concentrate followed by fine gold flotation. This process should recover in excess of 97% of the gold into a high grade concentrate. Intensive leaching of the concentrates and electro-winning will allow production of doré on site with estimated overall recovery in excess of 96%.

Variability testing on deep sourced core samples is not a viable option given the depth, cost and lack of core material available - this represents a risk to the gold recovery assumptions used in the PEA.

13.1 Introduction

The metallurgical flowsheet development for this project has relied heavily on the well recorded Blackwater Mine production history spanning some 40 years between 1908 and 1951. Metallurgical testing on limited diamond core samples and surface sampling of the existing mine waste dumps has been completed to support the conclusions drawn from the production records. The proposed ore processing flowsheet has a strong similarity to that used by past operators.

The Metallurgical test work focused only on ore characterisation and confirmation that the proposed flowsheet would achieve gold recoveries of at least 96%. Initial test work results indicate that gold recoveries up to 97% may be possible although this will require further metallurgical evaluation.

It is assumed that samples tested are representative of the whole ore-body. The particle size of samples retrieved from historical waste dumps was >40mm and the condition was generally good with some staining caused by organic matter on surface.

13.2 Historical Processing

The Birthday Reef was treated at Waiuta through two process plants, whose locations are shown in Figure 13-1. The first plant operated from 1908 to 1938, and was located in the Snowy River. Ore was transported from the Blackwater Shaft via an aerial tramway (Figure 13-1). In 1938 the second plant was built adjacent to the Prohibition Shaft which became the main haulage shaft in 1936. It was this second plant that has influenced the design of the proposed Blackwater Mine flowsheet. Plant descriptions have been sourced from mine statements from 1908 – 1951, and reports by J Henderson (1917), RJ Morgan (1929), EW Pearson (1942) and GP Hutton (1947). Figure 13-4 plots the gold production and throughput of the Blackwater process plant from 1908 to 1951 (New Zealand Mines Statements, 1908-1952).

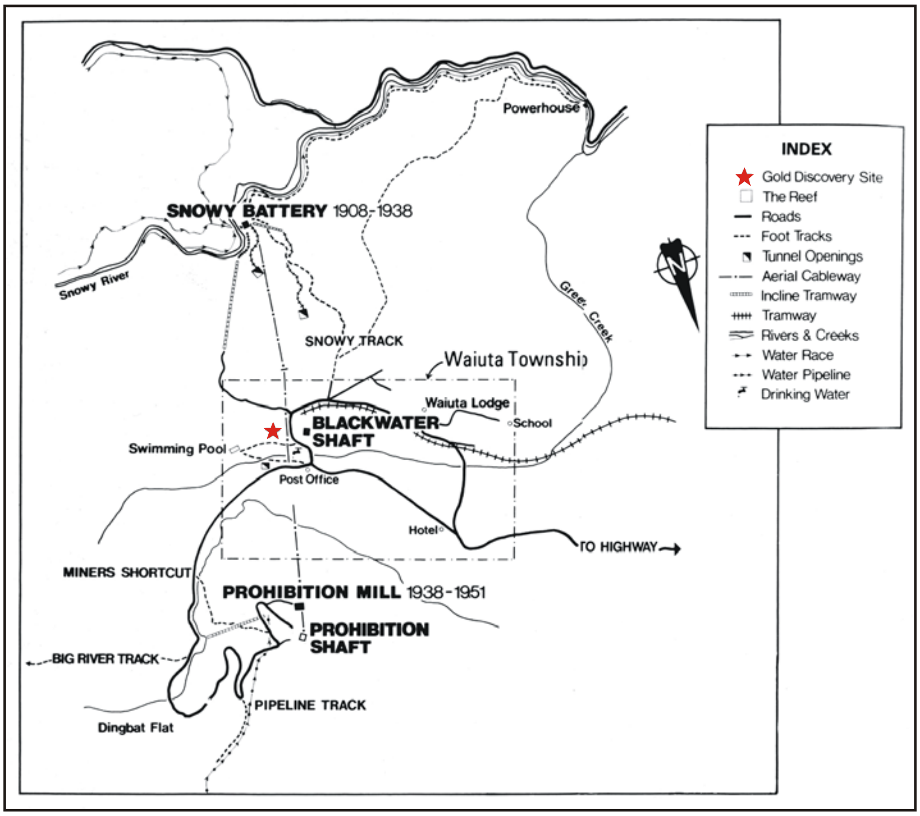

Figure 13-1: Infrastructure of the Blackwater Mine from 1908 to 1951

Historical reports describe the metallurgical flowsheet development at Blackwater as being heavily influenced by the testing and practice adopted at the Globe mine due to the perceived geographical similarities. The drive for including roasting and cyanide leaching was based more on the performance of un-oxidised Globe concentrates rather than specific testing on Blackwater ore itself. Current knowledge of the mineralogy at Globe around the original workings and the surrounding halo shows a significant difference in the style of mineralisation and the potential flowsheets needed.

13.2.1 The First Blackwater Plant

The first treatment plant was described as “simple and inefficient” due to the poor grind size control of the stamps and amalgamation methods. Ore was hand fed to mortars and crushed using slow running light stamps - the crushed pulp escaped through punched gratings of coarse mesh flowing over amalgamated plates. This recovered the majority of the gold. The plate tailings were separated into “slimes” and “sand” in a Spitz-lutten (early version of a classifier).

The sand (coarse fraction) from the Spitz-lutten was treated over a Wilfley table to recover gravity gold. The tailings from the Wilfley table were leached with cyanide solution in a vat before gold was recovered from the pregnant cyanide solution by precipitation on zinc shavings. The concentrate from the Wilfley table was treated over an amalgamation table to recover any free gold before being roasted in an Edwards roasting furnace. The roasted product was then leached in a strong cyanide solution with gold from the resulting pregnant liquor precipitated on zinc shavings.

The slimes from the Spitz-lutten were thickened in a Dorr thickener prior to leaching with a weak cyanide solution in agitated Pachuca tanks. The pregnant leach solution was separated by decantation and gold in liquor was precipitated onto zinc shavings. The thickened slimes were treated over blanket tables to recover fine gold prior to reporting to tailings discharge. The initial Snowy River Battery is shown in Figure 13-2 (Hancox,1985).

Henderson (1917) quoted gold recovery from the first plant as 89-90%, however Morgan (1929), noted the poor sampling quality and regimes making it difficult to confirm this recovery.

Figure 13-2: Snowy River Battery 1909

13.2.2 The Second Blackwater Plant

Gold losses and circuit inefficiencies resulted in the design and construction of an upgraded treatment plant incorporating grinding, gravity, flotation and leaching operations in 1937. The new circuit was completed and commissioned in July 1938. The process description was sourced from the 1938 Mines Statement and a flowsheet is shown in Figure 13-3 (Pearson, 1942).

Run-of-Mine (ROM) ore was delivered to the crusher feed bin. A Ross feeder fed the ore onto an 18" conveyor belt and into the primary jaw crusher. The ore was crushed to a 2" discharge size. A second 16" conveyor belt delivered the crushed ore to a 500t capacity mill feed surge bin. Ore was fed from the surge bin via a challenger ore feeder over a third conveyor belt and weightometer system into a 6ft x 6ft Marcy ball mill.

Cresylic acid, pine oil and xanthate were added to the ball mill feed so that the mill was used to assist in slurry conditioning. The mill discharge was to a ¼" mesh.

The Marcy ball mill discharge slurry was pumped over a series of primary blanket strakes to collect heavy concentrates and coarse gold. The primary strake concentrate was treated in an amalgam barrel to recover this coarse gold. The primary strake overflow was separated in a Dorr rake classifier where the underflow was delivered to an 11ft x 5ft ball tube mill and the overflow was delivered to the secondary blanket strakes.

The concentrate from the secondary blanket strakes was also treated in an amalgam barrel (as was the primary strake concentrate) to recover the coarse gold. The discharge from the ball tube mill (ground to 60 mesh – 250µm) was combined with the Marcy mill discharge and returned to the primary strakes.

The overflow from the secondary strakes was fed to a surge tank and then to a bank of six Fageren flotation cells. Concentrate recovered from the first two cells was thickened in the No.1 Dorr thickener for further treatment in the cyanide section while concentrate recovered from the last four flotation cells was combined with the overflow from the Dorr rake classifier and recycled to the secondary strakes. Tailings from the Fageren flotation cells reported as waste.

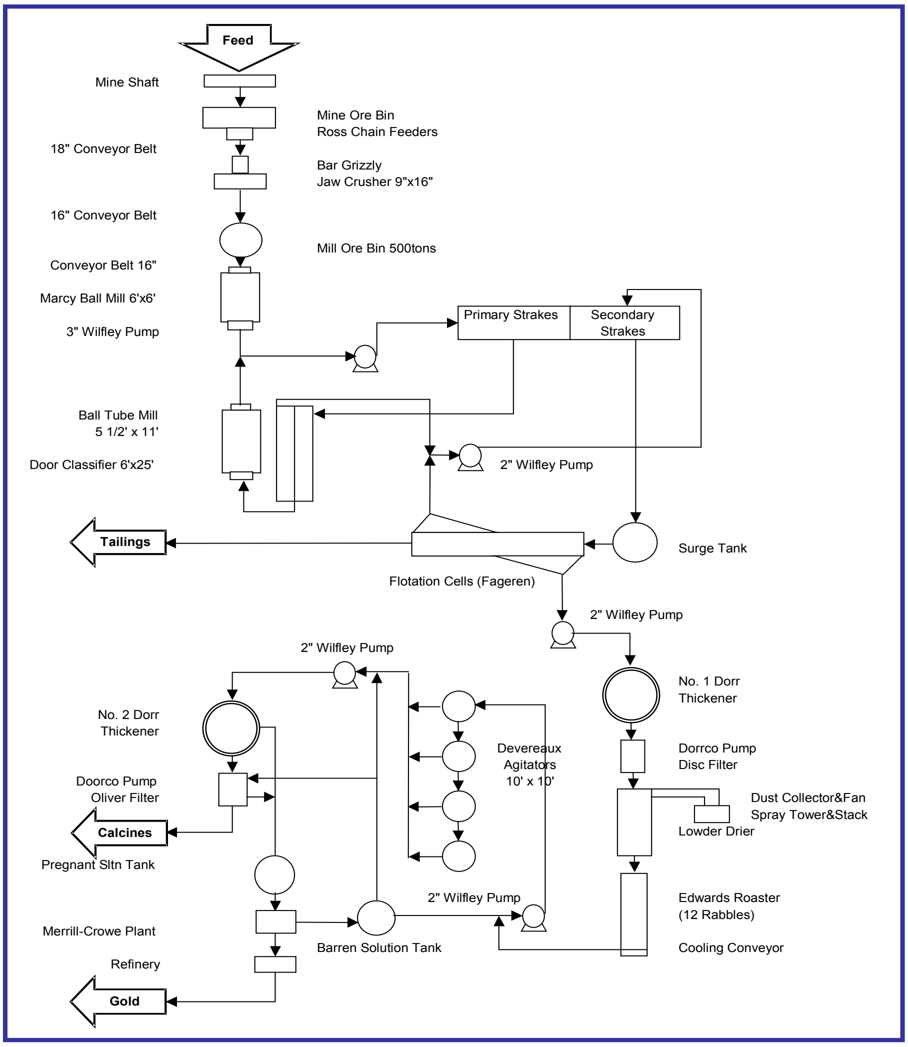

Figure 13-3: Prohibition Mill Flowsheet

The overflow from the No.1 Dorr thickener was returned to the flotation circuit and the thickened concentrate underflow was delivered to a disk filter. Dewatered concentrates from the disk filter were passed over a Lowden drier to remove excess moisture before being delivered via steel storage tank and screw feeder to an Edwards Roaster. Flue gases from the roaster were fed through a dust collector and condensing tower so that dust could be directed back to the No.1 Dorr thickener. In later years the mine extended the flume stack to remove the arsenic and sulphur laden fumes from the vicinity of the workings.

The roasted concentrates were cooled on a cooling conveyor prior to cyanide leaching in one of four Devereaux agitators. Cyanide leach was single stage. After a predetermined residence time the cyanide slurry was thickened in the No.2 Dorr thickener. The thickener underflow pulp was delivered to an Oliver filter.

The pregnant liquor from the No.2 Dorr thickener overflow and the Oliver filter were sent to a Merrill-Crowe plant where gold was precipitated using zinc dust. The zinc-gold precipitate from the Merrill-Crowe plant was collected in a bag filter, dried and then fluxed and melted in a tilting oil furnace to produce gold bullion.

The recovery of gold from the second Blackwater processing plant was reported by the late mining superintendent, JR Hogg as 96% gold recovery equating to 9.31dwt of gold recovered for 9.7dwt fed (Hogg, 1956). These recoveries were supported by GJ Williams (1965) who quoted the Blackwater gold recovery at 92.5% of which 80% of the gold was free.

13.2.3 Historical Production

Gold production from the Blackwater Mines was primarily dictated by mining rates from the underground operation. Ore processed was generally in the 40-50,000tpa range with gold poured in the 20-25,000ozpa range. Historical records reported that shortages of labour particularly during the two wartime periods had a significant impact on ore production and mine development leading to reduced subsequent production.

The graph in Figure 13-4 shows the mine production rates over the total operating life. Gold recovery estimates from different sources vary with the Prohibition mill achieving estimated recovery up to 95%.

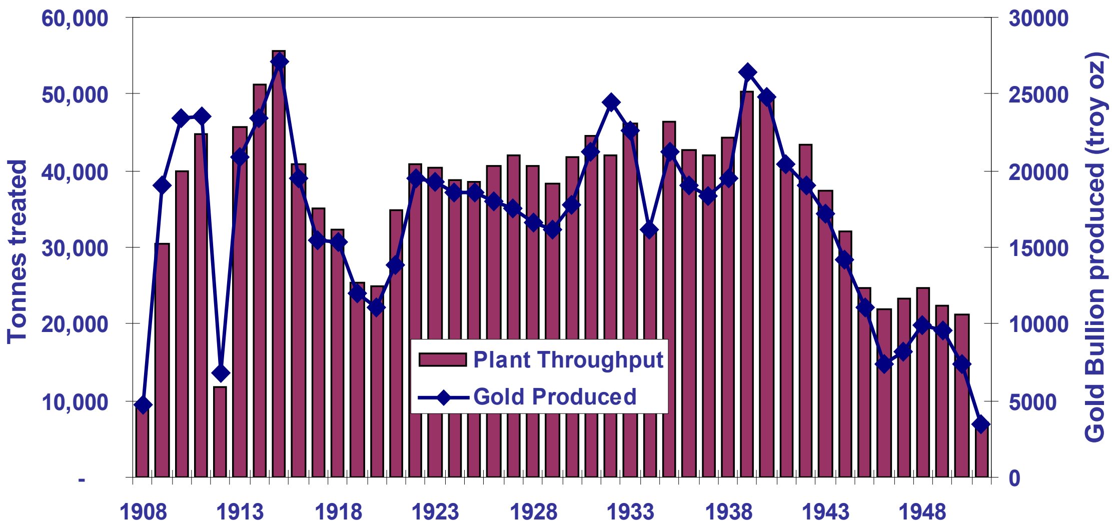

Figure 13-4: Historical mine production

For a total of 43 years, Blackwater Mines Ltd. extracted gold-bearing quartz from the Birthday Reef, producing 740,403oz from 1,582,379t of ore (14.55g/t Au recovered) at a recovery of 95%.

13.3 Mineralogical Review

13.3.1 Historical Mineralogy

The Birthday Reef is an essentially free milling ore containing a major part of its gold content as free gold recoverable by amalgamation (Morgan, 1929). The ore contains small quantities of metallic sulphides probably less than 1% and that these sulphides were entirely iron pyrites and arsenical pyrites. Some copper pyrite was observed but its occurrence was sporadic and the quantity so small that it was only of mineralogical interest and insignificant from the point of view of treatment of the ore (Morgan, 1929).

In July 1996, deep drilling at Blackwater resulted in two intersections of the Birthday Reef and two diamond drill core vein samples, the first from the parent hole WA11 and the second from the daughter hole WA11A. The WA11 hole recorded an intersection of 0.7m at 24g/t Au and the WA11A hole recorded an intersection of 0.65m at 63.4g/t Au. The core trays and ore intersections of the WA11 and WA11A samples are shown in Figure 13-5 and Figure 13-6 respectively.

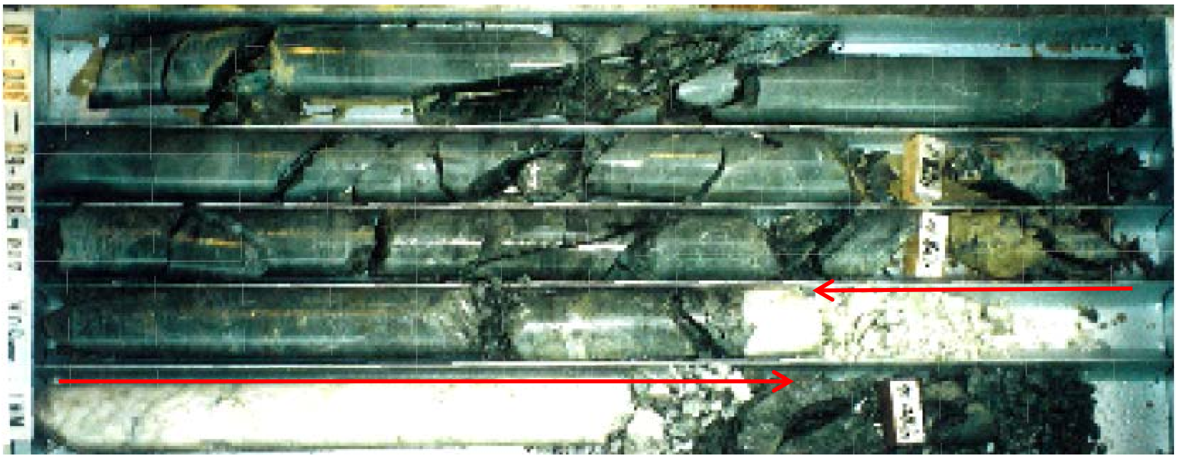

Figure 13-5: WA11 Core Sample (Reef intercept highlighted by red line)

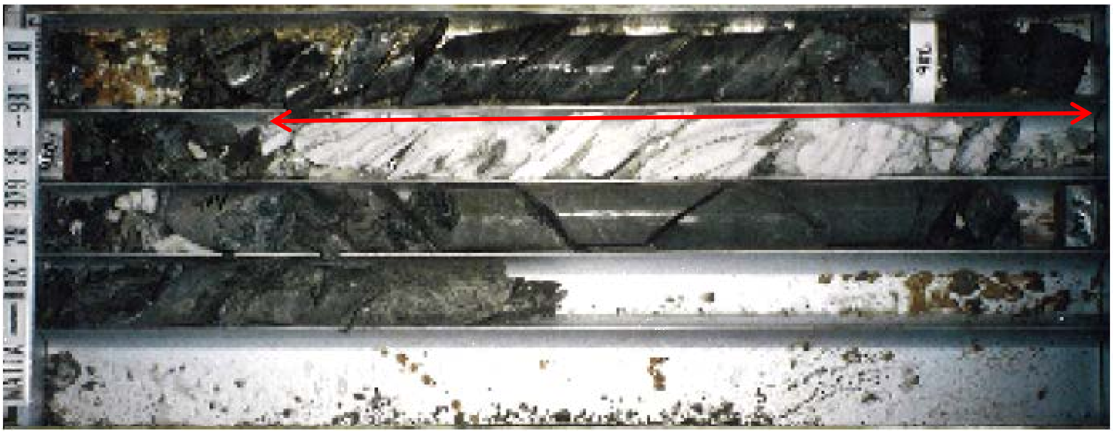

Figure 13-6: WA11A Core Sample (Reef intercept highlighted by red line)

Initial head assays and screen fire assays were carried out on both core samples in 1996 to 1997. The results of this investigation are shown in Table 13-1. The investigation showed that over 70% of the gold in both samples reported to the +75µm fraction, indicating that more than 70% of the gold contained in either of these samples could be expected to report to a gravity circuit concentrate. The initial screen fire assays also highlighted the wide variation in gold content between the two samples as a result of the spotty nature of gravity gold.

Table 13-1: Initial Head Assay and Screen Fire Assay Results for WA11 Intercepts

Analysis	Units	WA11		WA11A	
Intersection	m	979.6-980.3		217.75-218.4	
Interval	m	0.70		0.65	
Assays	Au g/t	24		63.4	
	Fe %	1.08			
	As ppm	202			
Head Assay	Au g/t	24.0	25.1*	63.4	56.0*
%Gold to +75µm	%	75.2	86.6*	73.3	73.3*

**Duplicate assay*

13.3.2 Mineralogy

For the 2003 test work the remaining WA11 ore sample was sent to Ammtec for analysis. The mineralogical analysis was completed by Roger Townend and Associates and the report on the analysis was included in the Ammtec report (Ammtec, 2003). The data in Table 13-2 shows the mineralogical analysis of the WA11 sample.

Table 13-2: Further Head Assay and Screen Fire Assay Results for WA11 Intercepts

Analysis	Units	Fraction 1	Fraction 2
Size	mm	+0.1	-0.1
Weight	%	86.9	13.1
TBE Sinks	%	1.3	3.5
Composition	Major	Pyrite, Arsenopyrite	Pyrite, Arsenopyrite
	Minor	Rutile, Chalcopyrite	Chalcocite, Pyrrhotite, Sphalerite
	Trace	Marcasite, Bi>Sb>Ni>Pb Sulphides, Sphalerite	Gold
	Accessory	Gold	Ore
Gold Occurrences	Number	19	10
	Liberated	11	7
	Sulphide Composites	4	3
	Non Opaque Gangue	4	-
Size	mm	+0.1	-0.1

In the +0.1mm size fraction the sulphide material was found to be predominantly unoxidised, liberated pyrite and arsenopyrite in crystals up to 0.5mm. Sulphide in quartz was also detected and marcasite was thought to be enclosed in pyrite. Chalcopyrite was present as liberated grains of 0.25mm.

In the +0.1mm sample nineteen occurrences of free gold were detected. Eleven of the free gold occurrences were liberated, angular particles of either hackly 0.1 to 0.25mm or flaky 0.25 x 0.025mm size. The other gold occurrences were inclusions in sulphide (pyrite, sphalerite, and chalcopyrite) and gangue.

The -0.1mm size fraction contained approximately 5% sulphides. The sulphides were largely liberated in the 0.05 to 0.15mm range. Most were angular and the pyrites frequently elongated. Composites observed were of chalcopyrite/pyrite and sphalerite/pyrite. Ten samples of free gold were detected. Seven were liberated, with shapes ranging from equant to hackly, and sizes from 20µm and rounded to 0.1mm and hackly. Three gold particles had small quantities of pyrite attached.

The gangue material was found to be over 80% quartz with the remainder a combination of feldspars (K Feldspar), chlorite and ankerite.

Figure 13-7 to Figure 13-9 illustrate three different occurrences of gold in the WA11 sample.

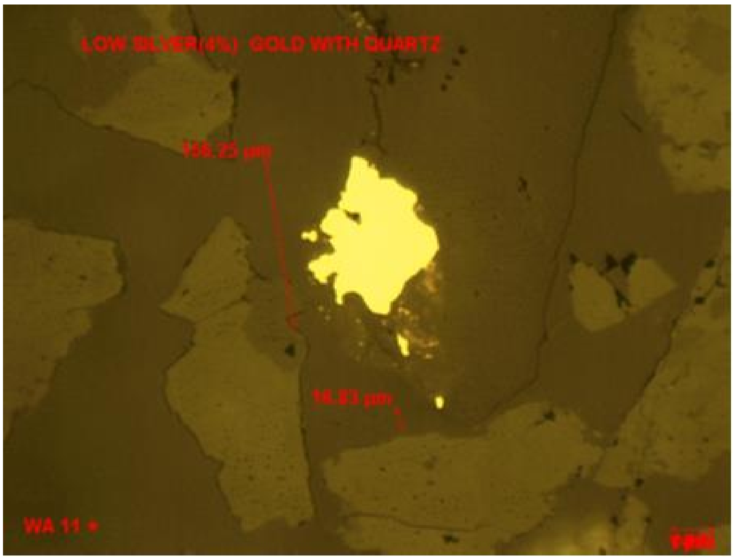

Figure 13-7: SEM Image of Gold with Quartz in WA11 Sample

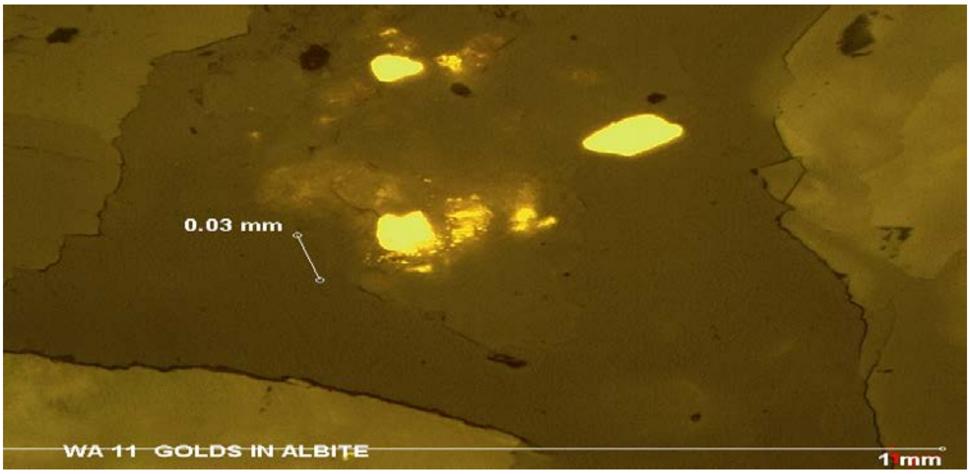

Figure 13-8: SEM Image of Gold with Albite in WA11 Sample

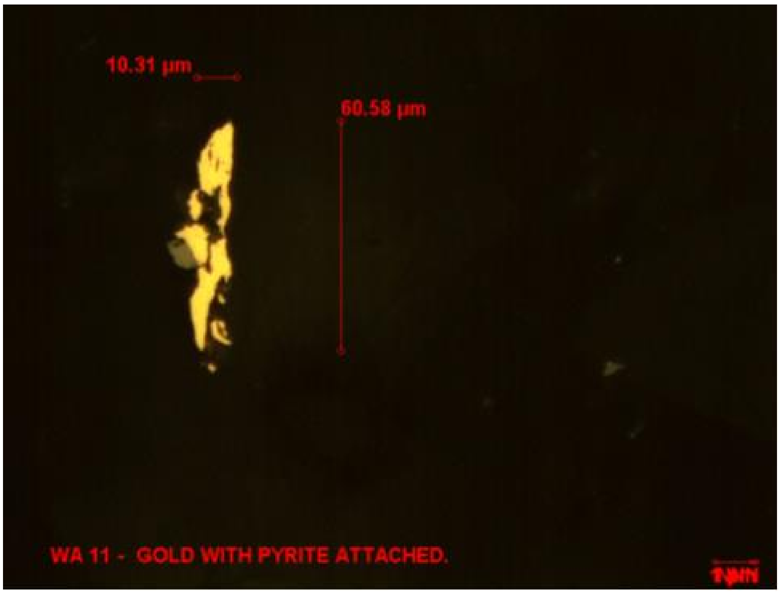

Figure 13-9: SEM Image of Gold on Pyrite in WA11 Sample

13.4 Metallurgical Test Work Review

13.4.1 Ammtec Metallurgical Test Work (Ammtec, 2003)

During 2003 metallurgical characterisation and test work were conducted on a sample of WA11. This sample was selected so that core from the larger WA11A hole could be left intact. The limited size of the metallurgical sample (~1.4kg) meant that the test work conducted was restricted but sufficient to get an indication of the effectiveness of the proposed flowsheet. All metallurgical test work was carried out at Ammtec Laboratories, Australia.

In addition to the mineralogical characterisation completed by Roger Townend and Associates further characterisation completed by Ammtec included multi element analysis and diagnostic analysis. The diagnostic analysis was done using a combination of amalgamation, flotation and cyanide leach.

The results of the multi element analysis are shown in Table 13-3 (Ammtec, 2003).

Table 13-3: Multi Element Analysis Results for WA11 Intercepts

Element	Units	Assay	Duplicate	Mean
Au	ppm	58.0	48.8	53.4
Ag	ppm	3		
As	ppm	300		
Ca	%	1.01		
Co	ppm	<5		
Cu	ppm	55		
Fe	%	1.12		
Mg	ppm	6200		
Na	ppm	1800		
Ni	ppm	23		
S	%	0.09		
Zn	ppm	228		

The diagnostic analysis program was carried out as a four-step process:

1. Grind to 106µm;
2. Quantify the gravity recoverable gold by amalgamation with Mercury;
3. Recovery of sulphides by flotation using the Globe Progress rougher-scavenger flotation scheme (Na_2CO_3 was not used because of the visual absence of "pug" material and clays in the ore sample); and
4. Differentiation of gold deportment to tailings and concentrate using cyanide leach.

The diagnostic analysis showed that 87% of the gold in the WA11 sample tested was gravity recoverable. Of the remaining material 10.4% was recoverable by flotation at a 1.74% mass pull. 60% of the flotation concentrate and 80% of the flotation tails were cyanide soluble. The back calculated head assay for this sample was 58.3g/t compared to the average head assay of 53.4g/t.

The Ammtec test work showed that 97% of the gold was recoverable by treatment through a gravity and flotation circuit and that 80% of the material remaining in the tails sample was cyanide soluble. Cyanide soluble material in the tails may be a result of free and possibly floatable material with surface tarnishing due to the age of the test sample.

It should be noted that as a result of the limited test sample this test work did not:

- Optimise gravity recovery and a feed size of 106µm was used to feed the gravity circuit;
- Optimise the flotation treatment scheme; and
- Include the effect of pressure oxidation on flotation concentrate prior to carbon-in-leach (CIL) and therefore the cyanide soluble recovery of the flotation concentrate does not reflect the expected CIL recovery if the concentrate was treated at Macraes Gold Mine (MGM), which has a pressure oxidation (POX) / CIL circuit.

13.4.2 Test Work on Prohibition Waste Dump Samples 2011

In 2010 a hand sampling program obtained a 12kg sample of quartz from the Prohibition waste dump area from material previously discarded as waste. The material was collected from the south west face, and from its location on the outer face, it is likely to represent material mined from the lower levels of the Prohibition shaft. The sample averaged 13.8g/t Au and 0.1% Sulphur.

A scoping test was carried out in the OceanaGold Macraes metallurgical laboratory with 5kg ground to a nominal size of 150µm and passed once through an L40 Falcon concentrator to produce a gravity concentrate. The gravity tails were then treated in a flotation cell with copper sulphate and SIBX collector to produce a flotation concentrate.

Recovery of gold to the combined concentrates was 98.6% with a gravity recovery in excess of 80% which is in line with the testing of the WA11 core undertaken by Ammtec previously.

A follow up hand sampling program collected quartz from both the Prohibition waste dump and also from the areas adjacent to the Joker bin and Blackwater lower adit. A sample of Greywacke waste which assayed 0.61g/t Au and 0.06% Sulphur was also collected from the waste dump for blending purposes.

A similar program was undertaken utilising the L40 Falcon unit in the laboratory to look at the effect of mass recovery on gold recovery with a modern, high G-Force, gravity concentrator. Feeds were prepared as a 50:50 mix of quartz and waste to simulate the expected stoped material from a hand-held operation with 100% dilution. This sample blend was similar to reported historical levels fed to the batteries. In general recoveries greater than 80% could be obtained at a 1% mass recovery (generally the upper end of a batch centrifugal concentrator) with recoveries approaching 91-92% at higher mass recovery above 3%.

Flotation of the gravity tails from this stage recovered approximately 65% of the remaining gold to yield a combined recovery over 96%.

Individual bottle roll leach tests were then carried out on individual gravity and flotation concentrates. Leaching was undertaken on unground concentrate samples using sodium cyanide and Leachwell catalyst and 3 successive leaches were carried out due to the high expected gold loadings. Leach recovery of gravity concentrate was 99.8% and on flotation concentrate 98.3% with overall recovery combined with the gravity/flotation stage better than 95%.

Based on the leach characteristics observed on this scoping test there does not appear to be a large refractory component of significance in the concentrates under more intensive leach conditions of the test compared to the historical vat leaching employed up to 1951. It is envisaged that a high intensity leaching unit such as an Acacia Reactor or Gekko ILR could be used to leach greater than 95% of the gold present into a solution for direct electrowinning.

Laboratory observations on the filtering of flotation tailings at a P_{80} of 150µm as a 50:50 blend indicate rapid filtration in the laboratory pressure filters and no signs of holdup from fine greywacke material as observed from clays at Reefton. Assuming the production ore is similarly free of pug-type material and given the mass flows required, the potential to filter the flotation tailings and then co-dispose of them in a dry-stacking arrangement with mine waste would appear to be a viable option. The operating and capital costs of this option are expected to be significantly better than a traditional wet storage tailings dam, and the arrangement affords the option to store underground where space is available.

13.4.3 Gekko Python Amenability Test Work 2011

Bulk samples of quartz and waste from the hand sampling program in 2010 were supplied to Gekko Systems to undertake a flowsheet evaluation program. The aim of this test work was to look at abrasion, crusher work index, grind sensitivity and gravity/flotation performance to allow consideration of a Python design.

The standard Python circuit consists of a two stage crushing circuit utilising a vertical shaft impactor (VSI) crusher to produce a fine product size followed by gravity separation via Inline Pressure Jig, and if required, coarse sulphide flotation.

The Quartz and Greywacke samples were evaluated with a VSI Amenability test to determine expected size distribution from a closed circuit and to determine appropriate work indices for design. Quartz and greywacke were then combined in a 50:50 ratio to provide feed to undertake gravity and flotation testing.

Progressive Grind Tabling (PGT) tests were conducted on the combined material and indicated that 80.8% of the gold could be recovered into 1.35% of the sample mass at a P_{100} of 600µm and P_{80} of 425µm. Further test work utilised a regrind down to a P_{80} of 104µm before the flotation stage and achieved an overall recovery to concentrates of 97.9%.

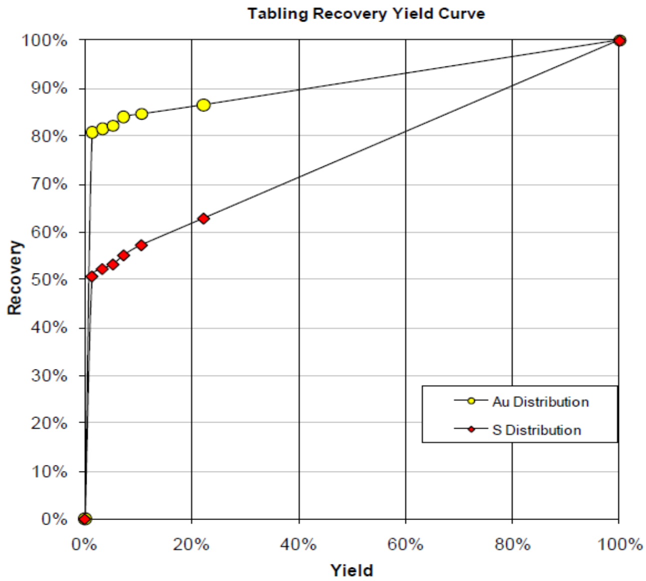

Figure 13-10: Tabling Recovery Yield Curve

Flotation tests were conducted on the PGT tails sample at the produced size and after a regrind to a P_{100} of 212µm. At the coarser size distribution approximately 60% of the gold in the gravity tail could be recovered, with the majority of the losses in the coarser fractions due to poor liberation. At a P_{80} of 106µm approximately 90% of the gold could be recovered to a flotation concentrate.

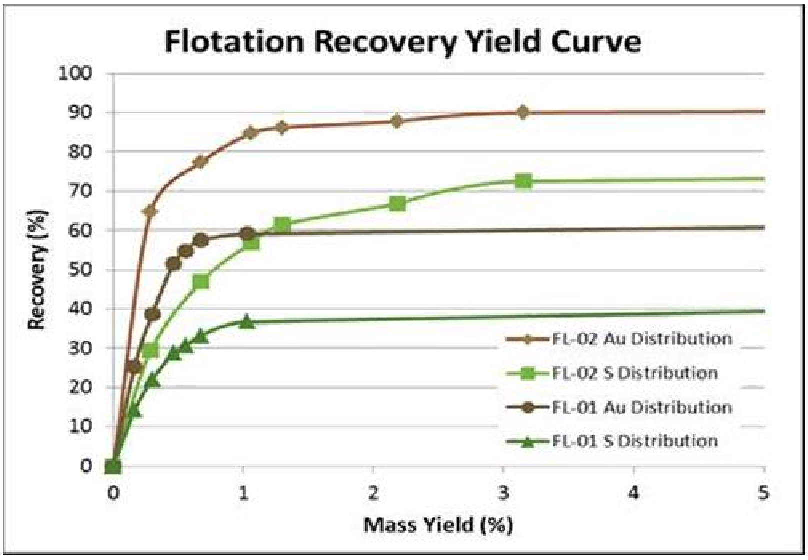

Figure 13-11: Flotation Recovery Yield Curve

As an alternative to flotation, a continuous Falcon Test was performed to simulate the use of a high-g centrifugal concentrator on the PGT tail to increase gold recovery. Approximately 21.6% of the gold could be recovered into 5% of the mass with poor recovery above 212µm again interpreted as due to poor liberation. Overall the use of a finer grind of 106µm and sulphide flotation was seen to yield a significantly better primary recovery of gold to a concentrate.

Table 13-4 shows a summary of the flowsheet options and achieved recovery in the test program. Results clearly show that a combination of gravity recovery in the comminution circuit followed by sulphide flotation will achieve a combined recovery in excess of 97%.

Table 13-4: 2010 Bulk Sample Recovery Results

Sample	Final Grind P_{80}	Head g/t	Au Recovery	Upgrade Ratio	Final Tail
PGT	430	5.66	80.8	338	1.09
PGT & Flotation FL01	430	6.15	89.5	232	0.66
PGT & Flotation FL02	106	5.79	97.9	127	0.13
PGT & Falcon	430	7.33	70.5	75.4	2.32

13.4.4 Optical Sorting Program 2011

Following a review of geological core, data and photographs of historical workings, the option of optical sorting was examined. This would allow mechanised mining to be utilised reducing mining costs without substantially dropping mill productivity from the effects of increased dilution. The general occurrence of gold within a single quartz structure along with assay data from the waste adjacent to the reef showing little mineralisation led to discussions with a number of equipment suppliers.

A secondary sampling campaign at the Blackwater site was undertaken in August 2010 to obtain approximately 4t of waste material and 500kg of quartz. Sample was sourced from along the old tramway route to the Snowy River battery from the No. 2 Level Adit. The materials were hand screened and washed into fractions of 9.5-19mm, 19-37.5mm, 37.5-53mm and 53-75mm. The quartz was similarly screened, and where necessary, crushed to produce a similar size range of material.

Bulk samples of around 400kg per size fraction were prepared containing approximately 20-25% quartz. This was expected to be a similar ratio to small scale mechanised mining of the ore-body. The material was shipped to Commodas Ultrasort in Sydney for testing in the optical sorting machine.

Table 13-5 shows the results from treating the test fractions through a commercial machine at normal production throughput rates.

Excellent performance was achieved above 19mm on the sorter with over 98% recovery of gold to the concentrate fraction with a rejection of more than 75% of the mass. The performance of the 9.5-19mm material was impacted by the presence of -9mm quartz incorrectly added to the sample before shipping to Sydney and production performance would be expected to be higher.

Various circuit configurations were simulated from the test results to look at the impact of optical sorting on the economics of overall mining and milling schedules. A simplified single sorter circuit (Figure 13-12) would provide the ability to upgrade mechanically mined, stoped ore to grades similar to historical hand held methods and improve the overall costs by a significant margin.

Table 13-5: Optical Sorting Results

Fraction		53-75mm	37.5-53mm	19-37.5mm		Total	9.5-19mm
	waste	144.4	200.6	169.3	240	394.2	188.7
	quartz	2	1.5	1.71	0.24	1.95	8.3
reject	*total*	*146.4*	*202.1*	*171*	*240.2*	*396.2*	*197*
	waste	21.4	3	3.0	1.48	9.5	12.6
	quartz	54	48.1	71.6	28.12	109.7	41.8
conc		*75.4*	*51.1*	*74.6*	*29.6*	*119.2*	*54.4*
total	quartz	56	49.6	73.3	28.4	111.7	50.1
total	waste	310.2	203.6	172.3	241.4	403.7	201.3
total	*mass*	*366.2*	*253.2*	*245.6*	*269.8*	*515.4*	*251.4*
mass split		**20.59**	**20.18**	**30.37**	**10.97**	**23.13**	**21.64**
feed au mg		1683.1	1490.0	2201.5	853.2	3354.6	1503.6
conc au mg		1620.2	1443.0	2148.5	843.6	3292.2	1253.5
Feed grade	g/t	4.60	5.88	8.96	3.16	6.51	4.99
Conc Grade	g/t	21.49	28.24	28.80	28.50	27.62	23.04
reject grade	g/t	0.42	0.23	0.31	0.04	0.16	1.27
Waste Rec	%	6.9	1.5	1.7	0.6	2.3	6.3
Quartz Rec	%	96.43	96.98	97.67	99.15	98.25	83.47
Gold Rec	%	96.26	96.85	97.59	98.87	98.14	83.37

	Quartz Distribution	tpa Quartz	Waste Distribution	tpa Waste	Quartz Recovery	Sorted Quartz to ore	Waste Recovery	Sorted W to ore
-10mm%	35%	17500	15%	18000	100%	17500	100%	18000
10-20mm	25%	12500	15%	18000	100%	12500	100%	18000
20-40mm	25%	12500	25%	30000	98%	12250	2%	600
40-60mm	10%	5000	35%	42000	98%	4900	2%	840
>60mm	5%	2500	10%	12000	97%	2425	5%	600
Total		***50000***		***120000***		***49575***		***38040***
g/t Au		30		0.1				

Case:	2m wide stope & 20mm screen		
Stoped grade	8.89411765	g/t	
Stoped tonnage	170000	tpa	
stoped gold	48612	oz	
Sotred quartz	49575	tpa	
Sotred waste	38040	tpa	
Sorted Ore	87615	tpa	
Sorted Grade	17.0	g/t	
Sotrted Rec	99%		
Mass fraction recovered	52%		
tonnes rejected	82385	T	
Rejected grade	0.25	g/t	
Mill Recovery	95%		
Oz lost	640		
	1500	$/oz	
	$ 959,636	%pa lost	
	60	$/t haulage, transport & processing cost	
	$ 4,943,100	pa savings on treatment	
	$ 3,983,464	pa cost benefit from haulage/processing	
Process plant throughput	24.3	tph	without sorter
	12.5	tph	with sorter

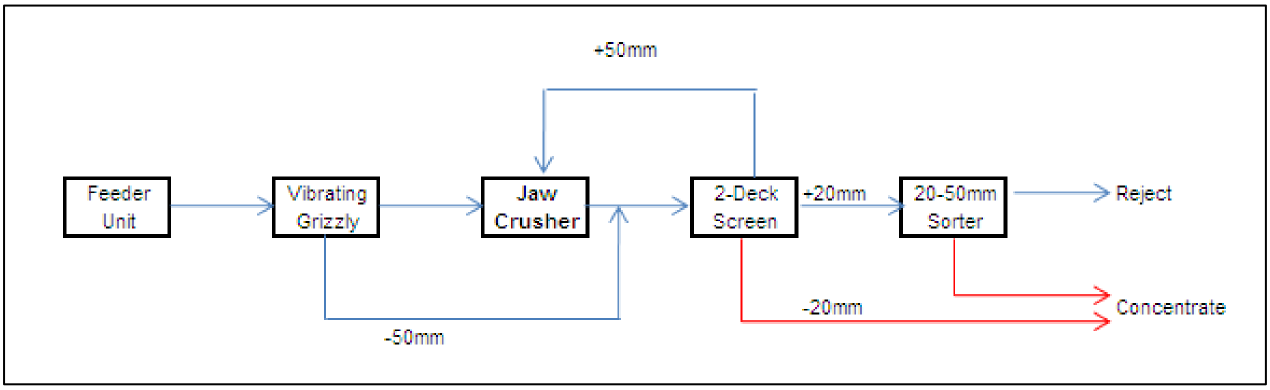

Figure 13-12: Simplified single sorter flowsheet utilising a jaw crusher and dual deck screen

A significant consideration on use of an ore sorter is the potential location on surface or underground to minimise potential haulage costs to the surface. Some time was spent considering the use of the Gekko Systems "Python" modular processing equipment with the concept of locating the primary crusher and sorter circuit underground. A similar sorter circuit was manufactured by Gekko for a project in South Africa.

The key risk on the flowsheet developed is the assumption that the gold occurs almost exclusively in the quartz reef and that the host Greywacke is essentially barren (0.1g/t is assumed in the balances for the optical sorter). The data available from the deep drilling holes shows there is no significant grade in the waste adjacent to the reef.

13.4.5 Gekko Flowsheet Validation Test Work 2013

A follow up program was undertaken by Gekko Systems to validate the 2011 findings and to provide adequate confirmatory data for design criteria for process design. A test program was initiated to perform:

- conventional 3-stage Gravity Recovery Gold (GRG) tests;
- a three-stage Continuous Gravity Recovery test;
- flotation testing at grind sizes down to 106µm to identify the optimum flotation feed size;
- Intensive leach tests on bulk flotation/gravity concentrates; and
- Electrowinning tests on direct leach liquor.

The results of the test program are intended to offer validation for the process design criteria and selection for the proposed flowsheet for ore treatment. Figure 13-13 from the continuous gravity recovery test shows the distribution of both gold and sulphur as a function of mass recovery. The proposed flowsheet currently envisages a moderate mass pull of 1.5% to gravity concentrate to target gold recovery in excess of 80%.

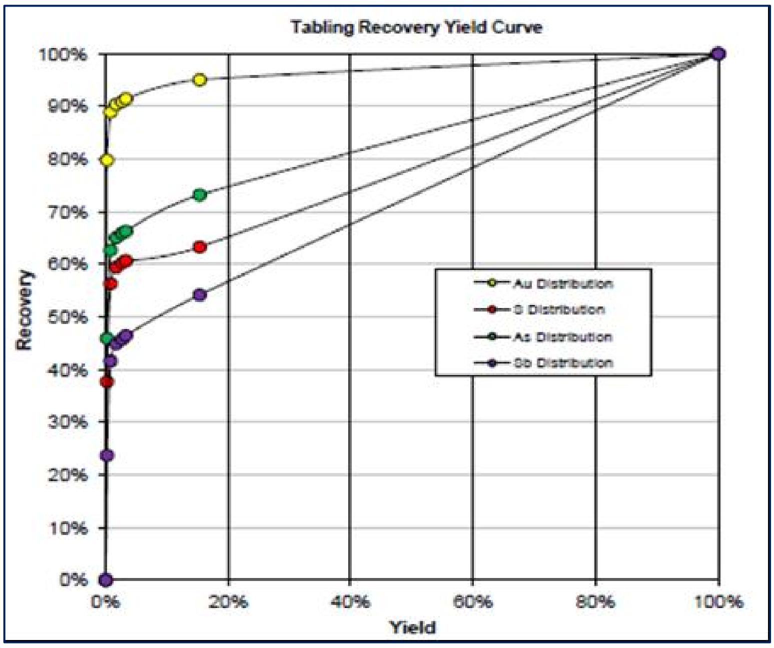

Figure 13-13: Plot of Yield against recovery for the CGR Test

Progressive flotation tests at grind sizes from 600µm down to 106µm on CGR tails showed a trend of increasing recovery with finer grind which is in line with earlier testwork. As shown in Figure 13-14 the benefit of grinding down past 150µm is marginal and therefore this grind size was selected for generating concentrate for intensive leach/electrowinning tests.

It is anticipated that combined gravity/flotation processing of the ore will generate recovery to primary concentrate exceeding 97%, and with allowances for leach/electrowinning losses, an overall circuit recovery above the 95% target in the economic models will be achieved.

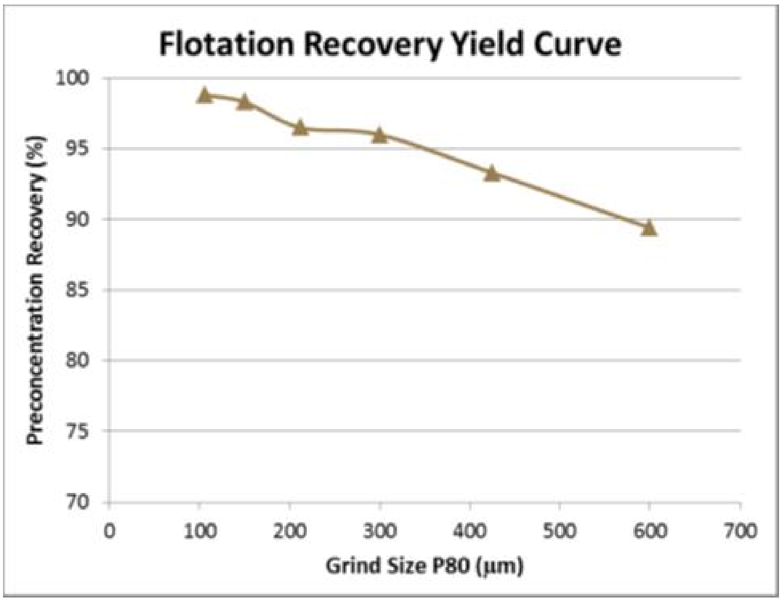

Figure 13-14: Plot of combined Gravity/flotation recovery against grind size

Intensive cyanide leach tests were carried out on the final concentrate product (3 stage CGR & FL05 concentrate) at different grind sizes to determine the effect on recovery. The intensive cyanide leaching results are summarised in Table 13-6.

Table 13-6: Cyanide Leaching Results

Sample	Calc Feed	Recovery @ 6hrs (%)	Recovery @ 12hrs (%)	Recovery @ 24hrs (%)	Residue (g/t)	NaCN (kg/t)
LOCE01 As Is	105	76.3	83.3	97.6	2.54	7.5
LOCE02 P_{100} 425µm	109	89.7	94.1	98.6	1.58	1.0
LOCE03 P_{100} 212µm	99.8	93.6	98.5	98.4	1.58	0.6
LOCE04 P_{100} 106µm	108	97.5	99.9	98.8	1.1	1.0
LOCE05 P_{100} 106µm	103	99.9	97.4	98.0	2.06	0.8
LOCE06 P_{100} 106µm	101	99.9	98.7	98.0	2.03	0.6

Leaching of the as-is concentrate gave the lowest recovery at 97.6%; this was after 48 hours of leaching. The highest recovery of 99.9% was achieved with a grind size of 106µm. Grind size impacted not only on recovery but also leaching kinetics as shown in Figure 13-15.

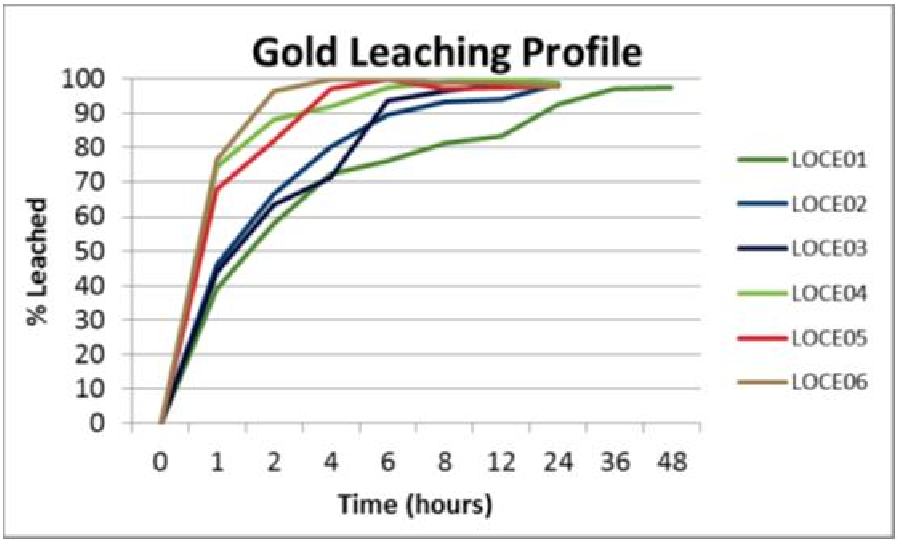

Figure 13-15: Gold Leaching Profile

As the presence of arsenic and antimony can negatively impact on leaching kinetics and recoveries, test LOCE 05 was conducted by repeating test LOCE 04, but using Proleach as an oxidant (instead of oxygen) as this can help negate the effect of arsenic in solution. Test LOCE 06 was carried out without any addition of sodium hydroxide or lime to control pH to negate the effect of antimony dissolution. Both of these tests improved the gold dissolution rate to 99.9% of gold in solution after 6 hours of leaching (Figure 13-15).

Pregnant leach solutions generated from the intensive cyanide leaching tests (LOCE 03 and LOCE 04) were used as feed for electrowinning. The results from the two electrowinning tests are summarised in Table 13-7.

Processing concentrate with a regrind stage would allow leaching to be completed in approximately 6 hours in a batch intensive leach reactor. Long residence times would be required without a regrind stage and would potentially drop plant recovery by 2%.

At room temperature (25˚C) electrowinning over a 6 hour period achieved a recovery of 97.4% of gold plated onto the steel wool. This improved to 98.3% with heating of the electrowinning solution to 60˚C. The barren electrowinning solution has a gold concentration of 1.3mg/L which decreased to 0.85mg/L with electrowinning at 60˚C.

The final solution composition was analysed by ICP and compared to the start solution to determine minor element behaviour. While there was some deposition of arsenic during electrowinning at room temperature, there was no deposition at 60˚C. Antimony was deposited on the cathode during both tests.

Table 13-7: Electrowinning Test Results

Test ID	Current (mA)	Au (mg/L)		Ag (mg/L)		As (mg/L)		Sb (mg/L)	
		Initial	End	Initial	End	Initial	End	Initial	End
EW (01)	50	49.6	1.3	2.0	<0.4	20	11	20	15
%Deposited		97.5%		>80%		45%		25%	
EW (02)	50	49.9	0.85	1.6	<0.4	37	56	20	13
%Deposited		98.3%		>75%		0%		35%	

The overall gold recovery achievable with intensive cyanidation on the final concentrate followed by electrowinning on the pregnant leach solution was calculated from the test results and is shown in Table 13-8. With a calculated gold head grade of 5.9g/t, overall recovery was calculated to be 96.5% which gives a final residue tail of 0.11g/t.

Table 13-8: Summary Leach/Electrowinning Performance

Test ID	Calc Au Head (g/t)	Mass Yields (%)	Pre Concentration Recovery (%)	Leach Recovery (%)	EQ Recovery (%)	Overall Recovery (%)
GR&FL05/LOCE04&EW01	5.9	6.12	98.3	99.9	97.4	95.6
GR&FL05/LOCE04&EW02	5.9	6.12	98.3	99.9	98.3	96.5

13.5 Risks

The key risk with the flowsheet developed is the assumption that the gold occurs almost exclusively in the quartz reef and that the host Greywacke is essentially barren (0.1g/t is assumed in the balances for the optical sorter). The recent data available from the deep drilling holes that intercept the reef suggests that this is the case.

Sliming of the Greywacke in the grinding process may have adverse impacts on the flotation and tailings filtration processes. Laboratory tests to date have not seen this occur however confirmation of the condition of the waste adjacent to the reef will improve confidence in this.

No refractory nature has been observed in the sulphides tested to date although a roaster was used at the Prohibition mill to treat flotation concentrates prior to vat leaching. It is possible that low levels of Stibnite have an effect of slowing the leaching process down and may have been present in some areas causing the implementation of the roaster (very important at Globe). The coarser grind of the older battery and reported problems with classification may also have led to the previous circuit configuration. If the refractory nature of the sulphide associated gold changes, the direct leach plan may not yield the high recoveries expected and campaigning the leach tails through the autoclave at Macraes Gold Mine would need to be considered.

Due to the limited core availability from the reef below the original workings recent test work has been performed on quartz samples obtained from the waste dumps adjacent to the Prohibition shaft and Joker Bin tramway. There is a potential that the performance of the reef may differ from the majority of recent samples tested, the recent programs did however produce results comparable to the core from hole WA11 tested in 2003.

13.6 Further Testwork

In order to reduce risk in design and scale up, additional test work is proposed including:

- Additional intensive leach tests on gravity/flotation concentrate to confirm parameters of a resin upgrade circuit prior to electrowinning as part of full processing at the Blackwater site on existing surface collected samples;
- Full flowsheet test on the remaining quartz core from the deep drill intercepts (approximately 10kg of quartz is available) on the final flowsheet to confirm recovery assumptions are achievable with material from the reef area to be mined;
- Additional filtration test work on flotation tailings samples to confirm parameters for filter equipment sizing; and
- Confirmatory flow sheet testwork on core retrieved from the resource definition drilling.

It is planned to complete the first of these three test works programmes during 2015 to allow finalisation of design criteria and detailed engineering of the process plant to be completed.

Upon commencement of the resource definition drilling, reef core will be available to complete additional variability testing of plant feed in the first two years of production to assist in production budget planning.

14 MINERAL RESOURCE ESTIMATES

The Blackwater resource estimate presented in this section has been based on a combination of deep diamond drilling and historical mine sample data.

A Mineral Resource estimate for the Blackwater Mine was released in May 2013 ("Technical Report for the REEFTON PROJECT, Located in the province of Westland, NEW ZEALAND, by OceanaGold Corporation with an effective date 24th May 2013), reported in accordance with the JORC 2004 Code. This was subsequently updated by a Resource statement as at 31st December 2013, released 26th March 2014, as part of the 2013 end-of-year Resource and Reserve statement. For the purposes of this Technical Report, the Resource statement as at 31st December 2013, inclusive of Table 1 disclosures in accordance with clause 5 of the JORC (2012) Code, is appended to and forms the basis for this PEA.

14.1 3D Block Model of Historically Mined Reef

In 2009, OceanaGold constructed a 5mN x 5mRL x 1mE ordinary kriged block model of reef grade, true thickness and contained gold (as width x grade). The model, shown in Figure 14-1, was based on channel samples obtained from an archived long section showing the underground workings, and horizontal channel samples of the reef as widths (in inches) and assay results (in pennyweight).

The channel samples derived from archive in many cases were not individual sample grades and thicknesses but rather values averaged from a number of samples over tens of metres (including development samples, rises and winzes). In most cases individual sample data were not available. These data therefore will exhibit less variability than would be seen with the raw values. Furthermore, the values have been top cut (with variable top cuts, at times estimated to be approximately 37 g/t Au (the cut-off strategies applied over the life of the Blackwater Mine are not well documented). Graham (1947), which discusses the sampling of the reef, confirms that the channel samples (at least for the annual ore reserve calculations) represent the grade and width of the quartz reef, and that marginal dilution was recorded separately.

The northings and RLs of the long sectional points were measured from an A0 copy. The eastings were then estimated by snapping these coordinates onto a three dimensional reconstruction of the underground workings. Once this was done, the eastings, northings, RLs, assays and widths were entered into the drill hole database. The widths were entered as horizontal sample lengths. The widths and assays were converted to metres and g/t Au respectively. The data were then kriged and imported into MINESIGHT 3D software.

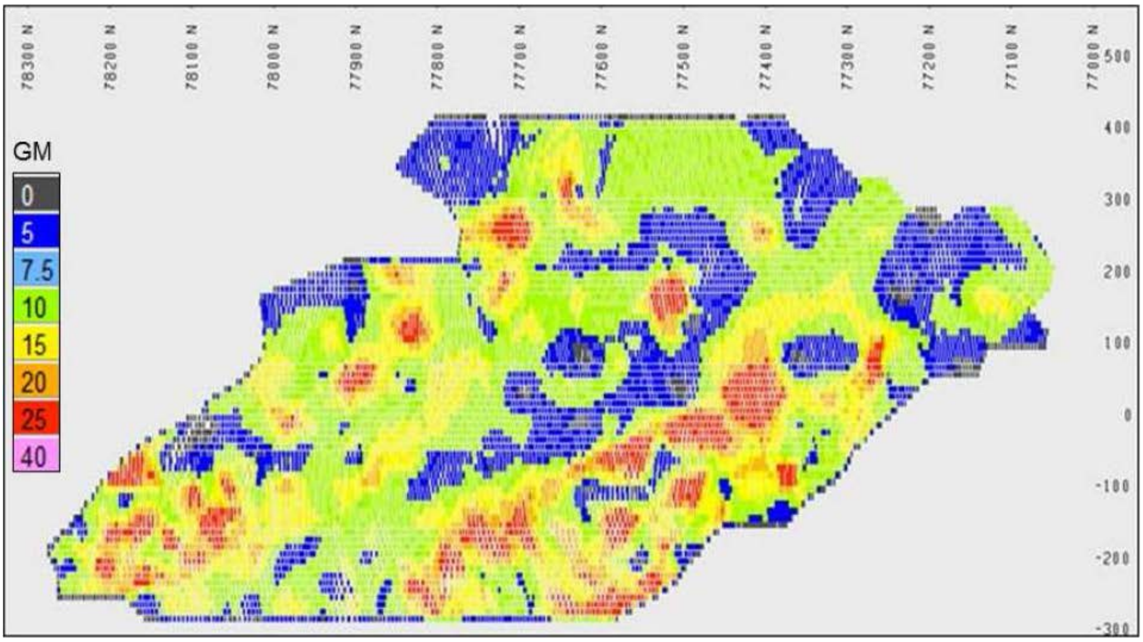

Figure 14-1: Blackwater Mine Long Section showing Block Modelled Gram-metres (Au g/t x width)

The model provides a valuable three dimensional summary of the historically mined resource. It also provides one approach to estimate the average grade of the total reef (i.e. including both the payable and non-payable reef). Note that the model does estimate resource a little beyond what is believed to be the true extent of mining, except for levels 4 and above where data is missing. Nonetheless, for this volume of modelled reef, the average estimated grade is 21 g/t Au.

14.2 Alternative Approaches to Estimating the Reef Grade for the Historical Mine

A number of other data-based estimates of the reef grade were completed after the release of the May 2013 resource estimate, as part of the Blackwater mining study. Due to poor record keeping during historical mining activities, no single source of data can be considered definitive. For this reason a number of estimates have been made, each placing varying reliance on the various data types. For example, historical face sample data have no QAQC and as such face sample grades should be treated with caution. Furthermore some grades are top cut, some not. It is not always clear whether, by how much, or why the grades have been top cut. Historical metallurgical recoveries and mining dilution are not known to a high level of precision.

14.2.1 15-16 Level Development Book

A copy of part of the development book covering face sampling from 15-16 levels and a number of winzes and rises up to 13 levels was obtained from the Hocken library in the past and electronically scanned. The data shows some inconsistency in the application of top-cutting. Many of the development face samples comprised more than one sample, for example a top, middle and bottom sample is collected at a face and assayed individually. These are then averaged to give the average face grade and width. In some cases the top cut is applied to the individual samples before averaging whereas in other cases the samples are averaged uncut and then the top cut applied to the average.

The data has been analysed and the following conclusions made:

- Most of the face samples have had a top cut of 23.5 dwt[4] applied (≈36.5 g/t Au).

[4] Pennyweight, where 1 pennyweight (dwt) = 1.555 grams (g)

- Application of the top cut is not consistent. In some cases it is applied to the individual samples; in other cases it is applied to the average of the 1-3 individual samples that make up each face sample (top, middle and bottom of a face).

A summary of the data analysis is shown in Table 14-1.

Table 14-1: 15-16 Level Development Book Data

Level	Average Width M	Arithmetic Average Cut g/t	Arithmetic Average Uncut g/t	Weighted Average Uncut g/t
15 North	0.76	21.85	26.09	26.53
15 South	0.80	21.72	30.51	32.72
16 North	0.52	21.77	27.09	31.15
16 South	0.76	21.02	26.41	27.09
Combined Levels	0.74	21.49	27.58	29.27
Rises, Winzes	0.78	23.35	30.91	38.44

The differences between cut and uncut average grades range from 5-11 g/t Au.

14.2.2 Payability Factor

Not all the strike length of the reef comprised payable quartz. The figure commonly quoted in the previous studies is a payability factor of 80%, where payability relates to the proportion of strike length (not tonnage) that was mined.

This figure was checked as part of this study by using the composite long-section as follows:

- For each level the total length of mined out area was measured. The mined out limits are defined by the northern limit of the northernmost stope and similarly the southernmost limit of the southernmost stope at any level;
- The length of each individual stope was then measured and added together for each level; and
- The payability was determined by dividing the total stoped length by the total mined out length. Results by level are tabulated in Table 14-2.

The average was 79%, near enough to the 80% payability factor that has been generally accepted. Note that payability refers to the strike length, and is not a tonnage based estimate.

Given that the payability was based on reef grade and width criteria, the payability in terms of tonnage is expected to be considerably higher. No records of mined reef widths versus widths of reef segments not mined could be located.

Assuming that non-payable reef typically averages half the overall reef width, tonnage-based payability is estimated to be 90%.

Table 14-2: Payability by Level

Level	Mined Length (ft)	Stoped length (ft)	Stoped %
1a	450	331	74
1	2,083	1,882	90
2	2,178	1,870	86
3	1,941	1,574	81
Low level	982	852	87
4	2,959	1,953	66
5	3,076	1,964	64
6	3,550	2,343	66
7	3,609	2,746	76
8	3,550	2,746	77
9	3,456	2,568	74
10	3,219	2,355	73
11	3,408	2,698	79
12	3,360	2,817	84
13	2,970	2,734	92
14	2,934	2,710	92
15	2,686	2,473	92
16	2,308	1,692	73*
Total	**48,719**	**38,308**	**79**

**16 Level was still being mined at the time the long-section was compiled and the actually payability is probably higher than shown.*

14.2.3 Alternative Estimate Based On Face Sample Data

The face sample data in Table 6-8 provides arithmetic data for the reef, but cannot be associated with 3D coordinates. The data covers both payable and non-payable reef. The combined face sample grade (i.e. excluding stope samples) for levels 11-16 and 4-13 is 21.9 g/t Au, if weighted by number of levels. These grades have been top cut, and uncut grades are not available. This grade estimate is within 4% of the block model grade estimate.

14.2.4 Alternative Estimate Independent of Face or Stope Sample Grades

Given the lack of QAQC for face / stope samples and the poor understanding of the top cut thresholds through the mine's history, it makes sense to generate a reef grade estimate independent of this data. The approach taken was to back-calculate the mined reef grade using the in-situ reef tonnage, mill-estimated tonnes, bullion and assumed metallurgical recoveries. A tonnage-based reef payability of 90% has been assumed (refer to Item 14.2.2). The estimate has been made using long-sectional area and average reef thickness (0.68m) to estimate in-situ tonnage.

The mined out area of Blackwater was digitised in Minesight software and a sectional area obtained. If the set of assumptions in Table 14-3 are used, the reef gold grade can be estimated. This estimate assumes that the non-payable reef has approximately half the width of the payable reef – it is known that the grade control criteria employed during mining were a combination of cut-off grade and width thresholds.

Table 14-3: Assumptions

Item	Value	Source
Tonnes milled	1,582,400t	Historical records
Recovered gold	740,400 oz.'s	Historical records
Gold recovery	90%	Average of range
Payability by Tonnage	90%	Assume non-payable reef 0.34m wide
Width	0.68m	Pearson (1942) average
SG	2.6	SG of quartz

Table 14-4: Estimated Gold Grade by Sectional Area

Area	Area	Payable Area (m²)	% of Total Area	Rec. Ozs	Mined Ozs	Payable Tonnes	Quartz Grade (g/t)
Above 2L	62,370	56,133	9	68,890	76,544	99,243	
2-4L (outside BM)	32,232	29,009	5	35,602	39,557	51,288	
BM area (mined)	575,720	518,148	86	635,908	706,564	916,086	
Total	**670,322**	**603,290**	**100**	**740,400**	**822,667**	**1,066,617**	**24.0**

Based on historically recorded milled tonnes of 1,582,400, the mine dilution is calculated as:

$$\frac{1{,}582{,}400 - 1{,}066{,}617}{1{,}066{,}617} = 48\%$$

This calculation is sensitive to the assumed width for the payable reef. For example, using a payable reef width of 0.64m rather than 0.68m would increase the estimated mined grade to 25.5 g/t Au.

As the non-payable reef is expected to have lower grade than the payable reef, a discount should be applied to estimate the total (payable and non-payable) reef grade. The difference between average grade determined for payable-only reef from stope samples and that from face sample data only (Table 6-8) is 1 g/t Au. This, although not definitive, is consistent with the grade of the non-payable reef being approximately half that of the grade of the payable reef. A 1 g/t Au grade discount is proposed to relate the back-estimated grade for payable-only reef to the likely combined payable / non payable reef grade. As such, an average grade of 23g/t Au is deemed to be an appropriate estimate for the grade of the reef within the Blackwater Mine. Sensitivities assuming 6 g/t Au and 18 g/t Au respectively for unmined reef grade, yield in-situ resource grades between 22.2 g/t Au and 23.4 g/t Au, so the impact is not large.

14.2.5 Reconciliation of Alternative Estimates

The sections above have presented a number of approaches to estimating the reef grade within the historical mining area. The first two estimate the combined payable and non-payable reef:

- 21 g/t Au from block model (top cut, uncut grades not available); and
- 21.9 g/t Au from face sample data (top cut, uncut grades not available).

There is no comprehensive set of uncut sample data, but it can be commented that estimates based on uncut grades would be higher.

The following estimates pertain only to the mined or payable reef:

- Stope samples average 22.9 g/t Au (top cut, uncut grades not available); and

- Long sectional back-estimated grade 23 g/t Au was estimated for the combined payable plus non-payable reef (independent of sample grades).

In summary, the combined payable / non-payable reef grade estimates range between 21 g/t Au (block model) and 23 g/t Au (back-calculated grades discounted by 1 g/t Au). Table 14-5 presents the sensitivity of back-calculated head grade (i.e. payable reef grade) to mining dilution and metallurgical recovery, using the calculation methodology shown in Table 14-4. Lower historical metallurgical recoveries would increase the back-calculated in situ reef grade. More recent investigations into the historical metallurgical performance of Snowy River and Prohibition plants suggests that gold recoveries below 90% were more likely than the 90% assumption used in this study. If this were the case, then it is more likely that the true reef grade would he higher rather than lower than the estimates presented above – refer to Table 14-5. Payability assumptions also affect the back-calculated grades.

Table 14-5: Back-Calculated Head Grade at Various Recovery and Dilution Factors

Recovery	Dilution			
	40%	50%	60%	70%
89%	23.0	24.7	26.3	28.0
90%	22.8	24.4	26.0	27.7
91%	22.5	24.1	25.8	27.4
92%	22.3	23.9	25.5	27.1
93%	22.1	23.6	25.2	26.8
94%	21.8	23.4	24.9	26.5
95%	21.6	23.1	24.7	26.2

For those estimates based upon sampling data, including the block model estimate, reasonable data integrity is assumed. No QAQC data is available so the integrity of the underground sampling is unknown; there is potential for grade bias, which may be in part the reason top cuts were applied (i.e. balancing cuts). Alternatively, top cutting could be masking higher grades. The little data that does allow comparison of cut and uncut grades reveals significant differences. For these reasons both sample based and mill back-calculated grade estimates have been considered.

The Long sectional back-calculated payable reef grade of 24.0 g/t Au is independent of sample grades and for this reason believed to be the most plausible estimate, acknowledging that assumptions are required to be made to enable calculation of this value. As discussed previously, a 1 g/t Au grade discount is proposed to relate this to the likely combined payable / non payable reef grade. As such, an average grade of 23g/t Au is deemed to be an appropriate estimate for the grade of the historically mined reef. (sensitivities assuming 6 g/t Au and 18 g/t Au for unmined reef grade, yield in-situ resource grades between 22.2 g/t Au and 23.4 g/t Au, so the impact is not large).

14.3 Mineral Resource Estimate

Because of the nature of the estimate (based on limited drilling, but supported by decades of historical mining data), the resource “volume” was presented as a simple plane, neither reflecting pinches and swells, nor structural disruptions. It wasn’t practical to attempt to interpret these pinches and swells given the paucity of drilling in the Inferred Resource area below 16 Level. The ordinary kriged block model (Item 14.1) of the historically mined reef however has been provided to give some insight into aspects of reef complexity (particularly pinch and swelling of the reef). Furthermore, conditional simulations are provided in Item 14.3.1 to present anticipated variability in reef grade, width and contained gold.

As with previous resource estimates completed since mining ceased at the Blackwater Mine, the 31 December 2013 resource estimate was based upon a projection of the historically stoped footprint below 16 Level of the Blackwater Mine workings. The projection depth is supported by 4 deep diamond holes (and their daughters) collared from surface in two campaigns in 1996 and 2010 to 2013 respectively. These are summarized in Table 14-6 and Figure 14-2.

The results are consistent with the range of historically mined widths and grades and indicate that the Birthday Reef continues for at least 680m vertically below the last worked Level of the Blackwater Mine. Historically, each vertical metre of the reef produced approximately 1,000 ounces of gold.

Table 14-6: Blackwater Deep Drill Hole Intercepts

Hole ID	From (m)	To (m)	Intercept (m)	True Width (m)	Grade (Au g/t)	Grade Width (g*m)	Comment
WA11	979.6	980.3	0.7	0.5	24.50	12.3	Parent Hole
WA11A	980.3	981.0	0.7	0.5	59.70	29.9	Daughter Hole
WA21A	1,315.9	1,316.8	0.9	0.5	23.30	11.7	Daughter Hole
WA22C	1,632.30	1,633.0	0.70	0.5	15.65	7.8	Parent Hole
WA22D	1,623.90	1,625.03	1.13	1.0	85.2	85.2	Daughter Hole
WA25	1,118.95	1,119.40	0.45	*0.35	31.8	11.1	Parent Hole
WA25	1,134.18	1,134.59	0.41	*0.3	62.4	18.7	Parent Hole
WA25	1,190.77	1,191.36	0.59	0.5	3.91	1.9	Parent Hole (BR)
WA25A	1,136.40	1,137.11	0.71	*0.5	134.00	67.0	Daughter Hole
WA25A	1,195.20	1,195.65	0.45	^0.4	61.90	24.7	Daughter Hole (BR)

** Indicates the upper intercept in each of the holes WA25 & WA25A interpreted as a fault repetition of the Birthday Reef. (BR) indicates the Birthday Reef intercept.*
^ Unorientated drill core. True width calculated using WA25 intercept.

The strike extent of Inferred Resource decreases at depth due to the limited coverage of resource drill holes at depth in the northern area of the reef (refer to Figure 14-2). There currently is no information to suggest that the resource will not develop similar strike lengths at depth to those seen within the historical mine, but the drill hole coverage precluded projecting the reef to the north at depth. This presents an exploration opportunity.

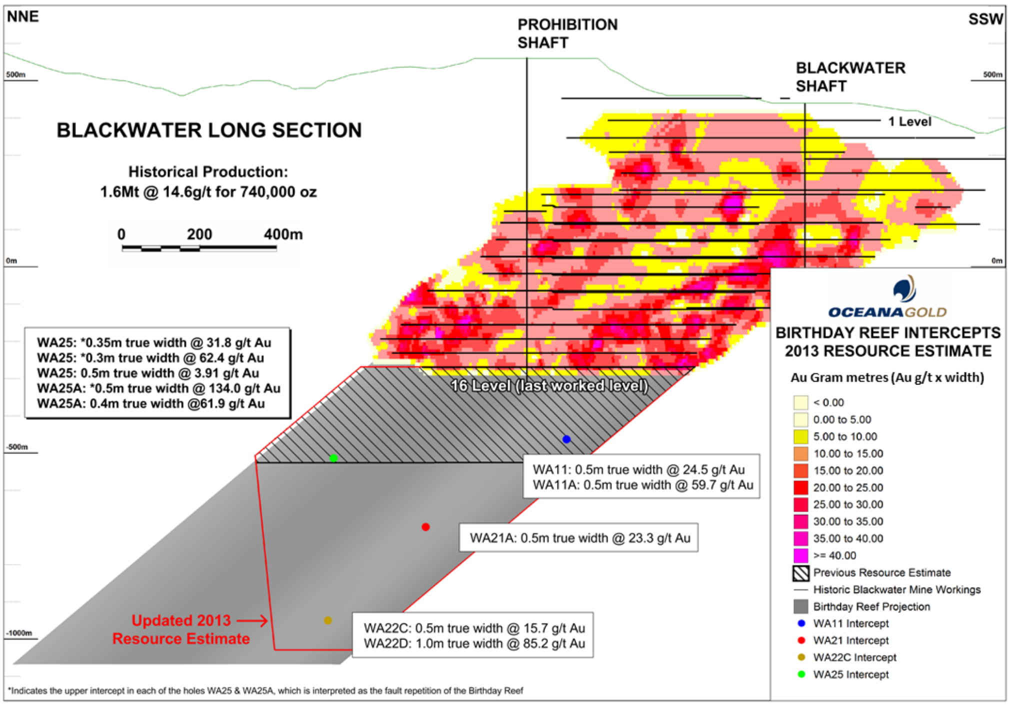

Figure 14-2: Blackwater Mine Long Section

Figure 14-2 shows Au gram-metres from historical workings, drill intercept locations with estimated true widths, gold assay results and the limits of the updated resource estimate.

To estimate the projected volume, a reef plane with a 900m strike length was projected to depth. Within this projected plane, the resource limit was broadly based on a 200m maximum distance to the nearest sample (see thick red line in the long section in Figure 14-2). The resource was extrapolated 100m below the deepest drill hole intersection (WA22) on the south west corner of the resource. The north east corner of the reef was excluded, but the reef was extrapolated approximately 200m down plunge. Approximately 15% of the resource is therefore extrapolated beyond actual sample locations.

The historical average (declustered via ordinary kriging) reef thickness of 0.68m was used to estimate the volume. A bulk density of 2.60 t/m³ was used throughout for the mineralisation, as the lode zone (reef) was documented to be quartz with some carbonate and sulphides, both of which are denser, but cavities are also documented.

Given the paucity of diamond core intercepts, the drill hole data was not directly used to estimate the grade of the unmined portion of the Birthday Reef.

The exploration drilling results (Table 14-6) fall within the range of historically mined thicknesses and grades, but represent a small number of exploration drilling intercepts. As detailed in Item 14.2.5, a comparison of various reef grade estimate methodologies suggests that a grade of 23 g/t Au (independent of underground sampling) is appropriate. No cut-off has been applied, and the estimate is geologically constrained within the reef volume with an average reef thickness of 0.68m. The estimate excludes all remnant mineralisation on or above the 16 Level.

While the projected depth of the resource is based upon four drill holes and their daughter holes, the availability of production records and three dimensional rectified channel samples provides valuable insight into the grade continuity and geometric complexity historically encountered.

The geological evidence of the projected resource is sufficient to imply but not verify geological and grade continuity. On this basis, the Blackwater estimate is classified as an Inferred Mineral Resource and is shown in Table 14-7. The assumptions that the average widths, average grades and average payability from the historical mining blocks are appropriate for the global Inferred Resource, but are unsuitable for detailed mine planning purposes.

Table 14-7: 31st December 2013 Blackwater Mineral Resource (Polygonal Estimate)

Birthday Reef			
Category	**Tonnes (Mt)**	**Grade (Au g/t)**	**Contained Gold (Moz)**
Inferred Resource	0.9	23	0.7

14.3.1 Grade Control Uncertainty and Risk

The kriged model presented in Item 14.1 provides a smoothed impression of the Birthday Reef and does not convey the likely local variability in terms of grade, thickness and contained gold. In order to gain some insight into this variability, a 1mE x 4mN x 4mRL three dimensional conditionally simulated model was constructed, using the same population of face sample data as was used for the kriged model. Note that the sample data used for these simulations in many cases appear to average a number of adjacent samples. This raw data is not available, so the simulations below will understate variability to some degree. The simulations (sequential gaussian) were generated in 2D - flattened to a constant easting. A single selected realisation (for grade and thickness) was then folded back into the undulating plane of the rectified historical underground workings (the grade distribution of the simulation honours that of the input sample grades, but will be lower than the back-calculated grade used for the resource – this grade was based on mill production and mining records).

The gold grade realisation in Figure 14-3 shows some structure in grade, but a high degree of local variability, suggesting that while visual control will define the width extent of the reef, that grade control sample-based cut-offs may not be successful (i.e. result in considerable misallocation). A minimum reef width threshold however might be able to be applied (Figure 14-4).

Drill core logging and assaying demonstrate that gold mineralisation is restricted to the reef and so in cross-sectional terms (i.e. reef margins), the mineralisation would be amenable to visual control.

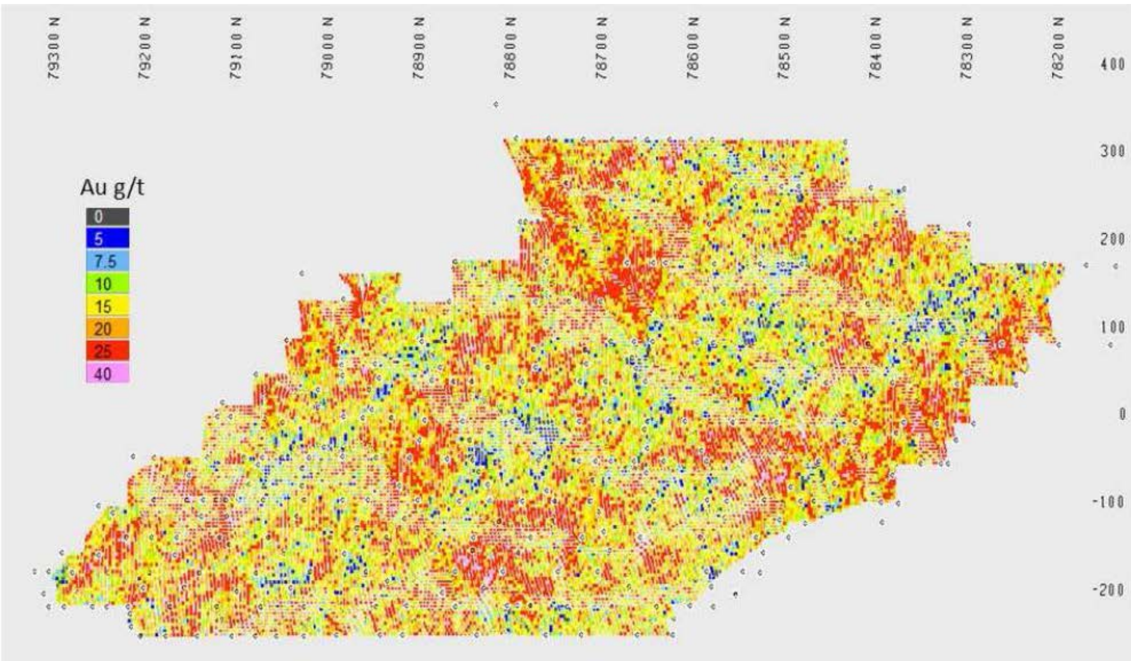

Figure 14-3: A Single Conditional Simulation of Gold Grade

The simulation of reef width distribution (Figure 14-4) shows more structure than the simulated distribution of grade. This suggests that the application of a minimum reef width threshold might be used to for grade control.

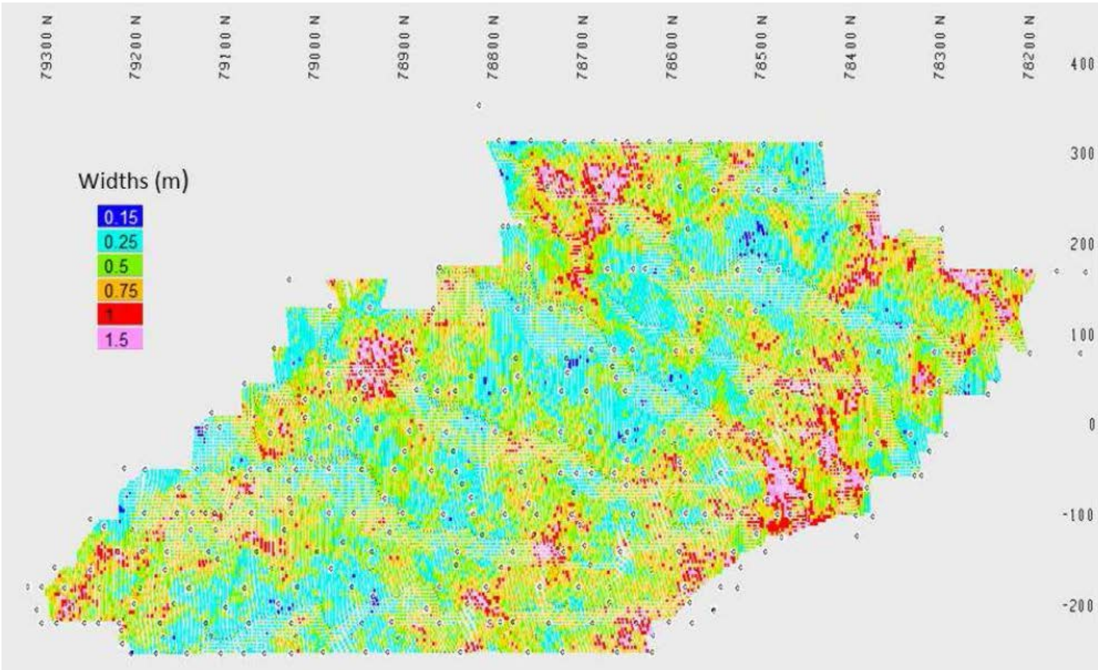

Figure 14-4: A Single Conditional Simulation of Reef Width

The distribution of contained gold (grade x width) was not simulated, but rather calculated directly from the product of simulated grades and widths; reef grades and widths were simulated independently. Contained gold was calculated from the product of these for each node. The use of independently simulated grades and widths for this calculation was felt to be appropriate given the poor correlation between reef grade and thickness. The resultant contained gold distribution is shown in Figure 14-5.

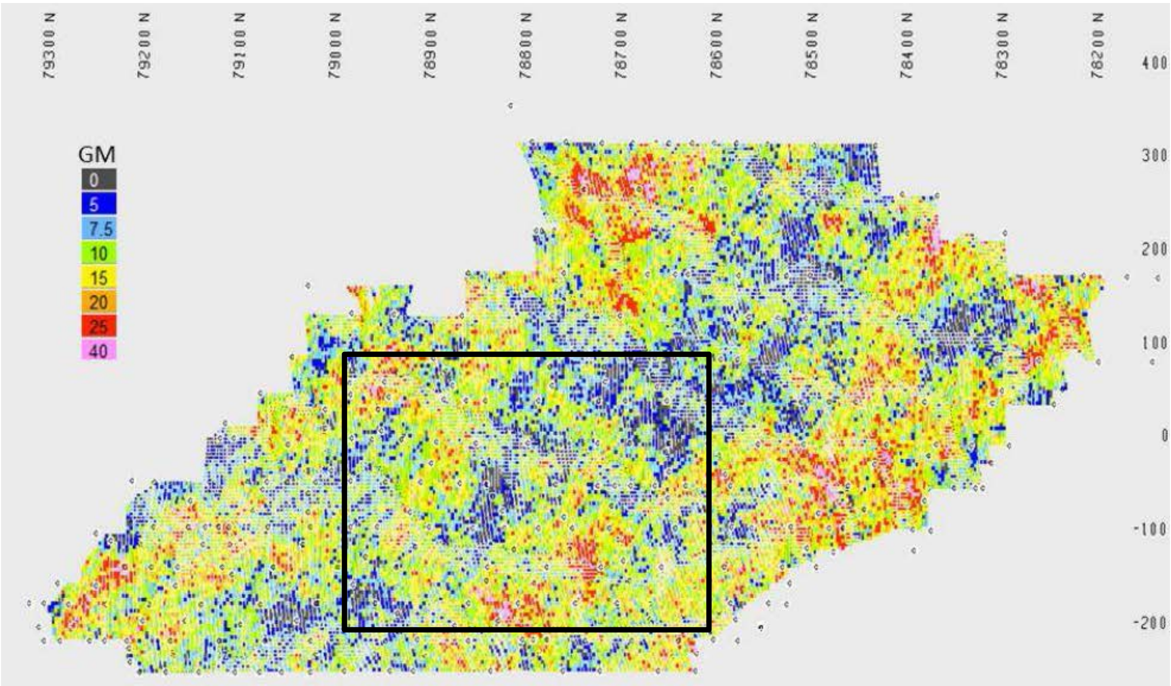

Figure 14-5: Contained Gold derived from Conditional Simulations of Width (m) x Grade (g/t Au)

An exploded view of the outlined area is shown in Figure 14-6. This shows structure in the simulations of contained gold. This predominantly reflects the underlying structure in the realisation of reef width distribution; extensive lengths of simulated reef contain minimal gold relative to the thicker regions of reef. This suggests that a minimum reef thickness threshold may allow selective mining of the reef.

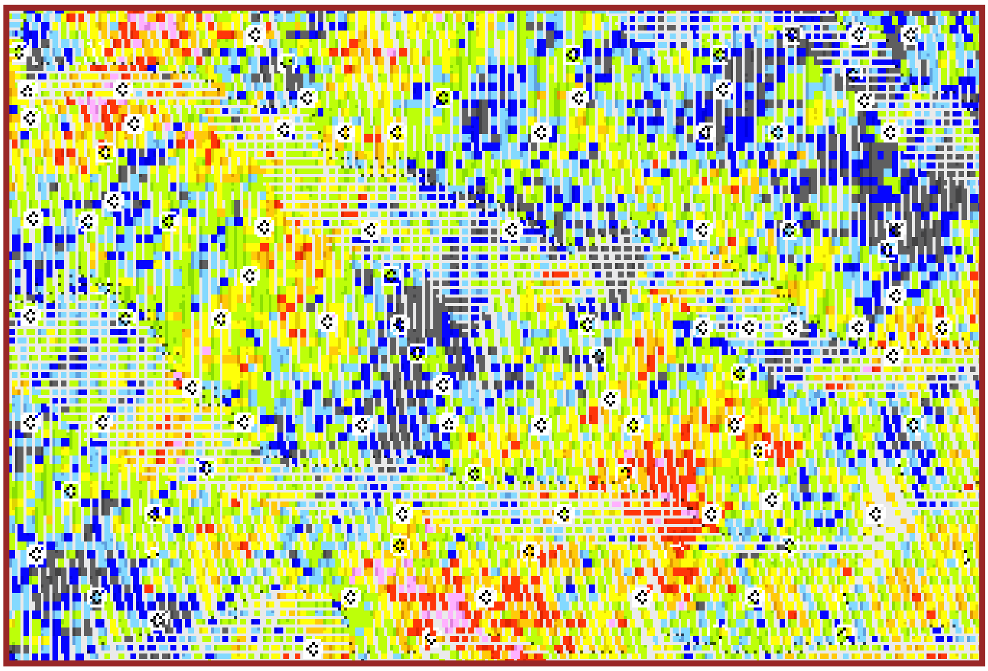

Figure 14-6: Exploded View of Figure 14-5

14.4 Grade Control and Resource Definition Strategies

Routine face mapping and sampling will be undertaken. As discussed in the Item 14.3, grade variability is expected to be such that grade control (in a long-sectional sense), based on cut-off grade may not be effective. A minimum reef thickness criterion however, might be possible.

The current mine design maintains the spiral access decline approximately 100m off the reef, with access to the reef provided by inclined or declined cross cuts. Development levels will be 100m apart vertically. A hanging wall (HW) drive will be mined along each level parallel to and 20-30m from the reef to each end of the reef with cross cuts to the reef every 60m. The HW drive will be developed first before any ore drive development is undertaken. Once the HW drive is completed, the ore drive will be mined and rises for stoping developed. Each stope panel will be 60m long with a rise at each end and a draw point in the centre of each panel into the bottom ore drive. The resue air-leg mining method will be used with the reef fired first and extracted, following which the waste is fired to fill the stope and provide the floor for the next lift.

The current mine resource definition plan has been costed on the basis of 25m x 50m spaced drilling, and is generally expected to keep the reef positional uncertainty to within 10m. In some instances short segments of reef might be fault-offset but not resolved by the resource drilling. It is anticipated that integrating face / backs mapping along each level coupled with infill drilling from the hanging wall drives as required will allow such offsets to be identified sufficiently ahead of time.

Mining will commence 100m below the historical mine (16 Level) mining up towards the old workings and leaving a 20m crown pillar under 16 Level. The proposed approach is to drill the preliminary top level 2.8m x 2.5m drive at 25m strike separations from the exploration drive and HW drive, to allow the development of this top ore drive to follow the reef. It is likely that the survey of the historically mined reef is sufficiently accurate to allow the 16 Level reef track to be projected downwards and integrated with the resource interpretation. This can be checked by infill drilling from the HW drive.

The development of the first ore drive will provide an opportunity to test the degree of spatial variability of the reef as well as provide a starting point from which to project mapped structural disruptions (fault off-setting / inflections / large scale pinch and swells) of the reef down to the next level. Once the ore drive on the next level(currently expected to be 100m below) has been developed along the reef, then air-leg stope panel design (nominally 100mRL x 60m strike) will be based on the mapping along both drives (or in the case of the preliminary drive the 16 Level historical mapping) and any drilling intercepts within the panel.. The air-leg mining method proposed will allow cut by cut mapping of the reef and enable mining of the ore drives to be closely controlled.

Most of the historical mine data that has been recorded spatially (and can be three dimensionally rectified) has been captured on levels that are separated 45m vertically. There is little information therefore on the short scale geometric and grade variability in the vertical direction. This vertical short scale variability is an unknown quantity, but has the potential to increase ore loss and mining dilution. With the air-leg mining method proposed each slice will advance 2m vertically at a time and the mine geologists will be able to map the backs of each slice and hence provide good geological control. Diamond drilling from the HW drive will be an option in areas of structural complexity or if the reef is lost.

14.4.1 Drilling Requirements

The mine design includes an exploration drive at around the -360m level and about 80m off the reef plane, enabling holes to be drilled up dip as well as down dip. It will provide the drilling platform for the initial 25m x 50m spaced drilling covering the first 200m vertically below 16 Level (Figure 14-7), proving up sufficient resource for the first 4 years of mine life at the planned production rate. Drilling the first 100m high panel would be completed before drilling commenced into the second panel.

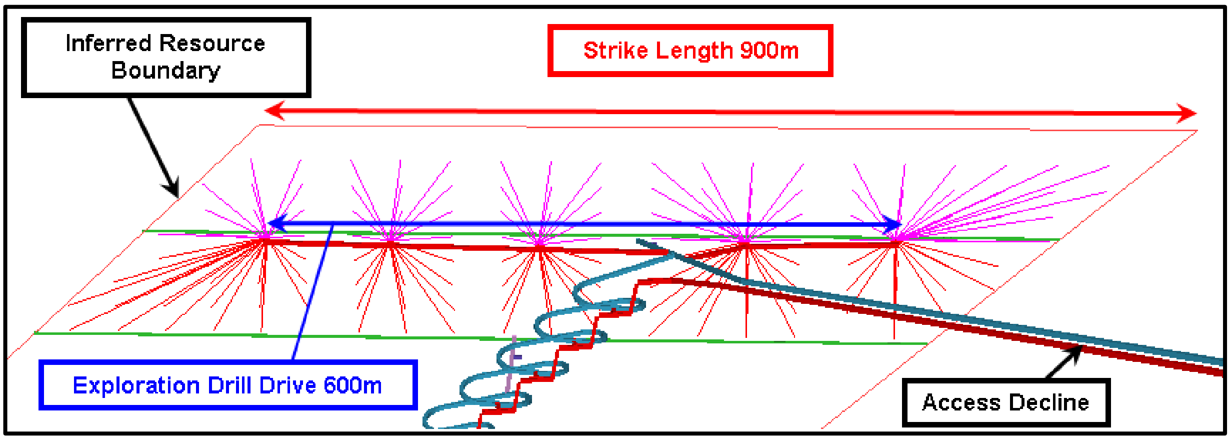

Figure 14-7: Initial Resource Definition Drilling

As the decline advances a similar exploration drive will be required about every 200 vertical metres for resource definition drilling of future mining panels. A total of 4 such drives will be required to cover the current expected extent of the reef at a rate of 1 every 2 years to maintain 4 years of resource ahead of mining. Costs have been included to also drill the possible extension at the northern end of the Birthday Reef, as identified in Figure 14-2. Total life-of-project resource definition expenditure has been estimated to be US$9M, commencing in the third year once underground drilling platforms have been established.

Closely spaced grade control drilling will be done on an “as required” basis from the HW drive on each development level which are spaced 100m apart vertically. Primarily this will be done to identify and quantify offsets in the reef both vertically and horizontally and track variations in reef thickness within a panel. An allowance of 2,500m of grade control drilling for each level has been costed, which at a cost of US$225/m for drilling and assaying totals US$4M for the life-of-mine.

It is recommended that the initial 25 vertical metres of reef be drilled to a 25m x 25m pattern to bed in the face / backs mapping grade control strategy. Following this initial period of mining, it may be that 25m x 50m will suffice for the majority of resource estimation to Indicated Resource classification. The northern and southern extremities of the reef however might require 25m x 25m, if or until mining demonstrates 25m x 50m is appropriate.

14.5 Risks

The Blackwater orebody is classified as an Inferred Resource and is based on the projection of historical workings at the Blackwater mine, which has been confirmed to continue at depth by 4 deep drill holes and their daughters. There is a risk that the remaining reef is either more complex, lower grade and / or thinner than that historically mined. Equally, there is an opportunity that the remaining reef has a higher grade and / or is thicker than that historically mined.

The proposed resue air-leg mining method poses some risks:

- The access decline is intended to be stood ≈100m into the HW of the Birthday Reef, with resource definition exploration drill drives to be mined nominally every 200m vertically, at ≈80m into the HW. Fault disruptions / offsets of the reef are expected to be common and many of these will elude the proposed broad resource drilling. The development of a HW drive prior to ore drive development will provide a drilling platform for closer spaced drilling to confirm the reef positon and test for any offsets;
- The 40-50m vertical spacing of historical level data provides little information about the vertical geometric regularity of the Birthday Reef (historical sample data suggests that down-plunge grade continuity will be reasonable). However, the proposed mining method with vertical advance of 2m for each slice and opportunity for backs mapping with each slice will give good mining control. Infill drilling from the HW drive could also be used to understand complex areas;
- The assumption that the average widths, average grades and average payability from the historical mining blocks is applicable to the Inferred Resource area. While justifiable for the estimation of a global Inferred Mineral Resource, is unsuitable for detailed mine planning; and
- The lack of confidence in the Inferred Mineral Resource estimate should be taken into account in all studies based on this estimate and in the reporting of the outcomes of those studies.

15 MINERAL RESERVE ESTIMATES

There is no Mineral Reserve estimate at the effective date of the PEA.

16 MINING METHODS

The mining study, including geotechnical assessment, on which this PEA is based, was completed by Mining Plus Pty Ltd. The geotechnical assessment and certain aspects of the mining method were reviewed by AMC Consultants Pty Ltd (AMC).

16.1 Mineral Resources Considered in the Mining Plan

The study references data from the historical workings located directly above the December 2013 Inferred Resource (Figure 14-2), and considers the Inferred Resource but excludes any possible extensions of the reef to the NNE or at greater depth.

As detailed in Item 14, there is no block model to support the mine design, however a wireframe was generated to represent the extents and orientation of the Inferred Resource. The wireframe has been delineated using an assumed strike and depth and uses data gathered from the historically mined levels located directly above the resource outline.

As detailed in Item 14 the assumed in situ grade of the reef is 23g/t Au at a thickness of 0.68m. It is expected that the reef will 'pinch and swell', with localised variations in grade and off-sets along strike. The resource grade includes adjustments for payability as described in the Item 14. Additionally, an allowance has been made to account for the localised areas of reduced thickness and/or grade which would prove uneconomic to mine. Production schedules generated have been prepared on the basis that 15% of the reef will prove to be uneconomic, due to a deficit in contained metal.

Whilst it is not possible to quantify the variances in this study, the selected assumptions are thought to be realistic.

16.2 Geotechnical Assessment

16.2.1 Introduction

Geotechnical input data is sparse and it is technically difficult to conduct detailed geotechnical drilling from surface on land not under the control of OceanaGold. Visual analysis of rockmass conditions either side of the reef and a review of geological core logs has been conducted. OceanaGold has conducted internal and external peer review of the proposed mining method, and AMC has conducted a review of geotechnical assessment and mining method.

There is limited geotechnical information to inform the expected ground conditions at Blackwater. Three holes were drilled along the alignment of the proposed access decline with only one intercepting the proposed decline depth. There has been no geotechnical laboratory test work conducted on core from the vicinity of the Birthday Reef. The geotechnical assessment below is therefore based only on core logging and core photographs.

A comprehensive geotechnical data collection programme is planned prior to the commencement of mining. Portal site investigation along with directional cover holes will be required to assess geotechnical and groundwater conditions prior to the commencement of the access decline.

The geotechnical assessment should therefore be read in the context of these limitations.

16.2.2 Geotechnical Data Review

AMC was engaged by OceanaGold in March 2014 to review the proposed mining method, (Air-leg Resue with flat-back drilling of waste) to determine its feasibility from a geotechnical perspective. AMC reviewed reports by Mining Plus and Kevin Rosengren & Associates Ltd (KRA). Additional data was provided to AMC, including spreadsheets with geotechnical logging for most of the holes referred to in the reports. AMC concluded that the proposed mining method was feasible though near mine geotechnical analysis is required prior to detailed mine design.

In 2013 a geotechnical assessment of the ground conditions at Blackwater was completed to determine design parameters for a proposed twin decline and for mining method selection and stope design parameters.

The KRA report, (Blackwater Mine Geotechnical/Mining Review, 2010), states that since the mine closed in 1951, there was no formal geotechnical information available. The only information available at the time for the Inferred Resource came from boreholes WA11 and WA11A, drilled in 1996.

16.2.3 Regional Geology

The Birthday Reef is a narrow quartz reef transgressing a bedded sequence of greywacke and argillite. The vein of ribbon-banded quartz averaging less than 1m thick which, apart from small offsets on late-stage faults, is a sheet-like body of northeast (NE) plunging shoots that is essentially continuous down-dip and along-strike for greater than 1km. It is characterised by an “oily bluish” colour, and by laminated bands of wall rock inclusions parallel to the margins of the vein (KRA 2010).

16.2.4 Local structures

The Birthday Reef strikes to the north-north-east, dips steeply to the north-west and plunges at around 40˚ to the north-east. The thickness of the reef ranges from 0.2m to 2.5m, with an average of 0.65m. About 90% of the reef intersections in the previously stoped area were less than 1m thick. The reef is dislocated by a series of major faults, striking 340˚ and dipping 65˚ to the north-east. The northern limit of the reef is defined by the Prohibition Fault (KRA 2010).

16.2.5 In-situ stress environment

The stress field at Blackwater has not been determined. Blackwater is situated very close to the Alpine Fault which is the tectonic boundary between the Australian and Pacific Plates. Large historical displacements on this structure are well known and the fault is active. Stress at this location is expected to be high at depth. Future works for assessment of the in-situ stress environment on the site need to be planned properly, and integrated with the drill program and other scopes of work, such as the decline excavation.

16.2.6 Decline Assessment

In 2013 Mining Plus completed a geotechnical study to determine the geotechnical design parameters for the proposed twin decline (mine access), and the underground workings. A desktop and literature review of geotechnical work from previous studies was completed.

In 2004 Golder Associates (NZ) Ltd assessed geotechnical aspects of the proposed decline. The decline was to be driven east (bearing 080°) from the Snowy River valley to intersect the base of the Prohibition Shaft which was developed during the first half of the 20th century. Four geotechnical holes were drilled to test ground conditions along the proposed decline. The holes were drilled to test ground conditions at the portal and through the historical Millerton workings in the initial part of the decline, and at potential raise bore sites along the decline's alignment. None of these holes intersected the currently proposed decline alignment.

In addition, the geology along the decline alignment (proposed in Golder Associate's report: draft R04645072-09) had been assessed by the Institute of Geological and Nuclear Sciences Ltd (IGNS, 2004). A cover sequence of glacial derived sediment blankets the full length of the proposed alignment. The decline was to be entirely within the Greenland Group rocks comprising alternating sandstone (Greywacke) and finer grained sediments (Argillite). The distribution of the geological units along the decline could not be estimated. Bedding and cleavage both strike NNE-SSW and are generally steeply dipping (60° to 80°) to the east (120° to 130°) and west (285° to 300°). The interpreted geological cross-section for the proposed Snowy decline (IGNS, 2004) is shown in Figure 16-1.

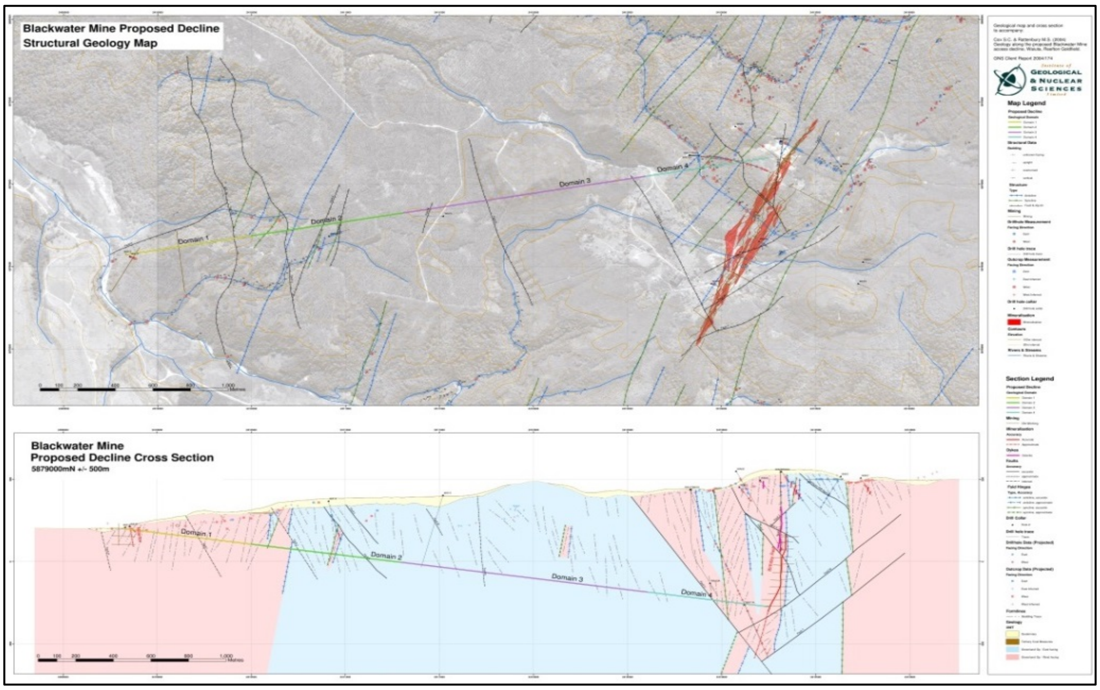

Figure 16-1: Geological Map and Cross Section (Looking North) for Proposed Access Decline[5]

The existing core from holes along the decline route (WA12, 13, 14 and 15) was re-logged for geotechnical information. Subsequently, historical logging by Golder Associates of the same holes was reviewed by Mining Plus and compared with the OceanaGold data.

[5] Waiuta, Reefton Goldfield - Cox S.C. & Rattenbury M.S. (2004)

Mining Plus observed that the cores had deteriorated significantly and recommended that the Golder logging should be used for geotechnical assessment. The limitation with this dataset is that only one hole (WA15) is deep enough to represent expected ground conditions in the decline path. Furthermore, the currently proposed decline has been moved about 200m north of the original route, consequently there is no directly applicable geotechnical information on expected decline conditions.

Given the lack of directly applicable geotechnical information, only general assessments of ground conditions and implied ground support requirements can be made at this stage.

The Q-system for rock mass classification was developed by Barton, Lien and Lunde. It expresses the quality of the rock mass in the so-called *Q-value*, on which are based design and support recommendations for underground excavations. The rock mass required is divided into Q ranges based on the level of available data from the 2004 field investigation campaign. For this PEA, only three Q ranges have been considered. The values of Q have been determined using representative parameters (Golder, DRAFT R04645072-09) and the results are summarized in Table 16-1, which represents a standard ground classification based on values of Q.

Table 16-1: Expected Q Index for Decline

Q ranges	Ground Classification
Q > 1	"Poor" or better
0.1 < Q ≤ 1	"Very Poor"
Q ≤ 0.1	"Extremely Poor" or "Exceptionally Poor"

16.2.6.1 Decline Dimension

The 3.3km long twin decline system from the west will be oriented across strike of the strata. The proposed decline profiles are:

- Main access decline 4m (W) x 4m (H) arched; and
- Ventilation decline 4m (W) x 4m (H) arched.

16.2.6.2 Decline Ground Support

Three major ground support zones are considered likely - Table 16-2:

Table 16-2: Description of Ground Support Categories

Support Category	Ground Classification	Bolt Ring Spacing (m)	Bolts per Ring (no.)	Bolt Length (m)	Mesh or Shotcrete Thickness (mm)
1	Q > 1 "Poor" or better	1.3-1.5	5 bolts in the backs 2 bolts in the walls	3.0	Mesh
2	0.1 < Q ≤ 1 "Very Poor"	1.3-1.5	5 bolts in the backs 2 bolts in the walls	3.0	Mesh + Fibre reinforced shotcrete, 50mm
3	Q ≤ 0.1 "Extremely Poor" or "Exceptionally Poor"	1.3-1.5	5 bolts in the backs 2 bolts in the walls	3.0	Mesh + Fibre reinforced shotcrete 75-100 mm

The rock mass shall be supported with 3m long grouted rock bolts installed in 1.5m spaced rings with bolts spaced 1.5m apart to within 2m of floor level. In very poor ground conditions, the standard size cross section (4m wide) must be reinforced by fibre reinforced shotcrete (50mm) over the exposed rock. When entering extremely poor or exceptionally poor ground conditions, the fibre reinforced shotcrete thickness will range from 75-100mm. Steel weld mesh shall be installed across the decline roof at the mining face. Mesh is designed to limit ground unraveling between the rock bolts and to protect personnel and equipment from minor rock fall. Mesh shall be welded galvanised steel mesh (standard 2.4m by 3m wide mesh sheets, 5.6mm diameter steel wire mesh welded on a 100mm grid), installed from shoulder to shoulder, to provide coverage from 2m above the floor.

The AMC peer review determined that the ground support standards suggested by Mining Plus are considered broadly appropriate for the three ground conditions categories they describe. AMC notes that additional ground support may be required for the intersections such as with stockpiles, passing bays (if used) and crosscuts linking the proposed twin declines. Typically, such spans would be supported with twin strand cable bolts to provide deep anchorage, high capacity rock reinforcement.

The design of such support requires information on the local structural conditions, but for budgeting purposes, a nominal grid pattern with a spacing of 2m by 2m through any wide span has been assumed for this study.

The distribution of ground support over the length of the proposed twin decline is shown in Table 16-3, and is an estimate for design purposes.

Table 16-3: Distribution of Support Classes (Updated from Golder, 2004)

Support Class	Chainage	Proportion	Total Metres
	0m to 100m	0%	
1	100m to 2,900m	50%	1,400m
	2,900m to 3,300m	0%	
	0m to 100m	70%	
2	100m to 2,900m	30%	1,190m
	2,900m to 3,300m	70%	
	0m to 100m	30%	
3	100m to 2,900m	20%	710m
	2,900m to 3,300m	30%	

The ground support distributions in Table 16-3 have been applied to the mine operating cost assumptions.

16.2.7 Stoping Assessment

16.2.7.1 Rock Mass Quality

Mining Plus states that 'the rock mass conditions in the mineralized zone appear to be generally poor'. In AMC's review, this general statement is consistent with the conditions that are apparent in the core photographs of drill holes intersecting the Birthday Reef, as appended to the KRA report.

The drilling of the deep holes was primarily to determine the presence, thickness and grade of the reef and not for geotechnical data collection. The country rock representing the hangingwall and footwall was not logged or photographed on site and was transported from drill site to Reefton in the back of a vehicle along unpaved roads. In the opinion of the exploration geologist it is almost certain that the core will have been damaged in transit.

The assessment (as reviewed by AMC) of likely ground conditions in the vicinity of the reef is represented in Table 16-4.

Table 16-4: Parameters to derive Rock Mass Quality (Barton et al, 1974)

Geotechnical Parameters		Average	Average for mineralised material	Average 5m above HW Contact
Rock Quality Designation	RQD	18	10	13
Joint Set Number	Jn	9	9	9
Joint Roughness	Jr	1	1	1
Joint Alteration	Ja	4	4	4
Average Greywacke UCS	MPa	144		
Average Argillite UCS	MPa	49		

In the Q system, the modified Q' value is calculated from Q' = (RQD/Jn)*(Jr/Ja).

Therefore Q' values near the reef are therefore likely to lie in the range 0.28 – 1.1.

16.2.7.2 Empirical Stope Design

Mining Plus conducted an assessment (reviewed by AMC) on using the Modified Stability Number (N') proposed by Potvin (1988) and based initially on Q':

N' = Q' x A x B x C where;

- Q' is the modified Q Tunnelling Quality Index (after Barton et al 1974);
- A is the rock stress factor;
- B is the joint orientation factor; and
- C is the gravity adjustment factor.

For the determination of modified stability numbers N', the following assumptions have been applied:

Rock Stress Factor (A)

The rock stress factor (Factor A) is determined from the ratio of the rock uniaxial compressive strength (UCS) over the maximum induced stress (σ_{max}), represented as:

$$Ratio = \frac{Uniaxial\ compressive\ strength\ (UCS)}{Maximum\ induced\ compressive\ stress(\sigma_{max})}$$

Factor A is calculated using the assumed mining induced stress field and material strength parameters for the expected stoping depth of 950m as the initial point of operations. The maximum vertical stress (σ_{max}) on the rock mass at 950m depth is estimated to be 22 MPa (see vertical stress measurements from mining and civil engineering projects around the world - after Brown and Hoek 1978). Intact rock strength is expected to be in the order of 140MPa, based on geological description of the Blackwater Gold Mine (Blackwater Gold Mine - Scoping Study, 2005). Testing will be required to confirm rock strength assumptions. Table 16-5 details the parameters used to determinate the ratio of UCS to induced stress, and Figure 16-2 shows the corresponding Factor A.

Table 16-5: Rock Stress Factor A

Expected Avg. UCS (MPa)	Depth (m)	σ1 (MPa)	UCS/σ1	Factor A
144	950	<25	5.76	0.50

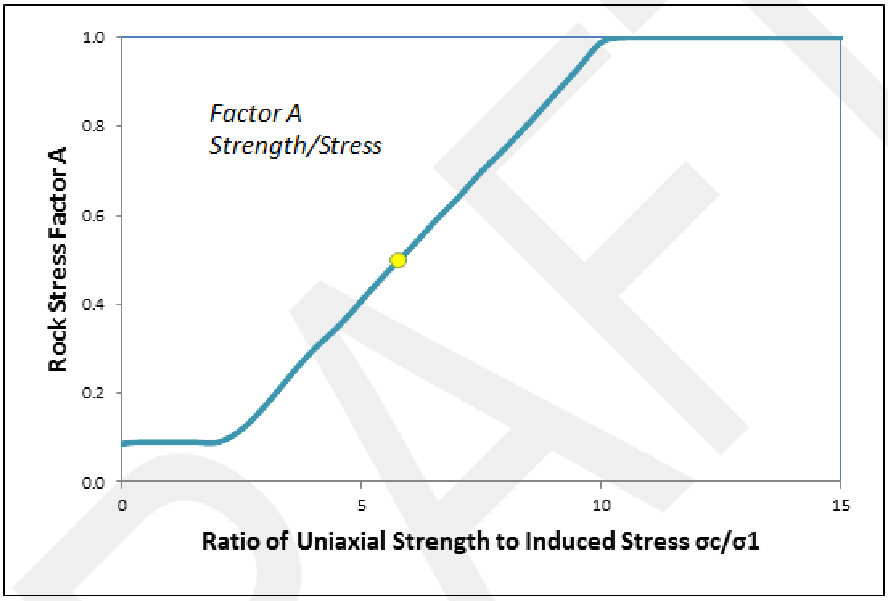

Figure 16-2: Rock stress Factor A

Joint Orientation Adjustment Factor (B)

Factor B is derived from an association with the ore-body joint orientation. Based on the orientation of geological structures at the Globe Progress Pit, the structures on site are commonly high dip, mostly parallel to the vein, which gives a Factor B of 0.3. This assessment will need to be reviewed once structural data becomes available.

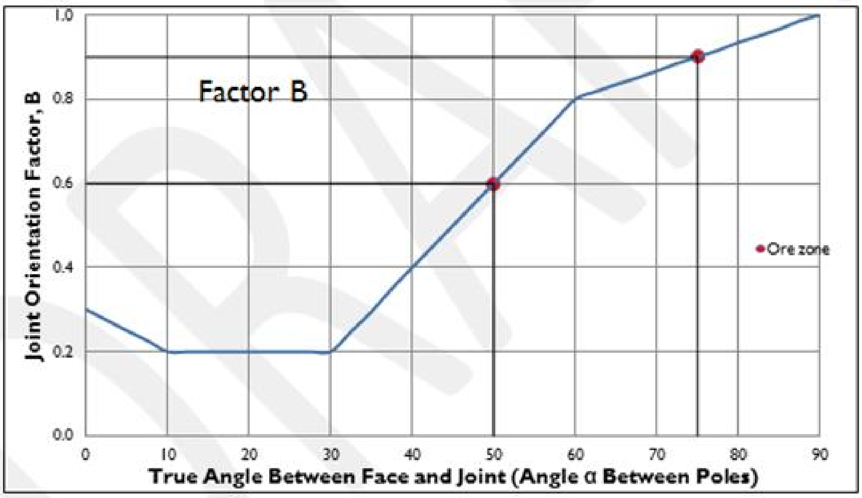

Figure 16-3: Adjustment factor B, (After Potvin, 1988)

Gravity and Failure Mode Adjustment Factor (C)

At this stage of the project and in the absence of further details of the structures, Mining Plus has assumed the deposit to experience 'slabbing' failure mode (Figure 16-4), and a deposit with a dip of 77° (Figure 16-5) determines the gravity adjustment factor to be 6.65.

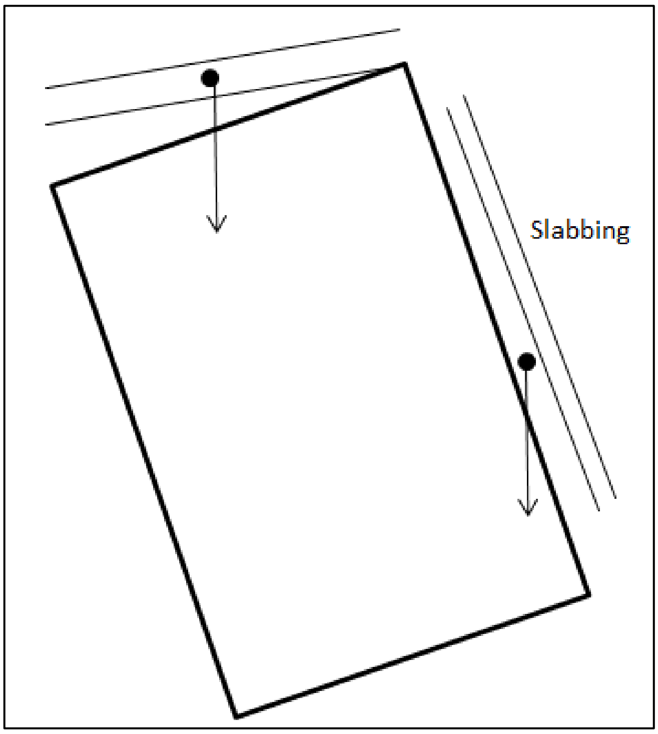

Figure 16-4: Slabbing is the mode of structural failure assumed in the study

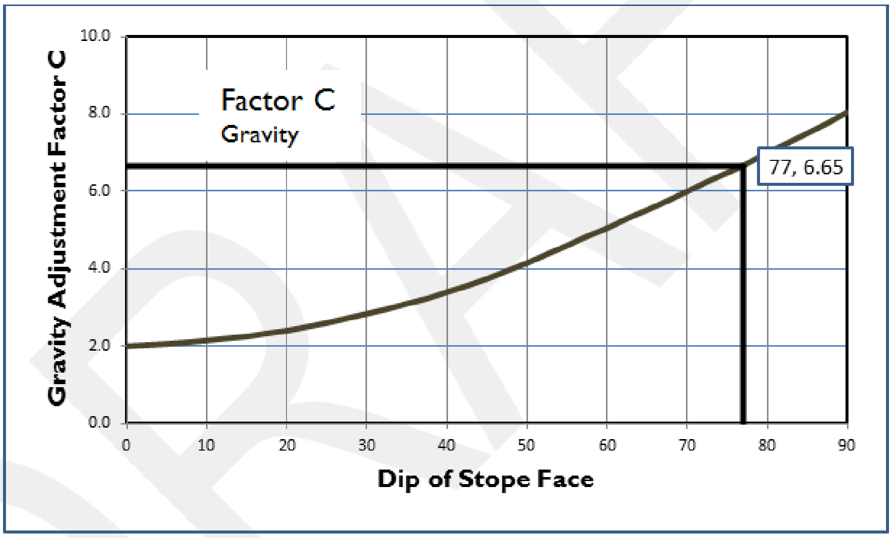

Figure 16-5: Gravity Adjustment Factor C

16.2.7.3 Stope Stability Summary

The analysis suggests critical hydraulic radii (HR's) for the ore-body zone defining the limit of the stable or stable-transition zone for stopes, for "walls" and "backs" dimensions. The wall values are summarised in Table 16-6.

Table 16-6: Stope Stability Summary

Mineralised material zone	Q'	A	B	C	N'	HR
Wall	0.28 – 1.1	0.5	0.3	6.65	0.28 – 1.1	1.6 – 2.6

16.2.7.4 Stope Span Recommendation

The implied hydraulic radius for unsupported stope walls is in the range 1.5 to 2.5 m. The implication is that only very small unsupported spans can be exposed. At a macro scale, conditions are not indicated to be amenable to large scale open stoping and mining methods that involve close wall support and small exposures are more applicable. Based on this finding, the primary mining method that has been selected is air-leg resue with waste flat-backing (refer to Item 16.5).

Once the access decline has been excavated and investigatory resource definition and geotechnical drilling has been completed the suitability of other mining methods will be revisited. As discussed in Item 14 there will be localised pinches and swelling of the ore-body, and also variability in the reef grade. Similarly, there will also be areas of better than average ground conditions and areas where conditions are worse.

In areas where the reef is thicker and ground conditions permit, it is envisaged that longhole benching will be employed as a mining method. In areas where ground conditions are considerably worse than average it is likely that a mechanised cut and fill mining method will be employed. Preliminary studies have been undertaken on these mining methods, and further work will be required once additional data is available from drilling.

As detailed in Item 14.5, an average width and grade have been assumed in the generation of the Inferred Resource detailed in Table 14-7, and this is not suitable for the purposes of detailed mine planning. As such, a single mining method has been considered when generating schedules for the base case scenario, being air-leg resue with waste flat-backing.

16.2.8 Portal Assessment

A boxcut will be required to establish the portal in competent rock. Based on topography a preliminary design was undertaken which positions the portal as shown in Figure 16-6. The location of the portal needs to be confirmed once the area has been surveyed in more detail, and geotechnical drilling has been completed.

There is a water channel (historical water race) above the boxcut's proposed location which currently provides sufficient drainage measures, but additional drainage works may be required once excavation of the boxcut and portal begins. Likewise, an appropriate drainage and diversion system around the box cut needs to be established in order to minimise or prevent the surface run-off water from entering the boxcut, with the potential to create issues in the underground operations.

A Wacker drilling survey over the proposed Snowy River decline site has been conducted by OceanaGold. A total of 114 wacker holes were completed along a line designed to transect the proposed portal site (Figure 16-6). The Wacker programme suggested the current planned portal site may not be in a desirable position, and moving the boxcut approximately 20m to the east may provide a preferable outcome. If required, an armoured tunnel with backfill will be installed for the first 20m or so.

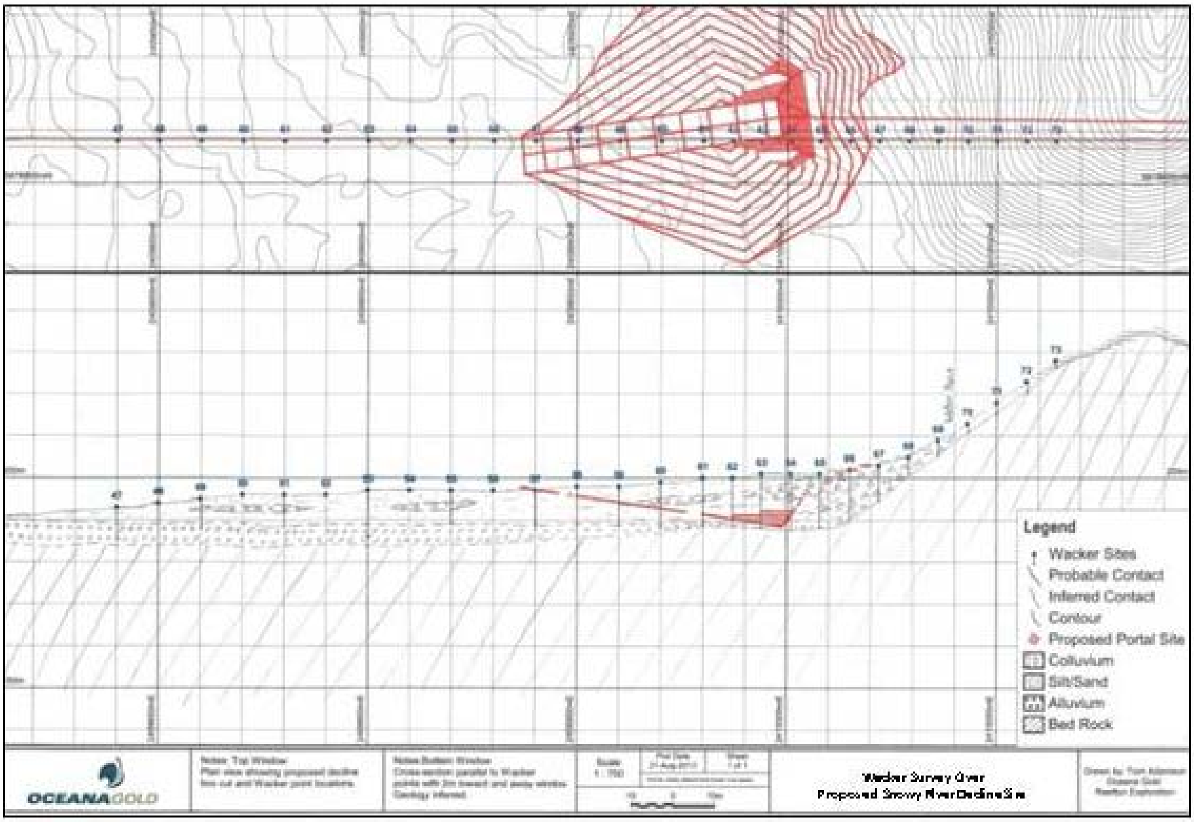

Figure 16-6: Wacker Survey over the proposed Snowy River Decline Site

Sampling at the eastern end of the boxcut (Wacker Site 73 on Figure 16-6) detects shallow soil cover over concealed bed rock. However, thickness of the unconsolidated material from surface increases consistently towards the west of the boxcut, reaching a depth of 12.7m at the proposed portal face.

Geotechnical site investigation drilling is required to confirm the preliminary ground conditions estimated by the Wacker survey program.

16.2.8.1 Required Standards

The proposed bore holes must reach specified depths. All core samples must be oriented during the extraction process. HQ is the suggested core diameter to allow for geotechnical analysis. If HQ is not possible, NQ will be the minimum core diameter that will allow for suitable geotechnical analysis, although more care will need to be taken when orienting the core due to the smaller size.

The collar locations should be placed along the baselines (drill lines) of the proposed decline path. The actual collar locations can be varied slightly (i.e. maximum of 20m in the X & Y directions) as required on site. Any additional lateral shift than this may require a hole to be re-designed or checked.

16.3 Mine Access (Exploration Decline)

The mine access arrangement was selected to provide the most favourable layout of the decline with respect to ventilation, egress, exploration drilling and future mine production phase. Various combinations of decline and shaft accesses were considered. Each access option was evaluated for capital cost, construction time, egress arrangements and ventilation for production and exploration.

16.3.1 Health and Safety Regulations

A key requirement is the final mine arrangement which must meet the New Zealand mining regulations.

The recently enacted Health and Safety in Employment (Mining Operations and Quarrying Operations) Regulations 2013, reg 172, require the operator of an underground metalliferous mining operation to ensure that, before stoping operations start at the mining operation, the operation has at least 2 egresses trafficable on foot (escapeways) that—

> *(a) are accessible from all stoping operations and lead to the surface;*
>
> *(b) are located strategically in response to the hazards that may arise at the mining operation and that will require evacuation;*
>
> *(c) allow for the passage of rescuers and rescue equipment, including stretchers;*
>
> *(d) are separated in such a way that a reasonably foreseeable event happening in one of the escapeways would not prevent persons escaping through the other escapeway; and*
>
> *(e) are maintained in a safe, accessible, and useable condition.*

The mine operator must ensure that a plan is made of the mining operation as at the date of commencement of the mining operation, which is reviewed and, if necessary, updated at least once every 3 months in relation to the parts of the plan that identify points of access, egresses, and refuges.

The selected main access (exploration decline) is twin declines from the surface at the Snowy River portal site to the Birthday Reef (Figure 16-7), for the following reasons:

- Complies with New Zealand Mining Regulations (Department for Health and Safety in Employment, 1999) and Guidelines in regards to requirements for primary ventilation and secondary means of egress;
- Favourable construction cost;
- Construction duration;
- The ability to integrate a primary ventilation rise near the base of the decline;
- Access to exploration platforms;
- Noise from the surface infrastructure and portal would have no impact on the local community;
- Allows for future production and ventilation requirements; and
- Matched to the preferred portal location.

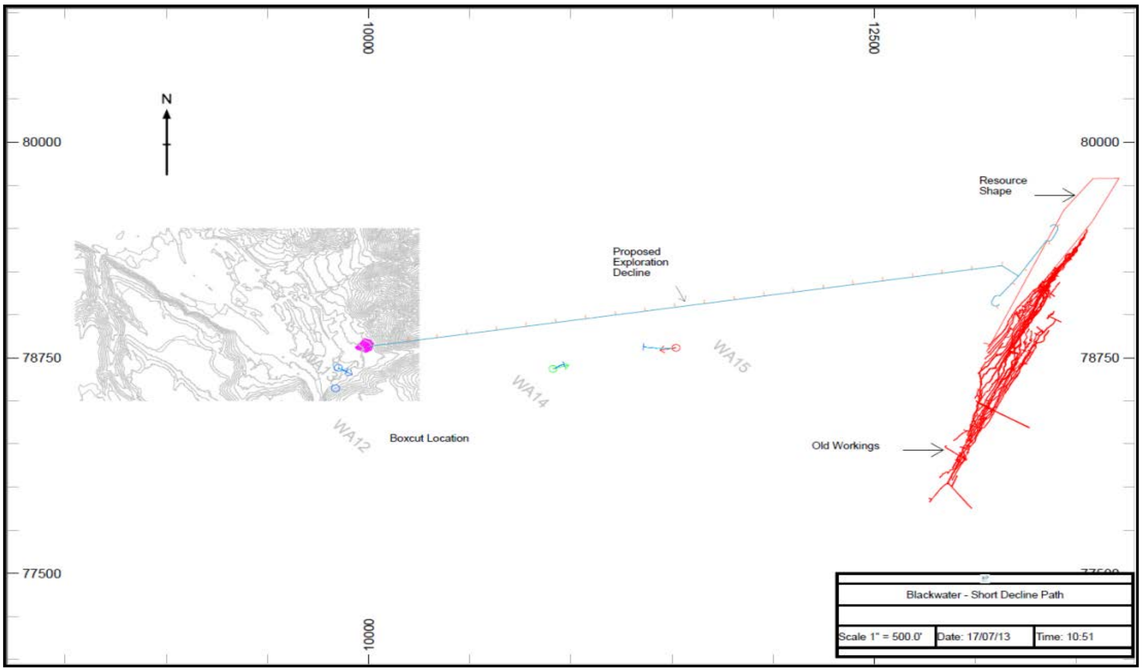

Figure 16-7: Mine Access Alignment of Twin Decline (Exploration Decline)

16.3.2 Construction Duration

The availability of multiple headings during construction of the twin declines will result in improved mobile equipment utilisation and a concomitant improvement in advance rates. The estimated time for construction of the twin exploration decline is 24 months which includes construction of the portal as well as the exploration drive adjacent to the reef, which will be used for resource definition drilling.

Initial resource definition diamond drilling will be undertaken when the access decline is approximately three quarters completed. A drill platform will be mined from the access decline to enable diamond drilling whilst decline excavation continues. Core from these earliest holes will be used for confirmatory metallurgical test work and preliminary assessment of the Birthday Reef. Figure 16-8 is a schematic showing an overview of the project timeframe.

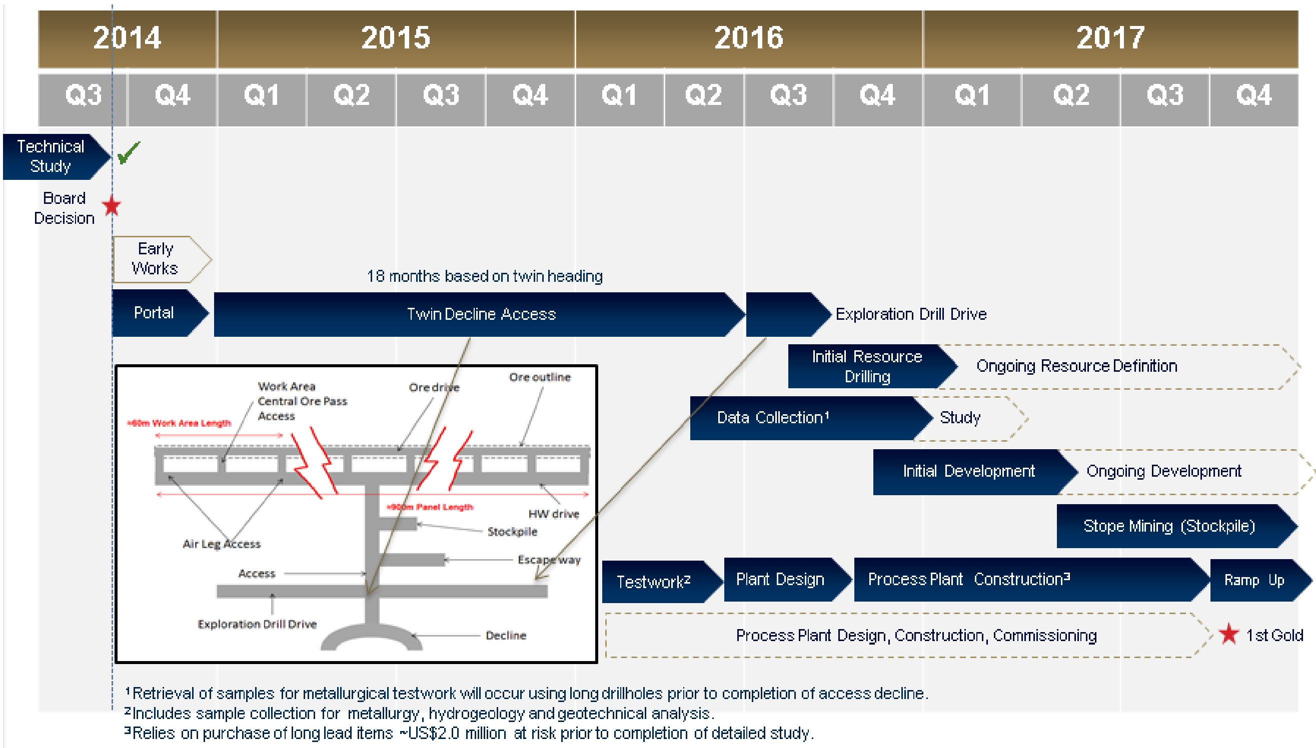

Figure 16-8: Project Timeframe

Cover drilling of sub-horizontal diamond drill holes in advance of the decline faces is proposed. This will provide prior indication of geotechnical and groundwater conditions.

16.3.3 Portal Location Review

The portal location is situated at the foot of the steeply grading hills of the Victoria Forest Park and the banks of the Snowy River are located approximately 200m away laterally.

The following observations were noted during the site visit:

- A water race (historical feature) bounds the northern side of the boxcut;
- Design changes to the boxcut and portal will need to consider the disturbance of the water race;
- The water race provides a natural water diversion for the boxcut;
- Two small unnamed creeks flow intermittently in the area of the boxcut; and
- The boxcut RL is approximately 10m above the banks of the Snowy River.

The summary findings conclude:

- No fatal flaws were identified for the portal location;
- Flood mitigating measures will be required once the boxcut excavation is completed;

- Geotechnical drilling is required within the boxcut design footprint, as well as a geotechnical investigation hole for the long-hole rise that connects the ventilation decline to the surface;
- Packer testing will be required immediately after the geotechnical drilling programme to test the permeability of the boxcut rock mass;
- A ground support management plan for the boxcut is required prior to construction; and
- Geotechnical considerations for the portal are discussed in Item 16.2.8.

16.4 Mine Development

The mine design for air-leg resue mining method is segregated into seven production panels, each 100m high. The strike length for the highest elevation production panel (immediately below a crown pillar to be left in situ below the historical Blackwater Mine workings) is approximately 900m along strike. A central spiral access decline is proposed, as seen in Figure 16-9.

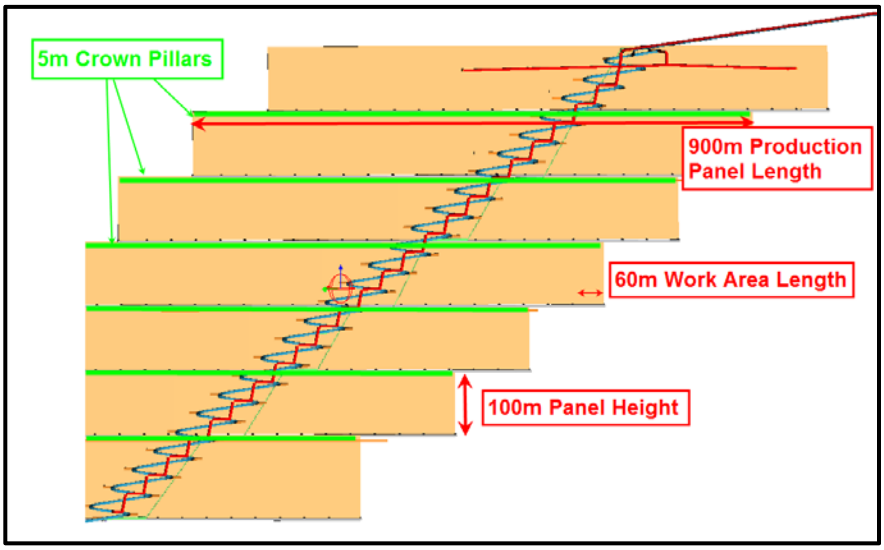

Figure 16-9: Air-Leg Resue Long-Section Schematic Layout of Ore Body, Looking East

At the base of each of the production panels a standard development layout is recommended, as illustrated in Figure 16-10.

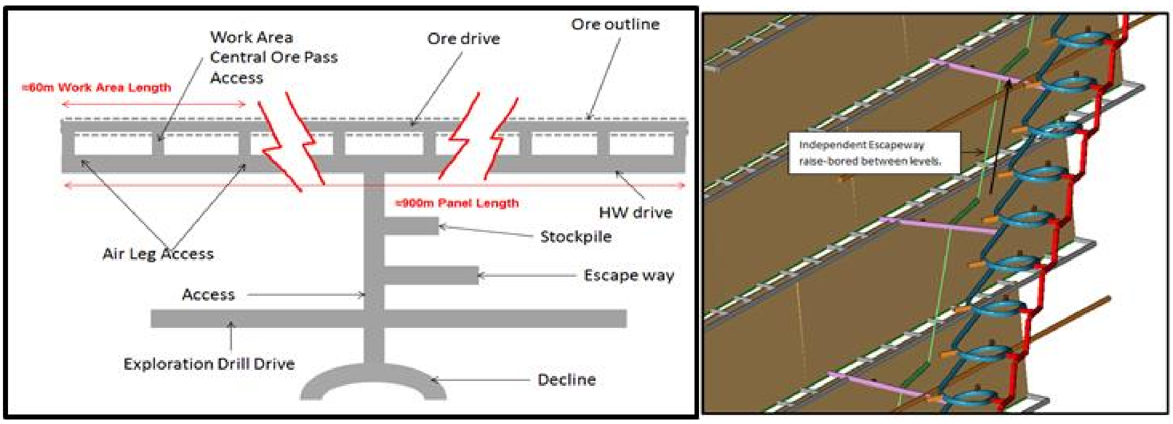

Figure 16-10: Air-Leg Resue Level Layout

Mechanised mining will be utilised for all development mining, including the decline and all assocated waste development to reach the ore drive.

The hanging wall drive is required for access to the stoping work areas, and also for loader access to each of the ore passes within the production panel. It will also be available for infill diamond drilling, if required to better define the reef in areas of complexity identified from the resource definition drilling.

Lateral ore pass accesses will be mined every 60m along strike, with dimensions of 4m x 4m to facilitate bogging with a 10t capacity loader. Accesses for air-leg miners will be mined between each of the ore passes to enable access to each end of every 60m long work area.

Once "on-ore", air-leg development will excavate the ore drive to establish the base of a production panel.

Adjacent to the orebody, a central spiral decline which is offset 100m in the hanging wall of the orebody will provide access to the working horizons. This offset distance is a conservative approach due to the unknown dislocation (offset) extents of the Birthday Reef. Evidence of off-sets has been sighted in the Levelplans of the old workings. Table 16-7 shows the development parameters used in the development design.

Table 16-7: Blackwater Development Design Parameters

Development Type	Profile	Width	Height	Shape	Gradient	Length	Notes
Decline	A	4.0	4.0	Arch	1 in 7	-	Spiral radius of 20m
Level Access	A	4.0	4.0	Arch	up to 1 in 7	100m	Graded for access
Stockpile	A	4.0	4.0	Arch	1 in 50*	17.5m	Graded for drainage
Sump	B	4.0	4.0	Arch	1 in 6 (down)	7.5m	-
Return Air Drive	B	4.0	4.0	Arch	1 in 50*	-	Graded for drainage
Escapeway Drive	B	4.0	4.0	Arch	1 in 50*	-	Graded for drainage
Ore Drive	H	2.8	2.5	Arch	1 in 100*	900m	Graded for drainage
Inter Level Return Air Rise	N	3.5	3.5	Square	> 70° dip	<30m	Backs to floor
Escapeway Rise	X	1.5	1.5	Circle	> 70° dip	120m	-
HW Drive	B	4.0	4.0	Arch	1 in 50*	-	Graded for drainage
Ore Pass Access	B	4.0	4.0	Arch	1 in 50*	-	Graded for drainage
Air Leg Access	B	4.0	4.0	Arch	1 in 50*	-	Graded for drainage
Air Leg Lifts	W	2.4	2.0	Square	1 in 100*	60m	Graded for drainage

16.5 Mining Method

The base case scenario mining method selected for the PEA and documented in this Technical Report is air-leg resue stoping. Development mining to access the reef is conventional mechanised mining, as discussed in Item 16.4. The reef will be mined along strike in the first instance to create 60m long work areas and then mined up-dip in 2m cuts.

Hand-held (air-leg) mining techniques are proposed along with resue firing which enables preferential segregation between ore and waste. The average reef width is assumed to be a constant 0.68m, and 50% dilution has been applied to occur with the ore firing, with a resultant ore firing width of 1.0m. Once fired onto a marker bed of cemented slurry, the ore will be scraped to a central ore pass using a scraper. Upon completion of ore scraping, 2m of waste will be fired and retained in the stope as fill for the next lift. The bulking factor of the waste when fired will swell to occupy the total stope mining width of 3m. Ground support will be installed as required.

16.5.1 In-Ore-body Materials Handling

Within each stope work area a scraper will be used to move blasted ore to a central lined ore pass (Figure 16-11).

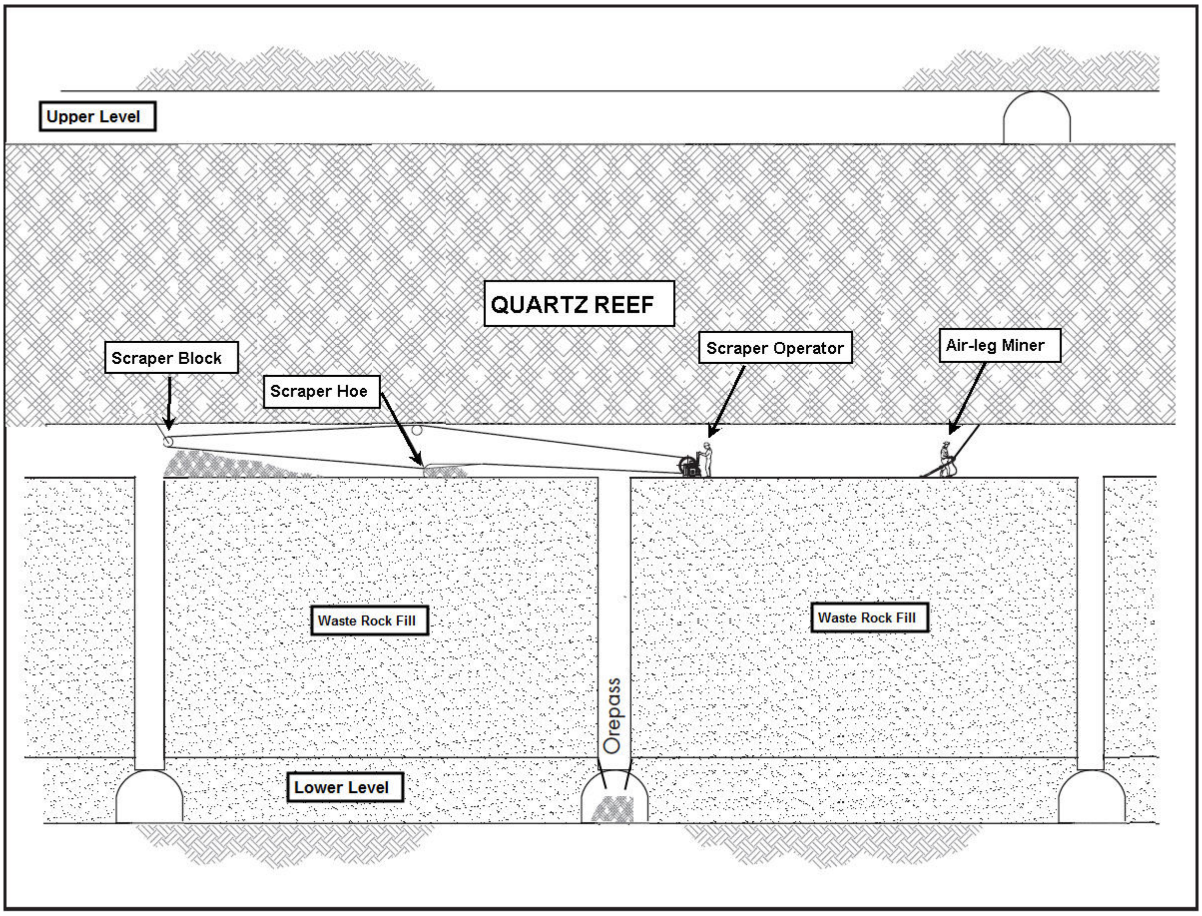

Figure 16-11: Air-Leg Resue Ore Scraping

The use of scrapers for movement of blasted material within the stopes will require pre-start checks to assess the condition of the scraper hoe, all snatch blocks and the scraper rope itself. The requirement for replacement ropes has been factored into the mining cost estimates along with additional scrapers and hoes for overhauls and major repairs minimising any impact to productivity and ensuring safe operating practices.

When scraping, the operator will set up a mesh guard in front of the scraper. The use of a cement slurry marker bed will reduce the likelihood of the scraper hoe or rope getting caught and will ensure maximum recovery of the high grade quartz reef.

Loaders will access the base of each ore pass via the hanging wall drive, tramming back to the central decline access for loading directly into trucks or from loader to stockpile to truck (Figure 16-10). All waste blasted during the resue stoping operations will be left within the stopes as a working platform to access the next mining lift (Figure 16-11).

16.5.2 Mining Sequence

The ore component (reef) of the stope is drilled out as a slot for 60m along strike using half uppers. At the same time 3m of waste is also drilled out at each end of the 60m work area, above the ladder ways (Figure 16-12). Ore pass cans and ladder ways are then installed or extended.

Scraper blocks and ropes are pre-installed in the shoulder away from the ore to provide protection during blasting. Once these activities are completed, the drilled gold bearing reef material for half of the work area is charged and fired, together with the waste portion above the ladder-way at one end.

The blasted ore is scraped back to the central ore pass and loaded from the base of the ore pass to surface using conventional LHD and trucks. On completion of scraping, a work area is established above the ladderway access at the end of the panel section that has been blasted by installing ground support.

The initial waste (3m along strike at the ends of the panel) which was fired with the ore is levelled. This creates a working platform for blasting the remainder of the waste within the work area.

Flat-back (horizontal) holes are then drilled and blasted in waste along the strike of the stope. The waste is thrown down into the void below and remains in the stope as backfill. After each 2.7m length flat-back cut, check-scaling is undertaken and ground support is installed before drilling the next flat-back round, progressing along strike.

Once flat-backing is completed along the first half of the work area, a marker bed of quick-setting grout slurry is placed on top of the blasted waste and the cycle repeats for the other half of the work area. The marker bed may not be required but has been costed to mitigate the risk of the scraper hoe becoming snagged, and to reduce the risk of ore loss into the blasted waste from the previous mining lift. These risks were noted during peer review of the mining method.

The sequence is illustrated in Figure 16-12 to Figure 16-17.

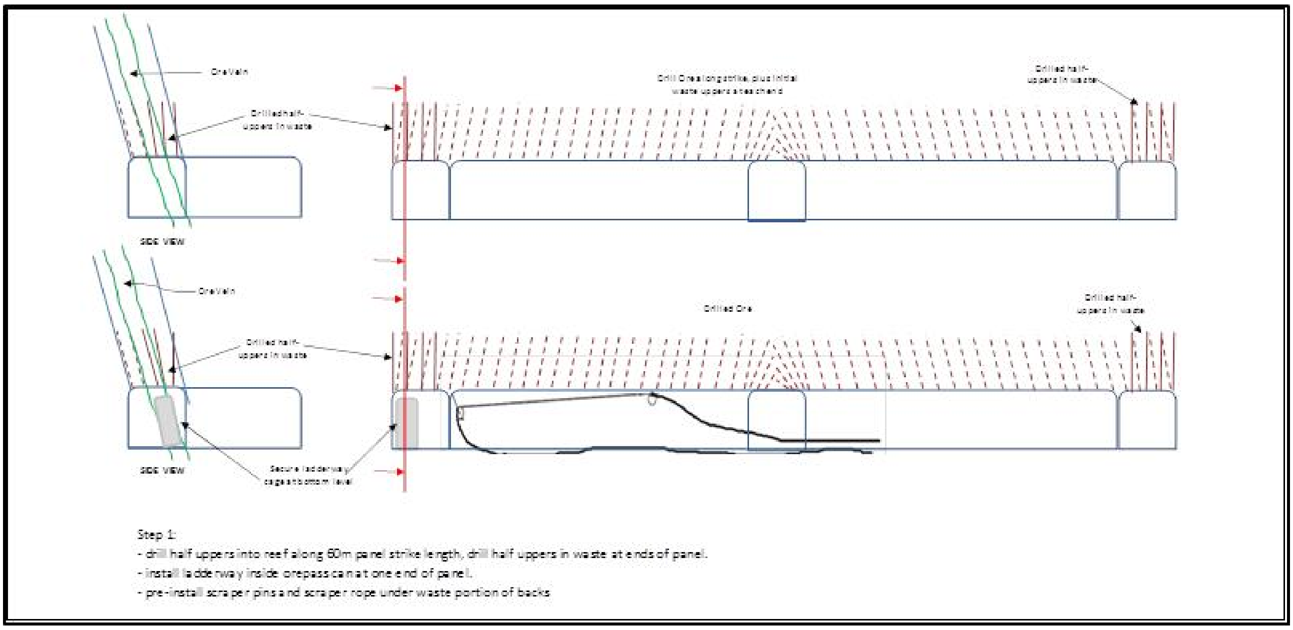

Figure 16-12: Air-Leg Resue Drill Quartz Reef along 60m Strike of Panel

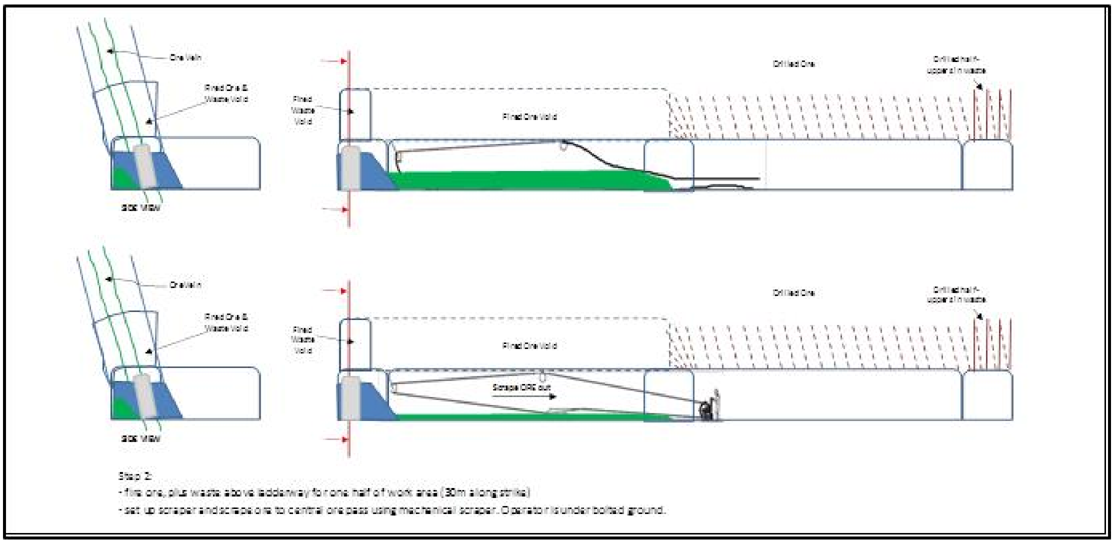

Figure 16-13: Fire Ore and Waste Above Ladderway. Scrape Ore to Central Ore Pass

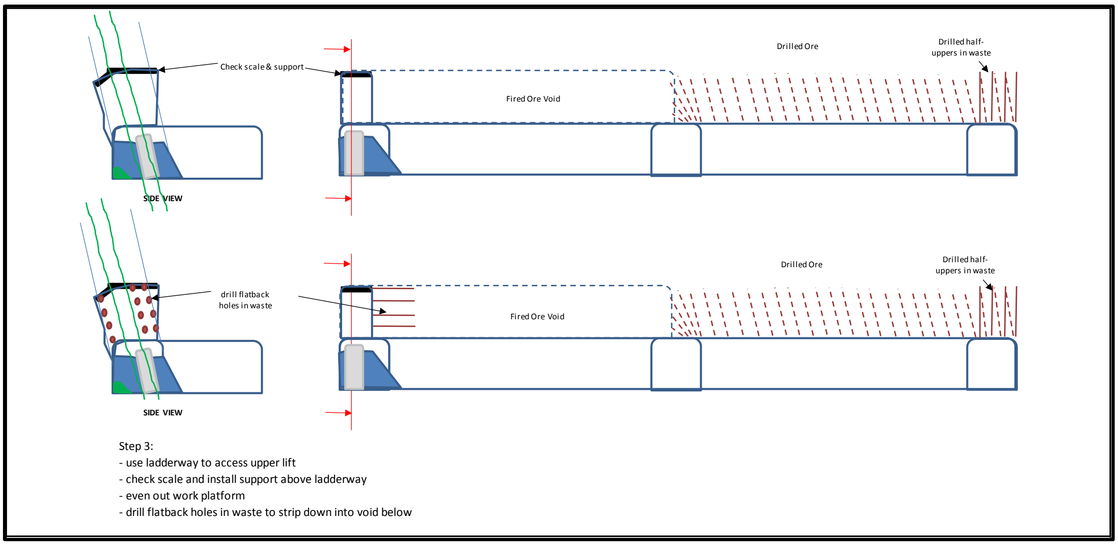

Figure 16-14: Check Scale and Install Support

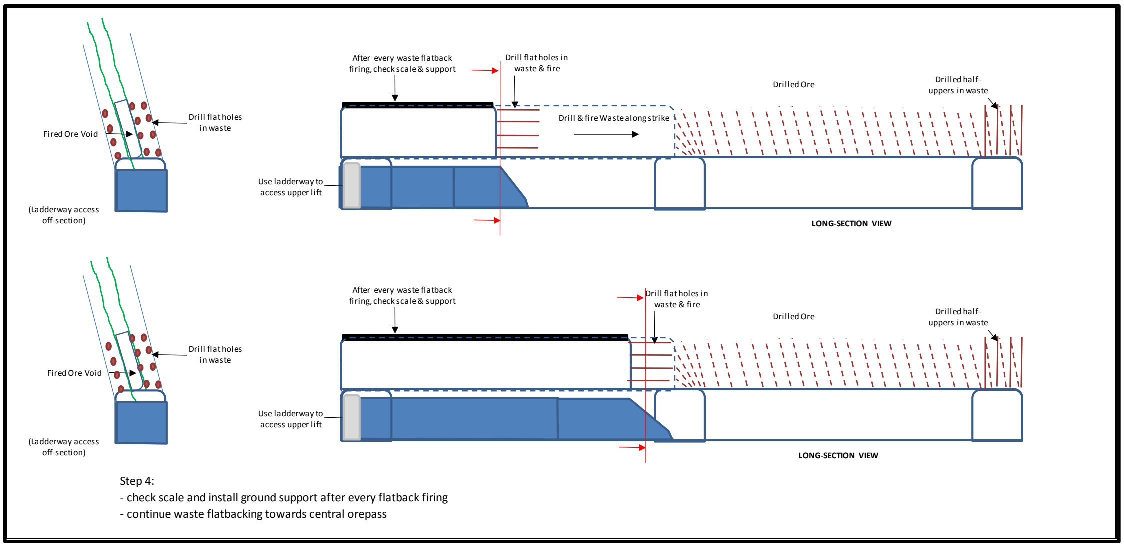

Figure 16-15: Drill and Blast Flat Back Waste. Check Scale and Support

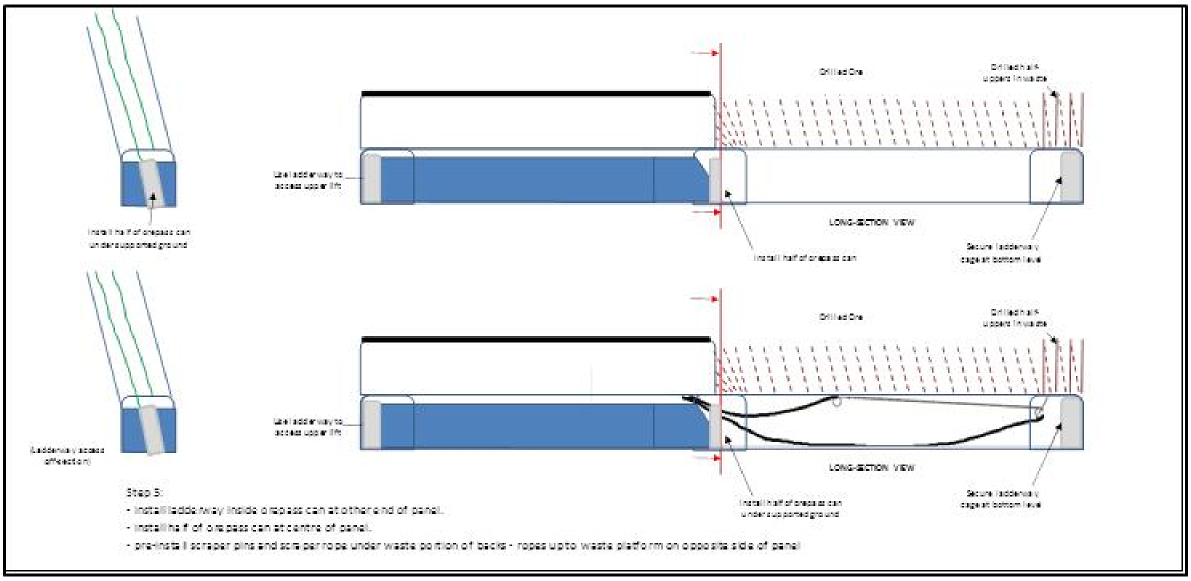

Figure 16-16: Establish Second Half of Panel

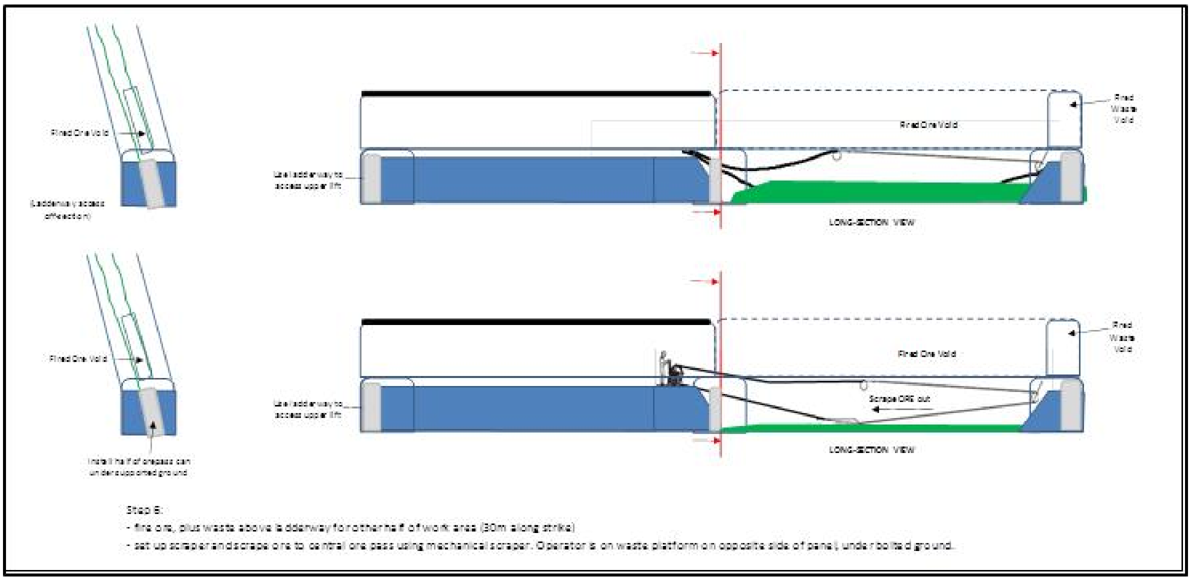

Figure 16-17: Fire Ore and Scrape to Central Ore Pass

16.5.3 Internal Peer Review

OceanaGold conducted an internal peer review process on the mining method and sought external opinion on the geotechnical aspects of the proposed mining methods. The outcome of the internal peer review raised 13 technical and safety queries, which were classified into either safety or economic queries and adjustments to study assumptions recommended. The final mine design adopted incorporates the adjustments recommended during this peer review.

The items highlighted during the mining method peer review and their respective corrective actions are listed in Table 16-8.

This peer review on mining method is not a replacement for the risk register but explains the development of the mining method during the study.

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Table 16-8: Peer Review Comments Cause and Effect

Item	Description	Category	Cause	Effect	Comments	Mitigation / Adjustment to Study
1	Manual handling	Safety	Manually intensive work	Personal Injury	Hand-held mining is recognised industry wide. Assess learnings from other hand-held operations.	Study now provides for additional headcount for support labour
2	Unit mining cost too conservative ($166/t)	Economic	Optimistic productivities	Lower project value	Three separate air-leg mining contractors have been consulted and compared with productivity cycle times calculated from 1st principles.	Adjust productivities, increase unit cost to $223/t (25% increase). Adjustments made to study assumptions to reflect 9hr productive shift thereby complying with NZ mining regs.
3	Decline productivity rate	Economic	Optimistic productivities	Lower project value	The development rate of the twin decline will depend on ground conditions. Blackwater has twin declines being mined simultaneously allowing for significantly higher equipment utilisation than single advance rate which has Jumbo utilisation times of <50% and loader <10%. The 4mx4m profile reduces ground	None
4	Portal geotechnical conditions	Economic	Poor ground conditions	Lower project value	Site investigation is recommended and already included in study.	None
5	Decline geotechnical conditions	Economic	Poor ground conditions	Lower project value	Cover' holes for geotechnical and ground water investigation are already proposed and costed into the study.	None
6	Stope geotechnical conditions	Safety / Economic	Poor ground conditions	Injury / lower project value	Firing the ore slot is likely to relax the waste material potentially causing a pendent effect. Need to assess then mitigate the likelihood.	Modelling analysis. Move ore slot to centre of waste, recommendation for ground support. External review of mining method (AMC).
7	Scraper operation	Safety	Rope breakage / snagged Scraper.	Injury or fatality	Set-up of scraper is important to de-energise the rope in the unlikely event of breakage. Break of rope could occur in two scenarios, too much pressure or lack of preventative maintenance. Guards at the block should be used to protect the operator. Usual safety awareness and precautions should be applied as with any mining equipment.	Include in the study provision for marker beds which will significantly reduce likelihood of scraper getting caught up in waste.
8	Scraper operation - dilution / ore loss	Economic	Scraper getting caught up on waste pile.	Lower project value	Marker bed, cement slurry hole from hanging wall drive above or concrete pump.	As item 6 above.
9	Ground Conditions - Dilution	Economic	Poor ground conditions.	Lower project value	The study has 51% plant utilisation and low trucking (~250t per day). Capacity is therefore available to deal with additional dilution.	Include in the study an increase in dilution to provide for additional dilution.
10	Dilution on either panel sides	Economic	Adjacent panel waste dilutes during mining	Lower project value	Ore cans are installed encapsulating ladder way to prevent waste from adjacent panels diluting ore slot area.	None
11	Material below COG	Economic	Lower than expected reef grade.	Lower project value	Historical gram metre data suggests that almost all of the reef will be recovered. This level of study (scoping) assumes a single grade and a single thickness. The economic model has provision for ~$24million of resource drilling ahead of mining panels.	Review ore loss assumptions during subsequent study phases.
12	Effectiveness of drilling pattern	Economic	Poor ground conditions	Lower project value	Drill and blast design should be reviewed and optimised in subsequent studies.	Review drill and blast assumptions during subsequent study phases.
13	Hours per shift assumption	Safety / Economic	NZ Legislation	Lower project value	NZ mining regs requires 30min unpaid meal break plus 3 x 10 mins rest break. In addition re-entry time, travel, meetings etc.	Adjust work hours in study from 10.5hrs to 9 hrs

16.5.4 External Review (AMC)

An assessment of the geotechnical aspects of the proposed air-leg resue mining method was undertaken by AMC Consultants (AMC). The review comments from AMC were taken into consideration when refining the air-leg resue mining method with flat-back mining of waste.

Based on the AMC review, the ore slot was moved to the centre of the stope – previously the waste was positioned in the hanging wall of the reef. The effect is expected to be reduced dilution and a reduction in risk associated with stability of the relaxed back and walls. Figure 16-18 illustrates a simple 2D model of the originally proposed slot location, using some reasonable assumptions about the stress environment, which would result in a de-stressed/relaxed area as a result of firing the slot (essentially a pendant effect).

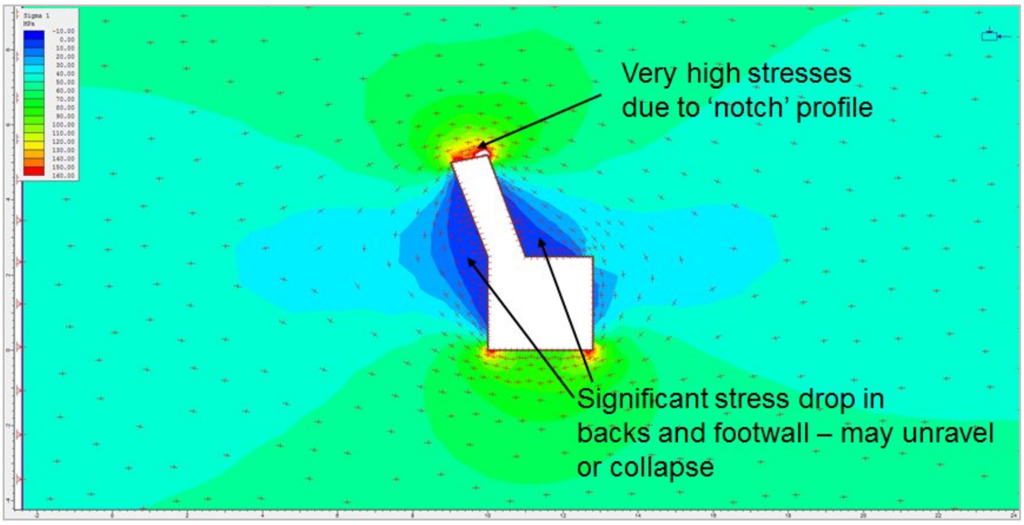

Figure 16-18: Initial Air-leg Resue Slot Position; Likely Stresses After Firing Up-holes

By positioning the ore drive so that the reef is more centrally placed (Figure 16-19), the zone of relaxed ground in the hanging wall of the up hole slot is smaller and hence easier to support. Allowing for the likely profile after blasting the slot, the footwall side of the backs should also be easier to control.

AMC concluded that the modified mining method proposed is considered feasible from a geotechnical perspective, with an acknowledgement that further geotechnical assessment in the vicinity of the decline and the ore-body will be required prior to detailed mine design.

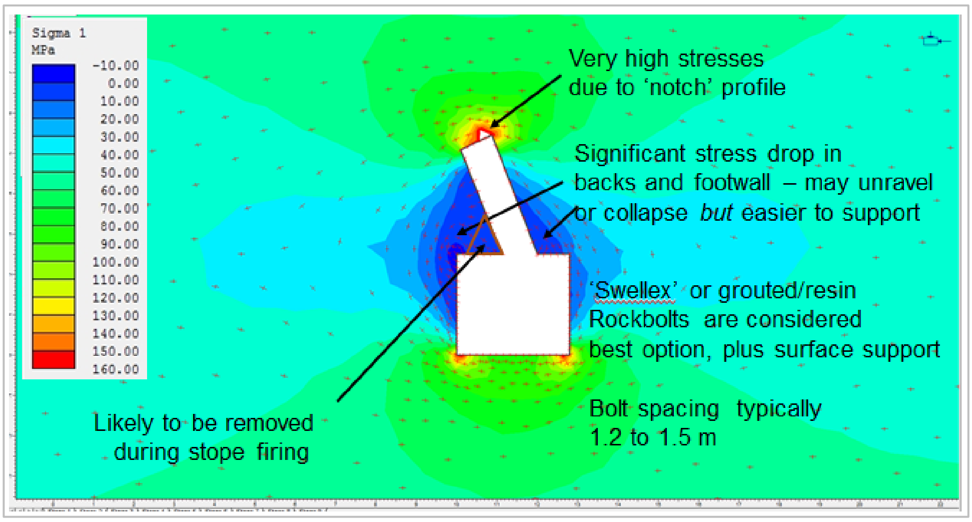

Figure 16-19: Modified Positioning of Ore Slot as Suggested by AMC

It is again noted that there is extremely limited geotechnical data available on which to assess the viability of any mining method. Figure 16-18 and Figure 16-19 were generated using assumed in situ stress conditions based on the depth of the ore-body and assumptions about stress gradients given the location of the Project. Until more detailed information is available, to be attained after the excavation of the access decline, it is not possible to provide a more detailed assessment of the mining method. As discussed in Item 16.2.7.4 other mining methods such as longhole benching and mechanised cut and fill will be considered for implementation once additional information is available.

16.6 Materials Handling (ex-Ore-body)

A material handling options assessment was undertaken to determine the most suitable materials handling method for the Blackwater project, with consideration given to:

- Initial capital expenditure;
- Ongoing production requirements; and
- Operating cost expenditure for Life of Mine.

The outcome of the assessment determined that trucking is the most effective method of materials handling for the following reasons:

- Low capital expenditure;
- Flexibility between pre-production activities and first ore;
- Skill levels of operators are quite low compared to other methods, and easily sourced; and
- Minimal surface infrastructure is required during the pre-production and steady state activities.

16.7 Mine Design

16.7.1 Cut-Off Grade

A cut-off grade has not been applied to the mine design. The assumption applied to the base case scenario is that the ore-body has a constant grade (23g/t Au) and reef thickness (0.68m) which are based on the Inferred Resource. Supporting rationale for the thickness and grade are discussed in Item 14. The Mineral Resource model that has been generated does not include allowances for local variations in thickness and grade. To account for the variability that will undoubtedly be encountered during mining, and in recognition of the historical payability factors that were encountered during mining operations between 1908 and 1951, it has been assumed that only 85% of the strike length of the ore-body will be extracted as payable ore. In addition, a sensitivity analysis was undertaken whereby the reef thickness and reef grade were flexed by ±15% and ±30%, to determine the impact on the Project NPV. The results are presented in Item 22.5.

Once resource definition drilling commences from the exploration decline it will be possible to derive a cut-off grade. Detailed revisions to the design are expected prior to commencement of mining operations, and also during the mining cycle as additional resource definition and geotechnical drilling results become available. With the proposed air-leg resue mining method it will not be possible to leave pillars within the working stopes, so mining widths will be modified to suit.

16.7.2 Production Panels

A 5m sill pillar will be left at the top of each production panel, as indicated in Figure 16-9. These pillars will provide a physical barricade to prevent waste rock material in the level above flowing down into the current work areas. Additional geotechnical work will be required to define the final sill pillar dimensions, with the current assumption of 5m thickness being a preliminary estimate.

16.7.3 Stope Design

Design parameters are shown in Table 16-9 with a schematic long-section shown in Figure 16-9. Trial mining and geotechnical data collection is recommended to validate the design assumptions. Geotechnical data will be collected from the planned underground resource definition drilling, to develop an improved understanding of the geotechnical conditions within and near the Birthday Reef.

Table 16-9: Stope Design Dimensions

Dimension	Value (m)
Production panel strike length	900
Production panel height	100
Work area strike length	60
Work area height	100
Ore drive height	2.5
Quartz vein average thickness	0.68
Ore resue width (incl. dilution)	1.0
Waste resue width	2.0
Resue lift mining height	2.0
Sill pillar height	5.0

It is possible that there may be some areas with better wall rock conditions. This may allow consideration of more productive, lower cost mining methods such as longhole narrow vein benching. Similarly, in areas where poorer ground conditions are encountered it is likely that a more costly mining method such as mechanised cut and fill will be employed.

16.7.4 Mining Width

The premise of the air-leg resue mining method is that sufficient waste is generated within the mining cycle to preclude the requirement to introduce fill from outside of the stope. As such, when the waste component of the resue is blasted the broken rock swells to occupy the void originally occupied by both the in-situ ore and waste components.

The host rock around the Birthday Reef is greywacke. An appropriate swell factor is 50-60% (Society for Mining, Metallurgy, and Exploration, Inc., 2011). The more conservative value of 50% was used for the PEA (the greywacke swell factor expected should be determined during early phase trial mining once the exploration decline is constructed).

To generate enough blasted waste to fill the void created by the ore and waste firing, the waste must be taken 2m wide. It should be noted that this width of waste stripping is not required to achieve a suitable width for safe and efficient mining, but rather to generate sufficient waste to fill the total void generated through the resue mining method.

Managing the waste mining width will be refined during short-term mine planning when true reef thickness within panels and swell factors are known. If the swell factor is a lower percentage, or if the width of ore is greater, then the width of waste mined during the air-leg resue mining cycle will need to be increased to generate sufficient broken material to fill the mined void.

16.7.5 Mine Dewatering

To allow efficient dewatering a sump is located on every level access. The sumps are vertically aligned to allow interconnecting drain holes between the levels.

Water is introduced to the Blackwater mine through groundwater seepage, previously mined areas and the use of drilling equipment. Dewatering is necessary to provide clear access to all working areas and to prevent accumulated water from causing corrosion or other damage to mobile and electrical equipment and other infrastructure.

The other significant potential source of water is the flooded historical workings immediately above the Project area. A dewatering programme will be implemented to remove the perched water by probe drilling from the access decline, and from the initial exploration drilling platform. Dewatering will occur in a controlled manner to ensure the safety of employees underground. Water from the historical workings will be treated as required to facilitate discharge to the environment – refer to Item 18.11.2 and Item 20.1.1

Dewatering will be achieved via pumps located throughout the mine, with water being delivered to a surface water storage dam. Design and costing of individual pumping networks has been completed for the PEA. Pump selection is based on operating head, required flow rates, pump type practicality, capital and operating cost.

16.7.6 Mine Services

Various services such as electricity, compressed air, water and communications are required to support underground operations. These services are reticulated throughout the mine or to specific areas in order to provide:

- Operation of electric fixed and mobile equipment such as drill rigs, pumps, lights and fans;
- Operation of pneumatic fixed and mobile equipment where on-board air compressors are not fitted;
- Dust suppression;
- Cleaning down of equipment and concreted area;
- Communication between personnel and equipment operators; and
- Conveying emergency messages to all personnel.

16.7.7 Ventilation

The decline portal to the uppermost level of the proposed Blackwater Underground has a twin decline system, with one decline acting as an intake and the other acting as the exhaust, and an exhaust fan installed on the surface providing through ventilation. Stockpiles between the declines serve to interconnect them, facilitating both access and through ventilation. As the declines advance and new interconnecting stockpiles are mined, the previous stockpiles are sealed to advance primary flow further down the decline. Primary ventilation will flow through the uppermost open stockpile, and secondary fans are installed above this open connection to push ducted auxiliary ventilation down to the working faces. The secondary fan installations are progressively moved down the declines as advance continues.

Figure 16-20 and Figure 16-21 illustrate the ventilation arrangement utilised whilst developing the twin declines, as modelled using VentSIM software. The blue arrow represents fresh air from the surface, whilst the red arrow represents return air. Modelling shows that sufficient flow can be achieved using 2 x 55kW secondary fans with 1,200mm vent duct.

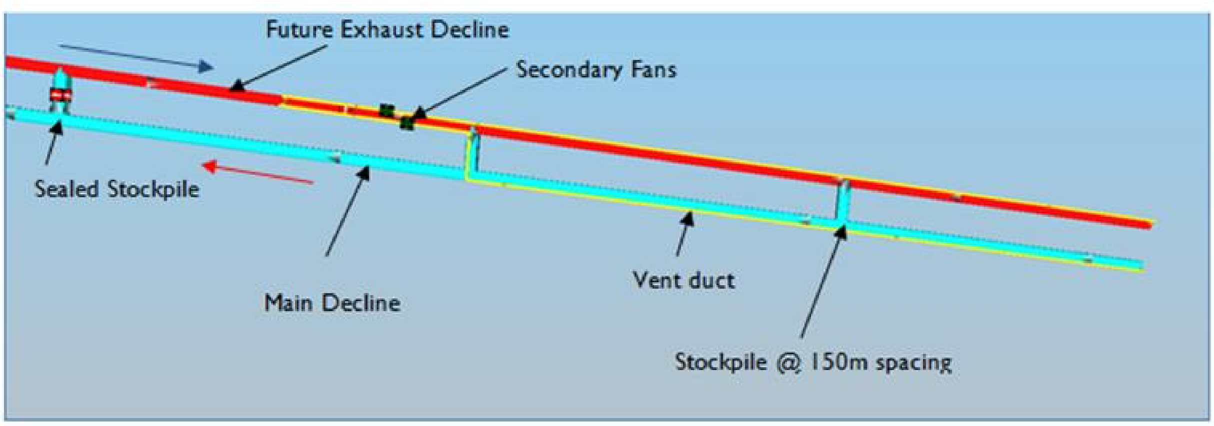

Figure 16-20: VentSIM Visual Model

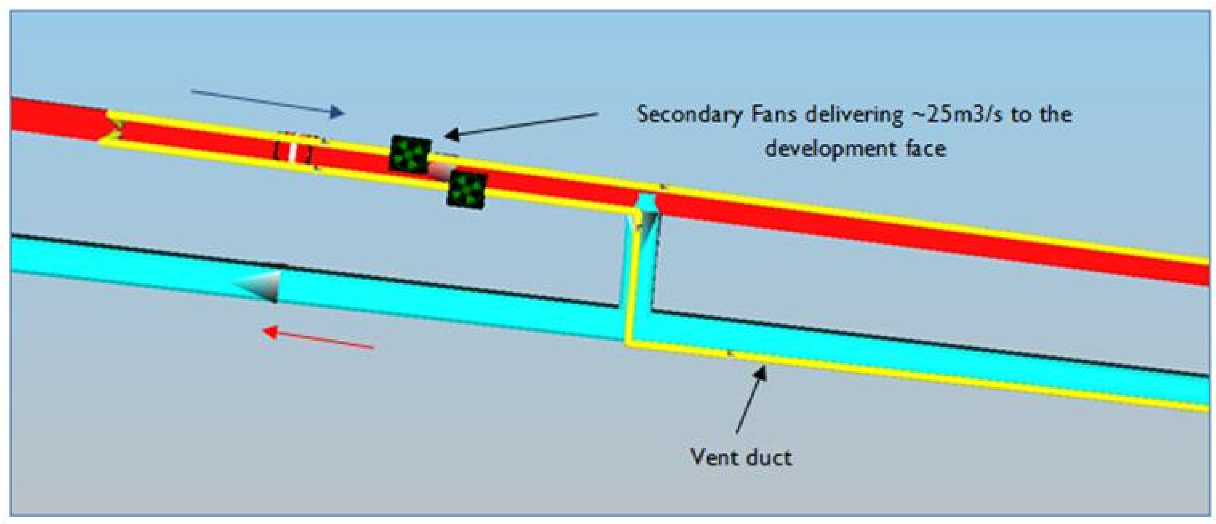

Figure 16-21: VentSIM Visual Model Detail

Once the top of the resource target has been accessed, from this point down a single spiral decline is proposed. Primary fresh air is provided from the decline and a series of long-hole rises between the loops of the decline are utilised for the return air. Secondary ventilation is used to provide fresh air to the production areas. This air is then re-used in the levels below and exhausted out the lowest return system (Figure 16-22).

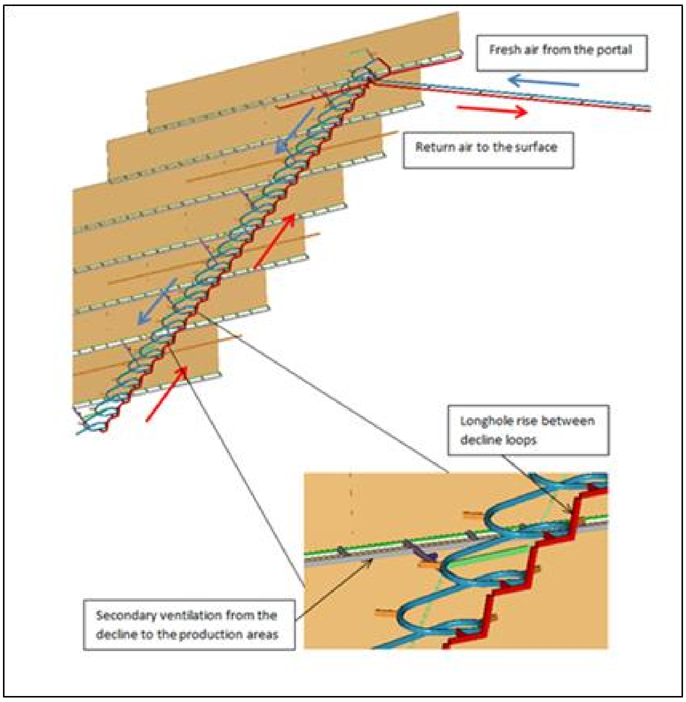

Figure 16-22: Ventilation Schematic

The decline will be extended on a “just-in-time” basis, such that the next production panel down-dip will be established just prior to the one above being completed. This allows sufficient time for resource definition drilling of the next panel down-dip prior to committing the capital expenditure required to advance the decline 100m lower to the next production panel. A production panel will be mined to completion once it has started, with the result that there will be a maximum of two production panels requiring access at any one time, and then only for a short duration whilst the upper panel is being completed and the lower panel is being established. This will result in minimal re-use of primary ventilation.

Within a production panel fresh air will be ducted to each ladder-way access for distribution into the individual work areas, with secondary ventilation fans located in the decline above the access crosscut to the mining panel. Orepasses not in use within the mining panel will act as exhaust raises. Exhaust ventilation will return via the hangingwall drive to the central access crosscut and then continue down the decline, before entering the primary exhaust raise network for discharge to the surface. Table 16-10 outlines the mine's primary airflow requirements, with the total mine airflow required being approximately 82m³/sec.

Table 16-10: Total mine airflow

Equipment Type	Model	Engine Rating	Utilisation	Qty	Air Flow Required
Jumbo	Atlas Copco Boomer 282	58	50%	1	1.5
Longhole Rig	Atlas T1 D Production Drill	75	5%	1	0.2
Truck	Sandvik TH320	240	80%	4	38.4
Loader	Atlas Copco ST1030	186	55%	1	5.1
Charge up	Normet Charmec	115	40%	1	2.3
IT Carrier	CAT IT28G	90	65%	1	2.9
Grader	CAT 12H Low Profile	104	50%	1	2.6
LV	Landcrusier	110	50%	10	27.5
Stores Truck	Isuzu NPS 300	113	20%	1	1.1
Total Air Flow Requirement (m3/sec)					**81.6**

Refrigeration may be required as mining progresses at depth. Limited information on thermal gradient was available to conduct an assessment of the requirement and it has not been included in the analysis. The contingency in the cost estimate is adequate to cover this cost, should the need arise.

16.7.8 Emergency Egress

An independent egress system between level accesses from the decline at 100m vertical spacing will be established. Each will require a 1.5m diameter raisebore, inclined at about 60˚ above horizontal, with each leg being ≈120m long (Figure 16-23).

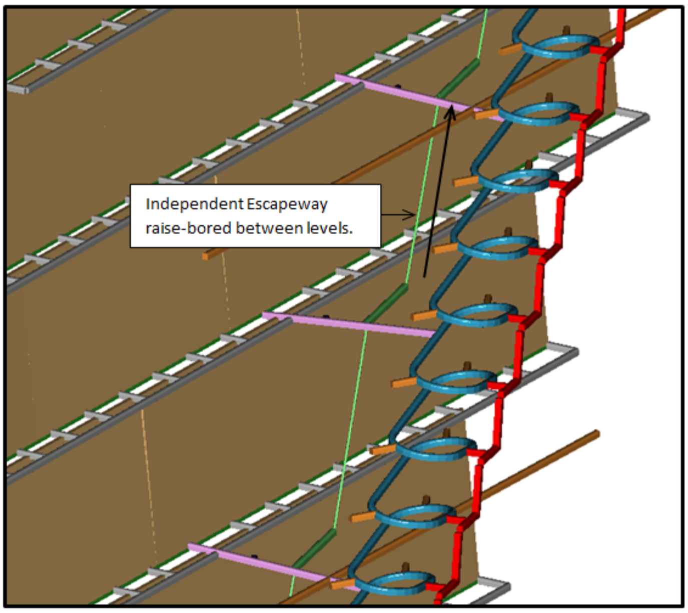

Figure 16-23: Escape Way Schematic

Installation of a SafEscape ladder way (Figure 16-24) has been included in cost estimates. The decline escape-way raises may be prone to stress-related damage as the mining front passes, causing damage to the ladder way. The proposed stand-off from the mining stopes must therefore be assessed and the decline egress ladder ways designed to minimise this risk.

16.7.9 Stope Access

Within the air-leg resue stoping work areas, a cost provision has been included for SafEscape ladder ways (Figure 16-24) at each end, enclosed within an ore pass liner, providing two means of egress from each work area.

Figure 16-24: SafEscape Ladder Way

The ladder ways would be installed prior to firing and tied off to the walls and backs as required to ensure rigidity that will stand up to a blast within close proximity. The blasting of the 2m lift would be timed in such a way as to “throw” the waste away from the ladder.

Following firing, access is achieved through the ladder way and some movement of blasted material may be required around the ladder way and liner to ensure adequate encapsulation and to establish a safe working platform prior to commencement of lateral waste flat back firing. The ladder way accesses are extended as each successive lift is extracted.

16.7.10 Ore Passes

Modular ore pass liners will be installed and extended as air-leg resue stoping continues upward within a work area (Figure 16-12). It is estimated that it would take one shift to extend the ore pass and pair of ladder ways for each stope work area.

16.7.11 Resource Drilling

Initial exploration will be from a dedicated exploration platform near the top of the production target (refer to Figure 14-7), while further diamond drill platforms will be established on every second level at intervals of approximately 200m vertically. The drill drives will be developed either side of the access, parallel to the hanging-wall drives for a length of approximately 600m (Figure 16-25).

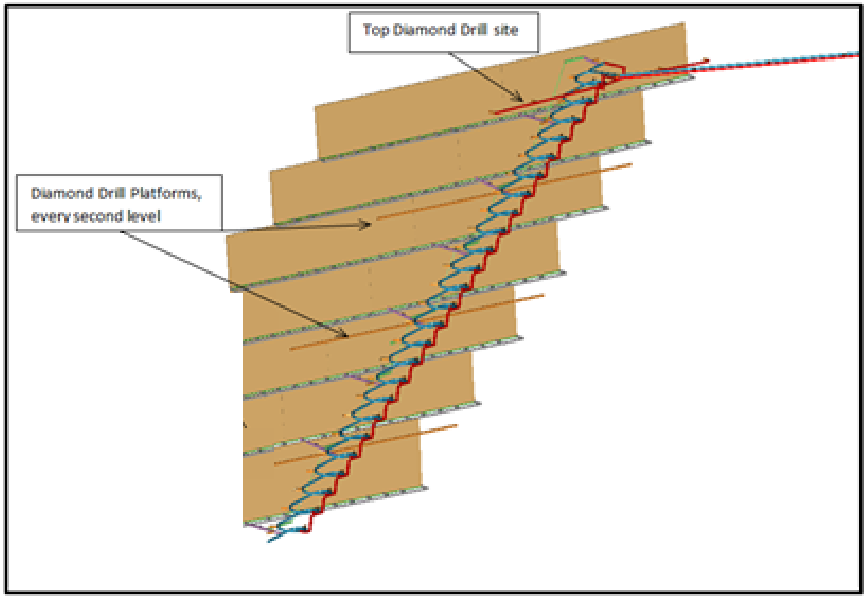

Figure 16-25: Exploration Drill Platforms

Upon completion of the access decline there will be a strong focus on rapid acquisition of data to enable commencement of mining operations. The decline will be stopped and the first exploration drill drive will be mined to the full extent before diamond drilling commences. Resource definition drill coverage at a density of 50m along strike by 25m up dip is planned. This coverage will be achieved in the uppermost stoping panel by 74 holes with a total length of 8km, and an average hole length of 107m (refer to Figure 14-7). Using two diamond drill rigs at a rate of 35m drilled per day, it will take under four months to completely drill the first stoping block, as indicated in Figure 16-8. This will provide OceanaGold with sufficient confidence to commit to mining operations.

As stated previously, geotechnical data will be collected from the planned underground resource drilling, to develop an improved understanding of the geotechnical conditions within and near the Birthday Reef.

16.8 Mine Planning

16.8.1 Production Sequence

The mine has been broken into seven production panels with nominal dimensions of up to 900m along strike by 100m up-dip (Figure 16-9). The production sequence by panel is top down, and within each panel is bottom up. Each production panel is broken along strike into work areas that are 60m long. Within a production panel there is currently no restriction on the number of work areas that can be simultaneously mined, and adjacent work areas need to be progressed upwards concurrently as they share a common access ladder way.

AMC noted that the mining sequence will form a series of sill pillars. These are progressively reduced in size as each stoping panel ‘closes’ on the extracted panel above. In this situation, the stresses will progressively increase as the sill pillars are reduced in size. It can be reasonably expected that ground conditions will become more difficulty to control over time as the pillars are reduced in size.

Squeezing ground conditions could be encountered, although this cannot be assessed with the current lack of information on the pre-mining stress field and rock material properties, including strength. Squeezing ground behaviour usually results in slower advance rates, higher support costs and delays due to rehabilitation.

The proposed mining depth (from 800m below surface to possibly as much as 1,600m) will require that stresses are investigated and that the mining sequence is developed to manage these as far as possible. It is possible that the overall sequence will need to be carefully managed to reduce rehabilitation and achieve acceptable levels of resource recovery, which could impact on the production schedule.

16.8.2 Development Sequence

Where possible, development is scheduled to be mined as late as possible and the hanging wall decline will be extended on a "just-in-time" basis to ensure that the next production panel is ready for extraction before the previous panel is completed, including a second means of egress, primary ventilation and materials handling development. The current schedule assumes that a production panel will be essentially complete before production commences on the next panel down dip. Once additional geotechnical information becomes available from underground diamond drilling this will be reassessed. The production sequence within a panel may need to be staggered 'en-echelon' to deal with the diminishing sill pillars discussed in Item 16.8.1, which may require an adjustment to the development schedule.

16.8.3 Mining Dilution

Initial on-ore development to establish each production panel (up to 900m long x 100m high) will involve mining an ore drive with dimensions of 2.8m (w) x 2.5m (h). It is planned that this will be mined by hand-held methods, but without resue firing.

Once the ore drive for a work area has been developed, air-leg resue stoping can commence. To maximise the ore head grade delivered to the mill, all stoping will be undertaken with resue firing, which involves separate firings (and material handling) of ore and waste material.

The average ore thickness is estimated in this study to be 0.68m, and it has been assumed that there will be planned dilution of 50% at zero grade, for a total ore resue mining width of 1m. Higher dilution may be encountered dependent upon stress encountered when firing the ore portion of the stope, and will be monitored as mining progresses.

All production holes for ore resue firing would be collared within the quartz reef rather than at the FW and HW contacts, so as to reduce the risk of excessive overbreak and dilution. Hole spacing and placement will be varied as experience is gained with the method, and the use of low impact explosives may be introduced as a further dilution control. This lower rate of dilution applies to all ore material mined up to the 5m sill pillars which are to be left in-situ below the level above.

16.8.4 Ore Recovery

Ore recovery has been determined to be 100% for on-ore development mining whilst establishing a level, due to there being no access restrictions and the ability to use a loader for material handling.

During air-leg resue stoping activities, ore material can be lost as a result of;

- remaining in-situ within the stope shape due to poor drill and blast techniques;
- material failing to rill down to the floor of the stope (unlikely given the steep dip of the ore);
- material left at the back of a stope where a scraper may not be able to retrieve it; and
- ore material being 'lost' within the waste working floor from the previous mining lift.

To mitigate the final point quick-setting cement slurry (a marker bed) will be laid down on top of the waste from the previous lift in order to provide an impermeable barrier, and to create a smooth running surface for the scraper hoe to run on. This will also reduce the risk of the scraper hoe or rope becoming caught on blasted material or rock bolts that could be present within the waste fill material.

A recovery factor of 95% has been applied to the stoping panels to account for the points above.

Overall recovery of the in-situ resource within each 100m high panel is calculated as follows:

- Sill pillar: 5m high @ 0% recovery;
- Initial ore drive: 2.5m high @ 100% recovery;
- Air-leg resue stoping: 92.5m high @ 95% recovery;
- Overall mined recovery: [(5mx0%) + (2.5mx100%) + (92.5mx95%)]/100m = 90%.

Recovery of the in-situ resource is calculated at 90%, with 5% remaining unrecovered in pillars, and the balance lost during stoping operations.

16.8.5 Cycle Time

Cycle time calculations were undertaken to determine appropriate productivity rates from stoping activities. Table 16-11 details the total cycle time for extraction of a single air-leg resue lift.

Table 16-11: Air-Leg Resue Stoping, Lift Cycle Time

Activity (in sequence)	Duration (shifts)
Drilling ore – half uppers	2.1
Drilling waste half uppers (above ladderways only)	0.5
Install laddertubes and ore pass cans	1.0
Blast ore & waste above ladderways	1.3
Scrape ore	5.4
Establish working platform	2.0
Waste flat-backing	10.2
Place slurry marker bed	1.0
TOTAL CYCLE TIME FOR A 2m LIFT	**23.5**

16.8.6 Productivity Rates

Single work area productivity rate achieved using air-leg resue stoping method is calculated as:

$$\left(\frac{324\ ore\ tonnes}{23.5\ shifts}\right) = 13.8\ ore\ tonnes/shift$$

Mining cycle efficiencies can be improved by having stoping work areas at the maximum length deemed geotechnically stable. Multiple work areas may operate simultaneously along strike for improved production efficiencies that come with additional work areas. Adjacent work areas must progress upwards concurrently to the next lift, due to the methodology of extending the access ways.

16.9 Production[6]

16.9.1 Production Schedule

The air-leg resue mining method utilises a twin decline design to access the initial exploration drilling platform, required to better define the resource prior to the production phase. Figure 16-26 illustrates the tonnes and grade profile, including the two and a half years pre-production for construction of the access decline and associated infrastructure, plus resource definition drilling. The waste shown does not include the waste left in the stopes as fill.

[6] The scheduled production volumes and grade discussed in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

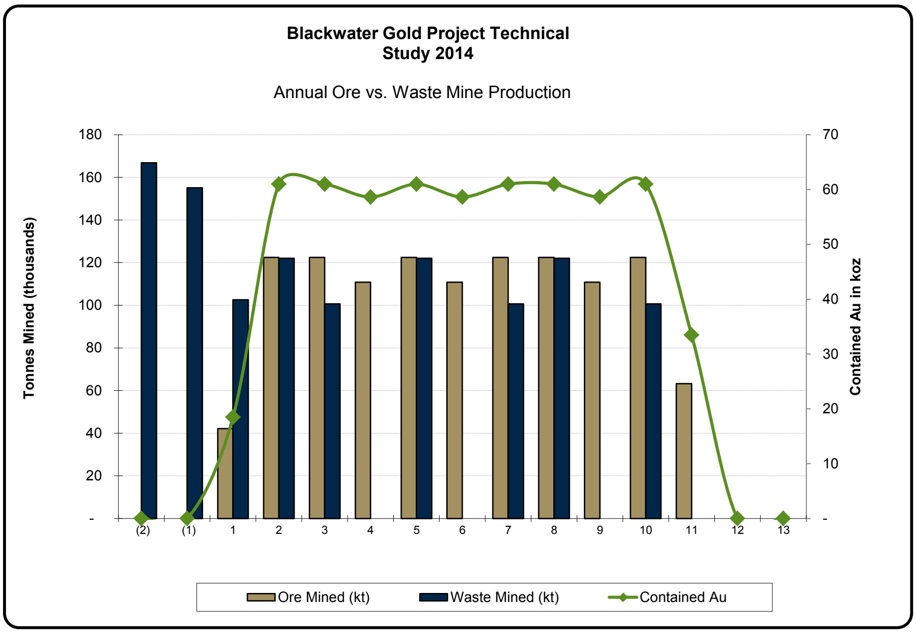

Figure 16-26: Life-of-Mine Ore, Waste Tonnes and Ounces Mined Profile

Figure 16-27 shows mine production and mill throughput by year. It has been assumed that all material is processed in the year that it is mined, with no stockpiles remaining at year end.

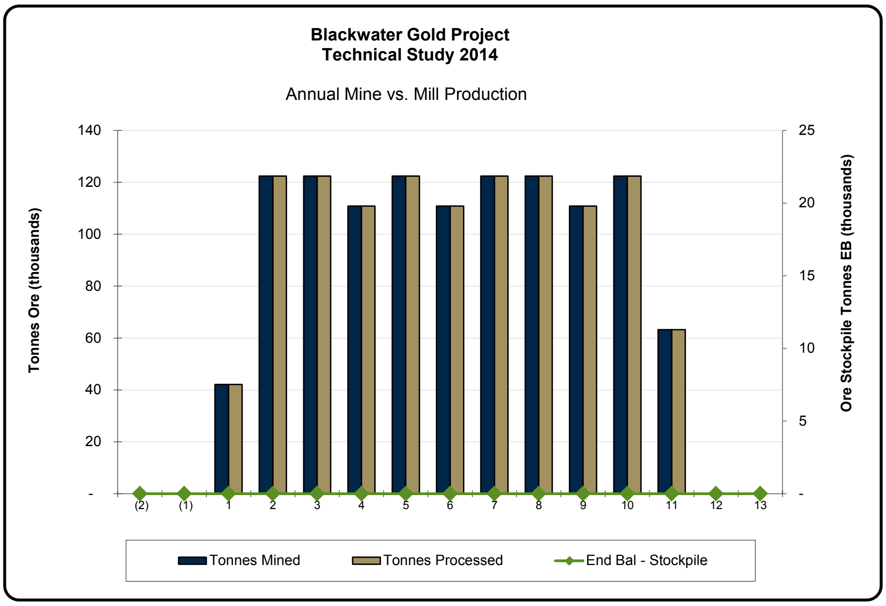

Figure 16-27: Mine and Mill Production

Due to the nature of the mining method and the number of personnel required during the production phase, the production profile is ramped up almost immediately. The production profile for the majority of the mine life alternates between 111-122kt per annum depending on the mine sequence. The final year of mining shows a reduced amount of tonnes and metal being produced by the mine, due to the final production panel being extracted.

16.9.2 Production Grade

Head grade throughout the mine life is expected to be in the order of 15g/t Au. The development carries more dilution and hence a reduced head grade, however the ounce profile will potentially increase during this period, as shown in Figure 16-27. The head grade and mined (diluted) grade are regarded as the same in this study.

Table 16-12: Air-Leg Resue Mineable Inventory

Description	Value	Units
LoM Ore Milled	1,172	kt
Max Daily Feed	335	tpd ore
Grade:	15.8	g/t
Contained Ounces:	594	koz
Processing Recovery:	96.0	%
Recovered Ounces:	570	koz

Table 16-12 shows the ounces by recovery category. The ounces not milled are residual in the sill pillars in each production panel or unrecovered during the air-leg resue stoping process.

16.9.3 Mobile Mining Equipment

The type of equipment to be utilised at Blackwater will be standard in order to achieve the following:

- Minimise spare part stock holding by volume and maximise holding by value;
- Allow for specific maintenance training on equipment that will facilitate flexible workforce rotation throughout the operations; and
- Improve machine availability for operations.

The quantity of fleet suggested for the project is provisional and aimed at providing indicative mining costs for the base case mining method. OceanaGold acknowledges that given the status of the mineral resource classification there is a high likelihood that fleet requirements could change to those stated in this PEA.

16.9.3.1 Loaders and Trucks

The suggested loader is the Atlas Copco ST1030 or similar. The ST1030 has a bucket capacity of 5.0m³ with a payload of 10t. The loader's height and width is suited to the proposed development dimensions. The ST1030 will be fitted with teleremote capabilities to be used in the event that alternative mining methods are employed, as discussed previously.

The suggested haul truck type is the Sandvik TH320. The TH320 has a payload capacity of 20t and is designed for high production, low cost-per-tonne hauling.

A single loader is calculated to be required throughout the mine life, with a second loader included in costing to ensure equipment availability and continuity of production in the event of a major breakdown. The number of trucks required increases with the depth of the mine, to a maximum of four. During the early stages of the project when it is calculated that only one truck is required, two trucks have been included in costings to ensure equipment availability and continuity of production in the event of a major breakdown.

16.9.3.2 Drill Rigs

A development jumbo drill will be used for both face drilling and rock bolting when excavating the exploration decline and capital development. Due to the rapid rate of development required, the jumbo will be equipped to drill the longest rounds practical, while still being able to install ground support in the drifts.

The recommended jumbo is the Atlas Copco Boomer 282, which is capable of both face drilling and rock bolting. A single development jumbo will have adequate capacity to complete all required development and ground support, based on a capacity of 20,000 drill metres per month.

A production long-hole drill rig will be required for drilling raises and for installation of cable-bolts. The rig will be underutilised due to all production stoping being undertaken by hand-held methods, but will still be required. The recommended production rig is the Atlas Copco Boomer T1D Long-hole drill, whose height and width is suited to the proposed drive dimensions.

Drilling within the stopes will be undertaken using rock drills and air-legs, which are low capital cost items. Provision has been made for spare rock drills and air-legs for each miner to ensure production continuity during maintenance activities.

16.9.3.3 Charge up

A charge up vehicle will deliver blasting products to the development face, and will incorporate a telescopic boom which allows them to reach any development face.

The Normet Charmec MC 605 is suggested for development face charging. The boom lift and platform are designed to enable reach up to 8.4m, and the rig's height width is suited to the proposed drive dimensions. A single Charmec unit should be sufficient to cater for the Life of Mine requirements.

Charging operations within the stopes will be by hand-held ANFO loaders, utilising compressed air venturi loaders to blow-load ANFO.

16.9.3.4 Grader

A single grader will be required for underground decline maintenance. The suggested grader for the Blackwater project is a CAT 12H Motor Grader or similar, which is suitable for the development dimensions proposed.

16.9.3.5 Integrated Tool Carriers (IT)

An IT will be required to complete service work underground such as:

- Installing and repairing vent bag;
- Installing mine services;
- Prep development faces prior to boring; and
- General mine service work.

The recommended IT is the CAT IT28G, which is suitable for the development dimensions proposed. A single IT will be adequate for the life of mine requirements.

16.9.3.6 Light Vehicles (LV)

Light vehicles will be used to transport operators to the underground workings and also inspect areas when required. Initially, only five LVs will be required during the excavation of the exploration decline, increasing to thirteen once ore production starts. There will be an additional requirement for LVs for surface use, including the process plant.

16.9.3.7 Stores Truck

Stores trucks transport equipment to the underground store such as:

- Vent bag;
- Mine services – hoses, poly pipe etc.;
- Explosives;
- Mesh; and
- Bolts.

The recommended stores truck is the Isuzu NPS 300, which has a transporting capacity of 4.5t and is fitted with 4WD capabilities. A stores truck will only be required once ore production begins, and a single unit will be adequate for life of mine requirements.

16.10 Manpower

16.10.1 Operational

The underground roster is based on two 12 hour shifts per day. The work cycle is based on a three panel roster which supports a roster that equates to two thirds working time. At any one time there will be one crew on dayshift, one crew on night shift and one crew on break.

When calculating manning requirements for the Blackwater project, the following assumptions were applied:

- One supervisor per crew;
- One operator per drill rig (Jumbo);
- One operator per loader;
- One operator per charge up rig;
- One operator per truck;
- Eleven air-leg miners per crew;
- Three scraper operators per crew;
- Three service crew per IT;
- One operator per grader; and
- One fitter per three heavy vehicles (truck, loader, jumbo, long-hole).

These numbers are per crew and hence multiplied by three (excluding maintenance personnel) to calculate the total manning required for the project.

During the initial decline development, there is opportunity for role sharing. During the pre-production period, three operators will utilise the single jumbo during the decline development stage. These operators will be multi-skilled and also operate as loader and charge-up operators. The service crew or a competent operator will be used on the long-hole drill rig to drill cable bolts as required.

Three loader operators will be required throughout the mine life. These operators will also assist with jumbo and charge-up requirements. Three truck drivers will be required for the pre-production period, ramping up to nine operators once production commences and increasing to twelve as the mine deepens.

Three charge-up operators will be required for the life of mine, and three grader operators will be required from the commencement of production to maintain the main haulage roads to the surface.

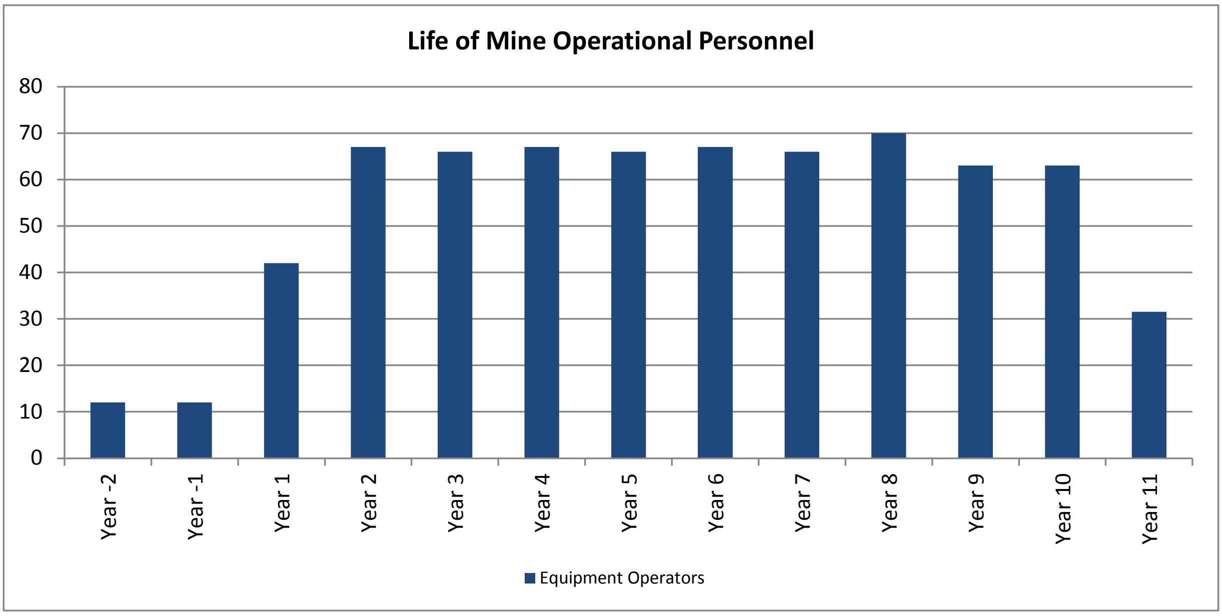

Figure 16-28: Operational Personnel by Year

16.10.2 Management and Technical

Management and technical personnel will be based on a residential style roster working five days on, two days off. They include underground management, engineers, surveyors, geologists, production and maintenance supervisors. There is an increase in technical personnel after the exploration decline access has been completed, resource definition drilling is undertaken and production commences. It should be noted that these labour numbers exclude personnel required in the process plant, and also exclude the diamond drilling contractors that would be employed throughout the mine life for resource definition and grade control drilling.

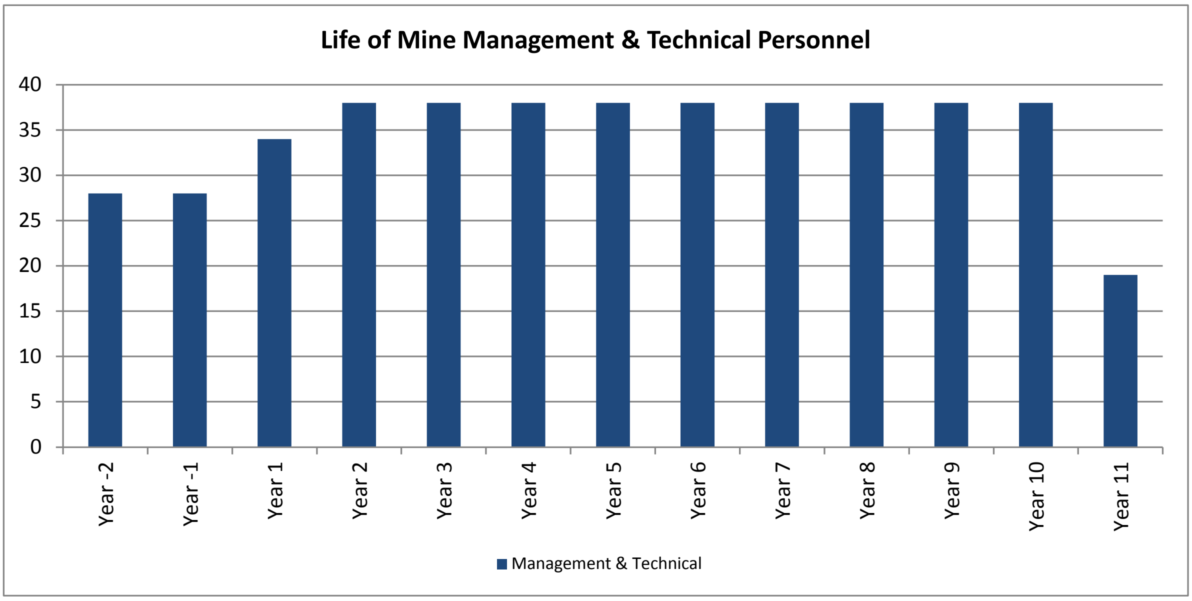

Figure 16-29: Management and Technical Personnel by Year

16.11 Risks and Opportunities

The primary risks are associated with ore-body variability in grade, width and payability. As detailed in Item 16.1, an allowance has been made to account for the localised areas of reduced thickness and/or grade which would prove uneconomic to mine. Production schedules generated have been prepared on the basis that 15% of the reef will prove to be uneconomic, due to a deficit in contained metal. The current lack of geology and geotechnical data precludes the ability to calculate more detailed modifying factors for the calculation of potential mining inventory. In addition, a sensitivity analysis was undertaken whereby the reef thickness and reef grade were flexed by ±15% and ±30%, to determine the impact on the Project NPV. The results are presented in Item 22.5.

Rock-fall hazards will exist. Trial mining will improve understanding of practicalities and risks associated with the mining method. The inclusion of a slurry marker bed will provide improve safe working practices associated with the scraper.

Where wall conditions are poor, significant fall-off can be expected during and after firing the ore from the de-stressed hanging wall and footwall of the 'slot', resulting in higher dilution. The actual outcome will depend on the amount of structure in the stope walls and its orientation, the condition of the joints and the effectiveness of the back support. If areas of particularly poor ground conditions are identified through resource definition drilling, the mining method employed will be changed to suit. Similarly, if improved ground conditions are encountered, a more cost-effective mining method would be utilised.

If the fall-off/overbreak occurs as suggested, this material will be removed with the ore thereby increasing dilution and potentially creating a shortfall of mullock to form the working platform. If this occurs it can be counteracted by varying the width of waste blasted to ensure that adequate waste is generated.

Mining at depths between 800m and 1,600m below surface is likely to be challenging. Considering the rockmass conditions and the expected stresses, adverse ground behaviour including high deformation or 'squeezing' behaviour should be expected. The mining sequence will need to be carefully considered to avoid exacerbating the stress conditions. Further studies including assessment of the extraction sequence (which currently involves closure on a series of sill pillars) is recommended for the next phase of study. This should include numerical modelling with an appropriate stress analysis program.

17 RECOVERY METHODS

Gekko Systems Pty Ltd (Gekko) was contracted by OceanaGold (New Zealand) Limited to carry out a Technical Study to assess the suitability of using a Surface Python Plant and Concentrate Treatment Plant (CTP), to produce smelted gold at their Blackwater site in New Zealand. This study is largely based on test work completed in February 2011 at Gekko's facilities in Ballarat.

The Gekko Python modular plant was utilised as the basis for the flowsheet development with the sizing adjusted based on the ore production rate of the mining method. The mining method investigated in the study is a hand held method producing at a nominal rate of 110ktpa at 16g/t Au (refer to Item 16).

The ROM ore is crushed to P_{80} of 2.4mm using a jaw crusher and Vertical Shaft Impactor (VSI) and is combined with ball mill discharge and fed to the primary gravity circuit (InLine Pressure Jigs).

The gravity concentrate collected is sent to final concentrate and the gravity tail is fed to cyclones. The cyclone underflow is fed to a ball mill where it is ground to P_{80} of 150µm. Additional water is added to make slurry at 36% solids before being fed to the flotation circuit to recover a sulphide concentrate.

The flotation tails are thickened then dewatered in a belt filter which is the battery limit for Client collection and disposal. Thickener overflow, filtrate and wash water report to the process water tank and then feed the process water system.

Design includes reagent preparation and distribution units for the collector (PAX), activator (copper sulphate), frother, cyanide, oxygen, sodium metabisulphite (SMBS), lead nitrate, sodium hydroxide, hydrochloric acid and sodium benzoate.

Combined flotation and gravity concentrate is transferred to a regrind mill in closed circuit with a DSM screen. The screen top size (+106µm) reports back to the regrind mill. The underflow (-106µm fraction) reports to the continuous InLine Leach Reactor (ILR) where cyanide is added to leach gold.

The ILR discharge flows to a plate and frame filter which carries out the solid/liquid separation of the pregnant leach solution and washing of the solids. The pregnant leach and wash solution reports to the G-REX resin circuit.

The leached filter solids are re-slurried with barren solution and process water. These are pumped to the detox circuit where cyanide is destroyed for safe final handling of the final tail residue. The gold in the pregnant leach solution is adsorbed onto resin in the G-REX column.

The barren solution from the column is recycled to the regrind mill and filter. The loaded resin is removed from the column and stripped with the gold recovered by electrowinning and smelting. The stripped resin is transferred back to the resin column or periodically acid washed in hydrochloric acid to remove impurities before being transferred to the barren resin holding tank and finally returned to the resin column.

The acid wash solution is neutralised with sodium hydroxide, then transferred to the barren process solution tank.

The capital cost of the supplied plant is **USD$20.5M ± 25%** for a 25tph P200 modular plant. This excludes further engineering and shipping of the supplied plant, and earthworks at the site.

The total processing plant operating cost at full production rate (Surface Python plus CTP) is expected to be USD$42 per tonne of mill feed, and be in the order of USD$4.6M per annum.

17.1 Introduction

Initial test work completed on the Blackwater project in 2011 was carried out on a blend of Quartz and Greywacke ore types (refer to Item 13.4.3). The aim was to determine if the ore was suitable for processing using a Gekko Python flowsheet, which incorporates a combination of gravity concentration and flotation to recover gold.

The testing involved thorough blending and preparation of the samples, progressive grind gravity test (PGT), combined gravity/Falcon testing and combined gravity/flotation tests. Results demonstrated that tabling of the feed was able to recover 81% of the gold into 1.4% of the mass at a P_{80} grind size of minus 450µm.

It was found a further 17% of the gold could be recovered by grinding the gravity tails to a P_{80} of 106µm to give a total gold recovery of 98% into approximately 4.5% of the mass. The calculated feed grade was between 5.6 g/t Au and 6.2 g/t Au and the combined gravity and flotation concentrate grade was 127 g/t Au giving an upgrade ratio of 20.

OceanaGold informed Gekko that the metallurgical results from the 2011 study were adequate to carry out the current technical study. Further laboratory test work conducted in June 2013 was to enable the recommendations from the initial laboratory study to be verified and to obtain leaching data that could be used to design the Concentrate Treatment Plant (CTP).

Due to delay with test results, the design of the CTP was based on assumptions applied from a similar processing plant which was designed and built by Gekko. These assumptions included using a six hour leach residence time, a 99% leach recovery (based on OceanaGold's internal test work results) and a similar concentrate throughput and gold content to the Ballarat Goldfields processing plant for design of the G-REX resin circuit and electrowinning circuit.

The June 2013 laboratory test work program used a combination of two quartz samples and Greywacke to create synthetic feed. The samples were thoroughly combined and prepared for a 3 stage GRG (gravity recoverable gold) test as well as continuous gravity testing with flotation of the gravity tails at selected grind sizes.

Three gravity and flotation tests have been carried out at minus 600, 425 and 300µm.

Further flotation tests have been carried out after grinding and gravity concentration at minus 212µm. The gravity tail was subjected to further grinding to look at flotation response at minus 212, 150 and 106µm.

The bulk concentrate from the minus 600, 425, 300 & 212µm size fractions was subjected to intensive cyanidation tests to determine leach kinetics.

17.2 Process Flowsheet

17.2.1 Surface Python

A concept layout for the Surface Python can be seen at Figure 17-1.

The ROM ore at a P_{100} of 250mm is withdrawn from the ore stockpile at a rate of approximately 25tph by a front end loader and discharged into the Surface Python feed bin. A variable speed chain conveyor discharges the ore onto the primary crushing feed conveyor. A belt magnet located above the conveyor belt removes tramp metal.

A weightomete located on the conveyor belt records feed to the Surface Python and is used to control the chain conveyor's speed to the required feed rate.

The primary crushing feed conveyor discharges onto a vibrating grizzly which scalps the ore at 50mm. The undersize is discharged onto the jaw crusher discharge conveyor, whilst the oversize passes into a jaw crusher, reducing the particle size from 250 mm to approximately 35 mm.

The jaw crusher is in closed circuit with a primary screen with 45mm aperture which returns the oversize back to the crusher for further size reduction.

An automatic belt sampler is installed on the Primary Screen undersize straight after it discharges onto the primary screen undersize conveyor. This ensures that a more representative feed sample is taken for gold assay before the material enters the concentrator, which otherwise may be problematic with standard manual sampling techniques of coarse gold ores. The primary screen undersize conveyor discharges onto the secondary screen feed conveyor.

The secondary screen wet screens product from the crushing area and Cleaner InLine Pressure Jig Tails at nominally 3mm, aiming to give a sizing with a cut point at $P_{80\ of}$ 2.4mm. Plus 3mm material at an estimated rate of 63tph for a 250% recirculating load is returned to a Vertical Shaft Impactor (VSI) where it is crushed and discharges onto a vibrating feeder which transfers it to the primary screen undersize conveyor. This stream transfers to the Secondary Screen Feed Conveyor where it joins with the Primary Screen Undersize as feed to the Secondary Screen.

The screen undersize at 34tph (P_{80} of 2.4mm) is pumped to the Ball Mill discharge hopper via a duty/standby pump. An automatic density sampler will record the slurry density and provide a sample. The Ball Mill discharge pump pumps the secondary screen undersize and ball mill discharge streams to the rougher InLine Pressure Jig (IPJ).

The rougher IPJ concentrate (gold and/or other heavy minerals) at approximately 7.5tph is pumped to a cleaner IPJ for cleaning. The tailings from the rougher IPJ is pumped to the grinding circuit cyclones.

The cleaner IPJ tailings at approximately 7.1tph flow under pressure to the secondary screen feed box.

17.2.2 Grinding and Flotation

In Line Pressure Jig cleaner concentrate at 0.7tph and 25% solids reports to the Gravity Concentrate Sump. This stream is pumped to the Concentrate Filter Feed Hopper and then to the Concentrate Filter. The concentrate is filtered to approximately 89% solids and is transferred by front end loader to either a concentrate stockpile or directly to the CTP feed hopper. The filtrate is pumped to the Process Water Tank for re-use in the Surface Python.

The grinding circuit cyclone generates a Cyclone Overflow with a P_{80} of 150µm, which flows to the Flotation Surge Tank. The Cyclone Underflow is gravity fed to the Ball Mill for regrinding. The Ball Mill grinds the underflow and discharges into the Ball Mill discharge hopper for pumping to the gravity circuit.

The Flotation Surge Tank gives the process approximately 2 hours surge capacity between the crushing/grinding/IPJ circuits and the flotation/filtration circuits. The slurry is pumped from the surge tank to the Flotation Conditioning Tank. The slurry overflows from the Conditioning Tank to a bank of rougher Flotation Cells.

Copper sulphate activator is added to the Cyclone Overflow. Potassium Amyl Xanthate (PAX) collector is added to the Flotation Conditioning Tank and frother is added to the feed box of the Flotation Cells. The Flotation Cells provide approximately 13 minutes residence time. An air blower provides the flotation air to the float cells with the air flow controlled by positioning valves on the blower discharge lines.

The flotation concentrate, containing 17% of the total gold to the Surface Python plant, in 3.1% of the weight, flows by gravity to the flotation concentrate sump. The concentrate is pumped through a Gekko Density Sampler, where the slurry density is periodically measured and sampled, to the Concentrate Filter feed Tank. The flotation tailing is pumped through a Gekko Density Sampler which measures the density and takes samples of the flotation tail. A flowmeter records the flowrate. This is then pumped to the Tailings Thickener where the underflow is thickened to approximately 56% solids.

The thickener underflow is pumped to the tailings filter for further dewatering to 86% solids, enabling the filtered tail to be reclaimed and handled by mobile equipment.

Flocculant from the flocculant mixing plant is added at the feed well to the tailings thickener to increase settling rates whilst maintaining a high underflow density. The expected underflow density is 56% solids though higher densities are possible, and which will reduce the amount of water reporting to the tailings filter and hence will reduce the overall water loss and work load of the filter. The tailings thickener and tailings filter overflows are pumped to the Process Water Tank.

The concentrate is processed in batches through the Concentrate Filter and is transferred to the CTP by front end loader for further treatment.

Recycle water from the Process Water Tank is pumped to the secondary screen, IPJ's and water addition points by the Process Water pump. The pump speed is automatically adjusted to provide a set pressure at the discharge of the pump.

Potable water is fed to the Gland Water Tank and a high pressure pump provides gland water at the required pressure for the system to the various slurry pumps. At each pump that requires gland water, an automatic isolation valve and variable orifice flow control valve (Maric valve) is used to provide the required gland water flow per pump.

The flotation reagent preparation and distribution system are as follows:

Collector

- The PAX collector is delivered to the plant in 1 tonne bulk bags, split opened inside a dust box by a bag breaker and discharged into the mixing tank. The tank has a closed top, sloped bottom and is fitted with an exhaust system to avoid poisonous gas build-up;
- The PAX is made up to a 20% w/w solution and is mixed by agitator over a set time period and transferred by pump to the Collector Holding Tank. The holding tank also has a closed top and sloped bottom and is fitted with an exhaust system; and
- Collector solution is dispensed from the holding tank via the collector dosing pump to the Flotation Conditioning Tank at the required process dosage.

Activator

- Copper sulphate activator is delivered to the plant in 25 kg bags and mixed with water using an agitator to a solution concentration of 20% w/w;
- The copper sulphate solution is transferred to the Activator Holding Tank by the activator transfer pump and dispensed to the cyclone overflow by the activator dosing pump at the required process dosage; and
- The copper sulphate solution is also pumped to the detox holding tank in the CTP for use in the cyanide destruction circuit.

Frother

- Frother is delivered to the plant in IBCs and is transferred to the Frother Holding Tank. It is dosed neat from this tank to the feed box of the Rougher Flotation Cells via a dosing pump at the required process dosage.

There are sump pumps in the bund area for each reagent to transfer spillages to the final tails sump in the flotation area.

Process control is achieved using an Allen Bradley PLC with SCADA interface. All motor starters and electrics are housed in the two or three standalone, air-conditioned, containerised Motor Control Centres.

External inputs to the Surface Python plant other than those mentioned above include:

- Potable water at approximately 10 m^3/h to the gland water tank;
- Mine/raw water is added upstream of a control valve located on the Process Water Tank which adds water to the tank to control it to the required level; and
- Potable water is required for the safety showers located around the plant.

Instrument air is produced by a reciprocating air compressor located in the Surface Python area. Process air is produced by a screw compressor located in the filter area. All tanks and hoppers include level sensors unless designed to overflow. All major flows are recorded by the SCADA system. All required pumps are variable speed controlled.

17.2.3 Concentrate Treatment Plant (CTP)

A concept layout for the Concentrate Treatment Plant can be seen at Figure 17-2.

The Concentrate Treatment Plant (CTP) processes the gravity and flotation concentrates produced by the Surface Python via:

- Regrinding the combined gravity and flotation concentrate;
- Leaching of the combined concentrate in an InLine Leach Reactor;
- Filtering and washing of the leach tail to recover gold in solution;
- Gold adsorption and stripping using Gekko Resin, followed by electrowinning and smelting; and
- Cyanide destruction of the plant residue.

17.2.4 Concentrate Grinding

Grinding will be carried out in a small 37kW ball mill (GRM 1500). The mill will be positioned on its own concrete pad adjacent to the ILR's. The combined gravity and flotation concentrate will be loaded into a feed hopper and discharged onto a reclaim conveyor which feeds into the mill. Barren solution from the Barren Solution Tank is added to the mill feed to make up a 63% solids slurry. Ground concentrate from the mill discharge is pumped to a DSM screen (300µm aperture) in closed circuit with the mill. The screen oversize flows by gravity back to the mill for regrinding. The screen undersize flows by gravity to a Gekko continuous ILR5000. An on-stream cyanide analyser measures the amount of cyanide in solution at the mill discharge sump and also at the discharge of the ILR.

17.2.5 Intensive Concentrate Leaching (ILR)

Solution from the Barren Solution Tank is pumped to the ILR feed chute for concentrate leaching. All the process chemicals required for leaching are added to this Barren Solution Tank. The resulting mixture continuously enters the ILR drums where it is tumbled at low speed for approximately 8 hours to leach the gold from the concentrate.

The process chemicals added to the Barren Solution Tank include 0.5% sodium cyanide and sodium hydroxide. Oxygen is added into the feed pipe of the ILR. Most of these reagents are pumped in from dosing pumps located in the Reagents area of the Surface Python plant.

Slurry overflows from the ILR drum and is then pumped into the Resin Filter Surge Tank from where it is pumped to the Resin Filter. The Resin Filter Surge Tank is designed to hold enough feed to the filter for approximately four hours. If the filter is down, slurry can be recirculated back to the surge tank using the filter feed pump. Filter cake from the Resin Filter discharges into the Detox Circuit re-pulping tank and the filtrate and wash solution flows to the Pregnant Leach Solution Tank prior to entering the Resin Circuit (G-REX).

17.2.6 Resin Adsorption, Stripping and Washing

Solution from the Pregnant Leach Solution Tank is pumped to the Resin Column where gold is recovered onto AuRIX resin. The resin is moved up through the bed of the column from bottom to top, counter-current to the flow of solution. Twice a day, one bed's volume of resin is withdrawn from the column and is passed over the Resin Dewatering Screen and into the strip vessel.

A batch of stripping solution (1.0M NaOH and 0.5M Sodium Benzoate solution) at 60°C from the Strip Solution Tank is pumped through the loaded resin, through the Electrowinning Cell and back into the Strip Solution Tank. At the completion of the strip, the solution is retained in the Strip Solution Tank and the resin is transferred to the Barren Resin Holding Tank. From there it is transferred back into the column once the next strip is ready to begin. Once a week, the resin is transferred from the Barren Resin Holding Tank to the Acid Wash Tank for washing to remove calcium salts and other acid soluble contaminants. It is then transferred back to the Barren Resin Holding Tank and into the column. Once the resin has been acid washed, the solution is neutralised with sodium hydroxide and pumped to the Detox Tank.

17.2.7 Electrowinning

The resin strip solution is pumped through the electrowinning cell. The current is passed through the solution in the electrowinning cell and the gold is deposited onto stainless steel cathodes. Solid gold is washed off the stainless steel cathodes, filtered, dried and then smelted and poured into doré bars.

17.2.8 Detox

In the re-pulp tank, the filter cake is mixed with excess barren solution and process water and agitated to make a 40% solids slurry. This slurry is pumped to the Detox Tank where copper sulphate, sodium metabisulphite (SMBS) and air are added in stoichiometric quantities to react and neutralise the Weak Acid Dissociable (WAD) cyanide in solution.

An on-stream cyanide analyser measures the WAD cyanide in solution in the re-pulp tank and also in the detox tank. The analyser is able to detect changes in the WAD cyanide concentration and control the level of detoxification via SMBS addition to the tanks. The detox slurry is then pumped to the detox tail facility for disposal underground as cemented marker beds between stope lifts or in disused drives.

The CTP reagent preparation and distribution system are as follows:

Sodium Cyanide

- The sodium cyanide is delivered to the plant in 1 tonne bulk bags, split opened inside a dust box by a bag breaker and discharged into the mixing tank. The tank has a closed top and agitator;
- The sodium cyanide is made up to a ~25% w/w solution and is mixed by agitator over a set time period and transferred by pump to the Cyanide Holding Tank; and
- Cyanide solution is dispensed from the holding tank via the cyanide dosing pump to the Barren Solution Tank at the required process dosage.

SMBS

- The SMBS is delivered to the plant in 1 tonne bulk bags, split opened inside a dust box by a bag breaker and discharged into the mixing tank. The tank has a closed top and agitator;
- The SMBS is made up to a ~10% w/w solution and is mixed by agitator over a set time period and transferred by pump to the SMBS Holding Tank; and
- SMBS solution is dispensed from the holding tank via the SMBS dosing pump to the Detox Tank at the required process dosage.

Sodium Hydroxide

- Sodium Hydroxide is delivered to the plant as 50% solution in IBCs and is transferred to the Sodium Hydroxide Holding Tank. It is dosed neat from this tank to the Barren Solution Tank, Resin Strip solution Tank, Detox Tank and the Acid Wash Tank via a ring main pump at the required process dosage.

Hydrochloric Acid

- Hydrochloric acid is delivered to the plant in IBCs and is dosed straight from the IBC to the Resin Acid Wash tank via a dedicated dosing pump at the required process dosage.

Sodium Benzoate

- Sodium benzoate is a non-hazardous chemical used to catalyse the stripping of gold from AuRIX resin. It is delivered to the plant in 25 kg bags and mixed with water using an agitator to a solution concentration of 20% w/w.
- The sodium benzoate solution is transferred to the Resin Strip Solution Tank by the benzoate transfer pump when required.

Oxygen

- Oxygen gas is produced by a packaged Vapour Swing Adsorption system and is dosed into the ILR feed pipes at a rate controlled by the dissolved oxygen metre at the discharge of the ILR.

17.2.9 Tailings Disposal

Tailings from both the flotation tailings thickener and detox circuits will be filtered to produce a cake with less than 14% moisture content. Filtered tailings will then be available to rehandle either for backfill underground or to be transported to the waste rock dump for co-disposal with coarse mine waste. Tailings from the detox circuit will be preferentially placed underground as cemented marker beds in the stopes or in abandoned hangingwall drives to eliminate risks associated with potential Acid and Metalliferous Drainage (AMD) generation in the waste rock dump.

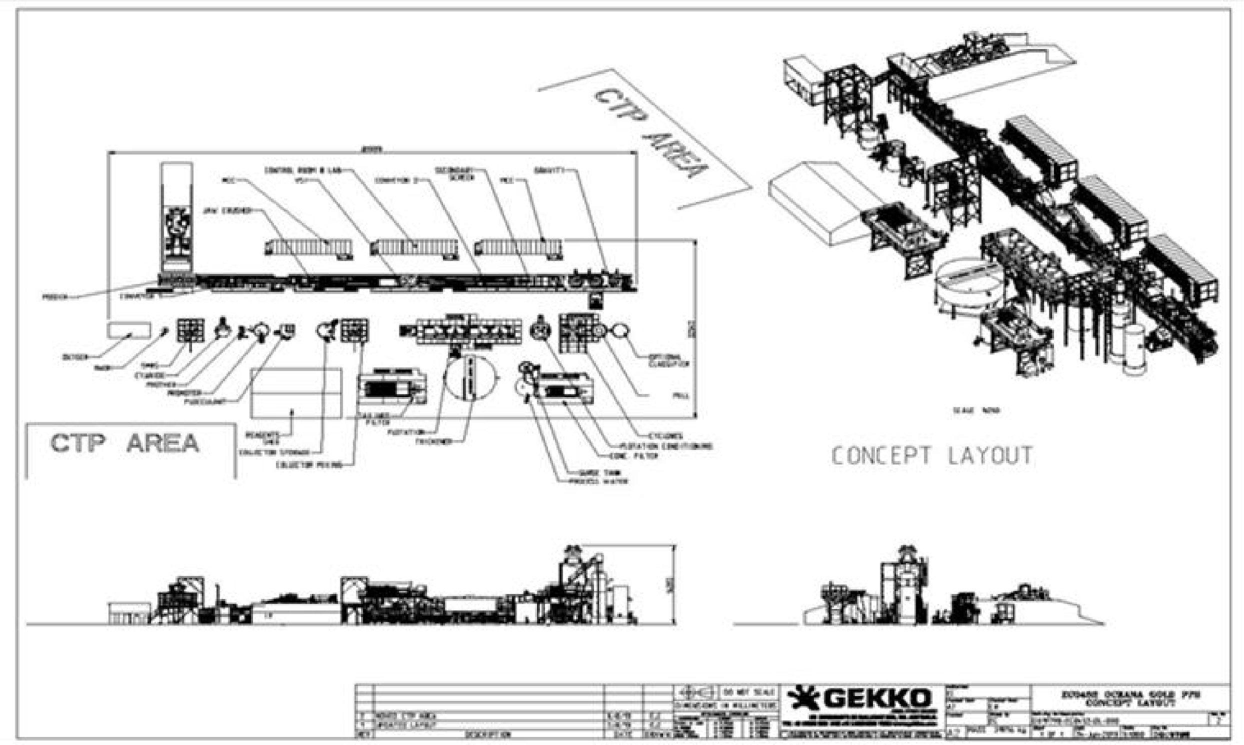

Figure 17-1: Surface Python Layout

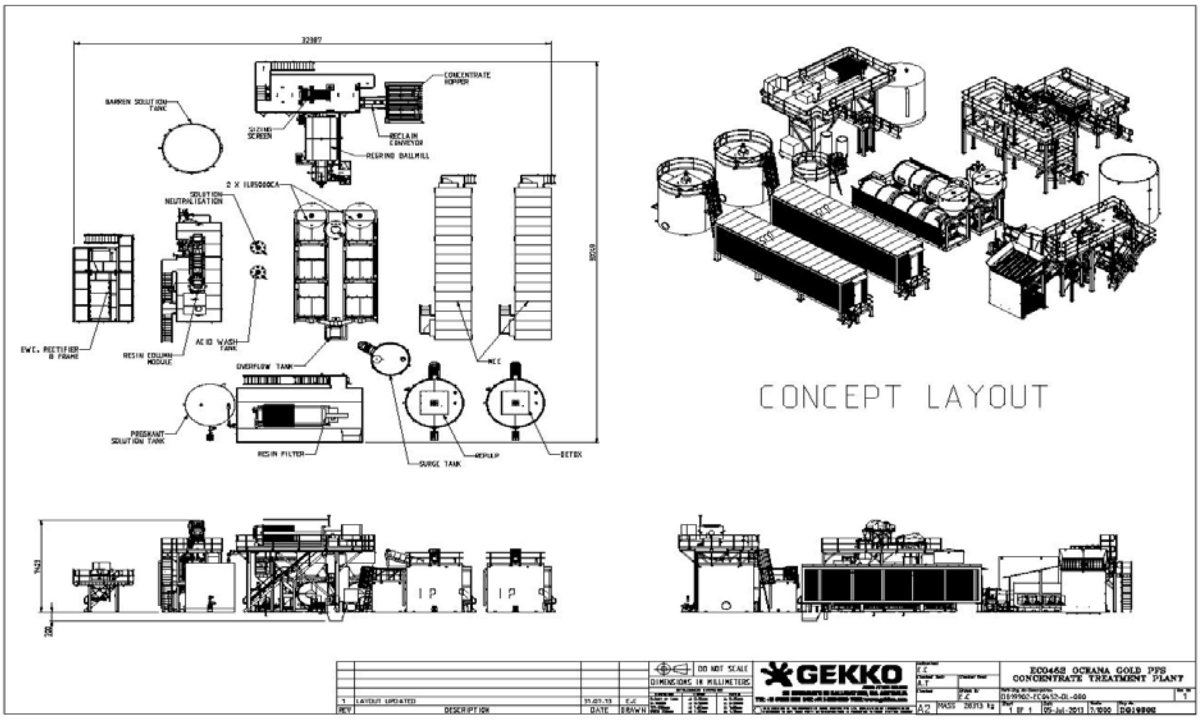

Figure 17-2: Concentrate Treatment Plant General Layout

17.3 Mass Balance and Design Criteria

The primary nominal inputs into the mass balance and design criteria are summarised in Table 17-1, while Table 17-2 details the estimated plant recovery.

Table 17-1: Mass Balance Input Summary

Stage	Units	P200 Python
Surface Python Feed	t/h	25
Surface Python Feed Au grade	g/t	16
Surface Python Concentrate Au grade	g/t	261
CTP Feed	t/h	1.5
CTP Tail Au grade	g/t	2.6
Plant availability	%	80
Plant annual throughput	tpa	120,000

Table 17-2: Plant Gold Recovery

Stage	Units	P200 Python
Au recovery from gravity	%	81.0
Au recovery from flotation	%	17.0
Overall Pythin Au recovery (gravity + flotation)	%	98.0
Au recovery from leaching	%	99.0
Au recovery from GRex and Electrowining	%	99.0
Overall Au recovery from Python and CTP	%	96.0

17.4 Selection Basis

The selection basis outlined below includes a description of how the flowsheet was developed in correlation with the laboratory test work from 2011, expected plant performance, operating strategy and potential bottlenecks.

17.4.1 Test Work Correlation to Flowsheet

The Blackwater Surface Python flowsheet has been developed directly from the test work completed in 2011 at the Gekko Metallurgical Laboratory with additional studies to confirm some more key design criteria due in July 2013. Key input data for the Surface Python flowsheet is listed below:

- Gravity gold recovery: 81%;
- Flotation gold recovery: 17%;
- Combined mass recovery: 4.5%;
- Cyclone overflow (flotation feed) particle size: P_{80} of 150µm;
- Bond crushing work index: 21.1 kWh/t;

- Bond ball mill work index: 17.7 kWh/t;
- Abrasion index: 0.76; and
- Recirculating load to VSI: 250%.

Major assumptions that have been made include:

- Mass split at jaw crusher feed grizzly: 50% to oversize;
- Tails thickener mass of water to overflow: 55%, solids density to underflow: 56%;
- Recirculating load to ball mill: 250%; and
- Tails thickener settling rate: 0.75t/m2/h;

The CTP flowsheet was not based on any test work results except the leach gold recovery of 99% which was a value given to Gekko by OceanaGold as a result of their own test work.

The majority of the CTP plant design has been developed using assumptions based on the Ballarat Goldfields CTP that was designed and built by Gekko for treatment of a similar coarse gold ore.

Assumptions that have been made include:

- Concentrate grind size: P_{80} of 106µm;
- Concentrate ball mill work index: 17.7 kWh/t;
- ILR gold extraction: 99%;
- ILR residence time: 8 hours;
- Resin maximum loading: 24,000 g/t or 8,000 g/m^3;
- Resin, electrowinning and smelting circuit gold recovery: 98%; and
- Reagent additions.

Subsequent to the process design work, the 2013 test work program validated the majority of the key parameters including:

- Primary grind target of 150µm for gravity/flotation recovery > 98%;
- Leach recovery at a 106µm regrind of 99%; and
- Electrowinning recovery at 60˚C of 99%.

17.4.2 Expected Plant Performance

The Blackwater Surface Python plant is expected to treat 25t/h of ROM ore over 4,992 hours per annum, 260 working days at 24 hours per day, 5 days per week (80% plant availability during scheduled operating time).

It is expected to achieve a gold upgrade factor of 20, concentrating the ore through comminution, gravity separation and flotation from a feed grade of approximately 16 g/t Au to a concentrate grade of 350 g/t Au in 4.5% of the feed mass.

The gravity circuit is expected to achieve a gold recovery of 81% in 1.4% of the feed mass.

The flotation circuit is expected to recover a further 17% of the gold into 3.1% of the feed mass.

The CTP incorporates regrind, gold leaching, resin adsorption, electrowinning and detox. The CTP plant is expected to treat 1.25 t/h of concentrate over 4,992 hours per annum, 260 working days at 24 hours per day 5 days per week (80% plant availability).

The leaching circuit is expected to recover 99% of the gold in the Blackwater concentrate which contains 98% of the total gold.

The resin, electrowinning and smelting circuit is expected to recover 99% of the gold from the pregnant ILR leach solution.

The overall poured gold recovery from the Blackwater Surface Python and CTP is expected to be 96%.

17.4.3 Potential Bottlenecks

Top size to the Surface Python is ~250mm based on the chain feeder size. Secondary breakage of oversize may be needed.

The ROM ore to the jaw crusher – if too much material is being fed to the crusher and is not going through the grizzly at the required rate, then the crusher may become over loaded.

The primary screen – if too much material is feeding the screen and not enough is reporting to the undersize then the aperture may need to be increased, which would then reduce the re-circulating load to the jaw crusher and potentially increase its crushing efficiency. Conversely, this would also have the effect of increasing the feed tonnes to the secondary screen and VSI.

The secondary screen – if the feed tonnes to the screen are increased due to an increase in the aperture of the primary screen and higher VSI recirculating load, then this screen may lose efficiency. This could further increase the amount of oversize to the VSI which would further increase the re-circulating load to the secondary screen. A solution to this would be to increase the aperture of the secondary screen as well. This would have the effect of reducing the re-circulating load to the VSI, but would increase the top size to the IPJ and Ball Mill.

InLine Pressure Jigs (IPJ's) – if the feed rate to the rougher jig is increased via an increase in the Ball Mill circulating load, then it may lose efficiency and the gravity gold recovery may decrease. If this occurs then it may need to be augmented with a second IPJ to process more tonnes.

Cyclones and Ball Mill – since the feed rate to the Rougher IPJ's is quite high at 70 t/h including the recirculating load from the Ball Mill and Cleaner Jig Tails, this puts a high rate of performance pressure on the hydrocyclones. If too much material is being fed to the hydrocyclones there is a chance the cyclone overflow to the flotation cells could coarsen in particle size and flotation gold recoveries may decrease since the recovery has been modelled on test work at a P_{80} of 150µm.

The direct coupling of the primary and secondary crushers with the milling circuit means frequent feed disruptions in these circuits due to oversize rock or trash in the ore, will directly affect throughput and flotation circuit stability. The surge tank before the flotation circuit is recommended to mitigate.

To fill the flotation feed surge tank, consideration should be given to designing for higher throughput at the front of the circuit e.g. 30 t/h, to enable the surge tank to progressively fill. Alternatively, the back end of the circuit, post the surge tank, could be downsized to ~20-22 t/h to better match the overall throughput and allow the surge tank to be filled.

The concentrate filter and CTP will be sensitive to concentrate mass and consideration should be given to installing a cleaner flotation circuit to enable optimum operation of the rougher flotation circuit to achieve recovery targets whilst achieving mass pull and grade targets with the cleaners.

Allowance should be made for a reasonable sized Surface Python concentrate storage area to allow for significant downtime in either of the two circuits.

The resin circuit will be at its capacity in this design. Increases in gold grade won't be able to be accommodated without slowing down concentrate treatment rate. Consideration should be made for the installation of an extra or larger resin column or the incorporation of a bowl centrifugal concentrator (BCC) and small batch ILR/EW circuit in the regrind circuit to reduce the load on the resin circuit. Alternatively another technology such as Merrill-Crowe could be used.

17.5 Technology Selection

Equipment selection in the processing plant has utilised proven technology in other operations. The choice of specific equipment types in some cases has been influenced by the philosophy of the Python processing philosophy of designing for high efficiency energy use and lower mill availability to reduce capital cost.

17.5.1 Crushing Circuit

Jaw/VSI crusher v's Jaw/Cone crusher – key drivers behind this decision are the reduced capital and improved energy efficiency when producing a -3mm product with a VSI. The VSI offers much higher reduction ratios in comparison to cone crushers which for design purposes has the reduction ratio limited to ~3. Thus there is likely to be 2 stages of cone crushing versus the single stage VSI. Even with two stages of cone crushing, it may not be possible to produce a fine enough product size minimise energy consumed in the ball mill.

17.5.2 Gravity Circuit

IPJ – the IPJ is a continuous gravity device that allows increased flexibility to deal with and maintain optimal gravity recovery in the dynamic environment that exists in such gravity circuits, i.e. when gold grade excursions occur with coarse gold the IPJ ensures that the gravity recovery is optimised. It's flexibility with mass yield is highly advantageous when dealing with variability in the sulphide content in various feed types.

17.5.3 Resin Circuit

Resin Circuit – the resin circuit was chosen as the preferred method for solution gold recovery in this circuit based on the following:

- Low leach liquor levels from the ILR reactors for the combined gravity/flotation concentrate grades;
- No access to a CIL circuit to bleed low grade gold bearing cyanide liquor into;
- Increased electrowinning cell efficiency and first pass recovery; and
- Reduced footprint compared to using carbon adsorption methods.

17.6 Capital and Operating Costs

17.6.1 Capital Cost

The installed capital cost of the processing plant for the P200 size is **USD$20.5M ± 25%** (Table 17-3).

Inclusions in the Gekko supplied direct capital costs:

- Control room;
- Laboratory;
- Project Management;
- Document Control;
- Supply of MCC's;
- Air conditioning in MCC's, Control room and Laboratory;
- GA drawings;
- P&ID drawings; and
- First fill of AuRIX Resin.

Items excluded in indirect capital costs:

- Civils and civils design;
- Site installation;
- RO water plant; and
- Filtered tailings conveying/loadout bunker.

In addition to the identified costs by Gekko, capital costs required to complete the installation have been estimated:

- Concrete/civils for 581m3 concrete $942,500;
- Mechanical/electrical installation 15,000 hours $750,000;

Table 17-3: Processing Plant Capital Cost Summary

Blackwater Processing Plant - CAPEX	US$M
Surface Python	
Crushing and Gravity	$4.1
Grinding	$1.7
Flotation	$1.3
Filtration	$1.7
Utilities/General	$0.5
Reagents	$0.7
MCC's, Control Room, Site Lab	$2.3
Total Direct Capital Cost Surface Python Plant	$12.2
CTP	
Regrind	$0.7
Leaching	$0.6
Resin Filtration	$1.1
GRex Resin	$1.0
Electrowinning	$0.8
Detox	$0.5
MCC's	$1.8
Total Direct Capital Cost CTP	$6.6
Total Direct Capital Cost	**$18.8**
Engineering. Construction/project management	-
Installation	$0.8
Concrete/civils	$0.9
Commissioning	-
Spares	-
Other	$0.03
Total Indirect Capital Cost	**$1.73**
Total Capital Cost	**$20.53**

Overhead costs for project/construction management, detailed engineering, spares, front end loader and operational readiness are incorporated into the Engineering & Design, Spares and Pre-Production Support areas.

17.6.2 Exchange Rate

The project's capital cost has been estimated based on the following exchange rates:

- 1 AUD = 0.92 USD
- 1 AUD = 0.68 Euro

17.6.3 Operating Costs

The operating costs are based on the mass balance, test work data and unit costs where available. Some of the costs for the CTP plant are based on assumptions that were made based on Gekko's experience designing and building a similar processing plant. The operating costs in this report are exclusive of contingency which does not affect the accuracy of this estimate.

General and administrative (G&A) and miscellaneous maintenance costs (small value items) are not included in this operational expenditure (Opex) estimate and are currently assumed to be part of owner's costs.

The operating costs were based on the total costs over LOM for the P200 Surface Python plant, an average throughput of 120,000t per year, an annual gold production rate of 60,000 oz. at 96% gold recovery. The processing plant operating costs are:

- Surface Python and CTP: USD$86.50/oz.
- Surface Python and CTP: USD$42/t of ROM feed.

Total operating cost is USD$49.3m over LOM and averages USD$4.6m per year.

The operating costs have been broken down into Labour, Consumables, Maintenance, Electricity and Laboratory (Assaying and Met test work) in Table 17-4.

Table 17-4: Operating Cost Summary

Area	P200 Cost 25tph		
	$/t	$/oz.	%
Maintenance	5.00	10	12
Electricity	4.08	8	10
Processing Plant Labour	22.18	46	53
Consumables	0.93	2	2
Reagents	3.21	7	8
Liners & Media	4.57	9	11
Assaying and Met Testwork	2.10	4	5
Total	**42.05**	**86**	**100**

The main costs are for labour (53%), maintenance (12%) and Liners/Media (11%) – refer to Figure 17-3.

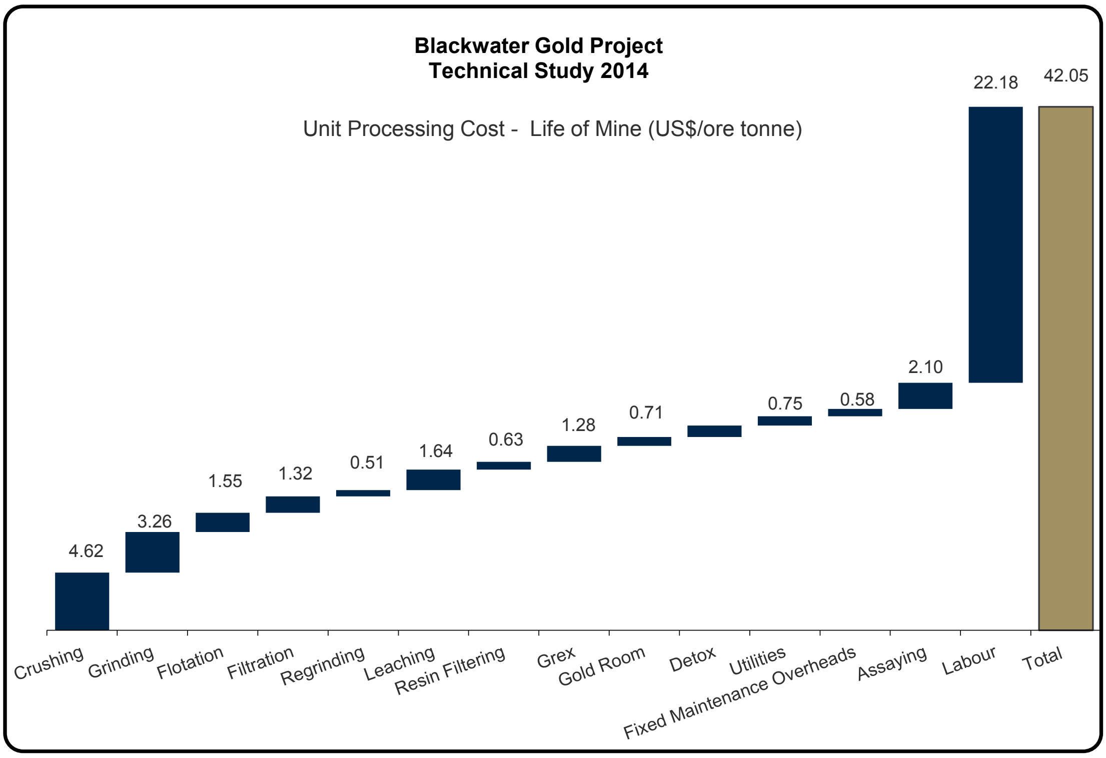

Figure 17-3: Summary of process plant operating costs

Due to the relatively high labour intensity for the processing of ore at low tonnage, labour has the highest impact on overall unit costs. During the ramp-up to full production and then closure at the end of mine life the effect of lower mill feed tonnes has a detrimental effect on labour efficiency, and ultimately on unit costs.

17.7 Further Study Work

Further engineering design is required to resolve:

- Flotation Surge Tank – this was included in the Surface Python circuit by Gekko to give two hours of uninterrupted processing time if any equipment upstream of the flotation circuit broke down and needed to be repaired (i.e. crushing, screening, gravity);
- If it is determined that possible lost gold production over two hours or flotation circuit instability is an acceptable risk, then this could be removed to save on Capital expenditure;
- Thickening and Filtration Test work – Preliminary filtration tests have been completed by Outotec to confirm the design parameters used by Gekko. For future engineering studies, these results will need to be reviewed and additional test work on reground concentrate completed so more accurate equipment sizing can be made, which will give a more accurate Capital expenditure estimate; and
- GRex and Electrowinning Circuit – leaching and electrowinnng tests were completed at a preliminary level to confirm recovery assumptions. Additional tests will need to be carried out to confirm the flowsheet and design criteria for this part of the CTP. This will also serve to revise and update equipment sizes, mass balance, capital and operating expenditures.

17.8 Risks

Inherent risks to meeting production schedules from increased rock competency and mechanical availability are offset with the required mill utilisation of approximately 52% with availability from the roster system of 71%. Additional hours are available to cater for lower throughput rates from ore hardness issues and to accommodate a lower targeted mechanical utilisation to reduce capital cost.

Tailings filtration capacity will have a significant impact on plant throughput with drops in filter availability directly impacting on plant utilisation and a requirement to meet a moisture level suitable for dry stacking or co-disposal of tailings. Scoping filtration tests have been undertaken by Outotec to allow sizing checks for a Larox PF style filter suitable to the duty to be made before detailed engineering is undertaken.

Preventing gold theft will be an on-going issue, given the head grade and coarse nature in the ore. The process plant site will be surrounded by a double security fence and utilise an access card system. This will restrict access to concentrates to plant personnel only. Additional layout and fencing may be required during final design to restrict casual access of personnel to high value material and procedures for checks on personnel and equipment leaving site.

Completion of a detailed co-disposal study to design the waste rock dump to allow encapsulation of all flotation tailings produced during the operating of the plant. The periodic availability of waste during horizontal development will require planning of dump designs to minimise risks of dusting and runoff. A suitable sized stockpile outside the plant security fence will be required to decouple the processes and security risk.

Mobilisation of heavy metals, in particular Arsenic and Antimony in the leach stage may increase levels of soluble metals in mill process water to levels above that acceptable for release. Compilation of a site wide water balance will allow modelling of water release rates and acceptable levels of metal ions in solution. Additional intensive leach testing would allow a better understanding of the ideal chemistry to minimise metal mobilisation or at least to characterise the levels to be expected and determine if a precipitation circuit is needed for water treatment.

Any potential long term AMD or heavy metal release from tailings will be addressed by segregating detoxified leach tailings to be stored as marker beads in stopes or in abandoned hangingwall drives underground. In the current plan approximately 65kt of concentrate tailings containing the majority of the sulphides in the ore will need to be disposed of underground. Sequestering this component underground should eliminate any issues in the surface waste dump and post closure underground flooding will ensure a long term anaerobic environment.

17.9 Plant Sizing

The P200 modular design from Gekko with a throughput rate of 20tph, is suited for the indicated mine production rate of 119ktpa, and also has the potential to process up to 25tph after revising laboratory test work. The P200 would allow the use of a proven, off-the-shelf design. This will reduce engineering and construction costs compared to the costs of trying to design a bespoke plant from a collection of new and second hand equipment. The Gekko P200 unit has been used as the basis for economic and physicals modelling.

17.10 Further Opportunities

The basis of the study costing assumes that all processing of Blackwater ore occurs on site. An alternative strategy to this would be produce a gravity/flotation concentrate at the Snowy Portal site and transport the material via road or rail to the Macraes process plant site, also owned by OceanaGold. At the Macraes plant the concentrate repulp and ILR reactors would be installed with an additional electrowinning cell in the gold room and existing facilities would be used for:

- Cyanide mixing and storage;
- Sodium Metabisulphite mixing;
- Inco cyanide detox circuit; and
- Eliminate the need for the GRex resin circuit.

Remote processing of the concentrate at Macraes would eliminate or defer in excess of USD$3M of capital expenditure. Operating costs for concentrate transport are expected to be in the order of USD$50/tonne of concentrate based on current haulage rates from Reefton.

Operational cost savings identified in the model include:

- $291,000 in reduced labour costs;
- $63,000 savings in resin consumables;
- $174,000 savings in maintenance on removal of the resin circuit;
- $126,000 savings in power on reduced power consumption; and
- $109,000 reduced cost on detox operation.

The current flowsheet incorporates a filtration and repulp/regrind stage between the primary ore processing and concentrate leach stages. With full processing at the Blackwater site it may be possible to replace the concentrate filter and repulp bin with a densifier cone/storage tank to handle surge capacity between the circuits with a number of benefits including:

- reduce the need for additional loader hours;
- security risk of stockpiling concentrate;
- inventory measurement issues; and
- reduced footprint and civil requirements.

The potential downsides are the loss of the ability to store or blend concentrate to reduce grade swings to the leach circuit and extra water addition to the CTP which will result in higher bleed volumes and hence higher reagent and detox costs. Consideration should be given during detailed design to the elimination of this circuit option.

At present a surge tank exists between the grinding and flotation circuits to reduce the impact on flotation from disruptions in the crushing/grinding areas with a 2 hour specified residence time followed by a conditioning tank. It is likely that the surge tank could be removed (or not constructed initially) and the conditioning tank utilised to reduce small scale surge flows to minimise upfront capital cost.

18 PROJECT INFRASTRUCTURE

This section outlines the infrastructure requirements for the Blackwater Project including site access infrastructure, site services infrastructure, processing infrastructure, administration and mine support infrastructure. Portal and underground infrastructure are not considered in this section, but are detailed in Item 16.

A site layout has been developed which considers all site restraints including lot boundaries, terrain, social and environmental. Infrastructure has been staged to manage capital cost cash flows and provide infrastructure to suit the development of the mining program.

The concept designs for infrastructure are based on input from OceanaGold's internal project development team and existing infrastructure at the operational Macraes and Reefton projects. Costs are considered accurate ± 25% and are largely based on quotations from local New Zealand suppliers.

18.1 Mine Site Infrastructure

18.1.1 Site General Arrangement

Access to the project for construction and operational traffic will be via State Highway 7 (SH7) and Snowy Road. The project area is 30 hectares and is bounded by Victoria Forest Park on the north and east, Snowy River on the west and Ngai Tahu land on the south.

The Blackwater Project general arrangement designed during this technical study is shown in Figure 18-1. Site Infrastructure is divided into five main areas:

- Site Services Area;
- Administration Area;
- Mine Services Area;
- Processing Area; and
- Mining Area.

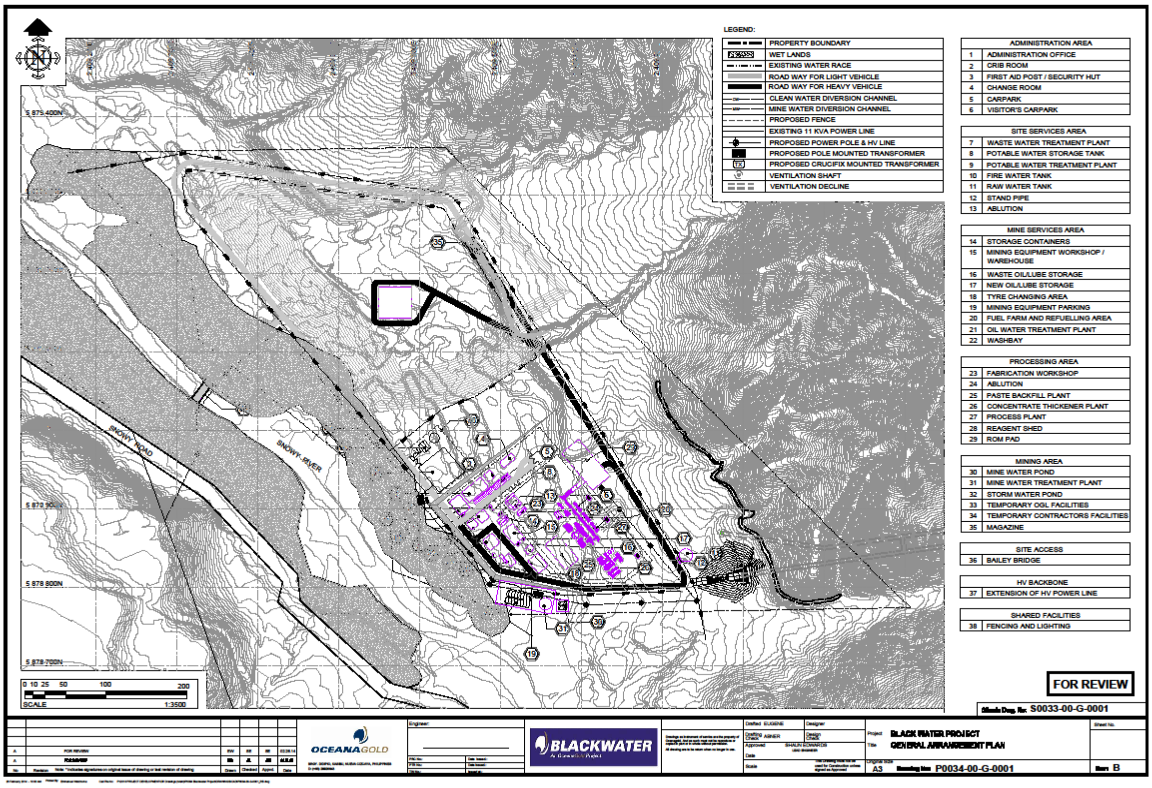

Figure 18-1: Blackwater Project, General Arrangement

The main infrastructure areas will be located on a built up rock fill pad (RL192m) on the Snowy River alluvial plain. The alluvial plain is the most suitable location for the permanent infrastructure as the area is relatively flat, lightly vegetated and in close proximity to the underground portal location. The pad level provides mitigation against the 1 in 1000 year flood level, as modelled by Golder Associates (Golder Associates (NZ) Limited, 2013) and requires an average of 3.5m of filling over the area. The pad will be constructed in stages from rock fill material sourced during decline development. Decline material will be blasted Greywacke material which is expected to provide a quality foundation for site infrastructure.

The site services area includes the raw water system, potable water system, waste water disposal system and high voltage electrical backbone and is located centrally to the processing area, mine services area and administration area to minimise distribution network requirements.

The administration area includes the administration office, crib room, first aid post/security post and change room. The administration area is located in close proximity to the processing and mine services areas such that administration facilities are easily accessible to those areas.

The mine services area includes the mining equipment workshop, mining equipment parking area, the mining equipment fuelling facility, mining equipment wash bay, tyre change facility and oily water treatment plant. The mine services area is located to the south of the processing area (approximately 200m from the underground portal) and has been positioned to keep mining equipment away from the processing and administration areas.

The processing area includes the process plant, concentrate treatment plant (CTP), reagent shed, fabrication workshop and ablutions. The processing area will be securely fenced and lit to ensure security for the safe handling of gold ore.

The mining area includes the ROM pad, underground portal entrance, waste rock dump (WRD), site raw water tank, mine water pond, mine water treatment plant, storm water pond and clean water diversion channels. The underground portal is located in the south eastern area of the Blackwater Project. Ore from the underground operation is delivered to the ROM pad which is located approximately 250m from the portal entrance to minimise haulage distance.

The ROM is located 8m higher than the process plant enabling direct feeding of ore into the crushing circuit. Waste material from the underground operation is stored in the WRD which takes up most of the northern area of the site.

18.1.2 Site Constraints

The most significant constraints on the site infrastructure layout design are as follows:

- Snowy River Flood Level: the 1 in 1000 year Snowy River inundation level is RL192m (Golder Associates (NZ) Limited, 2013). All life of mine infrastructure will be located above this elevation;
- Property boundary: the property is bound on the north and east by the Victoria Forest Park, south by Ngai Tahu land and the Snowy River on the west. The locating of infrastructure outside the current property boundaries was not considered due to uncertainty of land acquisitions, timing of any acquisitions and OceanaGold preference to keep infrastructure within the project area;
- Victoria Forest: the Victoria Forest Park extends into the eastern area of the property. The area is steep and heavily vegetated, therefore the infrastructure layout was designed to avoid unnecessary disturbance of the area which would create sediment and possibly slope stability issues;
- Portal location: the location of the portal and ventilation shaft were considered a “fixed” entity for the infrastructure layout design;
- Pine plantation: pine trees have been planted along the western boundary of the property by a previous land owner. Although the plantation can be cleared, the infrastructure layout was designed to avoid unnecessary clearing activities which would cause sediment issues plus have cost implications due to timber harvesting agreements made with the land owner. The pine trees also provide a visual buffer between the public road and the site;

- Existing Water Race: a post-1900 water race is located in the western area of the property. Although not legally bound to protect the works, OceanaGold has committed to protect the water race where practicable. Any requirement to modify or destroy the structure will be undertaken with input from Heritage NZ; and
- Wet lands: the existing wet lands along the western property boundary will be maintained as much as possible and utilised to dilute the treated water discharges from the project area.

18.1.3 Existing Site Infrastructure

There is no existing infrastructure at Blackwater site. Nearby infrastructure that services the property includes:

- 11kV transmission line which terminates just outside the entry to the site;
- The Snowy Road which passes by the site; and
- NZ telecom connection approximately 9km from site at the SH7 and Snowy Road intersection.

18.1.4 Site Layout Options Considered

During the study engineering work, variations of the site layout were considered and engineered to varying degrees in order to assess the relative feasibility between options. The main factors considered for the site layout are as follows:

- Topography;
- Vegetation;
- Geotech;
- Construction practicality;
- Construction cost; and
- Operability.

Topography is the main driver of the site layout. Infrastructure has been placed on the flatter areas to minimise earthworks and the geotechnical risks associated with cutting/filling infrastructure onto hillsides.

The main infrastructure areas have been located on the lightly vegetated Snowy River alluvial plain to minimise the clearing of indigenous beech vegetation located on the eastern hill slopes and the pine plantation located along the banks of the Snowy River. Clearance of indigenous vegetation requires special approval within the resource consent which will need to be applied for in advance of any clearing activities. Clearing of the pine plantation involves paying the landowner $60 000 NZD per ha for that right which is made up of $18 000 NZD per ha for the land parcel and a further $42 000 NZD per ha for the cutting rights as advised by OceanaGold consenting team.

The operational efficiency of the site has been maximised by locating interdependent facilities in close proximity to each other. The final layout constitutes the best over-all solution for the development of the project.

18.1.4.1 Administration Area Options Considered

The most significant priority when selecting the location for the administration was to segregate the area as much as possible from the mining and processing areas yet keep it close enough that the communal facilities such as the offices, crib room, change rooms, first aid post and warehouse are accessible.

18.1.4.2 Mining Services Area Options Considered

The priorities when selecting the location for the mine services area was proximity and ease of access to the mining operations (i.e. the portal and haul route) and clear segregation between heavy mining equipment and other site activities.

18.1.4.3 Processing Area Options Considered

The priority when selecting the location for the processing area was proximity to the portal to minimise ROM haulage. Another consideration was attaining a suitable height difference between the ROM and the process plant such that ore can be stockpiled and fed directly into the crushing circuit without ramping. The processing area is located adjacent the mine water pond and the storm water pond for easy connection to the either pond if required.

18.1.4.4 Mining Area Options Considered

The most significant priority when selecting the location of mining area infrastructure such as the mine water pond and storm water pond was topography. The mine water and storm water ponds are located at the low point of the property to allow easy drainage to the location and into the adjacent wetlands. The clean water diversion channel is located along the eastern disturbance limit of the property which diverts maximum volume of clean water around the project. The WRD is located in the northern area of the property as the wider northern area allows a more efficient WRD development. The magazine has been located in the far north of the property to maximise the clearance area between the magazine and main infrastructure areas.

18.2 Engineering and Design of Site Infrastructure

The level of engineering and design detail undertaken in this study is sufficient to demonstrate the technical viability of the proposed solution, in addition to allowing cost estimates in line with the nominated level of estimate accuracy of ±25%.

The site infrastructure has been designed to comply with applicable legal requirements, industry standards and best practice. The guiding design principals are to provide the maximum degree of safety, efficient use of capital, operational efficiency, minimal environmental impact and a high level of reliability, operability and maintainability. Proven infrastructure designs utilised at other OceanaGold sites were applied where appropriate.

18.2.1 Basis of Infrastructure Design

The primary sources of information used for infrastructure design are as follows:

- OceanaGold In house engineering expertise;
- LIDAR survey data;
- Processing design and information provided by Gekko;
- Mining design and information provided by Mining Plus;
- Organisation chart developed by OceanaGold with input from Gekko and Mining Plus;
- Reports and input from external consultants (Golders, Gekko, Mining Plus);
- Vendor supplied specifications and information;
- Experienced gained at other OceanaGold sites; and
- Best practice.

Blackwater Project Infrastructure is designed to operate fully independent of other OceanaGold operations. The site layout is based on assumptions that geotechnical conditions are suitable for construction as no geotechnical investigation had been conducted at the time of the study.

18.3 Project Schedule and Infrastructure Staging

18.3.1 General

Infrastructure development has been split into three stages to align with construction durations of the exploration decline.

- Stage one – Early Works Infrastructure (Pre-production year 1);
- Stage two – Support Services Infrastructure (Pre-production year 1); and
- Stage three – Process Plant Infrastructure (Pre-production year 2).

Rock from the decline development will be used to build up rockfill pad areas to (RL192m). The stage two rockfill area will be completed first, to open up area for the construction of permanent facilities to support the operation of the mine including administration areas, site services and mine services. Upon completion of the Stage two rockfill pad, rock from the decline development will be directed to the Stage three rockfill pad to create area for the construction of the processing plant (including CTP), processing support facilities and mining area infrastructure (stage 3).

18.3.2 Stage One Infrastructure

Stage one infrastructure will provide permanent site access, power and communications to the Blackwater site plus provide basic infrastructure for the establishment of facilities to support decline development work. Infrastructure included in Stage one includes:

- Establishment of temporary sediment controls;
- Extension of the high voltage power line into Blackwater site through to the box cut;
- Extension of communications (overhead line) from SH7/ intersection into Blackwater site through to the box cut;
- Establishment of site access road to the box cut;
- Installation of Bailey bridge;
- Establishment of temporary OceanaGold office facilities (relocated to form part of long term facilities);
- Establishment of temporary contractor facilities;
- Establishment of temporary site services such as potable water, septic system and power;
- Upgrade of SH7 and Snowy Road intersection; and
- Upgrade of Snowy Road (passing bays).

The Stage one infrastructure plan is shown on Figure 18-2.

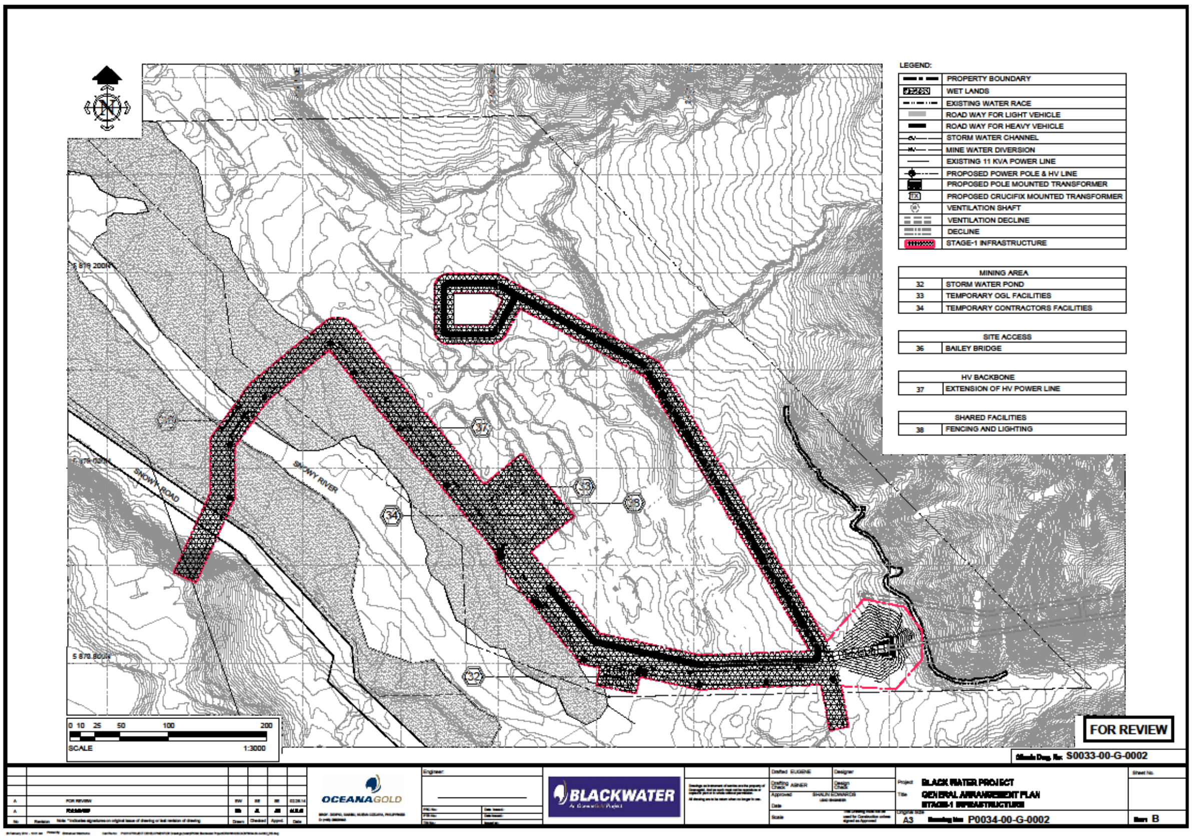

Figure 18-2: Stage 1 Infrastructure Plan

18.3.3 Stage Two Infrastructure

Stage two infrastructure will provide the life of mine service and administration facilities for the operation of the Blackwater Project.

Infrastructure included in Stage two includes:

- Establishment of Stage two site access roads;
- Extension of high voltage power to Stage two infrastructure areas;
- Establishment of site services area:
- Raw water system;
- Potable water system;
- Fire water system;
- Waste water treatment system;
- Establishment of mine services area;
- Mining equipment workshop/warehouse;
- Mining equipment washbay;
- Mining equipment tyre change facility;
- Mining equipment refuelling facility;
- Mining equipment parking area;
- Establishment of the administration area;
- Administration office;
- First aid post/security hut;
- Crib room;
- Change room;
- Explosives magazine; and
- Site raw water tank.

The Stage two infrastructure plan is shown on Figure 18-3.

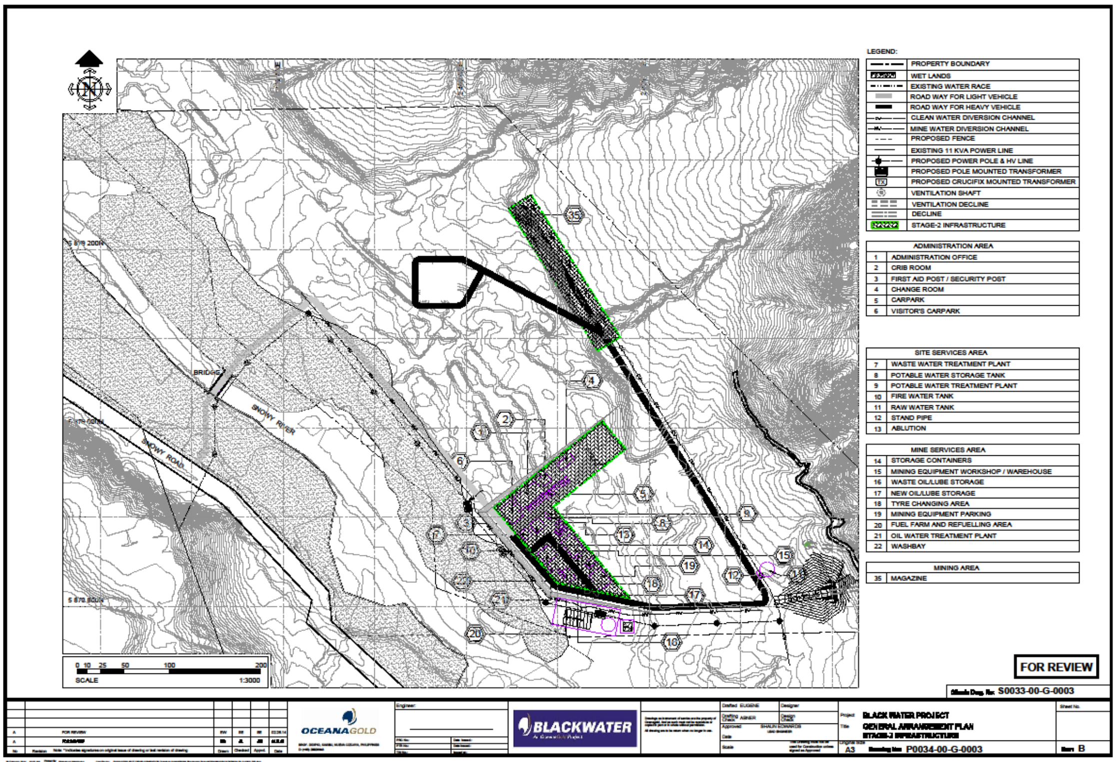

Figure 18-3: Stage 2 Infrastructure Plan

18.3.4 Stage Three Infrastructure

Stage three infrastructure includes the construction of the process area and infrastructure related to the treatment of underground mine water. Stage three infrastructure will be developed to coincide with the completion of decline development and the commencement of ore production. Infrastructure included in stage three is listed below:

- Extension of high voltage power to stage three infrastructure areas;
- Establishment of Stage three site access roads;
- Establishment of processing area;
    - Reagent shed;
    - Fabrication workshop;
    - Process plant;
    - Concentrate treatment plant;
- ROM Pad;
- Tailings impoundment area;
- Mine water treatment pond;
- Mine water treatment plant.

The stage three infrastructure plan is shown on Figure 18-4.

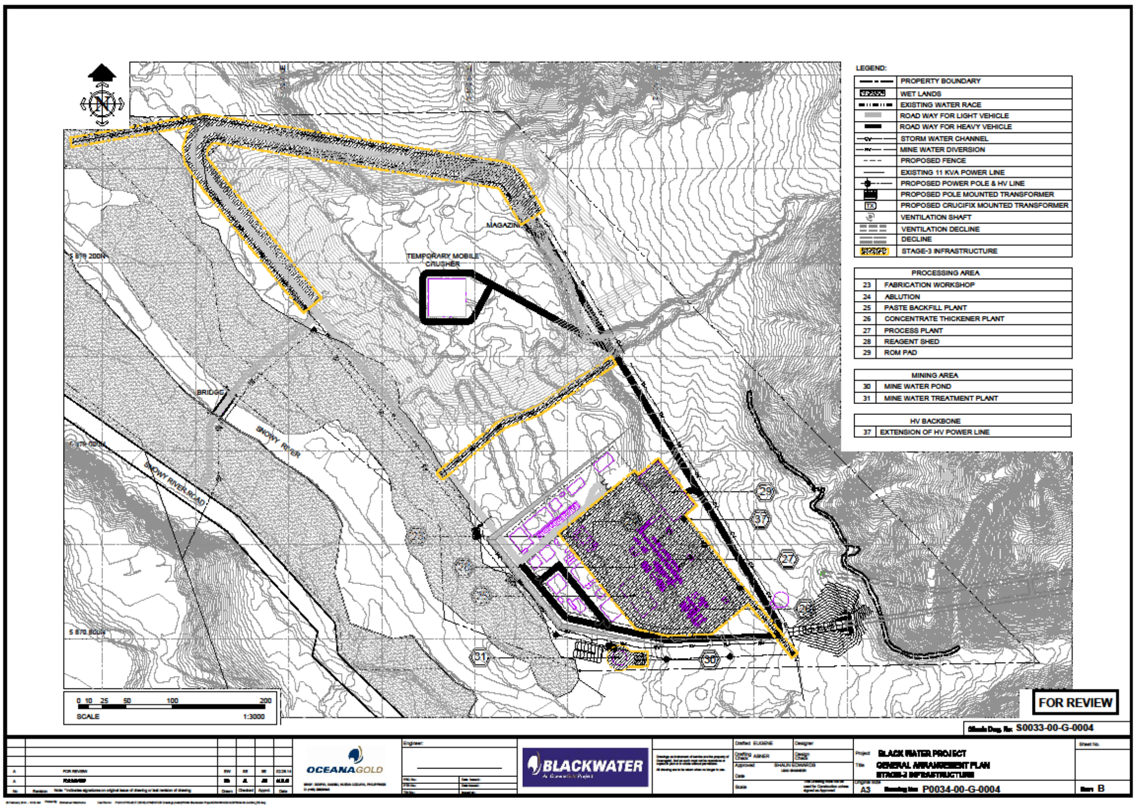

Figure 18-4: Stage 3 Infrastructure Plan

18.4 Site Earthworks and Civils

18.4.1 General

Earthworks for the Blackwater Project will be staged to defer capital costs and to avoid unnecessary early vegetation clearing and sediment generation. Bulk earthworks should be undertaken during the drier months to minimise earthwork costs, difficulties and sediment issues.

18.4.2 Tree Cutting and Vegetation Clearing

The Blackwater site will be cleared in a staged manner to minimise sediment generation. Site infrastructure is located primarily on the lightly vegetated Snowy River Alluvial plain and therefore clearing into the Victoria Forest area is minimal. Cleared vegetation will be pushed into stockpiles for controlled burning as the areas are opened up for construction. Clearing of vegetation from the WRD footprint will be undertaken progressively over the life of the mine and therefore only nominal allowance for clearing within the WRD footprint has been allowed.

Indigenous Beech vegetation and the pine plantation shall only be cleared with the required approvals in place. The estimated clearing area for each stage is summarised in Table 18-1.

Table 18-1: Clearing Area Estimate

	Stage One	Stage Two	Stage Three	Total
Surface Area (m^2)	60,000	22,000	44,000	**126,000**

18.4.3 Topsoil Stripping

In the absence of geotechnical information for the Blackwater site, topsoil stripping volumes have been estimated assuming 0.5m stripping depth over the full area of each stage. The WRD footprint will be stripped progressively during the life of mine as waste rock storage is required therefore only nominal allowance for early WRD topsoil stripping has been included in the estimate. The estimated topsoil removal volume for each stage is summarised in Table 18-2.

Table 18-2: Topsoil Volume Estimate

	Stage One	Stage Two	Stage Three	Total
Surface Area (m^2)	23,000	14,000	30,000	**67,000**
Topsoil Removal Volume (m^3)	11,500	7,000	15,000	**33,500**

Stripped topsoil will be stockpiled in a controlled manner inside the footprint of the WRD and used for rehabilitation of the WRD batters as the WRD develops. Topsoil stockpiles shall be clearly labelled and limited to less than 3m in height to avoid damaging the regenerative capacity of the material. This strategy will control sediment.

18.4.4 Pad Construction (RL 192m)

Stage one temporary facilities will be sited on existing ground at approximately RL188 – 190m however the stage two and stage three infrastructure areas will be built up (3m-4m) with blasted rock (Greywacke) sourced from decline development. In total, approximately 94,500 m^3 of blasted rock will be placed (layered and compacted) onto prepared foundation material to form a pad at RL192m. Based on the decline development schedule, the stage two and stage three pads will take approximately 14 months to complete. The estimated rockfill requirement for each stage is summarised in Table 18-3.

Table 18-3: Pad Rockfill volume Estimate (RL 192m)

	Stage One	Stage Two	Stage Three	Total
Surface Area (m^2)	N/A	11,500	15,500	**27,000**
Bulk Rockfill Requirement (m^3)	--	40,250	54,250	**94,500**
Pad Completion Date	--	7 months*	14 months*	**14 months***

18.4.5 Sheeting Material

Sheeting is required for roads and the sheeting of the main infrastructure areas. The volume of sheeting material required for each infrastructure stage is shown in Table 18-4 with the assumed thickness of sheeting required.

Table 18-4: Sheeting Volume Estimate

	Stage One	Stage Two	Stage Three	Total
Surface Area (m^2)	23,000	14,000	30,000	**67,000**
Sheeting Thickness (mm)	200	200	200	**200**
Sheeting Volume (m^3)	4,600	2,800	6,000	**13,400**

Sheeting material will be produced using waste material mined underground by an onsite crusher that will be located in the footprint area of the WRD. The onsite crusher will produce -50mm material.

18.4.6 Drainage

Approximately 2.8km of drainage channel is to be constructed around the site. A clean water diversion drain will be constructed along the northern and eastern most disturbance limits of the site to divert clean runoff around the site. A mine water collection channel will be constructed along the southern and western limits of the site to capture site runoff and direct to sediment ponds. The drains will be constructed progressively as the mine is developed. The estimated length of channels to be constructed in each stage is shown in Table 18-5.

Table 18-5: Drain Requirement

	Stage One	Stage Two	Stage Three	Total
Clean Water Diversion Channel (m)	410	190	550	**1,150**
Mine Water Diversion Channel (m)	680	180	800	**1,660**

18.4.7 Retaining Structures

Retaining structures are required along the eastern perimeter of the processing area and the ROM.

The processing plant retaining wall is expected to be approximately 100m long and 3.5m high though the requirement for retaining needs to be confirmed by geotechnical investigation. The retaining wall will be constructed of gabion baskets utilising alluvial cobbles sourced from the Snowy River.

The ROM pad/crusher retaining structure is expected to be 15m long and 7m high and constructed of reinforced concrete.

18.4.8 Concrete Supply

Quality concrete is readily available in the region. Allied Concrete, a reputable concrete supplier, provided quotation for 30Mpa ready mix concrete supplied to Blackwater site at \$234NZD/m^3 based on an expected requirement of 1,805m^3. The delivery time for concrete to site is approximately one hour and the maximum hourly batching rate at the Allied Concrete facility is 60m^3/hr. which should be sufficient for the expected size of project pours. A comparison quotation for onsite batching of concrete was also provided by Allied concrete however the supplied concrete unit cost is only marginally cheaper at 232NZD/m^3 and does not include mobilisation and demobilisation charges for the plant. It is therefore recommended that concrete is supplied from an offsite batching facility as the overall volume of concrete required and the proximity of the project to local concrete suppliers cannot justify an onsite batching facility. Estimated concrete volumes for the project are summarised in Table 18-6.

Table 18-6: Staged Concrete Volumes

	Stage One	Stage Two	Stage Three	Total
Concrete Volume (m^3)	90	725	990	**1,805**

18.5 Site Access Infrastructure and Logistics

18.5.1 General

Access to the Blackwater site is via State Highway 7 (SH7) and Snowy Road. A Bailey bridge and private access road will connect the site to Snowy Road. It is expected that imported goods from outside the south island of New Zealand will be shipped into Lyttelton Port, Christchurch from where the goods will be trucked to site.

Gold doré will be transported off site by road. The road network is illustrated in Figure 18-5.

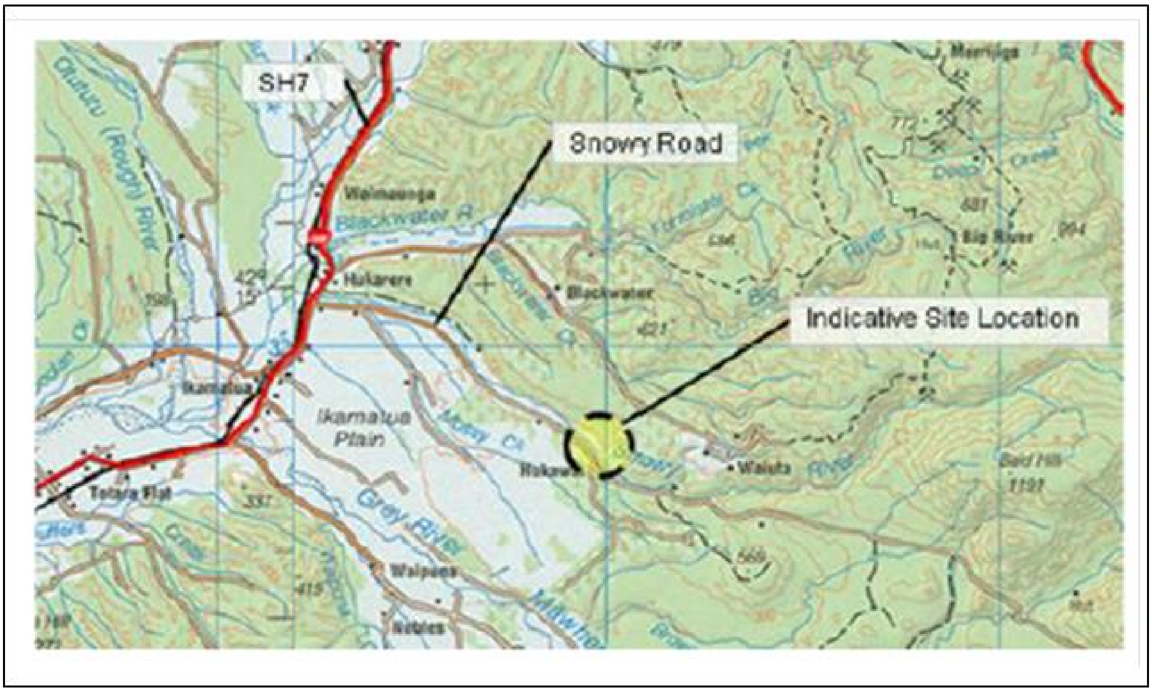

Figure 18-5: Snowy Road and State Highway 7 (SH7)

A traffic assessment of the primary access route to the Blackwater Project was undertaken in February 2013 by Beca Infrastructure Ltd (Beca). The primary aim of the assessment was to estimate the expected increase in vehicle movements during the construction and operational stage of the project and to make recommendation of any modifications/upgrades required to the existing road network to safely manage the increased vehicle movements. The major recommendations are listed below and will be implemented as part of stage one infrastructure development:

- Upgrade of the SH7 and Snowy Road intersection; and
- Addition of passing bays on Snowy Road.

18.5.2 State Highway 7 (SH7) and Snowy Road Intersection

SH7 is a Strategic Route that provides inter-regional connections between Canterbury and Nelson/Marlborough with the West Coast. SH7 intersects with Snowy Road at about 220 km from the start of SH7 at Waipara (route position 212/8.3). On NZTAA advice, the SH7/Snowy Road intersection will be upgraded to ensure that the road can safely manage the increased number of trucks travelling the route as follows:

- Undertake local road widening at the Snowy Road/SH7 intersection in accordance with 'Diagram E' in NZTA's Planning Policy Manual, to allow slow moving trucks to accelerate and decelerate clear of through traffic;
- Widen (to the north) the Snowy Road approach to the intersection with SH7 to enable trucks tuning left into Snowy Road to negotiate the turn at a reasonable speed and without crossing the Snowy Road approach centre-line; and
- Install truck-crossing signage on both the southern and northern approaches to the Snowy Road/SH7 intersection.

18.5.3 Snowy Road Passing Bays

Snowy Road is a rural road which intersects SH7 about 60km south of the SH7 Snowy River Bridge intersection. It is two-laned with a centre line for the first 2km from the intersection with SH7, with an approximate width of 5.5m. Thereafter it becomes a one-laned road, with approximately four m width. 1.5 km from the intersection with SH7 there is a one-laned bridge. Snowy Road is sealed for the first 8 km and then becomes gravel. The proposed haul road access to Blackwater Mine is located 8.5 km from the SH7 intersection. There are eight properties along that 8.5 km section of Snowy Road. The road provides access to the Snowy Battery Tramping Route. There are forest blocks located along the road. To manage the additional traffic movements the following improvements have been recommended by Beca and will be implemented by OceanaGold as part of Stage 1 infrastructure development:

- Erect a truck crossing sign at the mine access point on Snowy Road;
- Install gravel lay-bys on Snowy Road to create additional room for vehicles to pass, particularly when large equipment is being transported during construction of the mine. Gravel shoulder widening may be constructed on Snowy Road at other locations to provide passing bays, as required; and
- Undertake localised curve widening on Snowy Road to avoid large vehicles tracking onto the road berms.

18.5.4 Bailey Bridge

Access from Snowy Road into the site requires the crossing of the Snowy River. A Bailey bridge is to be constructed as part of stage one infrastructure development to enable vehicles to pass into the site without the need to disturb the river. A Bailey bridge is preferred to a floodway crossing due to lower maintenance requirement and low environmental impact. Based on the synthetic hydrograph data in Golders Water Management Report Ref No.1178110053-R-001-V7, the average maximum flow rate for the Snowy River is 55.81m3/s and occurs in May. The synthetic one in 50 years, 15 minute flow rate is 87.03m^3/s. Limiting the velocity in the river to 2m/s without causing back up requires a bridge of clear span approximately 25m.

A 30m span, single landed Bailey bridge with 4m deck width will be installed at the site. The Bailey bridge will have an operational load capacity of 50 ton. Heavy vehicles and oversized loads which cannot pass over the Bailey bridge will tram across the river. The bridge shall be hot dipped galvanised to minimise maintenance requirement over the life of mine. The bridge will sit on precast concrete foundations supplied as part of the Bailey bridge supply package. The abutments will be retained with gabion basket walls which will allow the span of the bridge to be minimised without risking abutment damage. Gabion baskets were chosen due to the abundant supply of cobbles in the Snowy River.

During the study, several Bailey bridge suppliers were contacted. The products supplied by manufacturers are very similar with little variation in designs and cost. All systems are designed for in gauge transportation to site (i.e. no oversize components).

The bridges can be assembled and launched using a 30t excavator or may be lifted into place with a crane. All manufacturers indicate an installation period of approximately three days if supervised by a representative of the manufacturer. A suitable Bailey bridge system is illustrated in Figure 18-6.

Installation of a pedestrian bridge was considered however it is expected most staff will commute to and from the site by vehicle and therefore the expected usage of the crossing would be unlikely to warrant the cost of construction. For this reason no pedestrian bridge has been included.

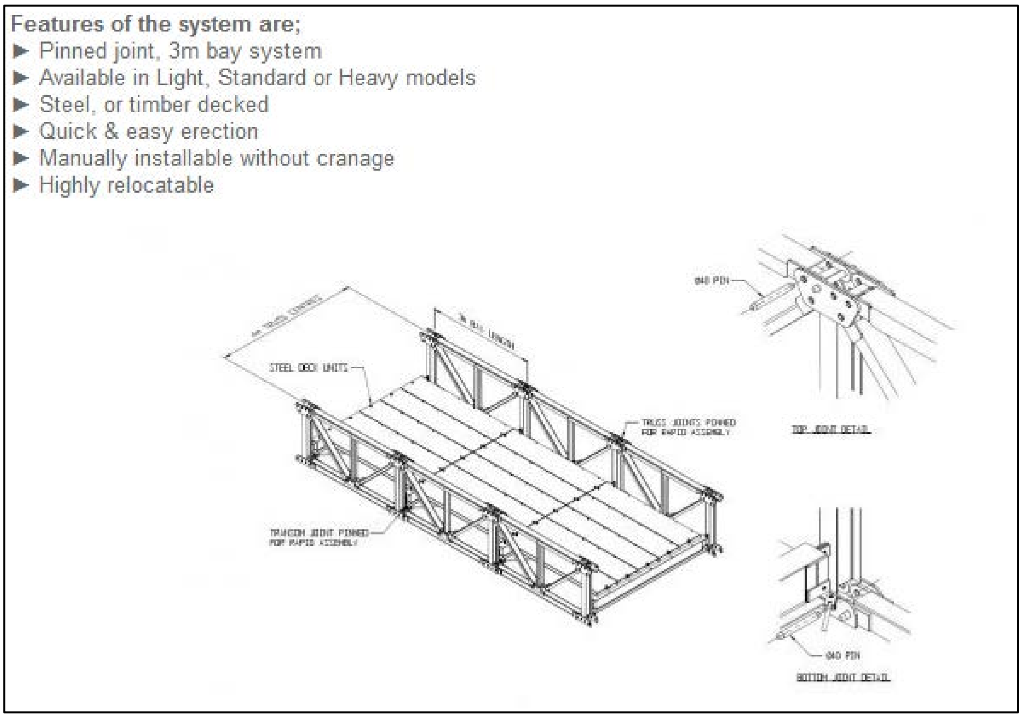

Figure 18-6: Typical Bailey Bridge Details

18.6 Site Services Infrastructure

The site services area is located centrally between the administration area, processing area and mine services area. The location was selected to minimise the size and cost of the associated distribution system plus the site is close to the Snowy River for raw water extraction and treated waste water discharge. The services considered in the infrastructure design include:

- Raw water (suitable for process water, dust suppression, wash down and fire water);
- Potable Water (suitable for all domestic uses);
- Electrical Power (Grid and backup);
- Waste water (domestic waste water) collection and treatment;
- Diesel fuel (Delivery, storage and distribution); and
- Compressed air (note that compressed air will not be reticulated from a central facility but rather will be produced by separate equipment at the demand centres as required).

The general arrangement of the site services area is shown in Figure 18-7.

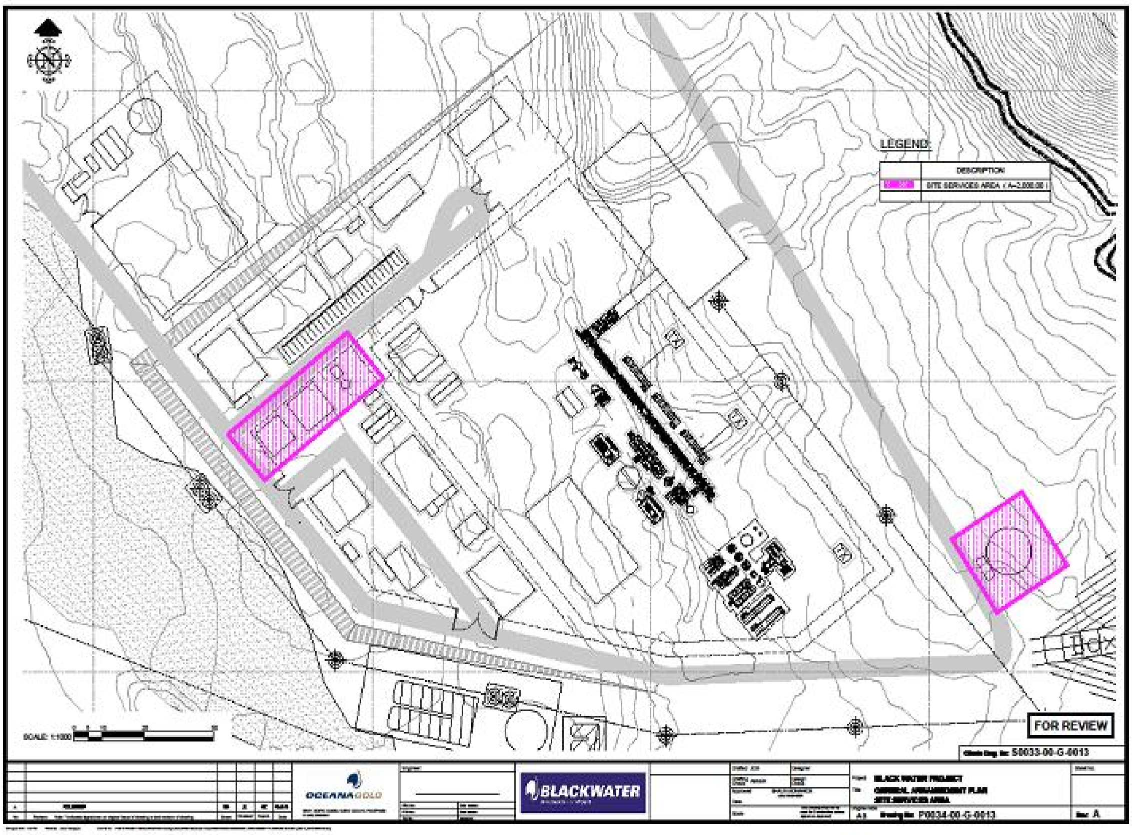

Figure 18-7: General Arrangement of Site Services Area

Mine water, including water from dewatering of the underground workings, is expected to be recycled. Resource consents have been obtained on the basis that any net water take will be less than 20L/s, which is the relevant regulatory threshold for consenting purposes. This range, coupled with the ability to recycle water, is expected to be adequate to meet all processing, mining and ancillary needs.

The expected average raw water requirement for the project will be sourced from a single borehole which will be located after hydrogeology study of the site. A back up borehole will be installed as a precautionary measure to ensure continuity of plant operation if the primary borehole experiences issues. An extraction point for raw water on Snowy River was considered however discounted due to seasonal variation to flow and quality.

The demand for potable water at the project is estimated to be 30kL per day which is based on 100 litres per person per shift per day.

The power grid in the project region is reliable with minimal outages expected and therefore backup power systems have been kept to a minimum. The estimated operating power requirement and backup power requirement for each of the main infrastructure areas is summarised in Table 18-7.

Table 18-7: Expected Operating Power and Backup Power Requirement

Requirement	Installed Power (kW)	Consumable Power (kW)	Backup Power (kW)
Administration Area	250	<250	0
Site Services Area	250	<250	0
Mine Service Area	250	<250	0
Processing Plant	2,382	1,491	0
Concentrate Plant	519	336	0
Mining Area	2,000	<2,000	<500
Total	5,651	<4,557	<500

18.7 Administration Area Infrastructure

The administration area is located at the entry to main infrastructure area in close proximity to the processing area and mine services area. The administration includes the following facilities:

- Administration Office and parking area;
- Crib Room;
- Change Room and ablutions; and
- First Aid Post/Security hut.

The layout of the administration area is shown in Figure 18-8.

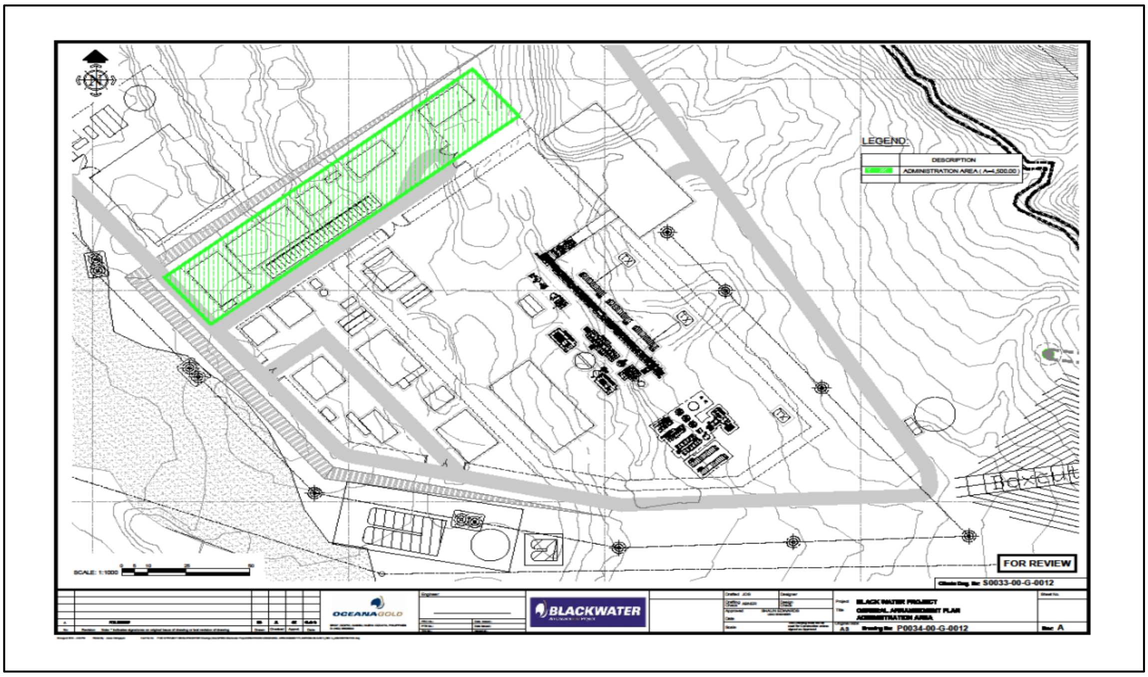

Figure 18-8: Administration Area, General Arrangement

The administration area will be serviced with the following:

- Potable water;
- Raw / Fire water;

- Power supplied at 415/240VAC;
- Waste water connection to the WWTP; and
- Communications network connection including high-speed internet.

18.8 Mine Services Area Infrastructure

The mine services area, which supports the mining operations, is located to the immediate west of the processing area. The area has been intentionally separated from the processing and administration areas to minimise interaction between the mining operations and other site operations. The mine service area includes:

- mining equipment fuelling area,
- mining equipment wash bay,
- mining equipment workshop/warehouse,
- new and waste oil storage areas,
- tyre change facility; and
- mining equipment parking area.

The general arrangement of the mine services area is shown in Figure 18-9.

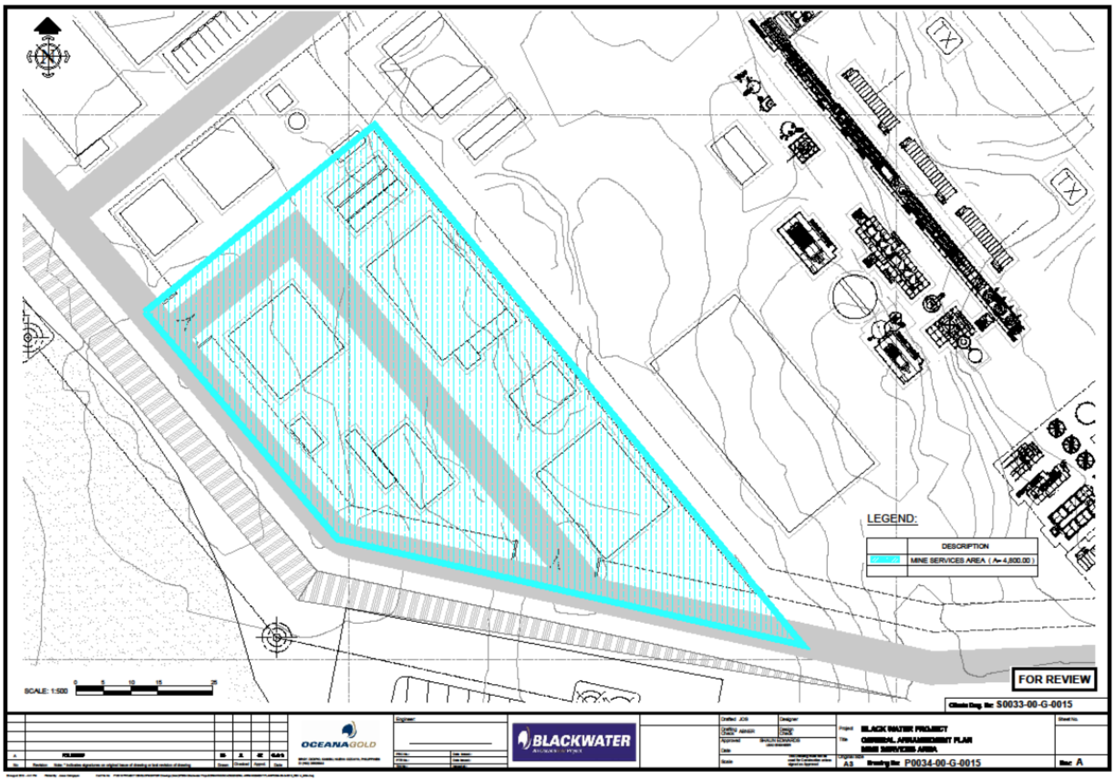

Figure 18-9: General Arrangement of Mine Services Area

The mine services area will be serviced with the following utilities:

- Potable water;
- Raw/Fire water;
- Diesel fuel;
- Power supplied at 415/240VAC;
- Waste water connection to the WWTP; and
- Communications network connection including high-speed internet.

18.9 Processing Area Infrastructure

The processing area includes the processing plant, Concentrate Treatment Plant (CTP), Reagent Shed and Fabrication Workshop. The area will be securely fenced with double perimeter fencing for security of gold doré on site. No allowance has been made for a helicopter landing area as gold transport from site will be by road. All facilities have been positioned such there is full perimeter access for maintenance works. The general arrangement of the processing area is shown in Figure 18-10.

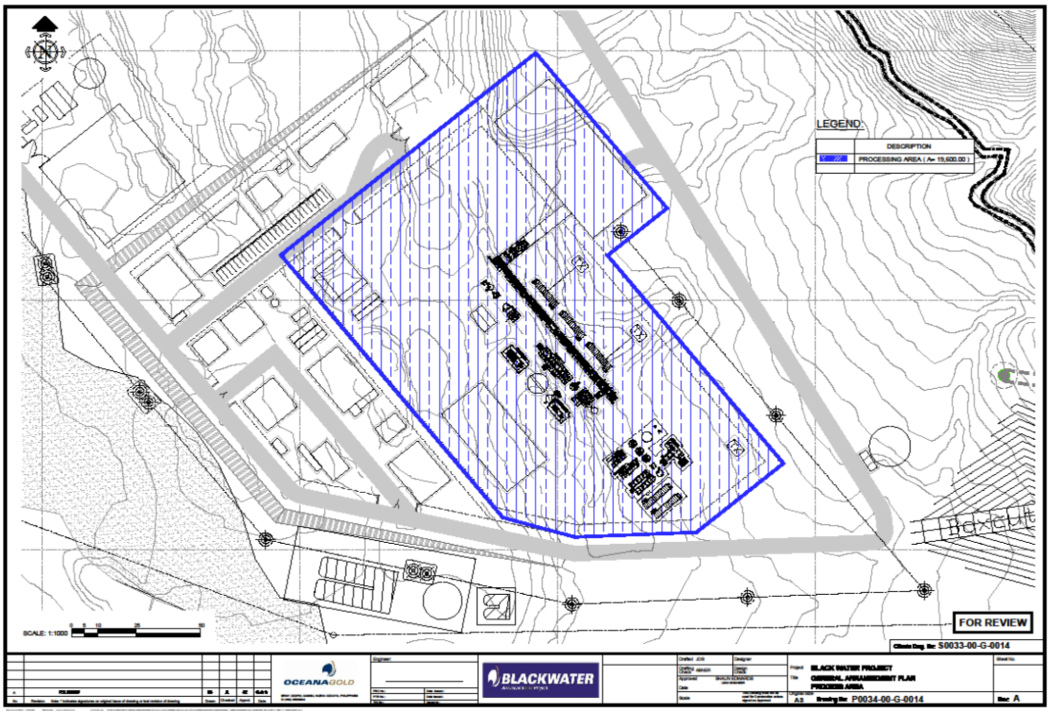

Figure 18-10: General Arrangement of Processing Area

The processing area will be serviced with the following utilities:

- Potable water;
- Raw / Fire water;
- Power supplied at 415/240VAC;
- Waste water connection to the WWTP; and
- Communications network connection including high-speed internet.

18.10 Disposal of Mine Waste Rock and Tailings

Current approvals allow for 1.1M m^3 of waste rock capacity. In the PEA it has been estimated that additional capacity of up to 25% may need to be consented to accommodate the additional volume associated with co-disposal of tailings (refer to Item 18.10.3.1), after allowing for a reduction in waste rock generated from a slightly reduced decline profile to that on which current resource consents were based. The updated technical reports supporting the ESIA have been obtained on this basis. They indicate that it is reasonable to assume that the variations to the resource consents to accommodate this additional capacity can be obtained. Specific issues associated with co-disposal of waste and flotation tailings in the waste rock dump are discussed in Item 18.10.3.1.

18.10.1 Mass Balance

A provisional mass balance has been developed indicating the quantities of concentrate and flotation tailings produced each year. The schedule of mine waste produced during drive development along with the quantities of tailings produced has been estimated in the base case production forecasts.

18.10.2 Waste Rock

Development waste from construction of the twin decline in the pre-production phase and vertical development during mining will be hauled to a Waste Rock Dump at surface, for co-disposal with flotation tailings. Waste generated during production will be retained in the stopes as backfill.

During the mining phase, the production of development waste is expected to be intermittent, based on a “just in time” vertical development mining strategy.

18.10.3 Tailings

18.10.3.1 Flotation Tailings

The flotation tailings will be thickened, and pumped to the tailings filter for further dewatering to produce a cake with less than 14% moisture content, enabling the filtered flotation tail to be reclaimed and transferred to the Waste Rock Dump, for encapsulation in the dump. The dump, including the tailings cell(s), is expected to be free-draining.

A number of additional studies are required to confirm the co-disposal strategy for flotation tailings:

- A detailed co-disposal study will be undertaken to design the waste rock dump to allow encapsulation of all flotation tailings produced during the operation of the plant. Flotation tailings are expected to match volumes of operating phase waste rock on a roughly 1:1 ratio. The periodic availability of waste during horizontal development will require planning of dump designs to minimise risks of dusting and runoff;
- Mobilisation of metals, in particular Arsenic and Antimony in the leach stage may increase levels of soluble metals in mill process water to levels above that acceptable for release. Compilation of a site wide water balance will allow modelling of water release rates and acceptable levels of metal ions in solution. Additional intensive leach testing would allow a better understanding of the ideal chemistry to minimise metal mobilisation or at least to characterise the levels to be expected and determine if a precipitation circuit will be needed for water treatment; and
- Tailings filtration capacity will have a significant impact on plant throughput with drops in filter availability directly impacting on plant utilisation and a requirement to meet a moisture level suitable for dry stacking or co-disposal. Scoping filtration tests have been undertaken by Outotec to allow sizing checks for a Larox PF style filter suitable to the duty to be made before detailed engineering will be undertaken.

18.10.3.2 Concentrate Tailings

Tailings from the detox circuit will be filtered to produce a cake with less than 14% moisture content. Filtered tailings will be available to mix on the surface with cement and transfer underground for use in the marker beds, as described in Item 16.5. Approximately 12,000t per year of tailings will be required for the marker beds and will consume all the leached concentrate tailings and a small portion of the flotation tailings. Sequestering the leached tailings in the cemented marker beds will likely provide a long term solution to minimise any AMD. Post closure, as the mine workings flood, the marker beds will be maintained under water in an anaerobic state to provide further long term protection from AMD.

18.10.4 Acid and Metalliferous Drainage

O'Kane Consultants (NZ) Ltd (OKC) were retained to complete a preliminary review of acid and metalliferous drainage (AMD) at the Blackwater Project. That review has been completed based on a limited amount of information, and further investigations will be required. In summary OKC have concluded that in general, assumptions made for the management of AMD, utilising the OceanaGold's experience at Globe Progress Mine near Reefton to predict water quality and the effects of acid and metalliferous drainage, appear reasonable.

Further work is required to characterise the rocks (and tailings) in regards to geochemistry and forecast water quality to reduce project uncertainties. Amongst other things, geochemical data is needed on the flotation tailings and concentrate tailings including acid base accounting and leach testing to derive potential contaminant loads. This information will be required to determine the stability of the tailings and waste rock under different environmental conditions.

The effects of AMD at Blackwater are likely to be minor, provided the potential risks are predicted and managed appropriately. Predictive test work is still required to confirm the conceptual approach.

Acid drainage from the Blackwater underground workings is potentially manageable due to the presence of carbonate minerals and hence elevated acid neutralising potential within the surrounding wall rock and waste rock.

Metalliferous drainage has potential to be an issue at the site, requiring management of contaminants such as As, Fe, SO_4 and other metals.

Given what is known about these risks, OceanaGold's concept proposals for a water treatment system to manage low dissolved oxygen mine waters elevated in As and Fe appear reasonable, subject to further work required to confirm the approach taken and ensuring that forecast water treatment requirements make adequate provision for additional load from the waste rock stack and any tailings. Management of the sludge produced needs further consideration

It should be assumed that any WRS at Blackwater will utilise paddock dumping to minimise longer term contaminant loads, consistent with the approach taken at the Globe Progress mine.

Pending further investigations, it should also be assumed that the waste rock pad used to elevate the site infrastructure and the waste rock stack itself will require a drainage system to direct to the basal drainage to the water treatment system, in order to mitigate any elevated Fe, As, and sulphate that may occur.

A waste rock management plan needs to be developed for the project that integrates tailings management, water management, treatment of water, and closure considerations

18.11 Mining Area Infrastructure

18.11.1 General

Mining area infrastructure includes the ROM pad, mine water pond, mine water treatment plant, storm water pond, clean water diversion channels and services that supply the underground operation.

The general arrangement of the mining area is shown in Figure 18-11.

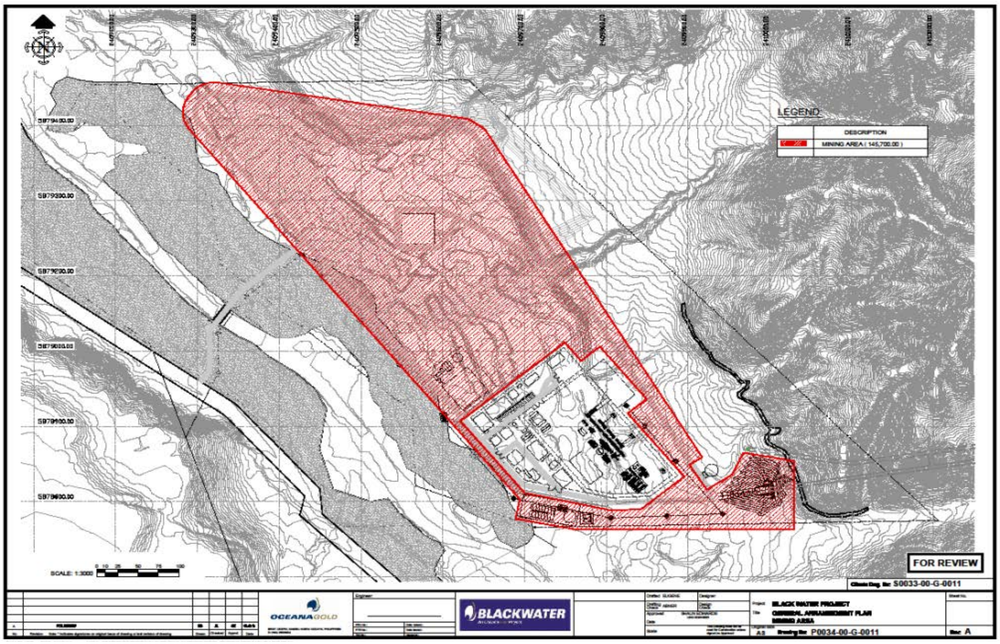

Figure 18-11: Mining Area Layout

The services considered in the infrastructure design include:

- Raw water;
- Potable water; and
- Electrical power (grid and backup).

18.11.2 Underground Mine Dewatering and Treatment

Based on the advice of Golder Associates (NZ) Limited, the initial required underground dewatering rate will be about 50l/s for a period of 6 months. This extraction rate will be required to dewater the old underground workings, estimated at 700,000m^3 of flooded voids, ready for underground development work. During normal operation the required dewatering rate will be expected to be <25l/s which will be made up of up to 10l/s ground water and up to 15l/s operational water.

Water samples from the historical workings indicate the water in the workings is likely to have elevated TSS, Arsenic and Iron levels making it unsuitable for direct release into the environment. Water from the underground operation will therefore be pumped into the mine water pond for sediment settling (TSS treatment) before being put through the Mine Water Treatment Plant (MWTP).

There is sufficient capacity on site for a 15m x 15m x 2m deep, lined mine water pond, which provides water storage capacity over and above that required by Golders' preliminary design. Based on 25l/s inflow rate, the pond will provide 5 hours retention for settling. A second pond may be required so that maintenance can be carried out on the primary pond.

Mine water will be extracted from the mine water pond and oily water will be separated for separate disposal. Discharge will be then put through the Mine Water Treatment Plant (MWTP). The flow rate will be adjusted to ensure the adequate level of pH treatment will be achieved. Treated discharge reports to the storm water pond for final dilution before release into a passive treatment wetland.

A design concept for a MWTP has been provided by Golders and has been tested in a small pilot study conducted as part of the site water management study. The process involves the neutralisation of the iron and arsenic with lime.

A 1km long clean water diversion channel will be constructed along the eastern and northern disturbance limit of the mine following the alignment of the haul road from the waste rock dump to the portal (Figure 18-1). The clean water diversion channel will divert the clean runoff from the northern and western Victoria Forest area around the site. The existing north south water race may be cleaned out and repaired. Any modifications to the water race will be undertaken in conjunction with Heritage New Zealand (the government authority charged with protecting historic places).

Runoff from the waste rock dump containing co-disposed tailings will need to be captured and options to either direct the water to the MWTP or to the storm water pond dependent upon water quality, will need to be available. Most other disturbed areas of the project area are intercepted by surface channels which report to the storm water pond. Surface water from site infrastructure such as vehicle servicing areas at the processing plant will be directed to the mine water pond.

The treated effluent discharge from the WWTP will be deposited into the storm water pond to maximise dilution before release into the wetland.

18.12 Risks and Opportunities

The following risks have been identified:

- geotechnical investigation of the main infrastructure areas is required;
- hydrogeological study of the site is required to confirm the raw water supply risk (capacity and quality);
- The current northern clean water diversion directs clean water runoff around the Blackwater Project site. The eastern discharge drains into an existing drainage channel on the Ngai Tahu land;

The following opportunities have been identified:

- Project management and engineering costs can be minimised by utilising OceanaGold Project Development Group (PDG) resources available in the Philippines;
- OceanaGold may consider constructing the process plant rather than buying packaged modular plant from Gekko;
- The requirement for surface storage magazine may be eliminated by using explosives stored at Reefton initially until an underground magazine is made available for storage of explosives;
- The high voltage and fibre communications extension to site could possibly be deferred to stage two if required; and
- The upgrade of the SH7/Snowy Road intersection and the installation of passing bays along the Snowy Road may be deferred later into the project if agreement can be reached with involved government bodies.

18.13 Further Study

The following areas have been identified for further study/investigation:

- Geotechnical investigation of the site; and
- Hydrogeological investigation of the site.

19 MARKET STUDIES AND CONTRACTS

19.1 Market studies

Markets for doré are readily available and the doré bars produced from the Blackwater Project could be sold on the spot market. Gold markets are considered mature, despite a current gold price that is lower than the 3-year trailing average. Gold hedging is an option.

19.2 Contracts

No contracts have been assigned considering the early stage of the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Based on technical reports and studies undertaken to support the consenting process to date, as updated to incorporate the addition of on-site ore processing and tailings disposal, the Blackwater Project will be consistent with regional and district plan objectives and will achieve compliance with appropriate monitoring standards for any adverse effects on the receiving environment. It will bring social and economic benefits to the local, West Coast and national economies.

The main environmental consents that are required to develop and operate the Blackwater Project are the regional and district council resource consents. These have already been granted based on a preliminary mine design that remains broadly applicable, with the exception of on-site processing and tails storage facilities which were not included in the resource consent application and will require changes to, or additional resource consents.

In the course of undertaking further geotechnical and site investigations the opportunity will be taken to undertake testing for existing contaminated ground as a precautionary measure.

20.1 Environmental and Social Impact Assessment

20.1.1 Environmental and Social impacts of the Blackwater Project

The Project will be consistent with the relevant objectives and policies of the district and regional plans, including those concerned with: Water, Solid and Hazardous Waste, Minerals, Air Quality, Poutini Ngāi Tahu, Soils and Rivers, Habitats and Landscapes, Natural Hazards, Heritage, Natural and Human Use Values, Surface Water Quantity, Ground Water, Hazardous Substances, Land Management, Lake and Riverbed Management, Sites Associated with Hazardous Substances and Contaminated Land, Odour and Dust. In summary, the following will be the environmental and social effects of the Project.

20.1.1.1 Quality of Receiving Waters

The company will design the Project, including:

- ore processing and cyanide detox facilities;
- co-disposal and encapsulation of flotation tailings in the waste rock stack;
- use and disposal of processed tailings in cement backfill and closed sections of the underground workings; and
- provision for sumps and wetlands

to meet appropriate water quality standards in receiving waters (the Snowy River), which are based on criteria to protect aquatic life in the sensitive river ecologies involved.

The mine water (both water from dewatering of the historical underground workings at Waiuta, and seepage from the ore-body during mine operations) can be treated for elevated iron and arsenic concentrations by encouraging the natural processes of ferrous iron oxidation. The cyanide detox circuit is also expected to precipitate some metals from the waste stream.

The process plant will be a closed water circuit with the only designed water discharges being in association with tailings disposal. Tailings from the flotation tailings thickener will be filtered to a cake with less than 14% moisture content and co-disposed in the waste rock stack. Tailings from the detox circuits will be dewatered and used in cement slurry in the underground workings or stored underground in closed workings.

The resulting water quality is predicted to be close to the existing water quality of the Snowy River and suitable for discharge.

20.1.1.2 Aquatic Ecology

The proposed water management systems will achieve two outcomes: the prevention of sediment laden water from entering the Snowy River, and the treatment of mine-affected water to a level that will ensure compliance with the existing water quality standards. These standards were derived in 2004 to protect ecological values of the Snowy River and remain appropriate.

20.1.1.3 Terrestrial Ecology

Mining facilities such as the waste rock dump, water storage pond, silt control drains, magazine and ancillary infrastructure will be constructed over a range of vegetation types from gorse scrub and rough exotic pasture to the sole native vegetation type of regenerating mixed beech forest. The Surface site is dominated by exotic plant species and is not considered to have significant conservation values.

Native vegetation clearance area will be up to 3.5 hectares out of about 13 hectares of this vegetation type existing at the site. This clearance will have a negligible effect on the ecological integrity of the surrounding area due to the very small size of clearance compared to the area of protected forest adjacent to the Surface Site (the Victoria Conservation Park of over 206,000 hectares).

Apart from vegetation clearance, likely impacts of the project on the ecological integrity of the surrounding area will be confined to site lighting, which will create an artificial extension of daylight hours for the bird populations using the area.

Both of these impacts are considered by OceanaGold's advisors to be insignificant. Any impact will be mitigated by a rigorous programme of weed control and restoration measures. The proposed native vegetation restoration programme will include beech forest restoration as well as incorporating wetland species appropriate to the area. This will increase the range of ecological niches at the site post closure, thereby enhancing the biodiversity values of the area.

National, regional and district planning objectives relating to significant indigenous vegetation and/or significant habitats for indigenous fauna do not apply to the project because the indigenous vegetation and fauna at the Surface Site are assessed as being not significant.

20.1.1.4 Visual Effects

Aspects of the project have the potential to be visible from the Snowy Road from adjoining areas of farmland to the west of the site, and from the lone farm house in the upper farmed extent of the valley. There are no other points or locations, such as walking tracks within Victoria Forest Park, from which the project will be visible.

The feature that may be most visible will be the waste rock dump. This will be mitigated during the operational phase by progressive rehabilitation and will be mostly hidden by the screening of the pine plantation that is at the Snowy Road site and on the immediately adjoining land between Snowy Road and the site. Apart from some minor clearance for location of site infrastructure the majority of the pine plantation is expected to remain in place for most of the life of the project. Tall, fast-growing vegetation will be planted to minimise the visual effect from the Snowy road.

During the closure phase all building and surface infrastructure will be removed and the only remaining features will be the waste rock dump and wetlands. Apart from the tall growing vegetation on the western side of the waste rock dump, the remaining parts of the waste rock dump will be restored using pasture grasses or native tree and shrub species.

20.1.1.5 Heritage

The Company will site Project elements to avoid compromising areas of greatest value and has adopted an approach, in consultation with the Heritage New Zealand, which seeks to protect in situ all existing heritage values and, at the completion of the Project, to remove all new infrastructure.

The site infrastructure as currently planned will not affect either of the two identified archaeological features: an historic water race and an adit. However, should the position of the portal or infrastructure be changed they may be affected. Therefore an authority to modify or destroy an archaeological site will be sought from the Heritage New Zealand as a contingency, and only activated if it is required.

20.1.1.6 Local Infrastructure Effects

OceanaGold will avoid imposing a financial burden on local infrastructure through its own financing of infrastructure improvements, namely:

- Meeting the cost of upgrading Snowy Road and the SH7-Snowy Rd intersection to provide safe and adequate access and cater for the minor increased usage caused by the development.
- Reducing any adverse effects of land clearance and construction of the waste rock dump by replanting progressively during mining and upon closure.
- Meeting the cost of providing an adequate supply of potable water for human consumption to the mine site for the duration of the project.
- Meeting the cost of an appropriate sewage system for the mine site for the duration of the project.

- Meeting the cost of sourcing raw water, through the installation of bores.
- Controlling storm water disposal at the mine site for the duration of the project.
- Meeting the cost of providing electricity supply to the mine site for buildings intended for human occupation.
- Meeting the cost of providing telephone links to and within the mine site.

20.1.1.7 Subsidence

The potential for subsidence was reviewed comprehensively in 2003 and this review has subsequently been updated to incorporate the current revision of the project. It was concluded that ground deformation (subsidence) from renewed mining followed by the decommissioning of the Blackwater Project would not have significant environmental effects.

20.1.1.8 Waste Rock Dump Stability

The design of the waste rock dump (WRD) will exceed the minimum factor of safety adopted as acceptable for design, set in the NZ Society Of Large Dams 2000 'Dam Safety Guidelines'. Liquefaction has been considered and preliminary analyses suggest that extensive liquefaction is unlikely in the area of the waste rock dump. During detailed design of the waste rock dump site specific geotechnical investigation will need to be carried out to confirm ground conditions, and hence liquefaction risk.

An application will be made to the Buller District Council for any building consent required for the WRD or any other structures.

20.1.1.9 Traffic Effects

The proposed Blackwater Project will generate a relatively low number of light and heavy vehicle trips on Snowy Road during the establishment and operation of the proposed activities. The current road network (SH7 and Snowy Road) is safely able to handle the increased volume of traffic given the improvements that the company proposes to make. With the proposed improvements to the road network, the traffic impact of the Blackwater Gold Project will be negligible.

20.1.1.10 Hazardous Substances

All hazardous substances and installations will be used and established in accordance with New Zealand, or relevant international, codes of practice or standards.

20.1.1.11 Gravel Extraction

Proposed consent conditions will mitigate any effects of this activity and will provide a comprehensive assurance that any adverse environmental effects can be adequately addressed. The Project is likely to require a lower volume of gravel than that allowed in the resource consents, with crushed waste rock used wherever possible.

20.1.1.12 Dust

The dust deposition from the proposed mining operation is expected to be less than that commonly experienced near unsealed roads with moderate to heavy traffic. Observation of farmland and native and exotic forests in such situations indicates very little effect on vegetation. Furthermore, with the high rainfall experienced in the mining area, any dust deposited on vegetation will be quickly washed off.

There is less knowledge about the effects of dust on birds. However, as the dust deposition levels due to the mining operation are expected to be less than some natural environments, the effects on birds are expected to be minimal.

20.1.1.13 Emissions to Air Generally

The proposed activity is considered to be consistent with the Resource Management Act (RMA) and the objectives and policies of the Air Plan. Overall the emissions generated from the revised Blackwater mining project are expected to be adequately mitigated and will not result in any adverse effects on the environment that are more than minor.

20.1.1.14 Blasting

A generic risk assessment has shown that all potential hazards or effects can be adequately managed. The blasting assessment does not identify any environmental impacts of the proposed blasting programme that are likely to cause adverse effects or discomfort to the single neighbouring house.

20.1.1.15 Noise

The main noise source from the development phase of mining activity will be construction of the bridge across Snowy River, particularly pile driving. The noise levels from piling have been assessed at properties nearest to the mine site and will comply with the limits set out in NZS 6803:1999 Acoustics – Construction Noise, with a large factor of safety at all times.

During the operational phase the main noise from surface work at the mine site will be the placement of waste rock from the mine in the waste rock dump, and processing of ore. Noise levels from these activities have been assessed at nearby properties and the Waiuta site which is frequented by visitors. Noise levels will be at or below the background sound in the area on a calm day.

20.1.1.16 Contribution to Tourism

The Blackwater Project has the potential to add to the 'story' told by the tour operators of historical, current and future mining of the Reefton Goldfields.

20.1.1.17 Community Sponsorship Programmes

In connection with its Globe Progress mine, OceanaGold currently provides financial support to a number of initiatives at the community level. During 2013-14, about $43,000 worth of grants were distributed to community groups including the Westland Mountain Bike Club, Friends Of Waiuta, Reefton Rodeo Club, Inangahua A&P Show, Reefton Workingmens' Club, Reefton Golf Club, Reefton Rugby Club, Reefton Junior All Blacks Rugby Club, the Kahuna Board Riders Club, Reefton Historic Trust Board, Eagles Golfing Society of Buller-Westland, West Coast Boys Hockey Primary A Team, Reefton Trotting Club "Reefton – Heart of Gold" Heart Stopper Challenge Team, DARE West Coast Incorporated, Reefton Youth Centre, Reefton Four Square (products for x2 raffles), Ikamatua Golf Club, Grey District Council, Academy School Books, Tourism Horizons Limited, and Reefton Bowling Club.

In addition, OceanaGold has committed $105,000 to the development of a new sports stadium for Greymouth to be known as the Westland Recreation Centre, $35,000 of which was paid in 2013.

The Project will help to underpin the continuation of OceanaGold's community grants programme.

20.1.1.18 Other Socioeconomic Benefits

The Blackwater Project will contribute to the "social fabric" of the Reefton, Buller and West Coast communities via staff, contractors and their families belonging to service clubs, sports clubs and other voluntary organizations. As well as fulfilling leadership roles and making other contributions within the community, the Project staff, contractors and their families will help to provide the critical mass to underpin the ongoing sustainability of the area.

20.1.1.19 Other National Economics Benefits

The Blackwater tenement is subject to the 1996 minerals programme and as such will generate a Crown royalty of 1% (ad valorem) or 5% (accounting profits) whichever is greatest. When profitable, the Project will also generate additional corporate income tax payments. To the extent that the Project leads to an overall increase in national employment the Government will receive additional income tax payments.

20.1.1.20 Summary of Positive Effects

The current proposal will have the following positive effects:

- It will provide economic benefit to the district and region via increased employment opportunities, wages/salaries and expenditure and associated economic welfare enhancing benefits associated with increased (or retained) levels of economic activity. It may contribute to tourism, provide other socioeconomic benefits such as staff contributions to the 'social fabric' of Reefton and underpin the continuation of OceanaGold's community grants programme.

- Creation of a wetland will improve water quality from surface water flows. Currently contaminated water from an adit linked to the historical underground workings runs across Department of Conservation (DOC) land to Snowy River. That runoff across DOC land will be avoided since water from the portal will be directed to the wetland and polished via natural processes before it flows to the river.
- OceanaGold will manage noxious plant species and replace them with pasture and other vegetation appropriate to the location.

20.2 Statutory Requirements for Environmental Consents

20.2.1 Background

This section discusses the requirement for resource consents under the Resource Management Act (RMA), New Zealand's principal environmental protection law. For the Project, a land use consent, discharge permits and water permits are required from the West Coast Regional Council (WCRC) and land use consent(s) are required from the Buller District Council (BDC).

The process of applying for resource consents includes a preparation phase where information is gathered about the proposed activities and their potential effects on environmental and social values, followed by lodgement of the application and an Assessment of Environmental Effects (AEE). The AEE is required to describe, in addition to the likely and potential effects of the proposed activities for which consent is sought, the available measures to avoid, remedy or mitigate any adverse effects. The councils assess the completeness of the application and if it is found to be incomplete will request further information from the applicant. Once the application is deemed to be 'complete', the processing of the application by the councils begins.

Based on their assessment of the potential effects of the activities proposed on the environment and on other parties, the councils decide whether the application is to be non-notified, limited notified (notified to only those parties directly affected by the application) or publicly notified. The consent of any landowner on whose land the proposed activities will take place is generally required as a matter of course, unless there will be no effects on them. Where an application is publicly notified, any member of the public may make a submission supporting or opposing the application and may ask that the council holds a public hearing to determine the outcome of the application. The decision to grant or decline a resource consent may be appealed, in whole or part, by the applicant or any person who lodged a submission (confined to the matters on which that person submitted).

The notification classification and the views of any subsequent submitters will affect the processing timeframes for the application, which can run from about a month for very simple non-notified applications with effects that are "less than minor" to 9 months or more for significant applications with material effects on third parties, environmental or social values. Any application to vary existing resource consent is processed in the same way and is subject to similar timeframes. An appeal from a first-instance decision to grant or decline resource consent or a variation to resource consent will extend those timeframes further.

20.2.2 Previous Consents

The Project was granted a suite of resource consents in 2004 from the West Coast Regional Council (WCRC) and the Buller District Council (BDC) to create a decline and establish surface infrastructure for mining at a DOC owned site adjacent to the Surface Site. These consents were rendered obsolete by the passage of time and/or the change in the location of the Surface Site.

20.2.3 Current Consents

In March 2013, a new resource consent application was made to the WCRC and BDC for the Project at its new surface location, (refer to Item 4.1 of this report for a discussion of surface rights). The redefined project brought other changes, chiefly including a larger waste rock dump and the avoidance of any surface disturbance at the Prohibition Shaft site or Waiuta Township. Consents were issued in May and July 2014.

Specifically, the resource consents as currently issued encompass four main components:

- Construction of a twin-tunnel decline from land at the Snowy River (the Surface Site), including establishment of surface mine infrastructure: water management structures, access roading (including a bridge across the Snowy River), a waste rock dump, provision of power and telecommunications, mine related buildings, a dangerous goods store, and some refurbishment of Snowy Road;
- Exploration drilling, trial mining, and dewatering of the historic workings at Waiuta;
- Mining of the ore-body including sorting and screening of the ore at the Surface Site, and transportation of the ore for off-site processing at either the Reefton mine or at the Macraes mine; and
- Mine closure including the removal of surface infrastructure, decommissioning sediment ponds to stock water dams, rehabilitation of any disturbed areas to their pre-mining state. The constructed wetland will remain.

The surface infrastructure layout for the project, for which resource consents were issued in May and July 2014, is shown in Figure 20-1.

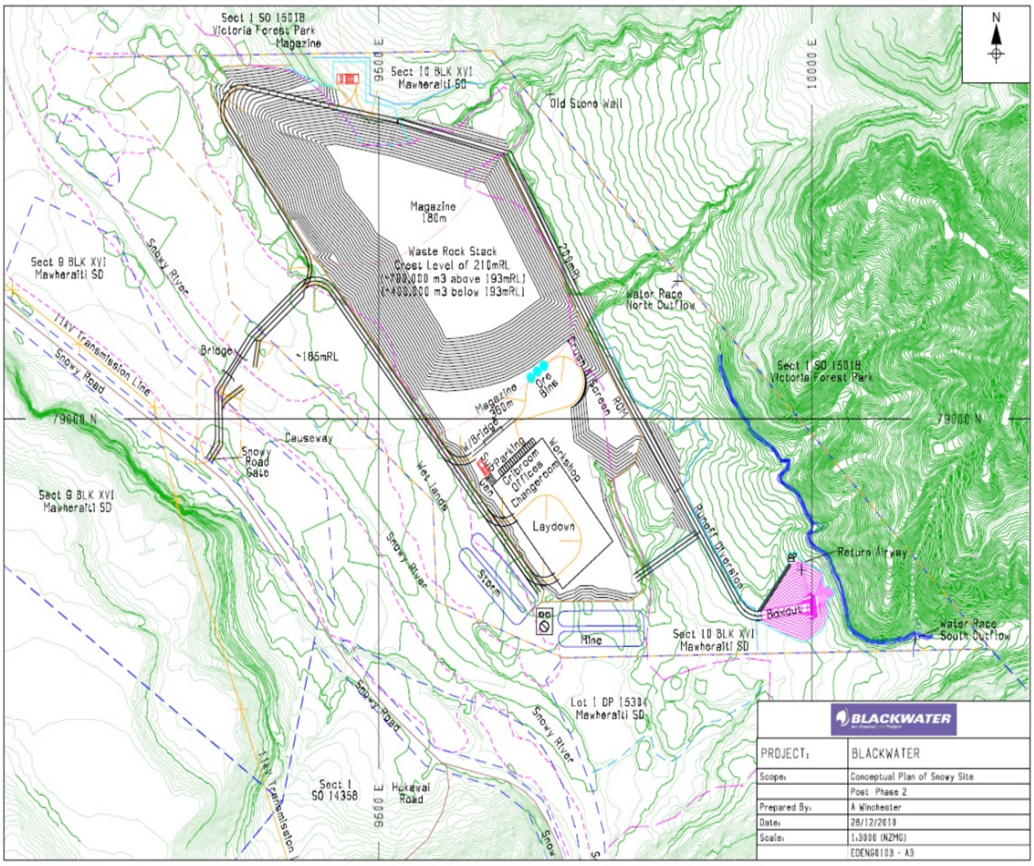

Figure 20-1: Surface Infrastructure Layout – March 2013

The application was notified on a limited basis to a number of affected parties, including local landowners, Iwi, DOC and Land Transport New Zealand. The company was able to obtain the written approval to the issue of consents from all of the affected parties, in some cases conditional on the payment of compensation (refer to Item 20.2.6). The Buller District Council and West Coast Regional Council have subsequently granted Land Use Consents in May 2014 and July 2014 respectively.

20.2.4 Future Consents

Variations to the current resource consents or new resource consents will be required to accommodate an on-site processing plant and tailings disposal method.

The revised surface infrastructure layout for the project is shown in Figure 20-2.

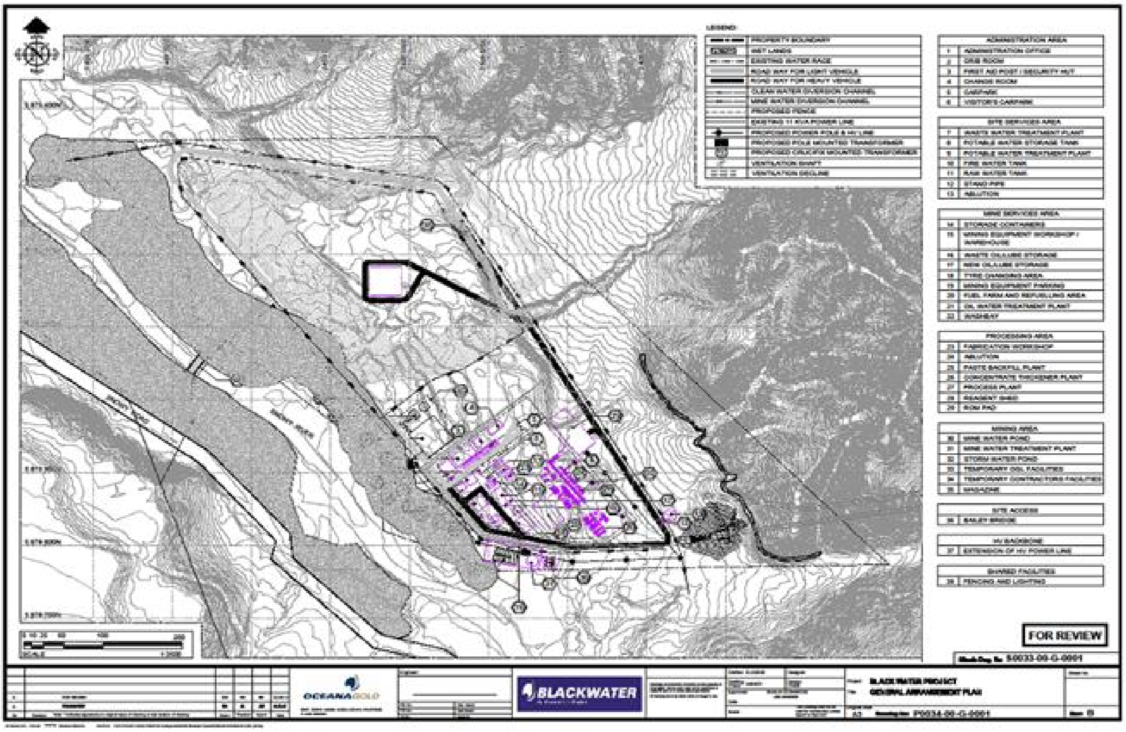

Figure 20-2: Surface Infrastructure Layout – April 2014.

OceanaGold has updated its ESIA to include on-site ore processing and tailings deposition and the updated technical assessments that OceanaGold has obtained to support that ESIA, while recommending further detailed investigations in some instances, appear to confirm that ore processing and tailings deposition will not result in unacceptable risk or adverse effects in terms of fundamental matters such as geotechnical stability, water and air quality, and ecology. OceanaGold is not aware of any reason why the technical assessments would indicate any significant risks or adverse effects. On that basis, the appropriate consent variations and / or new consents are considered to be attainable.

20.2.5 Consented Activities

The suite of resource consents for which the councils have granted their approval comprises those listed in Table 20-1 and Table 20-2.

Table 20-1: Buller District Council Land Use Consents RC130025

Type	Purpose
Approved Activity	To develop and operate an underground gold mine targeting the Birthday Reef below the abandoned Waiuta Township.
Land Use Consent	To develop and operate an underground gold mine targeting the Birthday Reef and associated works.
Land Use Consent	For Vegetation Clearance and incidental earthworks (excluding wetland) from 0.5ha up to 5ha per site, in total, over any continuous three year period. Modify riparian margin within 10m of the Snowy River riverbank for construction and use of Bridge across Snowy River to Blackwater Mine Site. Construct road on legal road reserve to link Snowy Road to access bridge and road improvements including the addition of passing bays and widening (for mitigation purposes).

Table 20-2: West Coast Regional Council Consents

Type	Purpose
Land Use Consent	To undertake land surface disturbance and earthworks associated with the construction, use, maintenance and rehabilitation of the access roads, and haul roads and a bridge over the Snowy River (including undertaking works in the riparian margins and bed of the Snowy River and the placement and maintenance of protection works at the abutments), install culverts, disturb the bed of an unnamed tributary and erect structures in the tributary. Cut and fill and undertake earthworks to create the mine site at the Surface Site, including construction, use, maintenance and rehabilitation of diversion drains.
Land Use Consent	To undertake land surface disturbance and earthworks associated with the construction, use, maintenance and rehabilitation of temporary and permanent silt ponds, sumps, bunds and treatment wetland.
Land Use Consent	To undertake vegetation clearance in the riparian margin of the Snowy River associated with the construction, use, maintenance and rehabilitation of the Blackwater Project including construction of infrastructure (including but not limited to pipelines and utilities), roads and a bridge over the Snowy River at or about map reference NZMS 260 K31 099 786.
Land Use Consent	To construct the Decline and undertake associated earthworks, at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786) in straight line to map reference NZTopo50 BT21 030 175 (NZMS 260 L31 134 792).
Land Use Consent	To extract gravel from the dry bed of the Snowy River.
Land Use Consent	To disturb the bed of the Snowy River for geotechnical testing and construction and use of a bridge over the Snowy River.
Water Permit	To divert stormwater around disturbed areas to silt ponds and to divert clean stormwater runoff from undisturbed areas to local surface drainage channels to minimise silt control requirements, at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).
Water Permit	To recycle surface water and groundwater for mine operational purposes, at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).
Water Permit	To take groundwater from the Decline for dewatering purposes (to maintain dry working conditions in the shaft and underground workings), at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).
Water Permit	To take water for use in mining, for dust control and for domestic purposes from the Snowy River.
Water Permit	To divert water for the purpose of constructing a bridge across the Snowy River.
Discharge Permit	To discharge surface water, groundwater and contaminants to land at the Surface Site (being water associated with drilling, underground operations, decline development, stormwater from the portal area, waste rock dump, ore stockpile and infrastructure area) in circumstances that will result in that water and contaminant entering the Snowy River at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).
Discharge Permit	To discharge up to 1.1M cubic metres of waste rock to land at the Surface Site at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).
Discharge Permit	Onsite discharge of sewage and grey water treatment overflow at the Surface Site at or about map reference NZTopo50 BT21 998 174 (NZMS 260 K31 099 786).

20.2.6 Compensation

A range of compensation measures have been agreed with various parties as part of securing affected party approvals for the issue of resource consents or as a result of arrangements dating back to the 2004 application where these are effectively re-triggered by the current consents. Generally the obligation to undertake the various measures agreed is conditional on the Project proceeding.

Consent conditions around the monitoring of conditions affecting aquatic life in the Snowy River have been agreed with Fish and Game.

Consent conditions have been agreed with Land Transport New Zealand (LTNZ), requiring the Snowy Road intersection to be widened in accordance with NZ Transport Agency rural road intersection rules, realignment of the centre of Snowy Road, full seal widening in accordance with NZ Transport Agency rural road rules to be undertaken on State Highway 7 south and north of the Snowy Road intersection and widening of the Snowy Road approach to the intersection with State highway 7 at an estimated cost of $200,000.

20.2.7 Other Environmental Consents

At this stage the company has not identified any archaeological sites which would be affected by the proposed Blackwater Project, for which authorities would be required from the Heritage New Zealand. However refer to Item 4.1.3.2 - Other Environmental Permits for a further discussion of the statutory requirement for authorities, should these be required, and for approvals issued under hazardous substances legislation.

20.3 Environmental Bonds

The company will be required to provide bank guarantees to secure the obligations to comply with the conditions of the resource consents, including remediation of the Surface Site. Refer to Item 4.1.3.3 of this report for a further discussion of the implications of this requirement.

21 CAPITAL AND OPERATING COSTS[7]

21.1 Capital Costs

The base case scenario identifies that two and a half years of “pre-production” are required to establish the access decline and initial underground exploration drilling platform. Capital expenditure in the first two years (the only years in which there is no mining of ore material) is estimated to be US$76M, and sustaining life of mine capital is estimated to be US$78M, including 15% contingency – refer to Table 1-3. Capital cost estimates are ±25% and assume that the mobile mining fleet is purchased rather than leased.

The US$76M cost estimate includes the cost estimate for the pre-production mining, surface infrastructure, process plant and contingency. Expenditure on resource definition diamond drilling is incurred from the third year onwards, once underground drilling platforms have been established. Total life-of-project resource definition capital expenditure has been estimated to be US$9M (plus contingency), and is included in the life-of-mine sustaining capital total of US$78.1M shown in Table 21-2.

To reflect uncertainty associated with the estimation of capital, a range of likely pre-production capital costs has been assessed. The low and high cases reported in Table 21-1 are based on ±30% range limits.

[7] The capital and operating expenditure discussed in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

Table 21-1: Summary Pre-Production Capex (US$M)

Item	US$ Millions		
	Base	+30%	-30%
Infrastructure	8	10	6
Processing	21	27	16
Mining	30	39	23
Management & Indirects	4	5	3
Operational Readiness	3	4	3
Contingency (15%)	10	13	8
Totals	**76**	**98**	**58**

Table 21-2: Base Case Pre-Production Capital Cost Summary (US$M)

Description	Year 1 ($M)	Year 2 ($M)
Processing Capex:		
Infrastructure & Power	4.1	2.1
Ore Processing		20.5
Total	**4.1**	**22.6**
Indirect:		
Engineering & Design	1.8	
Operational Readiness		3.1
Commissioning		0.3
Management & Indirects		3.7
Total	**1.8**	**7.1**
Underground Mining Capex:		
Development	10.1	10.4
Mobile Equipment	6.2	
Electrical Equipment	1.4	0.4
Infrastructure		0.5
Other	0.2	1.1
Total	**17.9**	**12.3**
Pre-production Capital Total:	**23.8**	**42.0**
Contingency Factor @ 15%	3.6	6.3
Sub Total per annum	27.4	48.3
Total Pre-Production Capital		**75.7**
LOM Sustaining Capital Total		**78.1**
Total Project Capital		**153.7**

21.1.1 Basis of Estimate

Capital costs used in this PEA were derived from a variety of sources including but not limited to comparative analysis of other operations, derivation from first principles, equipment quotes and factoring from other costs contained within this study. The cost estimates are for the base case scenario discussed throughout this report.

The accuracy of the estimates contained within this study varies due to the different methods of derivation used to estimate the costs; the capital costs are expected to be within a ±25% range.

All costs are expressed in first quarter (Q1) 2014 United States (USD) dollars. No allowance has been included for escalation, interest or financing fees or duties.

The capital costs are divided into five areas:

- Surface infrastructure and power costs;
- Ore processing costs;
- Underground mining costs;
- Contingency costs; and
- Indirect costs.

21.1.2 Surface Infrastructure and Power Costs

Blackwater initial direct civil infrastructure capital costs amount to $6.2M. This covers the infrastructure and facilities required to support the mine/mill operations including site preparation, civil work, services, roads, explosive facilities and electrical substation.

Power supplies in the region are sufficient for Project requirements, and no provision for additional power line construction has been included other than the power line from the Snowy River Road into the site and distribution throughout the site as detailed in Item 18.

21.1.3 Ore Processing Costs

Blackwater initial direct process plant capital costs amount to $20.5M. This covers all the process equipment and structures from mills to tailing filters, as well as pre-treatment and refinery facilities.

21.1.4 Underground Mining Costs

Blackwater underground mining capital costs for the life of the mine amount to $98M. Mining direct capital costs include pre-production mining, capital development costs, mine mobile equipment, and mine infrastructure. Development to be completed prior to the commencement of production was classified as pre-production (capitalised) development, including labour.

This development was assumed to be undertaken by contract mining crews with unit costs as shown in Item 21.2. A database of budget costs was used for mine equipment, mining infrastructure and fixed service equipment. Where necessary, the budget costs were factored to reflect current costs.

Capital requirements were allocated to both pre-production years, but during more detailed studies consideration should be given to allocating the capital requirements over additional years, as it is likely that payments for equipment will be required prior to the equipment being delivered to site unless a fleet lease option is adopted.

21.1.5 Contingency Costs

Blackwater contingency costs amount to $20M, and account for unforeseen costs within the project scope. Contingency costs were calculated using a factor of 15% of civil infrastructure, process plant, and mine infrastructure direct capital costs.

The contingency factors are considered appropriate for the level of engineering work performed in the preparation of this Report. Input variables used in calculating the contingency are a result of information gathered from previous projects and industry standards.

21.1.6 Indirect Costs

Indirect costs amount to $9M and include the following pre-production related expenditure.

- Engineering & design;
- Operational readiness;
- Commissioning; and
- Management & Indirects.

21.2 Operating Costs

Operating costs have been estimated using first principles derived from supplier quotations and/or benchmark data from OceanaGold and other similar operations. Site General and Administrative (Site G&A) costs refer to site-wide operational expenses rather that to expenses that can be directly attributed to functional departments.

Table 21-3 details the total life of mine operating costs in US$M and also a breakdown by cost per tonne of ore. Mining is inclusive of operating development, but does not include capital costs. The low and high cases in Table 21-3 are based on +/-30% range, to reflect the early stage of the project.

Table 21-3: Operating Cost Inputs

Item	Base Case US$M	Base Case US$/t Ore	+30% US$/t Ore	-30% US$/t Ore
Operating Costs				
Mining[1]	180	154	200	118
Processing	49	42	55	32
Site G&A	22	19	24	14
Selling Costs	3	2	3	2
TOTALS	**254**	**217**	**282**	**167**

21.2.1 Ore Processing Costs

The unit operating cost for ore processing at Blackwater is expected to be approximately US$42/t, with a breakdown provided in Figure 21-1. Low and high cases are detailed in Table 21-3.

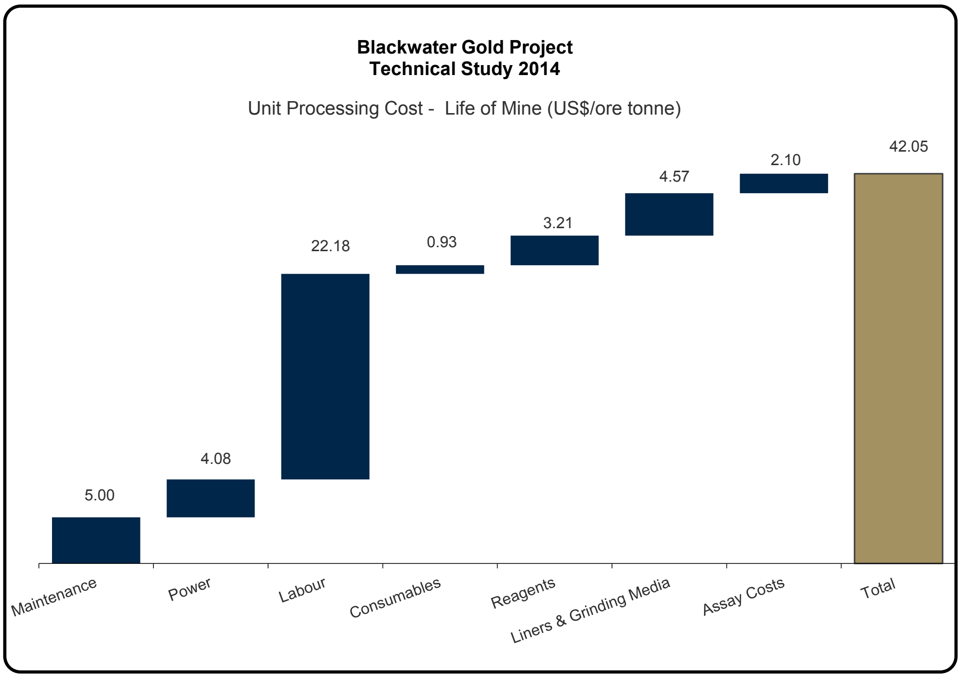

Figure 21-1: Operating Cost - Ore Processing

21.2.2 Mining Costs

The unit operating cost for mining at Blackwater is expected to be approximately US$154/t ore. The activity based analysis is reported from life of mine averages and summarised in Figure 21-2. Low and high cases are detailed in Table 21-3.

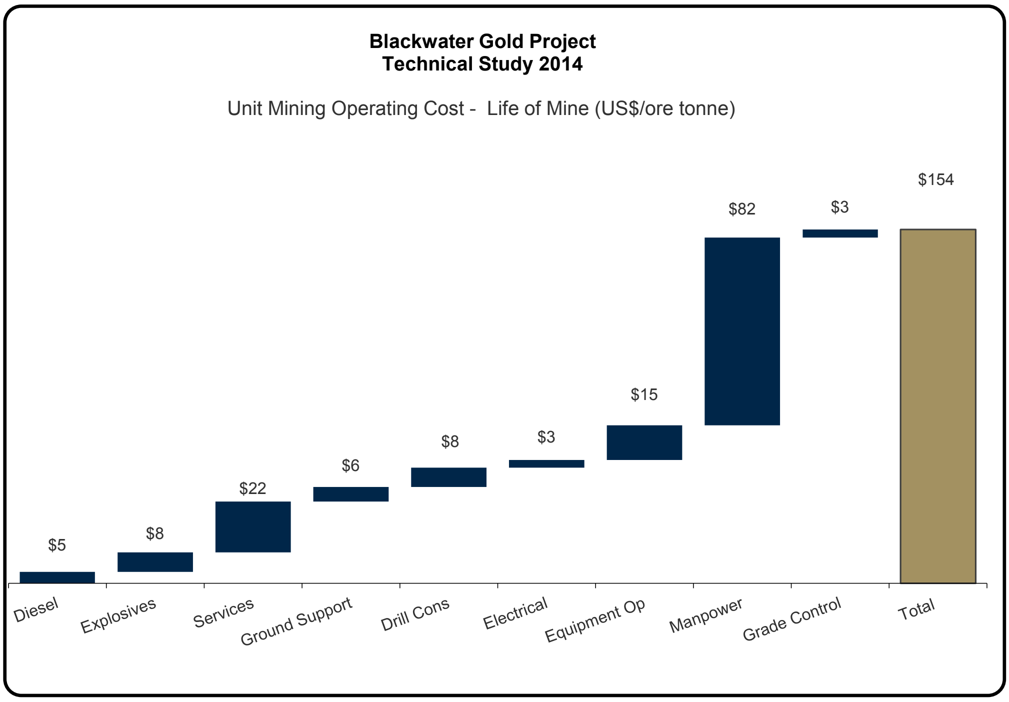

Figure 21-2: Operating Cost – Mining

22 ECONOMIC ANALYSIS[8]

22.1 Economic Assumptions

The principal assumptions and inputs used in this economic evaluation are listed in Table 22-1. The entire project is intended to be internally funded by OceanaGold cashflow, and 5% is the OceanaGold internal hurdle rate of return when assessing investment opportunities.

Table 22-1: Economic Model Parameters

Description	Value
Currency of economic model	USD
Preproduction period	2.5 Yrs.
Mine life (excluding pre-commercial production)	10 Yrs.
Available operating days per year	350 d/yr.
Discount rate (Mid period discounting)	5%
Base case gold price	US$1,300/oz.
Power cost	US$0.08/kWhr
Diesel cost	US$0.82/litre

[8] The financial analysis discussed in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

22.2 Cash Flow Forecast

22.2.1 Pre-tax Cash Flow

Development capital causes free cash flow to fluctuate every second year, as seen in Figure 22-1 and Figure 22-2.

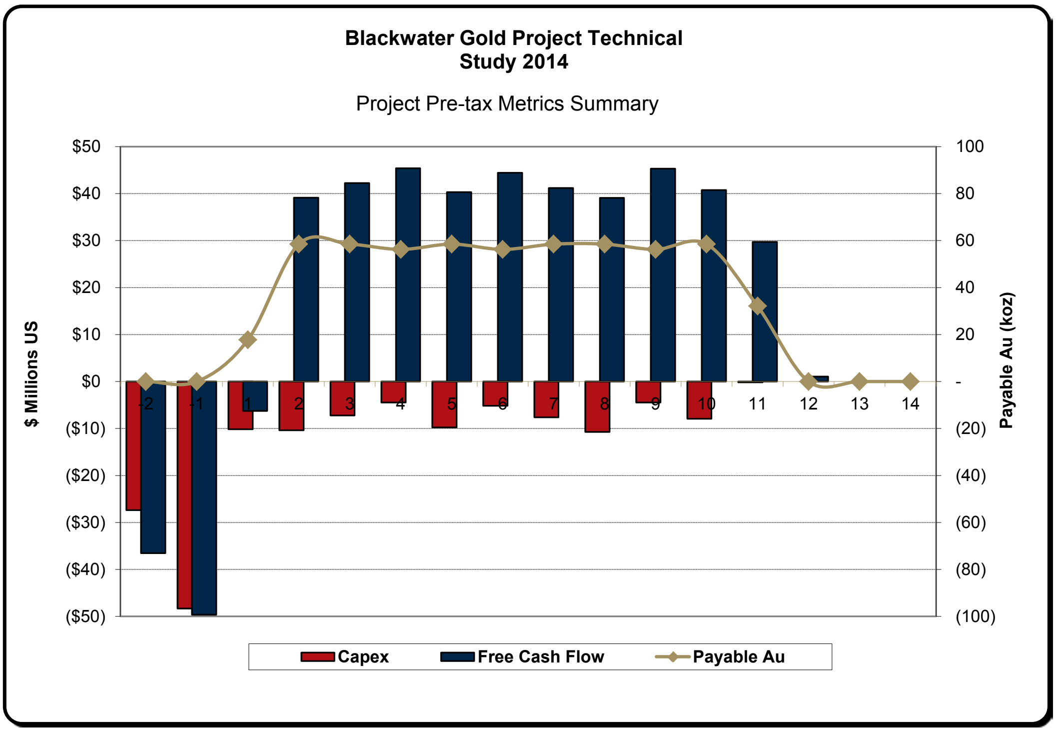

Figure 22-1: Gold Production and Pre-Tax Cash Flow

22.2.2 After-tax Cash Flow

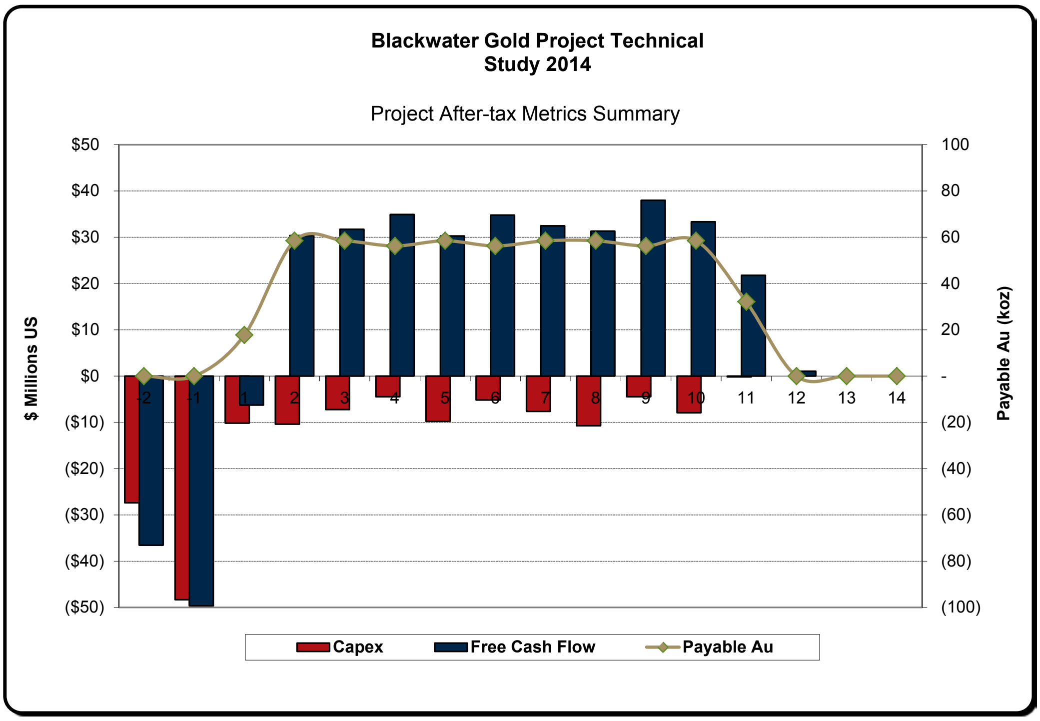

Figure 22-2: Gold Production and After-Tax Cash Flow

22.3 Financial Analysis

The economic evaluation for the base case indicates positive free cash flow, resulting in a base case after tax Net Present Value of $132M at a discount rate of 5%. Refer also to Table 22-4 for an assessment of the range of NPV values that were determined through flexing the width and grade of the Birthday Reef. The base case scenario demonstrates an after-tax internal rate of return (IRR) of 23%.

Based on the assumptions applied OceanaGold will be looking to achieve an all-in sustaining cost of approximately $600/oz for the base case scenario.

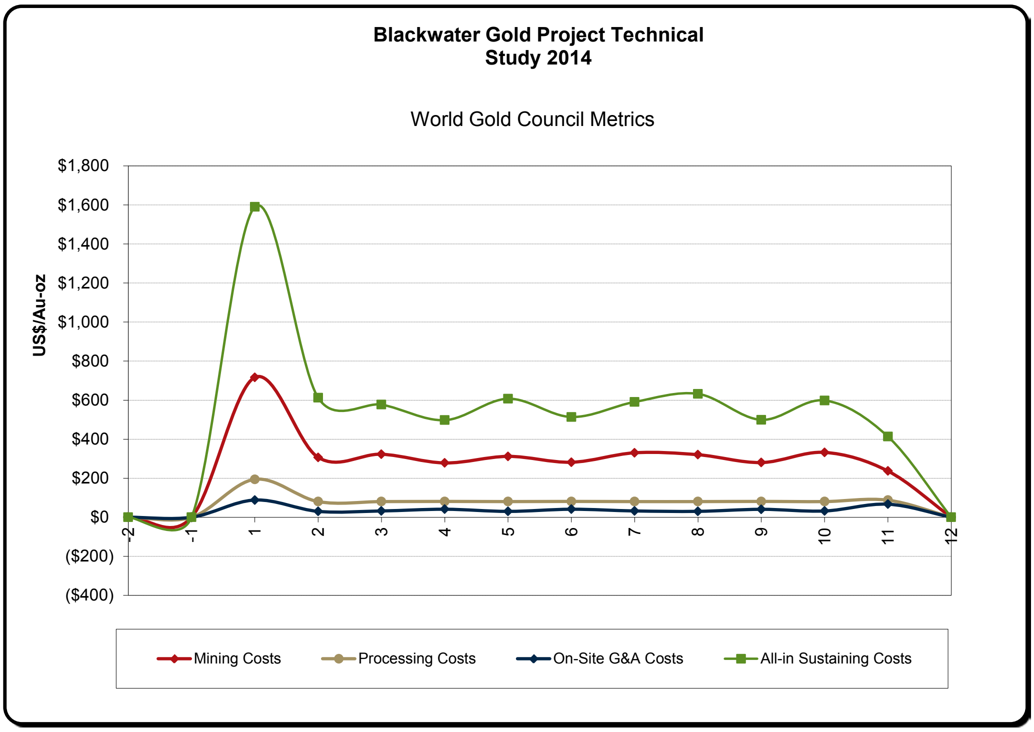

Figure 22-3: World Gold Council Metrics

22.4 Taxes and Royalties

The project does not include any existing tax losses accrued for Blackwater. It is possible that tax losses will be available to off-set what OceanaGold decides to invest in the exploration decline.

Table 22-2: Tax and Royalty Assumptions

Item	Key Criterion/Principal Assumptions
Crown Royalties	1% NSR during life of mine
Royalco Royalty	$9M
Depreciation method	Units of Production
Forward Appropriation	Utilized
Corporate Income Tax rate	28%
Escalation	Costs based on Q1 2013 in real terms; prices based on Q3 2013 real terms

22.5 Sensitivity Analysis

Deterministic sensitivity analysis has been assessed for the base case. Grade, total capex and total opex were flexed separately while holding all other factors constant. The results of the sensitivity are presented in Table 22-3, Figure 22-4 and Figure 22-5.

This data suggests that the project is robust, with the base case returning positive post-tax NPV ($40M) at revenue forecasts 25% less than estimated in the study.

Furthermore the evaluation suggests that the project could sustain significant increases in both capital and operating expenditure. The data reported in the sensitivity analysis is post-tax.

Table 22-3: Deterministic Sensitivity Data for NPV and IRR (post-tax)

NPV (US$million) @ 5%	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Revenues	40	59	77	95	114	132	150	169	187	205	223
Capital Costs	157	152	147	142	137	132	127	122	117	112	107
Operating Costs	164	158	151	145	138	132	126	119	113	106	100

IRR	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Revenues	11%	14%	16%	19%	21%	23%	25%	27%	29%	31%	32%
Capital Costs	30%	28%	27%	25%	24%	23%	22%	21%	20%	19%	18%
Operating Costs	27%	26%	25%	24%	24%	23%	22%	21%	21%	20%	19%

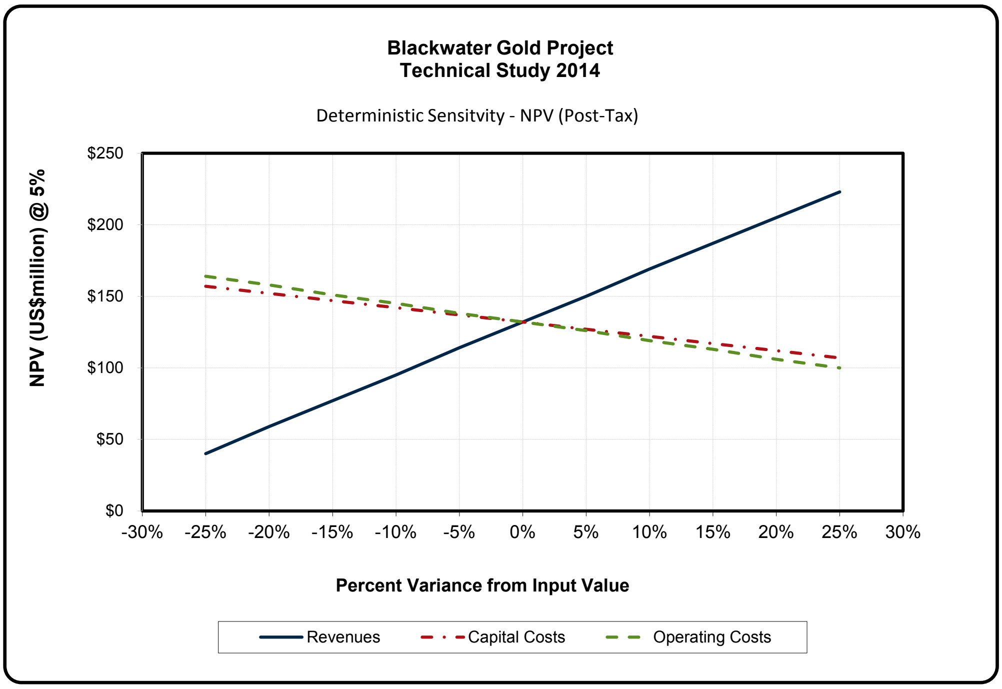

Figure 22-4: Deterministic Sensitivity Graph – NPV @ 5%

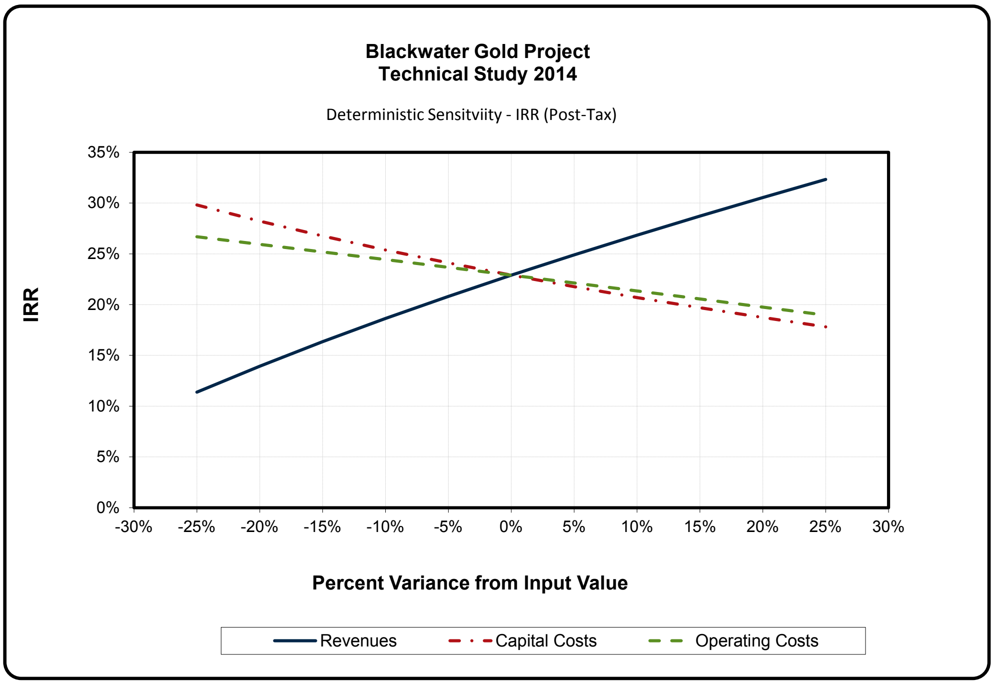

Figure 22-5: Deterministic Sensitivity Graph – IRR

Gold price scenarios for cumulative NPV outcomes are reported in Figure 22-6. OceanaGold's low case gold price (Base -20%, US$1040) and base case geological assumptions suggests an NPV of $58M.

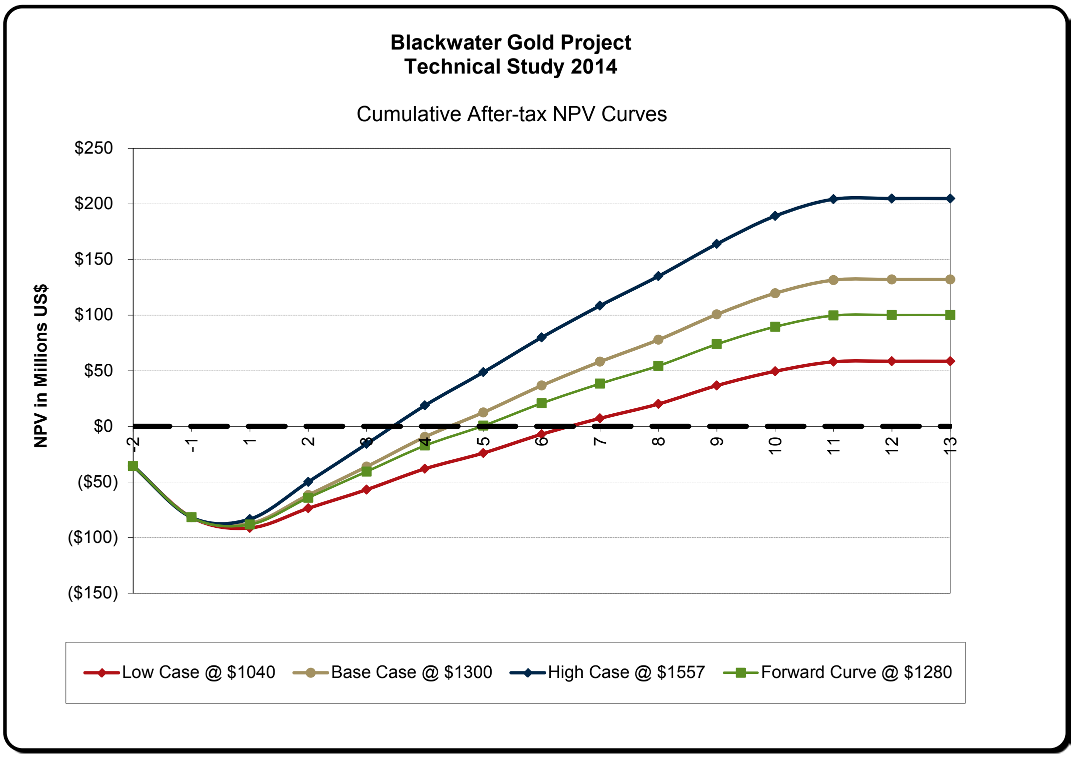

Figure 22-6: Cumulative After Tax NPV Curves @ 5% Discount Rate

In Figure 22-7, the base case NPV is reported for a range of discount rates.

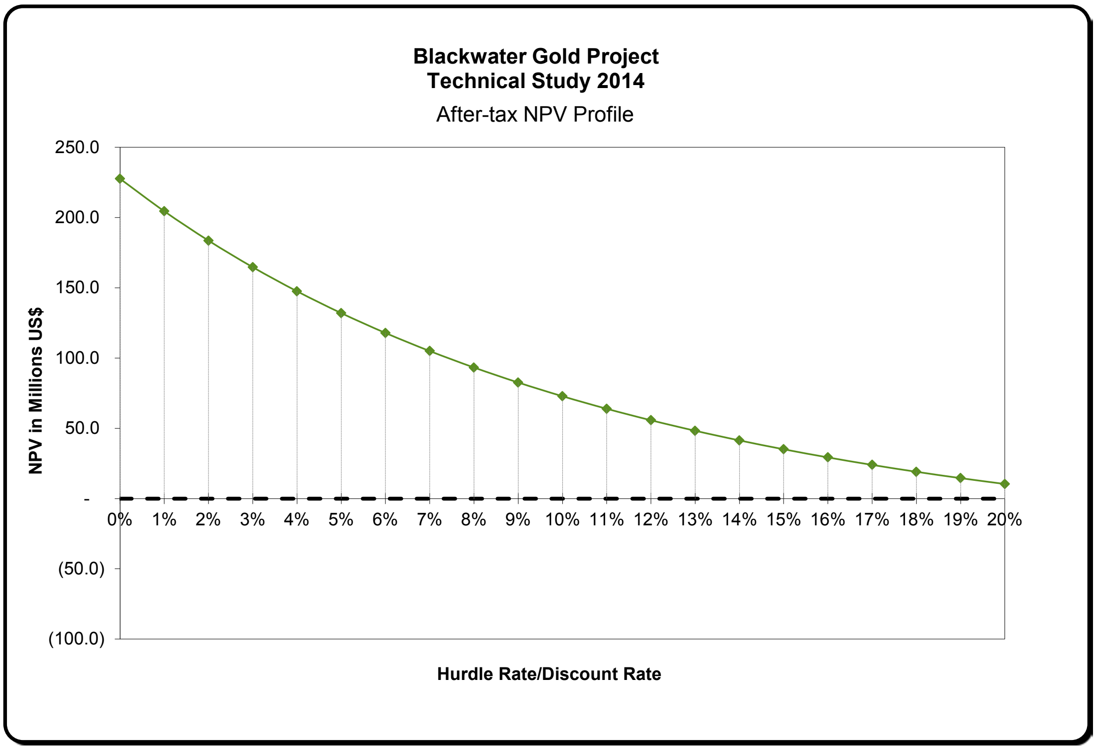

Figure 22-7: Discount Rate Sensitivity

As previously highlighted, the primary risk to the success of the Project is the variability of reef width and grade, plus the current limited knowledge surrounding ground conditions. To quantify the possible impact a sensitivity analysis was undertaken whereby the reef thickness and reef grade were flexed by ±15% and ±30%, to determine the impact on the Project NPV. The maximum impact on contained metal modelled is approximately ±30%, whether by flexing only width, only grade, or a combination of both. This range is considered the likely maximum variability that could be encountered when undertaking extraction of the Birthday Reef. The results are presented in Table 22-4.

Table 22-4: NPV Array – Flexing Reef Width and Grade

NPV ARRAY			Reef Thickness {Diluted Thickness} (m)				
			-30%	-15%	Base	15%	30%
			0.48 {0.75}	0.58 {0.87}	0.68 {1.00}	0.78 {1.17}	0.88 {1.30}
Reef Grade (g/t)	-30%	16			21		
	-15%	20		46	83	105	
	Base	23	42	90	132	159	199
	15%	26		133	180	212	
	30%	30			243		

Examination of Table 22-4 shows that NPV remains positive over the full range of reef width and grade scenarios modelled. It should be noted that within any of the modelled scenarios the grade and width were maintained as constants. In reality both width and grade will vary during the course of mining but with the current level of information available it is not possible to model such scenarios.

23 ADJACENT PROPERTIES

There are no adjacent properties that impact on the potential merit of the Blackwater Project.

24 OTHER RELEVANT DATA AND INFORMATION

OceanaGold is not aware of any other relevant data or required information for inclusion to make the report more understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS[9]

OceanaGold has delineated a significant gold project near the town of Reefton in the Buller District of the west coast of the South Island of New Zealand. 100% interest in the project is held through their wholly owned subsidiary OceanaGold (New Zealand) Limited (OceanaGold NZL).

OceanaGold has secured the required option agreements to acquire the surface land for the project and site access across privately owned land. Access across the Snowy River will require a Crown easement and a local authority licence. It is reasonable to assume these can be obtained.

The Blackwater Project has very good access and supporting infrastructure. The deposit is amenable to conventional underground mining methods with estimated mining recoveries of approximately 90%. It is expected that any future mining operations will be able to be conducted year-round. Process test work has shown gold recovery of 96% is possible using the treatment process outlined in the report.

The project has a ten year mine life after two and a half years of pre-production, and capital costs could be paid back within six years. The project is robust, returning positive post-tax NPV results over a range of inputs.

OceanaGold has been pro-active with regard to environmental and socioeconomic issues. Environmental monitoring, baseline studies and site investigations have been ongoing at the Blackwater Project site. Additional environmental baseline programmes are expected to continue as required through 2014. Consultation to date has included meetings with local councils and informal discussions with local land-owners.

Variations to resource consents will be required before the conclusion of the exploration decline to allow for ore processing and tailings co-disposal facilities.

The production target is currently based 100% on Inferred Mineral Resources and it is technically not possible to progress the project without investment in an exploration decline. The decline is required for the purposes of data collection for resource definition, metallurgical test-work, geotechnical assessment and groundwater management without which the project cannot advance.

There is extensive information from the historical Blackwater mine located directly above the Inferred Resource. This information along with OceanaGold's recent deep drilling programme provides sufficient confidence to justify investment in the exploration decline.

In the context of OceanaGold's ongoing operations in New Zealand and the Philippines the investment requirement for the exploration decline and associated works is not material.

25.1 Opportunities and Risks

The opportunities to improve the project include:

- Resource growth – the Birthday Reef at the Blackwater Project is currently open at depth and to the north. Future drilling may grow the resource beyond that which is currently estimated. Resource definition drilling to test the extension at depth and to the north has been included in the financial assessment of the project.

[9] The scheduled production volumes and grade and financial analysis discussed in this report and forming the basis for the conclusions in this Item must be read in conjunction with the cautionary statement on page 3, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

- Geotechnical review – advanced studies to identify and understand ground conditions which could allow an increase in the size of stable stope spans. Trial mining will assist with developing this understanding.
- Project management – utilisation of OceanaGold's Project Development Group (PDG) resources available in the Philippines.
- Process plant – OceanaGold have identified a number of initiatives to improve the process plant design.

The key risk factors are:

- This Technical Report (the "PEA") is preliminary in nature. The PEA includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realised;
- Geological continuity - fault disruptions / offsets of the reef are expected to be common and these may elude the proposed resource drilling. The development of a HW drive prior to ore drive development will provide a drilling platform for closer spaced drilling to confirm the reef position and test for any offsets. There is still a small risk that the HW drive itself could be wrongly located if the offsets are significant;
- Vertical continuity - the 40-50m vertical spacing of historical level data provides little information about the vertical geometric regularity of the Birthday Reef (historical sample data suggests that down-plunge grade continuity will be reasonable). However, the proposed mining method with vertical advance of 2m for each slice and opportunity for backs mapping with each slice will give good mining control. Infill drilling from the HW drive could also be used to understand complex areas or locate the reef if lost during stoping;
- Potential for refractory gold – no refractory nature has been observed in the sulphides tested to date although it is thought that a roaster was used at the Prohibition mill to treat flotation concentrates prior to vat leaching. It is possible that low levels of Stibnite may have been present in some areas, slowing the leaching process down and causing the implementation of the roaster (very important at the nearby Globe mine). The coarser grind of the older battery and reported problems with classification may also have led to the previous circuit configuration inclusive of roaster. If the refractory nature of the sulphide associated gold changes the direct leach plan may not yield the high recoveries expected and the Project would need to consider campaigning the leach tails through the existing autoclave at Macraes Gold Mine;
- Variability in ore hardness - inherent risks to meeting production schedules from increased rock competency and mechanical availability are offset with the required mill utilisation of approximately 52% with availability from the roster system of 71%. Additional hours are available to cater for lower throughput rates from ore hardness issues and to accommodate a lower targeted mechanical utilisation to reduce capital cost;
- Acid and metalliferous drainage – long-term AMD or heavy metal release from tailings will be addressed by segregating detoxified leach tailings to be used in the cement slurry for placed marker beds during the stoping process, and also can be stored in abandoned hangingwall drives underground. In the current plan all of the concentrate tailings containing the majority of the sulphides in the ore will be disposed of underground. Sequestering this component underground should eliminate any issues in the surface waste dump and post closure flooding by ground water should ensure a long term anaerobic environment. If co-disposal with surface waste is employed then this risk will need to be mediated;
- Additional mining dilution - where back or wall conditions are poor, significant fall-off can be expected during and after firing the ore from the de-stressed hanging wall and footwall of the 'slot', resulting in higher dilution. The actual outcome will depend on the amount of structure and its orientation, the condition of the joints and the effectiveness of the back support;
- Mining stresses – mining at depths between 800m and 1,600m below surface is likely to be challenging. Considering the rockmass conditions and the expected stresses, adverse ground behaviour including high deformation or 'squeezing' behaviour should be expected. The mining sequence will need to be considered to avoid exacerbating the stress conditions;
- Site infrastructure – No geotechnical investigation of the main infrastructure areas has been undertaken;

- A detailed hydrogeological study of the site has not been undertaken to confirm the raw water supply risk (capacity and quality); and
- Further work is required to characterise the rocks (and tailings) in regards to geochemistry and forecast water quality to reduce project uncertainties. This includes geochemical data on the flotation tailings and concentrate tailings including acid base accounting and leach testing to derive potential contaminant loads together with other investigations recommended by O'Kane Consultants (NZ) Ltd, who were retained to complete a preliminary review of AMD for the purposes of this PEA.

26 RECOMMENDATIONS

The results from this PEA demonstrate that the Blackwater Project is likely to be technically and economically viable and it is recommended that OceanaGold continue to advance the project by construction of an exploration decline for the purposes of further data collection. The costs of constructing the decline are discussed in Item 16 and Item 21.

26.1 Recommended Works Programme

26.1.1 Geology

The study recommends:

- Resource definition drilling collared from a dedicated drill drive at the end of an exploration decline. This will facilitate the upgrade of as much resource as possible from the Inferred Resource category to the Indicated Resource category, while continuing to increase the resource base laterally and at depth;
- Undertake a void drilling programme to determine the extents of the bottom level of the old workings, once a safe working procedure has been developed to mitigate against the risks associated with intersecting the flooded historical mine workings;
- Structural modelling to determine the offsets of the orebody; and
- Investigate the northern and southern extents of the orebody to determine whether the ore-body is structurally bound or continues beyond the resource shape.

26.1.2 Mining and Geotechnical

The study recommends:

- Geotechnical drilling programme for the boxcut;
    - Review the boxcut design and location. A deeper boxcut may be required if the fresh rock cover is insufficient to excavate the boxcut. Another option may be to move the boxcut location closer towards the water race to improve the quality of the rock cover; and
    - Review the ground support regime of the portal entrance. Steel sets and shotcrete may be required to provide long term support. Armco tunnelling may be required if the base of oxidisation is deeper than expected.
- Geotechnical logging of resource definition drill holes completed from underground, with associated rock testing and stress measurements. Collation of data required to improve confidence in determination of an appropriate mining method and extraction sequence;
- Review the proposed capital mine design layout once a block model and structural data set is available. This will assist in optimising the stand-off distance between the capital decline development and the stoping zone;
- Optimise the ore drive designs once any structural ore-body offsets are quantified. This may determine the start position of the cross cutting access from the decline development;
- In conjunction with further geotechnical analysis, determine the optimum level spacing and sill pillar dimensions, particularly if ground conditions are determined to be less favourable than is currently assumed;
- An assessment of an appropriate swell factor assumption for air-leg blasting of the host greywacke sandstone that will constitute fill within the stopes;

- Determination of a cut-off grade philosophy once a block model has been created, to assist with optimising the mine design;
- Determine the exploration decline excavation approach, whether it be owner-operator or a contract such as a fixed and variable, or a schedule of rates;
- Investigate opportunities to dewater the old workings above the current resource shape and opportunities to remnant mine the lower levels of the old workings; and
- Numerical modelling of the proposed extraction sequence with an appropriate stress analysis program.

26.1.3 Hydrology and Hydrogeology

The most critical knowledge gaps associated with the Blackwater project are associated with dewatering the historical underground workings. The study recommends:

- Undertake high flow gauging on Snowy River;
- Confirm hydrogeological conditions including the expected pressure difference and discuss with drilling experts;
- Directly sample and analyse water held within the historical underground workings;
- Undertake settling column testing of representative sediment laden mine water samples;
- A workshop that includes a Failure Mode and Effects Analysis for the mine plan, including closure considerations; and
- The development of a hydrologic and geochemical conceptual model for the site considering determined contaminant loads from mine water, waste rock dumps, and co-disposed flotation tailings, including development of a Waste Rock Management Plan including:
    - Development of a waste rock schedule for planning purposes such as construction and scheduling of the WRD;
    - Development of a conceptual waste rock stack construction plan that incorporates the flotation tails impoundment;
    - Integration of the waste rock geochemistry (quality and quantity) and forecasts (longevity) into the water management plan; and
    - Closure and consideration of ongoing passive treatment based on forecasts and longevity and maintenance of the passive treatment system.

26.1.4 Processing

In order to reduce risk in design and scale up, additional test work on samples of the reef collected from underground resource definition drilling is recommended including:

- Collection of a representative sample of reef material that will be mined and processed by the Blackwater plant, from underground diamond drilling following excavation of the exploration decline;
- Additional intensive leach tests on gravity/flotation concentrate to confirm parameters of the resin upgrade circuit prior to electrowinning as part of full processing at the Blackwater site;
- Assessment of the impact of a flotation concentrate cleaning stage to minimise the amount of leached material on concentrate recovery and mass pull;
- Variability testing using reef sample material to confirm design criteria of the final flowsheet, and to confirm recovery assumptions; and
- Mineralogy to be done on flotation concentrate from the reef samples, to understand how gold is locked up and to determine if product is refractory. i.e. requiring treatment through an autoclave.

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APPENDIX 1 - JORC CODE, 2012 EDITION TABLE 1

JORC Code, 2012 Edition – Table 1 report Blackwater Inferred Resource

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• Diamond drilling was used to obtain drill core samples that were collected at geologically defined intervals from sawn, and in some cases cleaved HQ and NQ size intervals. If samples were suspected to contain coarse grained gold the samples were assayed by screen fire assay otherwise the samples were fire assayed. The screen fire assay method involved pulverising 1kg of the sample then seizing that sample through a 100 um. There were then two portions of the sample, the >100 um fraction and the <100 um fraction with both fractions assayed. This procedure was used to quantify the coarse gold nature of the mineralization. Two quartz flushes were inserted between each sample and also underwent screen fire analysis. Coarse blanks were also inserted after each mineralized quartz vein. • Historical Blackwater Mine face and stope samples were used for comparative estimates of the reef but were not ultimately used as a basis for the reported resource estimate. No detailed sampling descriptions could be located, given that more than 60 years have passed since the stope and face samples were collected. The sample data used was as recorded on long sections.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Triple tube PQ, HQ and NQ diamond drilling was completed to obtain core samples. Excluding holes drilled prior to WA20, core was oriented using the Reflex ACT orientation tool. Downhole surveys were completed on all the holes with the most recent holes (WA20 onwards) being surveyed using the Reflex EZ-Trac.

Criteria	JORC Code explanation	Commentary
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• Diamond core recoveries were monitored throughout the program comparing core recovered with hole depth. Sample recovery was maximized by running a constantly monitored bentonite based mud program.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Diamond holes were both geo-technically and geologically logged in their entirety and logged to sub-meter detail where mineralization, alteration, lithology and geotechnical changes dictated. Core was photographed and stored in an enclosed shed.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Half core was taken for assay. Samples sent for assay were a minimum of 30cm for HQ, 40cm NQ and not greater than 1 m. Before sampling a cutting/sampling line is drawn on the core down the centre of the mineralised ellipse (if visible) to give a guide of where to cut to ensure a representative sample. • Core was split in half using either a diamond core saw or a cleaver. A cleaver was used when the material being sample was soft / unconsolidated (e.g. pug) or when it was thought that sawing may have an impact on obtaining a representative sample. • Before sampling the core, saw or cleaver used to split core was thoroughly cleaned between each sample. • In each case every piece of core that was split/cut in half had an equal chance of making it into the sample bag. A coin was flipped to ensure the random nature of this selection. • Laboratory duplicates (ie splits prior to pulverizing) showed good agreement except in one case; one screen fire check assay from WA25 did return a significantly higher assay than standard fire assay, due to the presence of coarse free gold in this sample, which was identified visually prior to assay. The drill hole assays themselves are not directly used for resource grade estimation.

Criteria	JORC Code explanation	Commentary
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• Diamond core submissions included a minimum of two blanks, one standard and at least one lab duplicate taken after coarse crushing of the sample. • 50g fire assay was completed on samples • Samples that were suspected to have or contained fine to coarse visible gold were sent to ALS Townsville. Submissions to ALS Townsville contained a minimum of two blanks, and one standard. Where intervals contained or were suspected to contain fine to coarse visible gold, each sample was followed with two quartz flushes. • On return of assay results, standard data was analysed and any failure of standards within a batch (i.e. standard results greater or less than two standard deviations from the certified standard value) were noted. • It was determined that reassay was not required for any of the batches submitted for assay.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Sampling was observed by at least two OceanaGold personnel to ensure protocols were followed and the samples were sealed and dispatched to laboratory for assay. • No holes were twinned, however daughter holes only reached a maximum of 10m separation from the parent, refer to the table in section 2 below for the down hole intercepts. • Assay results were disseminated from the lab to at least 4 company personnel with at least one if not two situated outside the Reefton Goldfield. Logging of geological information was completed directly into a laptop and that data was then uploaded into an aQuire database. Validation protocols are setup in aQuire to insure that the data is entered correctly. Assay results were directly imported into acQuire and then significant assays results were correlated back to the core to ensure depths and widths corresponded. • There was no adjustment to the assay data.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• Drill holes collar were surveyed using DGPS to a sub-centimeter accuracy using an independent contract surveyor. Except for drillholes WA10 and WA11 where collars were picked up by an independent contract surveyor but reported to 1 m accuracy. • New Zealand Map Grid was used.

Criteria	JORC Code explanation	Commentary
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• The projection of the Birthday Reef (ie the resource) has been intersected by four drill holes (and their daughter holes). The parent drill holes are typically separated by 400m. It is the strike and plunge extent and geological continuity demonstrated from the historical production data, combined with the predictability of the depth of drill hole intersections of the reef, that provide a compelling case for classifying the resource under JORC 2012. The geological evidence of the projected resource is sufficient to imply but not verify geological and grade continuity. On this basis, the Blackwater estimate is classified as an Inferred Mineral Resource. The resource was extrapolated 100m below the deepest drill hole intersection (WA22) on the south west corner of the resource. The north east corner of the resource was excluded, but the resource was extrapolated approximately 200m down plunge towards the north east corner. Approximately 15% of the resource is therefore extrapolated beyond actual sample locations. • The drill hole samples were not composited. • The face and stope samples presented on long sections from the historically mined reef were in some cases averaged (ie single grades and widths in some cases represented grades and widths averaged over a number of adjacent samples. In these cases, the original individual sample data were not available). Note however, that the resource grade estimate was not directly based on this sample data. The estimated average reef width was however based on this averaged data. It is believed that the overall variability of the reef width may be under-estimated. However the estimate of the average reef width is believed to be appropriate.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Intercepts were obtained that are oblique to the orientation of the mineralized vein therefore estimates of the true widths of the intercepts are based on the interpreted orientation of the vein from the known pierce points and from structural measurements taken from the oriented core.
Sample security	• *The measures taken to ensure sample security.*	• Samples were stored in the secure Hattie Street Office/ core shed until shipped to Australia for assay using local courier and FedEx.

Criteria	JORC Code explanation	Commentary
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• No external audits or reviews were completed on the sampling technique. Internal discussions were had with experienced OceanaGold personnel prior to sampling to establish documented procedures that were followed during the course of sampling.

Section 2 Reporting of Exploration Results

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

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	The Blackwater Mine is in the Grey District of the west coast of the South Island of New Zealand, 37km south of Reefton (by road) and 60km northeast of Greymouth. The mine is located in the abandoned township of Waiuta. Rights to prospect explore or mine for minerals owned by the Crown are granted by permits issued under the CMA. Crown-owned minerals include all naturally occurring gold and silver. The project is located within exploration permit EP40 542, covering an area of 4,308 hectares. OceanaGold has entered into an option agreement to acquire the land required for surface infrastructure works, which expires in April 2016. OceanaGold has received resource consent for the construction of an exploration decline and associated infrastructure works. Specific building permits will be required for site works related to the exploration decline. A variation to the resource consent is required to permit on site ore processing and co-disposal of filtered tailings. OceanaGold holds sufficient rights in the Blackwater Project and the main mining and environmental permits required to: • Acquire the necessary land access rights; • Undertake exploration activities and in due course, if those activities establish a suitable Mineral Resource secure a Mining Permit (Note: ordinarily an Indicated Mineral Resource is required to advance to a Mining Permit. Subsequent to completion of the exploration decline, and infill drilling from this access, OceanaGold

Criteria	JORC Code explanation	Commentary
		anticipate defining an Indicated Mineral Resource); • Construct the proposed Exploration Decline and undertake exploration drilling and mining in compliance with environmental laws. Prior to construction of an ore processing plant resource consents will be required to accommodate on-site processing and tailings storage facilities. The updated ESIA has not identified any reason why these additional facilities, provided they are appropriately managed, would not receive resource consents. Whilst there is currently no planned surface expression other than on the Surface Site, which is on land controlled by OceanaGold, any ventilation rise or other aspect of the workings day-lighting beyond the boundaries of the Surface Site, that may become necessary at any stage, these will require the relevant landowner's consent and environmental permits. Any surface ventilation work will have both a visual and vegetation impact. In 2004 as part of a previous attempt to reopen the mine, two ventilation shafts received permits and consents from the landholders. Third party rights to receive an interest in the project are confined to Crown royalties and royalties payable to Royalco Resources Limited. In both cases the royalties are fixed and quantifiable for the purposes of inclusion in the business plan. The underground workings of the proposed Blackwater mine will pass through land owned by various parties, including Crown land administered by the Minister of Lands on behalf of the Crown, public conservation land owned by the Minister of Conservation and land in private ownership. The law governing the requirement, if any, for landowner consent to mine under the surface of land is found in the Crown Minerals Act 1991. Under that Act OceanaGold will not require any access arrangements with the owners of the land through which the Blackwater Mine underground workings pass. The Blackwater EP is now in its 12th year, with a current 4 year term for appraisal purposes that runs through to 18 November 2016. The Crown Minerals Act 1991 allows a single further extension of the EP of up to 4 years for appraisal purposes, if certain conditions are met. Provided the permit remains in good standing (principally requiring the payment of annual fees and completion of work programme commitments), and assuming OceanaGold's exploration activities delineate the resource to the satisfaction of the Minister for Energy and Resources (ordinarily, for this purpose, an Indicated Mineral Resource will be required), OceanaGold has

Criteria	JORC Code explanation	Commentary
		a statutory right (section 32(3) of the Crown Minerals Act 1991), in priority and to the exclusion of all other parties, prior to the expiry of EP40542, to surrender the permit in exchange for a mining permit. The 2013 Minerals Programme (available at http://www.nzpam.govt.nz/cms/pdf-library/minerals-legislation/) governs the circumstances under which a mining permit is issued. The main set of criteria is as follows: *10.1* *(3)* *The Minister will ordinarily grant a mining permit if satisfied that:* *(a) the permit applicant has identified and delineated at least an indicated mineable mineral resource or exploitable mineral deposit, and* *(b) the area of the permit is appropriate, and* *(c) the objective of the mining permit is to economically deplete the mineable mineral resource or deposit to the maximum extent practicable in accordance with good industry practice.* The word “ordinarily” is intended to leave a discretion that allows the Minister of Energy and Resources to take into account a range of factors, as well as general discretion, as follows: *10.2 Matters that may be considered by Minister* *(1) In considering whether a mineral deposit has been sufficiently delineated to support the granting of a mining permit, or in assessing any proposed work programme30 (or modifiedwork programme), the Minister will ordinarily consider (but is not limited to) any or all of the following matters:* *(a) the geology and occurrences of minerals within the land to which the mining permit application (or application for extension of duration) relates* *(b) the applicant’s knowledge of the geology and extent of the mineral resource that the applicant proposes to extract* *(c) estimates of mineable mineral resources, which may include indicated and measured resources, probable and proved reserves, and the accompanying documentation on input*

Criteria	JORC Code explanation	Commentary
		data, methodology, quality control and validation of the mineral resource estimates *(d) inferred mineral resources* *(e) the applicant's mining feasibility studies, which include mine design, scheduling and production, resource recovery, and economic viability* *(f) project economics – in particular the financial viability and technical constraints, and the proposed level of expenditure in relation to the scale and extent of the proposed operations* *(g) whether the proposed mining operations are in accordance with good industry practice.* Once a mining permit is obtained, OceanaGold will be authorised to commercially extract the gold resource, subject to the conditions attending to the mining permit. A mining permit (MP) may be issued for a maximum period of 40 years. The Blackwater EP is currently in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Numerous parties have completed development studies and mineral resource estimates on the Blackwater Mine; Carpentaria Exploration Company 1975, James Askew Associates 1987, CRA Exploration1987, GRD Macraes 1991, Emperor Gold Mining 1992, Gemell Mining Engineers, John Dunlop and Associates 2002. CRAE and GRD Macraes both completed exploration drilling targeting the Blackwater workings and continuation of the Reef.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• Gold mineralisation at the Blackwater Mine is hosted within a quartz vein where about 70-80% of the gold is present as native gold, commonly occurring on the laminated host rock inclusions, with the remainder occurring as refractory gold locked in the lattice of arsenopyrite and pyrite. Surrounding the deposit is Ordovician Greenland Group rocks comprising an inter-bedded sequence of massive, jointed quartzose greywacke and indurated argillite. The mineralization is interpreted as Orogenic.

Criteria	JORC Code explanation	Commentary
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* o *easting and northing of the drill hole collar* o *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* o *dip and azimuth of the hole* o *down hole length and interception depth* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	

Hole ID	NZMG Coordinate*		Elevation (masl)	Hole Orientation		Daughter Depth Start (m)	Final Depth (m)	M Drilled
	East	North		Azimuth (Grid)	Dip			
WA10	2412835	5879174	438	90	-90		686.9	687
WA11	2412829	5879172	438	90	-65		1,171	1171
WA11A						644.4	1,011	527
WA21	2412888	5879439	528.681	83.5	-63.5		1378	1,378
WA21A						1,264.3	1324	60
WA22	2412888	5879439	528.68	65	-56		1121	1,122
WA22A						809.2	847	38
WA22B						815.2	863	47
WA22C						814.3	1675	861
WA22D						1,385.9	1641	255
WA23	2413278	5880086	540	143	-55		36	36
WA24	2413278	5880086	540	143.5	-51.5		363	364
WA25	2413278	5880086	540	140	-62		1282	1,282
WA25A						1036.2	1205	169
* GPS co-ordinates							**Total**	**7,997**

Criteria	JORC Code explanation	Commentary
		(table below)

Hole ID	From (m)	To (m)	Intercept (m)	True Width (m)	Grade (Au g/T)	Grade Width (g*m)	Comment
WA11	979.6	980.3	0.7	0.5	24.50	12.3	Parent Hole
WA11A	980.3	981.0	0.7	0.5	59.70	29.9	Daughter Hole
WA21A	1,315.9	1,316.8	0.9	0.5	23.30	11.7	Daughter Hole
WA22C	1,632.30	1,633.0	0.70	0.5	15.65	7.8	Parent Hole
WA22D	1,623.90	1,625.03	1.13	1.0	85.2	85.2	Daughter Hole
WA25	1,118.95	1,119.40	0.45	*0.35	31.8	11.1	Parent Hole
WA25	1,134.18	1,134.59	0.41	*0.3	62.4	18.7	Parent Hole
WA25	1,190.77	1,191.36	0.59	0.5	3.91	1.9	Parent Hole (BR)
WA25A	1,136.40	1,137.11	0.71	*0.5	134.00	67.0	Daughter Hole
WA25A	1,195.20	1,195.65	0.45	^0.4	61.90	24.7	Daughter Hole (BR)

** Indicates the upper intercept in each of the holes WA25 & WA25A interpreted as a fault repetition of the Birthday Reef. (BR) indicates the Birthday Reef intercept. ^ Unorientated drill core. True width calculated using WA25 intercept.*

Criteria	JORC Code explanation	Commentary
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal*	• Sample widths were constrained by the width of the quartz lode. No top cutting, no aggregation, or weighted averaging was applied. Au mineralization outside the quartz lode was minimal and not included in the resource calculation.

Criteria	JORC Code explanation	Commentary
	equivalent values should be clearly stated.	
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• True widths of the intercepts reported in the above table are based on the interpreted orientation of the vein from the known pierce points and from structural measurements taken from the oriented core.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Refer to the NI 43-101 Preliminary Economic Assessment Technical Report Blackwater Gold Project Reefton, Westland Province, New Zealand dated September 2014. Figures 10.2 and 16.2 present plan and sectional views respectively of drill hole locations.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Exploration results from diamond drilling associated with the Blackwater Mine are reported in the table above.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Samples of mineralised quartz were hand-picked from the surface of the Prohibition waste dump for initial laboratory testing. A bulk sample of 450kg of mineralised quartz material was recovered from adjacent to the original mine adit tramway between the mine and the Snowy River battery and used for the recent laboratory program to validate the recovery assumptions in the design criteria and replicate earlier test work on fresh core.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• Step out drilling was completed both north and south of the mineralization looking for extensions to mineralization along strike. To-date no significant mineralization has been defined outside of the identified mineralization. More work is recommended but has not yet been costed.

Section 3 Estimation and Reporting of Mineral Resources

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

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	• Drill hole data is entered via an AcQuire database interface which includes validation prompts. • Personnel are well trained and routinely check source versus input data during the entry process. • Mr J Moore believes that the personnel, systems, training and software in place are to industry standard.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• Mr J Moore has been involved in the wider Reefton project since 2001, and visited the Blackwater site in 2010 and visited the site again in 2013.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The Birthday Reef at Blackwater was mined over 750m (vertical) between 1908 and 1951. The historical sample data and mine reporting demonstrate remarkable consistency in terms of both grade and width. The unmined projection of the reef at depth (ie the resource) is however supported by considerably less data; Oceana have completed four deep parent drill holes with 6 daughters, and so have ten reef intercepts on which to estimate the resource which is known to continue to 680m below the mine workings (but the resource has been extrapolated a further 100m, that is 780m below the mine workings). Our confidence in the geological interpretation is based on: 1) the historically demonstrated geological continuity 2) that the projected reef was intercepted where predicted 3) the 1km strike continuity is consistent with large plunge continuity. • While the resource drilling was used to interpret the long sectional extent of the resource, it was felt that (given the structural controls on reef grade and particularly thickness, and given the uneven distribution of drill hole intercepts) that the average grade and width of the resource should be based on the historically averaged reef grade and thickness (this approach taken by Oceana produces a more conservative grade estimate than would be based on drilling results alone). The drilling intercepts do confirm however, the physical presence of the reef, and that the grade tenor and widths of the intersected reef are consistent with the range of grades and widths historically mined. • The assumption that the average widths, average grades and average payability from the historical mining blocks is applicable to the Inferred Resource area, while justifiable for the estimation of a global Inferred Mineral Resource is unsuitable for detailed mine planning. • A number of alternative estimates of the in-situ grade of the historically mined reef were

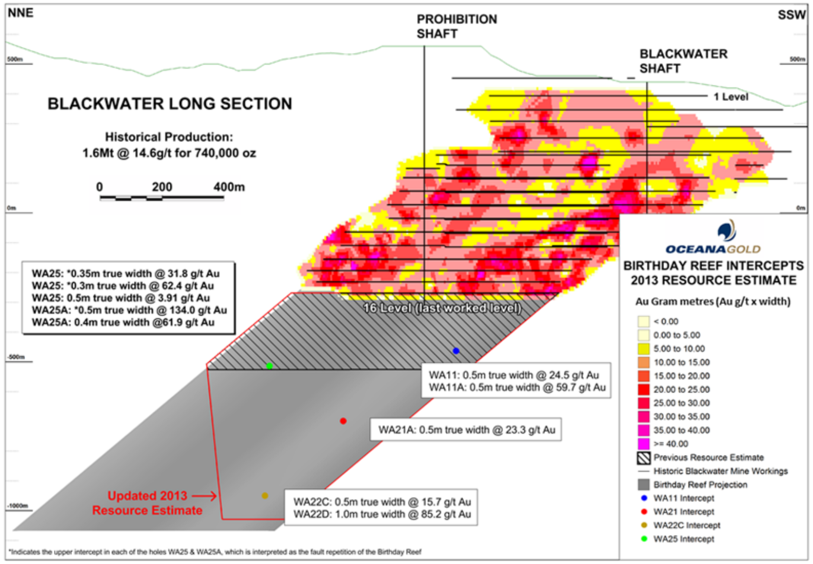
Criteria	JORC Code explanation	Commentary
		undertaken : 1) ordinary kriged sample grades 2) arithmetic averages of face and stope sample grades 3) by using reef payability to estimate the in-situ grade from the mill feed grade estimate (back-calculated from recorded bullion, assumed metallurgical recoveries and reef tonnages). The later was believed to be the most appropriate given that top-cutting of the historical sample grades has been widespread, but inconsistent and poorly documented. • Given that economic mineralization is restricted to the reef, interpretation of the resource wall boundaries is definitive. It is acknowledged however, that there will be shorter scale structural disruptions of the reef which cannot be resolved at the current drilling scale. • The reef grades and widths are related to faulting and reef / bedding relationships. As mentioned above, there will be shorter scale structural post-mineralisation disruptions of the reef which cannot be resolved at the current drilling scale. The impact of these during mining will be mitigated by ongoing development and stope mapping.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td>Dimensions</td><td>

- *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*

</td><td>

- The projected Mineral Resource has a strike length of 900m and has been projected to 780m (100m below deepest intercept) beneath the last worked level of the underground mine (which themselves extend approx. 740m below surface).
- The resource was extrapolated 100m below the deepest drill hole intersection (WA22) on the south west corner of the resource. The north east corner of the reef was excluded, but the resource was extrapolated approximately 200m down plunge, towards the north east corner. Approximately 15% of the resource is therefore extrapolated beyond actual sample locations.



</td></tr>
<tr><td>Estimation and modelling techniques</td><td>

- *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*
- *The availability of check estimates, previous estimates*

</td><td>

- As discussed above, a number of approaches to estimating the resource were evaluated. The approach taken is believed to be the most appropriate. The Blackwater resource estimate is based on a long sectional polygonal area calculation defined by the location of resource drilling reef intercepts. The drilling intercepts confirm the physical presence of the reef, and that the grade tenor and widths of the intersected reef is consistent with the range of grades and widths historically mined.
- The grade and width assumptions are based on averages of the historically mined reef and as such the estimate is not based on interpolation, domaining, top-cutting

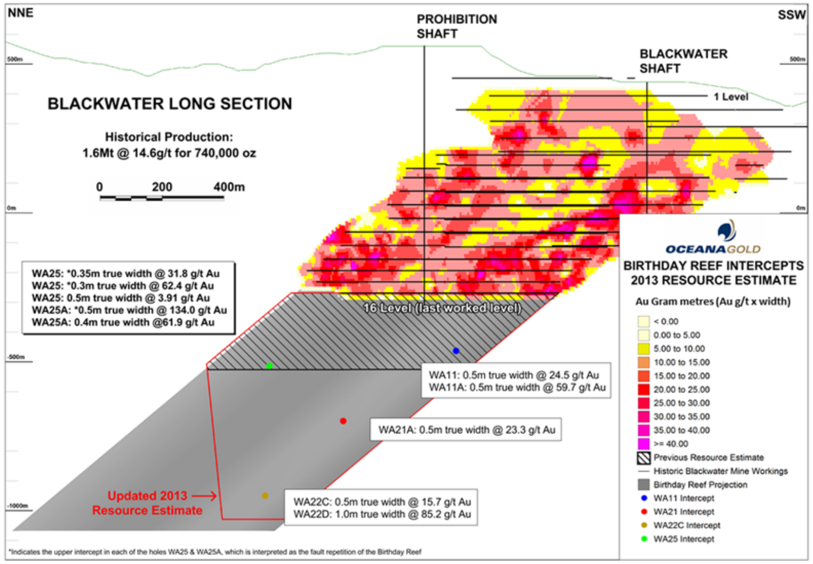
</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	assumptions etc. The estimated reef width was based on an ordinary kriged average of the historically mined reef sample lengths. The grade assigned to the resource however was based on a back-calculation of the mill feed grade (by making reef payability assumptions, the mill feed grade was then related to an in-situ reef grade, because not all the reef was mined). It is important to note that using the resource drilling grades directly would have resulted in a significantly higher grade, which Oceana felt would be inappropriate given the sparse and uneven drill coverage. • Long sectional estimates, with an average estimated (ordinary kriged) 0.68m reef width and 90% tonnage payability, give 1.07 Mt of mined reef resource (1.58 Mt was recorded as being milled, implying 48% mining dilution). 740.4 koz of gold were produced. Assuming 90% metallurgical recovery, 823 koz of gold would have been processed, which back-calculates to an undiluted mined reef grade of 24 g/t Au. • It is assumed that the grade of the reef historically left behind in the Blackwater Mine was lower than the grade of reef mined (reef was left behind due to local geological complexity, narrowing of the reef, and presumably in some cases, lower grade). The combined mined and unmined reef for the Blackwater Mine has been adjusted down to 23 g/t Au which is consistent with the grade of unmined reef being approximately half the grade of the mined reef. There is no data to directly support this correction, but given the likely selection criteria discussed above, some downward grade correction seems warranted. Sensitivities assuming 6 g/t Au and 18 g/t Au for unmined reef grade, yield in-situ resource grades of 22.2 g/t Au and 23.4 g/t Au respectively, so the impact is not large). • No by-product assumptions. • Neither the reef or Greenland Group country rock contain sulphur believed to pose an AMD risk or other deleterious elements. • The estimate is not block model based, and no SMU assumptions were made for resource estimation. • The Blackwater resource estimate is based on a long sectional polygonal area calculation defined by the location of resource drilling reef intercepts. • The width estimate is based on averages of the historically mined reef. The reef grade is based upon a back-calculation from gold production. As such the estimate is not based on top-cutting assumptions. • The estimated reef width and grade is based on historical production data. The limited resource drilling however is consistent with the range of widths and grades historically encountered.

Criteria	JORC Code explanation	Commentary
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• The estimated tonnages are dry.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• No cut-off grade has been applied to the resource estimate. It is anticipated the entire reef will be mined where possible, irrespective of cut-off grade, given the high anticipated nugget effect. There will be instances where the reef may pinch however.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• While no cut–off has been applied, the mineralisation is confined to the Quartz reef which is assumed will be mined in its entirety. However, areas where the reef pinches out or becomes impractically narrow, will not be economic on an accumulation basis.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• Metallurgical test work undertaken on samples of core and waste dump material have been used to demonstrate the recovery assumptions in the study are feasible. Historical production records demonstrate the consistent performance of the reef ore in the original batteries and the current flow sheet is applying modern technology to the established process. The process assumptions have been reviewed by OceanaGold technical staff and external consultants in the preparation of the technical study.
Environmen-tal factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of*	• Assay data on waste rock indicates no significant presence of sulphides or heavy metals that would pose a risk in the waste dump on surface. The process plant will separate the majority of the sulphide minerals present allowing them to be stored underground in cemented marker beds in the stopes minimising risks of metal mobilisation and AMD post closure. The remaining flotation tailings co-disposed in the waste rock structure are expected to pose a minimal risk from the removal of the sulphide minerals present. • Hydrology investigations have been used to model the site water balance and design passive water treatment systems to manage the potential of heavy metals in solution from mine dewatering.

Criteria	JORC Code explanation	Commentary
	these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• OceanaGold has approval for 1.1M m^3 waste rock dump, which appears adequate for the base case. In the PEA it has been estimated that additional capacity of up to 25% may need to be consented to accommodate the additional volume associated with co-disposal of tailings. OceanaGold considers that it is reasonable to assume that the variations to the resource consents to accommodate this additional capacity can be obtained.

Criteria	JORC Code explanation	Commentary
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• A bulk density of 2.60 g/cm3 has been assumed for the quartz reef, given that quartz has an SG of 2.65 - 2.66 g/cm3. A small (2%) allowance for fracturing has been made. • Given the nature of the resource estimate, no waste model was built. For the purposes of the study however, a bulk density of 2.70 g/cm3 was used. This was based on determinations at the Globe mine which is also hosted in greywackes (2.69 g/cm3) and argillites (2.73 g/cm3) of the Greenland Group.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• **"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."** While the projected depth and extent of the resource is based upon four drill holes and their daughter holes, the availability of production records and three dimensional rectified samples from the Blackwater Mine provides valuable insight into the grade continuity and geometric complexity historically encountered. The geological evidence of the projected resource is sufficient to imply but not verify geological and grade continuity. On this basis, the Blackwater estimate is classified as an Inferred Mineral Resource. The resource was extrapolated 100m below the deepest drill hole intersection (WA22) on the south west corner of the resource. The north east corner of the resource was excluded, but the resource was extrapolated approximately 200m down plunge, towards the north east corner. Approximately 15% of the resource is therefore extrapolated beyond actual sample locations.

Criteria	JORC Code explanation	Commentary
		 • Appropriate account has been taken of tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data. • Since Mr J Moore's involvement in the Blackwater project, primarily since 2010, there have been many discussions with other OceanaGold geologists and engineers, as well as with external resource geologists. The majority of these professionals concur with the classification of the resource as Inferred. The results presented in this Table 1 summary reflect Mr J Moore's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• There have been no audits or reviews of the 2014 resource estimate
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the*	• The volume of projected mineralization beneath the historical Blackwater mine constitutes the resource. This volume has been intersected by four drill holes (and their daughter holes). The parent drill holes are typically separated by 400m. It is the strike and plunge extent and geological continuity demonstrated from the historical production data, combined with the predictability of the reef intersections at depth, that provide a

Criteria	JORC Code explanation	Commentary
	relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	compelling case for classifying the resource under JORC 2012. Extensive 3D analysis of reef sample grades and widths reveals significant short range variability but remarkable large scale continuity. Give the resource drill hole spacing and short range variability seen in the production data, only global grades and widths have been estimated.

APPENDIX 2 – INDEPENDENT TECHNICAL REVIEW OF PEA BY AMC CONSULTANTS

In order to meet the requirements of ASX Listing Rule 5.16.6 the PEA is released with the report titled "Independent Technical Report for Blackwater Gold Project" dated October 21st, 2014 (the "ITR"), prepared by or under the supervision of M.G.Dorricott, Principal Mining Engineer. Mr Dorricott is an employee of AMC Consultants Pty Ltd and independent of OceanaGold as that term is defined in the VALMIN Code. Mr Dorricott is a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the JORC Code. The ITR forms this Appendix 2 to the PEA.

The reader is reminded that the Preliminary Economic Assessment (PEA) which is the subject of the ITR has been completed on a production target which is based solely on an Inferred Mineral Resource. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised. The stated production target is based on the company's current expectations of future results or events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met.

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Report

Independent Technical Report for Blackwater Gold Project

OceanaGold (New Zealand) Ltd

Prepared pursuant to ASX Listing Rule 5.16.6

AMC Project 314076
21 October 2014

Executive summary

AMC Consultants Pty Ltd (AMC) was asked to prepare an independent technical report (ITR) to be lodged with the public reporting of a production target for the Blackwater gold project to the Australian Securities Exchange (ASX). OceanaGold (New Zealand) Limited (OceanaGold) has completed a preliminary economic assessment (PEA) based on the results of its internal technical study. The PEA is to be lodged with the New Zealand Stock Exchange and ASX. AMC has reviewed the PEA and provided feedback to OceanaGold, resulting in some amendments to the PEA. The OceanaGold internal technical study was completed on the basis of an Inferred Resource[1] defined beneath the historical Blackwater mine.

AMC has prepared this ITR based on the PEA and AMC's enquiries and discussion with OceanaGold staff, which included a site visit by two AMC consultants. The ITR will serve as the basis for publically reporting the resulting production target and forecast financial information derived from a production target for the Blackwater gold project under ASX Listing Rules 5.16.6 and 5.17, and Section 8.7 of ASX Guidance Note 31[2]. Guidance Note 31 includes reference to the VALMIN Code[3] and ASIC Regulatory Guides 111[4] ("Content of expert reports") and 112[5] ("Independence of experts") to the extent those guidelines are applicable.

The Blackwater Mineral Resource[1], which is the basis of the current public report, is the responsibility of the OceanaGold Competent Person[1], Mr Jonathan Moore. The Resource has been previously reported under the 2004 JORC Code in a media release—"OceanaGold Announces Updated Resource & Reserve Statement"—dated 26 March 2014. AMC understands that it will be reported by OceanaGold in accordance with the 2012 JORC Code together with the PEA. OceanaGold's Inferred[1] Mineral Resource estimate is a satisfactory global Mineral Resource estimate for the depth extensions of the Birthday Reef. In this estimate, given the broad spacing of the resource drilling, it has not been possible to reproduce the local variabilities in reef thickness, grade, and payability[6] that were historically encountered in the Blackwater mine. For that reason, for the purposes of resource estimation, average reef thickness, grade, and payability have been used.

AMC is of the opinion that the Inferred Mineral Resource estimate, as reported by OceanaGold, is a reasonable estimate and is satisfactory, provided adequate account is taken of the inherent variability in the mineralisation, for use in the project study on which the reporting of the production target is based. The exploration decline and subsequent drilling programme is designed to provide information to improve the confidence in these matters.

The December 2013 Mineral Resource estimate is reported below.

Table ES.1 Mineral Resource estimate for Blackwater Birthday Reef as at December 2013

Classification	Tonnes (Mt)	Gold (g/t)	Au Million ounces (Moz)
Inferred Resource	0.9	23	0.7

The Birthday Reef was geologically constrained for the estimate of the Inferred Resource and all the mineralisation was reported without the further application of a grade cut-off. Source: OceanaGold Annual Information Form for the year ended December 31 2013, dated 31 March 2014.

1 As defined by the JORC Code. Australasian Joint Ore Reserves Committee (JORC), *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (The JORC Code), 2012 edn, effective December 2012, 44 pp., available <http://www.jorc.org/docs/JORC_code_2012.pdf>, viewed 16 September 2014.

2 Australian Securities Exchange (ASX), *ASX Listing Rules Guidance Note 31, Reporting on Mining Activities*, introduced 1 December 2013, 18 pp., available <http://www.asx.com.au/documents/rules/gn31_reporting_on_mining_activities.pdf>, viewed 16 September 2014.

3 The VALMIN Committee (VALMIN), *Code for the Technical Assessment and Valuation of Mineral and Petroleum Assets and Securities for Independent Expert Reports* (the VALMIN Code), 2005 edn, effective April 2005, 24 pp., available <http://www.valmin.org/valmin_2005.pdf>, viewed 16 September 2014.

4 Australian Securities & Investments Commission (ASIC), Regulatory Guide 111, Content of expert reports, October 2007, 30 pp., available <http://www.asic.gov.au/asic/pdflib.nsf/LookupByFileName/rg111-30032011.pdf/$file/rg111-30032011.pdf>, viewed 16 September 2014.

5 Australian Securities & Investments Commission (ASIC), Regulatory Guide 112, Independence of experts, October 2007, 22 pp., available <http://www.asic.gov.au/asic/pdflib.nsf/LookupByFileName/rg112-30032011.pdf/$file/rg112-30032011.pdf>, viewed 16 September 2014.

6 Payability is the proportion of strike length (not tonnage) that was mined in the historical mining and is assumed to also be applicable to the depth extensions.

AMC is of the opinion that the project is technically stalled and will remain so until the information to be obtained as a result of the exploration decline and subsequent programme is collected and analysed. This information will include:

- A better estimate of the in situ reef variability—increasing the resource confidence, instead of using the previous production areas as a proxy for the variability in the Inferred Resource below 16 level.
- A better understanding of the geotechnical parameters that will affect the mining methods.
- Further consideration of a mining method that is matched to the conditions.
- Results of further metallurgical testwork that is updated using representative samples.

OceanaGold has already tested the option of accessing the reef at depth through the existing historical workings, by refurbishing the historical Prohibition Shaft, in order to undertake further exploratory drilling from underground. Refurbishment of the Prohibition Shaft was started by contractor Combined Resource Engineering in September 2004. The attempted refurbishment was abandoned in November 2004. After encountering a blockage at 53 m, drilling from surface found the blockage to extend to at least 129 m, and it was decided to abandon the refurbishment project. This option for accessing the reef at depth is now ruled out.

In AMC's opinion, it is not practical or possible to collect the required information "by conventional exploration alone" to progress the project; that is, by further surface deep exploratory drilling. Such a programme would require the drilling of a large number of parent and daughter holes to provide the additional information required, and even then not all of the necessary data could be obtained. As an example, the geotechnical stress field information would not be reliably provided by deep diamond drilling.

In addition, access to the optimum surface drilling sites is not practically achievable due to land ownership restrictions, and topographical difficulties, and OceanaGold has experienced difficulty in controlling drillhole deviation. In any event, it is not possible to achieve the required density of intersections of the Blackwater Birthday Reef by surface drilling alone. Drilling these holes from the surface is prohibitively expensive. OceanaGold reports that the recent surface drilling has cost in excess of NZD10 million to eventually drill four successful parent holes and daughter holes, which achieved seven intersections of the reef and some apparent fault repeats in two holes.

Close access to the reef and surrounding rocks is considered necessary to gain the additional geotechnical information, and this cannot be obtained reliably from surface drillholes. Without the additional resource and geotechnical information, it will not be possible to select and design the most appropriate mining method, although the airleg resue method considered as the base case in the PEA provides a reasonable approach to the possible productivity of the mine.

Further representative samples to validate the metallurgical testwork will only become available from the underground drilling to refine the metallurgical parameters and plant specification. However, the production history and test results on similar mineralisation provide reasonable grounds for the current design.

Only when all of this information is collected and analysed will the project not be technically stalled.

AMC considers that the proposed mining method, including access, ventilation, and ore haulage systems are technically and operationally feasible and appropriate for the current level of study. They will need to be reviewed when resource definition is better-understood and more geotechnical information becomes available. AMC considers that the mining factors applied in converting the Inferred Mineral Resource to a production target are appropriate for the proposed mining method.

Recognising that variability in reef thickness and grade will occur, OceanaGold has modelled a range of grade and thickness scenarios and reported the corresponding net present values (NPVs). The results are presented as an array, ranging from a lower post-tax NPV of US$21 million to an upper post-tax NPV of US$243 million at a discount rate of 5%. AMC understands that the entire project is intended to be internally funded by OceanaGold cashflow, and 5% is the OceanaGold internal hurdle rate of return when assessing investment opportunities. The base-case scenario suggests a post-tax NPV of US$132 million.

The NPV remains positive over the full range of reef width and grade scenarios modelled, and the project financials seem robust. Both width and grade are expected to vary during the course of mining, but the current level of information makes it impossible to model such scenarios using geological modelling methods. AMC considers that the inputs to the economic analysis are reasonable, and that the presentation of the resulting NPVs as an array of possibilities is appropriate. The wide range of possible outcomes reflects the level of uncertainty at this stage of the project.

Table ES.2 NPV array (US$ million) by flexing reef width and grade

NPV array			Reef thickness {diluted thickness} (m)				
			–30%	–15%	Base	15%	30%
			0.48 {0.75}	0.58 {0.87}	0.68 {1.00}	0.78 {1.17}	0.88 {1.30}
Reef grade (g/t)	–30%	16			21		
	–15%	20		46	83	105	
	Base	23	42	90	132	159	199
	15%	26		133	180	212	
	30%	30			243		

Sensitivity analysis for a range of financial factors is reported in the economic evaluation Section 22 of the PEA, and is summarised in Section 10.6 of this ITR.

The production target and financial information derived from the Blackwater project production target referred to in this ITR are based solely on Inferred Mineral Resources. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated[7] Mineral Resources or that the production target itself will be realised. The stated production target is based on OceanaGold's current expectation of future results or events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met.

This ITR must be read in conjunction with the PEA on which it is based, and the reader's attention is drawn to the cautionary statements contained in the PEA.

[7] As defined by the JORC Code.

Quality control

The signing of this statement confirms this report has been prepared and checked in accordance with the AMC Peer Review Process. AMC's Peer Review Policy can be viewed at www.amcconsultants.com.

Project manager	The signatory has given permission to use their signature in this AMC document	21 October 2014
	Malcolm Dorricott	**Date**
Peer reviewer	The signatory has given permission to use their signature in this AMC document	21 October 2014
	Peter McCarthy	**Date**

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1 e-copy to Simon Griffiths, OceanaGold (New Zealand) Limited
1 e-copy to AMC Brisbane office

1 Introduction

1.1 Background

OceanaGold (New Zealand) Limited (OceanaGold) has completed an internal technical study to assess the technical and economic potential of the Blackwater gold project, near Reefton in New Zealand. Mr Simon Griffiths, of OceanaGold, requested AMC Consultants Pty Ltd (AMC) to prepare an independent technical report (ITR) to be lodged with the public reporting of a production target for the Blackwater gold project in accordance with the requirements of the Australian Securities Exchange (ASX). OceanaGold has completed a preliminary economic assessment (PEA) based on the results of its internal technical study. The PEA is to be lodged with the New Zealand Stock Exchange and ASX. AMC has reviewed the PEA and provided feedback to OceanaGold, resulting in some amendments to the PEA. The OceanaGold internal technical study was completed on the basis of an Inferred Resource[8] defined beneath the historical Blackwater mine.

1.2 Scope

AMC has prepared this ITR based on the PEA and AMC's enquiries and discussion with OceanaGold staff, which included a site visit by two AMC consultants. The ITR will serve as the basis for publicly reporting the production target and forecast financial information derived from a production target for the Blackwater gold project under ASX Listing Rule 5.16.6, and 5.17, and Section 8.7 of ASX Guidance Note 31[9]. Guidance Note 31 includes reference to the VALMIN Code[10] and ASIC Regulatory Guides 111[11] ("Content of expert reports") and 112[12] ("Independence of experts") to the extent those guidelines are applicable.

1.3 Competent persons and Specialists

The competent person for the ITR is Mr Malcolm Dorricott, AMC Principal Mining Engineer, who reviewed the mining and economic evaluation sections of the PEA.

Specialists who contributed to the preparation of the ITR included:

- Mr Peter Stoker, AMC Principal Geologist, undertook a high-level review of the Inferred Resource estimate to assess its suitability to be the basis for the production target.
- Mr Mike Sandy, AMC Principal Geotechnical Consultant, reviewed the geotechnical sections of the PEA.
- Mr Robert Chesher, AMC Principal Consultant, reviewed the metallurgical and processing sections of the PEA.
- Mr Peter Allen, AMC Principal Environmental Consultant, reviewed the infrastructure section, the environmental and community sections, and related permits and consents of the PEA.

Mr Peter McCarthy, AMC Chairman and Principal Mining Consultant, peer reviewed the ITR.

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8 As defined by the JORC Code. Australasian Joint Ore Reserves Committee (JORC), *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (The JORC Code), 2012 edn, effective December 2012, 44 pp., available <http://www.jorc.org/docs/JORC_code_2012.pdf>, viewed 16 September 2014.

9 Australian Securities Exchange (ASX), *ASX Listing Rules Guidance Note 31, Reporting on Mining Activities*, introduced 1 December 2013, 18 pp., available <http://www.asx.com.au/documents/rules/gn31_reporting_on_mining_activities.pdf>, viewed 16 September 2014.

10 The VALMIN Committee (VALMIN), *Code for the Technical Assessment and Valuation of Mineral and Petroleum Assets and Securities for Independent Expert Reports* (the VALMIN Code), 2005 edn, effective April 2005, 24 pp., available <http://www.valmin.org/valmin_2005.pdf>, viewed 16 September 2014.

11 Australian Securities & Investments Commission (ASIC), Regulatory Guide 111, Content of expert reports, October 2007, 30 pp., available <http://www.asic.gov.au/asic/pdflib.nsf/LookupByFileName/rg111-30032011.pdf/$file/rg111-30032011.pdf>, viewed 16 September 2014.

12 Australian Securities & Investments Commission (ASIC), Regulatory Guide 112, Independence of experts, October 2007, 22 pp., available <http://www.asic.gov.au/asic/pdflib.nsf/LookupByFileName/rg112-30032011.pdf/$file/rg112-30032011.pdf>, viewed 16 September 2014.

Certificate of competent person

As the competent person and co-author of the report titled "Independent Technical Report for Blackwater Gold Project" (Blackwater ITR) dated 21 October 2014, to which this certificate applies, I, Malcolm George Dorricott do hereby certify that:

1. I am a Principal Mining Engineer working for AMC Consultants Pty Ltd. My business address is Level 21, 179 Turbot Street, Brisbane Queensland 4000, Australia.
2. I graduated with a Bachelor of Engineering (Mining Engineering) degree in 1969 from the University of Queensland.
3. I am a Fellow and Chartered Professional (Mining) in good standing with The Australasian Institute of Mining and Metallurgy (The AusIMM).
4. I am a Registered Professional Engineer of Queensland (Area of Mining) in good standing.
5. I have worked in the mining industry for a total of 45 years since my graduation, including more than 10 years in underground base metals and precious metals mining operations. I have conducted numerous technical, strategic, and ore reserve reviews and other consulting assignments for gold mining operations and projects.
6. I have read the definition of "competent person" defined in Chapter 19 of the ASX Listing Rules (and the meaning of "Competent Person" in Appendix 5A of the JORC Code) and confirm that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be a "competent person" for the purposes of ASX Listing Rules.
7. My most recent personal inspection of the Blackwater gold project was in August 2014.
8. I prepared Sections 1, 3, 4, 5, 6, 10, and 11 of this report titled "Independent Technical Report for Blackwater Gold Project" dated 21 October 2014. The remaining sections were prepared by the named specialists.
9. I am independent of OceanaGold Corporation and any subsidiaries, other than providing consulting services.
10. Prior to providing consultancy services to OceanaGold in 2014, I have had no involvement with the Blackwater gold project.
11. I have read the VALMIN Code and the JORC Code, and the Blackwater ITR has been prepared in compliance with these codes to the extent that they are relevant.

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

MLDorricott

Malcolm George Dorricott
Principal Mining Engineer

Date of signature: 21 October, 2014

2 Mineral Resource estimate

2.1 Blackwater Birthday Reef Mineral Resource

The Blackwater Mineral Resource, which is the basis of the current public report, is the responsibility of the OceanaGold Competent Person, Mr Jonathan Moore. The Blackwater Mineral Resource as listed in Table 2.1 has been previously reported under the 2004 JORC Code in a media release—"OceanaGold Announces Updated Resource & Reserve Statement"—dated 26 March 2014. It will be reported in a subsequent market release to comply with the reporting conditions in the 2012 JORC Code and ASX Listing Rule 5.8, to support the statement of a production target for the Blackwater project. AMC has been provided with a copy of the internal documentation that supports the Mineral Resource.

AMC has reviewed the documentation that supports the public statement of the Blackwater Birthday Reef Inferred Mineral Resource. AMC notes that the extent of the resource, as shown in Figure 2.1, is determined by the recent deep drilling intersections of four holes with intersections of the reef in parent holes and also intersections in daughter holes (see Table 2.2). These intersections have been used to sensibly restrict the long. section area of the Birthday Reef Inferred Mineral Resource as follows:

> A reef plane with a 900 metre strike length was projected to depth. Within this projected plane, the resource limit was broadly based on a 200 metre maximum distance to the nearest sample", see the thick red line in the long section in Figure 2.1.

OceanaGold report:

> The historical average (declustered via ordinary kriging) reef thickness of 0.68 metres was used to estimate the volume.

Because of the perceived potential bias in the limited deep exploratory drilling intersection data set below the historical workings of 16 level, and the potential for errors in the face and stope samples from the historical workings, the decision was taken to back calculate the in situ grade from the historical production records. This, and the assumptions used, is well-documented in the OceanaGold internal documentation and the PEA. OceanaGold has documented the basis of the selection of the bulk density for use in the resource estimate, and in AMC's opinion, this is acceptable for a global Inferred Mineral Resource statement.

OceanaGold's Inferred Mineral Resource estimate is a satisfactory global Mineral Resource estimate for the depth extensions of the Birthday Reef. In this estimate, given the broad spacing of the resource drilling, it has not been possible to reproduce the local variability in reef grade, thickness, and payability[13], which was historically encountered in the Blackwater mine. For that reason, for the purposes of resource estimation, average reef thickness, grade, and payability have been used.

AMC considers that the Inferred Mineral Resource estimate, as reported by OceanaGold, is a reasonable estimate and is satisfactory, provided adequate account is taken of the inherent variability in the mineralisation, for use in the project study on which the reporting of the production target is based.

The Competent Person has pointed out the potential variability in grade, thickness, and payability likely to be encountered when the reef is intersected in development. The exploration decline and subsequent drilling programme is designed to provide information to improve the confidence in these matters.

The December 2013 Mineral Resource estimate is reported in Table 2.1.

Table 2.1 Mineral Resource estimate for Blackwater Birthday Reef as at December 2013

Classification	Tonnes (Mt)	Gold (g/t)	Au Million ounces (Moz)
Inferred Resource	0.9	23	0.7

The Birthday Reef was geologically constrained for the estimate of the Inferred Resource and all the mineralisation was reported without the further application of a grade cut-off. Source: OceanaGold Annual Information Form for the year ended December 31 2013, dated 31 March 2014.

[13] Payability is the proportion of strike length (not tonnage) that was mined in the historical mining and is assumed to also be applicable to the depth extensions.

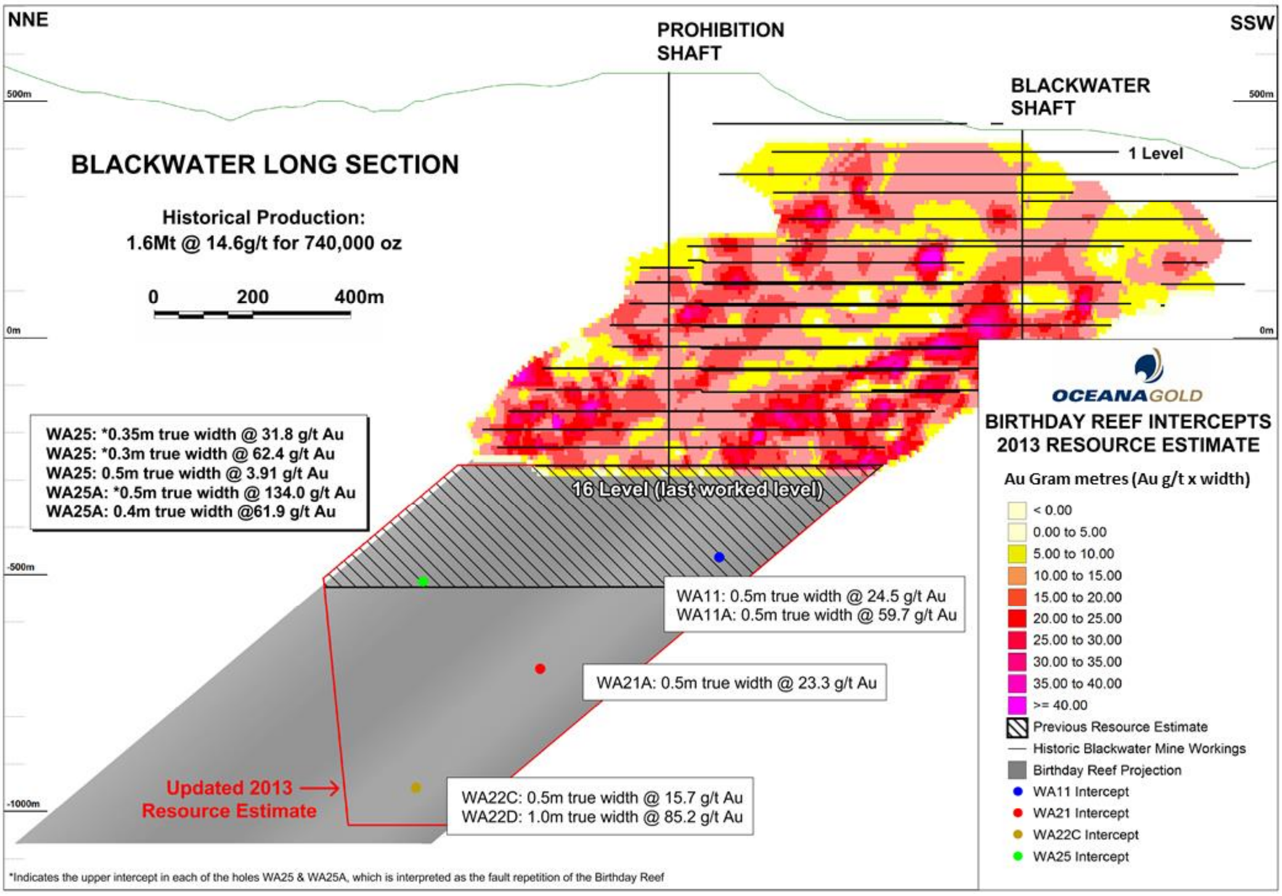
Table 2.2 Blackwater Deep drillhole intercepts

Hole ID	From (m)	To (m)	Intercept (m)	True width (m)	Grade (Au g/t)	Grade width ($g^A \times m$)	Comment
WA11	979.6	980.3	0.7	0.5	24.50	12.3	Parent hole
WA11A	980.3	981.0	0.7	0.5	59.70	29.9	Daughter hole
WA21A	1,315.9	1,316.8	0.9	0.5	23.30	11.7	Daughter hole
WA22C	1,632.30	1,633.0	0.70	0.5	15.65	7.8	Parent hole
WA22D	1,623.90	1,625.03	1.13	1.0	85.2	85.2	Daughter hole
WA25	1,118.95	1,119.40	0.45	*0.35	31.8	11.1	Parent hole
WA25	1,134.18	1,134.59	0.41	*0.3	62.4	18.7	Parent hole
WA25	1,190.77	1,191.36	0.59	0.5	3.91	1.9	Parent hole (BR)
WA25A	1,136.40	1,137.11	0.71	*0.5	134.00	67.0	Daughter hole
WA25A	1,195.20	1,195.65	0.45	^0.4	61.90	24.7	Daughter hole (BR)

* Indicates the upper intercept in each of the holes WA25 & WA25A interpreted as a fault repetition of the Birthday Reef. BR indicates the Birthday Reef intercept. ^ Unorientated drill core. True width calculated using WA25 intercept.

Source: Table 10-3 Blackwater Project PEA 2014, OceanaGold.

Figure 2.1 Blackwater mine long. section



2.2 Blackwater project technically stalled

This ITR has been prepared by AMC to accompany the public statement by OceanaGold of its intention to proceed with the construction of an exploration decline at its Blackwater project New Zealand, which includes the statement of a production target and forecast financial information derived from that production target. The production target is based solely on the Inferred Resources for the Birthday Reef at Blackwater as reported by the OceanaGold Competent Person Mr Jonathon Moore. This ITR is required to accompany the OceanaGold statement of the production target by ASX Listing Rule 5.16.6 and ASX Guidance Note 31. Its purpose is to demonstrate that it is acceptable to report a production target (and financial information derived from that production target) based solely on Inferred Resources in this particular instance.

Listing Rule 5.16.6 requires in part that the public report of the production target includes:

> a statement of the factors that lead the entity to believe that it has a reasonable basis for reporting a production target based solely on inferred mineral resources.

This requirement is expanded on the ASX Guidance Note 31 which states:

> Where a production target is based solely on inferred mineral resources, the following additional requirements must be satisfied:
>
> - disclosure of a statement of the factors that lead the entity to believe that it has a reasonable basis for reporting a production target based solely on inferred mineral resources. These factors should be limited to types of mineralisation where the project cannot be progressed through to a higher confidence level of mineralisation by conventional exploration alone prior to release of a production target;

The meaning of this requirement is generally interpreted to mean the project is "technically stalled"[14]. This meaning appears to have been accepted as the intended interpretation of the statement that appears in the guidance note:

> the project cannot be progressed through to a higher confidence level of mineralisation by conventional exploration alone prior to release of a production target.

AMC is of the opinion that the project is technically stalled and will remain so until the information to be obtained as a result of the exploration decline and subsequent programme is collected and analysed. This information will include:

- A better estimate of the in situ reef variability—increasing the resource confidence category, instead of using the previous production areas as a proxy for the variability in the Inferred Resource below 16 level.
- A better understanding of the geotechnical parameters that will affect the mining methods.
- Further consideration of a mining method that is matched to the conditions.
- Results of further metallurgical testwork that is updated using representative samples.

OceanaGold has already tested the option of accessing the reef at depth through the existing historical workings, by refurbishing the historical Prohibition Shaft, in order to undertake further exploratory drilling from underground. Refurbishment of the Prohibition Shaft was started by contractor Combined Resource Engineering in September 2004. The attempted refurbishment was abandoned in November 2004. After encountering a blockage at 53 m, drilling from surface found the blockage to extend to at least 129 m, and it was decided to abandon the refurbishment project. This option for accessing the reef at depth is now ruled out.

In AMC's opinion, it is not practical or possible to collect the required information "by conventional exploration alone" to progress the project; that is, by further surface deep exploratory drilling. Such a programme would require the drilling of a large number of parent and daughter holes to provide the additional information required, and even then not all of the necessary data could be obtained. As an example, the geotechnical stress field information would not be reliably provided by deep diamond drilling.

In addition, access to the optimum surface drilling sites is not practically achievable due to land ownership restrictions and topographical difficulties, and OceanaGold has experienced difficulty in controlling drillhole deviation. In any event, it is not possible to achieve the required density of intersections of the Birthday Reef by surface drilling alone. Drilling these holes from the surface is prohibitively expensive. OceanaGold reports that the recent surface drilling has cost in excess of NZD10 million to eventually drill four successful parent holes and daughter holes, which achieved seven intersections of the reef and some apparent fault repeats in two holes.

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[14] These words come from a statement that Steve Hunt, Chairman of JORC, made during the ASX/ASIC/JORC roadshows on the revised ASX Listing Rules and JORC Code in 2013.

Close access to the reef and surrounding rocks is considered necessary to gain the additional geotechnical information, and this cannot be obtained reliably from surface drillholes.

Without the additional resource and geotechnical information, it will not be possible to select and design the most appropriate mining method, although the airleg resue method considered as the base case in the PEA provides a reasonable approach to the possible productivity of the mine.

Further representative samples to validate the metallurgical testwork will only become available from the underground drilling to refine the metallurgical parameters and plant specification. However, the production history and test results on similar mineralisation provide reasonable grounds for the current design.

Only when this information is collected and analysed will the project not be technically stalled. This all requires some time after the completion of the exploration decline to collect and analyse the relevant information.

The ASX Guidance Note 31 also requires:

- disclosure of the level of confidence associated with the estimates of inferred mineral resources and the basis for the level of confidence. The basis for the level of confidence should be described in narrative form and be descriptive enough to inform the market of the reasons why the entity holds this confidence level;

AMC notes that OceanaGold has stated that it has reasonable grounds that its Blackwater Inferred Resource is a good global estimate of the expected mineralisation. OceanaGold states that this is based on the deep diamond drill intercepts of the Birthday Reef, which indicate continuity of the mineralisation beneath the historical Blackwater mine and the broad similarity of the grades with those reported from the historical production.

AMC is of the opinion that OceanaGold’s Blackwater project is technically stalled.

AMC agrees that the current Blackwater Inferred Mineral Resource provides a reasonable basis for the studies from which the production target is derived.

3 Site visits

Two AMC consultants undertook a site visit on 29 August 2014. Mr Malcolm Dorricott (the competent person) and Mr Peter Allen inspected drill core at the OceanaGold exploration office in Reefton, and visited the historical Waiuta site where they inspected the collars of the Blackwater and Prohibition shafts, the remains of the Prohibition mill, the old rock dump, and historical buildings and relics from the historical Blackwater mine. They also inspected the site of the proposed Blackwater gold project, including the site of the proposed surface facilities (process plant, infrastructure, waste dump, and tailings storage facility) and the proposed decline portal site.

Discussions were conducted with relevant OceanaGold staff and consultants.

4 Study status

OceanaGold has prepared a technical study at the level of a scoping study, based on an Inferred Resource estimate.

Based on this study, a National Instrument (NI) 43-101 preliminary economic assessment (PEA) technical report (S0031-REP-068-0) dated 15 September 2014 has been prepared.

AMC has reviewed the PEA as the basis for this ITR.

5 Cut-off parameters

No cut-off has been applied to the Inferred Resource estimate, which is geologically constrained within the reef volume with an average reef thickness of 0.68 m and an average grade of 23 g/t Au.

A cut-off grade has not been applied to the mine design. The assumption applied is that the production target has a constant grade (23 g/t Au) and reef thickness (0.68 m) for the base case.

It is expected that a cut-off philosophy will be developed once an Indicated[15] Resource block model has been created, to assist with optimising the mine design. Preliminary studies indicate this might be based on a metal accumulation (grade × thickness) criterion.

[15] As defined by the JORC Code.

6 Mining factors or assumptions

6.1 Conversion of Mineral Resource to a production target

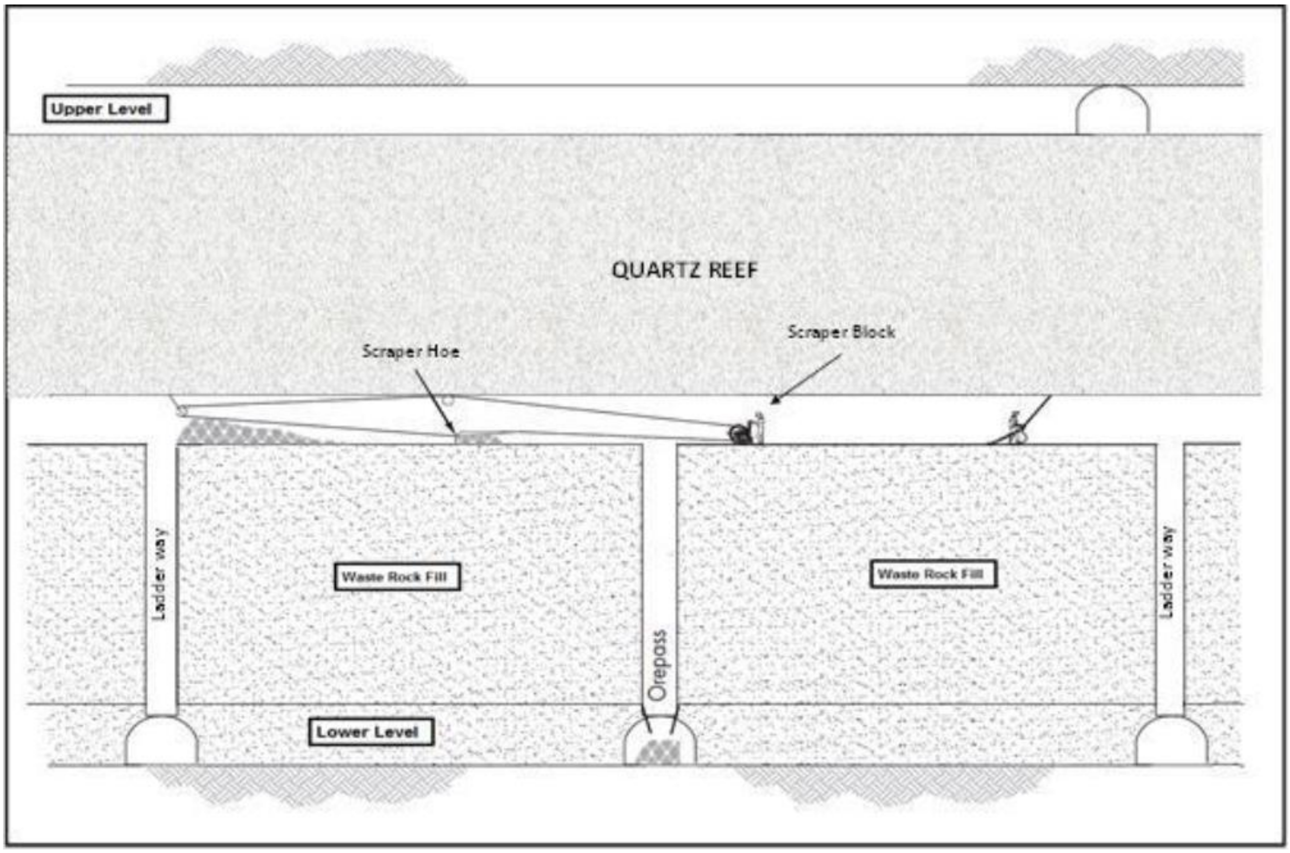
The Inferred Mineral Resource described in Section 2 has been converted to a production target on the basis of the technical study and the application of mining factors described in this section.

6.2 Mining method

Historically, the Birthday Reef was selectively mined by non-mechanised, overhand cut-and-fill mining techniques using the waste rock as backfill. This minimised the amount of barren rock that was hoisted to surface, and maintained a relatively high head grade to the processing plant. The airleg resue method proposed for future mining is similar, except that instead of hoisting the ore in a vertical shaft, it will be transported to surface in diesel-powered trucks via a decline roadway. Diesel-powered loaders will be used to load the ore into the trucks from the orepasses. Mechanised mining equipment will be used to develop the mine access roadways from surface down to the reef, and for other development used for exploration, level access, and ventilation. Initial development in ore at the bottom of each stoping panel will be by airleg mining.

Stoping will be mainly by airleg resue mining, whereby the narrow quartz reef is initially extracted with as little dilution from the country rock as possible, followed by blasting of the surrounding country rock to fill the void and provide a working platform for the next lift. The ore will be removed from the stope using scrapers into orepasses that are carried up through the broken waste material (Figure 6.1). A cement slurry marker bed will be placed on top of the waste rock to minimise dilution from the fill, and provide an improved surface for the scraper hoe. Ventilation of the stopes is provided via the ladderways as intakes and the orepasses as exhausts.

Figure 6.1 Section view of airleg resue stoping



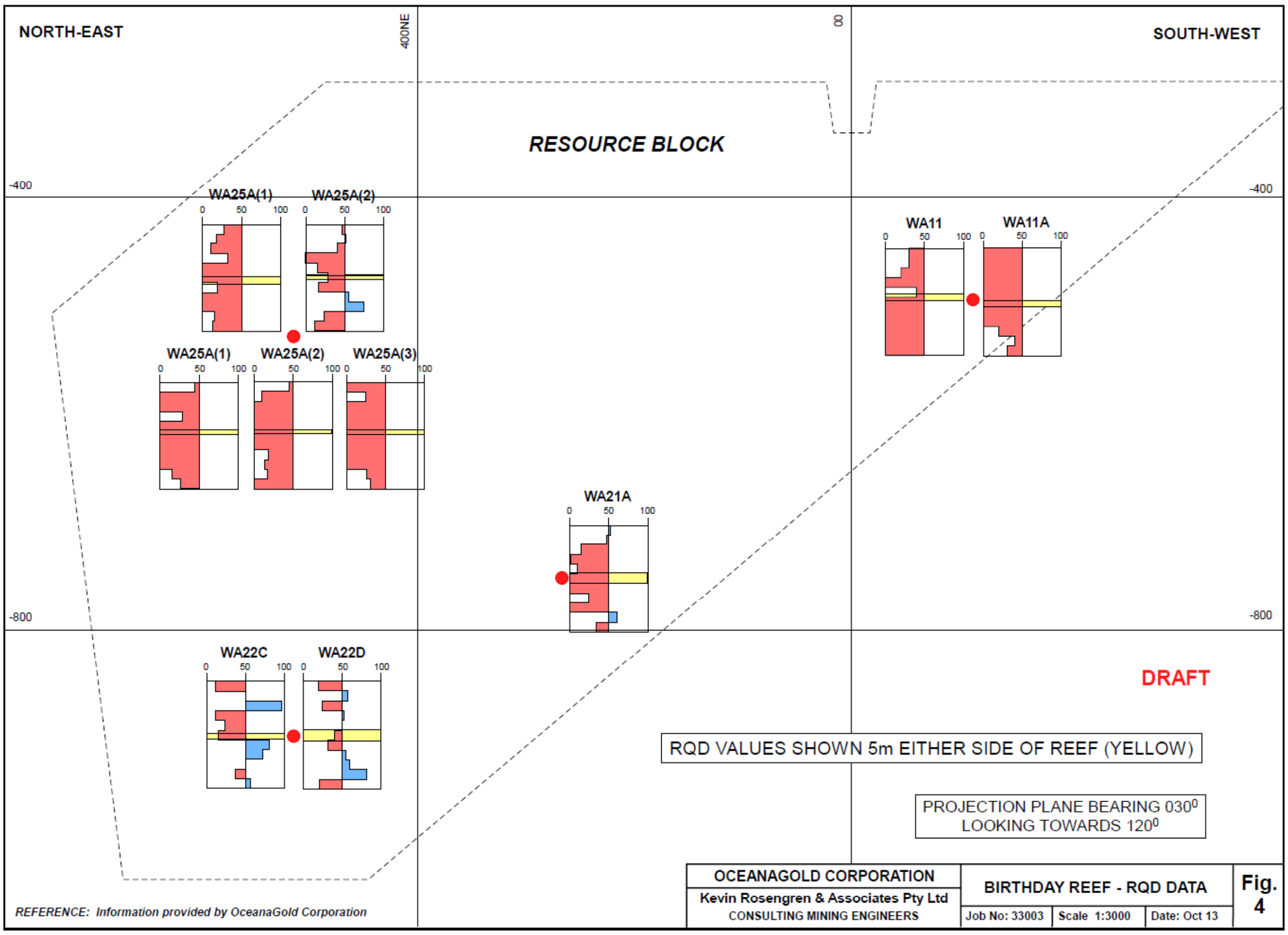
Other mining methods have been considered in previous studies, and these might be applicable in some sections of the reef. In wider areas, where ground conditions are good, narrow-vein bench stoping might be possible. This is a lower-cost, more productive method. In areas where ground conditions are poor, cut-and-fill mining might be necessary. This is a higher-cost method and would only be used if the grade of the reef could justify the additional cost.

6.3 Geotechnical assessment

6.3.1 Geotechnical data

As stated in the PEA, geotechnical data in the area of the study is sparse. A total of 10 intersections of the Birthday Reef were obtained from six parent diamond drillholes in 1996 and 2011–2013. The assessment of ground conditions in and near the reef is largely based on core photographs and geotechnical logging of these intersections, as presented in a Kevin Rosengren and Associates report (KRA, 2013). Figure 4 from that report is reproduced below as Figure 6.2.

Figure 6.2 Birthday Reef rock quality designation data (after KRA, 2013)



There are currently no site-specific geotechnical drillholes intersecting the proposed access decline situated to the west of the proposed underground mine. Conditions are currently interpreted from four holes that were drilled along the original decline alignment about 200 m to the south.

The PEA discusses the planned geotechnical data collection programme. This includes geotechnical site investigation holes for the current decline route (including the box cut and portal) and geotechnical logging of planned resource diamond drillholes to be drilled from underground access.

Rock material property testwork and stress measurements are also planned.

6.3.2 Ground conditions and support requirements

As mentioned above, no drilling information is available along the proposed decline. The information from the holes drilled on the original alignment has been used to infer conditions on the decline and to assign percentages for the proportion of development expected in each of three classes of ground conditions.

The PEA presents proposed support standards for each class. These seem reasonable for the planned development dimensions and rock mass quality categories described. AMC notes that a 2013 review by Mining Plus observed that the core had “deteriorated significantly”’. If the same materials are present along the decline, shotcrete or fibrecrete might be required even where the ground is otherwise amenable to support with bolts and mesh, in order to seal the rock.

Ground conditions near the mineralized zone are indicated to be generally poor. Q’ values, based on the rock mass classification scheme proposed by Barton et al (1974), are assessed in the PEA to lie within the range 0.28 to 1.1. The average Q’ value for the immediate 5 m into the hangingwall is stated to be 0.36.

There might be some areas of better rock mass quality, but the currently available information indicates that poor ground conditions might be widespread. AMC believes that the conservative assumption regarding ground conditions is appropriate, given the sparse availability of information.

6.3.3 Stope spans

The PEA describes how the “stability graph method” (as proposed by Potvin, 1988) was used to specify stope spans. Factors to account for stress versus rock strength, structural effects, stope wall inclination, and failure mode are applied to the rock quality (Q’) values to obtain “modified stability numbers” (N’). These suggest that stopes with a span (expressed in terms of “hydraulic radius”, HR) of 1.6 m or less should be stable for the range of conditions assessed at Blackwater.

The stability graph method is widely used in mining studies at scoping or pre-feasibility level to specify nominal design spans. The method is based on open stoping case histories that are mostly from projects with better ground conditions than those assessed for Blackwater in the PEA.

Reference to Potvin’s database indicates that only 20% of the case histories were in conditions similar to those assessed at Blackwater. Nonetheless, the use of the method is considered reasonable for a study at this level. The Q’ values and the suggested values for Potvin factors are considered reasonable.

6.3.4 Mining method

The currently proposed airleg resue mining method involves small spans of up to about 4 m (vertical) by about 3 m wide. This is in line with earlier AMC recommendations to assume a base case mining method involving small spans. Stopes will be mined in compartments or panels that are 60 m along strike. Only one half of each compartment (30 m) will be open over the 4 m vertical extent at any time.

The proposed stope hangingwall span of 4 m vertical by 30 m along strike has an HR of 1.8 m. This is slightly more than the span suggested by the stability graph assessment described above, which is for unsupported stope hangingwalls. Given that support will be installed in the lower part of the hangingwall (from the previous lift), this is considered acceptable.

After firing the ore slot in the backs and scraping the ore into a pass, the waste mining is conducted using flatback stripping. This is considered appropriate as it should minimise blast damage to the backs and allow good access for support.

During flatbacking, the vertical span will be about 2 m. This will constrain drilling and installing rock bolts into the backs and hangingwalls. Achieving bolt installations of adequate length might require the use of extension steels in some locations. It is unlikely that friction-type bolts of adequate length could be installed in the limited vertical height available.

The profile of the backs after firing the ore slot will potentially create zones of relaxation or significant stress drop in the stope walls. Support will be installed to manage this relaxed ground as far as possible, but there is still a risk that unravelling and instability will occur. This could lead to higher levels of dilution, increased support costs, and requirements to introduce additional material for backfilling.

Trial mining, as proposed in the PEA, will allow these concerns to be investigated. Drive profiles and the position of the ore slot relative to the oredrive could require modification to improve drive stability and reduce fall-off. Where very poor ground conditions are encountered, the PEA states that an alternative mining method would be employed: “In areas where ground conditions are considerably worse than average it is likely that a mechanised cut and fill mining method will be employed.” AMC considers this to be reasonable as it would involve very small wall exposures (typically no more than 3 m by 3 m unsupported at any time). This would be similar to the method that was apparently successfully used before the mine closed due to the failure of the shaft.

6.3.5 Extraction sequence

The PEA refers to an earlier AMC discussion on mining sequence issues:

> AMC noted that the mining sequence will form a series of sill pillars. These are progressively reduced in size as each stoping panel 'closes' on the extracted panel above. In this situation, the stresses will progressively increase as the sill pillars are reduced in size. It can be reasonably expected that ground conditions will become more difficult to control over time as the pillars are reduced in size.
>
> Squeezing ground conditions could be encountered, although this cannot be assessed with the current lack of information on the pre-mining stress field and rock material properties, including strength. Squeezing ground behaviour usually results in slower advance rates, higher support costs and delays due to rehabilitation.
>
> The proposed mining depth (from 800 m below surface to possibly as much as 1,600 m) will require that stresses are investigated and that the mining sequence is developed to manage these as far as possible. It is possible that the overall sequence will need to be carefully managed to reduce rehabilitation and achieve acceptable levels of resource recovery, which could impact on the production schedule.

It should be noted that if the stresses at Blackwater are high and the rock is weak and prone to severe squeezing, there might be a depth beyond which mining is not practical.

The PEA also discusses the potential for changes to the development sequence, that would be required if the extraction sequence needs to be modified to better manage the geotechnical conditions:

> Where possible, development is scheduled to be mined as late as possible and the hanging wall decline will be extended on a "just-in-time" basis to ensure that the next production panel is ready for extraction before the previous panel is completed, including a second means of egress, primary ventilation and materials handling development.
>
> The current schedule assumes that a production panel will be essentially complete before production commences on the next panel down dip. Once additional geotechnical information becomes available from underground diamond drilling this will be reassessed. The production sequence within a panel may need to be staggered 'en-echelon' to deal with the diminishing sill pillars, which may require an adjustment to the development schedule.

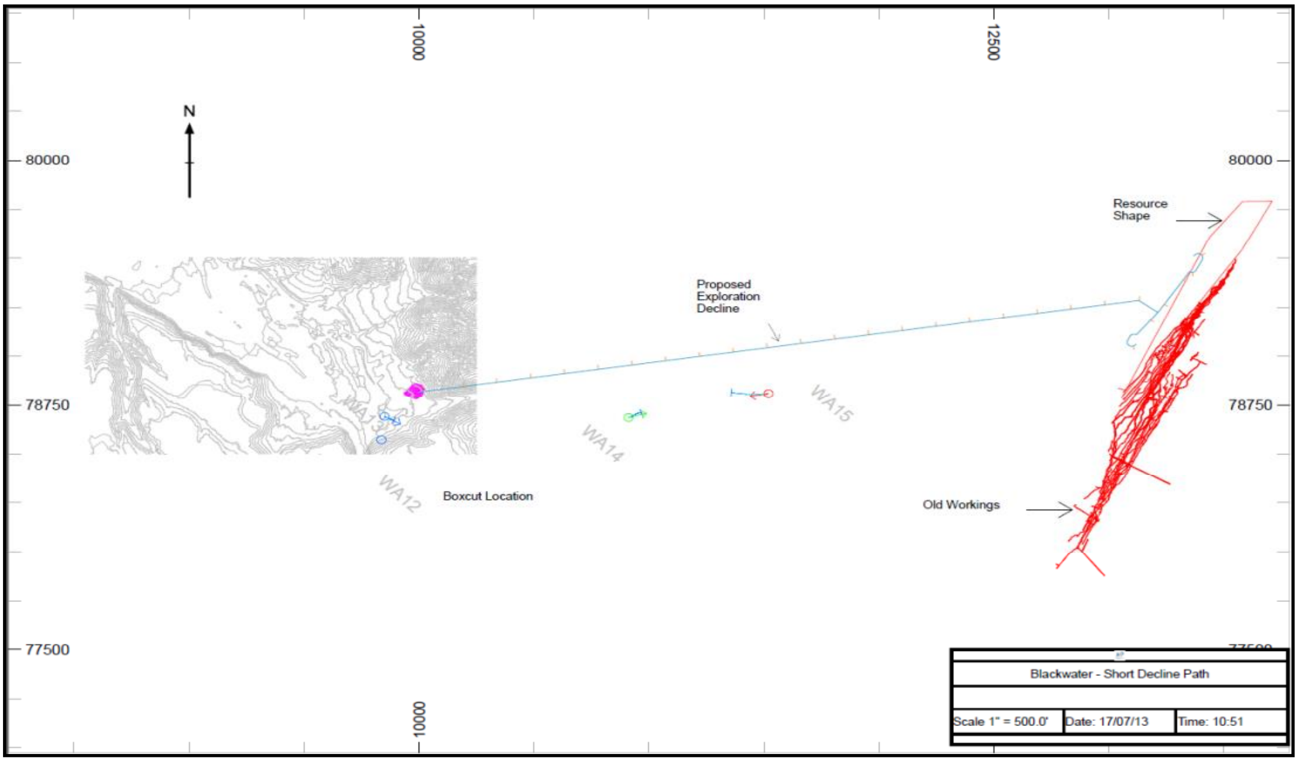
The discussion above is based largely on AMC's earlier reports and is still considered valid. The potential adverse effects on the development schedule, costs, productivity, and required development/stoping inventory cannot be better assessed until the proposed decline development and drilling programme have been undertaken.

6.4 Underground access and infrastructure

Because of limited surface access above the proposed mining operations, twin declines will be developed from the Snowy River site, with one serving as the main access and intake airway and the other serving as the return airway (Figure 6.3). This eliminates the need for ventilation raises to surface. Both declines are proposed to be 4.0 m high by 4.0 m wide with a gradient of 1 in 7. Concurrent mining of the two declines will provide effective ventilation during development without the need for raises, and better utilisation of the mining contractor with multiple faces available.

Mining-related infrastructure will include surface change room facilities, offices, stores, workshops, and magazines. These are part of the overall site infrastructure and are discussed in Section 9.

Figure 6.3 Plan view of twin exploration declines



6.5 Mining factors

The airleg resue mining method involves selectively mining a narrow quartz reef containing the gold prior to blasting the surrounding country rock as backfill. Inevitably, some country rock will be mixed with the ore and sent to the mill for processing, thus lowering the head grade of the material treated. An allowance of 50% for such material at zero grade has been made in the study.

Not all of the reef material will be recovered from the stope. Some will be left in non-recoverable sill pillars and some will become mixed with the country rock to the extent that its grade is too low to justify processing, and this ore will not be extracted or processed. An allowance of 10% has been made to account for this potential loss of ore.

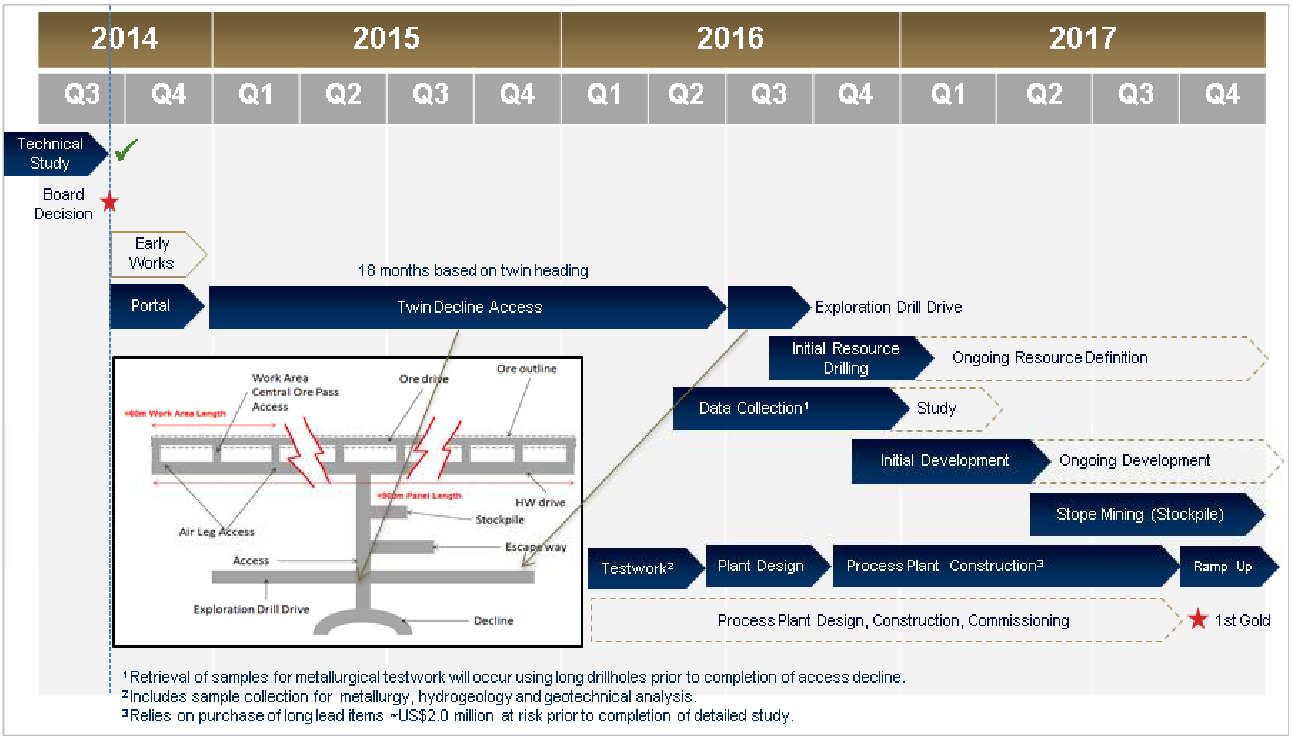
No minimum mining width has been applied to the reef, since it is really the metal accumulation (grade × thickness) that determines whether or not a particular section of the reef is economic or not. For this study, an average width and grade has been assumed for the reef. When the variability of the width and grade of the reef are better-understood from resource definition drilling, it might be possible to calculate a cut-off criterion based on metal accumulation.

6.5.1 Construction duration

The estimated time for construction of the twin exploration decline is 24 months that includes construction of the portal as well as the exploration drive adjacent to the reef, which will be used for resource definition drilling. Cover drilling of subhorizontal diamond drillholes in advance of the decline faces is proposed. This will provide prior indication of geotechnical and groundwater conditions.

Initial resource definition diamond drilling will be undertaken when the access decline is approximately three quarters completed. A drill platform will be mined from the access decline to enable diamond drilling whilst decline excavation continues. Core from these earliest holes will be used for confirmatory metallurgical testwork and preliminary assessment of the Birthday Reef. Figure 6.4 is a schematic showing an overview of the project timeframe.

Figure 6.4 Project timeframe



6.6 AMC comments

AMC considers that the proposed mining method, including access, ventilation, and ore haulage systems are technically and operationally feasible and appropriate for the current level of study. They will need to be reviewed when resource definition is better-understood and more geotechnical information becomes available.

AMC considers that the mining factors applied in converting the Inferred Mineral Resource to a production target are appropriate for the proposed mining method.

The project timeframe is ambitious, particularly in regard to the multiple programmes of testwork required, based on data collection from the resource drilling from the access decline. Any delays in collecting suitable representative samples, or delays in the metallurgical and geotechnical analysis, have the potential to impact the proposed start date for production.

7 Metallurgical factors or assumptions

7.1 Metallurgical testing

Ore from the Birthday Reef was historically known to be free-milling, with the majority present as coarse particles amenable to gravity collection and amalgamation. A small fraction was associated with pyrite (FeS_2) and arsenopyrite (FeAsS).

Gold-bearing ore from the Blackwater mines was economically mined for 43 years from 1908 to 1951. From 1908 to 1938, ore was processed at the Snowy Battery using amalgamation plates, Wilfley tables, and cyanide leaching of tabling tailings with gold deposition on zinc shavings using the Merrill-Crowe process. Gold recoveries of 89–90% were reported. A more efficient mill was constructed at Prohibition in 1938. The new flowsheet employed gravity collection and amalgamation for coarse gold recovery and flotation for recovery of fine sulphides. Roasting and cyanide leaching was used to recover gold from concentrates. Gold recovery was variously reported from 92.5% to 96%, with 80% of the gold free.

7.1.1 Ammtec Ltd testwork, 2003

In 1996, deep drilling at Blackwater produced two intersections of the Birthday Reef (WA11 and WA11A). A small sample of reef ore from WA11 was used to conduct indicative tests to confirm the effectiveness of a new flowsheet based on the old Blackwater plants. Testing was carried out in 2003 at the ALS Metallurgy (formerly Ammtec Ltd) laboratories in Australia.

The following results were shown:

- 97% gold recovery obtained by gravity and flotation.
- 80% of gold in flotation tailings was cyanide soluble.

7.1.2 Prohibition waste dump testwork, 2011

Testwork was carried out on a hand-collected sample (12 kg) of waste from the Prohibition waste dump (with mineralised material within the waste specifically selected for testing). The testwork was carried out by OceanaGold at the company's facilities in Macraes. The sample averaged 13.8 g/t Au and 0.1% S. It was processed with a Falcon L40 concentrator (gravity concentration) at a nominal size of 150 µm. Falcon tailings were processed by flotation. Recovery to combined concentrate was 98.6% with greater than 80% recovery from the Falcon unit.

A similar programme was undertaken on a reconstituted run-of-mine ore sample created as a 50:50 blend of hand-collected quartz and greywacke waste. The following results were reported:

- Combined recovery of 96% from Falcon centrifugal concentrator (91–92%) and flotation of concentrator tailings (65% recovery).
- Cyanide leach recovery of 99.8% on gravity concentrate.
- Cyanide leach recovery of 98.3% on flotation concentrate.
- No indication of refractory nature in the sample.
- Rapid filtration of flotation tailings, opening the possibility of filtration and dry stacking for disposal of tailings, rather than use of a wet tailings dam.

7.1.3 Gekko Systems testwork, 2011

Samples of quartz and greywacke were provided to Gekko Systems (Gekko) in Victoria for evaluation. The aim of the programme was to ascertain the amenability of Blackwater ore to processing using a standardised Gekko plant flowsheet (the Gekko surface python plant [Surface Python]). The standard layout consists of two-stage crushing (jaw crusher, vertical shaft impactor [VSI]), ball milling, inline pressure jig (IPJ), and flotation to produce a concentrate containing gold values. Concentrates are further processed using a standardised Gekko flowsheet (the concentrate treatment plant [CTP]) that utilises regrinding, cyanide leaching, resin adsorption and stripping, and cyanide destruction in plant tailings.

The following tests were carried out to confirm the suitability of the Surface Python for processing of Blackwater ore:

- VSI amenability test:
  - Quartz and greywacke samples were independently evaluated.
- Progressive grind tabling (PGT) tests:
  - 50:50 sample of simulated run-of-mine ore created.
  - 80.8% gold recovery into 1.35% of the tested mass, at P_{80} of 425 µm.
  - 97.9% overall gold recovery, after regrinding gravity tailings to P_{80} of 104 µm.
- Flotation tests:
  - 90% gold recovery, at a grinding P_{80} of 106 µm.
- Continuous Falcon test:
  - Use of a continuous Falcon unit to process PGT tailings without grinding was tested, but the method was unable to match the performance of grinding and flotation, and the option was eliminated.

7.1.4 Optical sorting evaluation, 2011

The potential for use of optical sorting techniques to upgrade bulk-mined, run-of-mine material was identified due to the historical close confinement of gold values to the quartz ore, and to the marked difference in colour between the quartz and surrounding greywacke waste. Bulk samples of greywacke waste and quartz were collected from the old tramway route from Blackwater to the Snowy Battery. Testing was carried out at Commodas Ultrasort Pty Ltd (Commodas) in Sydney.

Problems with sample preparation appear to have compromised the testing of finer-size fractions. However, coarse material showed the upgrading potential of the technique. In the +19 mm fraction, 98% of gold present was recovered to concentrate while 75% of feed mass was rejected.

Successful use of the technique would rely heavily on the assumption that no appreciable gold resides in the greywacke waste (as is simulated in the creation of synthetic feed ore by combining gold-bearing quartz and barren greywacke). This assumption would require definitive confirmation by sampling of Blackwater ore.

7.1.5 Gekko flowsheet validation, 2013

OceanaGold proposes use of two standard-format Gekko systems for the Blackwater plant; the Surface Python and the CTP. If the use of these packaged systems can be technically supported, capital costs for processing and plant construction time will both be reduced significantly, compared to full custom design and construction.

A further series of tests were conducted by Gekko in 2013 to validate the results of the earlier work, and to optimise and refine design criteria for the Gekko plants. Sample feed material was again created by combination of quartz ore and greywacke waste from the historical Blackwater site. The following results were obtained:

- The overall gold recovery achieved by gravity and flotation, followed by intensive cyanidation of concentrates was 96.5%, from feed with a gold grade of 5.9 g/t.
- 80% of gold in the feed could be recovered in a mass pull of 1.5% by gravity concentration.
- Grinding to a P_{80} of 150 µm is sufficient to achieve combined (gravity plus flotation) recovery to primary concentrate greater than 97%.
- Regrinding was confirmed as necessary to hold concentrate leaching time to a nominal six hours per batch, and to avoid gold recovery loss. Regrinding to P_{80} of 106 µm gave a leach recovery greater than 99%.
- Leach recovery was improved by 0.9% by electrowinning at elevated temperature (60° C rather than 25° C).
- Leach recovery can be negatively impacted by the presence of arsenic and antimony on the liquor. Testing confirmed that adjustments to solution chemistry to control the effects of arsenic and antimony result in leach recovery increasing from 97.5% to 99.9%.

7.2 Proposed flowsheet

OceanaGold proposes use of two standard-format Gekko systems for the Blackwater plant; the Surface Python and the CTP. General arrangement drawings for the two plants are shown in Figure 7.1 and Figure 7.2.

Figure 7.1 Surface Python general arrangement

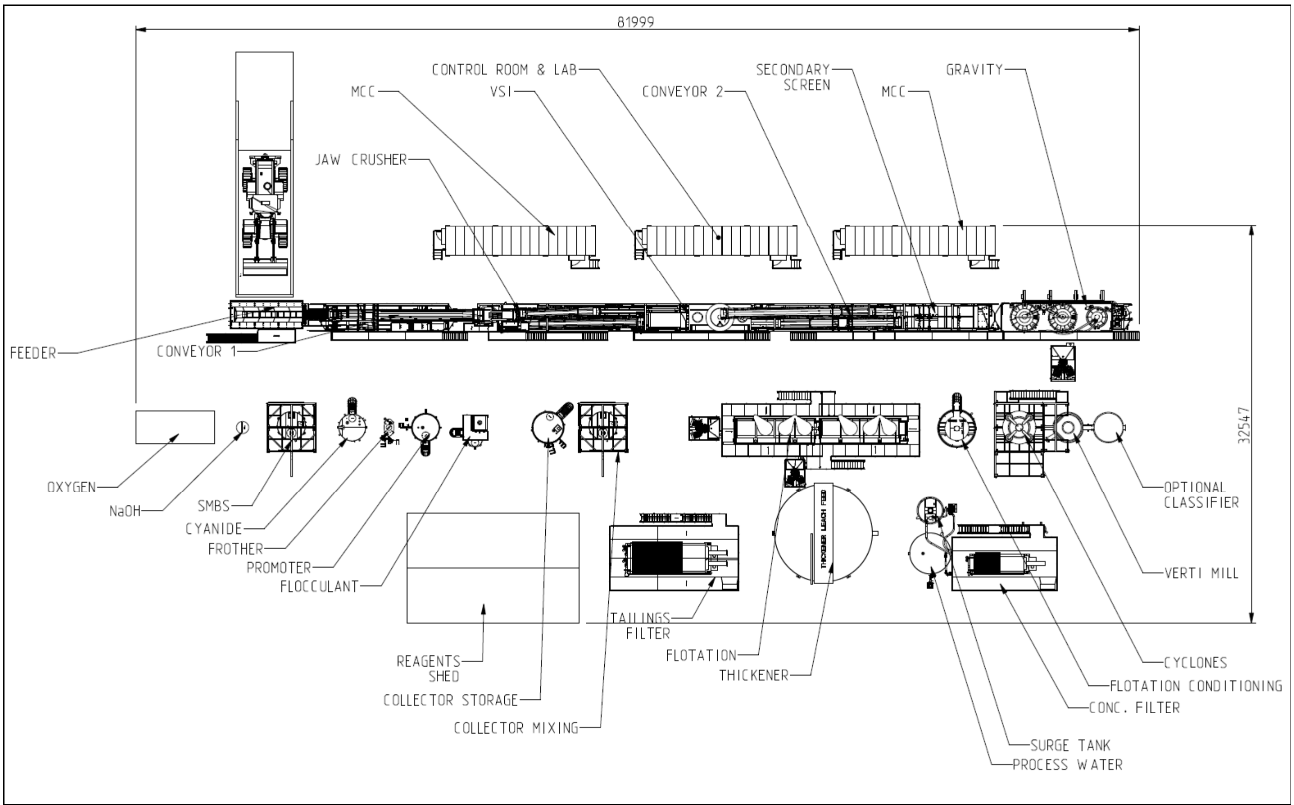
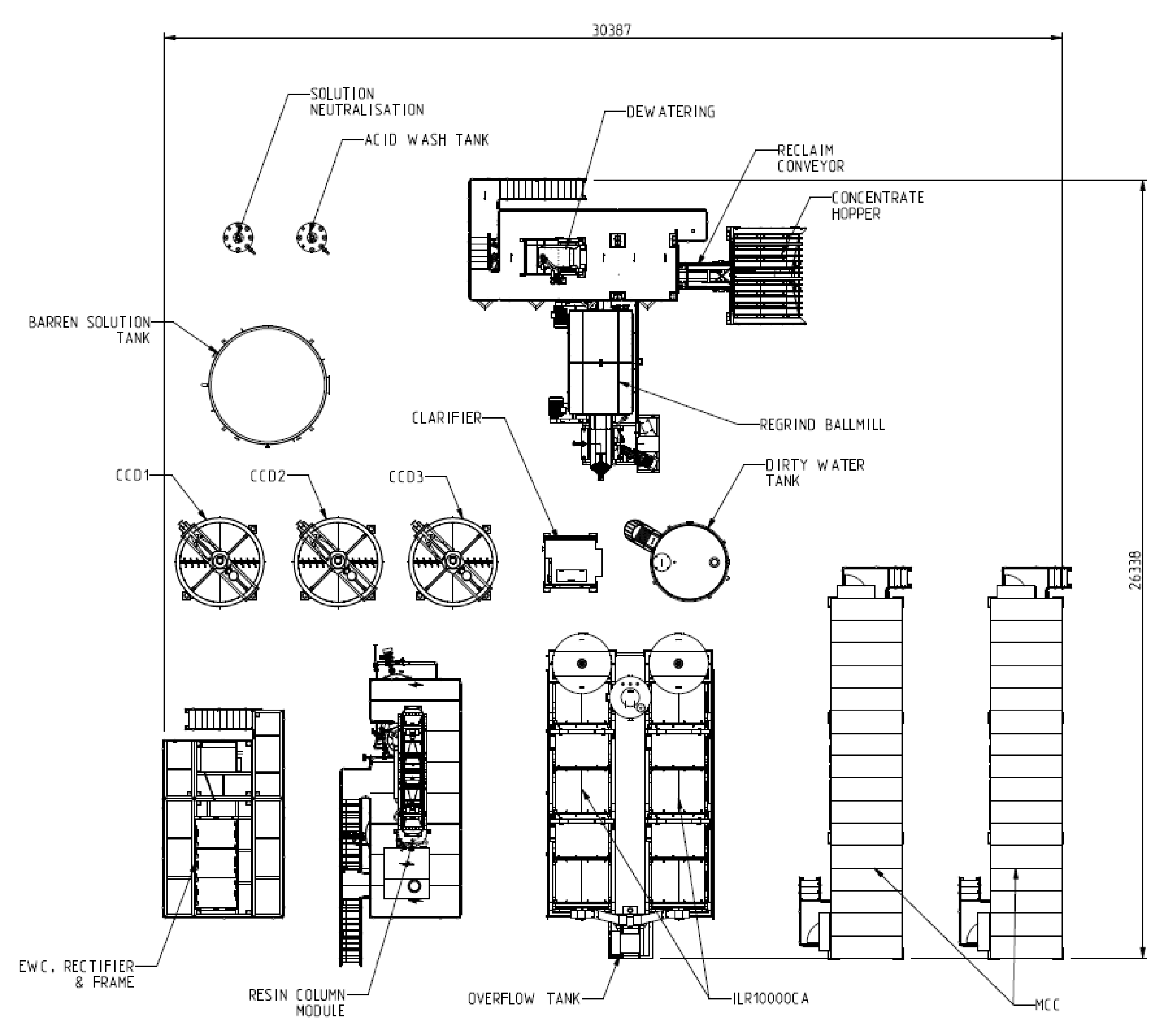


Figure 7.2 Concentrate treatment plant general arrangement



The plant has been sized to process 110–120 kilotonne per annum (ktpa) of ore at approximately 16 g/t Au that will be produced by the airleg resue mining method. The basic units of the plant are as follows:

- Run-of-mine (ROM) ore at P_{80} of 250 mm will be fed to the plant at 25 tonnes per hour (tph). Five days per week operation with 80% utilisation of time will result in a nominal throughput of 125,000 tonnes per annum.
- A jaw crusher with scalping vibrating grizzly screen will reduce size to approximately 35 mm.
- The jaw crusher is in closed circuit with a primary screen with 45 mm aperture.
- Primary screen undersize is fed to a secondary wet screen with aperture of 3 mm, producing a nominal P_{80} of 2.4 mm.
- Secondary screen oversize at +3 mm is fed to the VSI.
- VSI discharge reports to the secondary screen feed.
- Secondary screen undersize is pumped to the ball mill discharge sump, and then to the rougher IPJ.
- Rougher IPJ concentrate is pumped to the cleaner IPJ.
- Rougher IPJ tailings reports to grinding circuit (ball mill) cyclone feed sump.
- Cleaner IPJ tailings flows under pressure to the secondary screen feed.
- Cleaner IPJ concentrate reports to the gravity concentrate sump.
- Gravity concentrate is filtered and transferred by front-end loader to the CTP.

- Grinding circuit cyclone overflow at P_{80} of 150 µm flows to the flotation feed surge tank. The tank provides a nominal two hours of surge capacity.
- Cyclone underflow reports to the ball mill.
- Flotation feed is pumped from the surge tank to the conditioning tank where flotation reagents are added.
- Slurry overflows from the conditioning tank to the rougher flotation bank. The rougher bank provides 13 minutes of flotation residence time.
- Flotation tailings is thickened and filtered to produce cake with 14% moisture that is suitable for handling with mobile equipment and co-disposal with underground mine waste.
- Flotation concentrate is filtered and transferred to the CTP by front-end loader.
- Combined gravity and flotation concentrates are reground in a small ball mill in closed circuit with a Dutch State Mines (DSM) screen with apertures of 300 µm.
- DSM screen undersize flows by gravity to the concentrate intensive leaching reactor (ILR) where it is contacted with solution from the barren solution tank. Cyanide, lead nitrate, and sodium hydroxide are added at the barren solution tank, and oxygen is injected into the feed of the ILR. Residence time in the ILR is approximately eight hours.
- Oxygen is produced on-site using a pressure swing adsorption (PSA) system.
- Slurry from the ILR is filtered to separate pregnant leach solution (PLS) from leached filter cake.
- Filter cake is repulped and sent to the detox circuit where cyanide content is reduced using copper sulphate, sodium metabisulphite, and air. The circuit will utilise in-stream cyanide analysis and automatic control of reagents to ensure tailings meets required discharge standards.
- PLS is pumped to the resin column where it is contacted with gold ion-exchange resin (AuRIX) and gold is transferred to the AuRIX.
- Loaded AuRIX is removed batchwise from the column and stripped using heated (60° C) strip solution that is pumped through the column and the electrowinning cell where gold is plated out on stainless steel cathodes.
- Doré bars are smelted and poured using conventional gold room technology.

Primary input data for the design and sizing of the plant is shown in Table 7.1.

Table 7.1 Mass balance input summary

Stage	Units	P200 Python
Surface Python feed	t/h	25
Surface Python feed Au grade	g/t	16
Surface Python concentrate Au grade	g/t	261
CTP feed	t/h	1.5
CTP tail Au grade	g/t	2.6
Plant availability	%	80
Plant annual throughput	tpa	120,000

Nominal gold recoveries derived from the testwork and used in design and sizing of the plant are summarised in Table 7.2.

Table 7.2 Plant gold recoveries

Stage	Units	P200 Python
Au recovery from gravity	%	81.0
Au recovery from flotation	%	17.0
Overall Surface Python Au recovery (gravity + flotation)	%	98.0
Au recovery from leaching	%	99.0
Au recovery from resin adsorption, stripping, and electrowinning	%	99.0
Overall Au recovery from Surface Python and CTP	%	96.0

The plant flowsheet selected uses established technology in a somewhat innovative layout. In AMC's opinion, it is appropriate for the free-milling Blackwater ore. Good recoveries of coarse gold by gravity and fine gold by flotation are followed by intensive cyanidation dissolution from concentrates, and a cost-efficient recovery of the gold from solution using resin technology. This circuit arrangement has been used successfully by Gekko in a similar situation. The gold recoveries used to calculate plant performance are consistent with historical plant performance while processing Blackwater ore, and with testwork performed on available drill core and on remnant ore and waste material from the old processing sites.

7.3 Further study work

In AMC's opinion, a significant processing risk for the Blackwater gold project originates from the possibility that ore to be processed by the plant was not accurately represented by the samples used in the various testwork exercises conducted. The amount of drill core available was small, and the larger samples tested were synthetic combinations of historical remnant quartz, and historical greywacke collected from the old processing sites. While the similarity in metallurgical performance between the deep drill core samples and the synthetic feed samples lends credence to the contention that test results obtained from the overall program are valid, some risk must remain that the actual ore to be processed will perform differently, and that the plant will not yield the metallurgical performance claimed. AMC therefore recommends that representative samples of reef material that will be processed be collected from the exploration decline, and that the following testing be conducted:

- Variability testing using reef samples to confirm the design criteria of the final flowsheet, and to confirm recovery assumptions.
- Mineralogical examination of flotation concentrates from the reef samples, to understand how gold is locked up and to determine if product is refractory; that is, requiring treatment through autoclave leaching (as would be possible at the OceanaGold Macreas plant).

In addition, OceanaGold has identified further testwork to be completed:

- Additional intensive leach tests on gravity/flotation concentrate to confirm parameters of the resin upgrade circuit prior to electrowinning.
- Assessment of the impact on concentrate recovery and mass pull of a flotation concentrate cleaning stage to minimise the amount of leached material.
- These tests could also be conducted using the fresh reef sample material.

8 Licensing, environmental, and social

8.1 Mineral tenement and land tenure status and third-party interests

8.1.1 Mineral tenement status

The tenement status has been reviewed by Anderson Lloyd Lawyers and written advice (letter dated 15 September 2014) has been provided to AMC. Pertinent information regarding mineral tenure is provided below.

Rights to prospect, explore, or mine for minerals owned by the Crown are granted by permits issued under the *Crown Minerals Act 1991*. Crown-owned minerals include all naturally occurring gold and silver.

OceanaGold holds an exploration permit, EP40 542 (Blackwater EP), over an area of 4,308 hectares, which includes the relevant area of interest for the purposes of this ITR. AMC has sighted the formal instrument of title for Blackwater EP. Figure 8.1 shows the permit boundaries.

The Blackwater EP is now in its twelfth year, with a current four-year term for appraisal purposes that runs through to 18 November 2016. The *Crown Minerals Act 1991* imposes relinquishment requirements for the reduction in area of any exploration permit (EP) by up to 50% over the initial 10 years of the permit's life. Accordingly, the Blackwater EP has been reduced from its original size, on first grant in November 2002, of approximately 9,000 ha. The *Crown Minerals Act 1991* allows a single further extension of the EP of up to four years for appraisal purposes, if certain conditions are met.

Within the term of the Blackwater EP, the *Crown Minerals Act 1991* effectively establishes OceanaGold's title to the gold and silver in the Blackwater EP permit area, provided that OceanaGold complies with the conditions of the permit and the requirements of the Act and the *Minerals Programme for Minerals (Excluding Petroleum) 2013*. These requirements include the payment of annual fees, completion of the agreed work programme and, in due course, the payment of Crown royalties in respect of the gold and silver that is mined.

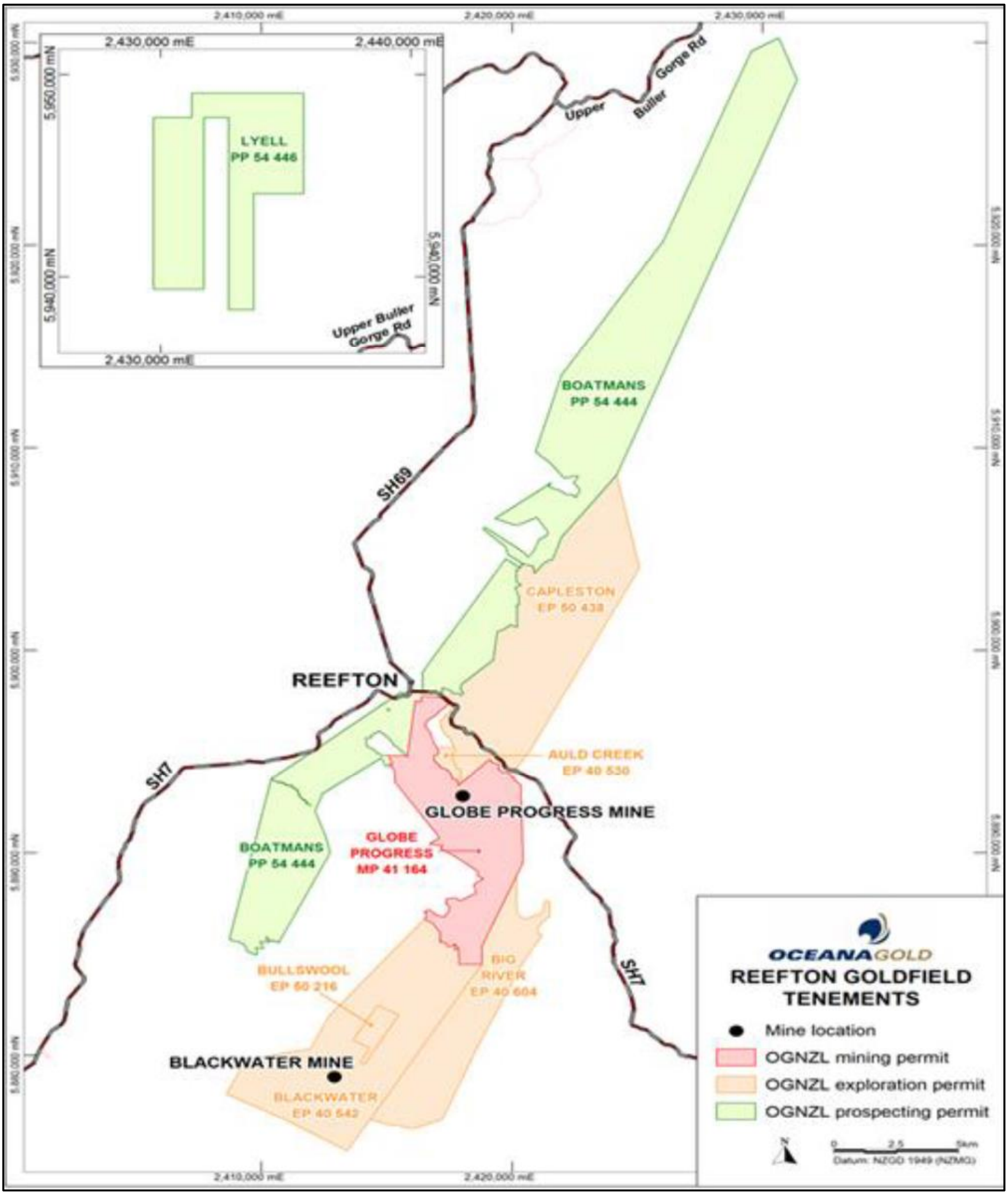
Under the *Crown Minerals Act 1991* (subject to the specific conditions of the permit), title to minerals is established through the following process:

- The holder of an EP has the right to prospect and explore for the Crown-owned mineral, in the land, and on the conditions, stated in the permit.
- If the holder of an EP satisfies the Minister that they have, as a result of activities authorised by the permit, discovered a deposit or occurrence of a mineral to which the permit relates, the permit holder has the right, on applying before the expiry of the EP, to surrender the permit insofar as it relates to the land in which the deposit or occurrence exists and to be granted in exchange a Mining Permit (MP) for that land and mineral. Ordinarily, an Indicated Mineral Resource is required to advance to a MP, however Inferred Mineral Resource is also taken into consideration, and OceanaGold anticipates that early results of resource definition drilling will form the basis for a MP application.
- The holder of a MP has the right to prospect and explore for and, in addition, a right to mine the Crown-owned mineral, in the land, and on the conditions, stated in the permit.
- The rights to prospect, explore for, and mine the Crown-owned mineral are exclusive to the permit holder.
- A second permit conferring rights in respect of the same land and the same mineral may only be granted to a person other than the permit holder with the prior written consent of the permit holder.
- The holder of a MP is the owner of all minerals lawfully obtained by or on behalf of the permit holder in the course of activities authorised by the permit.

A mining permit may be issued for a maximum period of 40 years. The duration granted will depend upon the extent of mineral reserves in the land, the permit holder's resources, and its work programmes.

New Zealand Petroleum and Minerals confirmed that (on 28 August 2014) OceanaGold is compliant with the general conditions and work programme conditions for Blackwater EP and that OceanaGold has already complied with some of the 168 month work programme conditions that are not due to be completed until 18 November 2016. AMC concludes that the Blackwater EP is currently in good standing, and that a clear pathway for conversion of the EP to a Mining Permit appears to be available.

Figure 8.1 Blackwater EP boundaries and adjacent permits



8.1.2 Real property tenure and underground mine tenure requirements

The underground workings of the proposed Blackwater mine will pass through land owned by various parties, including Crown land administered by the Minister of Lands on behalf of the Crown, public conservation land owned by the Minister of Conservation, and land in private ownership.

The law governing the requirement, if any, for landowner consent to mine under the surface of land, is found in the *Crown Minerals Act 1991*. This Act modifies the position under the general law, which would otherwise allow any landowner whose land was affected by mine workings, whether above or below ground, to withhold access if they wished or grant a licence to enter onto, or into, their land on whatever terms they deemed fit.

As a result of the code established by the *Crown Minerals Act 1991*, OceanaGold will not require any access arrangements with the owners of the land through which the Blackwater mine underground workings pass. Where land administered by the Department of Conservation is concerned, there will also be no requirement for a concession, where the land is within an EP or MP area, because the *Conservation Act 1987* (CA) provides that a concession is not required in respect of any mining activity authorised under the *Crown Minerals Act 1991*.

Notwithstanding the position under the *Crown Minerals Act 1991*, OceanaGold has consulted with relevant landowners in the course of obtaining resource consents, in relation to which both Department of Conservation and the local Rūnanga, Ngāti Wae Wae were held to be affected parties.

The Department of Conservation has given affected party approval to allow the company's resource consent applications to proceed. The local Rūnanga, Ngāti Wae Wae, were also consulted and provided affected party approval to the Blackwater resource consent applications.

8.1.3 Real property and indigenous land status

8.1.3.1 Snowy River site

OceanaGold has entered into an option agreement to acquire land required for surface infrastructure works at the Blackwater mine site (the Snowy River site). The land comprises Section 10 Block XIV Mawheraiti Survey, District Nelson Land District, NL10A/347, with an option to also acquire all or part of Section 9, over which access will be gained to the Snowy River site.

Rights to the Snowy River site (and associated private-land access) are secured until 21 April 2016, under the Access Arrangement for Mineral Prospecting, Exploration and Mining and Option Agreement between OceanaGold and the landowner dated 21 October 2010 (Access Arrangement) as varied by agreements dated 21 April 2013 and April 2014.

The Access Arrangement is capable of registration against the land concerned and contains protections against assignment without OceanaGold's consent, except on terms that bind a subsequent owner to honour the Access Arrangement. OceanaGold's option to purchase under the Access Arrangement is protected by a caveat against dealings.

The company may, at any time up to and including 21 April 2015 (or, upon payment of an additional option sum before that date, 21 April 2016), exercise the option by giving written notice to the landowner. That notice will bind both parties to execute an agreement in the form of the Real Estate Institute of New Zealand/Auckland District Law Society Agreement for Sale and Purchase of Real Estate.

The purchase of the Snowy River site will also be subject to receiving subdivision consent under the *Resource Management Act 1991* and regulatory consent under the *Overseas Investment Act 2005* which deems OceanaGold to be an "overseas person". The main points to note in relation to that latter process are:

- The process can be expected to take three to five months.
- Consent is likely to be conditional on ongoing reporting to the Overseas Investment Office. Conditions may also be imposed requiring divestment of unused land within set timeframes and undertaking heritage and/or ecological survey work on the site.
- The company has a successful track record of obtaining Overseas Investment Office consent for land purchases at Macraes, and it is considered unlikely that consent would not be obtained.

8.1.3.2 Blackwater River site access track

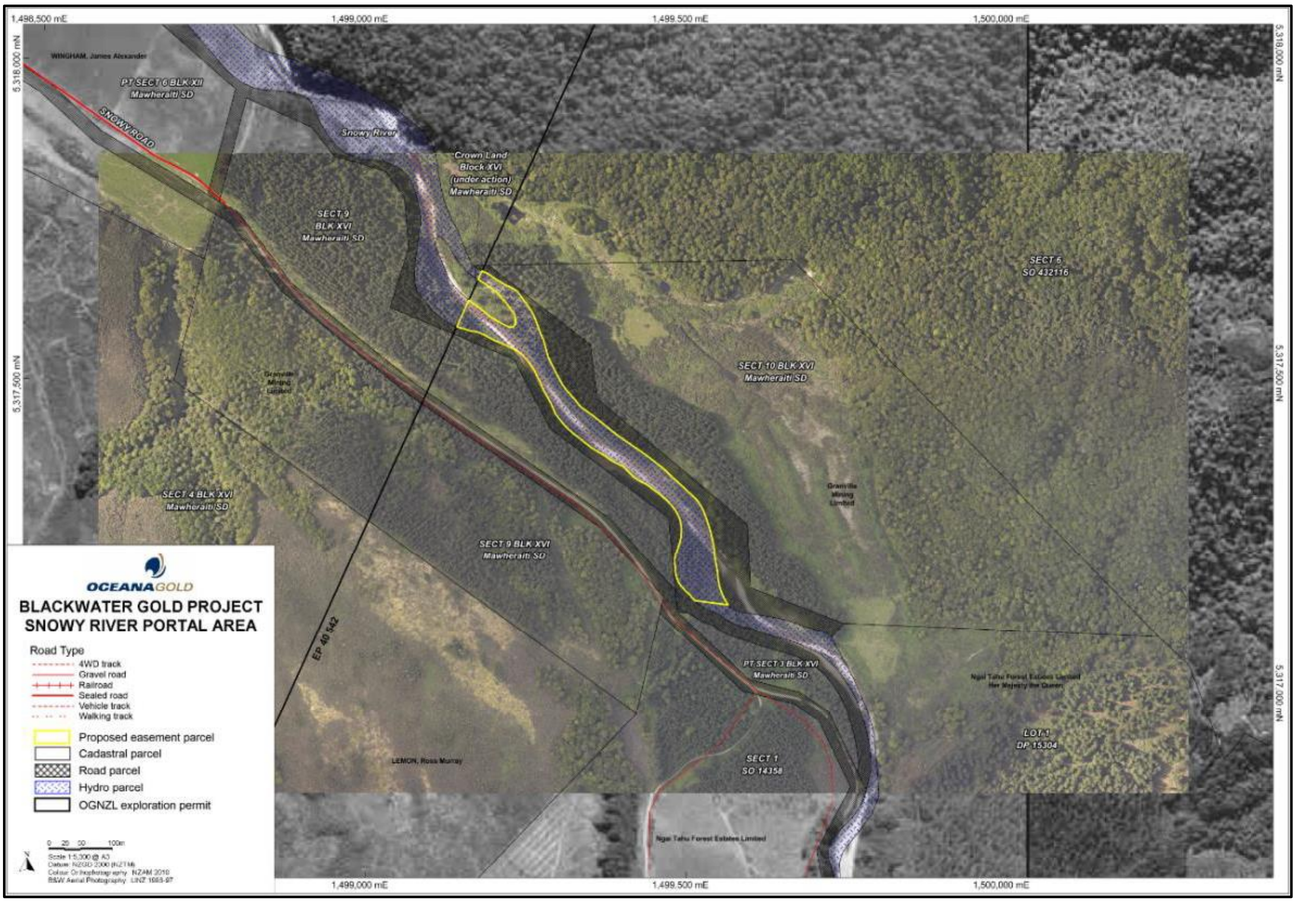
Access to the Snowy River site is from Snowy Road across land (and riverbed) owned by the Snowy River site landowner, the Crown and Buller District Council.

Private-land access to Snowy River site might be secured through seeking to agree an easement with the private landowner, or in the unlikely event that the company was unable to register an easement, exercising the option to purchase the land pursuant to the Access Arrangement.

The riverbanks and riverbed of the Snowy River are owned by the Commissioner of Crown Lands (refer to the areas shown as "hydro parcels" in Figure 8.2). A Crown easement will be required, to allow the company to construct a bridge across the river. This easement is unlikely to be difficult to obtain.

The Snowy River is flanked by areas of road reserve, owned by the Buller District Council (refer to the areas shown as "road parcels" in Figure 8.2). A Council Licence to Occupy will be required, to allow the company to access a bridge across the river. This licence is unlikely to be difficult to obtain.

Figure 8.2 Snowy River site real property ownership



8.1.4 Third-party interests

In addition to Crown royalties, the Blackwater EP is also subject to an agreement contained in a Deed of Novation with Royalco Resources Pty Ltd (Royalco). This is discussed further in Section 10.2.6.

8.2 Permits and approvals

8.2.1 Overview and approvals status

Mining and environmental planning approvals of mining projects in New Zealand are provided under the *Crown Minerals Act 1991* and the *Resource Management Act 1991*, respectively. The two approvals processes are tenuously interlinked, in that the Minister administering the *Crown Minerals Act 1991* must be satisfied that the permit holder will obtain necessary environmental approvals.

OceanaGold holds a suite of resource consents from the West Coast Regional Council and Buller District Council that authorise the proposed exploration and mining activities associated with the Blackwater gold project. Those consents were based on an earlier version of the project that did not include on-site ore processing and tailings deposition. Many of the existing resource consents will cover the revised project, some will require amendment, and some new resource consents will be required.

Legal opinion was obtained by OceanaGold (letter from Anderson Lloyd Lawyers dated 15 September 2014) regarding the existing and outstanding approvals and authorisations required for the project under the *Crown*

Minerals Act 1991 and the *Resource Management Act 1991* and provided to AMC. The legal opinion concluded:

- OG has updated its Environmental and Social Impact Assessment to include these elements and concludes that the necessary changes or additions to its consent to authorise these additional elements should be able to be secured. Provided the technical assessments that OG prepares to support these amendments to its consents confirm that ore processing and tailings deposition will not result in unacceptable risk or adverse effects in terms of fundamental matters such as geotechnical stability, water and air quality, and ecology, we concur that the appropriate consent variations and/or new consents should be attainable. We note that we are not aware of any reason why the technical assessments would indicate any significant risks or adverse effects.
- It is therefore our opinion that OG holds the necessary resource consents for all aspects of the proposed Blackwater Gold Project other than on-site ore processing and tailings deposition. Based on the information available to us at the date of this opinion we have no reason to think that OG will be unable to secure the necessary changes/new consents to incorporate on-site processing and tailings deposition into the overall project.

8.2.2 Approvals requiring amendment and other outstanding approvals

8.2.2.1 Resource consents

OceanaGold is in the process of updating its Environmental and Social Impact Assessment (ESIA) documentation to include consideration of on-site processing and tailings management. The ESIA will form the basis of applications to amend existing resource consents and application for new resource consents, as required.

8.2.2.2 Building consents

An application will be made to the Buller District Council for any building consent required for the project.

8.2.2.3 Heritage

A post-1900 water race is located in the western area of the property, which OceanaGold might reuse as a diversion drain. Although not legally bound to protect the works, OceanaGold has committed to protect the water race and undertake any developments of the structure with input from Heritage NZ.

8.2.2.4 Environment Protection Authority: handling storage and transport of hazardous goods

Permits for the transport, storage, and use of hazardous goods will be required for the project. These will be obtained as the project moves into development.

8.2.3 AMC conclusions regarding approvals status

AMC considers that there are reasonable grounds to expect that all necessary government approvals will be received within the timeframes anticipated in the project development programme.

The material outstanding approvals are the new resource consents and amendments to existing resource consents pertaining to on-site processing and tailings storage. Specifically, approval for a waste rock stack has been obtained, but approval for on-site processing and on-site disposal of tailings is yet to be obtained, and is currently in progress.

AMC considers that there are reasonable grounds to conclude that the consents will be achievable, given the following:

- The precedence for resource consents for this project and other OceanaGold projects in the Reefton area that do involve processing.
- The existence of established regulatory and permitting processes.
- The relatively small scale of the project.
- The impact management measures proposed for the project, and low likelihood of residual or unmanageable environmental impacts.

8.2.4 Compensation

A range of compensation measures have been agreed with various parties as part of securing affected party approvals for the issue of resource consents or as a result of arrangements dating back to the 2004 application where these are effectively retriggered by the current application. Generally, the obligation to undertake the various measures agreed is conditional on the project proceeding.

Consent conditions around the monitoring of conditions affecting aquatic life in the Snowy River have been agreed with Fish & Game New Zealand.

Consent conditions have been agreed with Land Transport New Zealand (LTNZ), requiring the Snowy Road intersection to be widened in accordance with NZ Transport Agency rural road intersection rules, realignment of the centre of Snowy Road, full seal widening in accordance with NZ Transport Agency rural road rules to be undertaken on State Highway 7 south and north of the Snowy Road intersection, and widening of the Snowy Road approach to the intersection with State Highway 7 at an estimated cost of NZ$200,000.

8.2.5 Environmental bonds

OceanaGold will be required to provide bank guarantees to secure the obligations to comply with the conditions of the resource consents, including remediation of the Snowy River site.

OceanaGold will be unable to activate its resource consents until it has furnished the Councils with a bank guarantee or guarantees (bonds) for the full amount of the estimated cost of having third parties undertake rehabilitation, covering the expected environmental impact of the company's forecast activities over the first 12 months of development operations (the bond sum).

The bond sum will be reassessed annually and increased or decreased to take into account activities to date and forecast activities for each successive year of operations. OceanaGold will be required to maintain bonds in place for the applicable bond sum applying from year to year. OceanaGold advises that it is in a position to furnish all necessary bonds based on its proposed activities.

A preliminary estimate assesses the likely bond amount for the first year of operations to be in the range of US$250,000 to US$500,000. AMC has reviewed the calculations and considers this to be a reasonable preliminary estimate of the bond.

8.3 Environmental factors

8.3.1 Environmental overview of the Snowy River site

An assessment of environmental effects (AEE) has been prepared for an earlier version of the project, and includes baseline environmental surveys of various environmental aspects of the project and the Snowy River site. The following assessment is based on information provided in the AEE and site observations by AMC.

OceanaGold (PEA September 2014) states that the most significant constraints on the site infrastructure layout design are as follows:

- Snowy River flood level: the 1 in 1000 year Snowy River inundation level is 192 m RL (Golder Associates (NZ) Limited, 2013). All life-of-mine infrastructure will be located above this elevation, and the portal will be 10 m above this level.
- Property boundary: the property is bound on the north and east by the Victoria Forest Park, south by Ngāi Tahu land, and the Snowy River on the west. The locating of the infrastructure outside the current property boundaries was not considered due to uncertainty of land acquisitions, timing of any acquisitions, and OceanaGold preference to keep infrastructure within the project area.
- Victoria Forest: the Victoria Forest Park extends into the eastern area of the property. The area is steep and heavily vegetated, therefore the infrastructure layout was designed to avoid unnecessary disturbance of the area, which would create sediment and possibly slope stability issues.
- Portal location: the location of the portal and ventilation shaft were considered a fixed entity for the infrastructure layout design.

- Pine plantation: pine trees have been planted along the western boundary of the property by a previous landowner. Although the plantation can be cleared, the infrastructure layout was designed to avoid unnecessary clearing activities that would cause sediment issues plus have cost implications due to timber harvesting agreements made with the landowner. The pine trees also provide a visual buffer between the public road and the site.
- Existing water race: a post-1900 water race is located in the western area of the property. Although not legally bound to protect the works, OceanaGold has committed to protect the water race and undertake any developments of the structure with input from Heritage NZ.
- Wetlands: wetlands created along the western property boundary will dilute the treated water discharges from the project area.

Based on AMC's site observations, the Snowy River site is on a single parcel of privately held land, is in a remote location, in an area previously used for mining and mineral processing, at least 1 km from the nearest residential receptor, located on a previously disturbed site, with good vehicle and power accessibility, and existing visual screening.

The ecological values of much of the site have likely been affected by previous and current land uses (alluvial mining, grazing, silviculture) and existing weed infestations (Gorse [*Ulex europaeus*]). As such, many of the usual social and environmental issues associated with mining/mineral processing will be avoided or mitigated by the location of the project.

8.3.2 Contaminated land and other potential environmental liabilities

The Snowy River site is downstream of the location of the historical Snowy Battery (gold ore processing plant) last operated more than 75 years ago. The method of gold recovery is not known, but might have been mercury amalgamation. Consequently, there might be a risk of mercury or other contamination at the site. Much of the site has historically been mined by alluvial mining processes, and subsequently was used for silviculture.

OceanaGold reports that, to date, no contaminated land assessments have been carried out. AMC considers that the historical land uses introduce a potential risk of land or groundwater contamination. OceanaGold plans to carry out contaminated land investigations as part of geotechnical investigations to be carried out on the site, prior to exercising its option to acquire the land.

8.3.3 Disposal of mine waste rock and tailings

8.3.3.1 Overview of waste rock and tailings

Current approvals allow for 1.1 Mm^3 of waste rock capacity. OceanaGold estimates that additional capacity of up to 25% might need to be consented to accommodate the additional volume associated with co-disposal of tailings, after allowing for a reduction in waste rock generated from a slightly reduced decline profile to that on which current resource consents were based.

The updated technical reports supporting the ESIA have been obtained on this basis. They indicate that it is reasonable to assume that the variations to the resource consents to accommodate this additional capacity can be obtained.

8.3.3.2 Waste rock

Development waste from construction of the twin decline in the pre-production phase and vertical development during mining will be hauled to a waste rock dump at surface, for co-disposal with flotation tailings. Waste generated during production will be retained in the stopes as backfill.

During the mining phase, the production of development waste is expected to be intermittent, based on a "just in time" vertical development mining strategy.

8.3.3.3 Flotation tailings

The flotation tailings will be thickened, and pumped to the tailings filter for further dewatering to produce a cake with less than 14% moisture content, enabling the filtered flotation tail to be reclaimed and transferred to the waste rock dump, for encapsulation in the dump. The dump, including the tailings cell(s), is expected to be free-draining.

A number of additional studies are required to confirm the co-disposal strategy for flotation tailings:

- A detailed co-disposal study will be undertaken to design the waste rock dump to allow encapsulation of all flotation tailings produced during the operation of the plant. Flotation tailings are expected to match volumes of operating phase waste rock on a roughly 1:1 ratio. The periodic availability of waste during horizontal development will require planning of dump designs to minimise risks of dusting and run-off.
- Mobilisation of metals, in particular arsenic and antimony, in the leach stage might increase levels of soluble metals in mill process water to levels above that acceptable for release. Compilation of a site-wide water balance will allow modelling of water release rates and acceptable levels of metal ions in solution. Additional intensive leach testing would allow a better understanding of the ideal chemistry to minimise metal mobilisation or at least to characterise the levels to be expected and determine if a precipitation circuit will be needed for water treatment.
- Tailings filtration capacity will have a significant impact on plant throughput with drops in filter availability directly impacting on plant utilisation and a requirement to meet a moisture level suitable for dry stacking or co-disposal. Scoping filtration tests have been undertaken by Outotec to allow sizing checks for a Larox PF style of filter suitable to the duty to be made before detailed engineering will be undertaken.

8.3.3.4 Concentrate tailings

Tailings from the detox circuit will be filtered to produce a cake with less than 14% moisture content. Filtered tailings will be available to mix on the surface with cement and transfer underground for use in the marker beds. Approximately 12,000 t per year of tailings will be required for the marker beds and will consume all the leached concentrate tailings and a small portion of the flotation tailings. Sequestering the leached tailings in the cemented marker beds will likely provide a long-term solution to minimise any acid and metalliferous drainage (AMD). Post-closure, as the mine workings flood, the marker beds will be maintained under water in an anaerobic state to provide further long-term protection from AMD.

8.3.3.5 Acid and metalliferous drainage

O’Kane Consultants (NZ) Ltd (OKC) was retained by OceanaGold to complete a preliminary review of AMD at the Blackwater project. That review has been completed based on a limited amount of information, and further investigations will be required.

OKC concluded that, in general, assumptions made for the management of AMD, utilising the OceanaGold’s experience at Globe Progress mine near Reefton to predict water quality and the effects of AMD, appear reasonable.

Further work is required to characterise the rocks (and tailings) in regards to geochemistry and forecast water quality to reduce project uncertainties. Amongst other things, geochemical data is needed on the flotation tailings and concentrate tailings including acid base accounting and leach testing to derive potential contaminant loads. This information will be required to determine the stability of the tailings and waste rock under different environmental conditions.

The effects of AMD at Blackwater are likely to be minor, provided the potential risks are predicted and managed appropriately. Predictive testwork is still required to confirm the conceptual approach.

Acid drainage from the Blackwater underground workings is potentially manageable due to the presence of carbonate minerals and, hence, elevated acid neutralising potential within the surrounding wall rock and waste rock.

Metalliferous drainage has potential to be an issue at the site, requiring management of contaminants such as As, Fe, SO_4, and other metals.

Given what is known about these risks, OceanaGold’s concept proposals for a water treatment system to manage low dissolved oxygen mine waters elevated in As and Fe appear reasonable, subject to further work to confirm the approach taken and ensuring that forecast water treatment requirements make adequate provision for additional load from the waste rock stack and any tailings. Management of the AMD treatment plant sludge produced needs further consideration.

It should be assumed that any waste rock stack at Blackwater will utilise paddock dumping to minimise longer-term contaminant loads, consistent with the approach taken at the Globe Progress mine.

Pending further investigations, it should also be assumed that the waste rock pad used to elevate the site infrastructure and the waste rock stack itself will require a drainage system to direct basal drainage to the water treatment system, in order to mitigate any elevated Fe, As, and sulphate that might occur.

A waste rock management plan needs to be developed for the project that integrates tailings management, water management, treatment of water, and closure considerations

8.3.4 Hazardous chemicals, cyanide management, and detoxification

8.3.4.1 Cyanide detoxification

OceanaGold proposes a cyanide detoxification process as part of its processing plant. The process is described as follows:

- In the repulp tank, the filter cake is mixed with excess barren solution and process water and agitated to make a 40% solids slurry. This slurry is pumped to the detox tank where copper sulphate, sodium metabisulphite, and air are added in stoichiometric quantities to react and neutralise the weak acid dissociable (WAD) cyanide in solution.
- An on-stream cyanide analyser measures the WAD cyanide in solution at the discharge of the repulp tank and also at the detox tank discharge. The analyser is able to detect changes in the WAD cyanide concentration and control the level of detoxification by copper reagent addition to the tanks. The detox slurry is then pumped to the detox tail facility for co-disposal.

Accordingly, the following hazardous chemicals will be transported, stored and used on site:

- Sodium cyanide
- Sodium metabisulphate
- Sodium hydroxide
- Hydrochloric acid
- Oxygen

OceanaGold operates a similar facility at its gold processing plant at Macraes in East Otago, and is considered experienced in the safe management of these chemicals and processes.

8.3.5 Groundwater seepage

OceanaGold (Golder Water Management report) advises that no hydrogeological studies of the site have been undertaken to confirm the raw water supply risk (capacity and quality). AMC understands that detailed groundwater studies have not commenced for the decline tunnels and Snowy River site, and that these will be considered as part of future geotechnical investigations. These studies will include investigation of potential yield for water supply.

Following completion of the access declines and dewatering of the historical workings, it is expected that groundwater flows to operational areas of the mine would decrease. Mine pumping records from the operational period of the historical workings, as well as discharges from the low level adit, are less than 10 L/s. It is expected that long-term seepage flows into the access declines would also total less than 10 L/s. Water flows requiring management in the operational area of the mine, including declines, are therefore in the order of:

- Less than 10 L/s decline water.
- 10 L/s main panel and access drive water.
- 15 L/s water introduced to the mine for operational purposes.
- Total of less than 35 L/s.

Construction of deeper mine drives and panels within the orebody would lead to additional groundwater flows to these workings. These flows would, however, be offset against reduced seepage into higher levels of the mine. The hydraulic conductivity of the rock mass will generally decrease with increasing depth.

The eventual total rate of groundwater flow into the mine is uncertain as the orebody currently remains open at depth and the eventual extent of the mine is not clear. It can reasonably be expected that the maximum groundwater flows into the mine during the ore extraction period would be substantially less than the flows managed during the dewatering phase of the historical underground workings (i.e. 50 L/s) and comparable to the inflows to the planned declines.

A sump is to be located on every level access. The sumps are vertically aligned to allow interconnecting drain holes between the levels. A mobile pumping station will be used in the deepest sump in the mine at any stage of development

8.3.6 Mine water management system

The overall philosophy of the proposed mine water management system is summarised as follows:

- A closed-loop processing facility, including the use of thickeners and filter presses in the tailings management system to recover process water for re-use.
- The isolation of historical mine water and orebody dewatering water from mine seepage, due to their differing chemical and settling properties, for treatment in the mine water treatment plant, and subsequent storage, reuse, or discharge to constructed wetlands and the Snowy River.
- Diversion (via diversion drains and berms) of clean water around the Snowy River site to existing drainage lines, to reduce the volume of mine-affected water, and maintain flow and quality of surface water drainage.
- Collection of underground mine-affected water (mine operational water and groundwater ingress) in sumps in the underground workings, and pumping to surface mine water treatment plant for treatment, and subsequently, storage, dilution, re-use, or discharge (if required).
- Collection of surface mine-affected water (site run-off, waste rock, and tailings drainage) in drains and channels, and treatment in sediment dams and the mine water treatment plant, for subsequent storage and re-use, or discharge.
- Installation of groundwater bores for raw water supply (for treatment to potable water standards) and for supplementation of process water supply, if required.
- Installation of a potable water treatment plant and distribution system for potable water supply, and a domestic wastewater collection system and treatment plant for domestic wastewater management.
- Installation of constructed wetlands for polishing wastewater treatment plant effluent prior to discharge to the Snowy River.

The Blackwater project will be designed to include sumps and wetlands, to meet appropriate water quality standards in receiving waters (the Snowy River), which are based on criteria to protect aquatic life in the sensitive river ecologies involved.

The mine water (both water from dewatering of the historical underground workings at Waiuta, and seepage from the orebody during mine operations) can be treated for elevated iron and arsenic concentrations by encouraging the natural processes of ferrous iron oxidation. The resulting water quality is predicted to be close to the existing water quality of the Snowy River.

8.3.7 Heritage

OceanaGold will site project elements to avoid compromising areas of greatest value and has adopted an approach, in consultation with the Heritage New Zealand, that seeks to protect in situ all existing heritage values and, at the completion of the project, to remove all new infrastructure.

A post-1900 water race is located in the western area of the property, which OceanaGold might reuse as a diversion drain. Although not legally bound to protect this feature, OceanaGold has committed to protect the water race where possible and undertake any developments of the structure with input from Heritage NZ.

The site infrastructure as currently planned will not affect any other identified historical sites for which authority to modify or destroy is required.

8.3.8 Other environmental factors

The following environmental factors have been considered in previous environmental impact assessments for the project:

- Flooding
- Receiving water quality
- Protection of aquatic ecology
- Terrestrial ecology
- Visual effects
- Noise and vibration
- Air quality
- Subsidence
- Traffic and transport

The assessments found that the potential impacts of the project were able to be managed (avoided or mitigated) through the implementation of routine or relatively straightforward impact management measures, such that the residual potential impacts to the environment were negligible or within regulatory limits.

8.3.9 Mine closure

After closure of the mine, the underground workings are to be allowed to fill naturally with groundwater. Once filled to the level of the decline portal, a discharge will occur from the portal instead of from the low level adit (untreated) as currently occurs. This discharge will flow to the primary mine water settling sump, which will remain as a stock water pond following mine closure. Water from the settling pond will discharge to the wetland (which will remain as a natural feature of the site), and ultimately to the Snowy River.

Approximately 800,000 m³ of waste rock will remain at the site following mine closure. The waste rock stack is to be progressively rehabilitated. Run-off and seepage is to be collected in a drain and directed to the primary storm water settlement sump, which will also remain as a stock water pond following mine closure. Water from this settling pond will discharge to the wetland (which will remain as a natural feature of the site), and ultimately to the Snowy River. Other temporary structures and buildings are to be decommissioned.

8.3.10 AMC conclusions regarding environmental factors

AMC considers that the pertinent environmental factors have been assessed to a level commensurate with, or higher than, the current status of the project. The majority of the key risks and corresponding management measures have been identified and described.

Additional work is required for the characterisation of waste rock and tailings, together with further design of the waste rock and tailings management, disposal, and closure facilities.

Preliminary tailings and waste characterisation together with preliminary designs of tailings and waste management facilities conclude that the tailings and waste rock will be manageable within the available area, and with the application of routine management designs.

8.4 Social factors

8.4.1 Key project stakeholders and stakeholder engagement

OceanaGold (AEE, 2013) reports that consultation has been undertaken outlining the changes to the project compared with the 2004 consented position.

During the research in connection with the changes and the preparation of this assessment of environmental effects, OceanaGold has consulted with a number of organisations and consent authorities. Those consulted by the applicant are:

- Department of Conservation
- Buller District Council
- West Coast Regional Council
- Friends of Waiuta Incorporated

- Ngāti Wae Wae
- Ngāi Tahu
- PF Olsen Limited, previously Timberlands West Coast
- Heritage New Zealand
- Fish & Game New Zealand (West Coast)
- Landowners on the Snowy Road
- The landowner of the Snowy River site.

AMC understands from discussions with project personnel that ongoing consultation with these stakeholders has continued since 2013, as the project is refined.

While AMC has not sighted records or meetings, grievance records, or issues identification processes, AMC considers that the views and requirements of stakeholders have been considered by direct engagement or by the public notification processes inherent in the consenting process for the project. The outcomes of some of these discussions are described in various parts of Section 8.2 and 8.3, and specifically in Section 8.2.5.

AMC understands that OceanaGold intends to continue to engage with these and other relevant stakeholders if and when the project moves towards development.

8.4.2 Local infrastructure effects

OceanaGold advises that it will avoid imposing a financial burden on local infrastructure through its own financing of infrastructure improvements, namely:

- Meeting the cost of upgrading Snowy Road and the State Highway 7–Snowy Road intersection to provide safe and adequate access and cater for the minor increased usage caused by the development.
- Reducing any adverse effects of land clearance and construction of the waste rock dump by replanting progressively during mining and upon closure.
- Meeting the cost of providing an adequate supply of potable water for human consumption to the mine site for the duration of the project.
- Meeting the cost of an appropriate sewage system for the mine site for the duration of the project.
- Controlling storm water disposal at the mine site for the duration of the project.
- Meeting the cost of providing electricity supply to the mine site for buildings intended for human occupation.
- Meeting the cost of providing telephone links to and within the mine site.

AMC notes that OceanaGold intends to construct its own infrastructure and avoid creating a financial burden to the existing community.

8.4.3 Community sponsorship programmes

In recognition of the important role the community plays in helping the Globe Progress mine realise its potential, OceanaGold provides financial support to a number of initiatives at the community level. During 2012 and 2013, grants were distributed to community groups including the Westland Mountain Bike Club, Friends of Waiuta, Reefton Rodeo Club, Inangahua A&P Show, West Coast Leading Light Business Excellence Awards, Soroptinist International Westland, Reefton Workingmens' Club, Reefton Golf Club, Reefton Rugby Club, Life Education Trust West Coast, the Kahuna Board Rider Club, Reefton Historic Trust Board, Inangahua Tourism Promotions, Greymouth SPCA, West Coast Pool Association, Reefton Netball Club, West Coast U13 Representative Hockey, Parfitt Kids, Inangahua Band Hall, Greymouth District Pony Club, Greymouth Heritage Trust, Reefton Sacred Heart School Parent Teacher Association (PTA), Reefton Trotting Club, and Reefton Volunteer Fire Brigade.

OceanaGold proposes that the Blackwater project will help to underpin the continuation of OceanaGold's community grants programme.

8.4.4 Other socioeconomic benefits

OceanaGold proposes that the Blackwater project will contribute to the "social fabric" of the Reefton, Buller, and West Coast communities by staff, contractors, and their families belonging to service clubs, sports clubs, and other voluntary organisations. As well as fulfilling leadership roles and making other contributions within the community, the project staff, contractors, and their families will help to provide the critical mass to underpin the ongoing sustainability of the area.

AMC agrees that the development of new mining projects in an existing mining area is likely to contribute to the social and economic maintenance and development of the Reefton and wider communities.

8.4.5 Other national economic benefits

OceanaGold notes that the Blackwater tenement is subject to the 1996 Minerals Programme and, as such, will generate a Crown royalty of 1% (ad valorem) or 5% (accounting profits), whichever is greatest. When profitable, the project will also generate additional corporate income tax payments. To the extent that the project leads to an overall increase in national employment, the government will receive additional income tax payments.

AMC agrees the contribution to the government revenue through royalties and taxes will provide a national economic benefit.

8.4.6 Corporate social responsibility initiatives

OceanaGold proposes to implement the following additional beneficial activities as part of the project:

- Treatment of mine-affected water from historical workings prior to discharge. Currently, contaminated water from an adit linked to the historical underground workings runs across Department of Conservation land to the Snowy River. The historical workings will be drained as part of the project, and drainage will be treated prior to discharge, rather than direct and uncontrolled discharge to the Snowy River.
- Voluntarily working collaboratively with the Department of Conservation and other parties on the proposed rehabilitation of historical contaminated land at the Blackwater mine site. This might include burial of contaminated material in disused mining voids.
- OceanaGold will remove noxious plant species from the site and replace them with pasture and other vegetation appropriate to the location as part of the rehabilitation, revegetation, and mine closure process.

AMC considers that these initiatives are likely to help build positive relationships and generate goodwill with some of the project stakeholders.

8.4.7 Grievance mechanisms and complaints records

AMC has sighted the complaints records for the existing Reefton gold project, as evidence of an existing grievance mechanism for OceanaGold projects in the area. AMC anticipates that OceanaGold will duplicate or extend the existing systems and procedures to the project.

8.5 Workforce availability and workforce accommodation

Labour will be recruited primarily from the local region, which has a strong mining history. The proposed workforce will be accommodated in private residences in the region, predominantly in the major centres of Reefton, Greymouth, and Westport, which are all within commuting distance of the project.

8.5.1 Summary of stakeholder consultation and social license to operate

AMC considers that OceanaGold has started, and continues to actively engage with key project stakeholders, as evidenced by agreements with landowners and inputs to the consent process for the project.

AMC considers that OceanaGold has made progress in developing its social license to operate the project, in terms of actively and openly engaging with project stakeholders, developing trust and goodwill, and by planning for the continuance and expansion of a variety of community contributions as part of the project.

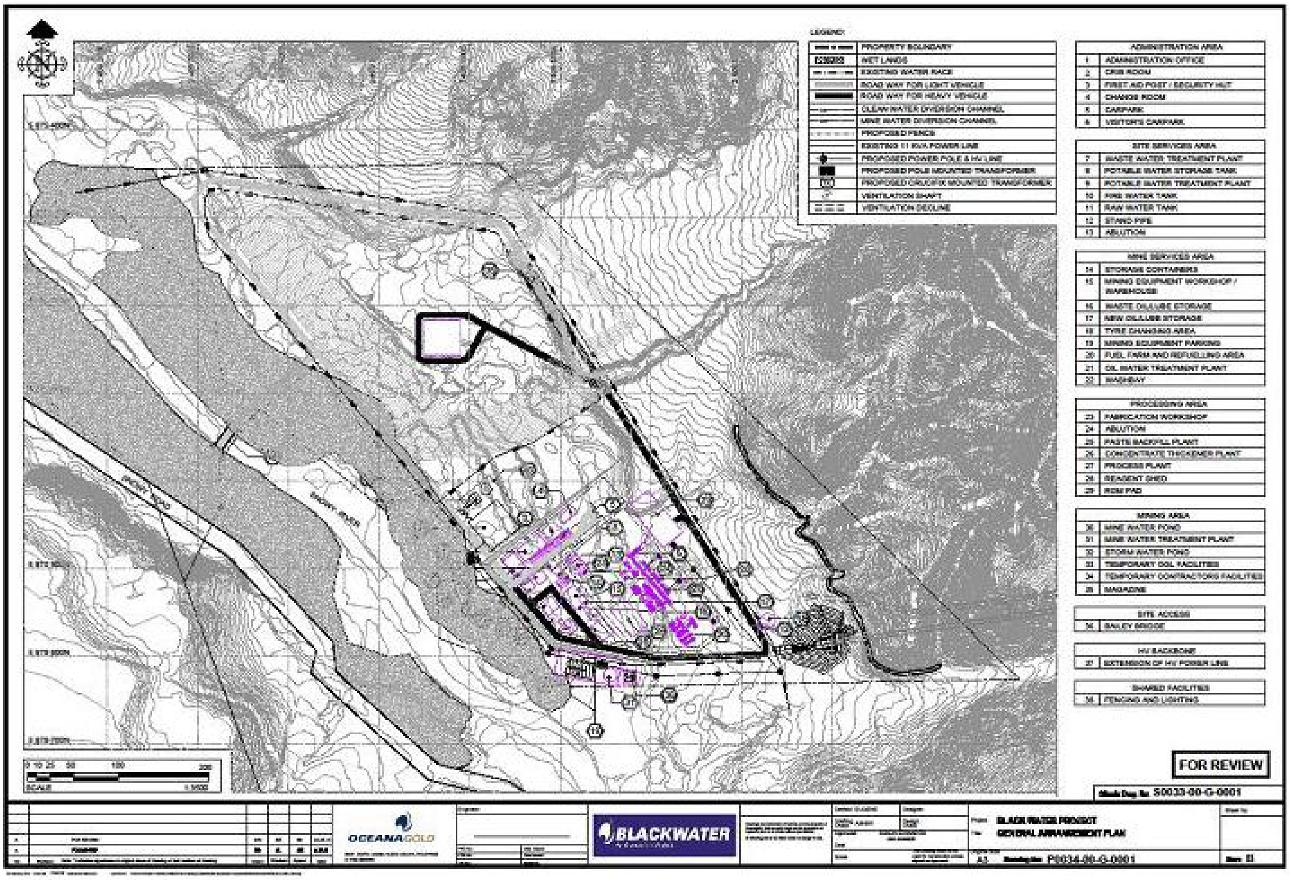
AMC concludes that OceanaGold has developed agreements and developed its social license to operate to a level that is commensurate with, or surpasses, the current stage of project planning.

9 Infrastructure

9.1 Overview of site infrastructure

The Blackwater project general arrangement designed during the project's technical study is shown in Figure 9.1. All mining infrastructure for the proposed Snowy site will be newly constructed. This will include a process plant, waste rock storage dumps, a tailings storage facility, a decline portal, administration buildings, maintenance buildings, and vehicle parking.

Figure 9.1 Blackwater project general arrangement



The main infrastructure areas will be located on a built-up rock fill pad (192 m RL) on the Snowy River alluvial plain. The alluvial plain is considered by OceanaGold to be the most suitable location for the permanent infrastructure as the area is relatively flat, lightly vegetated, and in close proximity to the underground portal location.

The pad level provides mitigation against the 1 in 1,000 year flood level, as modelled by Golder Associates (Golder Associates (NZ) Limited, 2013), and requires an average of 3.5 m of fill over the area. The pad will be constructed in stages from rock fill material sourced during decline development. Decline material will be blasted greywacke material, which is expected to provide a quality foundation for site infrastructure.

The site services area includes the raw water system, potable water system, waste water disposal system, and high voltage electrical backbone, and is located centrally to the processing area, mine services area, and administration area to minimise distribution network requirements.

The administration area includes the administration office, crib room, first aid post/security post, and change room. The administration area is located in close proximity to the processing and mine services areas such that administration facilities are easily accessible to those areas.

The mine services area includes the mining equipment workshop, mining equipment parking area, the mining equipment fuelling facility, mining equipment wash bay, tyre change facility, and oily water treatment plant. The mine services area is located to the south of the processing area (approximately 200 m from the underground portal) and has been positioned to keep mining equipment away from the processing and administration areas.

The processing area includes the process plant, CTP, reagent shed, fabrication workshop, and ablutions. The processing area will be securely fenced and lit to ensure security for the safe handling of gold ore.

The mining area includes the run-of-mine (ROM) pad, underground portal entrance, waste rock dump, site raw water tank, mine water pond, mine water treatment plant, storm water pond, and clean water diversion channels. The underground portal is located in the south-eastern area of the Blackwater project. Ore from the underground operation is delivered to the ROM pad, which is located approximately 250 m from the portal entrance to minimise haulage distance.

The ROM pad is located 8 m higher than the process plant enabling direct feeding of ore into the crushing circuit. Waste material from the underground operation is stored in the waste rock dump, which takes up most of the northern area of the site.

9.2 Availability of land for plant development

OceanaGold has entered into an option agreement to acquire land required for surface infrastructure works at the Blackwater mine site. The land comprises all or part of Section 10 Block XIV Mawheraiti Survey, District Nelson Land District, NL10A/347 (discussed in detail in Section 8.1.2).

The land parcel area is 30 hectares and is bounded by Victoria Forest Park on the north and east, Snowy River on the west, and Ngāi Tahu land on the south.

This parcel of land provides sufficient land area for the plant, waste rock dump, and other key features of the project, and retains two stands of silviculture between the site and the Snowy Road. Spare land is available within the existing land parcel, to the east of the planned site boundary.

AMC concludes that there is sufficient land for the development of the plant in a location convenient to the mine portal and power and transport infrastructure.

9.3 Power supply and telecommunications

An 11 kV transmission line has been constructed alongside the Snowy River Road, and is located outside the entry to the site. The transmission line was constructed for a previous iteration of the Blackwater project. OceanaGold advises that there is sufficient capacity in the transmission line to supply the Blackwater project.

OceanaGold advises that the electrical grid supplying the Blackwater project is considered reliable, and frequent or extended power outages are not expected. Consequently, back-up power will only be provided for critical underground operation systems (i.e. pumping and life support), which have an estimated power requirement of less than 500 kW. A 500 kVa back-up generator set (genset) will be provided at the portal.

There is currently no landline, fibre, or mobile signal communication available at the site. The nearest connection point to New Zealand telecom infrastructure is located approximately 9 km from the site. A 24-core, single-mode, all-dielectric self-supporting (ADSS) optical fibre line is proposed to be extended 9 km to site along the existing overhead 11 kV transmission line.

9.4 Water supply

There is currently no bulk water supply infrastructure in proximity to the site.

Process water is the main water demand for the project. A closed cycle will be used, with process water extracted from tailings by filter press (to provide a filter cake with approximately 14% water), and returned to the process water pond.

AMC understands that mine and process water supply is anticipated to be obtained from a number of sources:

- Rainfall draining from the site into sediment dams on-site, for storage and reuse.
- Mine water draining from historical workings of the Blackwater mine.

- Mine water draining from the new workings of the Blackwater mine.
- Groundwater from abstraction bore(s) to be installed on the Snowy River project site.
- Abstraction of water from the Snowy River.

Potable water will be sourced from a groundwater bore and treated in a potable water treatment plant to be installed on the site.

9.5 Site access and transportation

Access to the project for construction and operational traffic will be via State Highway 7 (SH7) and Snowy Road.

Access to the Snowy River site is via SH7 and Snowy Road. A Bailey bridge and private access road will connect the site to Snowy Road. It is expected that imported goods from outside the south island of New Zealand will be shipped into Lyttelton Port, Christchurch, from where the goods will be trucked to site. Gold doré will be transported off-site by road.

A traffic assessment of the primary access route to the Blackwater project was undertaken in February 2013 by Beca Infrastructure Ltd (Beca). The primary aim of the assessment was to estimate the expected increase in vehicle movements during the construction and operational stage of the project and to make recommendation of any modifications/upgrades required to the existing road network to safely manage the increased vehicle movements. The major recommendations were:

- Upgrade of the SH7 and Snowy Road intersection.
- Addition of passing bays on Snowy Road.

On New Zealand Transport Agency (NZTA) advice, the SH7/Snowy Road intersection will be upgraded to ensure that the road can safely manage the increased number of trucks travelling the route, as follows:

- Undertake local road widening at the Snowy Road/SH7 intersection in accordance with Diagram E in NZTA's Planning Policy Manual, to allow slow-moving trucks to accelerate and decelerate clear of through traffic.
- Widen (to the north) the Snowy Road approach to the intersection with SH7 to enable trucks turning left into Snowy Road to negotiate the turn at a reasonable speed and without crossing the Snowy Road approach centre-line; a flush or painted throat island might also be required.
- Install truck-crossing signage on both the southern and northern approaches to the Snowy Road/SH7 intersection.

Snowy Road has two lanes with a centre line for the first 2 km from the intersection with SH7, with an approximate width of 5.5 m. Thereafter it becomes a one-lane road, of approximately 4 m width. Approximately 1.5 km from the intersection with SH7, there is a one-lane bridge. Snowy Road is sealed for the first 8 km and then becomes gravel. The proposed haul road access to Blackwater mine is located 8.5 km from the SH7 intersection.

The following improvements have been recommended by Beca and will be implemented by OceanaGold as part of the infrastructure development:

- Erect a truck crossing sign at the mine access point on Snowy Road.
- Install gravel lay-bys on Snowy Road to create additional room for vehicles to pass, particularly when large equipment is being transported during construction of the mine. Gravel shoulder widening might be constructed on Snowy Road at other locations to provide passing bays, as required.
- Undertake localised curve widening on Snowy Road to avoid large vehicles tracking onto the road berms.

Access from Snowy Road into the site requires the crossing of the Snowy River. A Bailey bridge is to be constructed as part of initial infrastructure development to enable vehicles to pass into the site without the need to disturb the river.

A Bailey bridge is preferred to a floodway crossing due to lower-maintenance requirement. A 30 m span, single-lane Bailey bridge with 4 m deck width will be installed at the site. The Bailey bridge will have an operational load capacity of 50 t. Heavy vehicles and oversized loads that cannot pass over the Bailey bridge will tram across the river.

9.6 AMC conclusions regarding infrastructure

AMC concludes that the project can be connected to existing transport, power, and telecommunications networks by small-scale and easily constructed connections. Specifically:

- A short access track (less than 500 m) from the Snowy Road to the site and a Bailey bridge over the Snowy River.
- A short transmission line (less than 500 m) and connection to the existing transmission line running alongside Snowy Road.
- A 9 km extension of telecommunications lines from the nearest access point.

AMC concludes that there is a wide variety of mine water supply sources available to meet the moderate mining and processing water demands. Additionally, while groundwater investigations are yet to be completed for raw water supply bores, AMC considers that it is reasonable to expect that a substantial alluvial aquifer would exist in the vicinity of the Snowy River, and that groundwater supplies are likely to be available.

10 Costs and economic analysis

10.1 Capital costs

The base-case scenario identifies that two and a half years of pre-production activity is required to establish the access decline and initial underground exploration drilling platform. Capital expenditure in the first two years (the only years in which there is no mining of ore material) is estimated to be US$76 million, and sustaining life-of-mine capital is estimated to be US$78 million, including 15% contingency (Table 10.1). Capital cost estimates are ±25% and assume that the mobile mining fleet is purchased rather than leased. The confidence in the cost estimates relating to the surface infrastructure and processing plant is higher than that of mining, due to the level of detail that was required for the consenting process.

The US$76 million cost estimate includes the cost estimate for the pre-production mining, surface infrastructure, process plant, and contingency. Expenditure on resource definition diamond drilling is incurred from the third year onwards, once underground drilling platforms have been established. Total life-of-project resource definition capital expenditure has been estimated to be US$9 million (plus contingency), and is included in the life-of-mine sustaining capital total of US$78.1 million shown in Table 10.2.

To reflect uncertainty associated with the estimation of capital, a range of likely pre-production capital costs has been assessed. The low and high cases reported in Table 10.1 are based on +/–30% range limits.

Table 10.1 Summary pre-production capital expenditure

Item	US$ millions		
	Base	+30%	–30%
Infrastructure	8	10	6
Processing	21	27	16
Mining	30	39	23
Management and indirects	4	5	3
Operational readiness	3	4	3
Contingency (15%)	10	13	8
Totals	**76**	**98**	**58**

Table 10.2 Base case pre-production capital cost summary

Description	Year 1 (US$ millions)	Year 2 (US$ millions)
Processing capital expenditure		
Infrastructure and power	4.1	2.1
Ore processing	–	20.5
Total	**4.1**	**22.6**
Indirect		
Engineering and design	1.8	
Operational readiness	–	3.1
Commissioning	–	0.3
Management and indirects	–	3.7
Total	**1.8**	**7.1**
Underground mining capital expenditure		
Development	10.1	10.4
Mobile equipment	6.2	
Electrical equipment	1.4	0.4
Infrastructure	–	0.5
Other	0.2	1.1
Total	**17.9**	**12.3**
Pre-production capital total	**23.8**	**42.0**
Contingency factor at 15%	3.6	6.3
Subtotal per annum	27.4	48.3
Total pre-production capital	**–**	**75.7**
LOM sustaining capital total	**–**	**78.1**
Total project capital	**–**	**153.7**

10.1.1 Surface infrastructure and power costs

Blackwater initial direct civil infrastructure capital costs amount to US$6.2 million. This covers the infrastructure and facilities required to support the mining and processing operations including site preparation, civil work, services, roads, explosive facilities, and electrical substation.

Power supplies in the region are sufficient for project requirements, and no provision for additional power line construction has been included, other than the power line from the Snowy River Road into the site and for distribution throughout the site.

AMC considers that this allocation is appropriate for the project, which is close to good, existing infrastructure and services.

10.1.2 Mining capital cost

Blackwater pre-production mining capital amounts to US$30 million and capital costs for the life of the mine amount to US$98 million. Mining direct capital costs include pre-production mining, capital development costs, mine mobile equipment, and mine infrastructure. Development to be completed prior to the commencement of production was classified as pre-production (capitalised) development. It is expected that OceanaGold will deploy skilled labour from its existing underground operations in New Zealand.

Capital requirements were allocated to both pre-production years, but during more-detailed studies, consideration should be given to allocating the capital requirements over additional years, as it is likely that payments for equipment will be required prior to the equipment being delivered to site unless a fleet lease option is adopted.

AMC considers that the approach to estimating mining capital costs is appropriate and the estimate is reasonable.

10.1.3 Processing capital cost

The total capital cost of the processing facility is estimated to be USD20.5 million, as detailed in Table 10.3. An allowance of USD8.9 million across the project has been made for engineering, project management, and commissioning (Table 10.2).

The total cost is generally in line with typical industry costs for plants similar in nature and scale located in this region.

Table 10.3 Processing plant capital cost summary

Blackwater processing plant: capital expenditure	US$ millions
Surface Python	
Crushing and gravity	$4.1
Grinding	$1.7
Flotation	$1.3
Filtration	$1.7
Utilities/general	$0.5
Reagents	$0.7
Motor control centres, control room, site laboratory	$2.3
Total direct capital cost Surface Python plant	**$12.2**
CTP	
Regrind	$0.7
Leaching	$0.6
Resin filtration	$1.1
GRex resin	$1.0
Electrowinning	$0.8
Detoxification	$0.5
Motor control centres	$1.8
Total direct capital cost CTP	**$6.6**
Total direct capital cost	**$18.8**
Engineering. Construction/project management	–
Installation	$0.8
Concrete/civils	$0.9
Commissioning	–
Spares	–
Other	$0.03
Total indirect capital cost	**$1.7**
Total capital cost	**$20.5**

10.1.4 Other capital costs

Blackwater contingency costs amount to US$20 million and account for unforeseen costs within the project scope. Contingency costs were calculated using a factor of 15% of civil infrastructure, process plant, and mine infrastructure direct capital costs.

Indirect costs of US$9 million have been allowed to cover pre-production-related expenditure:

- Engineering and design
- Operational readiness
- Commissioning
- Management and indirects

AMC considers that the contingency and indirect costs are appropriate for the level of engineering work performed in the preparation of the PEA.

10.2 Operating costs

Operating costs have been estimated using first principles derived from supplier quotations and/or benchmark data from OceanaGold and other similar operations. The low and high cases in Table 10.4 are based on +/–30% range, to reflect the early stage of the project. The mining costs quoted for the base-case scenario do not include capital costs.

Table 10.4 Operating cost inputs (US$/t ore)

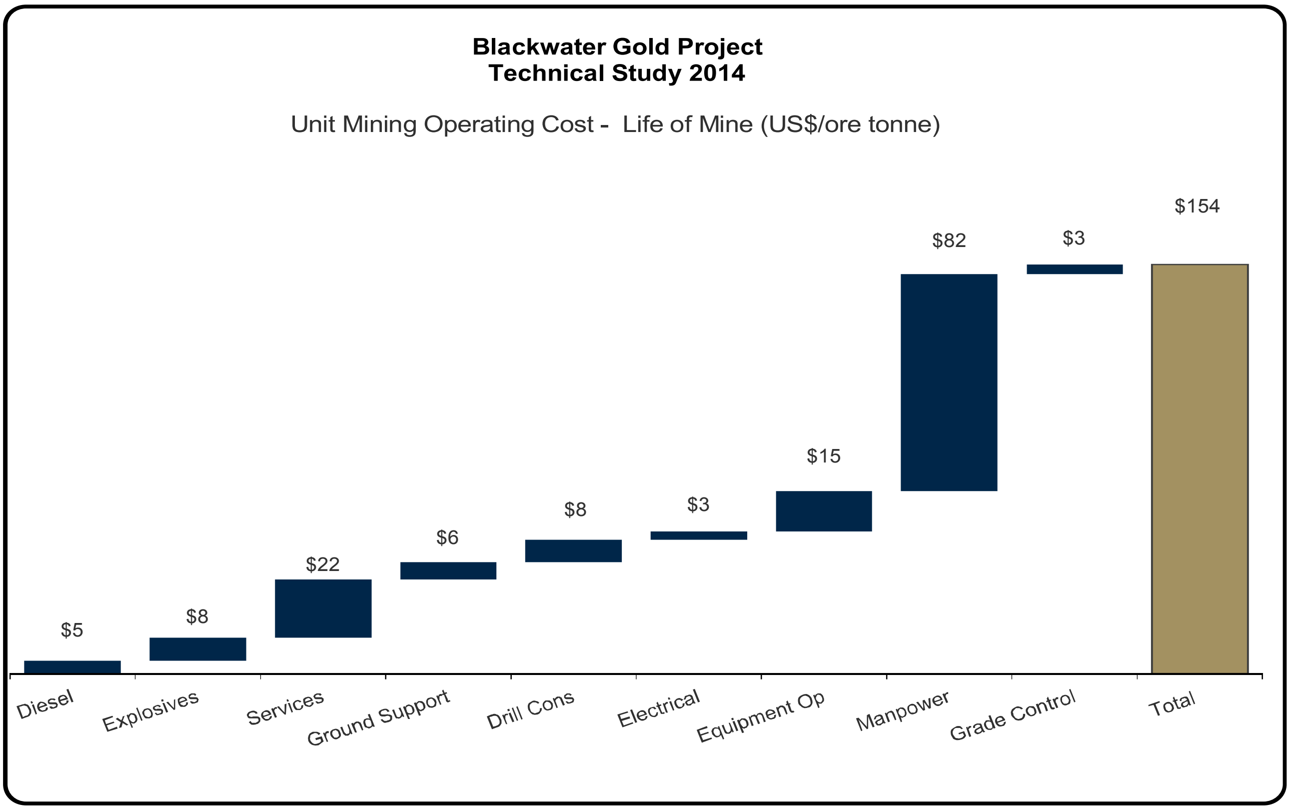
Item	US $/t ore		
	Base	+30%	–30%
Mining	154	200	118
Processing	42	55	32
Site general and administration	19	24	14
Selling costs	2	3	2
Totals	**217**	**282**	**167**

An allowance of 2,500 m of grade control drilling for each panel within the mine has been costed, which at a cost of US$225/m for drilling and assaying totals US$4 million for the life-of-mine. This is included in the mining operating cost in Table 10.4.

10.2.1 Mining operating cost

The unit operating cost for mining at Blackwater is expected to be approximately US$154/t ore. The activity-based analysis is reported from life-of-mine averages and summarised in Figure 10.1. AMC has compared these costs with its own databases and other similar mines with which it is familiar, and considers that the estimated cost is reasonable. These costs will need to be re-estimated when the likely ground conditions and reef structure are better-understood.

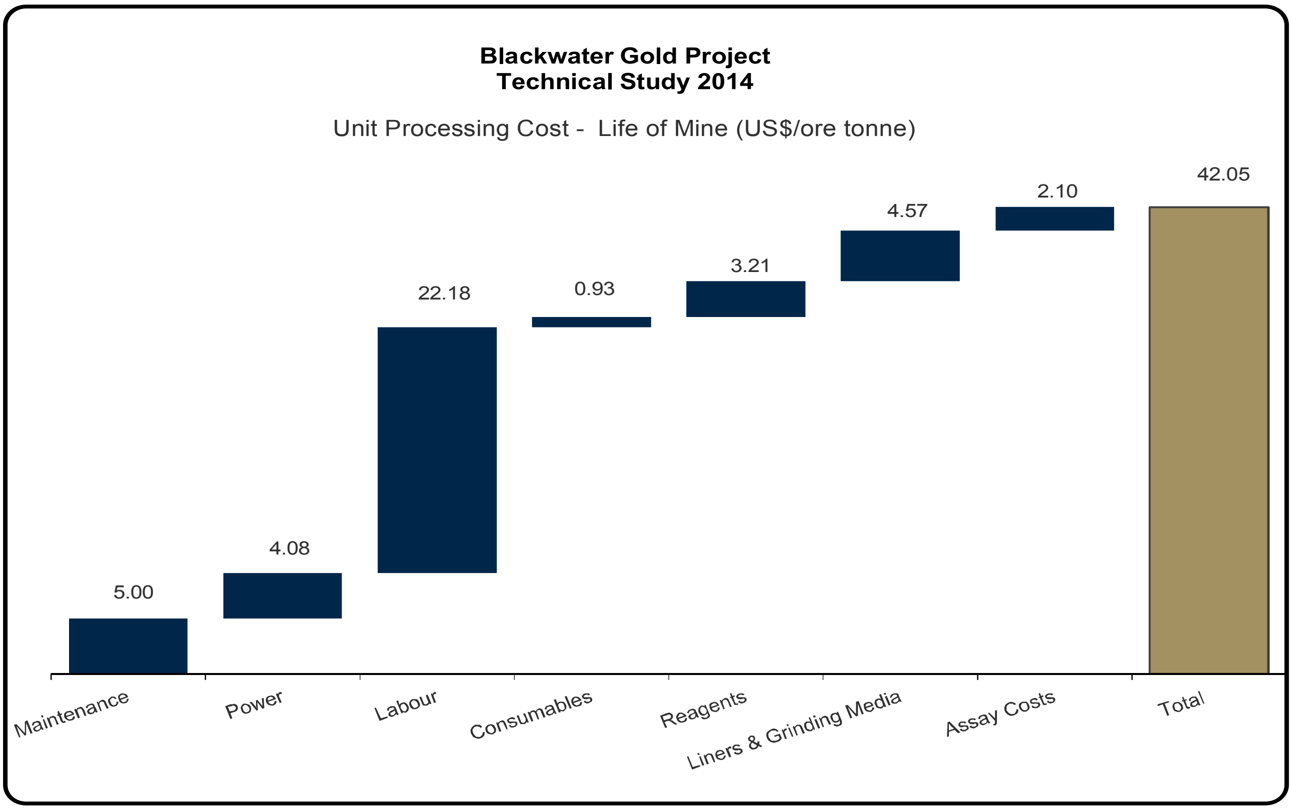
Figure 10.1 Operating cost: mining



10.2.2 Ore processing operating cost

The operating costs for ore processing are shown in Figure 10.2. Overall cost is estimated to be USD42.05/t. As noted in the PEA, the largest cost contributor is labour at USD22.18/t. This value is consistent with typical industry costs in Australia/New Zealand for a plant of this size. Likewise, power (at unit cost of USD0.08/kWh), maintenance, and wear components (crushing liners and mill liners) are in line with similar plants. However, the other costs associated with the operation of the plant are significantly lower than some other plants that use carbon-in-leach (CIL) technology for the recovery of gold from solutions rather than resin. AMC recommends verification of these costs with the supplier of the plant (Gekko).

Figure 10.2 Operating cost: ore processing



10.2.3 Other operating costs

This includes a site-wide general and administration cost, which captures all costs not included in the mining and ore processing costs including health and safety, security, and environmental costs. This is estimated at US$19/t of ore processed, which AMC considers reasonable for a mine of this size in this location.

Selling costs of US$2/t of ore processed are also allowed, which AMC considers reasonable.

10.2.4 Royalties

The *Crown Minerals Act 1991* provides for the payment of royalties to the Crown in respect of all gold and silver mined. Under Schedule 1 to the Act, those royalties are calculated in accordance with the 1996 Minerals Programme that applied when the initial prospecting permit (PP) or EP, giving rise to the subsequent MP, was granted.

In the case of the Blackwater EP, the initial permit was granted in November 2002, and came under the first minerals programme to be promulgated under the Act, being the 1996 Minerals Programme. Accordingly, the Crown royalties' payable under a subsequent MP will be calculated at the rate of 1% ad valorem or 5% of accounting profits, whichever is the greater within any given calendar year. Accounting losses can be carried forward, and at the ultimate expiry or surrender of the permit, there is reconciliation with provision for any overpayment of royalties to be refunded by the Crown. It is also permissible to amortise future expected rehabilitation costs over the expected life of the permitted operations, meaning the risk of a large correction at the end of the permit's life is reduced. Crown royalties are calculated on spot price for gold and silver, and take no account of the losses or gains associated with hedging.

In addition to Crown royalties, the Blackwater EP is also subject to an agreement contained in a Deed of Novation between Royalco and OceanaGold, under which an annual royalty of between 1% and 3% of gold produced is payable, according to the gold price at the time the royalty is due. Where the spot gold price is NZ$900 and above, the royalty is fixed at 3%.

The royalty reverts to 1.5% of annual gold production once an aggregate of 1,000,000 ounces of gold is produced from all of the Reefton tenements (including from the current Reefton mine).

The Royalco agreement grants OceanaGold an option to buy back the royalty over Blackwater EP for the sum of A$5,000,000, CPI adjusted from 14 May 1991. The option to buy back the Royalco royalty over Blackwater EP may be exercised at any time until OceanaGold makes a decision to mine EP 40 542 and applies to all gold produced from that decision (i.e. from EP 40 542).

10.3 Revenue factors

Project revenue for the base-case scenario is based on an assumed gold price of US$1,300/ounce. Although this is slightly higher than the current price, AMC considers it to be reasonable for the forecast period of production.

10.4 Market assessment

Markets for doré are readily available and the doré bars produced from the Blackwater project could be sold on the spot market. Gold markets are considered mature, despite a current gold price that is lower than the three-year trailing average. Gold hedging is an option.

10.5 Economic analysis

The economic analysis in the PEA is based on the production target that is derived from an Inferred Mineral Resource that is considered too speculative geologically to have the economic considerations applied to it that would enable an estimate of ore reserves.

The reef width and grade is expected to vary during mining, and a single resource shape has been generated to represent the Inferred Mineral Resource. A block model has not been generated for the production target. The mining study has, therefore, applied a constant thickness and grade as a base case and has assessed a range of possible outcomes.

Given that the production target is based on an Inferred Resource, a NPV range analysis has been completed to estimate the expected outcome for a range of reef thickness and reef gold grades. The selected range limits were based on ±30%, which is considered an indicative range in confidence for an Inferred Resource. Having due regard to these limitations, the outcome of the economic evaluation suggests robust economics for the base-case scenario reported. AMC understands that the entire project is intended to be internally funded by OceanaGold cashflow, and 5% is the OceanaGold internal hurdle rate of return when assessing investment opportunities. This is the discount rate used for the economic evaluation.

The maximum impact on contained metal modelled is approximately ±30%, whether by flexing only width, only grade, or a combination of both. This range is considered to be a reasonable maximum variability that could be encountered when undertaking extraction of the Birthday Reef. The results are presented as an array in Table 10.5, ranging from a lower post-tax NPV of US$21 million to an upper post-tax NPV of US$243 million. The base-case scenario suggests a post-tax NPV of US$132 million.

The NPV remains positive over the full range of reef width and grade scenarios modelled, and the project financials seem robust. Both width and grade are expected to vary during the course of mining, but the current level of information makes it impossible to model such scenarios using geological modelling methods. AMC considers that the inputs to the economic analysis are reasonable, and that the presentation of the resulting NPVs as an array of possibilities is appropriate. The wide range of possible outcomes reflects the level of uncertainty at this stage of the project.

Table 10.5 NPV array (US$ million) by flexing reef width and grade

NPV array			Reef thickness {diluted thickness} (m)				
			–30%	–15%	Base	15%	30%
			0.48 {0.75}	0.58 {0.87}	0.68 {1.00}	0.78 {1.17}	0.88 {1.30}
Reef grade (g/t)	–30%	16			21		
	–15%	20		46	83	105	
	Base	23	42	90	132	159	199
	15%	26		133	180	212	
	30%	30			243		

10.6 Sensitivity analysis

In the PEA, deterministic sensitivity analysis has been assessed for the base case. Revenue, total capital costs, and total operating costs were flexed separately while holding all other factors constant. The results of the sensitivities for NPV and internal rate of return (IRR) are presented in Table 10.6, and Figures 10.3 and 10.4. The evaluation suggests that the project could sustain significant increases in both capital and operating expenditure. The data reported in the sensitivity analysis is post-tax.

Table 10.6 Deterministic sensitivity data for NPV and IRR (post-tax)

NPV (US$ million) at 5%											
	–25%	–20%	–15%	–10%	–5%	0%	5%	10%	15%	20%	25%
Revenues	40	59	77	95	114	132	150	169	187	205	223
Capital costs	157	152	147	142	137	132	127	122	117	112	107
Operating costs	164	158	151	145	138	132	126	119	113	106	100
IRR											
	–25%	–20%	–15%	–10%	–5%	0%	5%	10%	15%	20%	25%
Revenues	11%	14%	16%	19%	21%	23%	25%	27%	29%	31%	32%
Capital costs	30%	28%	27%	25%	24%	23%	22%	21%	20%	19%	18%
Operating costs	27%	26%	25%	24%	24%	23%	22%	21%	21%	20%	19%

Figure 10.3 Deterministic sensitivity graph: NPV at 5%

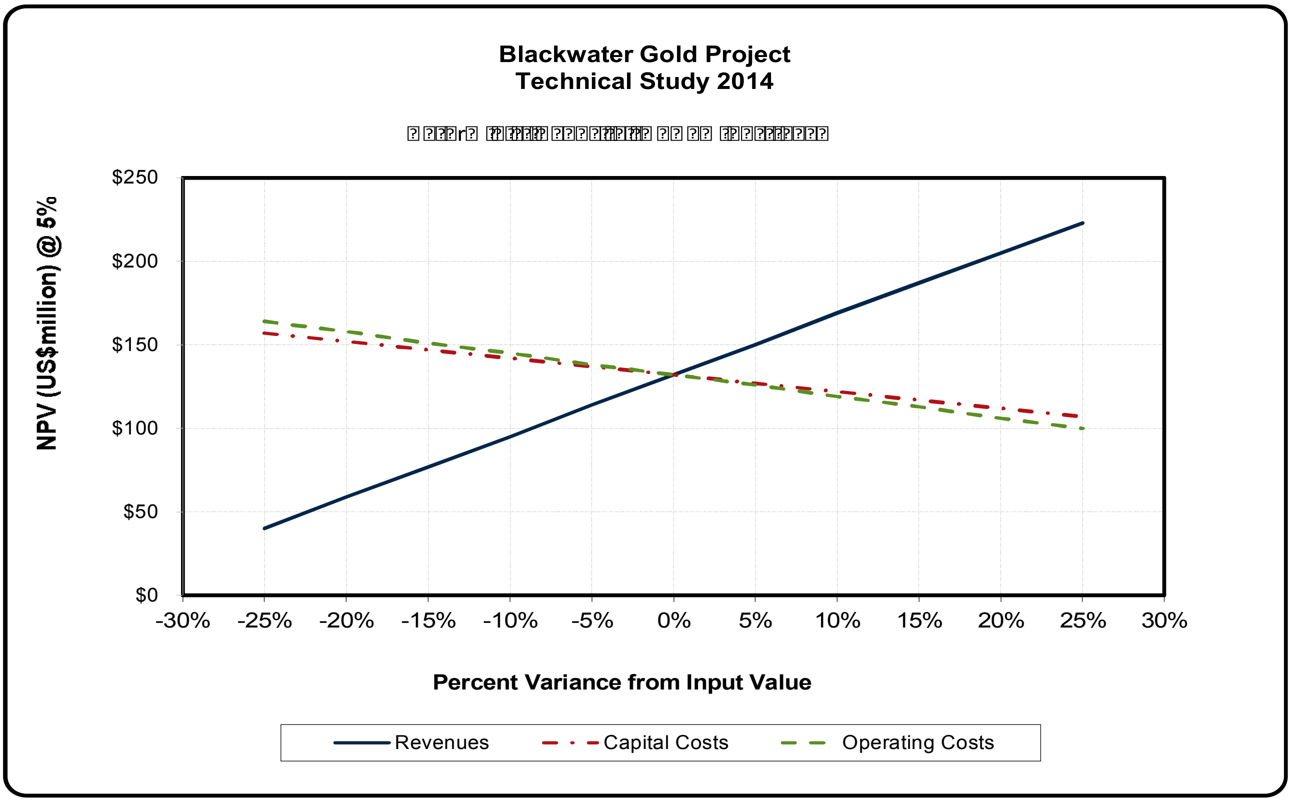
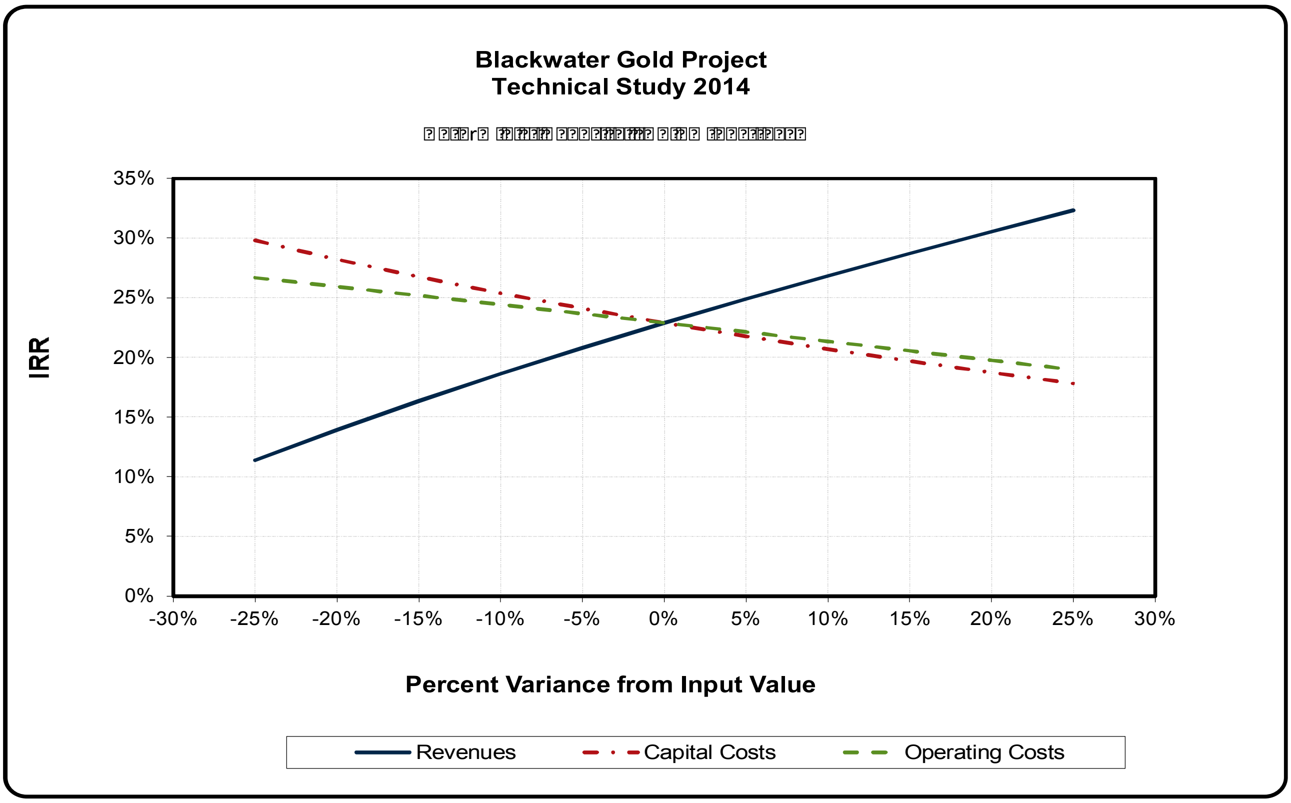
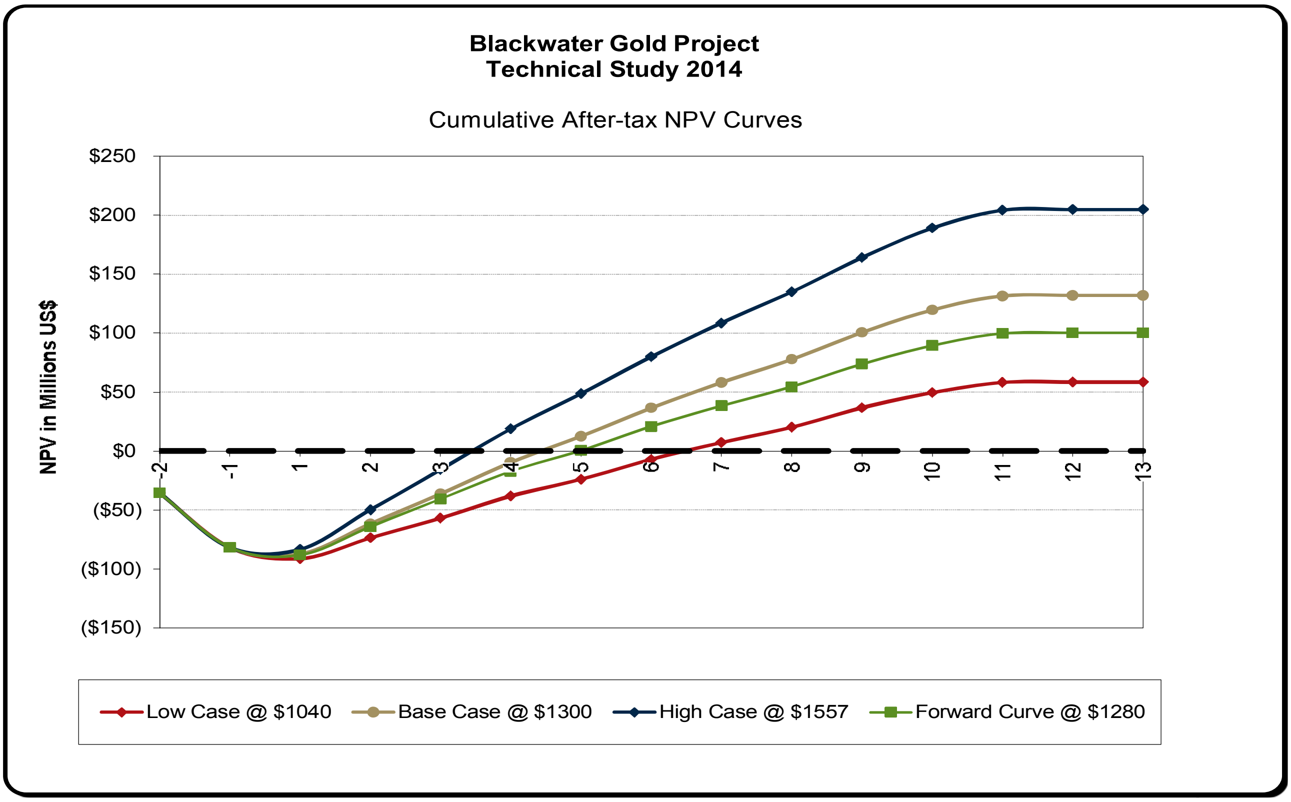


Figure 10.4 Deterministic sensitivity graph: IRR



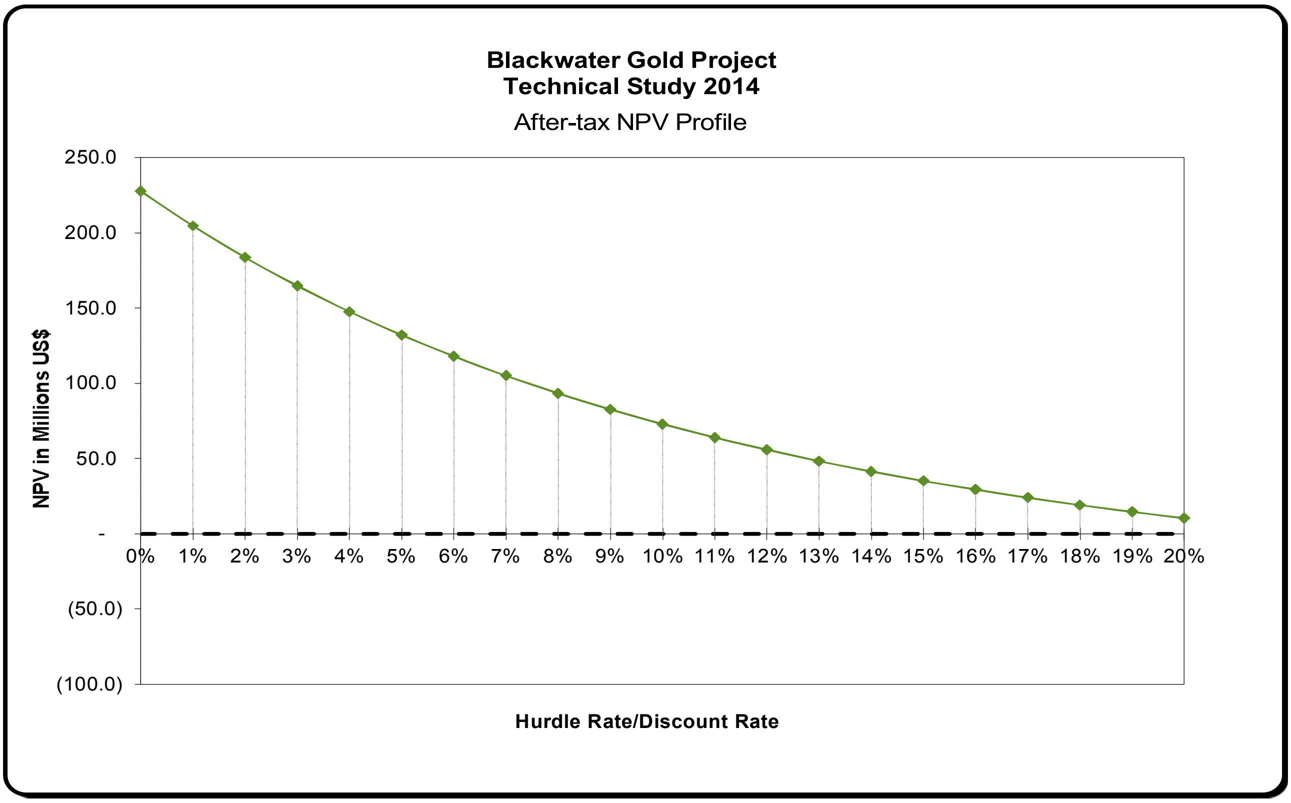
Gold price scenarios for cumulative NPV outcomes from the PEA are reported in Figure 10.5.

Figure 10.5 Cumulative post-tax NPV at 5% discount rate for different gold price scenarios



In Figure 10.6, the base-case NPV is reported for a range of discount rates.

Figure 10.6 Discount rate sensitivity



AMC considers that the sensitivity analysis demonstrates the effect of the key parameters likely to impact on project economics.

11 Other

11.1 Production target estimate risk from naturally occurring issues

Blackwater is situated very close to the Alpine Fault, which is the tectonic boundary between the Australian and Pacific Plates. Large historical displacements on this structure are well-known and the fault is active. There is a risk to the project from seismic activity, but given the scale of the proposed workings, it is unlikely to disrupt production for a sustained period. Damage to surface infrastructure is possible if a shallow seismic event occurs in close proximity to the project.

11.2 Classification

The classification of the Birthday Reef Mineral Resource as an Inferred Mineral Resource is appropriate, given the limited drilling and the use of the historical production and mining records to estimate the resource. AMC agrees with OceanaGold that the resource is a reasonable global estimate of the depth extension of the Birthday Reef below 16 level. OceanaGold reports that approximately 15% of the resource is extrapolated beyond actual sample locations. In the circumstances, AMC opinion is that this level of extrapolation is conservative.

11.3 Audits or reviews

This independent technical report is the result of AMC's independent review of the PEA and OceanaGold's internal technical report on the Blackwater project.

11.4 Discussion of relative accuracy/confidence

The Birthday Reef Mineral Resource is a global estimate of the mineralisation below 16 level constrained by four deep surface holes and daughter holes from those holes (Figure 2.1). The estimation of both tonnage and grade utilises historical face sampling to produce averages of thickness and payability. Grade is back calculated from historical production records using the historical mined shape, milling recoveries, and estimated dilution. Clearly, this is inherently dependent on the assumption that the Inferred Resource is similar to the historically mined area. This assumption is supported by the drilling intersections (Table 2.2).

In the development of the production target, OceanaGold has adopted a wide range of sensitivities to ensure the inherent inaccuracy of the Mineral Resource estimate is not overlooked in developing the production target.

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APPENDIX 3 - CAUTIONARY STATEMENTS AND TECHNICAL DISCLOSURES FOR PUBLIC RELEASE

General Cautionary Statement for Public Release

Certain information contained in this public release, including any information relating to the Company's future financial or operating performance and development and production timelines, production and cost estimates may be deemed "forward-looking" within the meaning of applicable securities laws. Forward-looking statements and information relate to future performance and reflect the Company's expectations regarding the future growth, results of operations, business prospects and opportunities of OceanaGold Corporation and its related subsidiaries. Any statements that express or involve discussions with respect to predictions, expectations, beliefs, plans, projections, objectives, assumptions or future events or performance (often, but not always, using words or phrases such as "expects" or "does not expect", "is expected", "anticipates" or "does not anticipate", "plans", "estimates" or "intends", or stating that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved) are not statements of historical fact and may be forward-looking statements. Forward-looking statements or information such as development and production timelines, costs estimates and production forecasts are subject to a variety of risks and uncertainties which could cause actual events, performance, achievements or results to differ materially from those expressed in the forward-looking statements and information. They include, among others, the accuracy of mineral reserve and resource estimates and related assumptions, inherent operating risks and those risk factors identified in the Company's most recent Annual Information Form prepared and filed with securities regulators which is available on SEDAR at www.sedar.com under the Company's name. There are no assurances the Company can fulfil forward-looking statements and information. Such forward-looking statements and information are only predictions based on current information available to management as of the date that such predictions are made; actual events or results may differ materially as a result of risks facing the Company, some of which are beyond the Company's control. Some of these risks and uncertainties include: general economic and market factors (including changes in global, national or regional financial credit, currency or securities markets); fluctuations in the price of gold; inability to obtain required consents, permits or approvals; changes or developments in global, national or regional political conditions (including any act of terrorism or war); changes in laws (including tax laws) and changes in GAAP or regulatory accounting requirements; and other risk factors as outlines in the Company's annual and interim filings. Readers are cautioned that the foregoing list of factors is not exhaustive. Although the Company believes that any forward-looking statements and information contained in this press release is based on reasonable assumptions, readers cannot be assured that actual outcomes or results will be consistent with such statements. Accordingly, readers should not place undue reliance on forward-looking statements and information. The Company expressly disclaims any intention or obligation to update or revise any forward-looking statements and information, whether as a result of new information, events or otherwise, except as required by applicable securities laws. All forward looking statements and information contained in this public release is qualified by this Cautionary Statement. The information contained in this release is not investment or financial product advice.

Cautionary Note Regarding Production Target

There is a low level of geological confidence associated with inferred mineral resources and there is no certainty that further exploration work will result in the determination of indicated mineral resources or that the production target itself will be realised. The stated production target is based on the company's current expectations of future results of events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met.

Cautionary Note Regarding Mineral Resources and Mineral Reserves

The Company's disclosure of Mineral Reserve and Mineral Resource information is governed by NI 43-101 under the guidelines set out in the Canadian Institute of Mining, Metallurgy and Petroleum (the "CIM") Standards on Mineral Resources and Mineral Reserves, adopted by the CIM Council, as may be amended from time to time by the CIM ("CIM Standards"). The disclosure of Mineral Reserve and Mineral Resource information for properties held by the Company is based on the reporting requirements of the 2012 JORC Code. CIM definitions of the terms "Mineral Reserve", "Proven Mineral Reserve", "Probable Mineral Reserve", "Mineral Resource", "Measured Mineral Resource", "Indicated Mineral Resource" and "Inferred Mineral Resource", are substantially similar to the JORC Code corresponding definitions of the terms "Ore Reserve", "Proved Ore Reserve", "Probable Ore Reserve", "Mineral Resource", "Measured Mineral Resource", "Indicated Mineral Resource" and "Inferred Mineral Resource", respectively. Estimates of Mineral Resources and Mineral Reserves prepared in accordance with the 2012 JORC Code would not be materially different if prepared in accordance with the CIM definitions applicable under NI 43-101.

There can be no assurance that the Inferred Mineral Resources will be converted into a higher category of Mineral Resources or that Mineral Resources that are not Mineral Reserves will ultimately be converted into Mineral Reserves. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Consequently Mineral Resources are of a higher risk than Mineral Reserves.

The PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Technical Disclosure

The estimates of Mineral Resources and the application of the Modifying Factors (mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental social and government) in the preparation of the PEA and the production target were prepared in accordance with the standards set out in the Australasian Code for the Reporting of Mineral Resources and Ore Reserves of December 2012 (the "JORC Code") and in accordance with National Instrument 43-101 of the Canadian Securities Administrators ("NI 43-101"). The JORC Code is the accepted reporting standard for the Australian Securities Exchange Limited ("ASX") and the New Zealand Stock Exchange Limited ("NZX").

Each Qualified Person and Competent Person referred to in this report has reviewed the information and data contained in this report and has approved it for dissemination.

The PEA and appended ITR have been filed with the relevant stock exchanges and are available for review at www.sedar.com under the Company's profile.