Internal Technical Report

for the

Snowy River Gold Project

Reefton

South Island

New Zealand

Date: September 2022

INTRODUCTION

This Internal Technical Report ("ITR") has been compiled to provide a summary of the Blackwater (now Snowy River) Mine and the work done to date on the depth extension of the orebody and re-starting the mine in 2024/5. The ITR is based on the JORC/NI 43-101 Preliminary Economic Assessment ("PEA") carried out by Oceana Gold Corp. ("OGC") in 2014, the 2013 deep drilling campaign by OGC, 2018 mining study by Mining One and Federation Mining Ltd. ("FML"), various metallurgical testwork and processing design studies by OGC and its consultants, including Mintrex, Gekko Systems and Golder Associates.

FML is a private Australian company (Tasman Mining Ltd. Is the 100% owned New Zealand subsidiary of FML). FML has a binding agreement with OGC to purchase the Snowy River Project for US$30 million in cash or shares at FML's election by December 31, 2023. In addition, FML has an exploration lease immediately to the north of the Snowy River tenement.

FML raised approximately A$100 million in capital in 2020 and has completed approximately 2,450 metres of the planned 3,300 metre twin declines to the orebody. This work is on time and on budget and has provided FML technical staff with invaluable geotechnical information. It is expected to intersect the historical Birthday Reef orebody in March 2023, at which time both stope development and orebody definition drilling will be carried out as precursors to production in 2024/5. On surface, FML has constructed substantial infrastructure, including access roads and bridge, grid power supply, workshops, offices, change house, water storage and filtering facilities, waste rock stack and laid out the area in preparation for processing plant construction.

The ITR includes 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 was prepared by or under the supervision of J.G. Moore, Chief Geologist, a full-time employee of OceanaGold (New Zealand) Limited and a Member and Chartered professional with the Australasian Institute of Mining and Metallurgy at the time of writing and as such qualifies as a Competent Person as defined in the JORC Code. It should be noted that the OGC 2014 PEA and this ITR are based on an Inferred Mineral Resource. There is a lower 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.

Specialist consultants who were engaged in preparing the OGC 2014 PEA are shown in Table . These reports utilise information sourced from the Blackwater Mine historical archives and reports prepared by previous owners of the Blackwater Project.

Table A: OGC 2014 PEA Specialist Consultants

Consulting Company	Work Package
Mining Plus Pty Ltd	Mining and Geotechnical Engineering
Gekko Systems	Ore Processing
Golder Associates (NZ)	Hydrology, 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

FML has updated this report where appropriate based upon new data and studies. These are summarised in Table .

Table B: FML Specialist Consultants

Consulting Company	Work Package
Mining One	Mining Design and Geotechnical
Pattle Delamore Partners (PDP)	Waste Rock Stack Design, Water Management, Air Discharges
Mine Waste Management	Ground Disturbance and Geochemistry
Frank Boffa	Landscape and Visual Effects
Ryder Consulting	Terrestrial and Aquatic Biology
Stantec	Transport and Traffic
Tonkin and Taylor	Noise and Vibration
Lane Associates	Bond Assessment and Closure Cost
MD	Assessment of Environmental Effects
Origin Consultants	Archeology and Historic Heritage
SCT	Geotechnical

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1 SUMMARY

The Snowy River Project ("the Project"), formerly named the Blackwater Project, is owned by OceanaGold Corporation ("OGC") through its wholly owned subsidiary, OceanaGold (New Zealand) Ltd. OGC is a publicly listed gold producer, with operating gold mines and a portfolio of assets located in New Zealand, the Philippines, the USA and Australia.

Federation Mining Ltd. ("FML") has consolidated the 2014 PEA by OGC, the 2018 Study by Mining One and other relevant studies to describe the technical and economic potential of the Snowy River Project and to support investment decisions for the development of an operating underground mine and surface processing plant.

The Snowy River Project is located near Ikamatua on the West Coast of the South Island of New Zealand. The deposit is a continuation of the Birthday Reef that underlies historical mine workings of the Blackwater Mine, operated from 1908 to 1951. Two main shafts were sunk in that period, the Blackwater and Prohibition. Ore was treated at the Snowy River Stamper Battery until 1938, when a modern CIP Plant (The Prohibition Mill) was constructed near the Prohibition Shaft (the main ore haulage shaft). Men and materials were transported via the Blackwater shaft.

In 1951, with the mining industry suffering from a shortage of miners and a low gold price, there was a ground failure in Blackwater Shaft, the main service and ventilation shaft. Management decided the capital cost of shaft rehabilitation was too high to contemplate so the mining operations were abruptly shut down. Over 43 years of operation, Blackwater Mines Ltd extracted gold-bearing quartz from the Birthday Reef, producing 740,000oz from 1.6Mt of ore (14.6g/t Au recovered grade) at a process recovery of approximately 90%, most of that by gravity methods. The orebody had been developed down to 16 level (approx. 650m below surface) and it is estimated that approximately 80,000 oz Au were delineated and left unmined in the lower levels as a result of the closure.

OGC conducted two drilling campaigns in 1996 and 2010 to 2013, successfully intersected the Birthday Reef with four deep diamond holes (and their daughters) collared from surface. These holes intersected the Birthday Reef at various depths down to 1,700 metres below surface, supporting the projected extension of the Birthday Reef. 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 by OGC and its consultants on material sourced from both the 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 followed by intensive leaching of the concentrates and electro-winning. This will allow production of gold doré bars on site with an expected overall recovery in excess of 96%.

To progress the then named Blackwater Gold Project, OGC undertook a PEA in 2014 to evaluate the value of mining the Inferred Mineral Resource. This included the costing of an exploration decline at the Snowy River site and drilling horizon that would dictate a 'go or no go' development decision. Given increased confidence in the resource, air-leg resue mining was the proposed method of mineral extraction. This was the preferred method based on the narrow geometry of the reef and limited stope span stabilities based on geotechnical analysis.

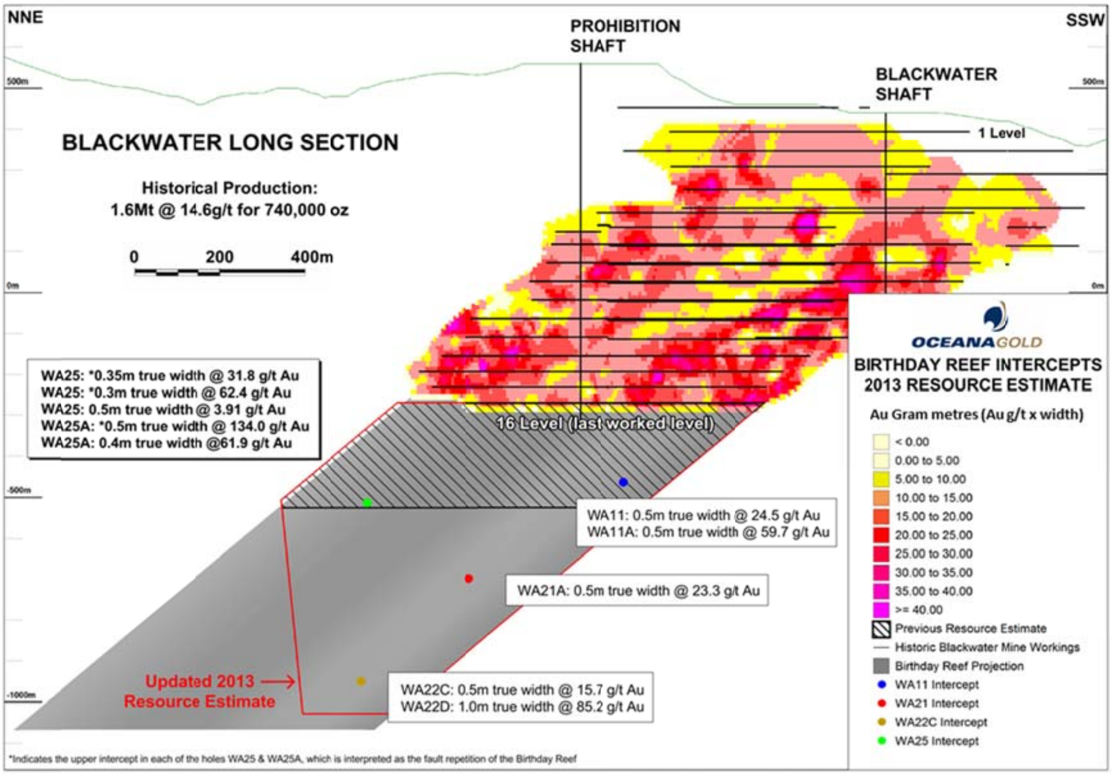
The Mineral Resource estimate is the basis for the PEA, the Mining One study and this ITR, is an 'Inferred Mineral Resource', being that part of a Mineral Resource for which quantity and grade (or qualities) are estimated on the basis of limited geological evidence and sampling. The Inferred Mineral Resource estimate is constrained within the reef with an assumed average thickness of 0.68 m and an average grade of 23 g/t Au.

Mining One was engaged in 2018 to conduct a study to evaluate the potential of exploiting the Birthday Reef through mechanised mining. This reduces the risks associated with air-leg mining and increases production capacity. The report evaluates up-hole retreat longhole stoping with the use of paste as backfill.

Due to the depth and geometry of the Birthday Reef and surface land ownership constraints, the conventional approach of drilling the mineralization from surface is not possible. In 2020 it was decided to develop a 3,300 metre twin decline to beneath 16 level of the Birthday Reef, develop a drill drive and develop down three levels, to confirm and upgrade the Mineral Resource. By the date of this ITR, July 2022, the twin decline was progressed approximately 2,250 metres (plus 500m of crosscuts) and Birthday Reef access is expected in 1Q23.

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 Snowy River (Blackwater) Mine Long Section



The 2014 PEA 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, updated for inflation to April 2022. Environmental studies have been completed and Resource Consent documentation filed with the relevant regulatory bodies. The mining permit was granted in late 2018 allowing the twin decline to be developed and the underground mining operations to commence. Resource Consents, or permits, to create a waste rock stack and water treatment facilities were granted in 2014. In June 2022 the final Resource Consent documentation were filed with the expectation of all operational permits being granted in 1H23.

Table 1.1 Summary Key Parameters

Description	Base
Mine life	10 Yrs.
Mining Method	Long hole stoping
Ore Processing Method	Gravity/Float/CIP
Ore Processing Recovery	96%
Ore Mined (per annum)	300,000t
Mill Head Grade (diluted)	7.9 g/t
Gold recovered	699koz
Peak Annual Gold recovered	73koz
Project IRR at US$1,650	45%
Project NPV(5%, $1650)*	US$258M

*Assumes US$/A$ 0.70

2 PROPERTY LOCATION

The Snowy River Gold 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 (Figure 2.1 and **Error! Reference source not found.**). 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.

Figure 2.1 South Island Map showing Snowy Gold Mine Location

The Mineral Resource ("Birthday Reef") 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 via State Highway 7 (SH7) from Hukarere to the hamlet of Blackwater (7km sealed) and on to Waiuta (7km unsealed).

The project is located within mining permit MP 60473, covering an area of 2,500 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 is being accessed by twin declines, which are being constructed from land adjacent to the Snowy River near Hukarere. The portals and mine site are accessed via the bitumen Snowy River Road, 8 km from the turn-off from State Highway 7.

Figure 2.2 Blackwater Snowy Gold Mine Local Location Map showing MP 60473 and EP 60460

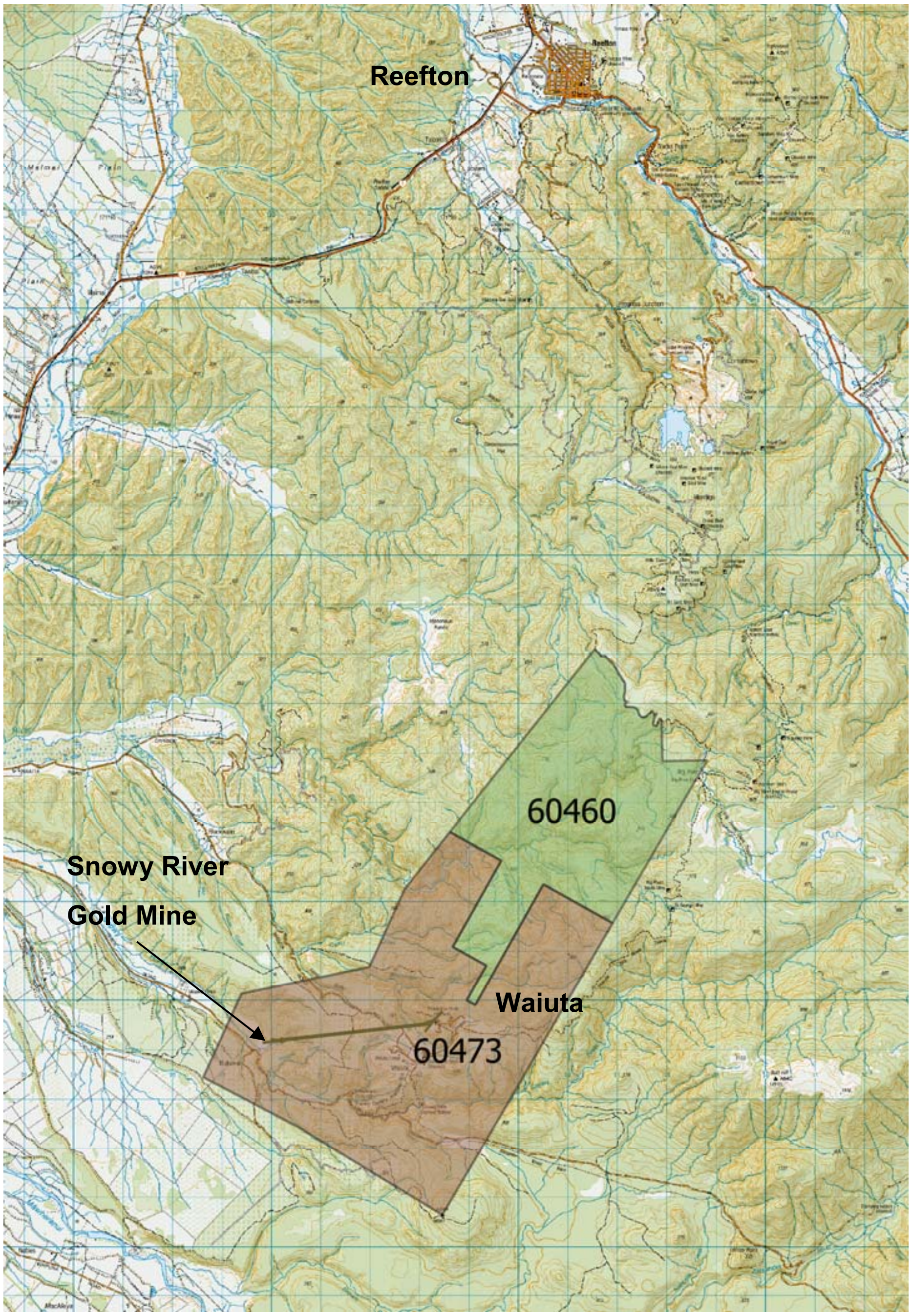


Figure 2.3 Snowy Gold Mine – Portals and site looking east towards Waiuta, June 2022

2.1. Physiography and Vegetation

The project area covers moderately steep country on the western foothills of the Victoria Ranges. The area is drained by numerous deeply incised creeks and streams, tributaries that flow into the Māwheranui (Grey) and Māwheraiti (Little Grey) river system. The portal site and associated infrastructure is located on the margins of alluvial floodplains formed by the Snowy River.

Most of the project area was logged and cleared to source timber for historic mining operations in the early to mid-1900s. The area is now largely covered largely by regenerating beech or beech/podocarp forest that forms part of the 206,000 hectares Victoria Conservation Forest Park. Part of the project area is in exotic pine forest that is owned and administered by Ngai Tahu Forestry. This forest has recently been harvested and replanted.

The Snowy River Mine portal site, offices, waste rock stack and settling ponds/wetlands area lies on private land that was historically alluvial mined. Most of the area at project start-up was gorse scrub and rough exotic pasture, with some regenerating mixed beech forest, located on the eastern hill slopes. Federation has begun planting of mixed native trees, shrubs, grasses and flaxes as part of planned progressive rehabilitation of the site.

Figure 2.4 Snowy Gold Mine – Portals June 2022

2.2. Climate

The Reefton area climate is moist and temperate, with the average annual rainfall of 2,219mm (recorded in Reefton from 1974 to 2004). Spring is generally the wettest season and late summer/early autumn the driest. 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.

3 LEGAL TENURE AND PERMITTING

The Snowy Gold Project comprises two permits, Mining Permit MP 60473 (Blackwater) and Exploration Permit EP 60460 (Bullswool).

OGC is the owner of the Mining Permit MP 60473 and is the registered Resource Consent holder at this point. FML has an exclusive option to purchase the Project before 31 December 2023 for US$30M cash or scrip ("the Option"), at FML's choice. FML has been appointed the 'Operator' of the permit by OGC. Since 2019 FML has been responsible for meeting permit work program obligations, developing the twin decline, construction of the waste dump and water treatment plant, infrastructure and environmental work.

Exploration Permit EP 60460 comprises a contiguous block of prospective rocks north of the MP 60473. The permit is held by FML's New Zealand subsidiary Tasman Mining Limited.

3.1. 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 a 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.

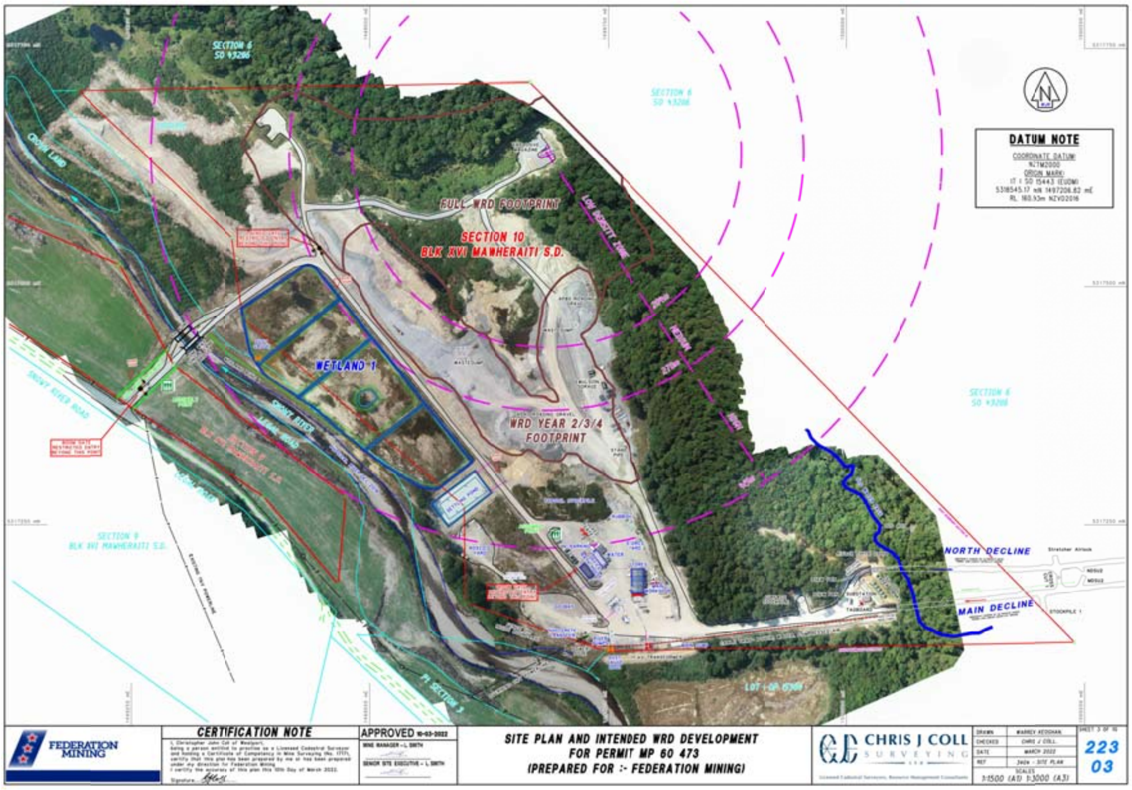
The following features of the Project are of relevance:

- The gold being targeted for 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 ("RMA");
- 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 RMA; 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.

3.2. Rights to Land

In legal terms the tenement comprises part of section 10 Block XIV Mawheraiti Survey, District Nelson Land District, NL10A/347.

Figure 3.1 Surface Site Layout



The Snowy Mine site is currently leased in perpetuity from Granville Mining (B McInroe - Owner) by OGC, with FML as a subcontractor of OGC. The terms of the lease allow OGC (FML) to carry out portal construction, decline development, necessary site works and mining/processing. The lease terms for the portal site also allow for exploration activities to be carried out (eg drilling).

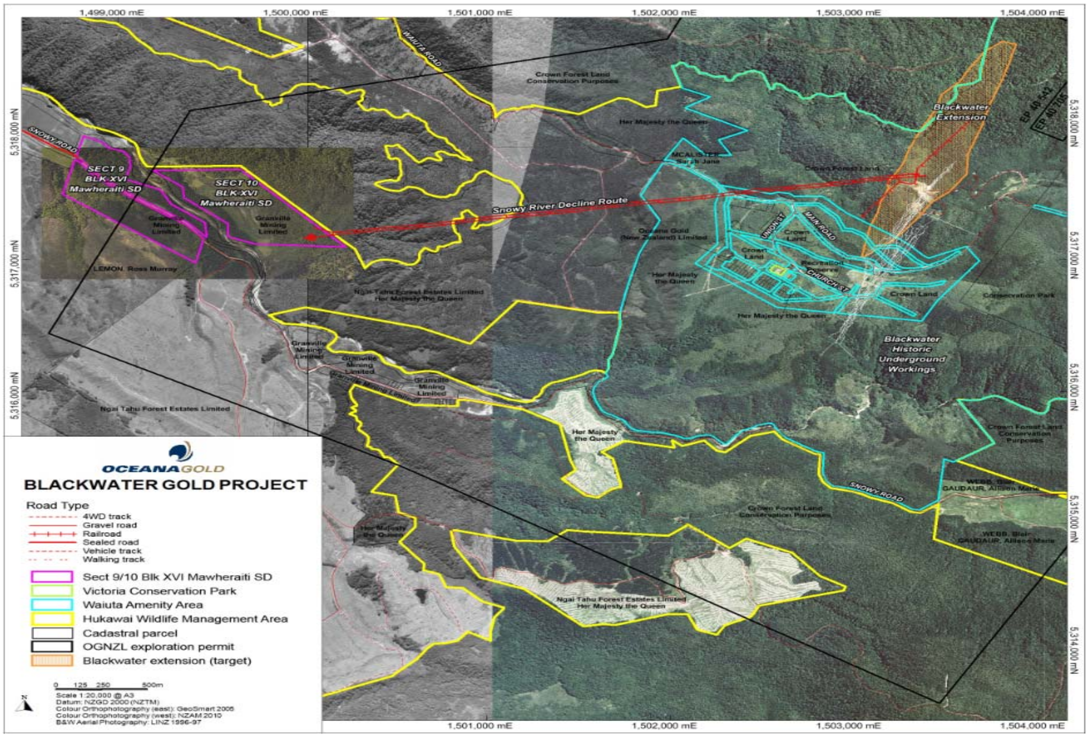
Road access to the project site was upgraded (sealed up to the site entrance from Snowy River Road) and a new bridge constructed over the Snowy River in 2020.

To undertake surface exploration work, an access agreement is required with the landowner. Land ownership is shown in **Error! Reference source not found.**. The major land owners for MP 60473 are the Crown (administered by the Department of Conservation) and Ngai Tahu Forestry (owned by the local Iwi). EP60460 is entirely on Department of Conservation land. FML/OGC have formal access agreements in place for minimum impact exploration work (geological mapping, soil sampling and wacker drilling) with both the Department of Conservation and Ngai Tahu Forestry.

3.3. Underground Workings

The underground workings of the proposed Snowy River 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.

Figure 3.2 Land Designation



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 FML will not require any access arrangements with the owners of the land through which the Snowy River Mine underground workings pass, provided that the Company's activities have no surface impact on those landowners.

Notwithstanding the position under the CMA, FML 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 RMA, the 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 Snowy River resource consent applications.

3.4. 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. Crown-owned minerals are administered by New Zealand Petroleum and Minerals (NZ P&M), a department of the Ministry of Economic Development.

The Snowy River/Blackwater Mineral Resource lies within Mining Permit MP 60473 (Blackwater). MP60473 was granted on the 19 December 2018 for a term of 20 years. It gives OGC the exclusive right to mine and explore for gold in the permit area. The permit shall transfer to FML upon exercise of its exclusive Option. FML, through its New Zealand subsidiary Tasman Mining Limited, has been appointed by OGC as the permit operator.

The permit is classed as a 'Tier 1' Permit under New Zealand Crown Minerals Act 1991 and the Minerals Programme (2013). The permit document stipulates a minimum work program and deadlines that must be met to maintain the permit in good standing. Project funding and COVID delayed the project timeline, so a change in consent conditions application was made to NZ P&M. This was approved in July 2023. The amended work program for MP 60473 (Schedule 3) is shown below:

1. Within 48 months of the commencement date of the permit, the permit holder shall (to the satisfaction of the chief executive):
    (a) secure access to and establish the portal site and appropriate surface infrastructure for decline construction;
    (b) construct an appropriate box cut;
    (c) construct a twin access decline from surface to access the Birthday Reef;
    (d) carry out continual drilling in advance of decline construction;
    (e) collect geo-mechanical, metallurgical, geochemical and environmental data appropriate for informing a Feasibility Study, optimised mine development and safe mine operation;
    (f) prepare a technical report detailing all work completed during this stage of the work programme in conjunction with QAQC information and data sufficient to demonstrate levels of accuracy and precision to be submitted to the chief executive in accordance with the regulations.

2. Within 72 months of the commencement date of the permit, the permit holder shall (to the satisfaction of the chief executive):
    (a) construct exploration drives once access is to the Birthday Reef is achieved;
    (b) update a portion of the mineral resource to at least an Indicated classification as defined under a recognised resource classification code as per Schedule 1 of the Minerals Programme;
    (c) update a portion of the mineral resource to at least an Indicated classification as defined under a recognised resource classification code as per Schedule 1 of the Minerals Programme;
    (d) complete a mine feasibility study;
    (e) complete a Mineral Ore Reserve estimate as defined under a recognised resource classification code as per Schedule 1 of the
    (f) complete construction of processing plant and other mine infrastructure to enable processing of first ore;
    (g) prepare a technical report detailing all work completed during this stage of the work programme in conjunction with QAQC information and data sufficient to demonstrate levels of accuracy and precision to be submitted to the chief executive in accordance with the regulations; and
    (h) provide the chief executive a forward looking work programme for the remainder of the permit's duration for approval by the Minister. The forward looking work programme shall include a commencement date of commercial mining and a minimum annual production rate that factors in:
        (i) the estimated Mineral Ore Reserves;
        (ii) the production schedule from the Feasibility Study;
        (iii) the mining method(s) to be used; and
        (iv) the need to follow good industry practice.

3. The permit holder shall, to the satisfaction of the chief executive, carry out the following work programme:
    a. in conjunction with annual reporting under the relevant regulations, unless otherwise already provided for, provide the chief executive with a plan in a digital format of all mine workings and planned development, and the timing of the development in line

with the guidelines on Completing and Submitting Plans on Mines and Tunnels (2017) or any varied guidelines that may subsequently be issued; and

b. Notify the chief executive of the discovery of any other deposits beyond the Blackwater deposit.

With regards to the permit work program commitment's, FML has completed items 1(a) (surface site infrastructure) and 1(b) (portal construction). Items 1(c) to 1 (f) are in progress and these program obligations will be met within the time frame stipulated.

Permits as of July 2023:

Mining Permit MP60473 (OGC)

Exploration Permit EP 60460 (FML/Tasman Mining Limited)

The Bullswool permit EP 60460 was granted for a 5-year term on the 19th of July 2018, covering an area of 1612.48 hectares. The permit area lies within reserves administered by the Department of Conservation. The permit is held by FML's New Zealand subsidiary, Tasman Mining Ltd.

EP60460 is in good standing and budgeted exploration for the summer 2022-23 field season will meet work program commitments. An extension of duration which gives another 5 years life to the permit will be applied for before the permits current expiry on the 19th of July 2023. In addition, if certain conditions are met, the Crown Minerals Act 1991 allows a single further extension of the EP of up to 4 years for appraisal purposes.

3.5. Health and Safety

Health and Safety for Mining Operations New Zealand is governed by the Health and Safety at Work (Mining Operations and Quarrying Operations) Regulations 2016 and the Health and Safety at Work Act 2015. This legislation is administered by a Crown agency, Worksafe, who undertake regular inspection of the operation.

FML has appointed statutory management and supervisory positions with the appropriate mining experience and certifications as set out in the regulations.

Within the Health and Safety at Work (Mining Operations and Quarrying Operations) Regulations 2016, a Principal Hazard Management Plan (PHMP) needs to be developed for any hazard arising at any mining operation that could create a risk of multiple fatalities in a single accident or a series of recurring accidents must be identified, managed and controlled. FML has PHMPs in place for the following principal hazards:

- Ground and Strata Instability.
- Inundation and Inrush.
- Roads and Other Vehicles Operating Areas.
- Tips, Ponds and Voids.
- Air Quality.
- Fire Explosion.
- Explosives.

FML has a Health and Safety Management system in place to manage safety at the project. This includes:

- inductions
- standard operating procedures (SOPS)
- training and assessment
- job hazard analysis (JHA)
- risk assessment

- hazard reporting
- incident reporting and investigation
- worker health monitoring

3.6. 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.

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

3.7. Royalco Royalty

In addition to Crown royalties, the Blackwater Mining Permit (MP) is also subject to an agreement contained in a Deed of Novation between Royalco Resources Pty Ltd (Royalco) and OGC, 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 Snowy River/Blackwater tenements.

The Royalco agreement grants OGC an option to buy back the royalty over Blackwater MP for the sum of A$5,000,000, CPI adjusted from 14 May 1991. The option to buy back the Royalco royalty over Blackwater MP may be exercised at any time until OGC makes a decision to mine MP60473 and applies to all gold produced from that decision (i.e. from EP 40 542). The FML Option over the Project transfers the Royalco purchase option to FML. It is estimated the Royalco royalty purchase price will range from A$11M to A$14M, payable concurrently with the Project Option exercise.

3.8. 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 HISTORY

The Blackwater gold deposit was discovered on the 9th of November 1905 by small prospecting group, who discovered a gold-bearing quartz vein in Greek Creek, a tributary of the Grey River. The discovery was made on King Edward VII's birthday, so it was loyally named the Birthday Reef.

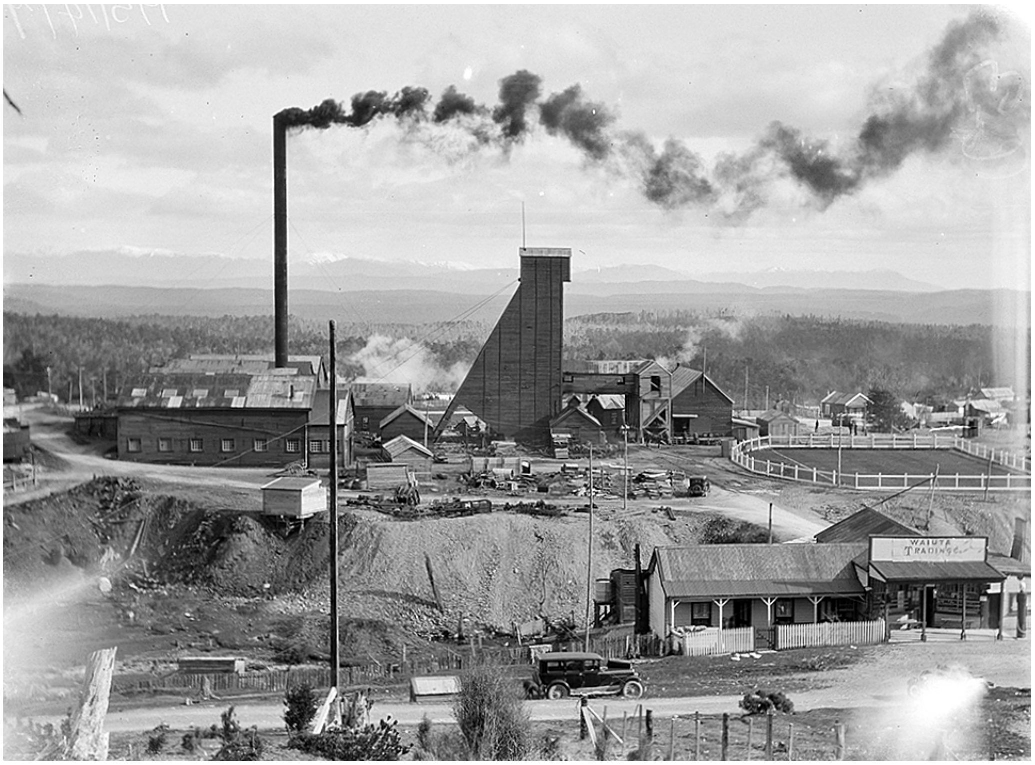
The discovery party optioned the claim to a local mining investor, Percy Kingswell for 2,000 pounds. Kingswell purchased the claim and then sold a six-month option for 30,000 pounds to Consolidated Goldfields (NZ) Ltd, a large British-owned company that dominated the Reefton quartz mining industry. The reef was initially prospected by trenching and developing a series of 100-foot spaced winzes and drives, which demonstrated the reef had substantial strike extent. Consolidated Goldfields exercised their option and purchased the claim. They later vended the project into a new company, Blackwater Mines Limited and in 1907 raised 50,000 pounds for working capital.

A thirty head stamp battery was built adjacent to the neighboring Snowy Creek and gold production began in 1908. Development was from the Blackwater Shaft (South Shaft) which was sunk to 563.7 metres. Levels were every 150 feet (45 metres).

Figure 4.1 Snowy River Battery and incline – looking south, 1930s

(Auckland Libraries Heritage Collections 1370-272-2)

Figure 4.2 Blackwater Shaft and Waiuta Township, 1930s, looking west



(Sir George Grey Collection Auckland Library 1370-272-15)

To the north of the Blackwater Mine, Blackwater North Mines limited was formed in 1911, aiming to mine potential northerly extensions to the Birthday Reef. A crosscut was developed from Coorong Creek (the Prohibition cross-cut) and a shaft sunk (the Prohibition Shaft). By 1917 the shaft had been sunk to 411m but no payable mineralization was discovered and the company ran out of money.

As time progressed it became evident the Birthday Reef plunged moderately north towards the Prohibition Shaft. Blackwater Mines acquired Blackwater North Mines in 1927. The Blackwater leases from Level 7 down where then extended into the northern Lease. By 1936 the Prohibition Shaft had become the main ore hoisting shaft, with the Blackwater Shaft largely being used for mine ventilation and dewatering. The Prohibition Shaft was sunk to 878 metres, with ore extraction down to the 16 Level (685 metres depth). A new modern processing plant was built in 1938 adjacent to the Prohibition Shaft (the Prohibition Mill, Figure 4.3).

The Blackwater mine closed in 1951, as a result of a ground failure in Blackwater Shaft below the Two Level. The shaft was used for mine dewatering and services. The mine had been in financial difficulties for many years due to wartime gold taxation, labour and materials shortages, low gold prices, labour disputes, and a failure in the Prohibition Shaft in 1949 that had required a lengthy repair. Due to these compounding issues, it was elected to close the mine rather than repairing the shaft.

Figure 4.3 Prohibition Mill, late 1930s

(Jos Divis)

Following the mine closure, the township of Waiuta was largely deserted and Blackwater Mines Limited was placed into receivership in 1952. In the ensuing years the project has been owned by several different companies (Table 4.1). Since the early 1990s the project has been owned by Macraes Mining Company Limited, Gold and Resource Developments (New Zealand) Limited and GRD Macraes. These companies are predecessor entities of OceanaGold (New Zealand) Ltd, which is the current registered owner of the Blackwater Mine, subject to FML's Option.

Table 4.1 Blackwater Mine Ownership History

Year	Mine Owners
1905	PN Kingswell
1906	Consolidated Goldfields Ltd through 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)

4.1. Historic Production

The Blackwater Mine was the largest underground producer in the Reefton Goldfield with a production of 740,000oz of gold (23t) from 1.6Mt of ore (recovered grade 14.6g/t Au), for the period 1908 to 1951.

Table 4.2 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 4.2 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

Year	Long Tons	Tonnes	Ounces	kg Au	Recovered Grade (g/t Au)	Back Calculated In-situ Grade (g/t Au)
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
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

4.2. Historical Blackwater Gold Mine Ore Reserves

Table 4.3 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 4.3 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 (28koz), allowing for 50% mining dilution.

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. 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. Figure 4.4 illustrates typical reef conditions and timber support when the Blackwater Mine was in operation during the early part of the twentieth century.

Figure 4.4 Underground in the Blackwater Mine

4.3. Historical Studies and Historic Mineral Resource Estimates

Several studies of the Blackwater Mine have been completed since the mine closed in 1951, and there have been a succession of historical resource estimates. These are summarised in Table 4.4, along with the work program recommended for each study. More detail on the various work programs completed is included in the following sections.

Due to the changing reporting requirements over the years, many of the old reports use antiquated terminology and methods. Estimates prior to 1995 in **Error! Reference source not found.** 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.

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

Table 4.4 Historical Resource Estimates

Year	Company	Resource Estimate Stage 2 Stage 3			Proposed Surface Drilling	Proposed UG Development	Proposed UG Drilling
		Tonnes (Mt)	Grade (g/t)	MOz			
1975	Carpenteria Exploration	0.59	21.9	0.416			
1987	James Askew	0.32	21.5	0.220	3 parent holes		
					9 intersections		
1987	CRA Exploration	1	10 - 15	0.3 - 0.5	4 parent holes		
					12 intersections		
1991	GRD Macraes	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	Emperor Gold Mining Company	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	GRD Macraes	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	GRD Macraes	1.7	22 - 25	1.2			
1994b	GRD Macraes	0.31	21	0.21			
1994	Gemell Mining Engineers					1,000m reef driving	total cost
						500m in cross cutting	$2.35M
						10 drill cuddies	
1995	GRD Macraes	0.18	22	0.13	2 parent holes		
					6 intersections		
1996	GRD Macraes	0.31	21	0.21			
1997	GRD Macraes	0.46	21	0.31			
2003	OceanaGold (NZ) Ltd	0.48	22	0.3	1 parent, 1 daughter	Re-furbish shaft	25 holes
						To 17LevelDrill drives	
2005	OceanaGold (NZ) Ltd	0.94	9.8	0.28	1 parent, 1 daughter	New shaft Decline & drives	25 holes
2011	OceanaGold (NZ) Ltd	0.48	22	0.3	6 parent holes plus daughters	Decline & Drives	9,000m

4.4. 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’.

4.5. 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.

4.6. 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.

4.7. 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 (29koz) 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 (50koz) between the 16 and 17 Levels, for a total of 180,000t at 13.6g/t Au in ore reserves (79koz). 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).

4.8. 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 (30koz) between 16 and 17 Levels and then 'geological reserves' of 1,000m by 400m by 1m (at 2.5t/m^3) for 1Mt at 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.

4.9. 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.

4.10. 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.

4.11. 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 4.5 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 4.6 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.

4.12. 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.

4.13. 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.

4.14. 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 platform 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.

4.15. 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.

4.16. 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 4.7 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**	

4.17. 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.

4.18. 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 enroute 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.

4.19. 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 110 ktpa 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 4.8 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 OGC listed 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.

4.20. 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.

4.21. 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. 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.

4.22. 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 (inches) and assay results (pennyweight per ton) 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 4.9 Analysis of Level Face Sampling, Stope Widths, 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 4.5 Grade and Width Distribution in 431 Samples of Reef on Levels 11-16

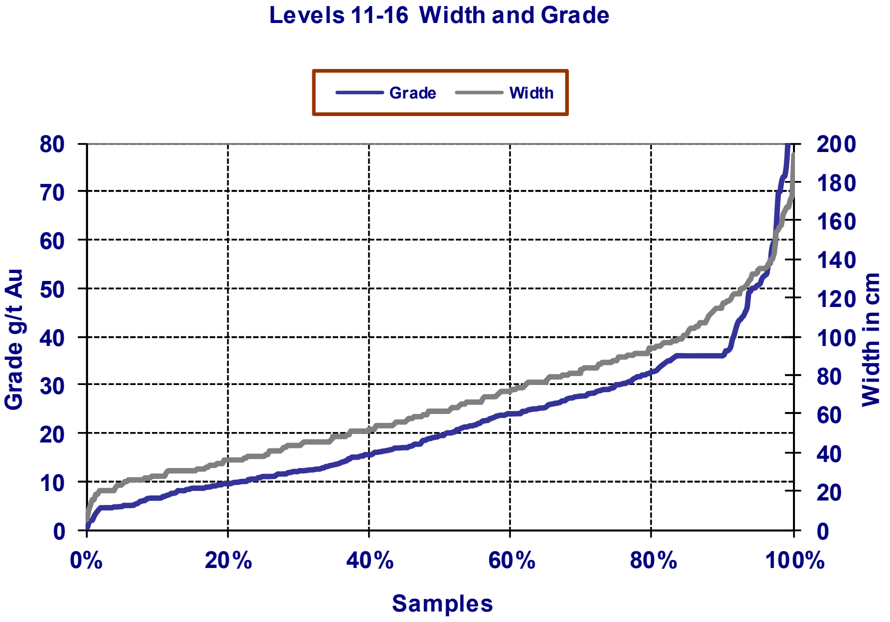


Figure 4.6 Grade and Width Distribution in 1,083 Samples of Reef on Levels 4-13

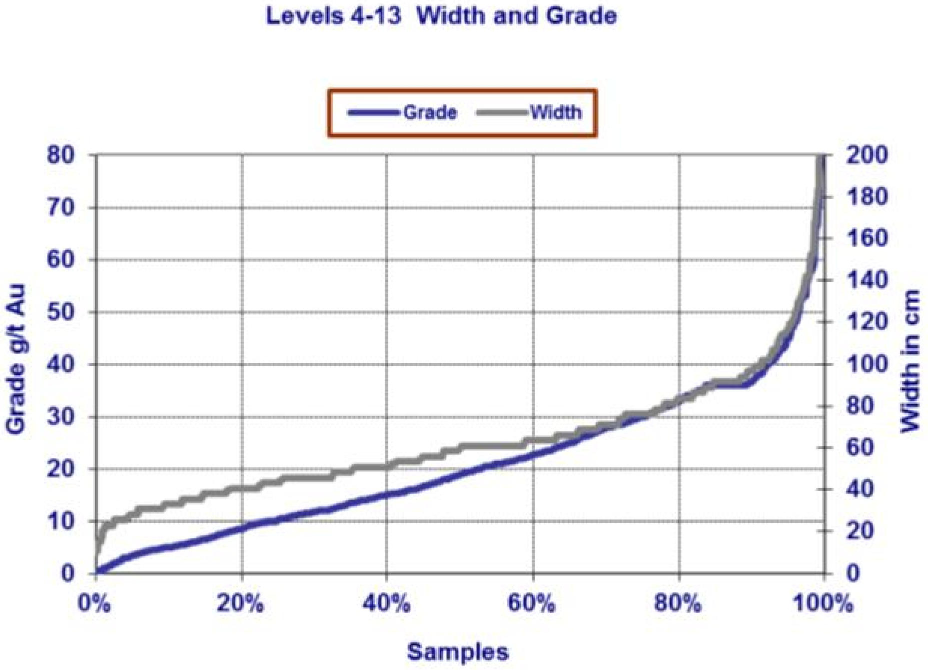


Figure 4.7 Grade and Width Distribution in 334 Samples from Stopes on Levels 1-16

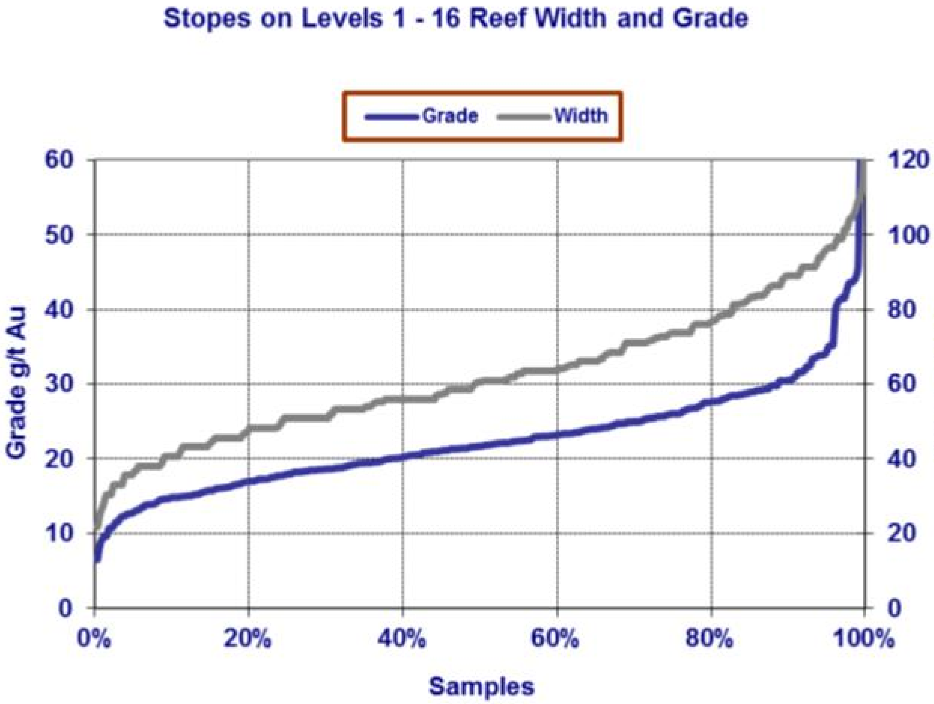


Figure 4.8 Grade versus Width Distribution of Reef Samples on Levels 11-16

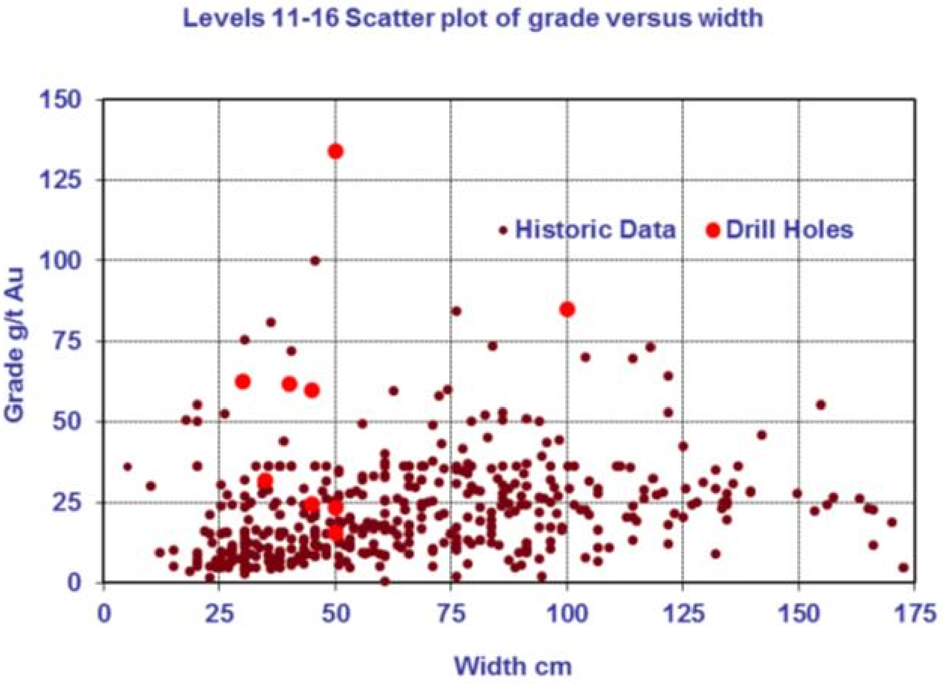


Figure 4.9 Grade versus Width Distribution of Reef Samples on Levels 4-13

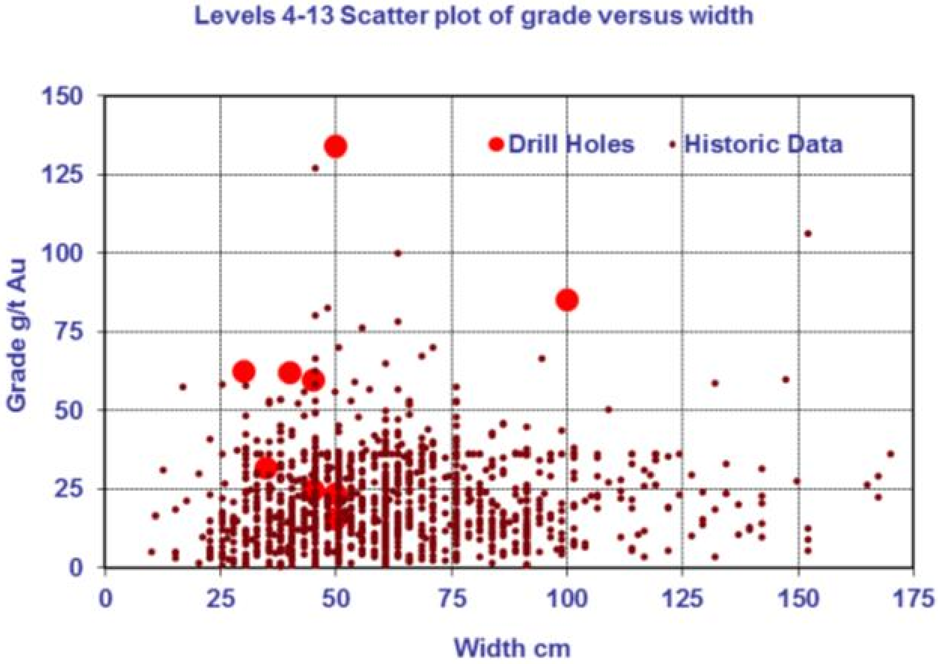
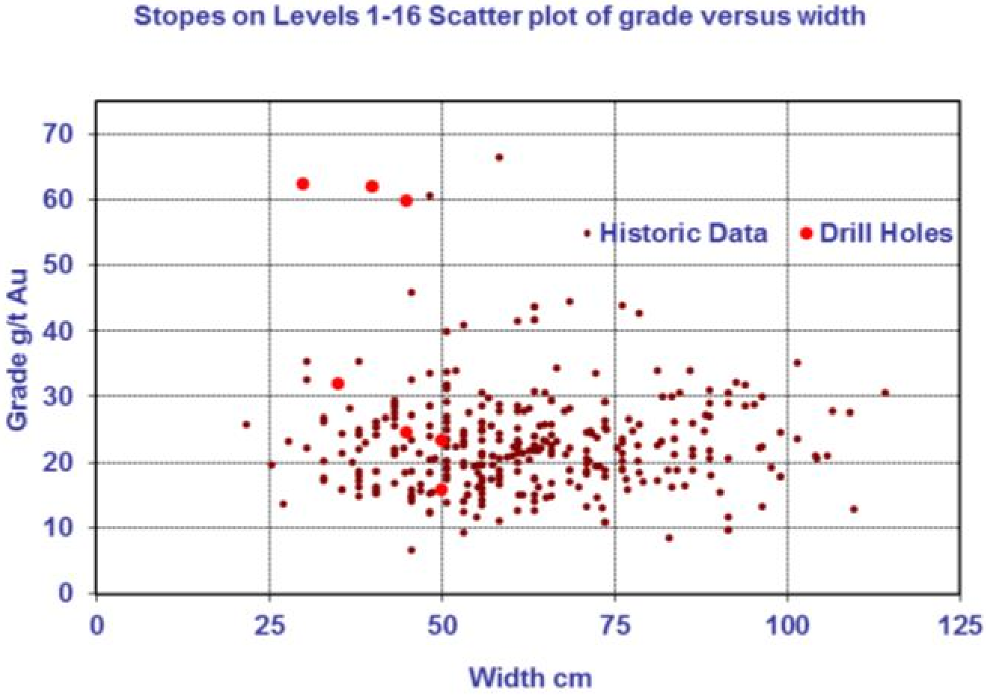


Figure 4.10 Grade versus Width Distribution of Reef Samples on Levels 1-16



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 4.8 through Figure 4.10) 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.

5 GEOLOGICAL SETTING AND MINERALISATION

The Birthday Reef is an orogenic mesothermal quartz vein-hosted gold deposit situated in the western limb of a regionally significant anticline, proximal to the anticline hinge. Mineralised veins tend to strike sub-parallel with the regional geological structure. The deposit is hosted within a sequence of Ordovician age folded turbidite sediments (The Greenland Group).

5.1. 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.

5.2. 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).

5.3. 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).

5.4. Igneous Rocks

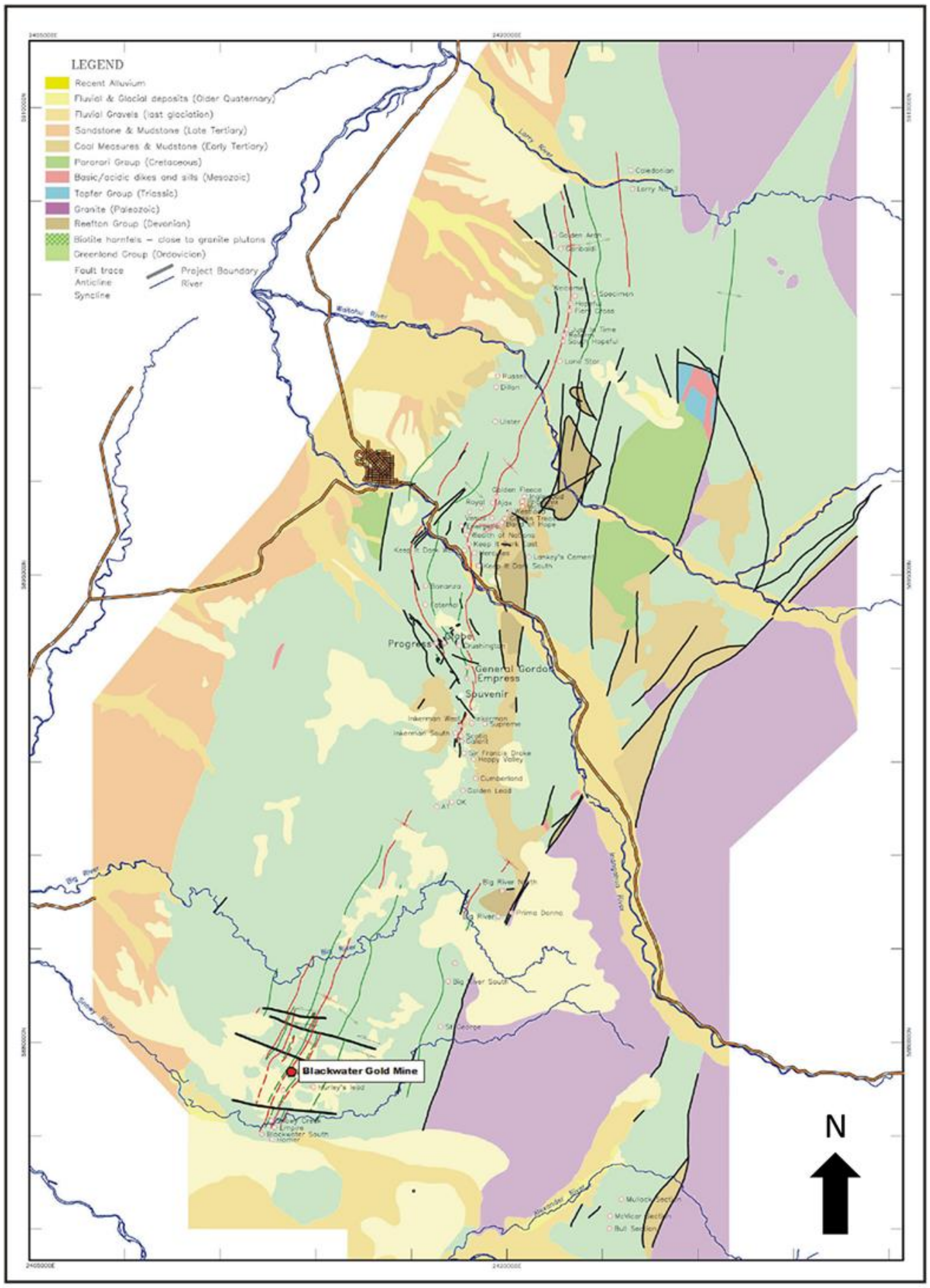
Igneous activity followed the deformation and metamorphism, with the emplacement of widespread Karamea Suite S-type granitoids in the mid-Devonian (375Ma), with a second minor period of granitoid intrusion in the Carboniferous (330Ma). A third intrusive event, the Rahu Suite, comprising relatively small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

5.5. 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 5.1 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.

5.6. 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 (**Error! Reference source not found.**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 channel ways and traps for ascending mineralised fluids.

5.7. 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 5.1). 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.

5.8. Local and Property Geology

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.

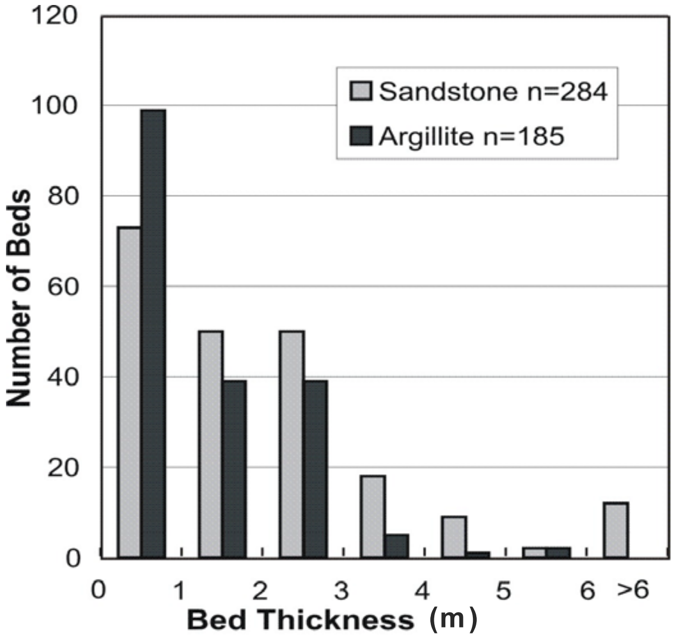
5.9. Lithology

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 5.2).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 phyllosilicate 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 5.2 Drill Hole WA11 Bed Thickness (Cox, 2000)



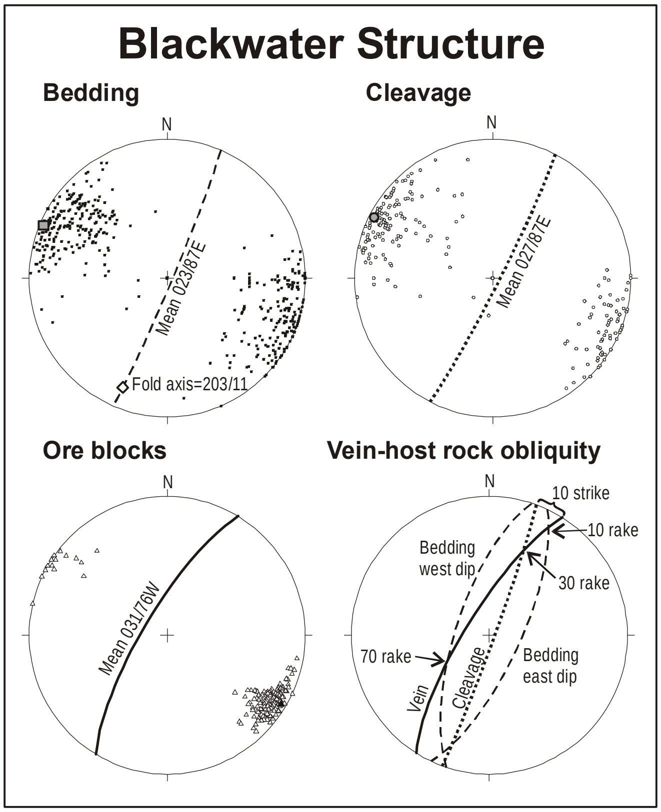
Unconformably overlying and masking the Greenland Group basement rocks are a series of Pleistocene fluvioglacial terrace deposits. These are horizontally bedded, well-consolidated moraine outwash gravels comprising mainly greywacke, argillite and granite cobbles and boulders in a clayey-sand matrix. 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.

5.10. 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 5.3). 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 5.3 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 - 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.

5.11. 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 5.3).

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 (**Error! Reference source not found.**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 selected previous reports and analysed historic plans and drill core.

Table 5.1 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 have inferred a displacement of up to 10m on the North West Fault, whist Morgan (1929) Inferred only minor displacement.

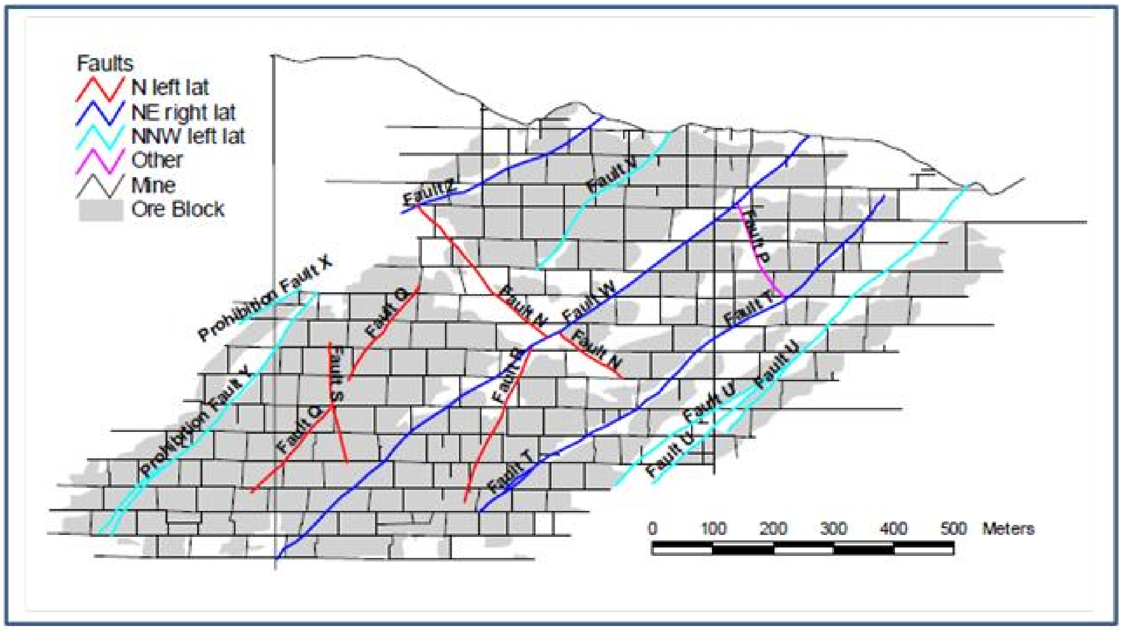
Murfitt’s (1975) reef parallel shearing and interpretation are valid, although not discussed by other authors. Recent drill holes indicate that shearing along the reef is localised to within 15cm of the hanging wall and less in the foot wall of the reef.

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, **Error! Reference source not found.**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 Anticline 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.

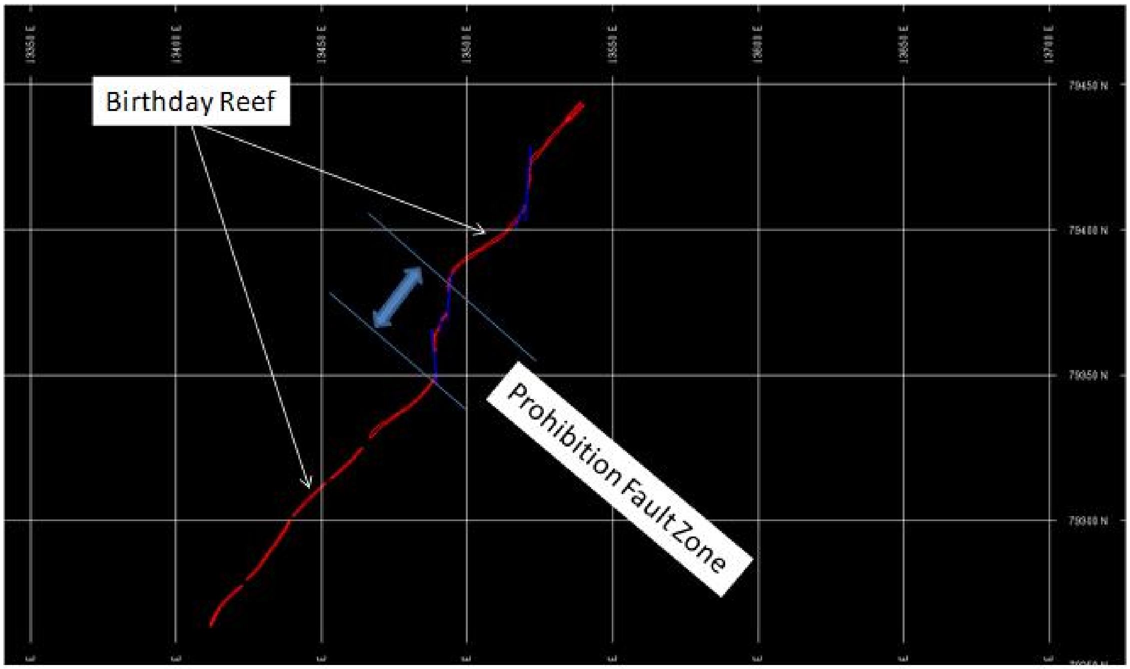
Figure 5.4 Blackwater Mine Late Stage Faults (Cox & Rattenbury, 2004)



The disruption of the Birthday Reef by the Prohibition Fault Zone and other parallel fault zones is depicted in **Error! Reference source not found.**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 stoping shown on Figure 5.4. 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 stereo plot 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 (**Error! Reference source not found.**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 5.5 Birthday Reef and Prohibition Fault on 14 Levels



5.12. 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 5.6 and Figure 5.7) 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 5.6 Blackwater Reef Lens Lengths - 11-26 Level

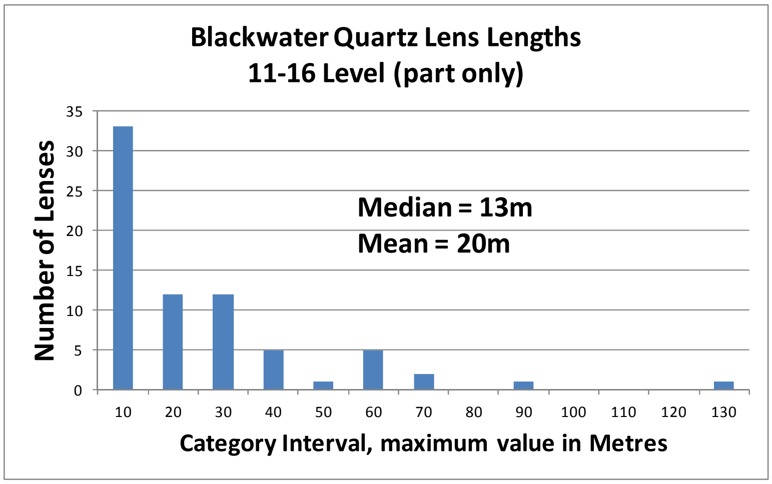
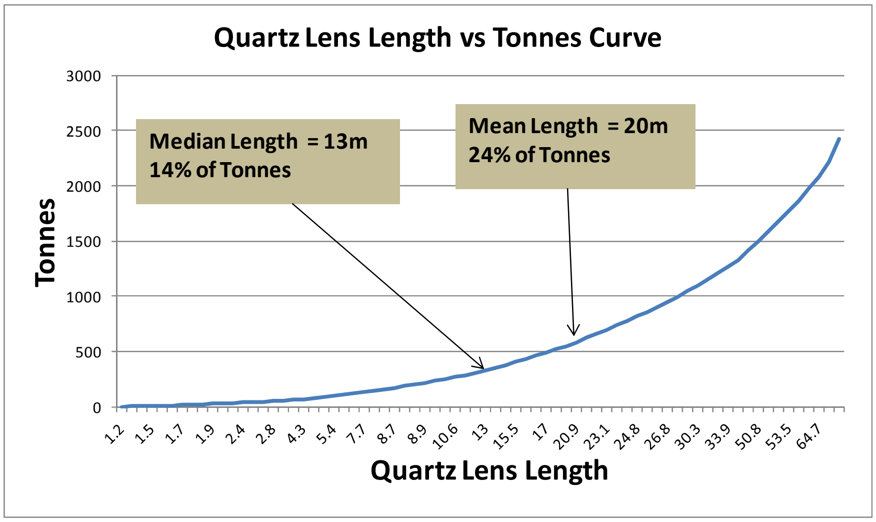


Figure 5.7 Blackwater Reef Tonnes vs Lens Length – 11-16 Level



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 (**Error! Reference source not found.**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 - 27.0 g/t Au.

The 16 Level sampling data (Figure 5.10) 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 5.8 Level 16 Face Grade

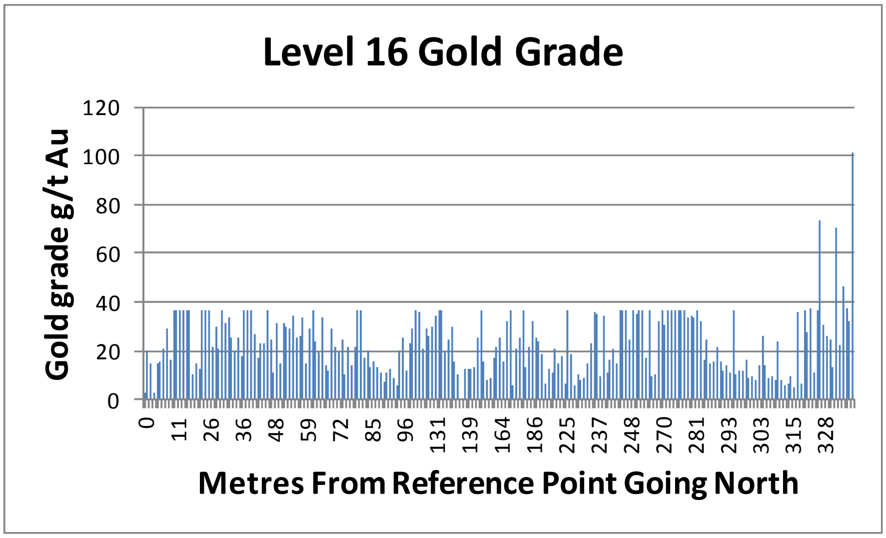


Figure 5.9 Level 16 Face Reef Thickness

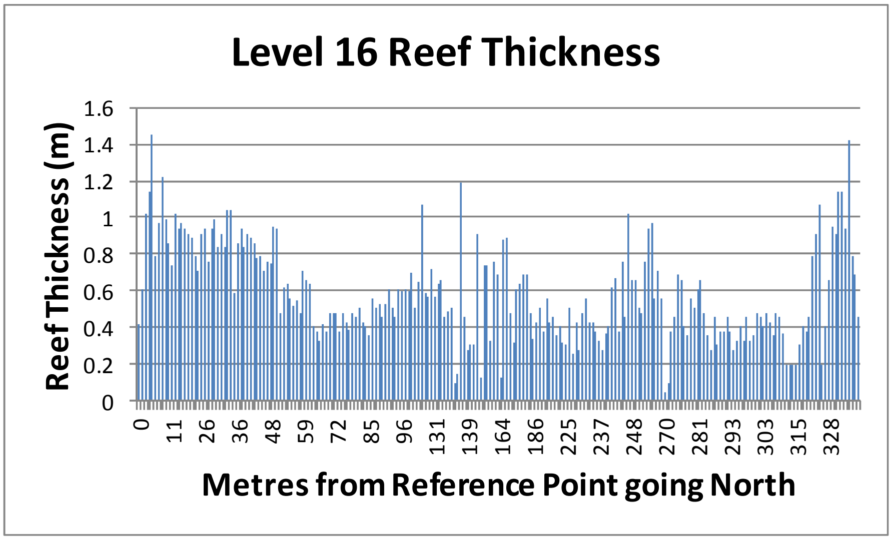
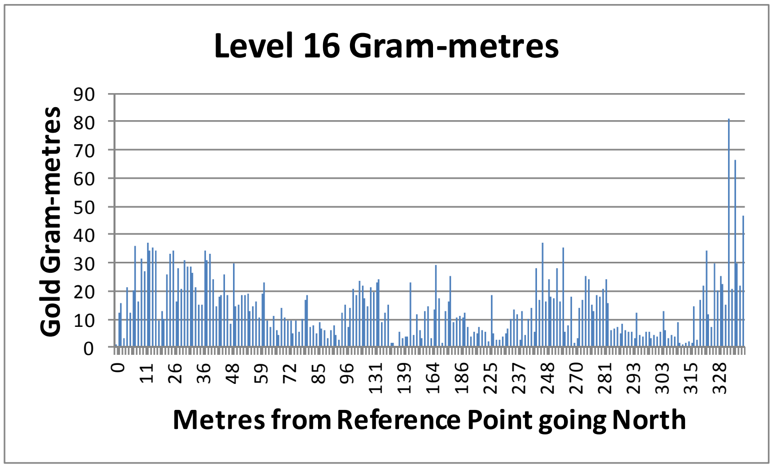
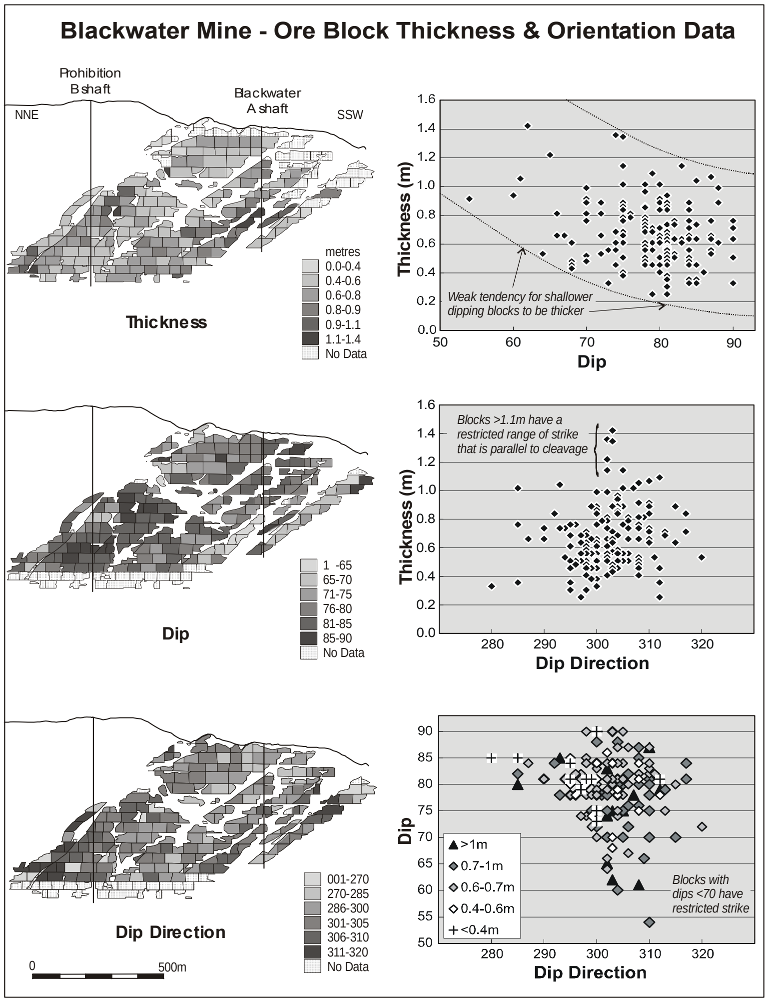


Figure 5.10 Level 16 Face Grade-Thickness



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

Figure 5.11 Blackwater Mine Stope Thickness and Orientation Data (Cox, 2000)



6 EXPLORATION

Exploration for significant undiscovered mineralisation in the vicinity of the Blackwater Mine has been carried out by several companies since the mine was discovered in 1905.

6.1. Historic Exploration

During the historical working of the mine there was limited geological mapping and no drilling. Work was primarily focused on driving the reef, with little exploratory driving outside of the known reef track. In the vicinity of the reef there was limited work to identify strike extensions or parallel veins. The only significant exploratory drives were:

- the Joker and Low Level Adits which transect the footwall of the reef for 200 metres on the south eastern margin of the Birthday Reef. These were largely used for access and haulage to the upper mine levels and for dewatering.
- Prohibition Adit was excavated east of the Birthday Reef in 1907, attempting to intercept the Birthday Reef and/or potentially parallel reef(s). Six reef tracks were encountered, but none were noted to contain payable stone (Downey, 1928).
- The McVicars adit was driven for 183 metres from Prohibition Road, searching for mineralization north of the Prohibition Fault. Carpentaria Gold sampled the adit, with the best assay returned 0.9 g/t Au.
- An unnamed 250m long crosscut to the east from the Blackwater Shaft on the Three Level, which apparently did not intersect ant significant mineralization.
- The Battery Crosscut which apparently intersected several reef tracks, none of which were contained “stone above cut-off grade” (Murfitt, 1975)

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. The only significant recorded gold production is from the Homer Mine (363 oz Au), 1 km to the south of Blackwater, and the Millerton Mine (1,675 oz Au), 3km to the west.

Figure 6.1 Blackwater Historic Mine Workings

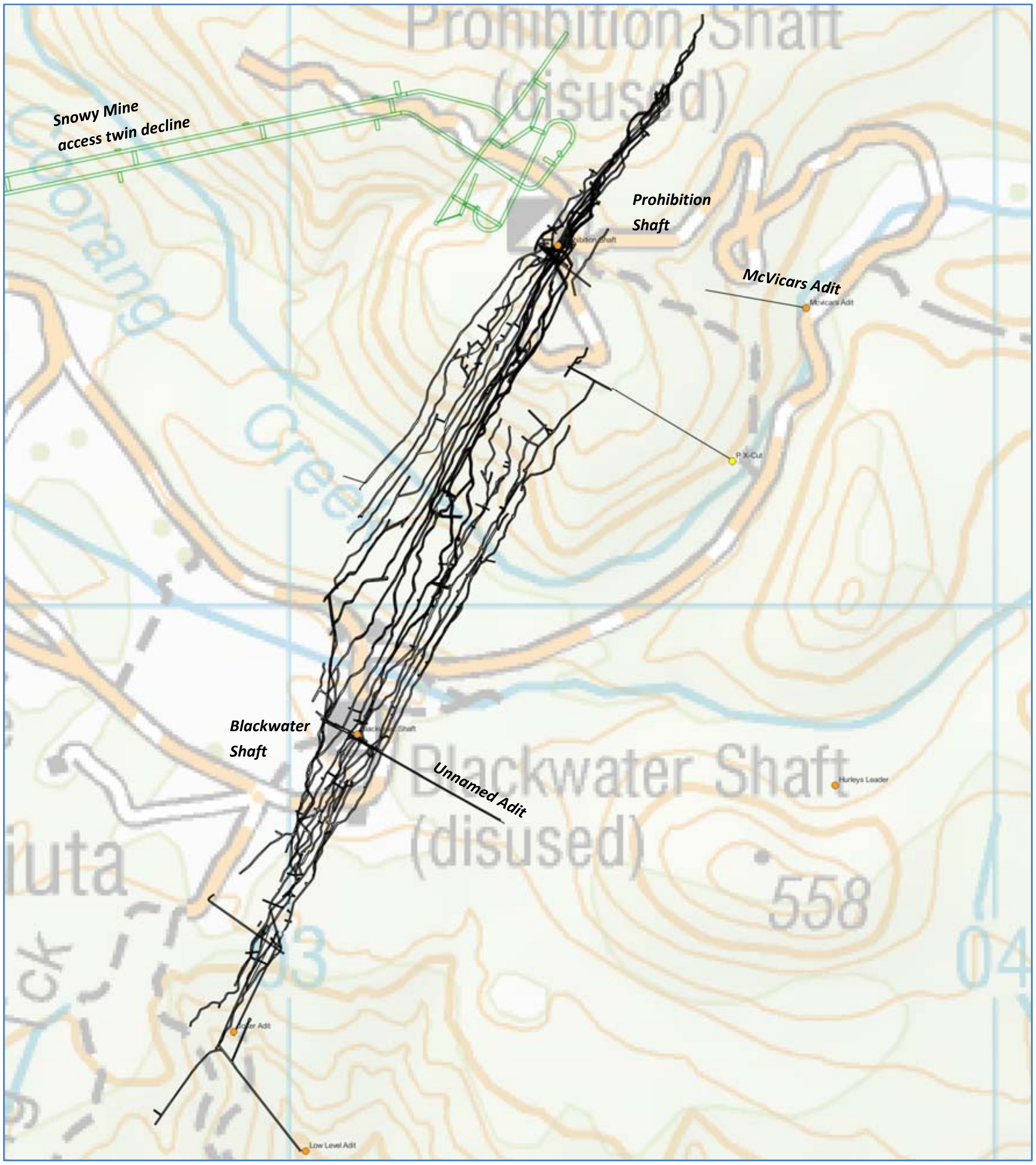
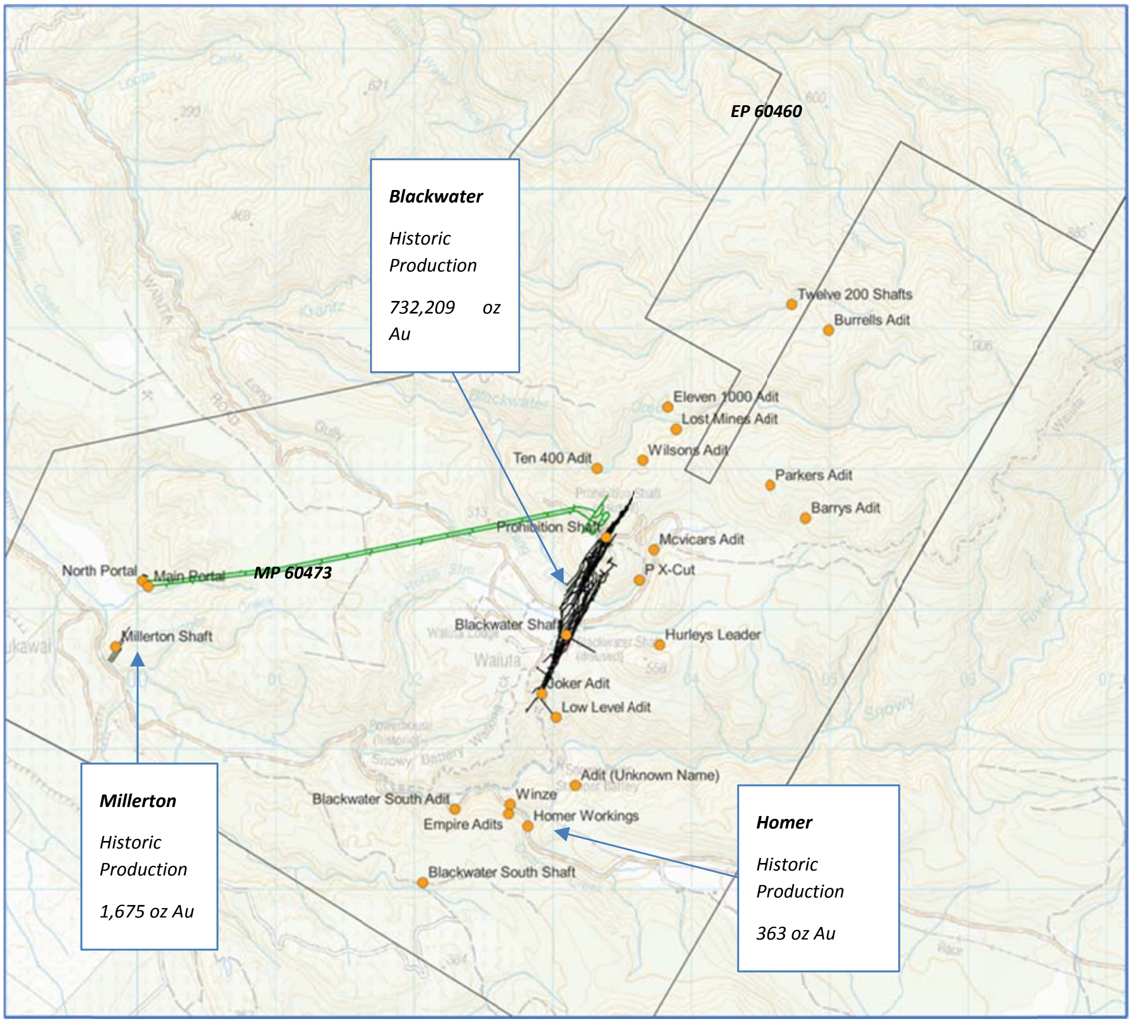


Figure 6.2 Blackwater Historic Mine Workings and recorded gold production



Blackwater

Historic Production

732,209 oz Au

Millerton

Historic Production

1,675 oz Au

Homer

Historic Production

363 oz Au

6.2. Modern Exploration

Several companies have explored the project area in modern times. In the late 1980s and early 1990s CRA Exploration carried surface geochemical sampling and small drilling programs focused on the Birthday Reef and potential strike extensions. The project was acquired by GRD Macraes (later to become OGC) in the early 1990s. Since then, work was largely focused on drilling strike and down dip extensions to the Birthday Reef, with small drilling programs testing potential new mineralized positions.

Structural mapping of the creeks and the slopes surrounding the Birthday Reef has been carried out by various geologists with mapping by OGC being the most comprehensive. Structural assessment of geological data from both underground and surface exposure has guided exploration drill targeting.

6.3. Drilling

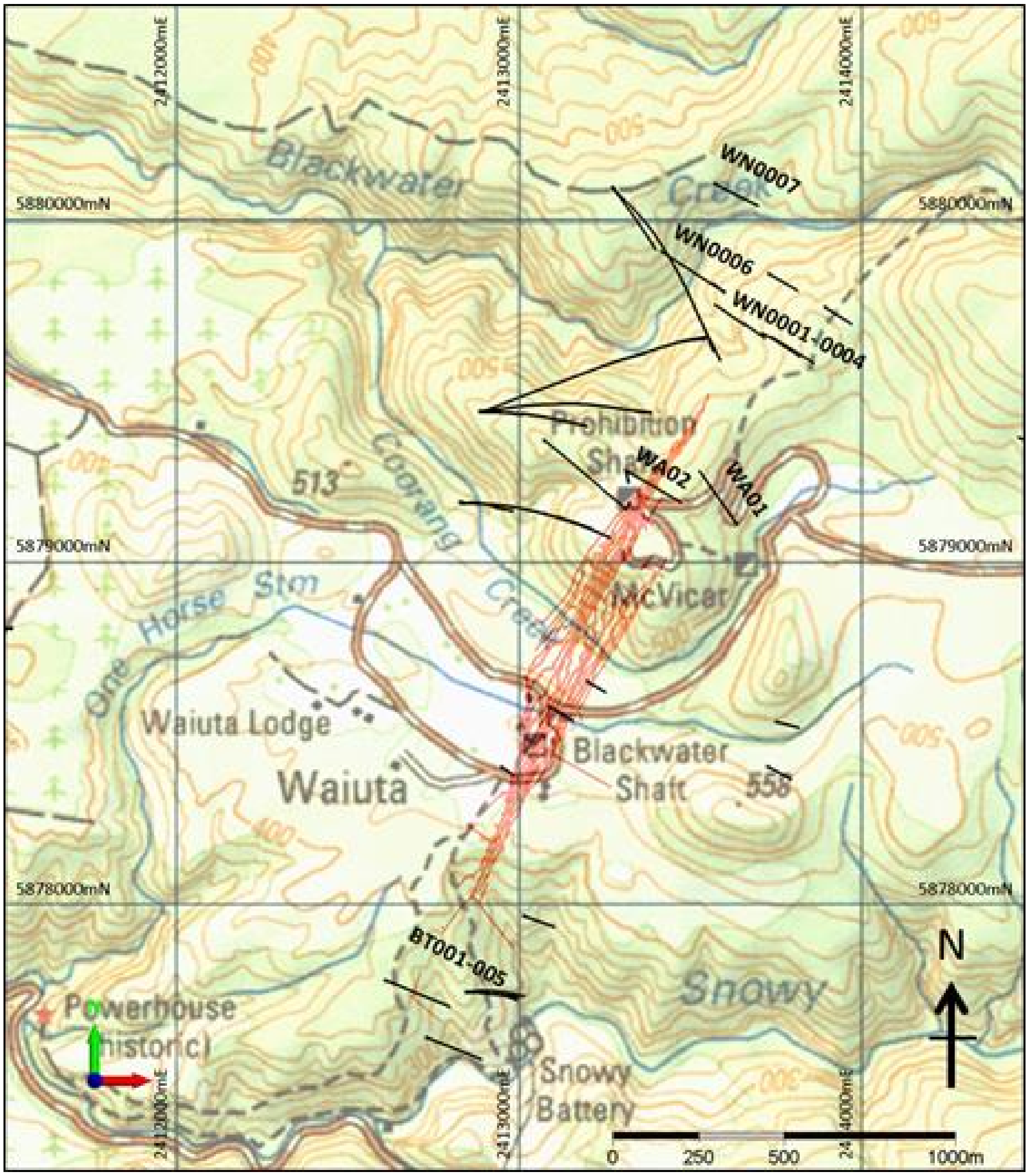
Drilling by OGC and its predecessor company GRD Macraes was largely focused on identifying extensions to the Birthday Reef, both down-dip and laterally.

WA01 and WA02 targeted a possible extension of the Birthday Reef immediately northeast of the mine lode, 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.

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.

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. Targeting of the reef in these deep holes was challenging, compounded by unplanned drill hole deviation in conjunction with reasonably acute drill hole intersection angle with the Birthday Reef. Daughter holes were drilled off the parent holes to re-evaluate zones of interest encountered in the parent, typically achieving 10m separation from intersection of the Birthday Reef from the parent hole.

Figure 6.3 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 OGL note there was high confidence that there had been no rotation of core in the barrel.

All the Blackwater diamond drill holes were systematically 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 6.1.

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. Drill holes WA21 to 25 were down hole surveyed using a REFLEX EZ-SHOT electronic single shot camera.

Table 6.1 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 6.2 and Figure 6.4). The pad sites are located approximately 0.5km to 1.5km north of the historic Waiuta mine.

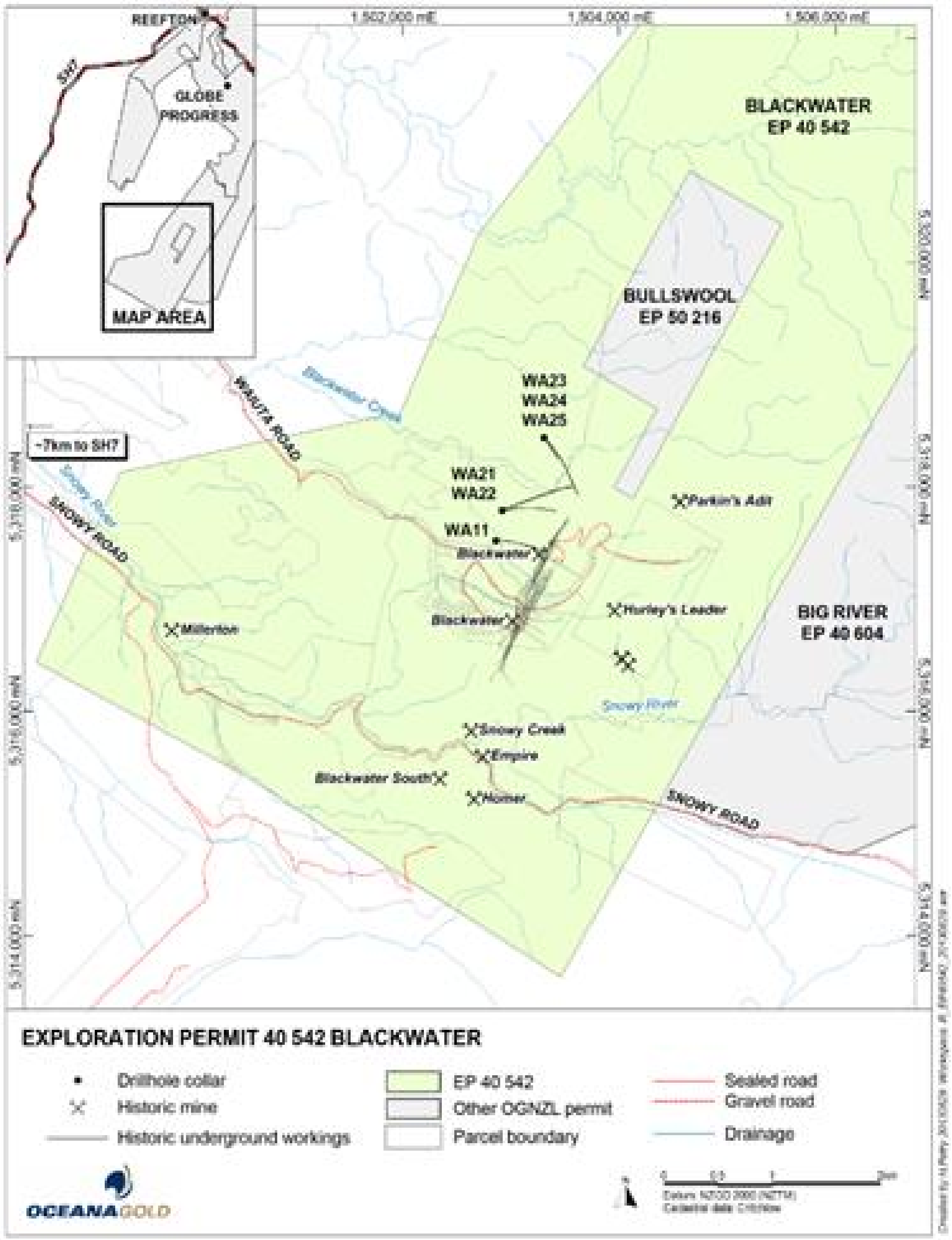
The sites are accessible by travelling south from Reefton on State Highway 7 and turning East onto the Waiuta road. After 9km travel on the Waiuta road, the road turns from tar sealed to gravel. The drill sites are accessible from the gravel road.

Table 6.2 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 6.4 Blackwater 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 6.3. 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 6.3 Blackwater 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)

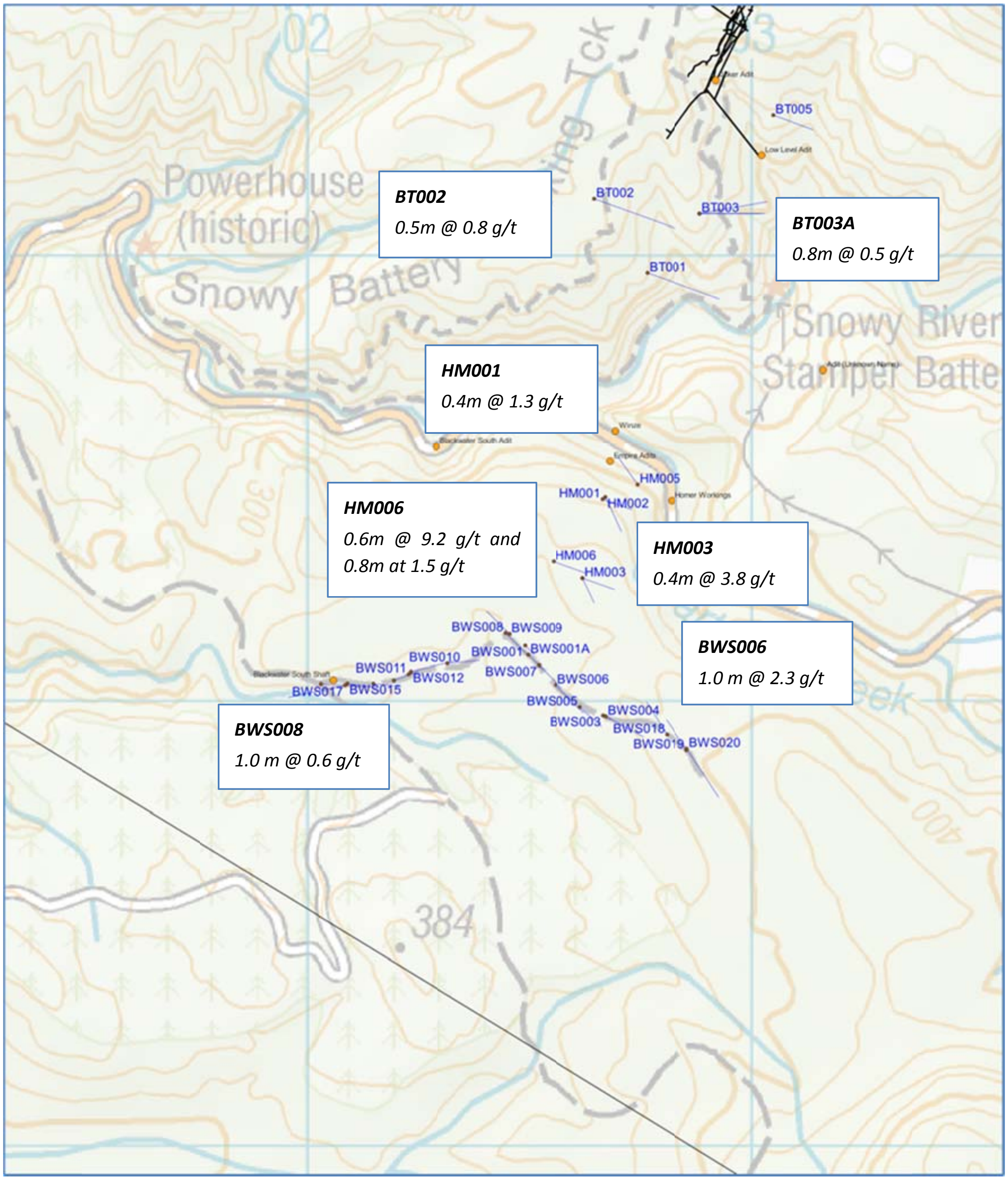
** Indicates the upper intercept in each of the holes WA25 & WA25A interpreted as a fault repetition of the Birthday Reef. (BR) indicates the Birthday Reef intercept.*

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

Drilling programs were completed by OGC testing the area south of the Birthday Reef. Several diamond holes were drilled testing for extensions to the reef and for potential sub parallel veins near the Snowy Battery site. Several zones of quartz veining and weak gold mineralisation were intersected with the best results returned from BT002 (05.m at 0.8 g/t Au) and BT003A (0.8 m at 0.5 g/t Au).

Drilling programs were also completed testing around the Homer workings south of the Snowy River, along strike from the Birthday Reef. Drilling returned several zones of interest with best results from HM006 (0.6m at 9.2 g/t Au and 0.8m at 1.5 g/t Au), HM003 (0.4m at 3.8 g/t Au) and HM001 (0.4m at.3 g/t Au). A fence of RC holes was drilled to the south of Homer, designed to test the stratigraphic position of the Birthday Reef. Results were generally disappointing with the best results from BWS006 (1.0 m at 2.3 g/t Au) and BWS008 (1m at 0.6 g/t Au).

Figure 6.5 Blackwater South – Drilling Results



6.4. Surface Exploration and Chemistry

FML hold interests in two contiguous permits, MP 60473 (Mining Permit with option to purchase from OGC) and EP60473 (Exploration permit held by Tasman Mining, FML's subsidiary). The following section summarizes exploration work that has been completed on this lease package outside of the main Blackwater Mine/Birthday Reef area.

Modern exploration of the project area began in the 1980s when CRA Exploration (CRA) began working in the area. CRA carried out reconnaissance exploration work that included drainage geochemistry, reconnaissance mapping and rock sampling, a photo-geological interpretation, aeromagnetics, and radiometrics.

In the southern part of the EP 60473 small programs of more detailed gridding, soil and rock geochemistry, detailed outcrop mapping, and trenching were completed. No significant gold mineralisation was identified (Lew, 1986; Patterson, 1987; Lawrence, 1988a,b; Agnew & Lew 1989).

CRA's New Zealand exploration assets were acquired by Macraes Mining Company Limited (MMCL). Their work in the 1990s was largely focused on identifying potentially open pit minable gold deposits and on re-development of the Blackwater/Waiuta Mine. Work was largely focused around the Blackwater/Birthday Reef area. Outside of this, exploration work was largely geological and structural mapping along stream sections (Stewart, 1996).

OceanaGold (formerly MMCL) completed the majority of exploration work on the permits between 2007 and 2015. Drilling programs were carried out testing the depth extensions to the Birthday Reef, with smaller programs testing strike extensions to the reef, sub parallel veins, and around the Homer/Blackwater south area. Other work included geological mapping, geochemical sampling, wacker drilling and diamond drilling.

Prospect areas in EP60460 are shown in Figure 6.6. The area now covered by EP 60 460 initially comprised three permits. The southern parts were held under EP 50 216 (Bullswool) and EP 40 705 (Krantz Creek) which we later both combined into a single permit. The northern part was incorporated within EP 40 542 (Blackwater). Work is documented in Gardener, 2012, Hood Hills et al, 2010, Jober and Mustard, 2010, Lotter, 2011, Oceangold, 2014 and Reynolds and James, 2005.

Several geological mapping programs were completed. In June 2010 EP 50216 and the surrounding permits was mapped as part of a larger mapping project covering the southern Reefton goldfield for OceanaGold (Scott, 2010). The Krantz Creek Shear was identified in the eastern part of EP 40705. Additional regional / prospect scale mapping within the creeks was completed in 2012 with a further 41 structural measurements collected. Fifteen geological observations were made in EP 50 216 between December 2012 and December 2015 as part of a programme to better understand the geological setting of the area. A geological mapping and aeromagnetic interpretation program was completed on the northern part of EP60460 in 2014, with more detailed mapping around the Honeys Reef prospect.

Rockchip, channel and float samples were taken during field mapping programs. Sampling was largely focussed on the southern part of the EP60460 around the Bullswool and Krantz Creeks, with more limited sampling in the northern parts of the license. The best results returned was 12.1 g/t Au from a sample of mullock material in Krantz Creek. In the northern part of the license area the only significant assays returned were from Honeys Reef, which were low grade (up to 0.3 g/t Au).

In 2011 several lines of wacker drilling were completed in the Krantz Creek over the interpreted strike of the Krantz Creek Shear. 187 samples were collected of which 4 returned gold grades > 0.05 ppm with numerous other samples returning anomalous arsenic grades up to 757 ppm As. Wacker drilling defined a strong arsenic anomaly associated with the shear zone. Wacker drilling utilises a petrol powered jack-hammer to drive a steel sampling rod into the ground. The sampler and rods are then pulled out of the ground with a hand operated jack. This is a method for penetrating overburden material and obtaining a sample of bedrock or near bedrock.

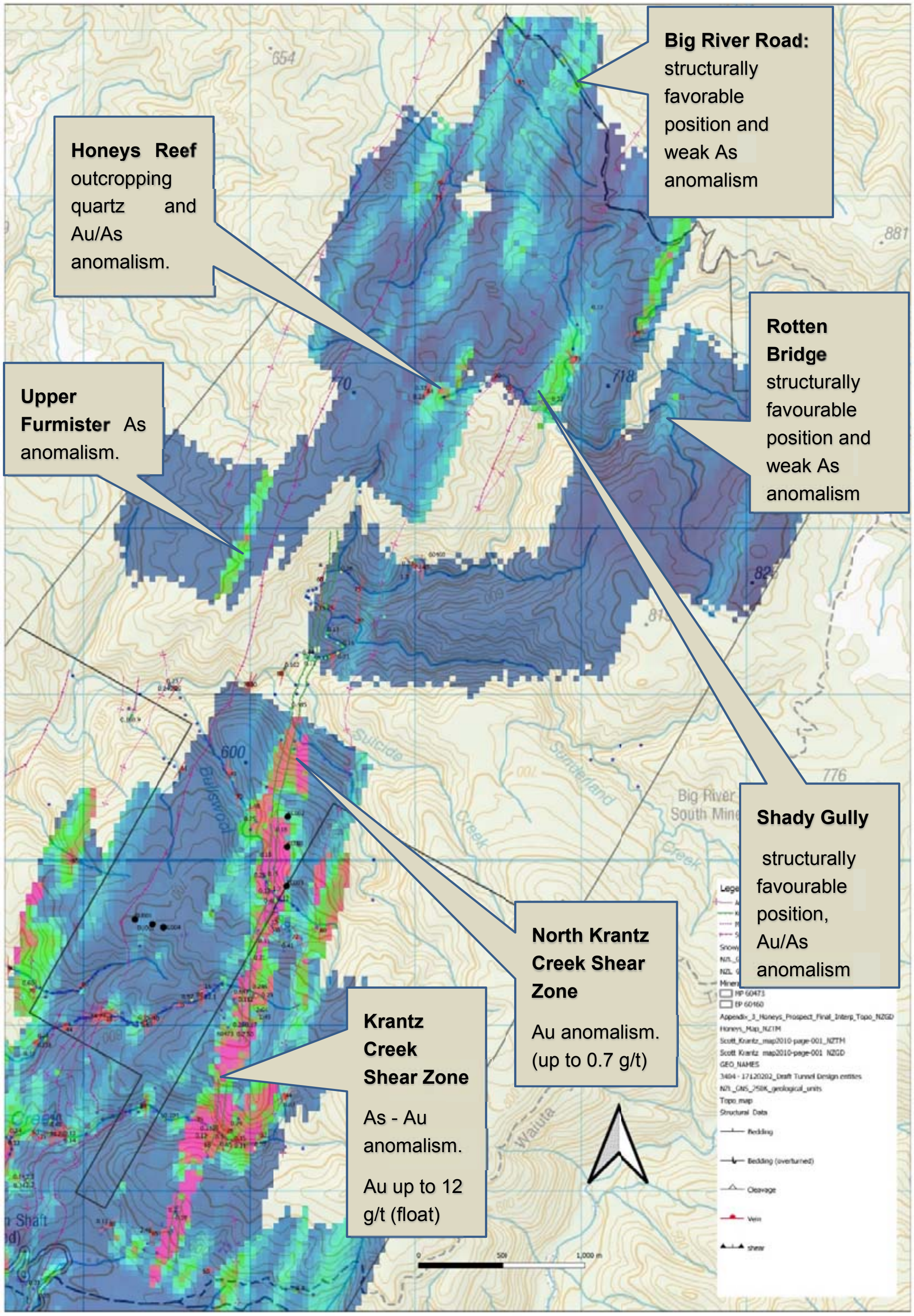
Two diamond drilling programmes were completed in 2012 (Figure 6.7). Three holes (508.2m) were drilled testing the Krantz Creek Shear. KC003 returned the only significant results (>0.5 g/t Au), intersecting 3m at 1.2 g/t Au from 15m from within the weathered Krantz Creek shear zone. A fence of four diamond drill holes (544.3m) were drilled to the west of the Krantz Creek Shear testing a structurally prospective corridor for Blackwater style mineralisation. Although two holes failed to reach target depth the program was interpreted to have tested the target area. No significant results (>0.5 g/t Au) were returned from the drilling program.

Historically, geochemical sampling was not systematically completed over much of the permit area due to the limited outcrop and the thickness of the fluvioglacial gravel cover. Methods such as soil sampling were viewed by previous workers to be of limited effectiveness, as soils developed over gravel overburden were interpreted not to geochemically reflect any potential basement mineralization. Wacker drilling was utilized extensively to obtain bedrock samples through cover. This method is effective, but is time consuming and labour intensive, hindering its effectiveness in covering large areas as a first pass geochemical exploration tool.

However, in more recent times other explorers in the district have had success using soil sampling as a first pass exploration tool. Elements such as arsenic (As) (and potentially gold) may have more mobility from bedrock into the overlying transports rock soil profiles than previously thought. In particular, arsenic may have mobility from basement lithologies into overlying transported cover rocks and soil profiles. Soil samples are much quicker to collect, allowing larger areas to be systematically tested.

In addition, advances in portable XRF technology (PXRF) mean that accurate, low detection limit, multi-element geochemical sample analysis can be easily and quickly obtained. This has significantly reduced the speed and cost of obtaining multi-element geochemical data sets. Arsenic is a useful pathfinder element, which can be accurately detected with a PXRF. Arsenic anomalism varies depending on the deposit style. Narrow quartz-vein hosted ‘Blackwater-style’ mineralisation typically has a more subtle anomalous halo of arsenic (typically several metres at > 50 ppm As). The shear-hosted and more structurally complex ‘Globe-Progress-style’ mineralisation has a much broader and higher-grade arsenic halo due to more extensive arsenopyrite rich alteration (up to 10 of metres >100 ppm As).

Figure 6.6 Plan showing prospect areas and contoured arsenic results (July 2021)



Big River Road: structurally favorable position and weak As anomalism

Honeys Reef outcropping quartz and Au/As anomalism.

Upper Furmister As anomalism.

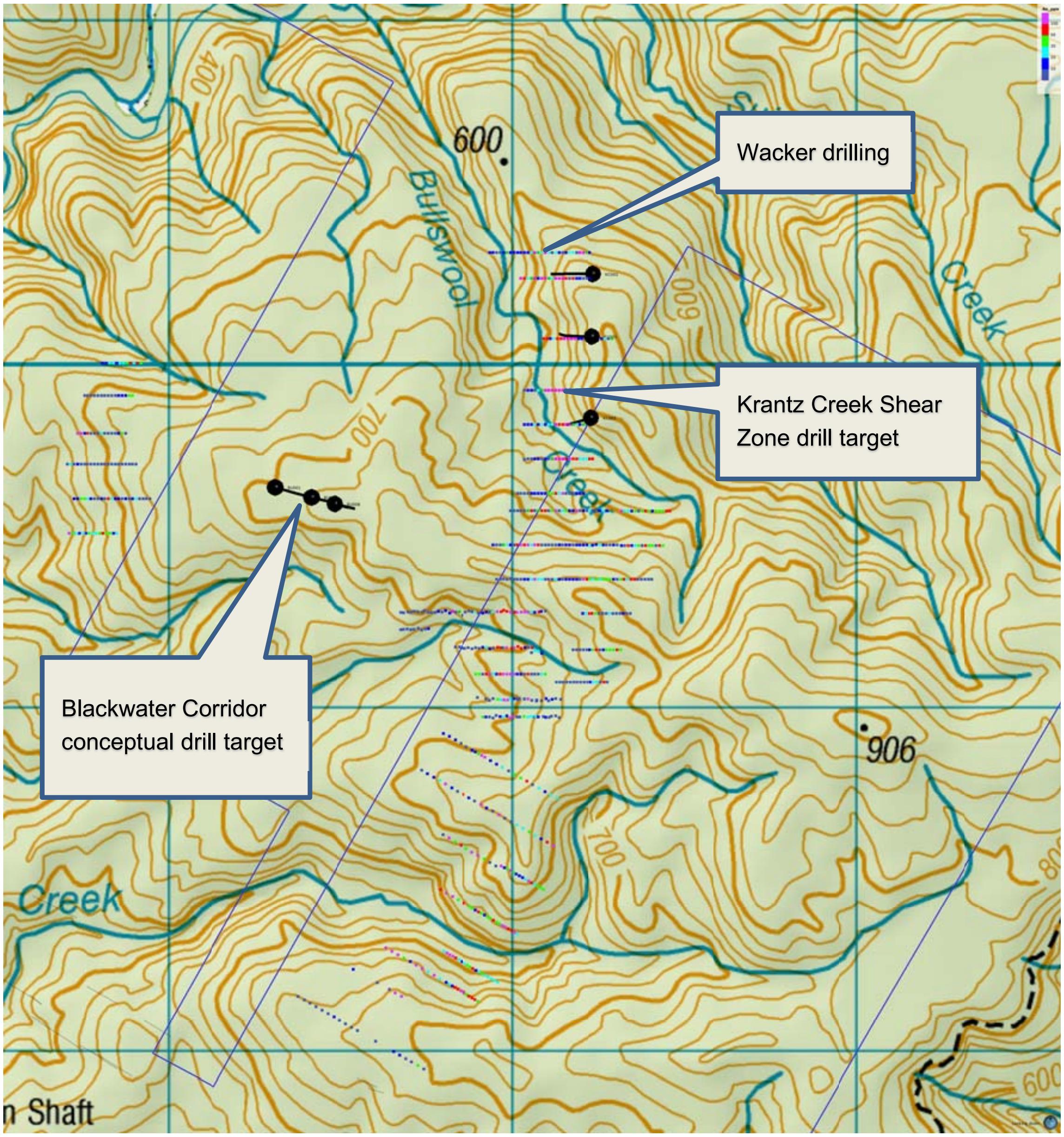
Rotten Bridge structurally favourable position and weak As anomalism

Shady Gully structurally favourable position, Au/As anomalism

North Krantz Creek Shear Zone Au anomalism. (up to 0.7 g/t)

Krantz Creek Shear Zone As - Au anomalism. Au up to 12 g/t (float)

Figure 6.7 EP 60460 – Bullswool/Krantz Creek area



Since 2021, FML's exploration work program has focused on reconnaissance soil sampling and mapping throughout the lease package. The aim of this is to systematically obtain geochemical data sets that can assist in identifying targets for future drilling. Soil sampling lines are planned on a nominal 400 metre line spacing, with infill to a tighter spacing based on results. Systematic 'gridding' of the areas also provides the opportunity to identify areas of outcrop and mineralization that may not have been identified by previous workers.

Federations work program to date has focused on:

- Northern part of EP 60460. Wide space traverses have identified several areas of weak As anomalism (**Error! Reference source not found.**6). Further infill and geological mapping is planned.

- Northern Krantz Creek Shear Zone (Figure 6.8 and Figure 6.9) A series of soil lines were run over the interpreted northern extension of the Krantz Creek Shear Zone. Soil sampling has identified a significant arsenic in soils anomaly and weak gold anomaly. Further lines to the north and infill lines are planned

- Southern Krantz Creek Shear Zone (Figure 6.10). A single soil line testing the interpreted southern position of the shear zone returned anomalous arsenic results. Infill to the north and south is planned.

- Blackwater South/Homer Area (Figure 6.10). A strong arsenic in soils anomaly has been identified with elevated gold values in some areas, Further infill lines are planned along with geological mapping

- Snowy Portal area (Figure 6.10). Soil lines returned low arsenic results. No further work planned.

Federation has further soil sampling and geological mapping programs planned for the 2022-23 summer field season. A small wacker drilling program will also be completed in EP 60460. Planned soil lines are shown in Figure 6.9 and Figure 6.10

Figure 6.8 EP 60460 and MP 60473 – Historic drilling, tunnels, lease boundaries and 2021/22 soil sampling arsenic results

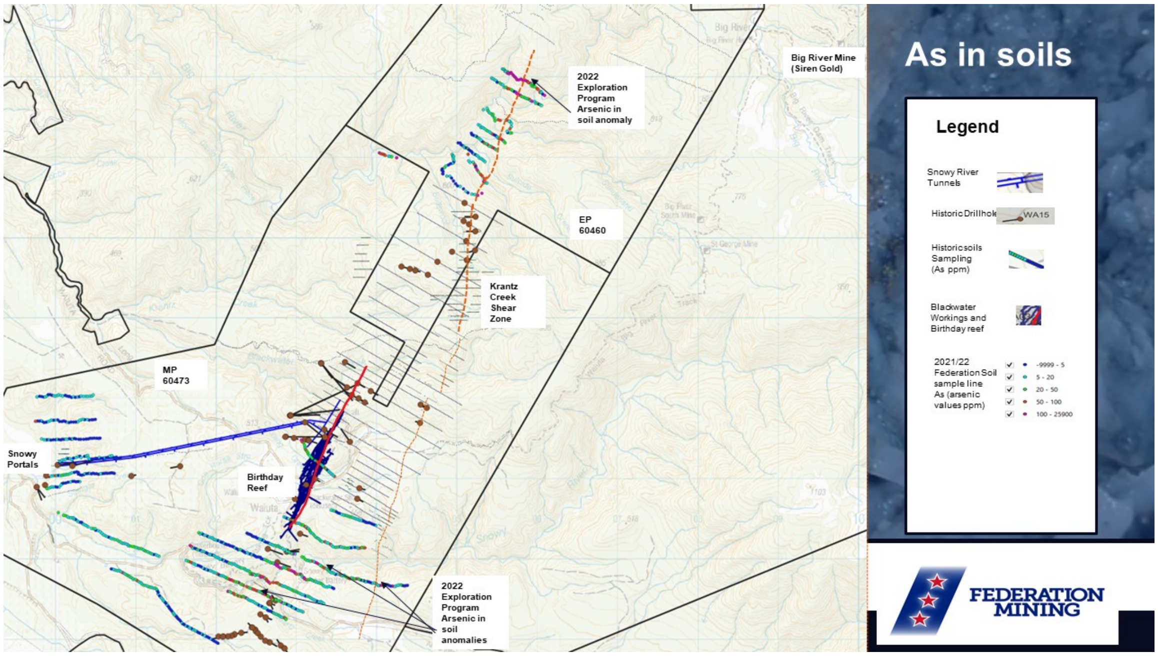


Figure 6.9 EP 60460 – Plan showing 2021/22 soil sample arsenic results, and planned soil lines

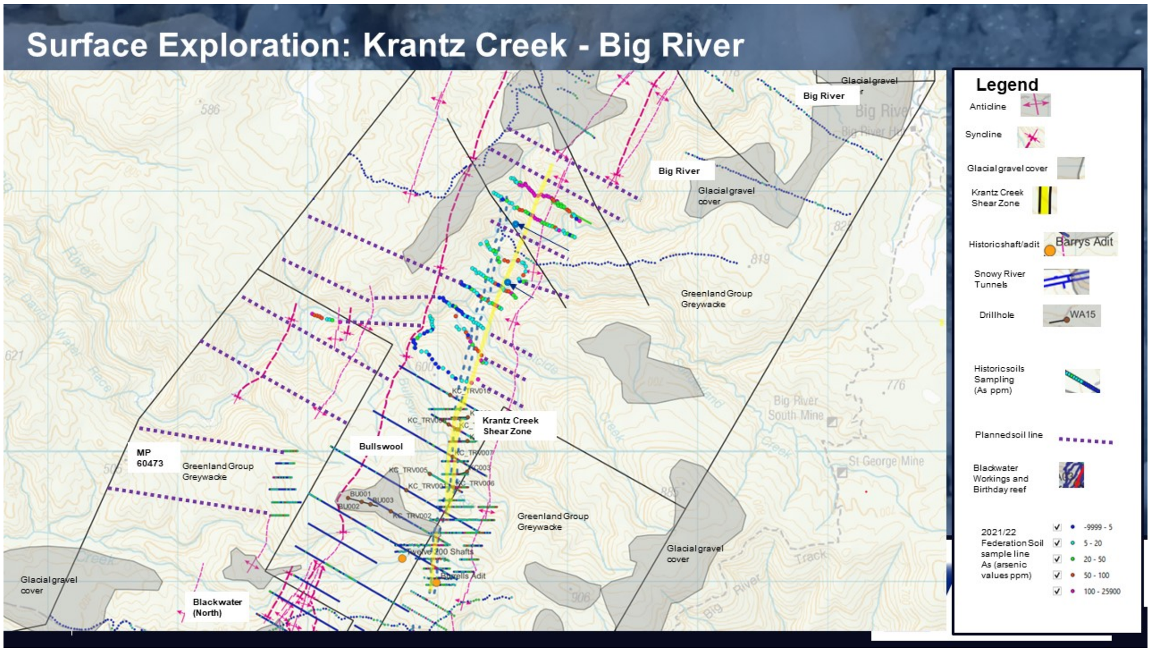
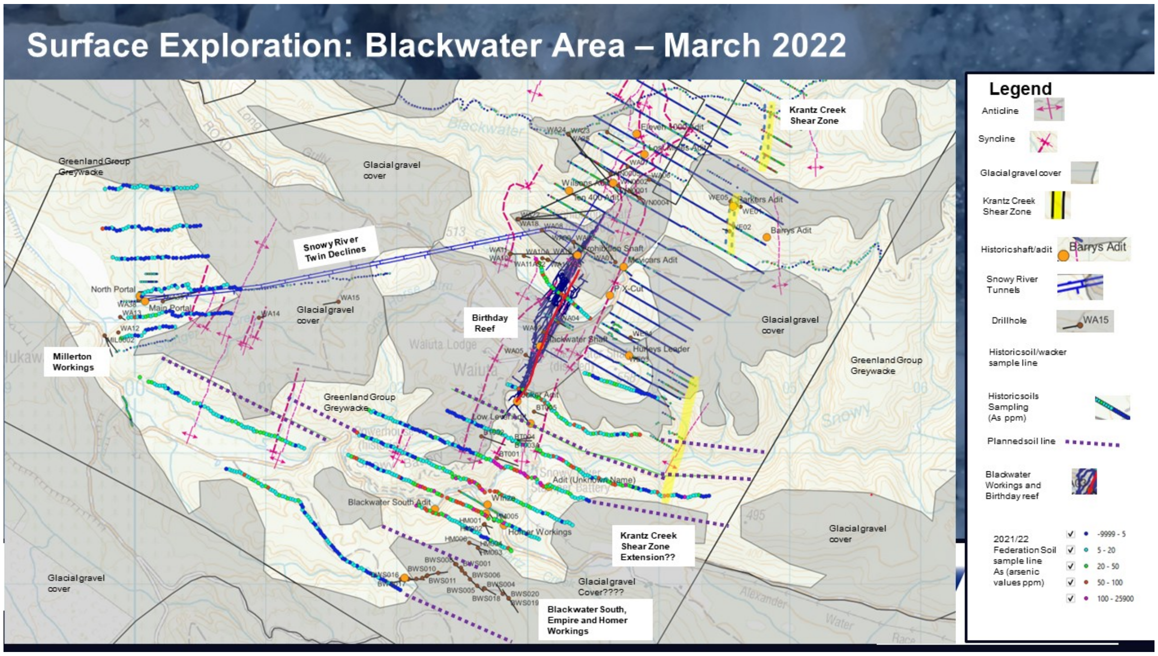


Figure 6.10 MP60473 – Plan showing 2021/22 soil sample arsenic results, and planned soil lines



7 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section summarises sample data used to inform the 2013 Blackwater Mineral Resource estimate, which was reported in accordance with the JORC Code (2012). This formed the basis of

OGCs 2014 PEA for the Blackwater project. The Competent Person for the MRE was J.G. Moore, Chief Geologist, a full-time employee of OceanaGold (New Zealand) Limited at the time of writing. Mr. Moore was of the opinion that the procedures followed by OGC for sample preparation, security and analytical measures followed accepted industry standards, and that the database was suitable for Resource Estimation.

7.1. Logging and sampling

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 7.1.

Table 7.1 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 7.2 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 7.3, which shows that core recovery of the Reef ore zone was in most cases 100%.

Table 7.2 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 7.3 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.

7.2. 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 7.4.

Table 7.4 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

7.3. 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 7.5. 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 analysed 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 analysed 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 **Error! Reference source not found.**.

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.

Equation 7.1 Calculation of Au Total with Screen Fire Assay (1kg)

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

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

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 7.5 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

7.4. 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.

7.5. 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. The QA/QC results for the respective laboratories are tabulated in Table 7.6 and

Table 7.7.

Table 7.6 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 7.7 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

7.6. 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.

7.7. 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 OGC.

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.

7.8. Database

OGC stored all drill hole assay, survey and geology data is stored in an Acquire database. The geological wireframes and drill hole locations use truncated NZMG coordinates whereby the first two digits are removed for both eastings and northings.

The OGC database has been migrated into an Access Database by FML. Data has been translated into the NZTM2000 Transverse Mercator projection. This is based on the NZGD2000 datum, uses the GRS80 reference ellipsoid, and is the standard projection within mainland New Zealand.

8 DATA VERIFICATION

The Competent Person for the Blackwater 2013 MRE, Mr J.G. Moore, reviewed the existing data of all available past and recent reports. 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. As outline in Section XX the Competent Person for the MRE, Mr JG Moore, was of the opinion 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, the basis for this ITR, 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 (JORC 2012). The Competent Person for the MRE, Mr JG Moore, believed 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.

9 MINERAL RESOURCE ESTIMATES

The most recent Mineral Resource estimate for the Blackwater Mine was initially 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). This was reported in accordance with the JORC 2004 Code. This was subsequently updated and reported in accordance with the JORC Code (2012) in the OGC’s 2013 end-of-year Resource and Reserve statement, dated 31st December 2013 (released 26th March 2014).

For the purposes of this ITR, the Resource statement as at 31st December 2013, forms the basis for this report. The Competent Person for the report was J.G. Moore, Chief Geologist, a full-time employee of OceanaGold (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. JORC Table 1 disclosures in accordance with clause 5 of the JORC (2012) Code are included in Appendix 1.

The Blackwater resource estimate presented in this section is based on a combination of deep diamond drilling and historical mine sample data. The deep diamond drilling demonstrates continuity of the vein at depth, but drilling hole spacing is too broad to use in any meaningful geostatistical estimation.

OGC's methodology was to utilise the historic mine records to estimate the likely in situ grade/width of the reef, and then apply this to the area tested by deep diamond drilling (polygonal estimate). To support this, they undertook extensive studies to validate the thickness, continuity and tenor of gold mineralisation within the historic Blackwater mine. This work is summarised in the following sections.

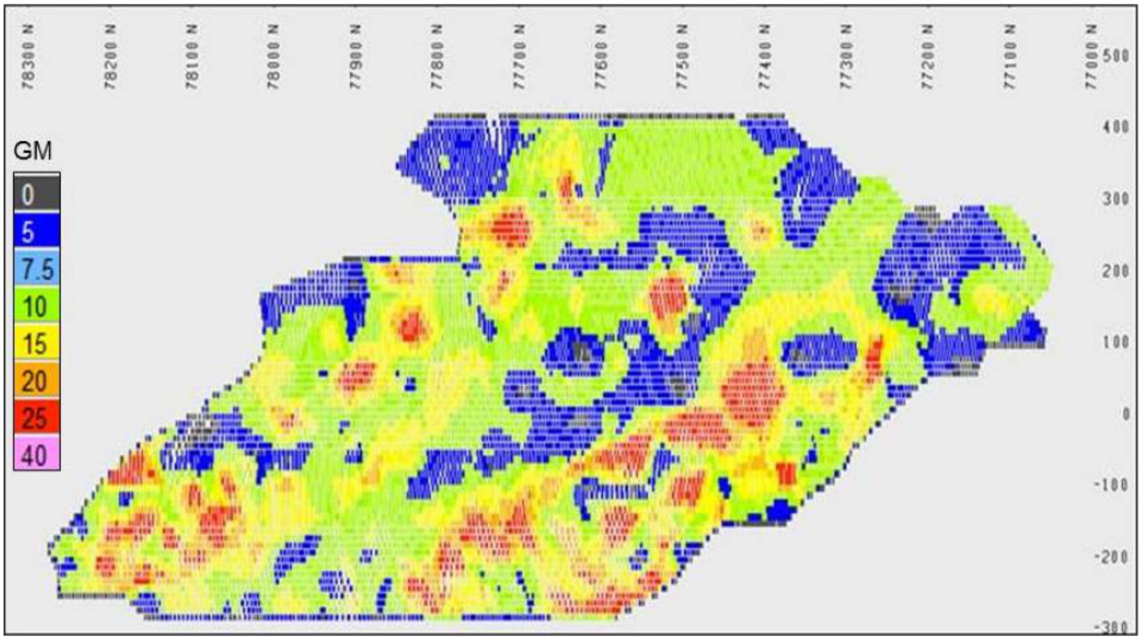
9.1. 3D Block Model of Historically Mined Reef

In 2009, OGC 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 9.1, 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 9.1 Blackwater Mine Long Section showing Block Modelled Grade-Width (g-m)



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.

9.2. Alternative Estimation of 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. 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 QA-QC 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.

9.3. Mine 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[1] applied (≈36.5 g/t Au).
- 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 9.1.

Table 9.1 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.

9.4. 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 9.2.

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 9.2 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.*

9.5. Alternative Estimate Based On Face Sample Data

The face sample data 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.

9.6. 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. 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 9.3 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 9.3 Assumptions

Item	Value	Source
Tonnes milled	1,582,400t	Historical records
Recovered gold	740,400 oz	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 9.4 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 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.

9.7. Grades 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 9.5 presents the sensitivity of back-calculated head grade (i.e. payable reef grade) to mining dilution and metallurgical recovery. 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 be higher rather than lower than the estimates presented above. Payability assumptions also affect the back-calculated grades.

Table 9.5 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.

9.8. Mineral Resource Estimate – OGC 2013

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 historical 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.

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.

Due to 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. Furthermore, conditional simulations were conducted to estimate anticipated variability in reef grade, width and contained gold.

Table 9.6 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. 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.

Figure 9.2 Snowy River/Blackwater Mine Long Section

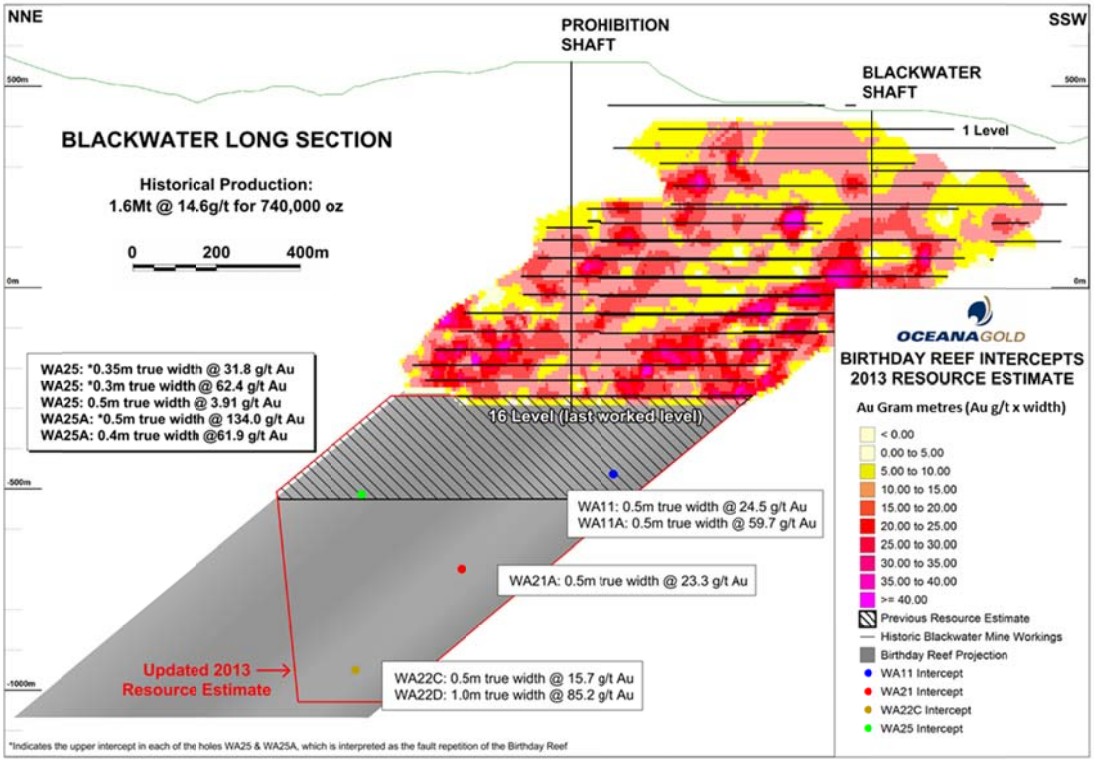


Figure 9.2 shows Au gram-metres from historical workings, drill intercept locations with estimated true widths, gold assay results and the limits of the updated resource estimate.

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 red line in the long section in Figure 9.2). 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 fall within the range of historically mined thicknesses and grades, but represent a small number of exploration drilling intercepts. 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 Snowy River estimate is classified as an Inferred Mineral Resource and is shown in Table 9.7. 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 9.7 Blackwater Mineral Resource 31/12/2013 (Polygonal Estimate)

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

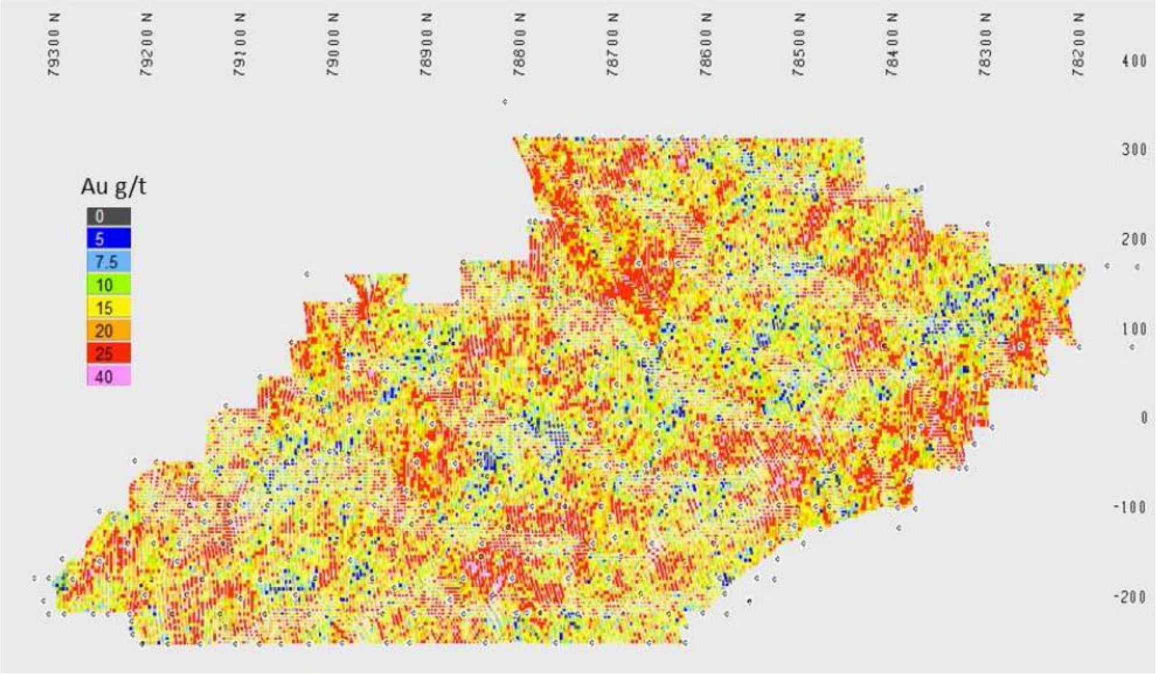
9.9. Control Uncertainty and Risk

The kriged model 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 9.3 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 9.4).

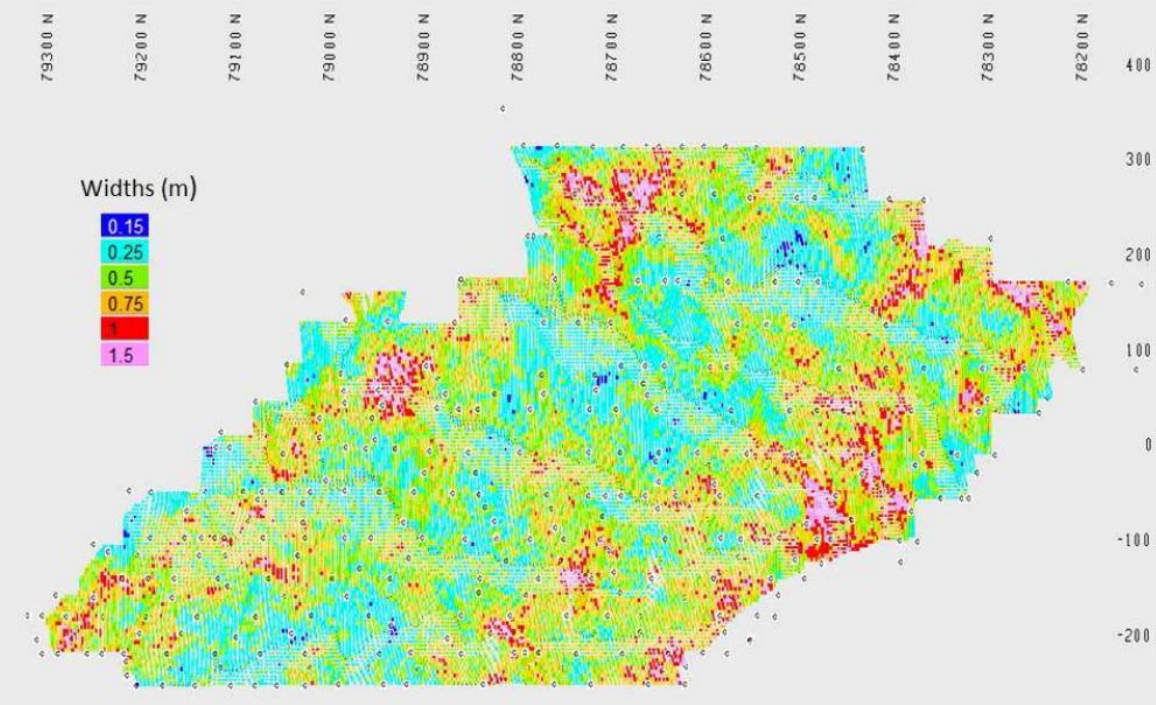
Drill core logging and assaying demonstrate that gold mineralisation is restricted to the reef and so in cross-sectional terms (i.e. reef margins), the mineralisation would be amenable to visual control.

Figure 9.3 Single Conditional Simulation of Gold Grade



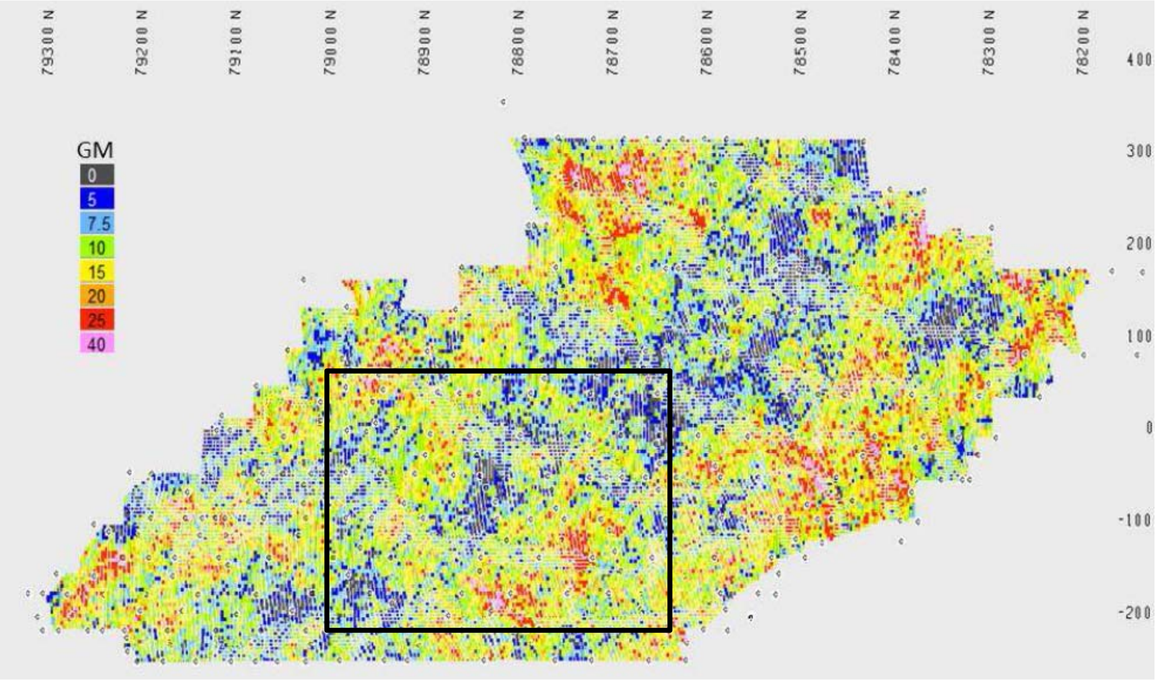
The simulation of reef width distribution (Figure 9.4) shows more structure than the simulated distribution of grade. This suggests that the application of a minimum reef width threshold might be used to for grade control.

Figure 9.4 Single Conditional Simulation of Reef Width



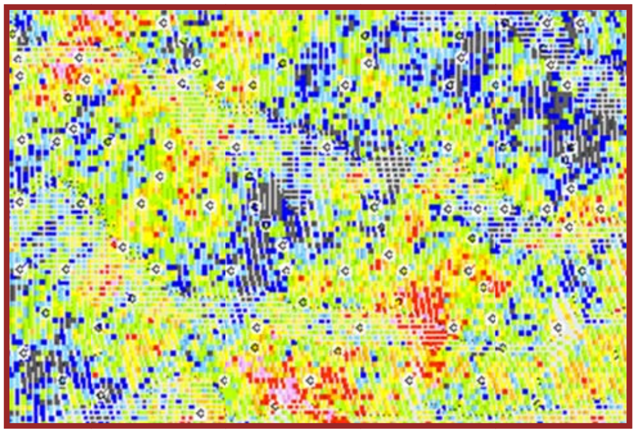
The distribution of contained gold (grade x width) was not simulated, but rather calculated directly from the product of simulated grades and widths; reef grades and widths were simulated independently. Contained gold was calculated from the product of these for each node. The use of independently simulated 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 9.5.

Figure 9.5 Contained Gold derived from Conditional Simulations of Grade-Width (g-m)



An exploded view of the outlined area is shown in Figure 9.6. 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 9.6 Exploded View of Figure 9.5



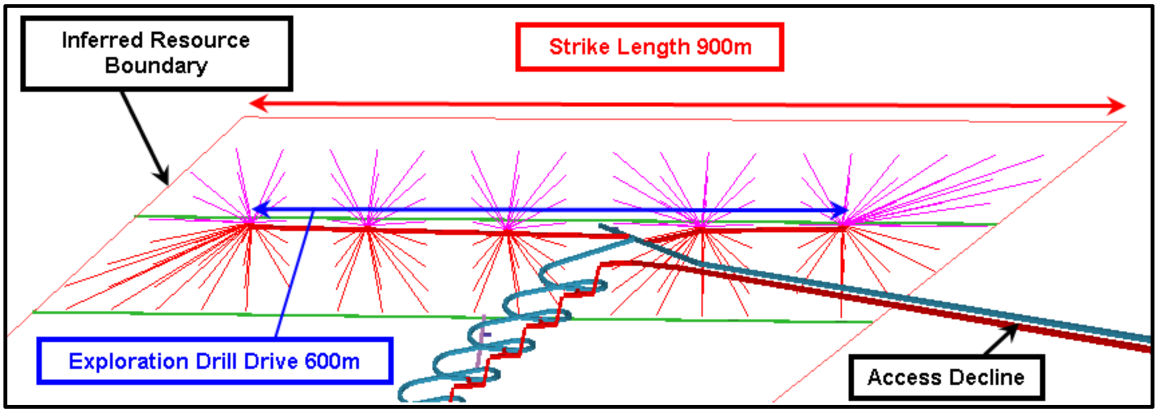
9.10. Resource Definition Strategies

OceanaGold 2014 PEA

OGC planned an exploration drive at around the -360m level, 80m off the reef plane, to enable resource definition drilling of the reef. The drive was positioned to enable holes to be drilled up dip as well as down dip. It was designed to provide the drilling platform for the initial 25m x 50m spaced drilling covering the first 200m vertically below 16 Level (Figure 9.7), proving up sufficient resource for the first 4 years of mine life at the planned production rate. Drilling the first 100m high panel was planned to be completed before drilling commenced into the second panel.

OGC 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.

Figure 9.7 OGC Planned Initial Resource Definition Drilling



As the decline advances, OGC envisaged 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 would 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.

OGC noted that closely spaced grade control drilling would 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.

Federation Mining 2023 Planned Drilling Program

The decline allows for resource drilling and future deeper drilling. . As of mid-August 2022, 2.45km of the planned 3.3km decline access had been completed (Figure 9.8 and Figure 9.9).

The twin access tunnels from Snowy River are at a 1 in 7 gradient, which means the drill drive is positioned at the -280m elevation (higher than the original OGL drill drive position). The drive has been moved to a 120m to 140m stand-off from the Birthday Reef to allow deeper drilling coverage (previously 80 metres).

Drilling will initially begin in March 2023, with the drilling of several shallow angle holes targeted 50 metres below the historic workings from Crosscut 17. These are designed to obtain geological, geotechnical and hydrogeological data and validate the position of the vein and the historic workings.

Development of the tunnels will continue, with drilling from the planned drill drive expected to commence in April 2023. Three underground diamond rigs will be utilised to drill the 16,000 metre resource definition program. This program is scheduled to be completed by September 2023. The planned resource drilling program will collect geological, geotechnical, hydrogeological, and metallurgical data for feasibility study work, and form the basis of an updated Mineral Resource.

The resource definition program is focussed on drilling the central panel of the reef (600m strike by 300m down dip) on 50m x 50m spaced centres (Figure 9.10). This drilling, combined with the demonstrated geological continuity of the vein and grade from historic mining, will form the basis of an Indicated Mineral Resource. Drilling is designed to upgrade 200K to 250K oz Au to the Indicated Resource Category. Infill of the northern and southern extents of the vein is planned from future decline and mine access drives. This drilling is planned for 2024.

The drill drive has been designed to allow drilling to 300 metres below the historic workings (Figure 9.10). Deeper areas can be targeted with core drilling from Cross Cut17 (southern part) or from a planned extension and crosscut from the drill drive (northern part) (Figure 9.11).

Figure 9.8 Schematic Cross Section of the Snowy River Mine declines, planned drilling and historic workings looking north

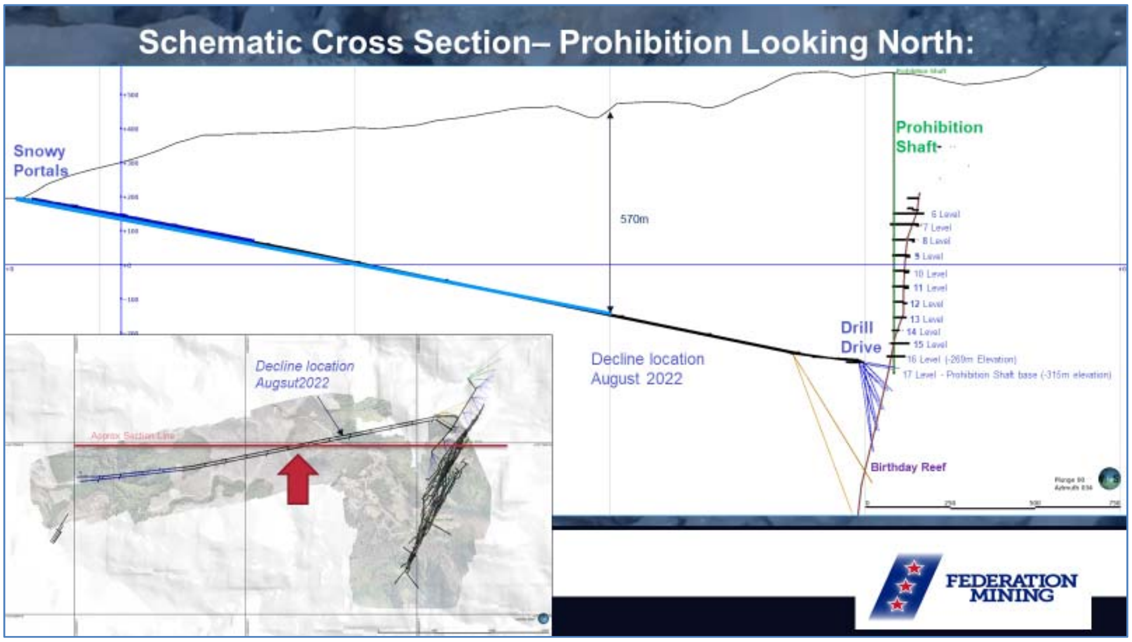
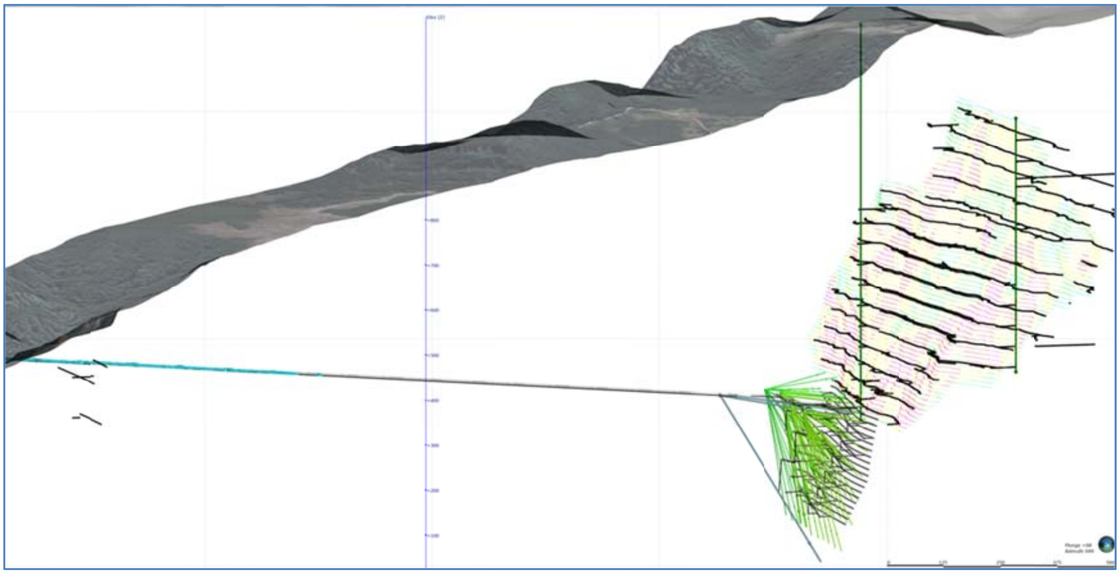
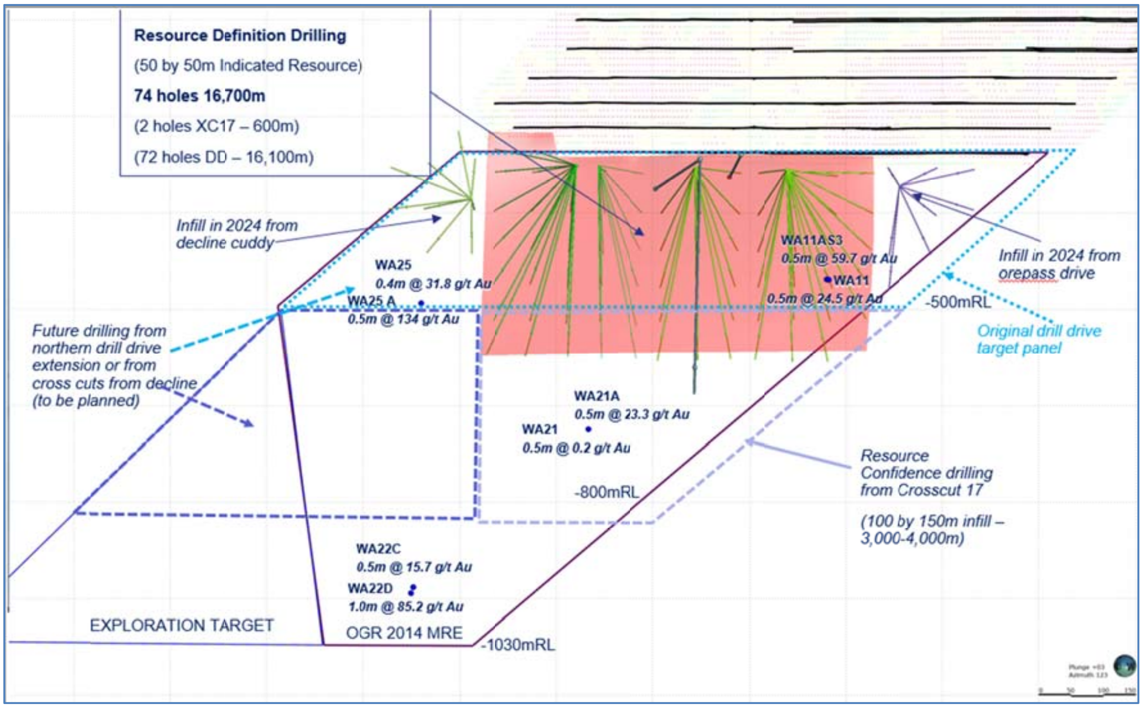


Figure 9.9 FML Planned Resource Definition Drilling



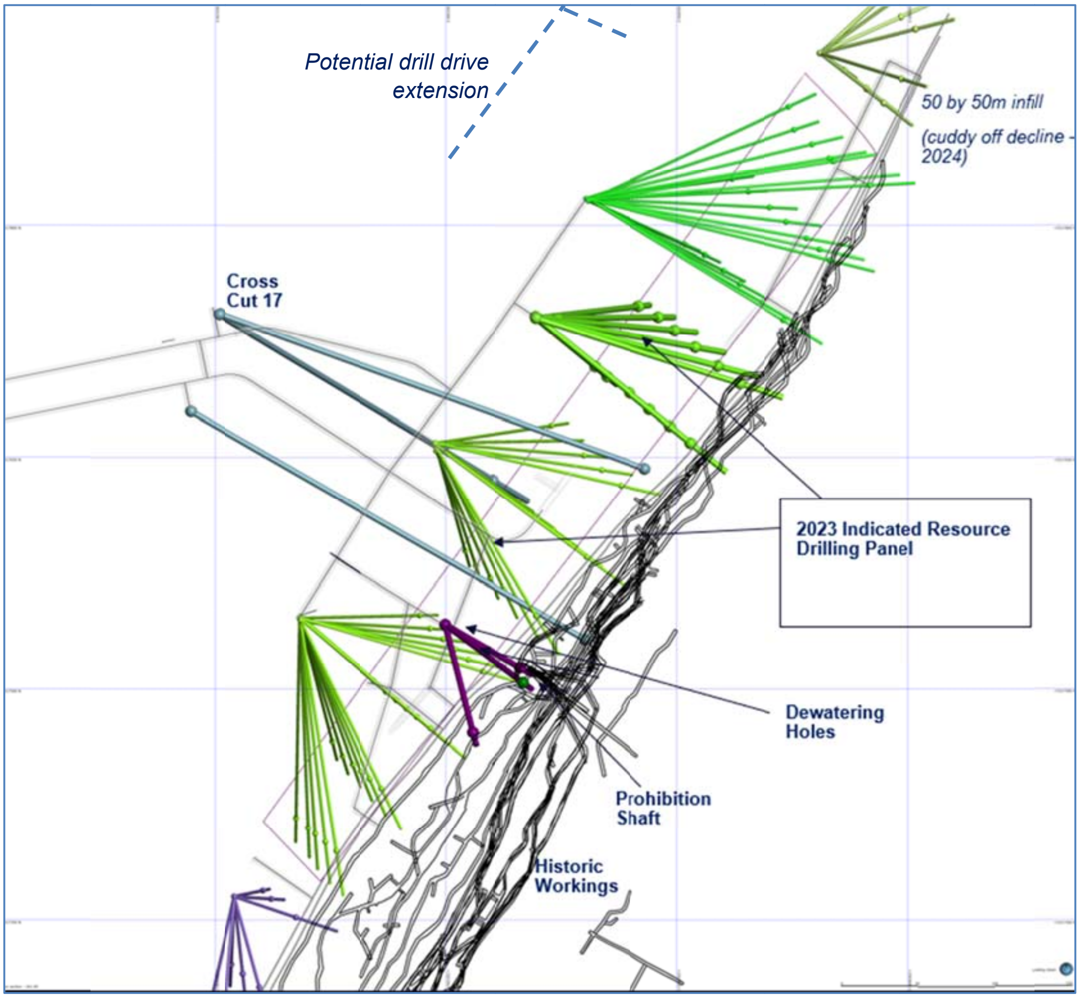
Oblique cross section view looking northeast, showing access tunnels, planned drillhole traces, historic workings and OGL historical mined grade block model.

Figure 9.10 FML Planned Resource Definition Drilling



Long section looking east. Pale pink is target panel for 2023 Resource definition drilling. Target areas for future drilling shown.

Figure 9.11 FML Plan view of 2023 drilling program and historic workings



9.11. Grade Control Strategies

OceanaGold 2014 PEA

OGC planned to mine using hand-held airleg mining methods, with mining on 100m by 60m (strike) airleg panels. For resource definition drilling, OGC were planning 25 x 25m and 25m x 50m spaced drilling to keep the reef positional uncertainty to within 10m. Level spacings were 100m. They envisaged that in some instances short segments of reef might be fault-offset but not resolved by the resource drilling that would require additional infill drilling.

OGC noted that 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.

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.

Federation Mining Proposed Grade Control

FML's planned mine design is based upon mechanised uphole retreat longhole stoping with paste backfill. Development is planned on 13.5m spaced levels. Ore driving on reef is planned with drive size of 2.5m (width) by 3.5m (height), which gives a 10-metre separation between levels. Individual stopes are planned to be 10m by 10m, which will be then paste filled to allow neighbouring pillar extraction.

FML plan to geologically map and sample each face in the ore drives for grade control. FML is currently utilising iPad based Rockmapper software for mapping and sampling in the decline and will use it for ore development. The software allows for mapping of geological features onto three dimensionally registered face photos and image scans. This data can then be easily used to build grade control block models for stope design and extraction. Leapfrog Geo will be used for building 3D models of the vein, and Leapfrog Edge for gold grade modelling and block estimation. Mine design work is currently being done using Deswik, which interfaces with Leapfrog.

Given the generally high-grade nature of the vein it is likely that most development headings on the reef will be sent as ore. Of note is that OGL in their PEA have suggested that vein thickness might be a possible cut-off criteria. Further work on this is needed.

Resource definition drilling is planned on 50m by 50m spaced centres, with infill to 25m spaced centres as needed. It is envisaged that this density of drilling, coupled with systematic geological mapping and sampling of the reef development will generally be sufficient for ore development and grade control. However, it is likely that there will be some more structurally complex or faulted areas that may require additional information. FML envisage utilising a small portable core rig, capable of drill short holes (up to 60m) to infill these areas, either from the ore drives, or from nearby hanging wall access drives.

Historically the reef was offset by several faults. Geological mapping on the ore development levels, coupled with drilling data, will assist in identifying fault location and displacements. These faults and will be modelled in Leapfrog, and can then be used as predictive tools for the development of new levels.

It is anticipated that as the mine reaches steady state production there will be constant work for two drill rigs. This would entail a larger rig drilling 140m to 500m holes for grade control, resource definition and exploration, and a smaller mobile rig drilling shorter (30m to 80m) infill holes in areas of complexity.

9.12. Risks

The Birthday Reef 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.

OGC noted the following risks in their PEA assessment.

- 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 position 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. FML notes that this risk will be somewhat mitigated by the planned 13.5m level spacing, which allows a lot more on-reef development and the ability geologically map the reef and potential offsetting faults. These can then be modelled and used predictively on the next levels. 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.

10 MINE DESIGN

Mining One was engaged by FML to assist in assessing mining methods, schedules, and costs for the Snowy River Gold Project situated on the West Coast of the South Island of New Zealand. The PEA was based on the OGC proposed air-leg resue mining strategy. The Project requires underground exploration drilling to increase confidence in the resource and requires a 3.3 km twin decline to establish access to the orebody.

The objective of the mining option study by FML is to evaluate an alternative strategy of mining the Snowy River deposit (Birthday Reef), namely, mechanised uphole retreat longhole stoping with paste backfill.

The study evaluates and provides the following:

- An outline of the revised mining method;
- Direct comparison of the two proposed mining strategies from the initial decline development and exploration drilling to production;
- A mining schedule that would support consistent production of 300 ktpa based on industry productivities for narrow vein mining equipment
- An estimate of capital, operating and processing costs;
- A conceptual cash flow model;
- Comparison of the potential value of the two proposed mining methods;
- A review of surface works and planning completed to date;
- Outline of permits and legal agreements between OGC and FML.

Table 10.1 Key parameters comparison

	Unit	PEA	MiningOne	FML Analysis
In situ resource grade	g/t	23	23	23
Au Price	US$/oz	1300	1300	1650
Mine Life	Yrs	10	15	10
Pre-production period	Yrs	2.5	3	3
Ore production rate	tpa	120,000	300,000	300,000
Head grade	g/t	15.8	7.9	7.9
Process recovery	%	96	96	96
Ore process method		Gravity/Float/RIS	Gravity/Float/CIP	Gravity/Float/CIP
Product		Doré	Doré	Doré
Au recovered	koz	570	987	699
CAPEX	US$	154	138	137

OPEX	US$/t ore	154	86	155
Process Cost	US$/t ore	42	25	29
NPV	US$M	193 (pre-tax @ 5%)	207 (pre-tax @ 8%)	258 (post-tax @ 5%)
IRR	%	29	32	45

Note: CIP (carbon in pulp) and RIS (resin in slurry)

10.1. Geotechnical Assessment

Introduction

Geotechnical data for the project was very limited before FML began development of the access declines in late 2020. Earlier assessment work was largely based on the small data sets of drilling data from the project, combined with surface geological mapping. Most drilling was focused around the Birthday Reef. Two holes were drilled along the decline path by OGL, and two core holes were drilled at the portal and from Stockpile 1 along the decline path by FML. Little geotechnical data is available from the Waiuta mine as very little geological mapping was undertaken.

The geotechnical assessment below is based largely on the small dataset of historic core logging, core photographs and geological mapping. Where appropriate this has been updated based on new data collected by FML in the over 5,000 metres of tunnel development that have been completed to date.

Geotechnical Data Review

AMC was engaged by OGC in March 2014 to review the then 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.

In 2018 Mining One Consultants was engaged by FML to review the previous studies by Mining Plus, KRA and AMC, and to propose revisions to the previously proposed mining method.

Tunneling began in late 2020, and since then several additional reviews and studies have been undertaken on the project. P Keall and Associates (Mining Geotechnical Engineer) completed the following reports and studies:

- Evaluation of Shotcrete for Primary Support
- Ground Support Regime review
- Snowy Gold Project Site visit Jan 2021
- Snowy Gold Project Site visit Feb 2021
- Snowy Gold Project Site visit March 2021
- Snowy Gold Project Site visit May 2021

- Snowy Gold Project Site visit June 2021

From September 2021 the geotechnical consultants SCT have been regularly visiting the project. Reports and studies completed include:

- Snowy River Mine Geotechnical Site Inspection October 2021
- Snowy River Mine Geotechnical Site Inspection 27 January 2022
- Snowy River Mine Geotechnical Site Inspection 10 March 2022
- Snowy River Mine Geotechnical Site Inspection 29 April 2022
- FED5382a_Report_Review of Snowy River Mine Pillar Design for Main Declines
- FED5382B_Report_Snowy River Mine Ground Support Design Assessment

In 2021 a student dissertation project title as part of a Professional Masters in Engineering Geology was completed by Declan Pearson. The title was 'Geotechnical Characterization of Greenland Group Bedrock in the Snowy River Mine Tunnels, Waiuta'. The work focused on geologically/geotechnically mapping several areas of the tunnel between 490m and 940m chainage, with UCS and point load tests on representative samples.

During excavation, FED have been regularly recording geological and geotechnical information by mapping development faces and completing campaigns of backs mapping of the tunnels. An extensive database of geological and geotechnical information has now been compiled.

For the first 1.2 km of the tunnels, mapping data was recorded onto face logging sheets to collect geological and geotechnical information. Faces were also photographed where possible, and images georeferenced for visualisation in Leapfrog. From 1.2km chainage (January 2022), face mapping has utilised RockMapper^{TM}. This is an iPad cloud-based application for photographing and mapping underground. The software also can generate 3D Lidar scans of development. The software allows geological/geotechnical information to be collected in the field, and then easily imported into Leapfrog for visualisation and interpretation.

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).

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 0.68m. 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).

In-situ stress environment

The stress field at Blackwater has not been determined. Blackwater is situated close to the Alpine Fault (30km) 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. 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.

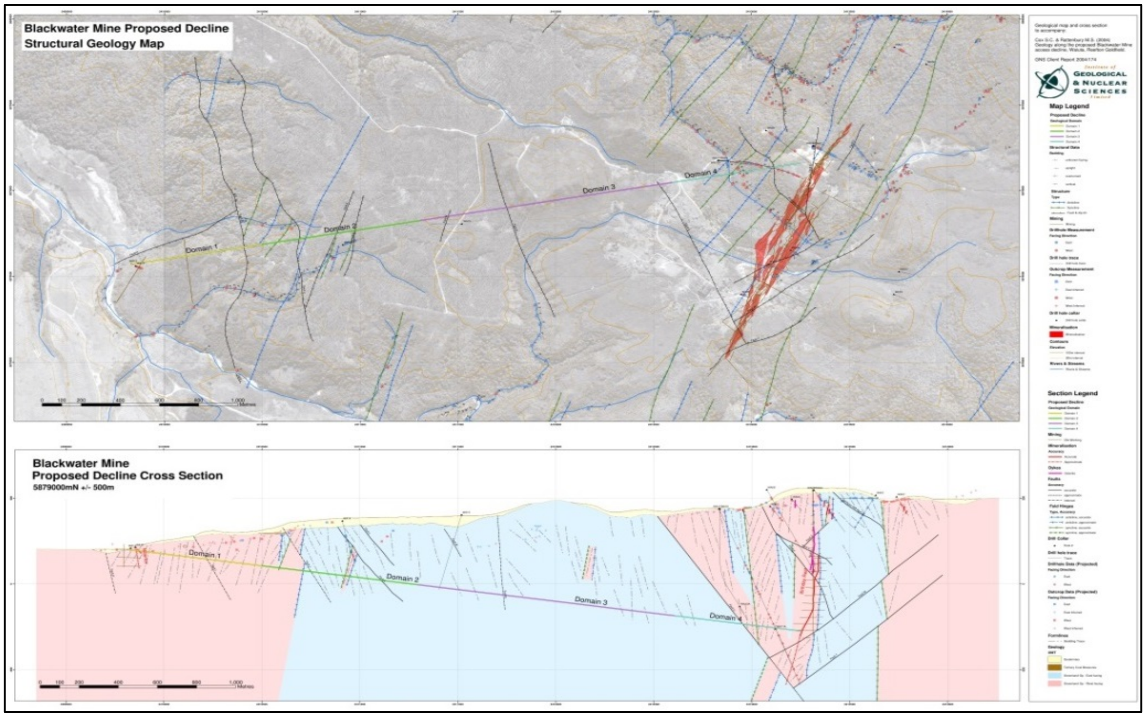
Decline Assessment

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 10.1.

Figure 10.1 Geological Map and Cross Section (Looking North) for Proposed Access Decline



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 OGC data.

Mining Plus observed that the drill core 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.

Mining Plus noted that given the lack of directly applicable geotechnical information, only general assessments of ground conditions and implied ground support requirements could be made at that point.

Mining Plus and Golders studies utilise the Q-system for rock mass classification. This 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. Based on Golders 2004 field investigation campaign ground conditions in the tunnel were expected to lie within three Q ranges shown in Table 10.2 Expected Q Index for Decline. These ranges were used in later assessments of anticipated d ground conditions in the tunnels and recommended ground support regimes.

Table 10.2 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"

Decline development began in late 2020. Since then, FML routinely geologically and geotechnically maps decline faces as the development advances. This includes a visual assessment of Q values using an app developed by the NGI (Norwegian Geotechnical Institute). This gives minimum and maximum Q values for a face based on assessment of rock mass, rock quality, defect type and spacing, stress and water. Q values for the tunnel from Crosscut 5 to Cross Cut 10 are shown in Figure 10.2 and Figure 10.3. Q values intersected in the tunnels are similar to Mining Plus and Golder's earlier assessment work.

Figure 10.2 Q values from Main Decline Crosscut 5 to Cross 10; rockmass classification

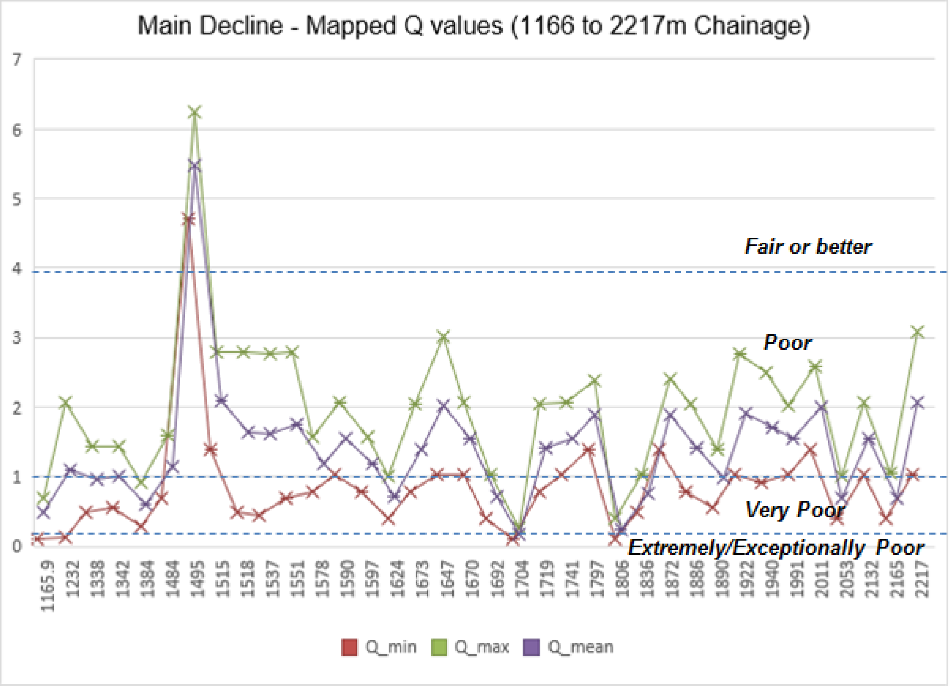
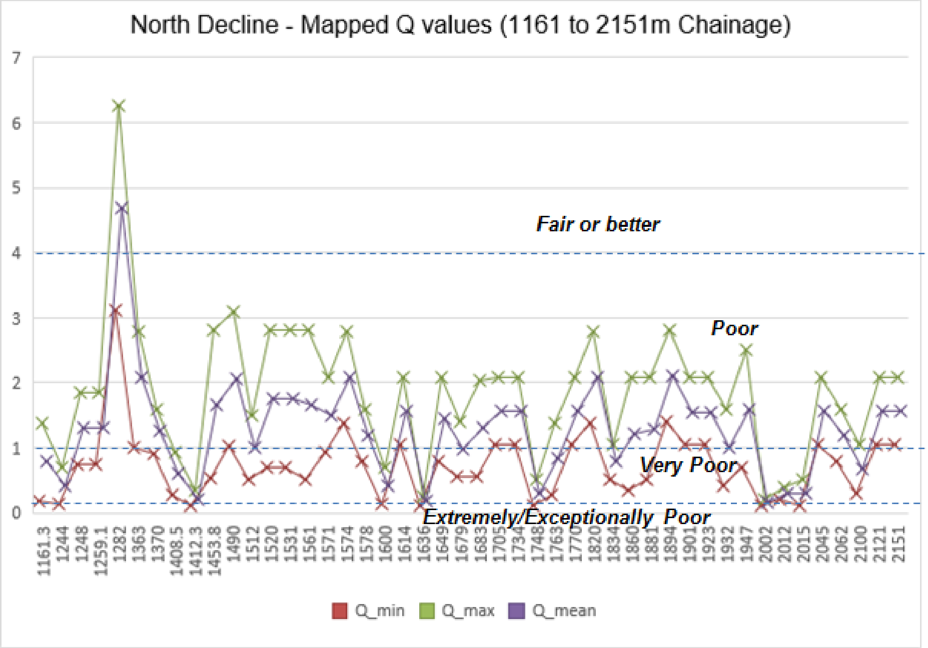


Figure 10.3 Q values from North Decline Crosscut 5 to Cross 10; rockmass classification



Decline Dimension

The 3.3km long twin decline system from the west is oriented across strike of the strata. Mining Plus proposed decline profiles were:

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

- Ventilation decline 4m (W) x 4m (H) arched.

FML revised the tunnel designs in 2020. The standard decline profiles are now:

- Access decline – 4.8m (W) x 5.2m (H) arched. (Main Decline)
- Ventilation decline – 4.8m (W) x 5.2m (H) arched. (North Decline)

These profiles are for the development of the decline only and will need to be re assessed before any production mining begins. Ground support designs have also been modified to suit these larger tunnel dimensions.

Decline Ground Support

Mining Plus proposed three major ground support regimes shown in Table 10.3.

Table 10.3 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

Mining Plus recommended that the rock mass 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 was assumed.

The distribution of ground support over the length of the proposed twin decline is shown in Table 10.4, and is an estimate for design purposes.

Table 10.4 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%	

More recently FML has revised the ground support regimes based on the experience of mining the tunnels to date, and on reviews by geotechnical consultants (SCT and P Keall & Associates). These Manager’s Support Rules specify the ground support required in all development. There are four (4) basic support types for development drives and one (1) for intersection cable bolting:

- Class I - Mesh/split sets
- Class II- Mesh/split sets +/- shotcrete (50mm)
- Class II – Shotcrete (75-100mm)/mesh/split sets
- Class IV -Shotcrete to floor (120mm)/mesh/split sets
- I+ – Cable bolting intersections.

Ground support utilizes rings of >46mm 2.4m to 3.0 m friction bolts (split -sets), with 10mm diameter galvanized mesh. Bolts spacing is 1 metre in each ring, with rings typically on 1.4m or less spacing. Fibre reinforced shotcrete is used in poorer grounds areas. Twin strand (6m) grouted cable bolts are used to provide deeper support at intersections, and in areas of very poor ground.

Ground Support Installation is managed under a TARP (Trigger, Action, Response, Plan) that sets out the conditions under which a change in Support Type is to occur. The TARP provides a description of ground condition indicators which, where observed separately or individually and may indicate a change in Support Type for individual headings is needed. Ground support regime selection responsibility lies with the Mine Superintendent or deputy in consultation with the Project Geologist, Tunnel Manager and Supervisors. If the ground support regime is being reduced then a risk assessment may be required.

Quality control and monitoring procedures are in place to monitor ground support installation and performance. This includes systematic testing of split sets (pull testing) to ensure they meet design criteria, testing of shotcrete quality and installation, convergence monitoring, and visual inspections.

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 deep exploration drill holes were primarily to determine the presence, thickness and grade of the reef and not for geotechnical data collection.

The assessment (as reviewed by AMC) of likely ground conditions in the vicinity of the reef is represented in Table 10.5.

Table 10.5 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.

The values above appear similar to FML observations in the tunnels to date (**Error! Reference source not found.**FML will be collecting more geotechnical information as part of the resource definition drilling program in 2023. This will allow more accurate assessment of rock quality around the stopes for future mine study work.

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:

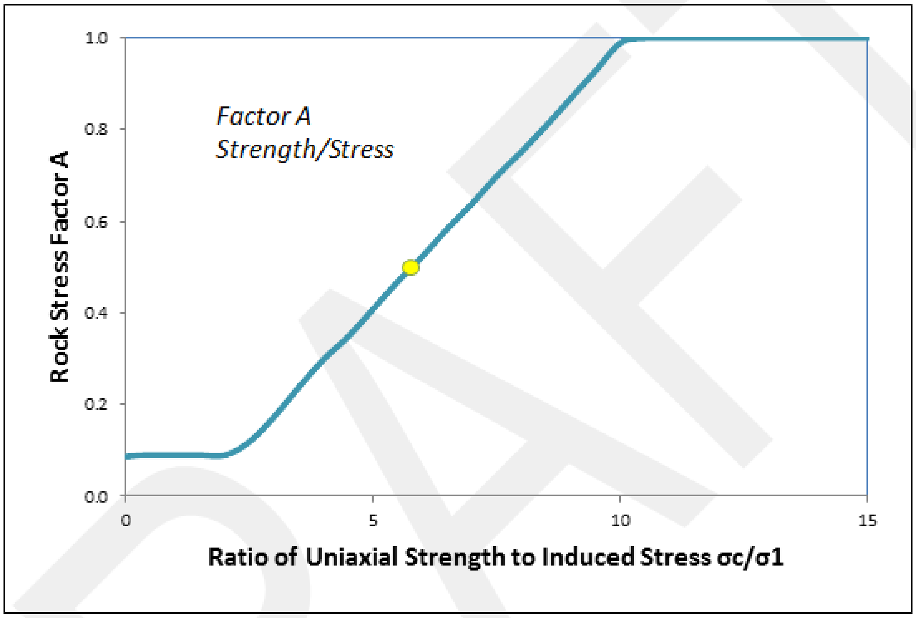
$$Ratio = \frac{Uniaxial\ compressive\ strength\ (UCS)}{Maximum\ induced\ compressive\ stress(\sigma_{max})}$$

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 10.6 details the parameters used to determinate the ratio of UCS to induced stress, and Figure 10.4 shows the corresponding Factor A.

Table 10.6 Rock Stress Factor A

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

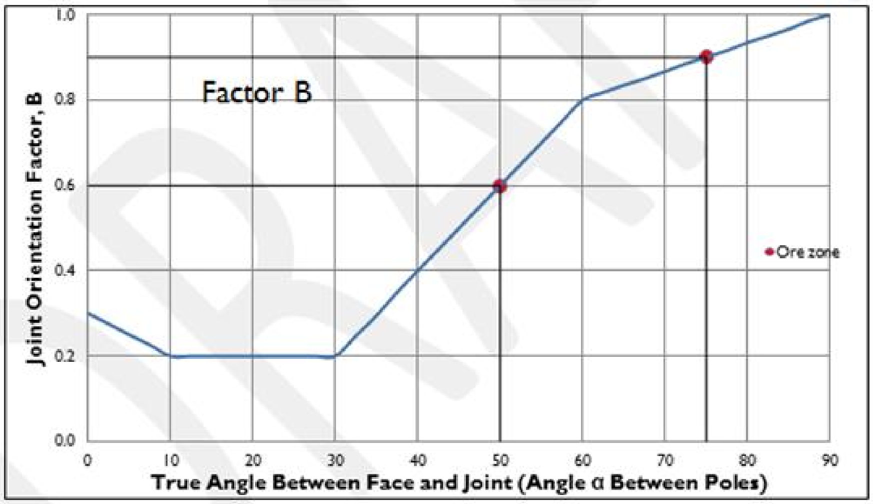
Figure 10.4 Rock stress Factor A



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 10.5 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 10.6), and a deposit with a dip of 77° (Figure 10.7) determines the gravity adjustment factor to be 6.65.

Figure 10.6 Slabbing is the mode of structural failure assumed in the study

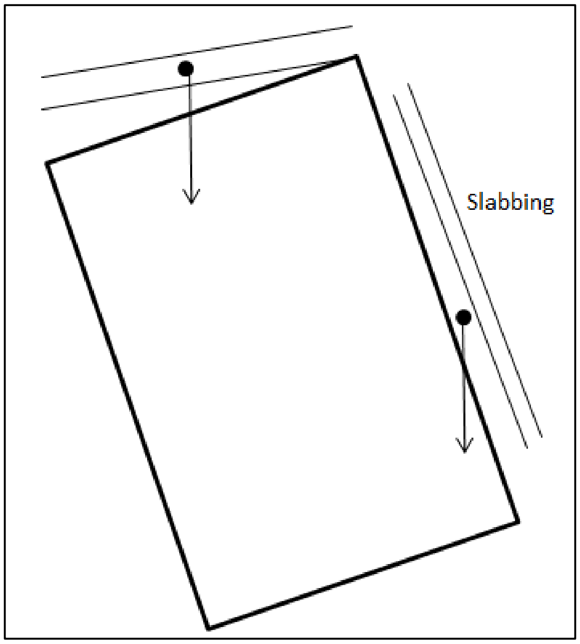
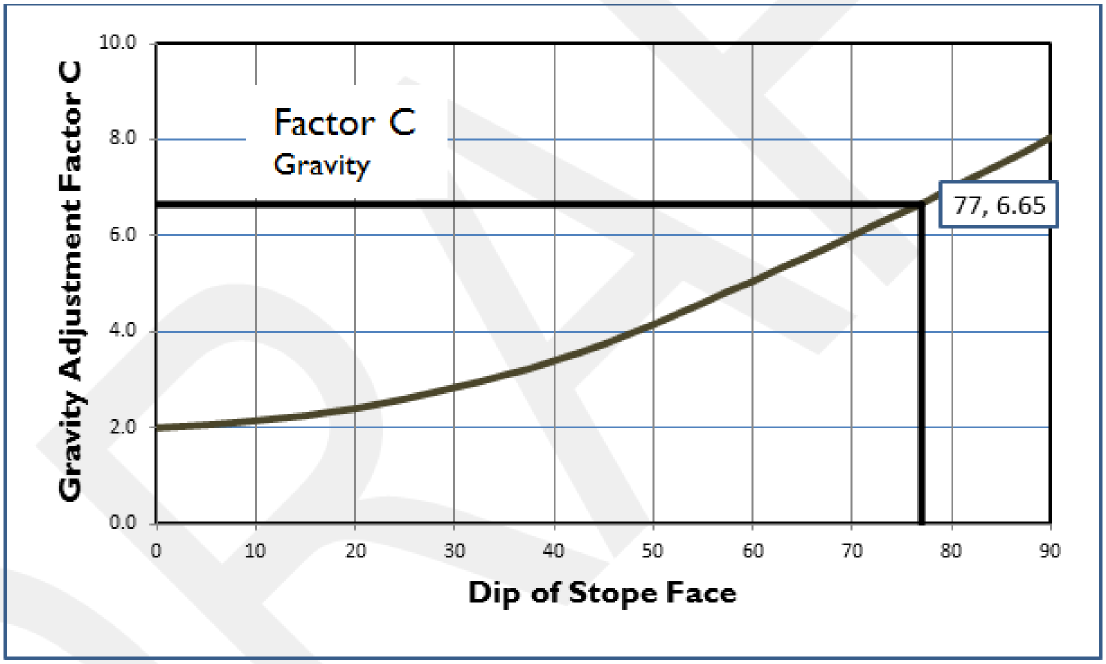


Figure 10.7 Gravity Adjustment Factor C



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 10.7.

Table 10.7 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

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. FML intends to use small scale mechanised open stoping with limited exposure to personnel, thereby reducing the risks involved in utilisation of air leg mining methods.

Preliminary studies have been undertaken on these mining methods, and further work will be required once additional data is available from drilling.

Required Standards

FML’s economic analysis is based on the 2018 Mining One Study with updated capital and operating costs. The LOM schedule and mine design work by Mining One is based on a ‘top-down’ up-hole stoping retreat mining method with pastefill. Stopes will be drilled with up-holes by a longhole drill rig and mined retreating towards the accesses. Stopes will be subsequently filled before the next stope in sequence is mined beside it on the same level, or below it on the level below.

The mine design includes a single ‘figure 8’ decline skewed northward with depth to remain offset from the centre of the orebody while following the ‘plunge’ of the orebody. The decline branches into north and south sub-level drives which provide access to three consecutive stoping levels. Level access then connects the sub-level drives to the orebody. The north and south sub-level drives provide two means of access to each stoping level. The additional infrastructure was included to reduce excessive production loading tramming lengths and to optimize ore production across increased mining fronts (four mining fronts per level; two north and two south). Each level is 13.5 metres from ‘floor-to-floor’ with 3.5 metre high ore drives and 10 metre high stopes.

A crown pillar has been designed every nine stoping levels (every three sub-levels) to enable concurrent mining fronts on underlying levels. Additional stoping availability is key to achieving the required 300ktpa production target.

Ore drives span approximately 915 metres along strike, with 455 metres between each level access from the north and south. This effectively reduces the longest level tram from 450 metres to 230 metres. Ore passes are also positioned every three stoping levels to minimise loader tram distances.

Fresh air will be drawn in through and down the access decline. Return air will be drawn out by return air rises (RAR), positioned every three sub-levels off the northern extents of the decline, and then up the parallel return air decline to surface. Secondary ventilation will be required for each operating level that will be exhausted through the RAR and up the return air decline.

Stopes have been designed 10 metre high between levels with a 10 metre strike length. Stope widths are 1.8 metre minimum including allowances for internal planned dilution. This is roughly 504 t per stope with an assumed density of 2.8 t/m3. With an assumed in-situ reef grade of 23 g/t at 0.68 metre wide, stope width parameters create a diluted stope grade of 8.7 g/t. No allowances have been made for overbreak (unplanned dilution). The proposed stope dimensions have a wall hydraulic radius of 2.5 metre (100 m2 / 40 m).

Mining Fleet

The proposed mining method will involve a conventional mechanized mining fleet to complete the twin decline, access the orebody and mine the orebody along strike. Narrow vein jumbos, production drills and loaders will be required to develop and exploit the orebody through open stoping. The proposed equipment types are listed in Table 10.8 with their corresponding mining schedule rates.

Estimated CO2 emissions per 300,000 tonnes of ore processed is 2500 t CO2e per year, equivalent to 0.03 tonnes CO2 per ounce production. 2019 average emissions for gold mines worldwide was 0.08 tonnes CO2 per year, equivalent to 0.04 tonnes CO2 per ounce.

At this point battery powered narrow vein mining equipment is not available but it is anticipated that Snowy River will change over to zero emission electric equipment by about 2026 or 2027. NZ grid power to site comes from hydroelectric generation.

Table 10.8 Equipment fleet and resource capacities

Equipment	Capacity of fleet
Jumbo	200 m / month
Narrow-vein jumbo (3)	400 m / month
LHD	1200 t / day
Narrow-vein LHD	300 t / day
Narrow-vein longhole drill rig	480 m / day
Truck	72,000 tkm / month / truck

A comparison of the fleet numbers for the OGC and FML studies is presented in Table 10.9 with their associated power requirements in Table 10.10.

Table 10.9 Equipment fleet comparison by project phase

	Pre-Production Phase		Production Phase	
Equipment	**PEA**	**Mining One**	**PEA**	**Mining One**
Large jumbo	1	2	1	2
Small jumbo	-	-	-	3
Air-leg	-	-	30	-
Scraper	-	-	15	-
Long hole drill rig	1	-	1	4
Large loader	1	1 (Asset hire)	1	2
Small loader	-	-	-	4
Trucks	1	2	4	5
Normet	1	-	1	-
IT	1	1	1	2
Grader	1	1 (Asset hire)	1	1
Light vehicle	5	2	16	8
Stores truck	1	-	1	-

Note: PEA fleet figures only include pre-production and expansion (excl. sustaining)

Table 10.10 Equipment power requirements

Equipment	PEA (Units)	Mining One (Units)	PEA (kW)	Mining One (kW)
Large jumbo	1	2	150*	206
Small jumbo	-	3	75*	209
Air-leg	30	-	-	-
Scraper	15	-	-	-
Long hole drill rig	1	4	80*	320
Compressor	3	1	540*	180
Primary fan	1	1	250	350
Secondary fan	20	10	440	550
Pumps	19	30	697	221
TOTAL			2,232	2,036

(*) where kW ratings aren't supplied in PEA technical model (only kWh), post-PEA assumptions utilized.

Mining Method

The mining method selected for the Mining One study is long hole open stoping with the use of paste backfill. The reef will be mined along strike and then retreated back in 10m intervals from the top down. A slot raise will be formed adjacent to the paste filled stope with a cylindrical compressed air balloon and blasted with three firings per stope. Drill and blast practices will need to be optimized to reduce stope widths and prevent excessive dilution. Narrow vein tele-remote LHDs will be used to extract stope ore to be transported to a single sub-level stockpile. A combination of conventional and remote bogging is considered with remote bogging assumed to be 30% of the total bogging.

An ore pass is located every three stoping levels on the north and south sub-level accesses to mitigate long tramming distances. Once extracted, stopes will be paste filled to improve stope wall stability and reduce potential dilution.

Table 10.11 Mining method stope design parameters

	PEA	Mining One Study
Production panel strike length	900 m	900 m
Production panel height	100 m	10 m
Work area strike length	60 m	10 m
Work area height	100 m	10 m
Ore drive height	2.5 m	3.5 m
Quartz vein average thickness	0.68 m	0.68 m
Ore resue width (incl. dilution)	1.0 m	N/A
Waste resue width	2.0 m	N/A
Resue lift mining height	2.0 m	N/A
Sill / crown pillar height	5.0 m	10.0 m (Recoverable)
Stope width (*)	3.0 m	1.8 m

(*) While the stope width in the PEA is listed as 3.0 m the actual ore extracted width will be 1.0 m, the remaining width is for access and generated waste will be used as backfill material.

The top down mining sequence suggested in the Mining One study is largely dictated by the pastefill backfill regime. Measures have been included to ensure a minimum number of stoping fronts are available to maintain required production rates. Dual level access will provide four stoping fronts on each level.

An instantaneous fill rate of 40 m^3/hr has been assumed. In the schedule, a nine day period has been allowed for the fill cycle from end of production to commencing the following stope. The fill sequence with each stage and duration is listed below.

Fill sequence:

- Wall build – three days
- First fill and cure – two days
- Final fill and cure – four days

The adjacent stope along strike can commence production following the final fill and cure period of four days. The fill will still behave plastically but will have sufficient cohesion to support its self-weight when exposed which has been based on industry experience.

The reticulation system will require further investigation for final specification of the paste plant. The paste will be distributed underground through a steel pipe run down the decline and through service holes for supply to production areas. This will need to be reviewed in relation to the paste plant pumping capacity given the distance and low angle nature of the decline. The use of paste backfill will reduce the tailings volumes that will be stored on the surface and will ensure higher mining recovery.

Crown pillars have also been designed every nine stoping levels to allow for mining to commence on underlying levels and effectively open up additional stoping fronts. These crown pillars are essential to the mining sequence as schedule iterations failed to achieve the required production rate without them. The orebody will be retreated back in 10 m stope intervals.

In addition to the wall build there will be a cavity monitoring survey (CMS) of the stope void carried out, for volume determinations, and drilling of fill and breather holes into the void. At the end of the filling process a development cut will be required to remove the fill wall and adjacent fill to expose the drive for drilling.

Material Handling

The Mining One study considers mechanized material handling techniques through the use of LHD equipment suited to narrow vein deposits. Broken ore from uphole stoping will be mucked by a combination of conventional and remote bogging operations. A remote component of 30% has been assumed given the small stope geometries. Blasted rock will be trammed to the stockpiles located on the hangingwall sub level access drives. Alternatively, it will be transferred via ore passes located on every third stoping level to reduce tramming distances.

Mining Schedule

The top down retreat schedule was evaluated for sequence constraints, along with equipment optimization to ensure the 300 ktpa target production rate could be achieved.

Development within the mining schedule was set to As-Late-As-Possible (ALAP) to defer capital costs and maximize project value. Resource levelling was derived from the capacities mentioned in Table 10.8. Truck fleet capacities assumed 90% Availability and 80% Utilization of Availability, giving an Efficiency (or Utilization) of 72%.

Ventilation

The decline portal to the uppermost level of the proposed Snowy River Mine 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 10.8 and Figure 10.9 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 10.8 VentSIM Visual Model

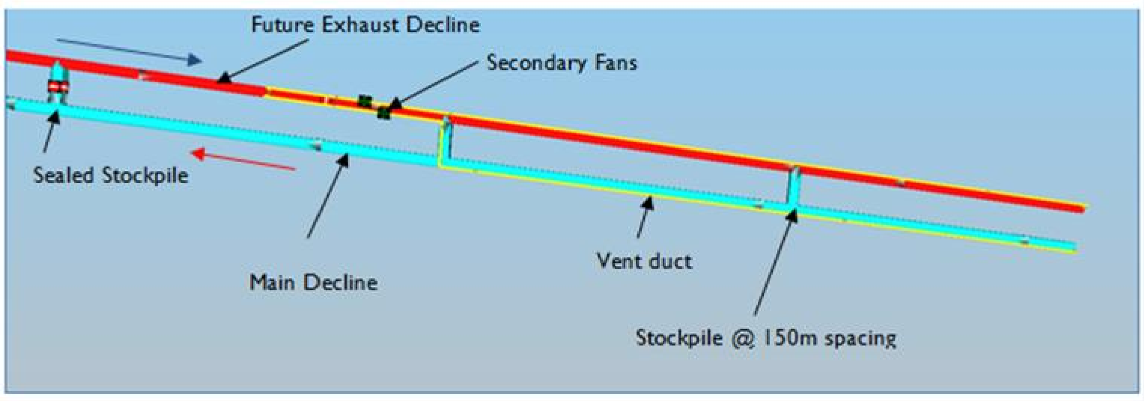


Figure 10.9 VentSIM Visual Model Detail

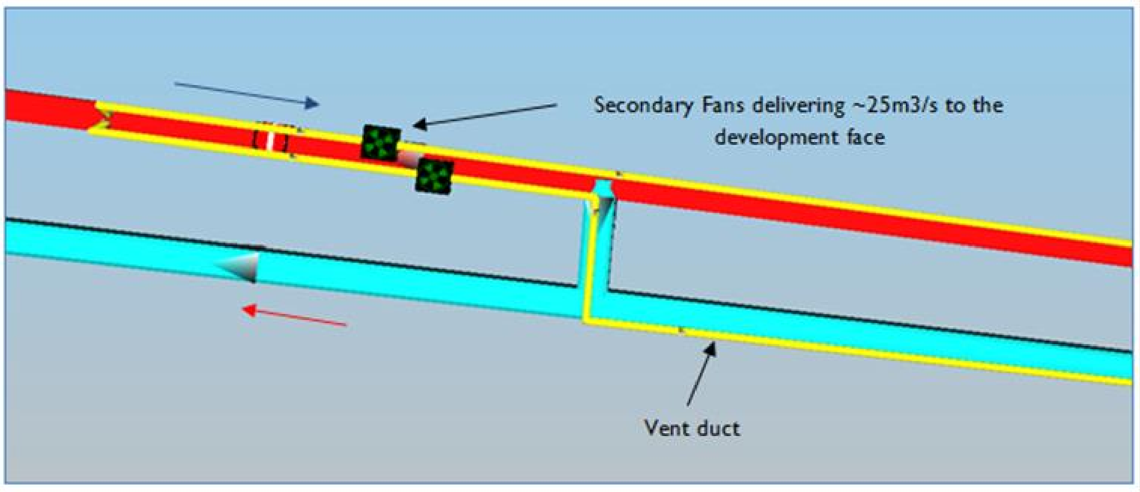
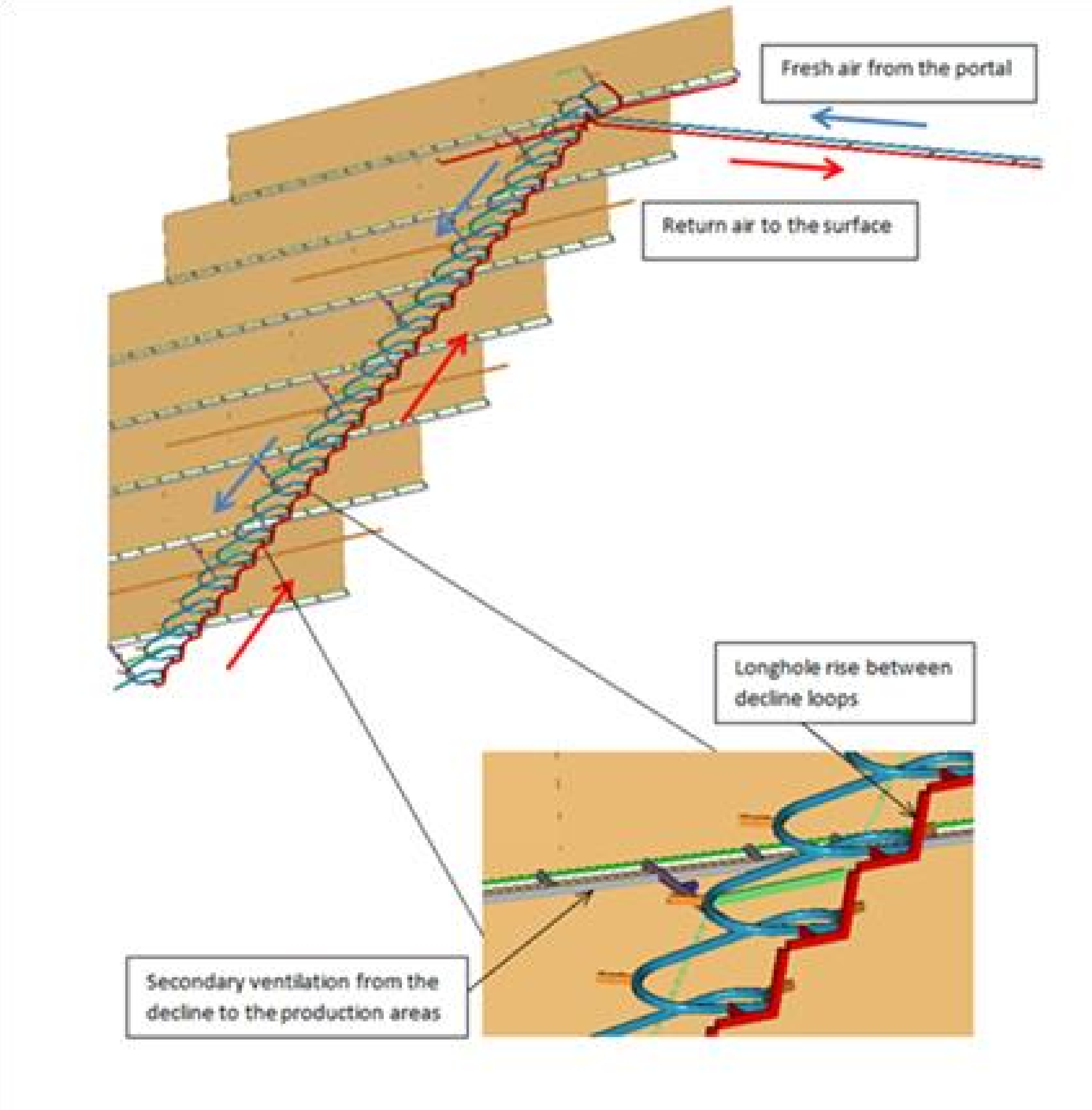


Figure 4.1: VentSIM Visual Model Detail

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 10.10 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 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. Ore passes 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.

Emergency Egress

An independent egress system between level accesses from the decline will be established. Each will require a 1.5m diameter raise bore, inclined at about 60° above horizontal. Installation of SafEscape ladder ways 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.

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 mine workings 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.

Dewatering will be achieved via pumps located throughout the mine, with water being delivered to a surface water storage dam.

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 assuming all the old workings were not filled and are open, 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 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. 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.

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.

Decline Development and Drilling Program

The orebody is being accessed via 3.3 km twin parallel declines - a main access decline (MAD) and a return air decline (RAD). Both declines are being mined concurrently allowing for adequate ventilation and an increased number of development work areas.

The initial twin declines development phase has been separated into four parts; portal establishment, main declines to bottom, primary pump station and diamond drill drives.

Table 10.12 Pre-production development breakdown

	Unit	Portal	Declines	Pump Station	DD Drive
Portal	m	20	-	-	-
MAD	m	150	3,030	-	-
RAD	m	150	3,030	-	-
XC	m	20	400	-	-
Geotech cuddies	m	-	200	-	-
Pump station	m	-	-	30	-
DDD	m	-	-	-	400
Geotech	drm	-	3,945	-	-
DDH	drm	-	-	-	14,400

The main declines to the bottom represent the majority of the development phase. The declines have cross cuts (XC) developed every 180 m connecting the MAD to the RAD. The twin decline is offset by 20 m from centerline-to-centerline. Additional drill cuddies (20 m in length) will be established at XC17 approximately 250m from the orebody to facilitate diamond drilling to assess upcoming groundwater and geotechnical conditions. Decline development spans 21 months at an average advance of 330 m development per month. The primary pump station will be situated at the base of the decline. The north and south diamond drill drives (DDD) will span three months. The diamond drill drive design is currently set at 400 metres in length, at a distance of 100m from the orebody.

The diamond drilling program will commence upon the completion of the DDD. Initially, a single rig has been scheduled which will ramp up to a potential maximum of four rigs throughout the program. Twenty metres per shift has been assumed for the drilling capacity.

Both the PEA and Mining One study recommend the use of conventional mechanized mining for the development of the twin decline and accompanying waste infrastructure. Table 10.13 below provides a comparison of the decline design parameters. The only subtle discrepancy is the drive profile dimensions which are larger in the current concept study to accommodate larger mechanized equipment.

Table 10.13 Twin decline design comparison

Design Parameter	PEA	Mining One Study
Decline profile	4 mW x 4 mH, arched	4.8 mW x 4.8 mH, arched
Gradient	1:6	1:7
Length	3.2 km	3.2 km
Orientation	W-E	W-E
Cross cut frequency	150 m	150 m
Twin decline offset	20m	20m

Diamond drilling of 32.4 km was proposed in the Mining One study beginning with one drill rig and ramping up to four rigs. Table 10.14 outlines the design considerations for the PEA and Mining One study. FML has modified the drilling plan to outline sufficient mineralization to enable financing of the Project construction completion. It is anticipated that a 4 month (14km) drill program would outline a panel of 60,000 m2 or approximately 440kt at a diluted grade of 7.9 g/t. An allowance has been made for 1200 m per month (the equivalent of 1 diamond drill rig) of grade control drilling upon the commencement of production for the life of the project.

Table 10.14 Pre-production ore definition diamond drilling program comparison

Design Parameter	PEA	Mining One Study	FML Analysis
Drill drive length	600m	500m	400m
Location	-350 mRL	-250 mRL	-250mRL
Drill fleet	2	1-4	3-4
Duration (months)	4	10	4
Drill rate (m/day)	35	40	40
Resource target (Mt)	0.9	1.3	0.95
Drill (km)	8.0	32.4	14.4

Mining Method Comparison

The mining method proposed by Mining One for the exploitation of the Birthday Reef involves fully mechanized mining for both the development and production stages of the project. This is the fundamental difference that sets it apart from the strategy put forward in the 2014 PEA prepared by OGC. The PEA proposed hand-held (air-leg) mining techniques for stoping activities, specifically the use of resue blasting techniques to facilitate the separation of ore and waste.

Table 10.15 Advantages and disadvantages of proposed mining methods

Mining Method	Advantages	Disadvantages
Air-Leg resue	• Decreased costs (less capital expenditure, backfill with waste rock) • Highly selective • Reduced dilution (resue blasting) • Minimized stope development (100 m panels)	• Increased safety risk (personnel exposure to rockfalls including from the face, intensive manual handling) • Scraper operation safety concerns (stored energy) and inefficient means of ore recovery • Skilled labour shortage (experienced operators retiring, limited uptake in trade) • Greater impact on operators body (machine vibration) • Industry tendency to dissociate from air-leg mining (regulations may limit or ban use of method) • Decreased production rates • Mining cost uncertainty
Uphole open stoping with pastefill	• Higher safety compared to resue mining (less labour intensive and non-entry mining method) • Higher production rates through mechanization • Accurate cost estimates	• Increased costs (capital equipment, pastefill requirements) • Increased stope dilution • Decreased mining selectivity

While a higher production rate is achievable through mechanized mining, in the Mining One Study LOM is five years longer (15 years) than that proposed through air-leg mining (10 years). This is attributable to the Mining One assumption that recovered ounces increases from 570 koz to 977 koz based on a positive view of the resource upside. FML has assumed for its base case scenario the inferred resource of 700 koz, bringing the LOM to 10 years.

Total ore tonnes mined is 3.5 times greater in the Mining One stoping study (4.06 Mt) compared to the PEA (1.17 Mt). This is largely due to the selectivity offered by resue blasting effectively reducing tonnes and dilution for an increase in stope grade. With 13.5 metre floor-to-floor spacings, the current study also requires additional capital and operating development. In contrast, the PEA strategy has allowed for seven 100 m high production panels. Stable stope spans are the constraining factor here and separate waste blasting permits continuous mining up dip. Mechanized mining however will require paste backfill at set intervals.

Table 10.16 Comparison of mining parameters of PEA and Mining One study

Mining Parameter	PEA	Mining One
Mining Method	Air-leg Resue	Longhole stoping
Backfill	Waste rock	Pastefill
Ore production (per annum)	120,000 t	300,000 t
Mine life	10 yrs	15 yrs
Ore drive	2.8 mW x 2.5 mH	2.5 mW x 3.5 mH
Stope dilution (planned)	50% (of 0.68 m)	0% (of 1.8 m), dilution included in 1.8m stope width
Development dilution (planned – all lateral)	0%	5%
Development recovery	100%	100% *
Stoping recovery	95%	100%
Gold recovered	570 koz	977 koz
Gold recovered (per annum)	58 koz	65 koz *

(*) 100% stoping recovery justified through top down mining sequence, material unrecovered from stope will be recovered below.

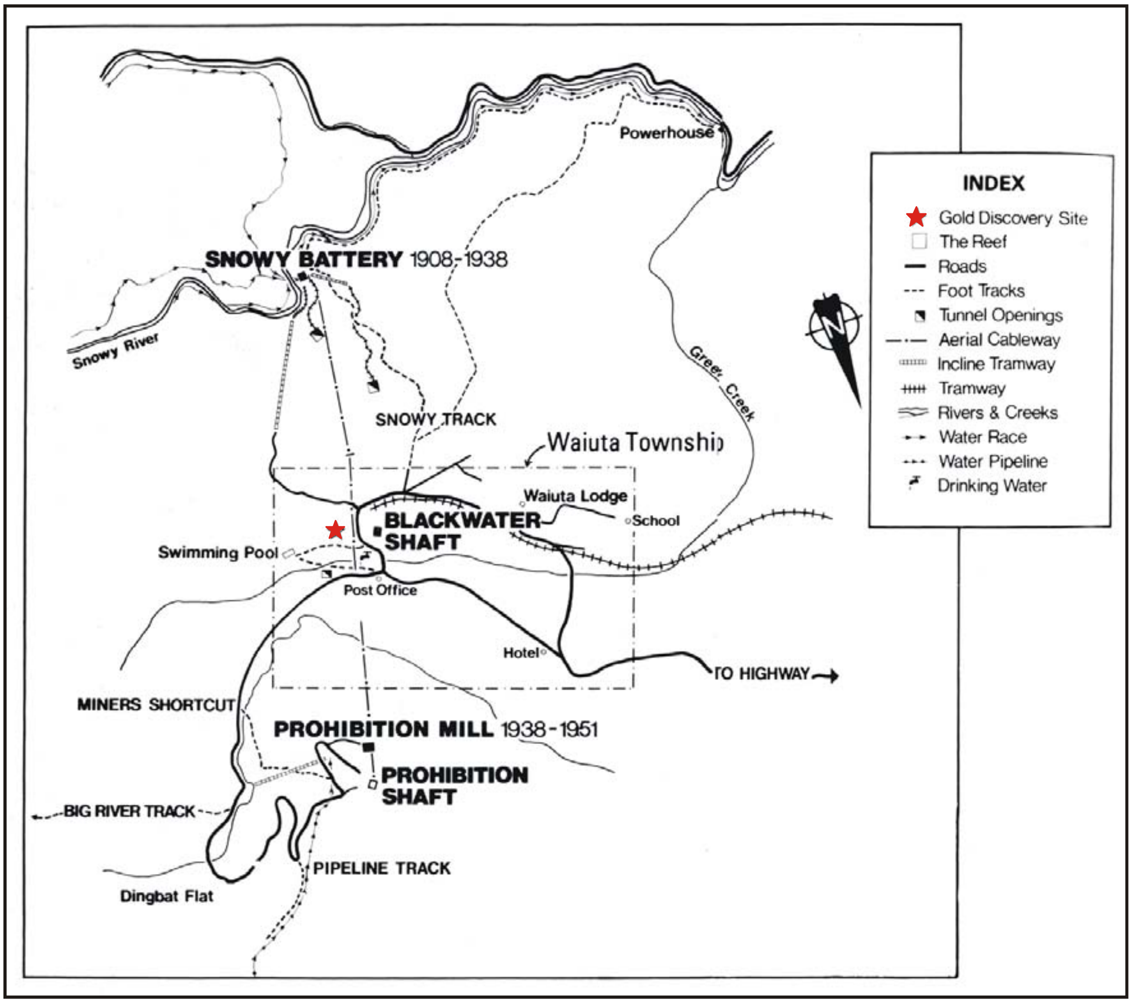
65 koz. recovered per annum represents an average over the 15 year life of mine.

11 PROCESSING

11.1. Historical Processing

The Birthday Reef was treated at Waiuta through two process plants, whose locations are shown in Figure 11.1. 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 (**Error! Reference source not found.**). 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 11.4 plots the gold production and throughput of the Blackwater process plant from 1908 to 1951 (New Zealand Mines Statements, 1908-1952).

Figure 11.1 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 Progress Mine around the original workings and the surrounding halo shows a significant difference in the style of mineralisation and the potential flowsheets needed.

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 11.2 (Hancox,1985).

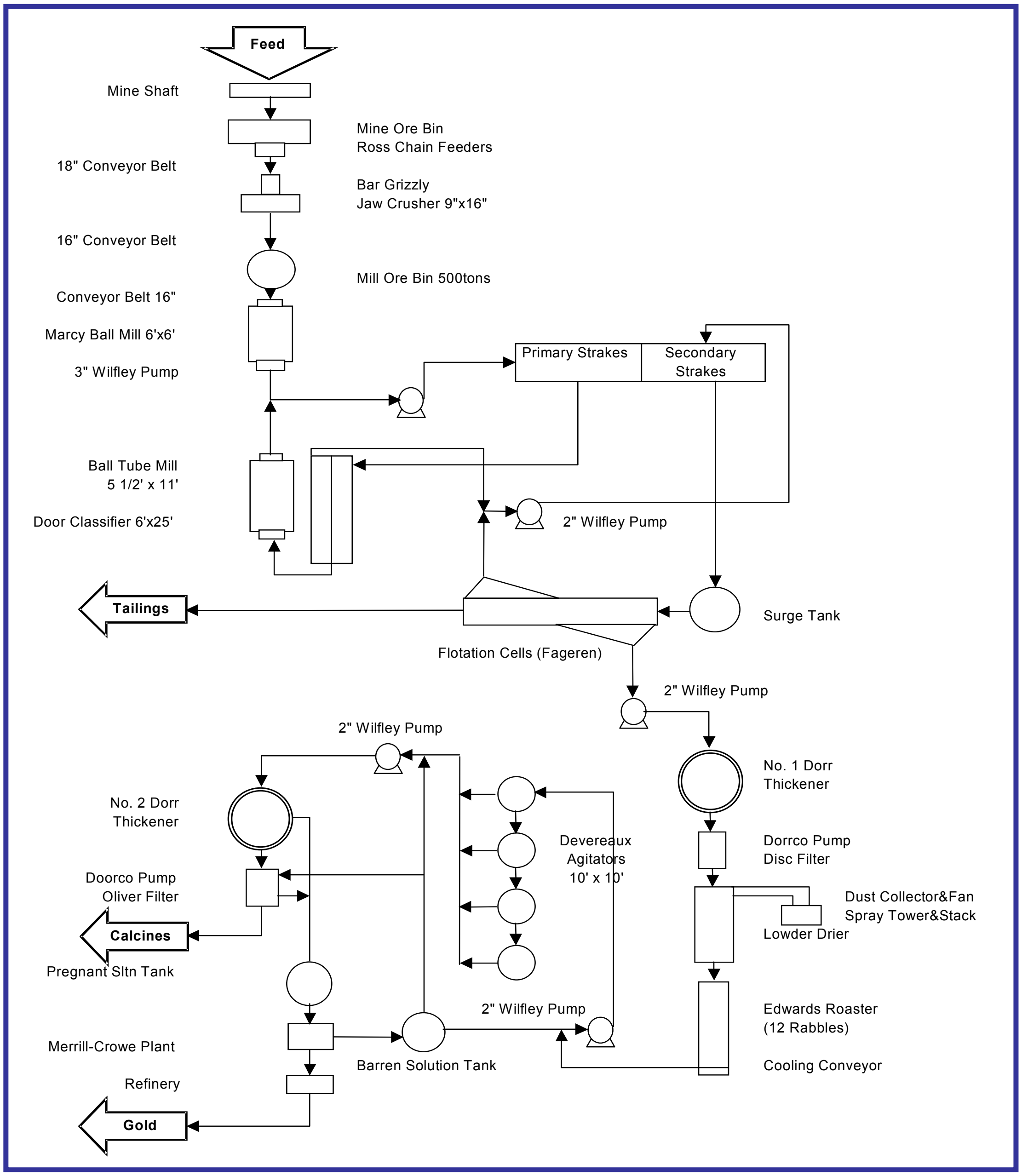
Henderson (1917) quoted gold recovery from the first plant as 89-90%, however Morgan (1929), noted the poor sampling quality and regimes making it difficult to confirm this recovery.

Figure 11.2 Snowy River Battery 1909

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 11.3 (Pearson, 1942).

Figure 11.3 Prohibition Mill Flowsheet



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.

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.

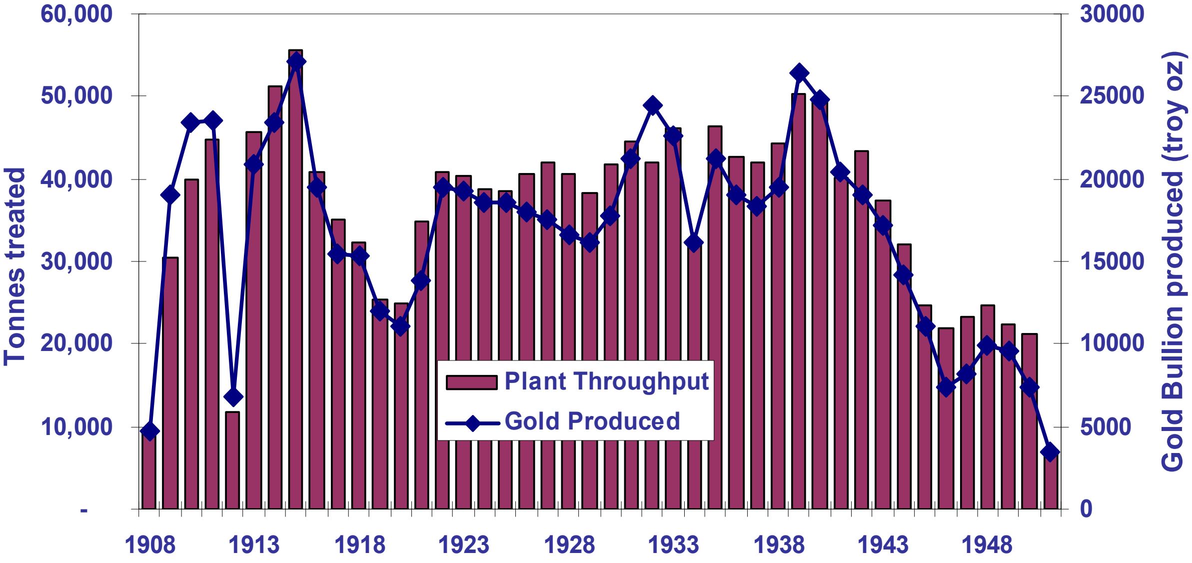
The recovery of gold from the second Blackwater processing plant was reported by the late mining superintendent, JR Hogg as 96% gold recovery equating to 9.31dwt of gold recovered for 9.7dwt fed (Hogg, 1956). These recoveries were supported by GJ Williams (1965) who quoted the Blackwater gold recovery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

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 11.4 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 11.4 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%.

11.2. 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 11.5 and Figure 11.6, respectively.

Figure 11.5 WA11 Core Sample

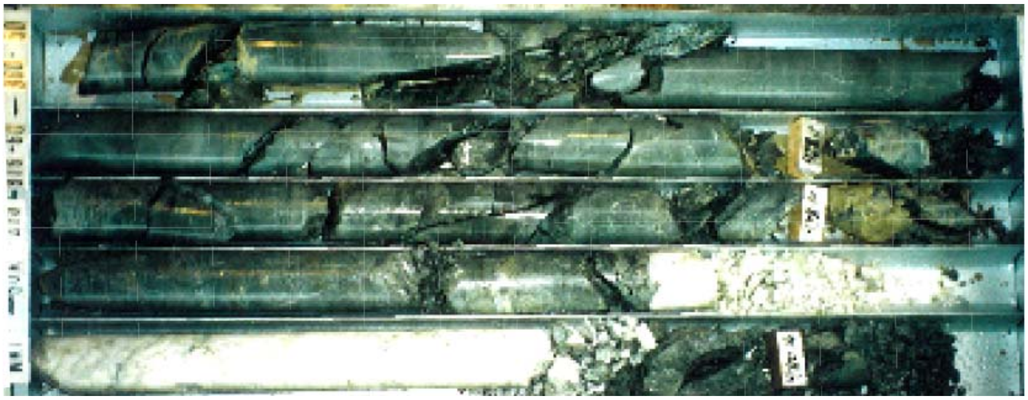
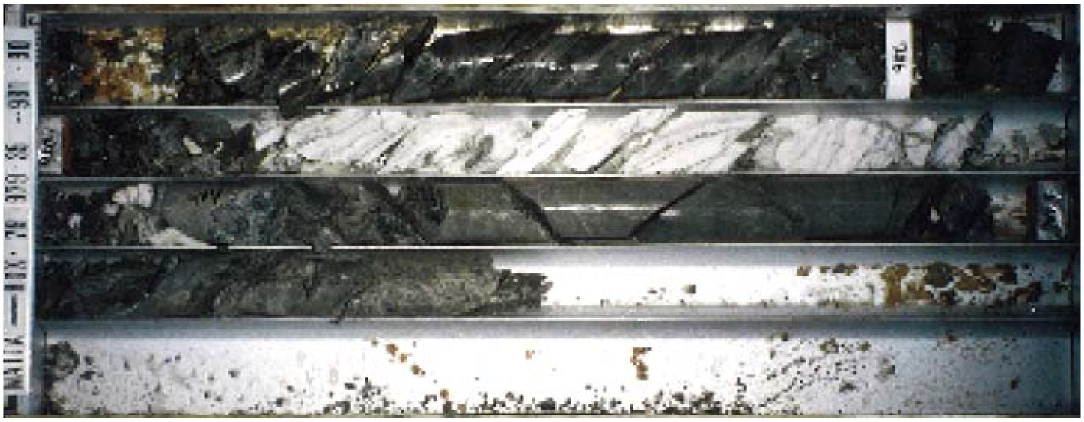


Figure 11.6 WA11A Core Sample



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 11.1. 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 11.1 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*

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 11.2 shows the mineralogical analysis of the WA11 sample.

Table 11.2 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 11.7, Figure 11.8, Figure 11.9 illustrate three different occurrences of gold in the WA11 sample.

Figure 11.7 SEM Image of Gold with Quartz in WA11 Sample

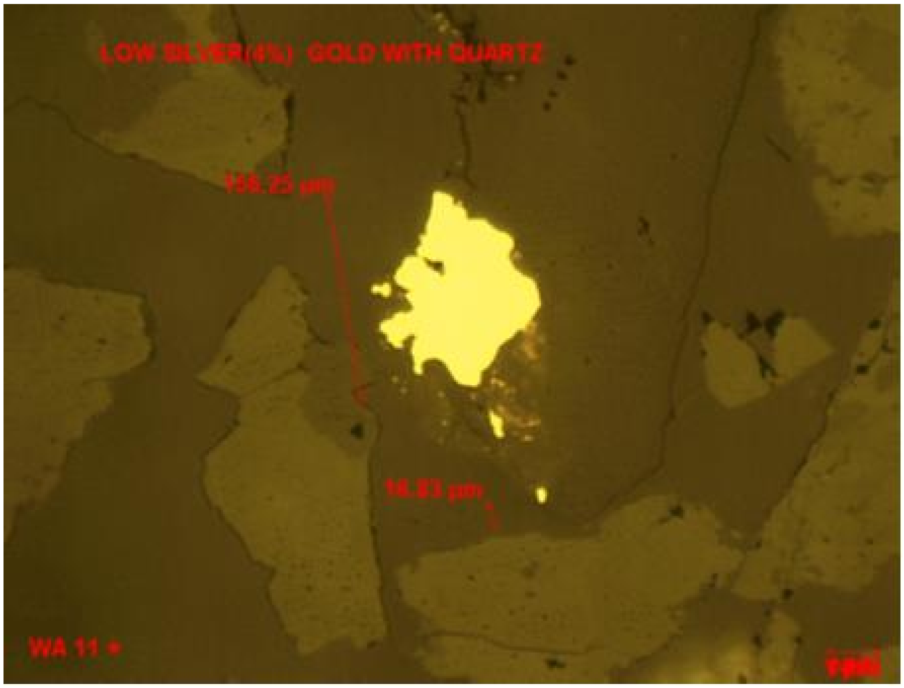


Figure 11.8 SEM Image of Gold with Albite in WA11 Sample

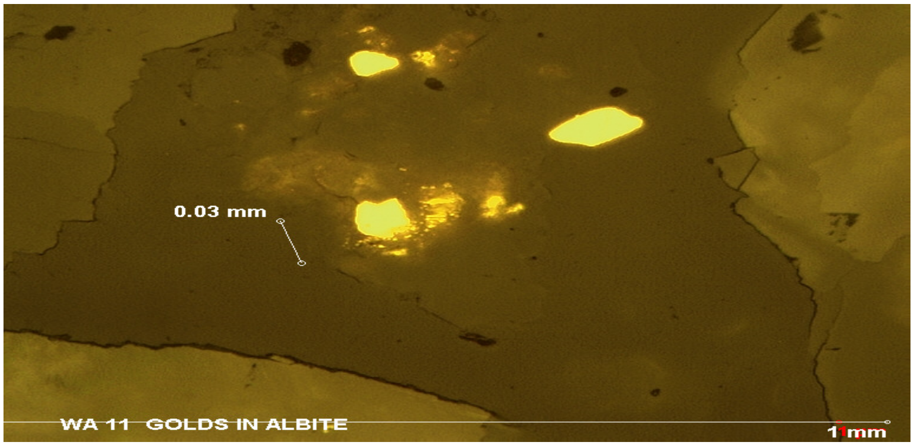
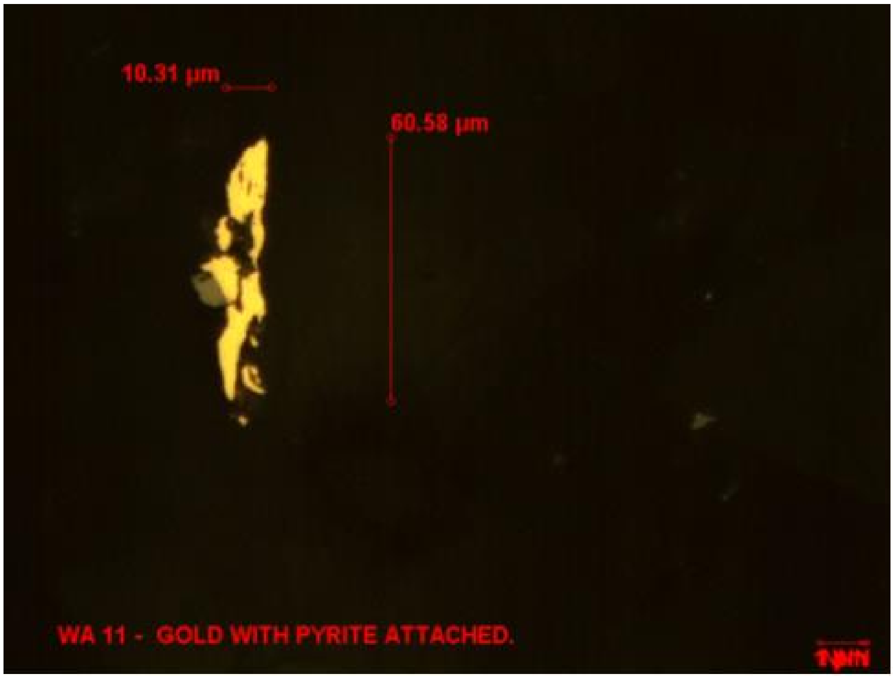


Figure 11.9 SEM Image of Gold on Pyrite in WA11 Sample



11.3. Metallurgical Testwork

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 orebody. 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.

11.4. 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.

Table 11.3 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;

11.5. 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 OGC 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.

11.6. Gekko Python Amenability Test Work 2011

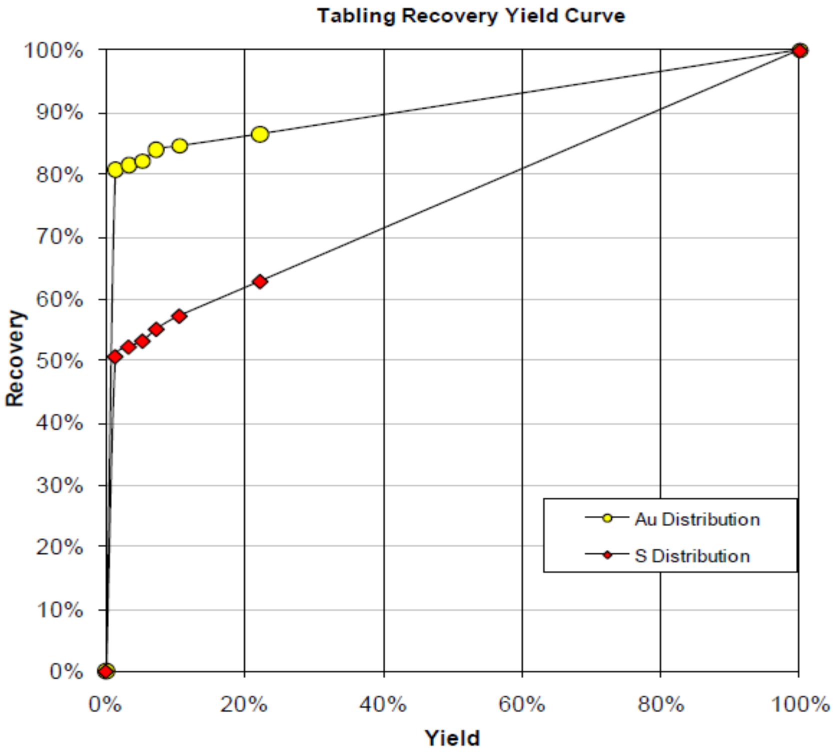
Bulk samples of quartz and waste from the hand sampling program in 2010 were supplied to Gekko Systems to undertake a flowsheet evaluation program. The aim of this test work was to look at abrasion, crusher work index, grind sensitivity and gravity/flotation performance to allow consideration of a Python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

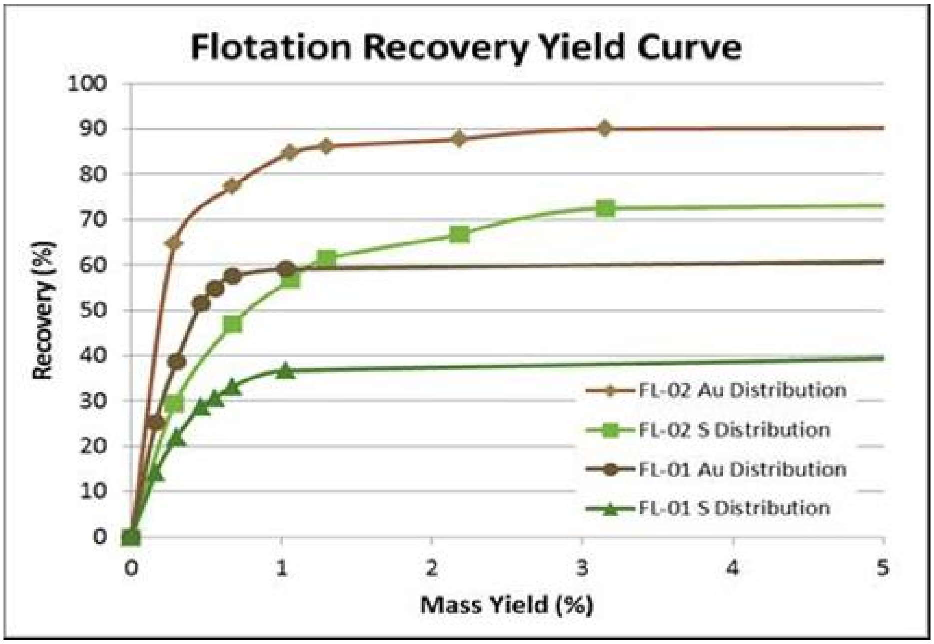
Progressive Grind Tabling (PGT) tests were conducted on the combined material and indicated that 80.8% of the gold could be recovered into 1.35% of the sample mass at a P_{100} of 600µm and P_{80} of 425µm. Further test work utilised a regrind down to a P_{80} of 104µm before the flotation stage and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

Figure 11.10 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 11.11 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 11.4 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 11.4 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

11.7. 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 11.5 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 11.12) 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 11.5 Optical Sorting Results

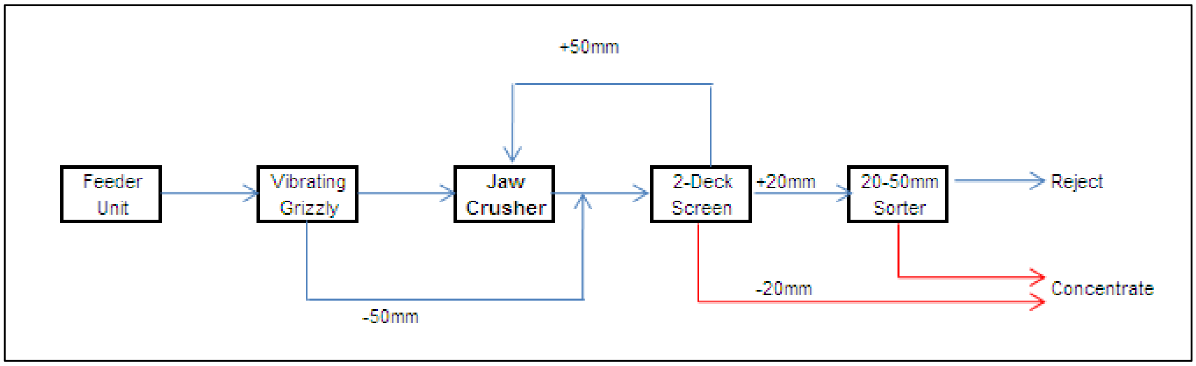
Fraction		53-75mm	37.5-53mm	19-37.5mm		Total	9.5-19mm
	waste	144.4	200.6	169.3	240	394.2	188.7
	quartz	2	1.5	1.71	0.24	1.95	8.3
reject	*total*	*146.4*	*202.1*	*171*	*240.2*	*396.2*	*197*
	waste	21.4	3	3.0	1.48	9.5	12.6
	quartz	54	48.1	71.6	28.12	109.7	41.8
conc		*75.4*	*51.1*	*74.6*	*29.6*	*119.2*	*54.4*
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

Table 11.6 Optical Sorting Results II

	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 11.12 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.

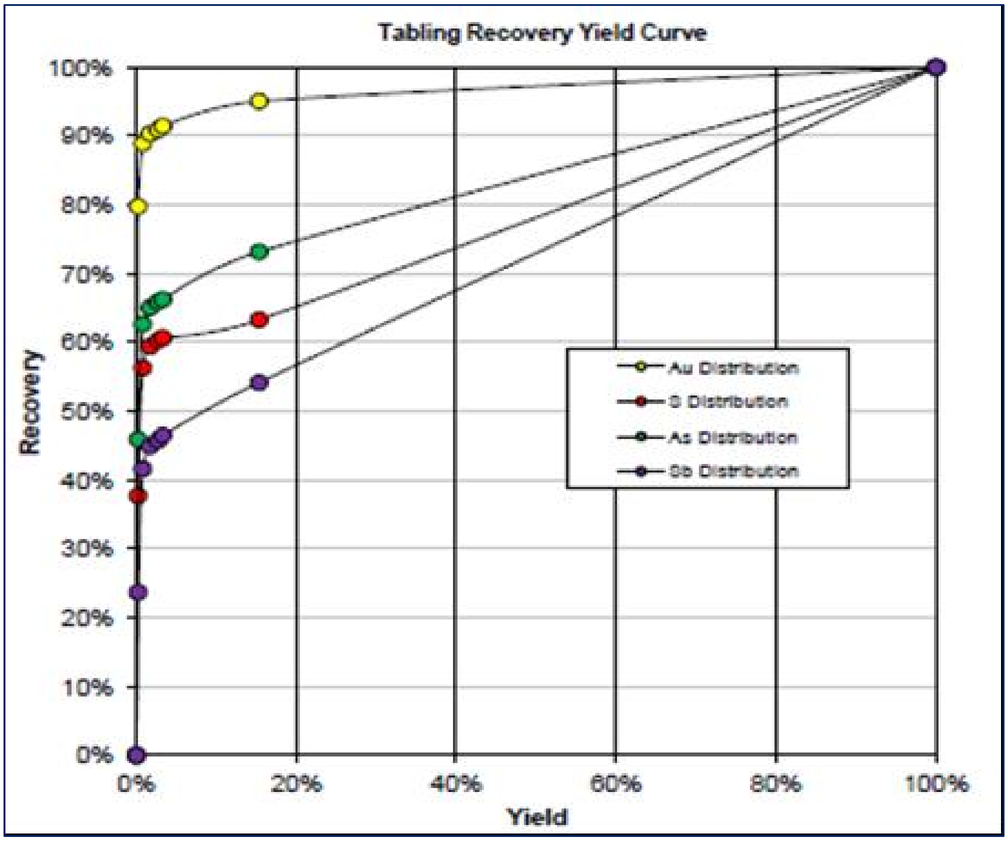
11.8. Gekko Flowsheet Validation Test Work 2013

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 11.13 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%.

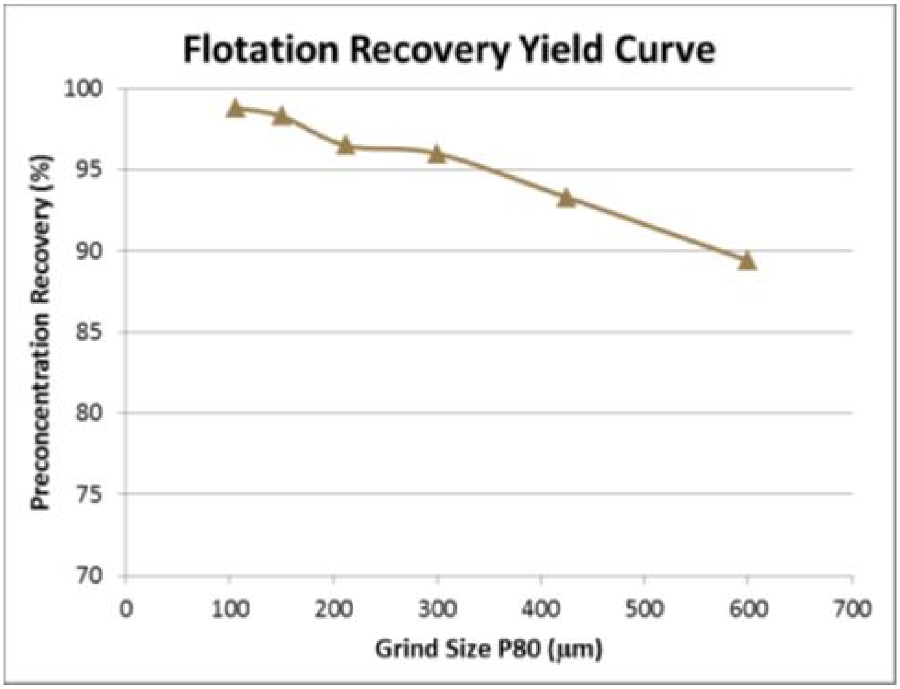
Figure 11.13 Plot of Yield against recovery for the CGR Test



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 11.14 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 11.14 Plot of combined Gravity/flotation recovery against grind size



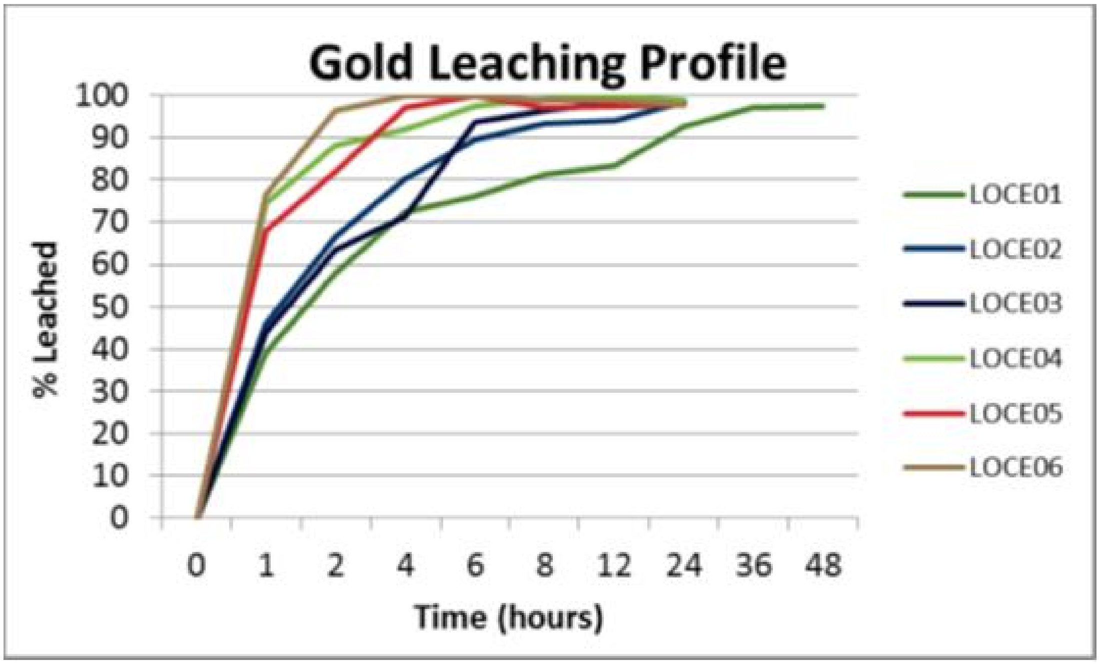
Intensive cyanide leach tests were carried out on the final concentrate product (3 stage CGR & FL05 concentrate) at different grind sizes to determine the effect on recovery. The intensive cyanide leaching results are summarised in Table 11.7.

Table 11.7 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 11.15.

Figure 11.15 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 11.15).

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 11.8.

Table 11.8 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%	

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.

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 11.9. With a calculated gold head grade of 5.9g/t, overall recovery was calculated to be 96.5% (final residue tail of 0.11g/t).

Table 11.9 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

11.9. Flowsheet 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 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. 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.

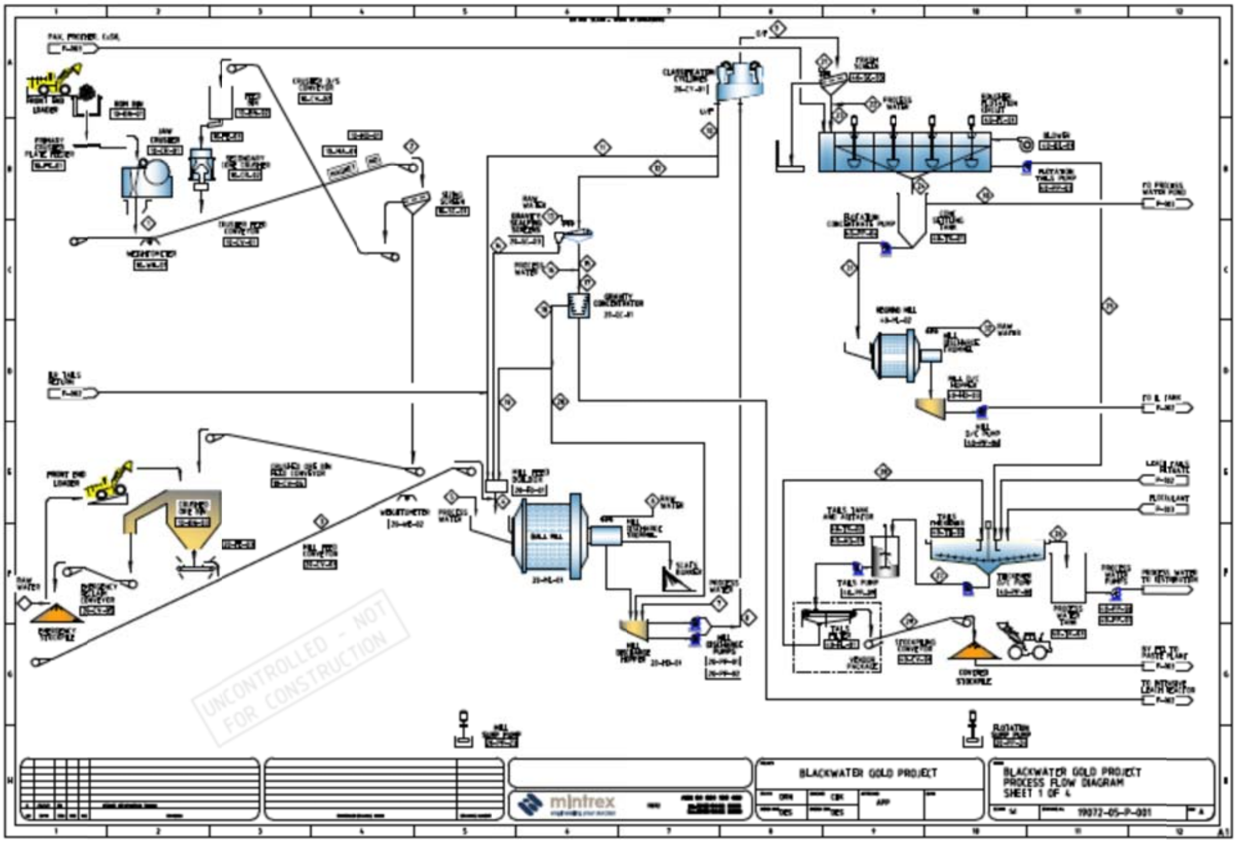
11.10. 2019 Mintrex Plant Study

Mintrex was commissioned by FML in 2019 to revise the 2014 PEA proposed processing plant concept. The Mintrex Study addressed many of the issues and risks associated with the Gekko design, in particular the high wear experienced in other Python plants (in particular with the VSI crusher), the small ROM feed size (<250 mm) and the challenge of operating a resin circuit.

Key parameters included:

- Increase plant throughput from 120,000 tpa to 300,000 tpa at a mill head grade of 7.9g/t Au. Plant operation at 49 t/h, operating 7 days per week, at an availability of 98%.
- Majority of tailings to be returned underground as paste backfill.
- Equipment selected for the specific duty rather than to "fit in" with the Gekko Python/CTP process package.
- Jaw crusher upgraded from the PEA to accept up to 500 mm rock.
- Conventional crushing and comminution circuit consisting of a jaw crusher and cone crusher feeding an overflow ball mill to achieve a primary grind P80 -150 µm.
- Centrifugal concentrator in the grinding circuit to recover liberated gold. Gravity concentrate intensively leached and electrowon.
- Flotation of gravity tails to produce a sulphide flotation concentrate at an expected mass pull of 3% (same rate determined by Gekko for the 2014 PEA)
- Flotation concentrate reground to a P80 -106 µm followed by intensive cyanidation leach
- Adsorption of gold onto carbon in CIP tank Pumpcells.
- Conventional acid wash/elution to recover gold from the carbon followed by electrowinning.
- Gold cathode sludge from electrowinning smelted on site.
- CIP tailings detoxified using the INCO/Air process before being filtered in a plate and frame filter and used to generate a paste for backfilling in the mine.
- Flotation tailings thickened with underflow filtered and used for paste plant backfill or co-disposed with mine waste.
- Capital cost (including paste plant) was estimated at A$44M (US$31M) and was re-estimated by Mintrex in December 2021 at A$54M (US$38M).
- Operating costs are estimated at A$41/t (US$29/t) (2022).

Figure 11.16 Mintrex Flowsheet – Crushing, Grinding, Gravity & Flotation Shown



11.11. Mine Drainage

In 2013 O’Kane Consultants (NZ) Ltd (OKC) were retained to complete a preliminary review of acid mine drainage (AMD). That review was completed based on a limited amount of information, and further investigations are required. In summary OKC concluded that assumptions made for the management of AMD were 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 Snowy River are likely to be minor, provided the potential risks are predicted and managed appropriately. Predictive test work undertaken in 2022 indicates that mine water discharge will be neutral.

Acid drainage from the old Blackwater Mine underground workings is 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. Technical reviews undertaken during 2022 have found that contaminants from the mine and waste rock stack can be treated onsite using an active and passive water treatment system.

The active water treatment system has been designed by a New Zealand based consultancy firm and is planned to be constructed ahead of the processing plant commissioning.

The passive water treatment system is a series of ponds is in operation now and assists in the settling and removal of contaminants found in the development phase such as suspended solids.

11.12. 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 AMD generation in the waste rock dump.

12 ECONOMIC MODELS COMPARISON

The fundamental assumptions and inputs in the PEA, Mining One (and Mintrex) studies and recent FML analysis are outlined in Table 12.1.

Table 12.1 Economic model parameters

Description	PEA	Mining One Study	FML Analysis
Currency	USD	NZD	AUD
Pre-production period	2.5 yrs	3 yrs	3 yrs
Mine life	10 yrs	15 yrs	10 yrs
Operating days/year	350	365	365
Discount rate	5%	8%	5%
Gold price (US$)	1,300	1,300	1,650
Power cost (US$/kWh)	0.08	0.11	0.16
Diesel cost (US$/l)	0.82	1.12	1.25
Capex (US$/oz)	270	142	202
Opex (US$/oz)	446	499	664
Project IRR (%)	29	32	45
Project NPV (US$M)	193	207	258*

*Notes: * - FML NPV/IRR is post-tax and includes purchase price of Project and RoyalCo royalty*

Exchange Rates: A$:US$ - 0.70 ; NZ$:US$ - 0.64 for recent FML analysis

12.1. Economic Analysis

The results of the economic analyses discussed in this section represent forward-looking information. 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 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.

The base case scenario estimates production start up in 4Q24, allowing time for completion of the twin decline, development of 3 levels, a drill drive and an initial underground drilling program. Capital expenditure from January 2023 through to commercial production is estimated to be A$97M (US$68M). Sustaining life of mine capital is estimated to be A$53M (US$37M) including closure and rehabilitation costs.

Capital cost estimates are ±25% and assume that the mobile mining fleet is purchased rather than leased. In addition, the exercise of the option to purchase will add a further A$42M (US$30M) and the Royalco royalty purchase an additional estimated A$11M (US$8M).

Table 12.2 Base Case Pre-production Capital Cost Summary from January 2023 (A$M)

Description	2023	2024
Build: (UG development, waste dump, WTP)	14.5	13.3
Plant	2.9	51.4
Capital Equipment purchases	4.1	8.5
Drilling Program	1.1	1.1
Total	22.7	74.3

Operating costs have been estimated using first principles derived from supplier quotations and/or benchmark data from OGC and other similar operations and updated to 2022.

Table 12.3 Operating Cost Inputs (A$/t Ore)

Item	A$/t
Mining	158
Processing	41
G&A	23
Total	223

The base case scenario at US$1,650/oz gold price estimates a post-tax NPV of A$338M.

Figure 12.1 Project NPV (A$M) – Gold Price range

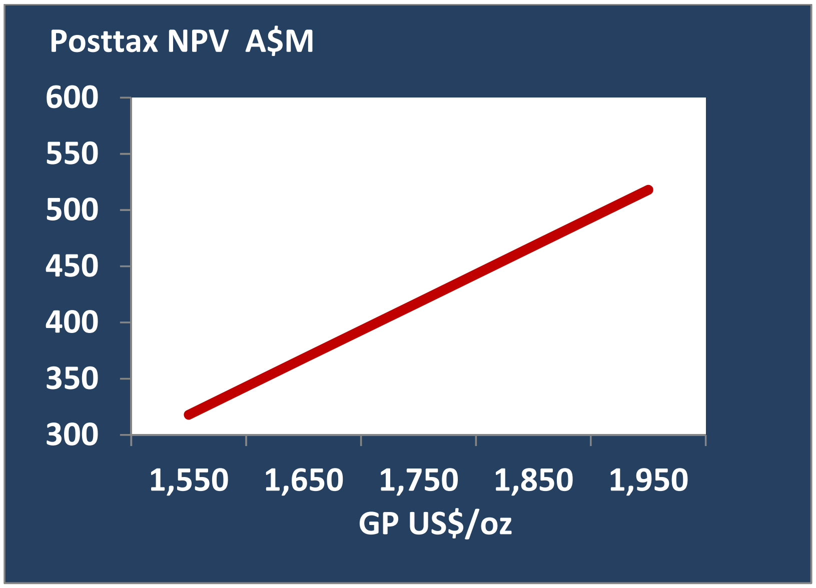
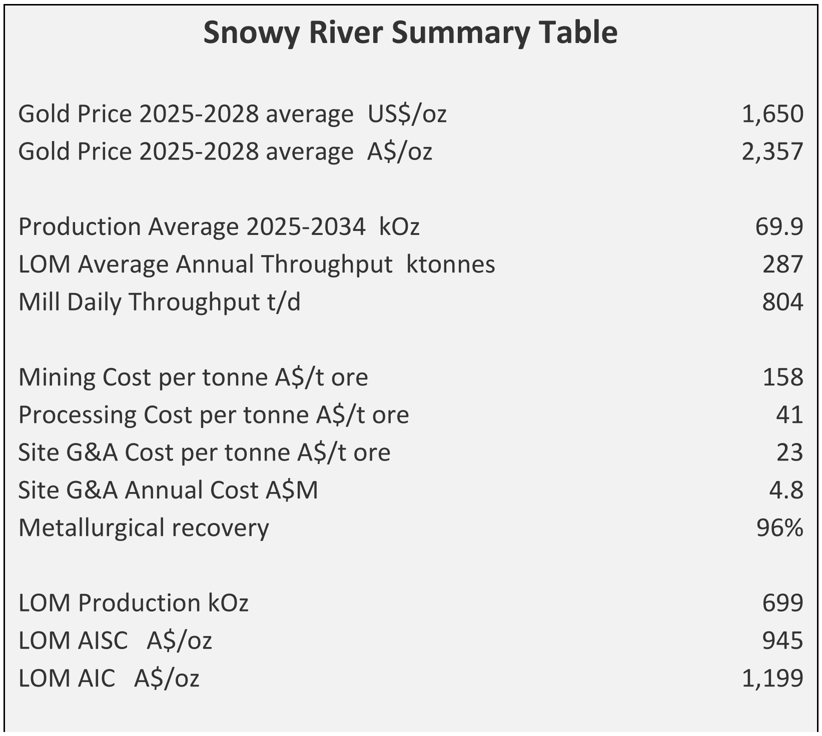


Table 12.4 Base Case Economic Results



Snowy River Summary Table

Gold Price 2025-2028 average US$/oz	1,650
Gold Price 2025-2028 average A$/oz	2,357
Production Average 2025-2034 kOz	69.9
LOM Average Annual Throughput ktonnes	287
Mill Daily Throughput t/d	804
Mining Cost per tonne A$/t ore	158
Processing Cost per tonne A$/t ore	41
Site G&A Cost per tonne A$/t ore	23
Site G&A Annual Cost A$M	4.8
Metallurgical recovery	96%
LOM Production kOz	699
LOM AISC A$/oz	945
LOM AIC A$/oz	1,199

Capex from Jan 2023 A$M	97
Sustaining Capex A$M	35
Closure Cost A$M	19
Base gold price US$	1,650
Project NPV(5%) A$M	368
Project IRR	45%
LOM Estimated Tax Paid A$M	209

12.2. Income Tax Assumption

Net income from the Snowy River 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%.

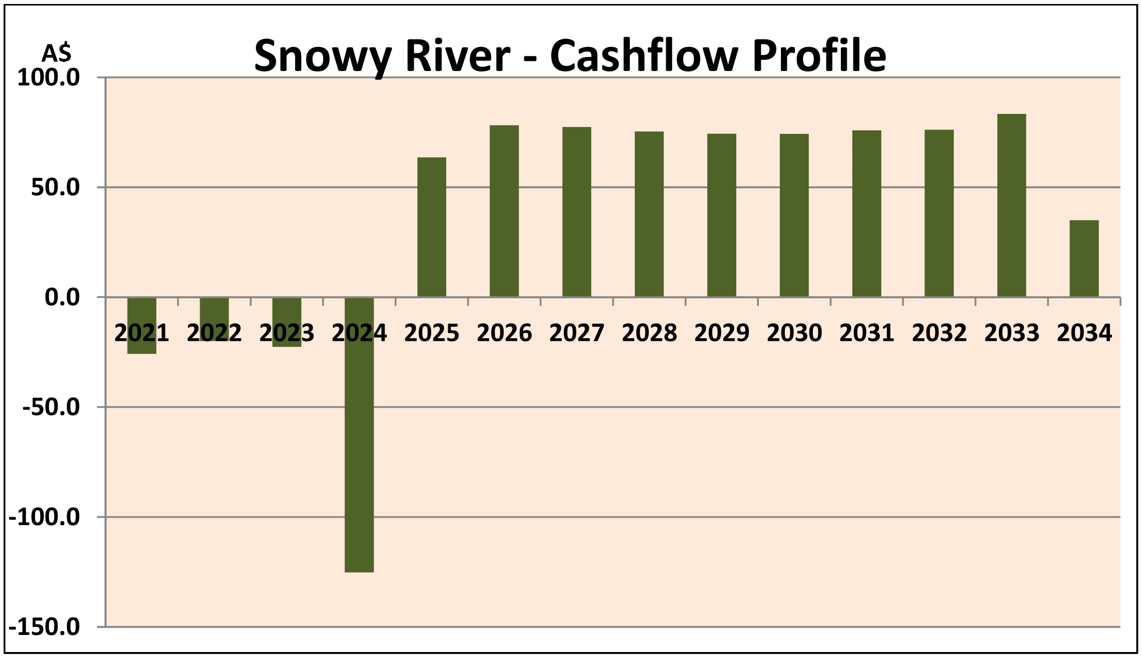
Exploration expenditure is tax deductible in the year in which it is incurred. Development expenditure (including, in some cases, expenditure on exploration assets that subsequently continues to be used in the development and operational phases of a project) is spread for tax deduction purposes. The 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.

The Mining One economic evaluation assumed a base case scenario of 970 koz Au. FML has assumed the current Inferred resource of 730 koz as its base case and considers the 1m oz case as an upside scenario (not included in this ITR).

The FML analysis assumes a base gold price of US$1,650/oz and applies normal capital deductions to calculate the New Zealand taxation using a rate of 28% of net income. Included in the Project economics is the US$30M purchase price in cash to OGC and the A$11M estimated cost of purchasing the RoyalCo 3% royalty.

The FML base case scenario of 699koz produced at AISC of US$662 (TC US$839/oz), the Project has an NPV (5%, $1650) of US$258M, and IRR of 45%.

Figure 12.2 Snowy River base case undiscounted cash flow (FML analysis)



Note: 2024 cashflow includes purchase price of Project (A$42M) and RoyalCo royalty (A$11M)

12.3. Capital Costs

The base case scenario for the Mining One study has a three year 'pre-production' period that includes development of the twin decline, diamond drill platform and ore definition drilling program, totaling capital expenditure in the initial three years of the project of approximately US$70M.

Capital costs used in the Mining One study were obtained through several sources including equipment quotations, first principal derivations, other relative operations and cost factoring and are deemed to be within an accuracy of ±50%. FML updated that initial capital requirement in 2022 to approximately US$100m to reflect changes in scope and inflation since 2018.

Table 12.5 Estimated project capex comparison

	PEA (US$M)	Mining One (US$M)	FML Analysis (US$M)
Pre-production			
Infrastructure	8		7
Processing	21	31	34
Mining	30	28	42
Management and Indirects	7	9	7
Contingency	10 (15%)	2 (10%)	10 (10%)

Subtotal	76	70	100
LOM Sustaining capex	78	68	36
LOM Project Capital	154	138	136

Note: Exchange rates used vary from 2014 PEA to 2018 Mining One Study to 2022 FML analysis

12.4. Operating Costs

Operating costs used in the Mining One study were derived from supplier quotations and the use of first principle derivation. FML updated operating cost estimates in April 2022.

Table 12.6 Snowy River Project Operating Costs

Item	OPEX	PEA	Mining One	FML Analysis
Mining	US$M	180	348	318
	US$ / Ore t	154	86	111
Processing	US$M	49	102	82
	US$ / Ore t	42	25	29
Site G&A	US$M	22	34	47
	US$ / Ore t	19	8	16
Selling costs	US$M	3	Inc. in G&A	5
	US$ / Ore t	2	Inc. in G&A	1
Totals	US$M	254	484	452
	US$ / Ore t	217	119	157

Note: Exchange rates used vary from 2014 PEA to 2018 Mining One Study to 2022 FML analysis

Based on the Mining One estimate, the 2018 unit operating cost for mechanized mining was US$86/t, below the OGC 2014 estimate of US$154/t for air-leg (resuing) mining. The labour required for an air-leg operation significantly increases the operating cost for this method. In order to reach the desired production capacity, the PEA requires on average 22 air-leg miners per shift. Mechanized mining will produce 2.5 times the production capacity with 20 miners per shift. FML has adjusted the mining cost to US$111/t to reflect 4 years of inflation.

Table 12.7 Unit operating cost per tonne of ore mined comparison

	PEA (US$/ore t)	Mining One Study (US$/ore t)	FML Analysis (US$/ore t)
Diesel	5.00	3.57	5.40
Explosives	8.00	1.74	2.25
Services	22.00	0.19	0.65
Pastefill	-	9.41	12.15
Ground Support	6.00	1.44	1.85
Drill Cons	8.00	4.38	5.65
Electrical	3.00	7.91	10.20
Equipment Op	15.00	12.70	15.50
Manpower	82.00	36.75	47.50
Grade Control	3.00	7.51	9.70
Total	**154.00**	**85.60**	**110.85**

12.5. Processing Plant Costs

A Scoping Study level costing estimate for a gold processing plant on site at Snowy River was supplied by Mintrex. A proposal was initially put forward for the PEA with several changes applied relative to the current scenario:

- Plant throughput increased to 300,000 tpa;
- Bulk of mine tailings being returned underground for paste backfill; and
- Process flowsheet developed at equipment unit level as opposed to the complete Gekko Python process package proposed in the PEA.

Operating cost has been evaluated in more detail than capital cost to an estimate closer to ±25-30%. An operational schedule of five days a week, 24 hours a day has been assumed for the Snowy River processing plant. Planned maintenance has been accounted for on Saturday dayshift to address issues identified on a weekly basis. This will consist of processing plant operators, maintenance personnel and administrators. The operating roster is expected to be 10 x 12 hours shifts per week for 120 hours per week. Three crews have been costed working a two-week on one-week off roster. A 10% contingency has been applied to account for leave and sickness coverage. For the Mintrex/Mining One study, salaries were based on 2018 New Zealand processing operations plus an on-cost of 13.5%. Note: NZD:USD exchange rate of 0.64 applied.

Table 12.8 Processing Plant Operating Costs

Operating costs	Annual (US$M)	US$/t (processed)	US$/oz. (recovered)
Plant labour	2.15	7.15	29.68
Maintenance	1.07	3.55	14.74
Mobile equipment	0.40	1.34	5.59
Power	1.76	5.87	24.35
Consumables	1.47	4.90	20.33
G&A	0.69	2.31	9.61
Total	**7.50**	**25.12**	**104.30**

A comparison of the unit processing cost of the two studies for the Snowy River project is shown in Table 12.9. Based on the information supplied by Mintrex, the unit processing cost for the Mining One study is US$25.12/t, below the processing cost of US$42.07/t proposed in the PEA, an estimate considered logical, given the throughput difference. G&A costs were excluded from the PEA as they were assumed part of the owner's costs. The lower ore process tonnage in the PEA has the greatest impact on the labour unit cost compared to the Mintrex/Mining One estimates. In 2022 FML adjusted the unit cost to A$41/t (US$29/t).

Table 12.9 Unit processing cost per tonne of ore mined comparison

	PEA	Mintrex
Maintenance	5.00	3.55
Power	4.08	5.87
Labour	22.18	7.15
Consumables	0.93	4.90
Reagents	3.21	-
G&A	-	2.31
Liners and Grinding	4.57	-
Assay Costs	2.10	-
Mobile equipment	-	1.34
Total	**42.07**	**25.12**

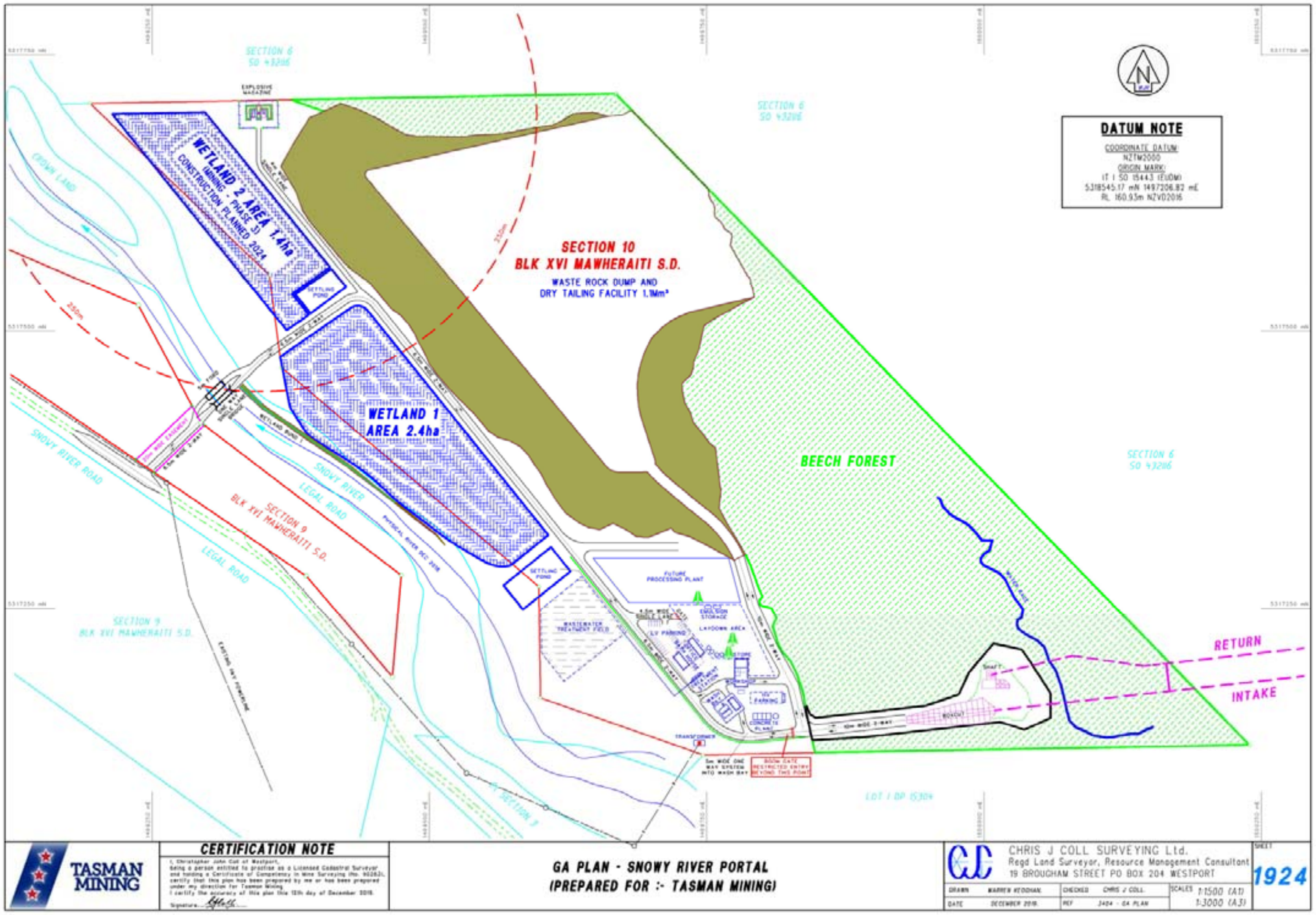
13 SURFACE WORKS

All required surface infrastructure for the Snowy River project will be newly constructed and will include but is not limited to;

- Processing plant;
- waste rock stack incorporating co-disposal of dry filter cake from ore processing ;
- pastefill plant;
- administration buildings;
- maintenance buildings;
- vehicle parking;
- a decline portal;
- power and water supply;
- water filter/water treatment;
- Explosive magazine; and
- Snow River bridge.

An overview of the surface infrastructure at the proposed Snowy River site is shown in Figure 13.1. The surface layout has been designed taking into consideration mining lease boundaries, overlying topography, social and environment factors. Allowances have only been made for the cost of the pastefill plant and the magazines in the economic model.

Figure 13.1 Surface infrastructure overview at Snowy River site



Earthworks on site commenced in July 2019. Tree clearing was completed mid-August and earthworks and decline portal preparation continued through 2020. Safety Management Systems including Principal Hazard Management Plans (PHMP's) and Principal Control Plans (PCP's) were implemented in January 2020. In December 2020 the twin decline development commenced. As of July 2022 approximately 5,000 linear metres of development have been completed (4,500 m of decline plus 500 m of cross-cuts). Infrastructure in place includes road to the site, bridge across the Snowy River, administration offices with satellite and mobile phone connections, employee parking, change house, explosives magazine and mobile equipment workshops.

In progress are the water treatment pond excavations and lining, water treatment plant, waste rock stockpile incorporating tailings disposal area, and preparation of plant site.

13.1. Power

Power (initial 500kW allocation) is supplied to site by Rutherford via power poles installed along Snowy River Road. NZ grid power was connected to site in late 2020 and energy supplied to site is generated by hydroelectricity.

Without the need for significant infrastructure upgrade, 4MW of power will be available to the Snowy River site. The processing plant will require 3.2 MW (source: Mintrex).

Underground mining power requirement totals 2 MW. A substation and/or potential infrastructure upgrades may be necessary in the future.

14 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 Snowy River 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 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.

14.1. Environmental and Social impacts of the Snowy River 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.

14.2. 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 water quality criteria for in direct discharge have been developed with input from New Zealand based water specialists and consultation with the local Iwi.

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 in the passive water treatment system. The cyanide detox circuit is also expected to precipitate some metals from the waste stream. The remaining contaminants will be managed through the planned active water treatment system

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 or taken underground as paste fill. Sulphide tailings from the detox circuits will be dewatered and used in pastefill for 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 indirect discharge back to the Snowy River.

14.3. 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 contemporary water quality standards consistent with New Zealand Regional and National Guidelines.

14.4. Terrestrial Ecology

Existing facilities such as the waste rock dump, water storage pond, silt control drains, magazine and ancillary infrastructure was 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.

The current surface footprint has required the clearing of 10.47 hectares of land of this there was only 0.73 hectares of the beech forest for the portal and box cut area, the remaining areas cleared were gorse and broom.

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 FML/OGC's advisors to be minimal. Any impact will be mitigated by a 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.

14.5. 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 Ione 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 passive water treatment ponds. 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.

The visual effects were reviewed during 2021 by a New Zealand landscape architect and Regional plan guidelines have been built into the final Waste Rock Stack design prepared by PDP in 2022.

14.6. 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 does not affect either of the two identified archaeological features: the historic water race and an adit.

14.7. Local Infrastructure Effects

FML 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.

14.8. Subsidence

The potential for subsidence was reviewed comprehensively by OGC 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 Snowy River Project would not have significant environmental effects.

14.9. 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.

The waste rock stack is consented under the existing Resource Consent and Federation made a decision to install a compacted clay liner on the base the waste rock stack to protect ground water, the compacted clay liner is based on an engineering design by PDP.

PDP have prepared an amended waste rock stack design to allow for the depositing of processing waste in the centre of the stack during the processing phase of the operation. The existing waste rock stack design has been subject to at least geotechnical reviews.

14.10. Traffic Effects

The proposed Snowy River Mine 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.

During 2021, Stantec were engaged to review the potential traffic impacts from onsite processing, this report was provided to the New Zealand Transport Agency and District Council. The overall reduction in heavy vehicle traffic due to onsite processing was considered an improvement from the current consented environment.

14.11. 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.

14.12. Gravel Extraction

During the early works construction gravel was extracted from the Snowy River to assist with the installation of the access bridge and use in civil construction operations prior to the installation of the site buildings.

The Project is fortunate to have access to crushed waste rock which can be used for road base wherever possible.

FML is engaging with a number of interested parties and Government agencies in relation to the removal of waste rock for use on public roads throughout the West Coast.

14.13. 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.

During 2021, consultant PDP was engaged to undertake a technical review of potential emissions to air during the processing phase of the project, this report outlined the mitigations required to manage potential emissions and these will be documented in the air management plan for the operation.

14.14. Emissions to Air

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. During 2021, consultant PDP was engaged to undertake a technical review of potential emissions to air during the processing phase of the project, this report outlined the mitigations required to manage potential emissions and these will be documented in the air management plan for the operation. Greenhouse gas emissions were calculated using Australia NGER guidelines and New Zealand emission factors. NZ grid power to site is sourced from hydroelectric power generation and Federation Mining is also considering installing a local run of river hydroelectric plant to supply the mine.

14.15. Blasting

A detailed 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 Environmental and vibration monitoring currently occurs to ensure blasting operations do not have an impact on nearby residents. The results of the environmental monitoring are provided to the Regional Council as required under the Resource Consent.

14.16. Noise

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. During 2021 Noise levels were assessed for the production phase and a report was prepared for the Regional and District Council which indicated a minor increase in noise for two nearby landholders from mobile equipment works on the Waste Rock stack. A management plan will be prepared to ensure the impact of noise is minimised for our nearby residents.

14.17. Rehabilitation

Rehabilitation is an integral part of all mining operations at the Snowy River Mine and a key issue considered in all mine planning. FML has prepared a cost estimate and closure plan for the life of mine for the Snowy River operation with the assistance of a New Zealand based consulting firm.

An existing environmental bond is held by the West Coast Regional Council which is reviewed annually and assessed by a third party. This bond is currently NZ$360,000, anticipated to increase to NZ$3,000,000 when the mine reaches full production, and will be adjusted down as progressive rehabilitation is completed. Environmental bonds remain in place for 10 years post the conclusion of Resource Consents at which point FML has proposed the establishment of a post relinquishment bond to the value of NZ$200,000 held by the Regional Council to ensure the cost of monitoring or maintenance on the passive water treatment ponds. The project does not require disturbance of any new areas and will use existing, already disturbed land around the existing surface infrastructure.

Approach

The approach to rehabilitation taken for the Snowy River Mine will be progressive and includes:

- Careful design and construction of the waste rock stack to integrate with the existing landform character of the area;
- Keeping the surface area disturbed to a minimum while remaining compatible with day-to-day operations;
- Rehabilitating the site to a safe and stable condition as soon as practical;
- Removal and restoration of the haul roads used during the project if these are no longer required for future projects and the surrounding land is also rehabilitated;
- Containing and treating all contaminants on-site in such a manner that they do not pose a long-term safety or environmental hazard; and
- Achieving a suitable and sustainable post-mining land use.

Implementation

The implementation of the closure strategy at the conclusion of operations will include:

- The access to the portals secured for safety;
- The surface infrastructure will be removed, and the site rehabilitated as per agreed consent conditions;
- The bridge and site access road will be retained as part of the land access agreement;
- The water treatment ponds to provide ongoing management of site run off; and
- The waste rock stack will be rehabilitated with native plants and integrated with existing landform character.

FML will work with OceanaGold to transition the existing partnership with the Department of Conservation on the pest management project in adjoining landholdings from mid-2024 and for the duration of production.

Landscape Mitigation and Remediation at Project Closure

The waste rock stack is located in front of a naturally elevated landscape which will allow the stack to be keyed into the existing topography, and once rehabilitated, appear to be part of the natural hill side. At mine closure, the mine portals will be sealed, and defunct sections of the access road (not required by the landholder) removed and restored during the closure phase of the project. The surface structures will be removed, and all tunnels will be secured to ensure public safety. All ancillary infrastructure not required to remain under consent or land access agreement will be removed. If there is subsurface subsidence on the slopes above the portal, any large slump scraps that arise will be contoured and secured for safety.

FML has a strategy to compensate for the temporary removal of trees from the Snowy River Mine site for the purposes of mining operations. Native trees will be planted progressively on the perimeter of the site to provide compensation and assist with minimising visual effect from surface operations. FML has partnered with the “Trees That Count” project to support the planting of native trees on the West Coast in cooperation with community planting groups. To date FML has sponsored the planting of around 500 native trees on the West Coast which will assist to offset 113 tonnes of CO2 emissions. FML has a target of sponsoring “Trees That Count” or similar program with the aim of enabling the planting of 1000 trees per year for each year of production as part of their sustainability commitments. The Company acknowledges that climate related risk has the potential to impact the businesses and communities they operate within. FML will work to identify and actively manage climate change impacts, both in terms of physical and transition risks present in their operations.

14.18. Contribution to Tourism

The Project has the potential to add to the ‘story’ told by the tour operators of historical, current and future mining of the Reefton Goldfields.

FML has been working with the local tourism information centre and School of Mines to improve facilities for visitors and provide regular information for tourist on the mining operations

14.19. Other Socioeconomic Benefits

The Snowy River 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.

FML will continue to provide support for local community groups to assist their service delivery and support of the communities in which we operate.

14.20. Other National Economic 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.

14.21. 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 with the workforce expected to grow to 150 staff, 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.
- Creation of a passive water treatment system which 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.
- FML will manage noxious plant species and replace them with pasture and other vegetation appropriate to the location.
- FML has committed to planting 1000 native trees per year offsite on the West Coast for the duration of the production phase of the operation.

15 PERMITTING

OGC, as the registered owner of the Project, holds the permits that have been exercised under the project deed. FML is listed as the mine operator under New Zealand law.

15.1. Mining Permit

Almost all minerals in NZ are owned by The Crown and as such The Crown Minerals Act 1991 sets out the broad legislative framework for the issuing of permits to prospect, explore and mine Crown-owned minerals within New Zealand.

An application for a Minerals Permit (60473.01) was granted to OGC dated 19th December 2018. This permit has been granted for a period of 20 years from that date and is classified as a Tier 1 permit (complex higher return mineral activities). Conditions of the permit require the permit holder to pay annual permit fees, royalties and obtain relevant consents before commencing the operations. There is also a minimum Work Programme with information that is required to be submitted to NZP&M (New Zealand Petroleum and Minerals) within certain timeframes, which include the decline development and diamond definition drilling within 48 months and a mine feasibility study within 72 months of commencement

A Mineral Permit can be extended by applying to NZP&M at least 6 months prior to the expiry and can be extended by such time as the Minister considers reasonable to enable the permit holder to economically deplete the discovery. This is a common occurrence in NZ.

15.2. 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 Snowy River 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.

16 RESOURCE CONSENTS

Within New Zealand most activities are permitted by the issuance of a Resource Consent unless those activities are already classed as a “Permitted Activity” by the respective Regional or Local Council. The physical location of the project/ activity determines which councils will review and approve any activities that use natural resources, such as land, water, air, etc.

The Resource Management Act 1991 details all these requirements. Typically, the ‘Regional Council’ will manage those aspects such as Land use, Air and Water, including the taking of and discharging to any of these resources. The local councils focus on the management of site access and impacts to residents, neighbours and other effected parties, they also manage site vegetation/ clearing and the rehabilitation of the site post all activities.

OGC applied for resource consents to the West Coast Regional Council (RC13042) and the Buller District Council (RC130025) associated with the location of the surface infrastructure for the Blackwater Gold Project dated the 30 July 2014 and 20 May 2014 respectively. Under the consents, the consent holder is required to provide the following current management plans:

- Contingency and response plan;
- Rehabilitation plan;
- Construction and operational plan;
- Water management plan;
- Air management plan; and
- Annual work plan.

The following documents were provided by FML to the Buller District Council:

- The first Annual Work Plan (AWP) covering the period of 1st of March 2019 to 29th February 2020 (revised version provided 8th March 2019)
- Construction and Operational Management Plan
- Contingency and Response Management Plan
- Rehabilitation Management Plan
- Rehabilitation Costs Assessment undertaken by Lane Associates Ltd

The plans supplied to the West Coast Regional Council are as above with the addition of:

- Water management plan
- Air management plan and
- Environmental monitoring program.

The AWP has been approved by both the Buller District Council and the West Coast Regional Council This effectively “Exercised the Consents” and thus they are now active and will remain current until the following:

- Buller District Council Consent RC13025 – The Mine Operator provides 6 months’ notice of Intent to Cease
- West Coast Regional Council RC13042 – Current Expiry date of 1 January 2034. The consent can be extended by application to the WCRC at least six months before the expiry date
- A Rehabilitation Bond for NZ$370,000 is lodged with WCRC (held by WCRC on behalf of both councils)

There is a formal requirement to provide an annual review of the previous year’s Annual Work Plan, a submission of the next 12 months work plan, a summary of all environmental monitoring for the previous 12 months and a review of the rehabilitation bond calculation based on any further disturbances or areas that have been rehabilitated. These reviews have occurred annually as required under the consent.

The other plans are reviewed, and any updates made are required to be supplied to the respective councils.

The consents are able to be transferred in whole or part to another operator or person at anytime.

16.1. Future Consents

FML has been preparing a Resource Consent Application over the past 12 months to build and operate a processing plant onsite at Snowy River. The application was submitted to West Coast Regional Council and Buller District Council on the 17th of June 2022.

FML understands the importance of being a responsible sustainable mining company in New Zealand and has been actively engaging with the community and stakeholders on plans for the future of the Snowy River Mine over the past year. FML has engaged New Zealand based environmental and technical specialists to assist in preparing the Resource Consent Application to minimise impact on the environment while ensuring maximisation of the economic opportunity for the West Coast.

The Snowy River workforce is expected to grow from 50 to 150 personnel during production which will also bring greater opportunities for partnerships with local New Zealand businesses.

The following companies and organisations assisted in the preparation of technical reports and the Resource Consent application,

- OceanaGold – the current consent holder
- Mitchel Daysh- Resource consent management advisor
- Pattle Delamore Partners (PDP) – Water management, Waste Rock stack design & air emissions
- Mine Waste Management – Geo chemistry and water management
- Tonkin & Taylor – Noise and vibration
- Ryder Consulting – Water quality,
- Stantec – Traffic
- Origin Consulting – Heritage
- Frank Boffa - landscape design
- Lane Associates – Closure planning
- Environmental Legal – Legal Advisor

The Regional and District Council have reviewed the submitted Resource Consent application and FML has had initial dialogue on draft conditioning with both councils while the application is reviewed by nominated affected parties Department of Conservation and the local Iwi group.

Under the Resource management Act, affected parties have the opportunity to review the Resource Consent application and all associated technical reports. All affected parties have undertaken a site visit and are currently reviewing the application documentation prior to providing the respective councils with feedback.

It is anticipated that this process will be concluded by the end of calendar year 2022 and subject to affected party support amended consents conditions could then be issued by the Regional and District Council.

16.2. Consented Activities

Table 16.1 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. 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 16.2 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 Snowy River 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 storm water around disturbed areas to silt ponds and to divert clean storm water 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, storm water 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).

16.3. 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.

16.4. Other Environmental Consents

The company has not identified any archaeological sites which would be affected by the proposed Snowy River Project, for which authorities would be required from the Heritage New Zealand. This review was completed in 2021 by an New Zealand based consultancy and provided to the regulator.

16.5. Environmental Bonds

The company have existing resource consents which require the lodgement of an environmental bond with the Regional and District Councils 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 will be reassessed annually and increased or decreased to take into account activities to date and forecast activities for each successive year of operations.

The current bond for Snowy River mine is jointly held by West Coast Regional Council and the Buller District Council to the value of NZ$370,000.

The company will be required to maintain bonds in place for the applicable bond sum applying from year to year. FML is able to furnish all necessary bonds based on its proposed activities.

16.6. 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 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.

17 LEGAL AGREEMENT BETWEEN OGC AND FML

OceanaGold (New Zealand) Limited (OGNZL) and FML entered into an agreement, *The Blackwater Project Deed,* dated the 3rd July 2018. In this agreement, OGNZL has exploration rights and assets relating to the Blackwater project area. OGC agreed to allocate US$3,000,000 of funds to FML to finance the establishment costs required to further the Snowy River Project to a point where further funds can be secured to begin construction of the underground access. Assuming FML is successful in raising the required funding, an option is available for the purchase of OGNZL's remaining share of the Snowy River assets for US$30,000,000.

The deed is comprised of two phases:

- Phase One that commenced from the effective date whereby OGNZL was obliged to provide FML with the seed funding for the project. OGC also provided access to personnel and assist in the application to convert the exploration permit into a mining permit. Phase One ended upon the commencement of Phase Two.

- Phase Two commenced after FML raised sufficient funding for construction of the twin decline. This was confirmed in writing by OGNZL to FML that the above conditions were satisfied. During Phase Two, OGNZL has continued to provide access to its personnel. FML began mining development operations (the twin decline) and must meet project deliverables as outlined in the deed schedule. Phase Two will end hence terminating the deed if FML exercises the option to purchase the Snowy River asset (Decision to Mine) or end of business on 31 December 2023.

Termination of the deed is also permissible following failure to meet key milestones, breach of obligations or a change in key personnel. Upon termination of the deed, neither party will have any claim against the other party with the proviso that there is no breach of the project deed.

Table 17.1 Project phase outline

Project Phase	Tasks
Phase 1	Seed funding Surface works and planning Secure Phase 2 funding
Phase 2	Develop decline and diamond drill platform

	Diamond drilling program Resource estimation Technical study
Phase 3	Decision to Mine Funding Process plant construction Production development Fleet acquisition

18 CONCLUSIONS

Notwithstanding the confidence limitations associated with an Inferred Resource, there is considerable evidence to suggest continuity of the Birthday Reef approximately 1,000 metres along strike and 650 metres down dip from the historic 16 level of the Blackwater Mine. The anticipated continuity of geology, grade, width and conversion of Mineral Resources to Mineral Reserves will be confirmed by completion of the twin decline, development of 3 new levels and a drilling program.

The 2014 PEA, 2018 Mining One study and this ITR suggests that the Snowy River Project is technically and economically viable.

The Snowy River Project has very good access and supporting infrastructure. The Birthday Reef is amenable to conventional modern underground mining methods.

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

The project has a base case 10 year mine life beginning 4Q24. The base case scenario is robust, returning positive post-tax NPV results over a range of inputs.

Environmental monitoring, baseline studies and site investigations are ongoing at the Snowy River Project site. Extensive consultation to date has included meetings with local and regional councils, local land-owners and regulatory agencies.

FML, through OGC, received the required consents to construct the twin decline, waste dump and water treatment facilities, undertake drilling and underground mining operations. Documentation to achieve resource consents to allow for ore processing and tailings co-disposal facilities was submitted in June 2022.

18.1. Opportunities and Risks

The opportunities to improve the project include:

- Resource growth – the Birthday Reef at the Snowy River 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.
- 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.
- Potential for ore sorting – this will be confirmed after on ore strike development is achieved in 2023 and final plant design is completed.

The key risk factors are:

- The OGC 2014 PEA, Mining One 2018 Study and FML 2022 Analysis are preliminary in nature. They include Inferred Mineral Resources that are too geologically speculative 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 proposed project development 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 a minor 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 limited information about the vertical geometric regularity of the Birthday Reef (historical sample data suggests that down-plunge grade continuity will be reasonable). 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. 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 or shipping the concentrate offshore;
- 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 HW 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 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 will be higher as depth continues from 800m to 1,600m below surface. 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;
- 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 2014 PEA.

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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 OGC 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 Snowy River 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 core shed until shipped to Australia for assay using local courier and FedEx.
Audits or	• *The results of any audits or reviews of sampling techniques and data.*	• No external audits or reviews were completed on the sampling technique.

Criteria	JORC Code explanation	Commentary
reviews		Internal discussions were had with experienced OGC 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. FML/OGC have 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. Documentation pertaining to the variation application was filed in June 2022. FML/OGC holds sufficient rights in the Snowy River 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; • Construct the proposed 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

Criteria	JORC Code explanation	Commentary
		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 FML/OGC, 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 Snowy River 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 FML/OGC will not require any access arrangements with the owners of the land through which the Snowy River 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 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), FML/OGC have 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-

Criteria	JORC Code explanation	Commentary
		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 modified work 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 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*

Criteria	JORC Code explanation	Commentary
		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.* 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.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td>Drill hole Information</td>
<td>

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:

easting and northing of the drill hole collar
elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar
dip and azimuth of the hole
down hole length and interception depth
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.

</td>
<td>
<table>
<tr><th rowspan="2">Hole ID</th><th colspan="2">NZMG Coordinate*</th><th rowspan="2">Elevation (masl)</th><th colspan="2">Hole Orientation</th><th rowspan="2">Daughter Depth Start (m)</th><th rowspan="2">Final Depth (m)</th><th rowspan="2">M Drilled</th></tr>
<tr><th>East</th><th>North</th><th>Azimuth (Grid)</th><th>Dip</th></tr>
<tr><td>WA10</td><td>2412835</td><td>5879174</td><td>438</td><td>90</td><td>-90</td><td></td><td>686.9</td><td>687</td></tr>
<tr><td>WA11</td><td>2412829</td><td>5879172</td><td>438</td><td>90</td><td>-65</td><td></td><td>1,171</td><td>1171</td></tr>
<tr><td>WA11A</td><td></td><td></td><td></td><td></td><td></td><td>644.4</td><td>1,011</td><td>527</td></tr>
<tr><td>WA21</td><td>2412888</td><td>5879439</td><td>528.681</td><td>83.5</td><td>-63.5</td><td></td><td>1378</td><td>1,378</td></tr>
<tr><td>WA21A</td><td></td><td></td><td></td><td></td><td></td><td>1,264.3</td><td>1324</td><td>60</td></tr>
<tr><td>WA22</td><td>2412888</td><td>5879439</td><td>528.68</td><td>65</td><td>-56</td><td></td><td>1121</td><td>1,122</td></tr>
<tr><td>WA22A</td><td></td><td></td><td></td><td></td><td></td><td>809.2</td><td>847</td><td>38</td></tr>
<tr><td>WA22B</td><td></td><td></td><td></td><td></td><td></td><td>815.2</td><td>863</td><td>47</td></tr>
<tr><td>WA22C</td><td></td><td></td><td></td><td></td><td></td><td>814.3</td><td>1675</td><td>861</td></tr>
<tr><td>WA22D</td><td></td><td></td><td></td><td></td><td></td><td>1,385.9</td><td>1641</td><td>255</td></tr>
<tr><td>WA23</td><td>2413278</td><td>5880086</td><td>540</td><td>143</td><td>-55</td><td></td><td>36</td><td>36</td></tr>
<tr><td>WA24</td><td>2413278</td><td>5880086</td><td>540</td><td>143.5</td><td>-51.5</td><td></td><td>363</td><td>364</td></tr>
<tr><td>WA25</td><td>2413278</td><td>5880086</td><td>540</td><td>140</td><td>-62</td><td></td><td>1282</td><td>1,282</td></tr>
<tr><td>WA25A</td><td></td><td></td><td></td><td></td><td></td><td>1036.2</td><td>1205</td><td>169</td></tr>
<tr><td colspan="7">* GPS co-ordinates</td><td>Total</td><td>7,997</td></tr>
</table>
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</tr>
</table>

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

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

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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.

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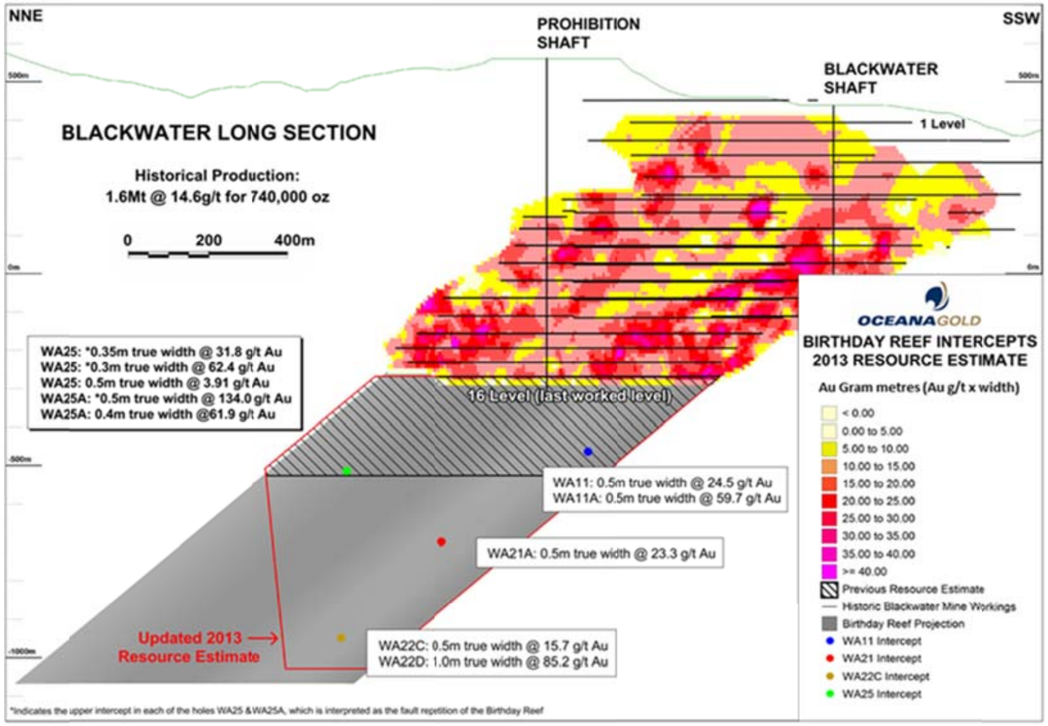
Criteria	JORC Code explanation	Commentary
	equivalent values should be clearly stated.	
Relationship between mineralisation widths and intercept lengths	- *These relationships are particularly important in the reporting of Exploration Results.* - *If the geometry of the mineralisation with respect to 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.
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 Project 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; OGC 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 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.

Criteria	JORC Code explanation	Commentary
Dimensions	- *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.*	- 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.  WA25: *0.35m true width @ 31.8 g/t Au WA25: *0.3m true width @ 62.4 g/t Au WA25: 0.5m true width @ 3.91 g/t Au WA25A: *0.5m true width @ 134.0 g/t Au WA25A: 0.4m true width @61.9 g/t Au WA11: 0.5m true width @ 24.5 g/t Au WA11A: 0.5m true width @ 59.7 g/t Au WA21A: 0.5m true width @ 23.3 g/t Au WA22C: 0.5m true width @ 15.7 g/t Au WA22D: 1.0m true width @ 85.2 g/t Au *Indicates the upper intercept in each of the holes WA25 &WA25A, which is interpreted as the fault repetition of the Birthday Reef

Criteria	JORC Code explanation	Commentary
Estimation and modelling techniques	• *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 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.*	• As discussed above, a number of approaches to estimating the resource were evaluated. The approach taken is believed to be the most appropriate. The Snowy River 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 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 OGC 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 Snowy River resource estimate is based on a long sectional polygonal area

Criteria	JORC Code explanation	Commentary
		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 OGC technical staff and external consultants in the preparation of the 2014 PEA.
Environmental 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.	- The Project has approval for 1.1M m^3 waste rock dump, which appears adequate for the base case. In the 2014 PEA it was estimated that additional capacity of up to 25% may need to be consented to accommodate the additional volume associated with co-disposal of tailings.

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.

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		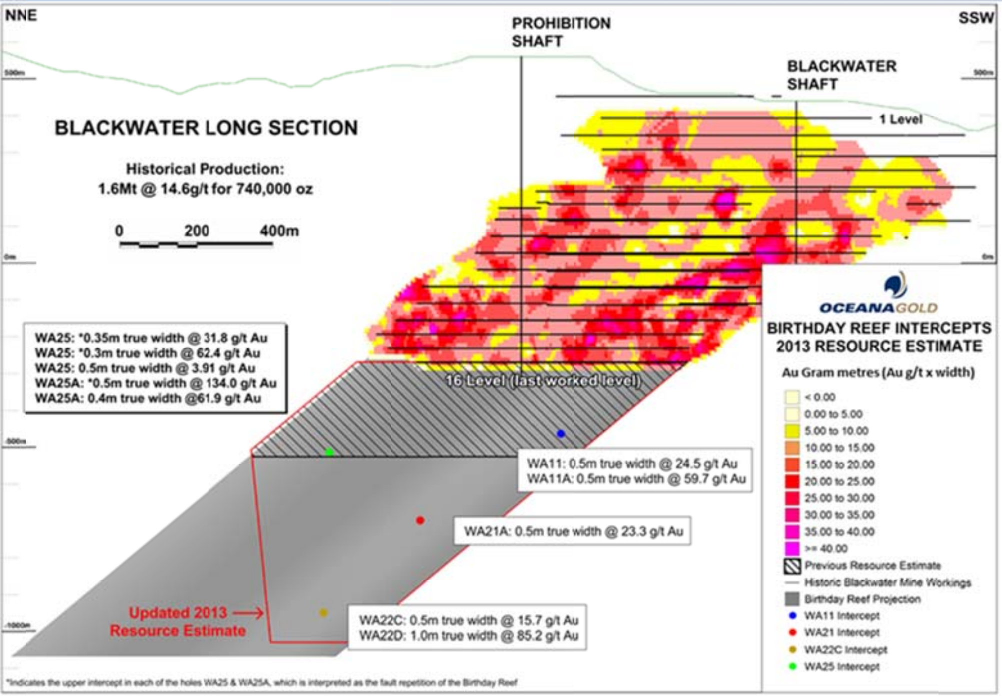 • Appropriate account has been taken of tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data. • Since Mr J Moore's involvement in the Blackwater project, primarily since 2010, there have been many discussions 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

Criteria	JORC Code explanation	Commentary
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 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.*	• 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 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.