Waihi District - Martha Underground Feasibility Study

NI 43-101 Technical Report

Effective Date: 31st March 2021

STU-059-REP-002-0

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Technical Report Certification

The effective date of this Technical Report and sign-off is 31st March 2021.

Trevor Maton — Date 31st March 2021

David Carr — Date 31st March 2021

Peter Church — Date 31st March 2021

Forward Looking Information

This report contains forward-looking statements. All statements, other than statements of historical fact regarding OceanaGold Corporation or Waihi Operations, are forward- looking statements. The words "believe", "expect", "anticipate", "contemplate", "target", "plan", "intend", "project", "continue", "budget", "estimate", "potential", "may", "will", "can", "could" and similar expressions identify forward-looking statements. In particular, this report contains forward-looking statements with respect to cash flow forecasts, projected capital, operating and exploration expenditure, targeted cost reductions, mine life and production rates, potential mineralisation and metal or mineral recoveries, and information pertaining to potential improvements to financial and operating performance and mine life at the Waihi Operations that may result from. All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to such assumptions, the forward-looking statements are inherently subject to significant business, economic and competitive uncertainties and contingencies. Known and unknown factors could cause actual results to differ materially from those projected in the forward-looking statements. Such factors include, but are not limited to: fluctuations in the spot and forward price of commodities (including gold, diesel fuel, natural gas and electricity); the speculative nature of mineral exploration and development; changes in mineral production performance, exploitation and exploration successes; risks associated with the fact that the Waihi Operations is still in the early stages of evaluation and additional engineering and other analysis is required to fully assess their impact; diminishing quantities or grades of reserves; increased costs, delays, suspensions and technical challenges associated with the construction of capital projects; operating or technical difficulties in connection with mining or development activities, including disruptions in the maintenance or provision of required infrastructure and information technology systems; damage to OceanaGold Corporation’s or Waihi Operations reputation due to the actual or perceived occurrence of any number of events, including negative publicity with respect to the handling of environmental matters or dealings with community groups, whether true or not; risk of loss due to acts of war, terrorism, sabotage and civil disturbances; uncertainty whether the Waihi Operation’s will meet OceanaGold Corporation's capital allocation objectives; the impact of global liquidity and credit availability on the timing of cash flows and the values of assets and liabilities based on projected future cash flows; the impact of inflation; fluctuations in the currency markets; changes in interest rates; changes in national and local government legislation, taxation, controls or regulations and/or changes in the administration of laws, policies and practices, expropriation or nationalisation of property and political or economic developments in Canada; failure to comply with environmental and health and safety laws and regulations; timing of receipt of, or failure to comply with, necessary permits and approvals; litigation; contests over title to properties or over access to water, power and other required infrastructure; increased costs and physical risks including extreme weather events and resource shortages, related to climate change; and availability and increased costs associated with mining inputs and labour. In addition, there are risks and hazards associated with the business of mineral exploration, development, and mining, including environmental hazards, industrial accidents, unusual or unexpected formations, pressures, cave-ins, flooding and gold bullion, copper cathode or gold or copper concentrate losses (and the risk of inadequate insurance, or inability to obtain insurance, to cover these risks).

Many of these uncertainties and contingencies can affect OceanaGold Corporation’s actual results and could cause actual results to differ materially from those expressed or implied in any forward- looking statements made by, or on behalf of, OceanaGold Corporation. All of the forward-looking statements made in this report are qualified by these cautionary statements and OceanaGold Corporation and the Qualified Persons who authored this report undertake no obligation to update publicly or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.

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1 SUMMARY

1.1 Overview

This Technical Report has been prepared to conform with the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). The purpose of the Report is to disclose the results of the Mineral Resource and Mineral Reserve estimates and the completed Feasibility Study on Martha underground. It will be lodged with SEDAR in accordance with TSX requirements.

The Waihi Project area is situated within the world class gold mining town of Waihi located on the North Island of New Zealand. The Mineral Resources within Waihi are currently estimated to be 1.83 Million ounces of gold in the Measured and Indicated Mineral Resources categories at an average grade of 4.13 grams per tonne gold and a further 1.4 million ounces of gold in the Inferred Mineral Resource category at an average grade of 4.4 g/t Au. The Mineral Reserves are currently estimated to be 0.62 million ounces gold, supporting a mine life of 8 years.

The previous NI 43 101 technical report for the Waihi operation was filed in August 2020. This technical report prepared in accordance with Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) for the Waihi operation (“Technical Report”) summarises work completed during the past nine months mainly covering the conversion of Mineral Resources to Mineral Reserves for the Martha underground deposit. This report supports Mineral Resources and Mineral Reserves estimates as at December 31, 2020.

OceanaGold is undertaking exploration drilling at MUG and WKP for the purpose of identifying potential further discoveries and resource conversion to increase mining inventories and extend mine plans.

1.2 Introduction

1.2.1 Waihi

Mining has played an important role in the history of the Waihi town since gold was first discovered in 1879. Since then the Martha deposit within the heart of the Waihi town has produced close to 6.9 Moz of Au. Historical underground mining of the Martha veins took place from 1882 to 1952 and an open pit mine extracted ore from the upper portions of the vein system from 1988 to April 2015. When mineralisation was discovered in the Favona deposit located approximately 2 km east of the Martha deposit a modern-day underground mine was developed, and extraction of ore commenced in 2004. This underground mine is still currently in operation having extracted ore from numerous vein systems in-between the Favona and Martha ore bodies including the Trio and Correnso ore bodies (producing approximately 1.1 Moz Au). Ore is processed on-site adjacent to the existing Favona underground portal. Recovery of gold at Waihi is achieved through a conventional SAG Mill-Ball Mill grinding circuit followed by Carbon in Pulp (CIP) leach circuit with a conventional elution and electro-winning circuit.

The OceanaGold, Waihi operation holds the necessary permits, consents, certificates, licences and agreements required to develop and mine the Martha underground mine.

1.2.2 WKP

WKP is an exploration project situated approximately 10km north of the Waihi town. A Mineral Resource estimation has been calculated for this area based on 43, 900 m of diamond drilling. The project is currently at an exploration phase with no permits or consent established for mining.

1.3 Reliance on Other Experts

The authors, Qualified and Non-Independent Persons as defined by NI 43-101, were engaged by OceanaGold to study technical documentation relevant to the Technical Report, to contribute to or review the Technical Report on the Waihi operation, and to recommend a work programme if warranted.

The authors believe the information used to prepare the report and formulate its conclusions and recommendations is valid and appropriate considering the status of the operation and the purpose for which the Report is prepared. The authors, by virtue of their technical review of the project's exploration potential, affirm that the work programme and recommendations presented in the Report are in accordance with NI 43-101 and the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") technical standards.

1.4 Property Description, Location and Ownership

The Waihi and WKP projects are located on the North Island of New Zealand. All naturally occurring gold and silver minerals in New Zealand are owned by the Crown. Rights to prospect, explore or mine for these minerals are granted by permits issued under the Crown Minerals Act 1991 (CMA). Mineral exploration permits provide a permit holder the exclusive rights to explore for the specified minerals in an area. Minerals mining permits grant the holder to exclusive rights to mine for the specified minerals.

1.4.1 Waihi

The Waihi mine is located within the township of Waihi situated within the Hauraki District. The Waihi project is managed by OceanaGold (New Zealand) Limited, a 100% owned subsidiary of the OceanaGold Corporation. All gold mining activities in Waihi including the current underground mining operation, the ore processing plant, tailings facility and the inactive Martha open pit are within the existing Favona Mining Permit 41 808 (MP 41808).

MP 41808 extends across an area of 1572.59 hectares characterised by urban and rural land use. Land ownership is variable including parcels owned by OceanaGold, private landowners and government organisations. Permission is required by the landowner for access to the land.

OceanaGold holds a suite of resource consents from the Hauraki District Council (HDC) and Waikato Regional Council (WRC) relating to mining and associated discharge activities for the Mining Licence and Extended Project areas. Resource Consent for underground mining of the remnant mineralisation around the Martha vein system was granted on the 12th December 2018.

1.4.2 Wharekirauponga

The WKP Project is located approximately 10km to the north of Waihi, held under mining permit 60541 (MP 60541). The project is located on land owned by the Crown and administered by the Department of Conservation (DOC) as a conservation/forest park. Exploration activity requires an access arrangement with DOC in addition to resource consents from district and regional councils. Under the Crown Minerals Act 1991 exploration is restricted to low impact and higher impact activities including, but not limited to drilling, bulk sampling and trenching. All required mineral tenures, access agreements and consents have been obtained for the current work programmes. Known environmental liabilities are managed through stipulated conditions in the DOC access agreement and Regional and District Council Consents.

1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

Waihi town is situated 132 km from Auckland International Airport, an approximately 2-hour drive sealed road. The Waihi town is a large resource centre with a supermarket, two petrol stations and a sizeable retail hub. The township has a population of 4,527 (2017 census). The nearest domestic airport is 66 km to the south in Tauranga city.

The WKP project is remotely situated within the Coromandel ranges. It can be accessed by an easy to moderate hiking track (approximately 1.5-hour hike) from a road end which is an approximately 20-minute drive from the coastal town of Whangamata (or a 45-minute drive along a tar sealed road from Waihi). There is no infrastructure at the WKP project.

The climate within the Coromandel Peninsula is temperate. Mean monthly temperatures range from 8.9^{0}C in July to 18.9^{0}C in January. The Coromandel ranges reach over 600 m above sea level in places and run along the length of the Coromandel Peninsula. These ranges receive on average 2000 mm of rainfall per annum, with approximately 31% of rainfall expected within the winter months between June and August and 22% of rain in the summer months between December and February.

Road, rail and air networks cover the country with road transport being the dominant method of passenger and freight transport. Bulk freight is mainly transported by coastal shipping and rail.

New Zealand's system of utilities is extensive. The electricity sector uses mainly renewable energy sources such as hydropower, geothermal power and increasingly wind energy. 82% of energy for electricity generation is from renewable sources, making New Zealand one of the lowest carbon dioxide emitting countries in terms of electricity generation[1].

1.6 Project History

1.6.1 Waihi

Waihi is a historic mining centre. The original Martha mine began as an underground operation in 1879 and by 1952, about 12 million tonnes of ore had been mined to yield 1,217 tonnes of gold-silver bullion. The historic mine extracted four main parallel lodes (the Martha, Welcome, Empire and Royal) together with numerous branch and cross lodes.

Exploration drilling between 1979 and 1984 by Waihi Mining and Development Ltd. and AMAX Exploration Ltd. identified large open pit reserves within the confines of the historic mining area. Following the granting of permits, the Martha Mine open pit operation commenced operation in 1988 as an unincorporated joint venture between subsidiaries of Normandy Mining Limited Group and Otter Gold Mines Ltd. The Otter Gold holding was acquired by Normandy in 2002 and the Newmont Mining Corporation acquired full ownership of the Waihi Mine in 2002 through the acquisition of the Normandy Mining Group. OceanaGold obtained the Waihi property as an operating open pit mine, underground mine and process plant in October 2015.

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[1] Ministry of Business, Innovation and Employment, Energy in New Zealand, 2019 Calendar Edition

The Martha Mine open pit started its operations in 1988 under the Mining Licence 32 2388. The pit produced 22 Mt at 3.1 g/t Au (2.2 Moz.) between 1988 and April 2015 when a pit wall failure suspended open pit mining. There is no open pit mining currently active in Waihi. Mining Licence 32 2388 expired in July 2017 and was amalgamated into the existing Favona Mining Permit 41808.

The Favona Mining permit 40418 (MP 41 808) was granted in March 2004 for a duration of 25 years to mine the Favona ore body situated approximately 2 km east of the Martha ore body. Underground mining resumed in Waihi some 52 years after the closure of the Martha underground mine when the Favona decline was developed in 2004 and the subsequent mining of the Favona, Moonlight, Trio, Daybreak and Correnso ore bodies. An Extension of Land to MP 41 808 was granted in 2006 and 2017. Resource Consent for underground mining of the mineralisation around the Martha vein system was granted on the 12th December 2018. Development of MUG commenced in mid 2019.

1.6.2 WKP

Early prospecting and mining at WKP were attempted between 1893-1897, but only 19 oz. of bullion was recovered from a 14-ton test parcel and mining was soon abandoned. Modern prospecting and exploration ignited again in 1978-1993 by Amoco, BP and others which included 5,500 m of drilling in 23 drill holes. Newmont acquired a controlling interest in the property in 2005 and started reconnaissance geological mapping, sampling, CSAMT geophysics and drilling campaigns targeting high-grade underground minable veins. In 2010, hole WKP-24 intersected the main T-Stream vein containing 156 m at 1.6 g/t Au. Wide spaced follow up drilling confirmed the presence of three prospective vein zones each striking more than 1 km in length, namely the Western Vein, the T-Stream Vein, and the East Graben (EG) Vein. Newmont completed 7,000 m of diamond drilling in 15 holes intersecting locally high-grade Au mineralisation in each hole. Newmont ceased exploration in 2013 and the prospect remained idle until 2016 when OceanaGold acquired Newmont's New Zealand assets. Exploration then continued with additional geological mapping, sampling and geophysics leading up to further diamond drilling (WKP40 to WKP 68). Drilling intersected significant Au mineralisation including but not limited to 7.6 m (true width) averaging 10.84 g/t Au in the first hole (WKP40). Drilling since has highlighted significant gold mineralisation and a Mineral Resource estimate has been calculated. A mining permit was granted over WKP on the 5th August 2020.

1.7 Geological Setting and Mineralisation

Both the Waihi and WKP Projects are located within the Coromandel Peninsula which hosts over 50 gold and silver deposits that make up the Hauraki Goldfield. The peninsula is built up of Miocene to Quaternary volcanic rocks (the Coromandel Volcanic Zone) overlying a Mesozoic basement. It is bound to the west by the Hauraki Rift, a large graben filled with Quaternary and Tertiary sediments, and to the south by volcanics deposited by the presently active Taupo Volcanic Zone (TVZ).

The Coromandel Volcanic Zone (CVZ) is of Miocene to Pliocene in age and formed during three main phases of volcanism. The first phase constitutes the widespread andesites and dacites of the Coromandel group (18-3 Ma). The second phase encompasses the predominantly rhyolitic units of the Whitianga Group (9.1-6 Ma) and the third phase is dominated by Strombolian volcanoes and dykes of the Mercury Bay Basalts (6.0-4.2 Ma). Epithermal veins and hydrothermal alteration are observed within the Coromandel and Whitianga Groups.

The Au-Ag deposits of the Waihi District and WKP are classical low-sulphidation adularia-sericite epithermal quartz vein systems associated with north to northeast trending faults.

Larger veins have characteristically developed in dilational sites in the steepened upper profile of extensional faults with narrower splay veins developed in the hanging wall of major vein structures. Moderate to steeply dipping veins are characterised by 200 m to 2000 m of strike, 170 m to 700 m vertical range and typically 1 m to 5 m vein widths (but up to 30 m locally). The main gold bearing minerals are electrum and silver sulphides developed within quartz veins.

The geological control on mineralisation is well understood and is sufficient to support the estimation of Mineral Resources. The current experience and geological knowledge of the area is also considered sufficiently acceptable to reliably inform mine planning.

1.8 Deposit Types

The deposits discovered in/near Waihi to date are typical of epithermal vein gold – silver deposits. In the opinion of the Qualified Persons (QPs), features that the Waihi deposits display that are typical of epithermal gold deposits include:

- Gold-silver mineralisation is hosted within multiphase quartz veins.
- Host lithologies for veins are hydrothermally altered volcanic units including andesite (Waihi) and rhyolite (WKP),
- The upper portion of veining along the Favona deposit in Waihi contains an intact siliceous sinter sheet.

1.9 Exploration

Work completed since 1986 has comprised surface reconnaissance exploration, geological and structural mapping, geochemical sampling, airborne, ground and downhole geophysical surveys, surface and underground drilling, engineering studies and mine development.

Recent diamond drilling has largely focused on the Gladstone, Martha and WKP deposits. OceanaGold continues to drill in the Waihi area, with 25 km of drilling planned for resource infill and reserve conversion for the Martha Underground project and 6 km planned for the WKP project in 2021.

The exploration programs completed to date are appropriate to the style of the deposit and prospects.

1.10 Drilling

Approximately 268,685 m of diamond core has been drilled within the Martha and Gladstone projects since 1980 (as at February 2021). The WKP project has had 47,139 m of diamond drilling in 100 holes drilled since 1980 (as at February 2021).

Additionally, 86,074 m have been drilled in 4,445 reverse circulation grade control holes during the open pit operation.

Surface holes are collared using large-diameter PQ core, both as a means of improving core recovery and to provide greater opportunity to case off and reduce diameter when drilling through broken ground and historic stopes. All drill core was routinely oriented below the base of the post-mineral stratigraphy by plasticine imprint or using the Ezimark or Reflex core orientation tool.

All drilling data in Waihi is recorded in terms of Mt Eden Old Cadastral grid (MEO). This is the grid utilised for all underground and exploration activity within 3 km of the Waihi Mine beyond which New Zealand Map Grid (NZMG) has been utilised (including WKP). Drillhole collars in Waihi and WKP are surveyed using a total station by a registered professional land surveyor.

All diamond drill core is logged including lithologies, alteration, veining, structure, geotechnical and recovery fields. Core is then photographed, and sample intervals chosen.

1.11 Sample Method & Analysis

The Mineral Resource estimates of individual projects in Waihi and WKP use a combination of sampling techniques including:

- Martha Underground (MUG): DD core, Reverse Circulation (RC) chips from exploration drilling, RC chips from open pit grade control drilling, and grade control channel samples,
- Gladstone Project: DD core, RC chips from exploration drilling, and
- WKP Project: DD core.

Once logged and photographed, any diamond core to be sampled is cut lengthways in half and one half is bagged for analysis and the other retained for storage in a core shed. Since mid-2006, sample preparation of drill core has been carried out at the SGS Waihi laboratory. Some of the WKP core was prepared at the Westport SGS laboratory on the South Island (only holes WKP40-WKP45). Current standardised sample preparation consists of crushing to 80% passing 3.35 mm, rotary splitting to 800 g, then ring pulverising to 90% passing 75 µm. Of the pulverised material approximately 300 g is sent for analysis. Pulps are assayed by SGS for gold by 30 g Fire Assay with AAS finish. Additional analyses for As, Ag, Sb, Cu, Pb, Sb and Zn are also often assayed using a 0.3 g Aqua Regia digest followed by an ICP-MS instrument finish.

Underground channel samples are marked up and chipped off with a hammer by an ore control geology team. Sample intervals are based on geology. Chip samples are sent to SGS for analysis of Au and Ag only.

Drill core QAQC samples include one standard, one blank and one crush duplicate every 17 samples. Underground channel QAQC samples include one standard, one blank, one crush and one field duplicate every channel (on average every 6 samples).

The Waihi protocol requires the QAQC Certified Reference Material (CRM) standards to be reported to within 2 Standard Deviations of the Certified Value. The criterion for preparation duplicates is that they have a relative difference (R-R1/mean RR1) of no greater than 10%. Blanks should not exceed more than 4 times the lower detection method of the assay method. Failure in any of these thresholds triggers an investigation and re-assay.

In addition to routine quality control procedures, umpire assays were carried out at Ultratrace Laboratories in Perth.

The sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation and mine planning purposes. The quality of the analytical data is reliable and sample preparation, analysis, and security are performed in accordance with exploration best practices and industry standards.

1.12 Data Verification

The data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation.

1.13 Metallurgical Test Work

Metallurgical test work and associated analytical procedures were performed by recognised testing facilities, and inhouse OceanaGold metallurgical testing facilities, and the tests performed were appropriate to the mineralisation type. Samples selected for testing were representative from a range of depths within the deposit of the various types and styles of mineralisation within the Waihi and WKP areas. Sufficient samples were taken so that tests were performed on adequate sample mass. Average recoveries have been assumed based on test work completed. These recoveries are appropriate to be used in support of Mineral Resource and Mineral Reserve estimation, based on the drill hole spacing and sample selection.

Historical metallurgical results on the MOP5 deposit support an expected gold recovery assumption of 90% for treatment through the existing process plant flowsheet. Metallurgical recoveries of 90% for WKP and 71% for GOP have been estimated based on recent testwork results. Testwork will continue as part of future evaluation phases for these projects.

Recovery models were developed for each of the Martha underground vein structures based on the leach testwork results conducted on the 2019 and 2020 samples. Multiple Linear Regression (MLR) was used to predict gold recovery with the explanatory variables being gold head grade and arsenic content in the feed. An average recovery of 94.9% was estimated for the MUG Mineral Reserve.

1.14 Mineral Resources Estimate

1.14.1 Reporting Date

Mineral Resources for the Martha Underground, Wharekirauponga, Gladstone and Martha Open Pit are reported as at December 31, 2020.

1.14.2 Qualified Persons

The Mineral Resources quoted here were prepared by, or under the supervision of Peter Church, Principal Resource Geologist for OceanaGold, with assistance from the OceanaGold geology team.

1.14.3 Mineral Resources

The modelling process employed in the grade estimation for all the Waihi projects is performed using numerous Vulcan and Leapfrog processes summarised in the steps outlined below:

1. Input data Validation
2. Update lithological domains, geologic model construction
3. Data selection, Drill hole data selection from an AcQuire drill database
4. Exclusion of unwanted drill holes by data type
5. Flag data files by lithology
6. Composite drill holes to fixed length composites within defined geological boundaries, typically 1 m using length weighting
7. Exploratory data analysis by domain, generation of domain and data type summary statistics
8. Variography
9. Assign top cuts by domain and data type to input data files
10. Block Model construction based upon lithological wireframes
11. Run estimation for all domains for Au, Ag, As, Resource Classification
12. Assign density, mining depletions, back fill grade, stripping of negative values from non-estimated blocks, assignment of grade to dilution domains

13. Classify model

The model is estimated using Vulcan software. Estimations were performed in individual lithological domains using length weighted downhole composites. Vulcan software version 11.0 has been used to construct the Martha Underground, WKP and Gladstone models. MineSight® software version 9.10-01 was used to construct the Martha model.

Sub-blocking with either ordinary kriging (OK) or inverse distance weighting to the second power (ID2) or third power (ID3) methods are used for all underground models. With the data density which exists in Correnso, Martha and the surrounds ordinary kriging, and tetra-unfolding - using ID2 or ID3 estimates both achieve comparable results. The method of unfolding was adopted for the estimation of vein models as a way of dealing with the sinuous character of the veins.

The underground block models are rotated in bearing to align with the dominant strike of the veins and they are run using Vulcan software. Sub-blocking is used to define narrow veins and to maintain volume integrity with the geology solids. The grade estimation for all models is strictly controlled by the geology, with both sample selection and estimation of blocks limited to domains defined by the geology interpretation solids. Gold is estimated using one of the following methods; either - a single pass with a combined channel and drilling dataset; OR - two-pass estimation using a combined dataset with short search range first, then followed by a second pass using drill hole data only with longer search ranges to estimate blocks not estimated in the first pass.

Mineral Resources were classified to Australasian Joint Ore Reserve Committee (JORC)[2] Code categories. For all projects the Resource Classification is based on the average distance of the block to the closest three holes within specified ranges, with the ranges having been determined through drill spacing analysis of mineralisation continuity and site experience with similar veins.

The process included steps to remove isolated small clumps of blocks or isolated individual blocks of different classifications that cannot be realistically mined separately. The Martha underground project has been extensively mined historically, leaving abundant historic voids that interact with the remnant mineralisation. In order to adequately capture resource risk for this project the depleted insitu resource is evaluated through a stope optimisation process to define the potential future mineability of the resource. The stope shapes generated are based on an incremental cut-off and a stated gold price assumption. The stope shapes are then used to define the portion of the resource that has an appropriate average drillhole spacing for reporting purposes.

[2] The definitions of Ore Reserves and Mineral Resources as set forth in the JORC Code have been reconciled to the definitions set forth in the CIM Definition Standards. If the Mineral Reserves and Mineral Resources were estimated in accordance with the definitions in the JORC Code, there would be no substantive difference in such Mineral Reserves and Mineral Resources.

Table 1-1 :Classification Criteria

Project	Drill Spacing for Measured Resource	Drill Spacing for Indicated Resource	Drill Spacing for Inferred Resource
Martha Open Pit ELB	20 metres	50 metres	100 metres
Gladstone Open Pit	15 metres	30 metres	52.5 metres
Martha Underground	20 metres	40 metres	60 metres
WKP	15 metres	50 metres	70 metres
Correnso	10 metres	30 metres	60 metres

Mineral Resource classifications are based solely on gold using a combination of data density, spatial arrangement of the data, quality of estimation, and geological interpretation. Mineral Resource classification reflects the confidence levels in the supporting data.

Mineral Resources are inclusive of Mineral Reserves and are presented in Table 1-2 to Table 1-4.

Table 1-2 :Open Cut Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Measured	0	0	0	0	0
Indicated	6.75	1.82	13.3	0.40	2.89
Measured & Indicated	**6.75**	**1.82**	**13.3**	**0.40**	**2.89**
Inferred	5.4	1.8	17	0.3	3.0

Table 1-3 :Underground Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Measured	0.04	4.92	14.8	0	0.02
Indicated	7.00	6.35	17.9	1.43	4.01
Measured & Indicated	**7.04**	**6.35**	**17.8**	**1.44**	**4.02**
Inferred	4.4	7.7	17.3	1.1	2.5

Table 1-4 :Combined Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Measured	0.04	4.92	14.8	0	0.02
Indicated	13.74	4.12	15.6	1.82	6.91
Measured & Indicated	**13.8**	**4.13**	**15.6**	**1.83**	**6.93**
Inferred	9.9	4.4	17	1.4	5.4

Notes to Accompany Mineral Resource Table

- A gold price of NZ$2,394/ oz (US$1,700/ oz @ USD:NZD 0.71) for all Resources;

- MUG Resources are reported below the MOP5 design and are constrained to within a conceptual underground design, based upon the incremental cut-off grade of 2.15 g/t;
- WKP Resources are constrained to within a conceptual underground design based upon the cut-off grade of 2.5 g/t Au;
- MOP5 and GOP resources are reported within conceptual pit designs based on cut-off grades of 0.5g/t and 0.56g/t respectively.
- The tabulated Resources are estimates of metal contained as troy ounces;
- No dilution is included in the reported figures and no allowances for processing or mining recoveries have been made;
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly;
- Development mining in MUG commenced in 2019 following receival of Resource Consents in December 2018. Mining has not commenced at any of the other projects; and
- Resources that are not Mineral Reserves have not demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

Mineral Resources are reported on a 100% basis. The resource estimate is sub-divided for reporting purposes into an open cut resource that includes material within the limits of the Martha and Gladstone pits and an underground resource within the Correnso area and Martha underground area. The resources are depleted for mining as at December 31, 2020.

1.15 Mineral Reserve Estimate

1.15.1 Reporting Date

Mineral Reserves for the Martha underground and Correnso underground are reported as at December 31, 2020.

1.15.2 Qualified Persons

The Mineral Reserves quoted here were prepared by, or under the supervision of Trevor Maton, Study Manager for OceanaGold.

1.15.3 Mineral Reserves

Mineral Reserves are being declared for Martha underground based on the recent completion of the Feasibility Study. The Mineral Reserves were classified using the 2014 CIM Definition standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining shapes created during the mine design process.

The Reserves were compiled with reference to the NI 43-101 and Joint Ore Reserve Committee. The basis for the estimation of Mineral Reserves is a metal prices of NZD2,112 per oz (US$1,500 per ounce) for gold. The technical and economic viability of the reported Mineral Reserves is supported by studies which meet the definition of a Feasibility Study (FS). All permits and consents are in place for the extraction of the Mineral Reserve.

Mineral Reserves were classified using the 2014 CIM Definition standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining shapes created during the mine design process. The Mineral Reserves for Martha underground and the Mineral Reserves for Correnso are summarised in Table 1-5.

Table 1-5: Mineral Reserves

Reserve Area	Class	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
MUG	Proven	-	-	-	-	-
	Probable	4.46	4.33	13.5	0.62	1.94
Total MUG		**4.46**	**4.33**	**13.5**	**0.62**	**1.94**
Correnso	Proven	0.04	4.92	9.2	0.01	0.01
	Probable	0.02	6.01	10.2	0.00	0.01
Total Correnso		**0.06**	**5.25**	**9.50**	**0.01**	**0.02**
Total Mineral Reserve		**4.52**	**4.34**	**13.5**	**0.63**	**1.95**

Notes to Accompany Mineral Reserve Table:

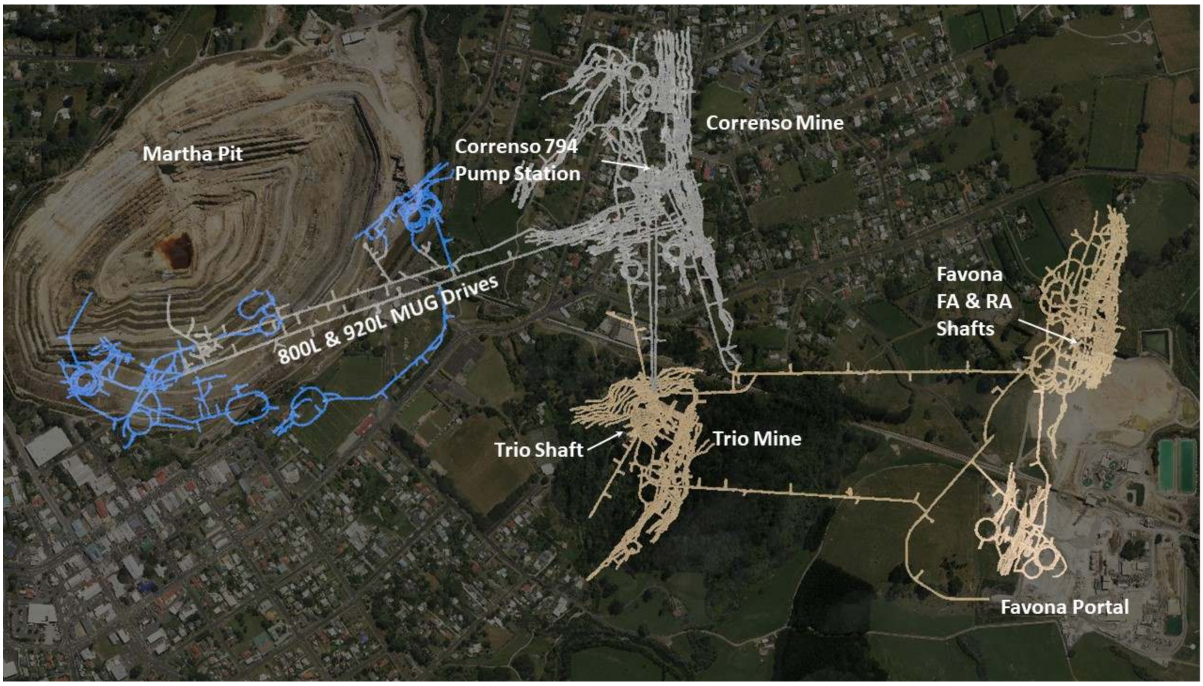
- Mineral Reserves are reported on a 100% basis;
- Mineral Reserves are reported to a gold price of NZD 2,112/oz;
- Tonnages include allowances for losses resulting from mining methods. Tonnages are rounded to the nearest 10,000 tonnes;
- Ounces are estimates of metal contained in the Mineral Reserves and do not include allowances for processing losses. Ounces are rounded to the nearest thousand ounces;
- Rounding of tonnes as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content;
- Tonnage and grade measurements are in metric units. Gold ounces are reported as troy ounces.
- Underground Mineral Reserves are stated using 2.4 to 3.3 g/t Au cut-off. Mining recovery ranges from 67% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated within a dilution zone. The dilution ranges from 0% to 7% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash flow model.
- The underground Mineral Reserves were estimated by Trevor Maton, MAusIMM (CP), QP.

1.16 Mining Method

Permits and consents have been granted for MUG project and all selected mining methods are in accordance with the license, permit and consent conditions, principally related to placement of backfill, blast vibration limits, method of working and hydrogeological controls. Exploration drives were completed on 800 m RL and 920 m RL in 2018. Development of MUG commenced in mid-2019 and 2,169 m of lateral development and a 120 m ventilation raise were completed by the end of 2019.

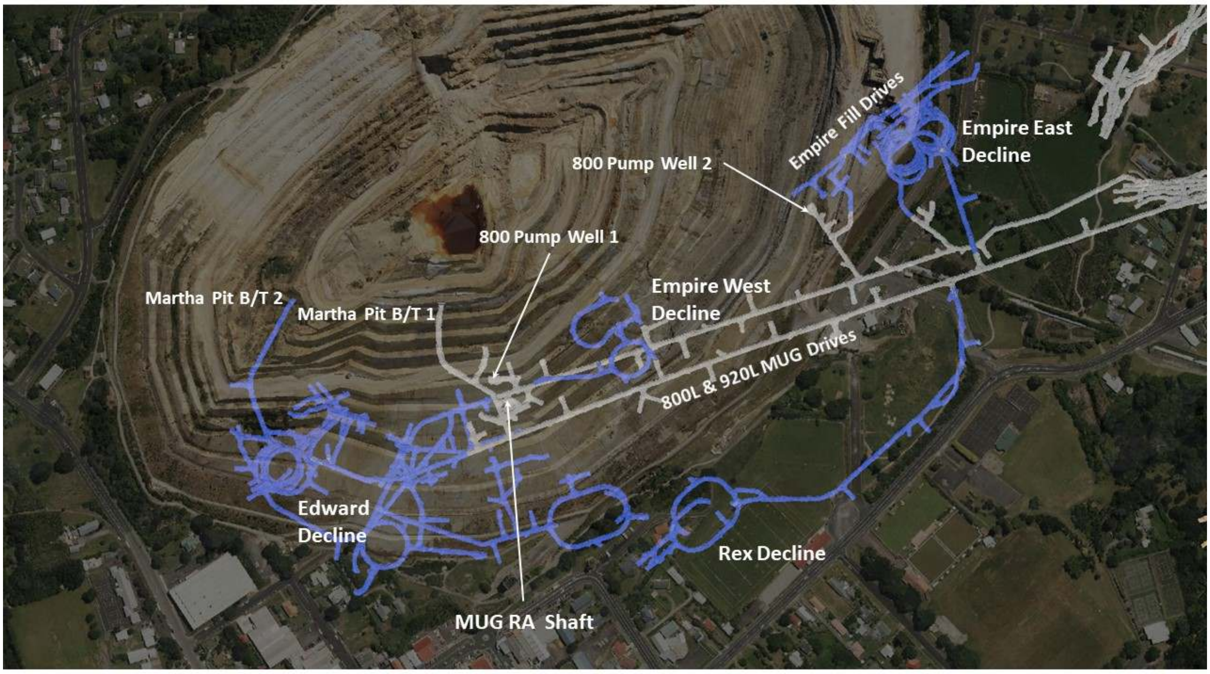
The Martha underground is accessed via the existing Favona portal through the existing Trio and Correnso workings and shares the ventilation development and shafts as well as the Correnso workshop, Trio cribroom and dewatering systems. Exploration drives were completed on 800 m RL and 920 m RL in 2018. Refer Figure 1-1 for the extent of mine development as at the end of 2020.

Figure 1-1: MUG Mine Access Development



Development of Martha underground commenced in mid-2019 and 2,169 m of lateral development and a 120 m ventilation raise were completed by the end of 2019 and a further 7,554 m of lateral development completed in 2020. Two breakthrough openings into the pit for ventilation and escape were also completed. The extent of MUG development is shown in Figure 1-2.

Figure 1-2: MUG Mine Development to End 2020



Development up to end 2020 has been focussed on ramp accesses for Edward, Empire, Rex and Royal mine zones, ventilation connections, pumping well access drives, drilling platforms and back fill drives as well as the breakthroughs into the pit.

The mine design is shown in plan view in

Figure 1-3 and long section in Figure 1-4.

Figure 1-3: Martha Underground Overview Plan View

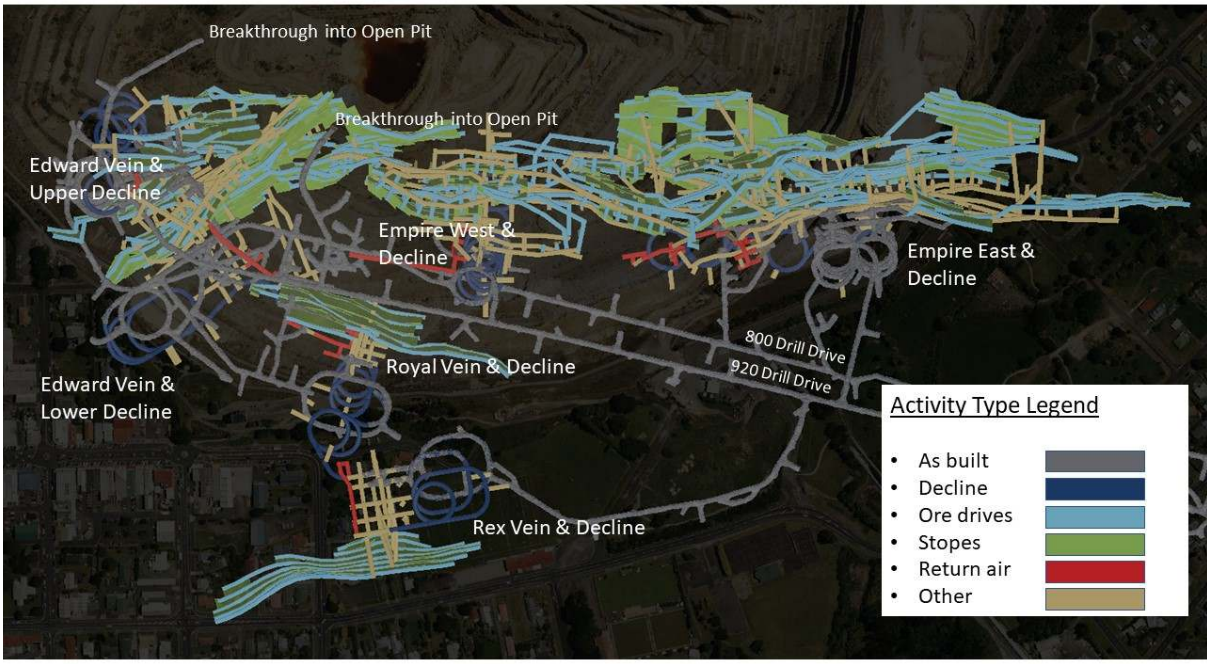
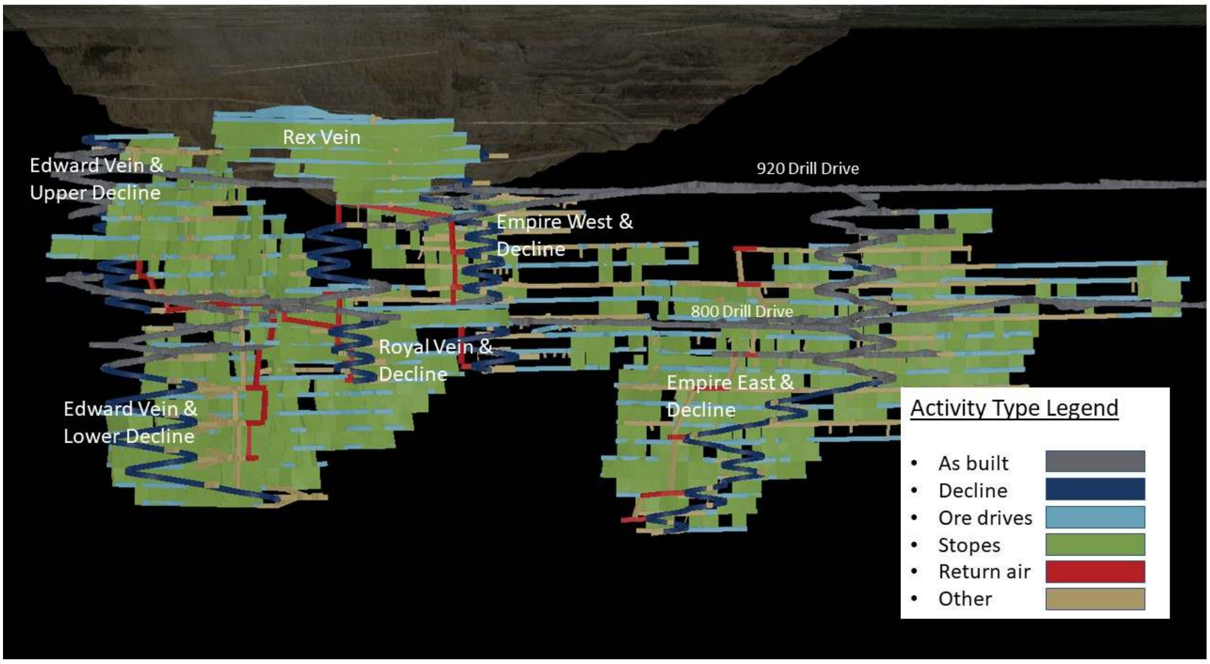


Figure 1-4: Martha Underground Overview Long Section



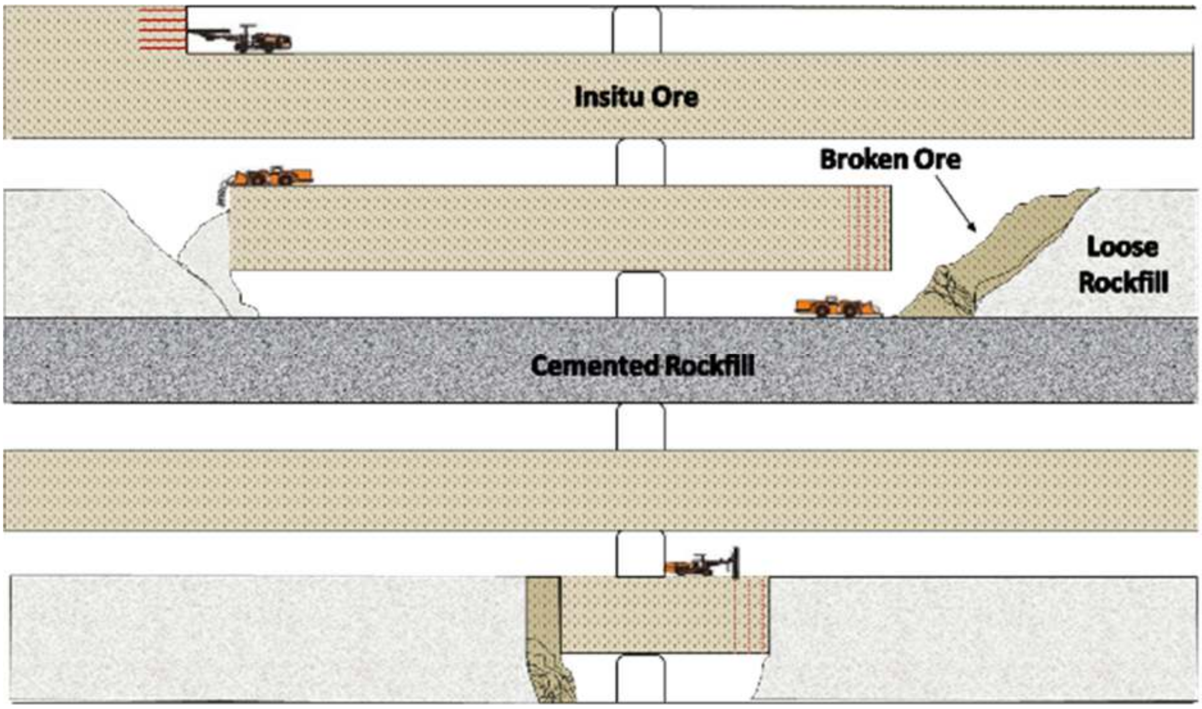
Based on the proposed mining method and equipment, historical experience and orebody geometries, the development strategy for all underground areas involves mining of declines for access to five main stoping blocks. Access drives will be mined to develop drilling and loading levels, generally targeted to intersect the orebodies centrally. Access drives will be spaced at 18 m vertically over the height of the mine. Each access drive will have a dedicated sump, substation recess and development for escape and return air raises. Ore drives will be developed in both directions along strike from the access drives. Stockpiles will be mined off the decline and in levels for truck loading. The development design used for the Feasibility Study is aligned with current operating practices at Waihi.

Mining method selection work for the Martha underground was undertaken by SRK in 2011, 2016 and 2017 and confirmed by Entech in 2018 and by OceanaGold in 2020. Four mining methods are proposed for the mine:

1. Modified Avoca with rockfill in previously unmined areas.
2. Modified Avoca with rockfill in remnant areas adjacent to collapsed stopes separated by an intermediate pillar.
3. Modified Avoca with rockfill in remnant areas adjacent historical stopes filled with engineered fill (CRF / CAF)
4. Bottom up side ring method with CRF/CAF/RF where skins adjacent to historical backfill are extracted.

Much of the Mineral Reserve can be extracted using the modified Avoca mining method, refer Figure 16-18, similar to the methods employed at Favona, Trio and Correnso. The modified Avoca method with RF is a semi-selective and productive underground mining method, and well suited for steeply dipping deposits of moderate thickness. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralisation. Access is required centrally within stope panel to allow for mining to progress longitudinally. Down holes are drilled and loaded with explosives and the stope is blasted, with broken material falling to the bottom drive for extraction. Conventional bogging of the broken material can occur until the brow is exposed. At this point, remote controlled LHD’s are employed required to remove the blasted material from the stope. Stope structural support is provided through a combination of cable bolting and uncemented RF. It is not planned to leave rib pillars unless there is limited access to the sub-level or recommended to maintain overall mine stability.

Figure 1-5: Modified Avoca Mining Method



Source SRK Consulting Ltd

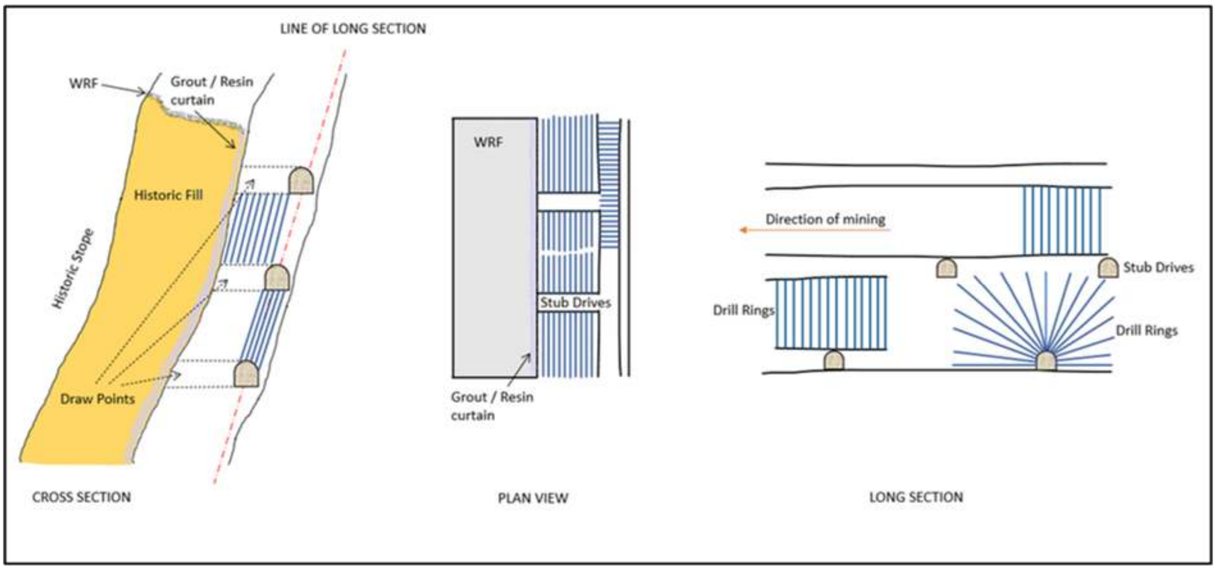
A small proportion of the Mineral Reserve will involve the extraction of remnant skins in the footwall or hangingwall of previously mined (historical) stopes, or the extraction of both remnant skins. Historical backfill may also be mined and experience with OP mining shows this material may be above the cut-off. However, as it is currently classified as Inferred Resource it is not included as Mineral Reserve.

Following detailed studies over the last nine years, three methods are proposed for the extraction of remnant areas, adjacent to historic workings, namely:-

1. A modified Avoca method whereby the historic stope is backfilled with CRF prior to stoping and the remnant skin is extracted by conventional modified Avoca using RF in a bottom up sequence that exposes the CRF.
2. A modified Avoca method adjacent the collapsed historic stope where backfill with CRF is not feasible and a stand off from the historic wall of 3.5 m maintained with lower estimated recoveries and higher dilutions.
3. A remote side ring method where the historic backfill is extracted together with remnant wall rock in a bottom up sequence. The side ring method is described below.

The side ring mining method for the extraction of remnant skins will use conventional remote drilling and loading methods, combined with remote LHD equipment. This method involves additional waste development adjacent to the remnant stopes, which increases overall development quantities and mining costs. Entech concluded that once established, the method is expected to achieve acceptable mining recovery with few safety issues anticipated. The proposed mining method is illustrated in Figure 1-6. This method is employed in the Empire west area and comprises a very small proportion of the Mineral Reserve.

Figure 1-6: Remnant Mining Method



Source Entech Consulting Ltd, 2021.

1.17 Recovery Methods

Recovery of gold at Waihi is achieved from the use of leaching and adsorption following a conventional SAG Mill-Ball Mill grinding circuit. The plant has been successfully running for 30 years with a well-established workforce and management team in place. The Processing Plant has the capacity to treat up to 900,000 tonnes of Martha underground Mineral Reserve ore per annum.

Ore from the surface and underground mine is stockpiled at the ore pad before being fed to a jaw crusher located directly above the mill into the SAG mill. Ore is fed to the SAG mill along with lime, water and steel balls. Once the ore has reached the final product size it is thickened to higher density slurry in a thickener before the leaching process begins.

The pre-leach thickener increases slurry density to approximately 37% to 40% solids prior to the CIP circuit, which comprises of five leach and seven adsorption tanks. The leaching tanks provide a total residence leach/adsorption time of 24 hours for Martha open pit ore and 48 hours for Correnso / Martha underground ore.

The loaded carbon is fed into an elution column where the carbon is washed at high temperature and pressure to remove the gold and silver from the carbon and into the water. The cathodes are periodically harvested and rinsed to yield a gold and silver bearing sludge which is dried, mixed with fluxes and put into a furnace at 1200^{O}C. Once the sludge is molten it is poured as bars of doré bullion ready for shipment to the Mint.

The process plant is also suitable for both processing other resources from Waihi and WKP. Flotation and ultrafine grind continue to be being evaluated as a method to improve process recoveries.

1.18 Project Infrastructure

The project is an active mining project with the majority of the infrastructure required for its ongoing operation already in place. Site access from major ports, international and domestic airports and roads are well established at the Waihi site. Supplies, equipment, and materials are trucked to the sites via the paved roads. As this is a gold project there are no concentrate shipping constraints. There are no material logistic limitations impact the project.

MUG will use the facilities in place established at Waihi operations in 1988 and upgraded in the late 1990's and from 2004 for the Favona mine. Existing facilities comprise:

- Two separate tailings storage facilities,
- numerous silt and collection ponds,
- stockpile facilities,
- mine access roads,
- water treatment facilities,
- fuel supply and storage,
- Favona underground administration and change house,
- Martha pit surface conveying and loadout facilities,
- Favona surface and underground workshops,
- Trio underground cribroom,
- surface explosives magazines,
- existing process plant, and
- access via the Favona, Trio and Correnso mines and existing ventilation shafts.

New surface infrastructure comprises raising of the TSF's, duplicating the existing 33kV incoming power line, construction of a cement batch plant and refurbishment of the open pit crusher and overland conveyor.

Project infrastructure will require small raises of the existing TSF's. TSF2 has a planned finished crest elevation of 159.5 m RL (3.5 m raise) and the planned crest of TSF1A is 182 m RL (9 m raise). The embankments have both been constructed from overburden material obtained from mining Martha pit. TSF2 was constructed first and provided tailings storage from 1989 to 2000. TSF1A has since provided tailings storage. TSF1A and TSF2 are permitted by the Mining Licence, TSF1A has a Building Consent allowing it to be constructed to 177.25 m RL. TSF2 has a Building Consent allowing it to be raised to 161 m RL. A further lift on TSF1A to 182 m RL is planned through a combination of downstream and centreline construction techniques and requires consenting.

A cement batch plant is proposed to be constructed close to the existing polishing pond stockpile entry.

Power is supplied through the local utility. The power supply is provided from the national grid and supplied to the company substation at the mill location and mine locations. The company has backup generation available to support the main lines if needed. The mine is currently allocated 12 MW but during peak holiday seasons the mine is restricted in its power draw to 9 MW and some areas of the operations are shut down during these times.

It is planned to duplicate the 33 kV Powerline from the Waikino Substation (11 km's north of Waihi Township) to the Waihi Town Substation and from there to a proposed new 33 kV / 11 kV Substation at the WPP.

1.19 Market Studies and Contracts

Contracts are in place covering underground mining, transportation and refining of bullion, purchase and delivery of fuel, electricity supply, explosives and other commodities. These agreements conform to industry norms.

OceanaGold maintains a number of operating permits for the importation of reagents into New Zealand. New Zealand has an established framework that is well regulated and monitored by a range of regulatory bodies.

Risk associated with renewal of importation permits is regarded as manageable.

1.20 Environment Studies, Permitting and Social or Community Impact

1.20.1 Environmental & Permitting

The Waihi operation holds the permits, water rights, certificates, licences and agreements required to conduct its current operations and to extract the Mineral Reserve. A small raise on TSF1A will require a building consent and resource consent.

1.20.2 Environmental Technical Assessment

Modern mining has operated in Waihi since 1987 to today. During this time, mining occurred only within MOP and most recently at Correnso and Favona underground with mining currently occurring in the MUG project that is located largely under the Martha Pit.

Over this period OceanaGold has undertaken technical assessment and monitoring of relevant amenity aspects and implemented mitigation strategies where required. This approach will continue for the assessment and ongoing management of the proposed projects. The amenity aspects include, but are not limited to, the following: blasting, vibration, terrestrial ecology, aquatic ecology, water management, ground settlement, air quality, property, heritage and socio-economic impacts.

1.20.3 Social / Community

The Waihi community has experienced a history of mining operations both open pit and underground and new projects during this time. With the close proximity of a community to these operations, the need to understand and manage how mining affects the community and society at large is integral to successfully operate within a town. This has led to mining becoming part of the social fabric and identity of Waihi today, creating strong community connections built through trust by sharing information. To identify and analyse how mining affects the Waihi community, reliable information is gathered through various monitoring methods across a broad section of disciplines.

1.20.4 Cultural

OceanaGold (and its predecessors) have engaged with Iwi who have a special interest in Waihi in relation to various mining proposals over the last 30 years, and engagement is currently occurring in relation to proposed projects.

The nature of the engagement between OceanaGold and Iwi has been wide-ranging. It has included the establishment of Memorandums of Understanding, which have sought to recognise the relationship of Iwi with some of the areas that are subject to mining activities and their role as kaitiaki (and for Iwi to recognise the perspectives and interests of the company). It has also included engagement in relation to the potential cultural effects of resource consent applications for new mining proposals (e.g. through the preparation of cultural impact assessments), acknowledging that at times there has been a need for Iwi to submit on resource consent applications by the company. OceanaGold has also worked with Iwi to implement ongoing cultural awareness training of its staff.

OceanaGold will continue to engage with Iwi during the operational phase of Martha underground.

1.21 Capital and Operating Costs

1.21.1 Revenue

- The estimated operating cost per tonne for the Reserve case is USD 115 /t.
- Processing plant production rate of 0.9 Mtpa has been scheduled.
- Gold Price: US$1,500 /oz.
- Exchange Rate: 1 NZD = USD 0.71.
- Metallurgical recovery average of 94.9% for MUG but varies based on Au and As grade,
- Royalty payments include higher of 1% of net sale revenue or 5% accounting profit to the Crown, and 2% to a third-party specific to a localised area of Rex.
- Revenue is recognised at the time of production.

1.21.2 Mineral Reserve Cost Model

Capital costs are developed for growth and sustaining capital. Growth capital represents pre-production underground mining and capital required to increase production. OGC developed the sustaining capital cost estimate to account for underground mine development, mine equipment and TSF construction capital costs through the LOM, by applying the same estimating methodology as for growth capital.

The capital cost estimate for the FS has an expected accuracy of ± 15%. Underground capital mine development costs are well known through the sites operating history as is the costs of salaries, wages, ground support, drilling, blasting and mobile plant consumables.

The estimate includes direct and indirect costs (such as engineering, procurement, construction and start-up of facilities) as well as owner’s costs and contingency associated with mine and process facilities and on-site infrastructure.

The following areas are included in the estimate:

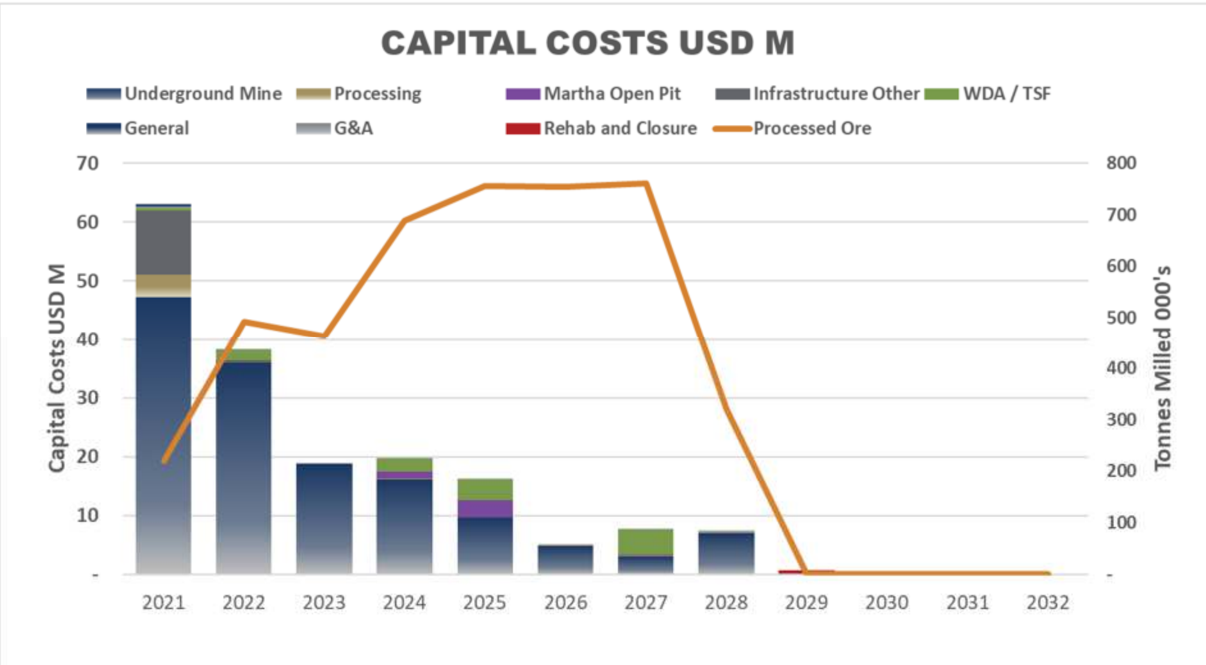
- Mine (underground mine development, equipment fleet finance leases, backfill plant and supporting infrastructure and services).
- Process plant replacement of existing Waihi SAG Mill shell (currently being fabricated with known costs).
- Tailings Storage Facility raises to TSF1A and TSF2 estimated by independent consultants.
- On-site infrastructure (water treatment and distribution, electrical substation and distribution, and other general facilities).
- Pit rim works including relocation of public roads, estimated by independent consultant.
- Refurbishment of the overland crusher and conveyor system, estimated by OGC.
- Property purchases above the Rex orebody.
- Duplication of the 33kV line from Waikino to Waihi with a buried cable and new substation.
- Incremental mine site rehabilitation.
- Engineering work, being in the range of 25‑30% of total engineering for the project, was carried out to support the estimate.

The capital costs including sustaining capital is outlined in Table 1-6 and shown by year in Figure 1-7. The range of accuracy for the capital cost estimate is ± 15%.

Table 1-6: Capital Costs Initial and Sustaining

Summary Capital Expenditure Schedule	Growth MUG LOM Estimate USD M	Sustaining MUG LOM Estimate USD M
General and Administration Costs	0.00	2.20
Processing	0.00	4.51
Open Pit Mining Martha Pit	0.00	4.20
Underground Mining Martha	25.50	114.37
TSF Constructions	0.00	12.19
Infrastructure	12.39	0.00
Rehabilitation	0.00	1.37
Total	**37.89**	**138.84**

Figure 1-7: Capital Cost Estimate



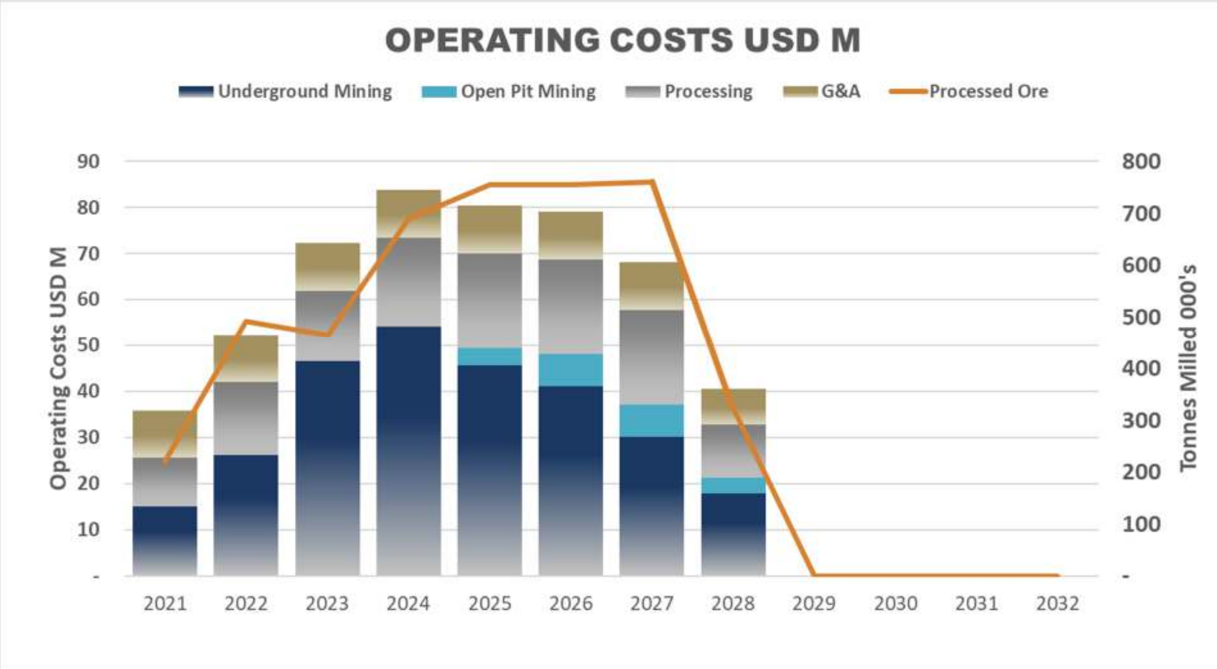
The operating cost estimate is +/- 15%. This level of accuracy is attributed to the site operating history over a range of conditions. Table 1-7 summarizes the estimated operating costs and is approximately USD 115 /t for the Mineral Reserve.

Table 1-7: Operating Costs

Summary Operating Expenditure Schedule	LOM Estimate USD M	LOM Estimate USD / tonne
General and Administration Costs	80.11	17.97
Processing	134.29	30.12
Open Pit Mining Martha Pit	21.29	4.78
Underground Mining Martha Underground	276.86	62.10
Other & stockpile	0.00	0.00
Total	**512.54**	**114.97**

The estimation of operating costs by year together with the Mineral Reserve processed are shown in Figure 1-8: Operating Costs for Project.

Figure 1-8: Operating Costs for Project



1.22 Economic Analysis

1.22.1 Cash Flow Analysis

Reserve case assumptions for economic projections include a gold price of US$1500 /oz and USD 0.71 exchange rate. The key economic results are as presented in Table 1-8, using 1 January 2021 as the reference commencement date.

Table 1-8: Key Economic Metrics

Financial Metric	Unit	Reserve Case
Gold Price	$/oz	1500
Exchange Rate	USD:NZD	0.71
Before-Tax		
$NPV_{5\%}$	USD M	143
Internal Rate of Return	%	47
LOM Cumulative Free Cash Flow	USD M	193
After-Tax		
$NPV_{5\%}$	USD M	99.4
Internal Rate of Return	%	36
LOM Cumulative Free Cash Flow	USD M	139
Payback Period	years	3.9
Cash Costs C1	USD/oz.	839
AISC	USD/oz.	1107

1.22.2 Reserve Case

The LOM projections for the Reserve case gold price assumption free cashflow, are shown in Figure 1-9 and Figure 1-10.

Figure 1-9: Cash Flow Profile Reserve Case

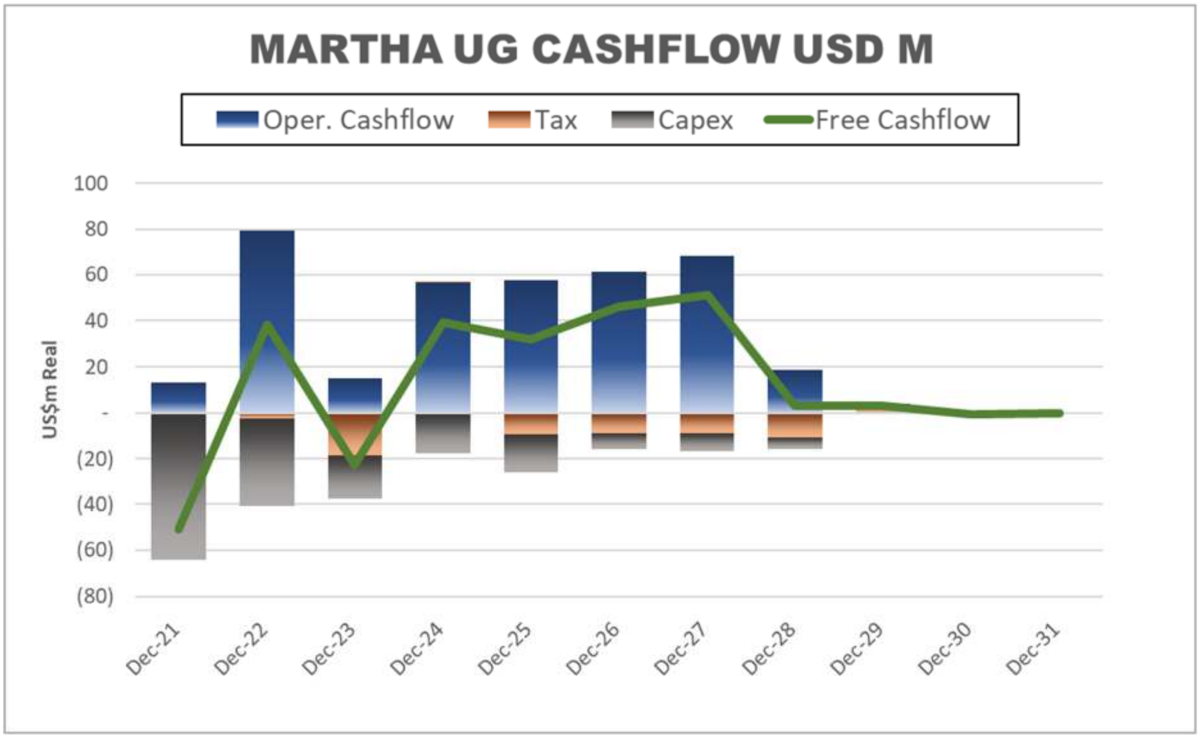
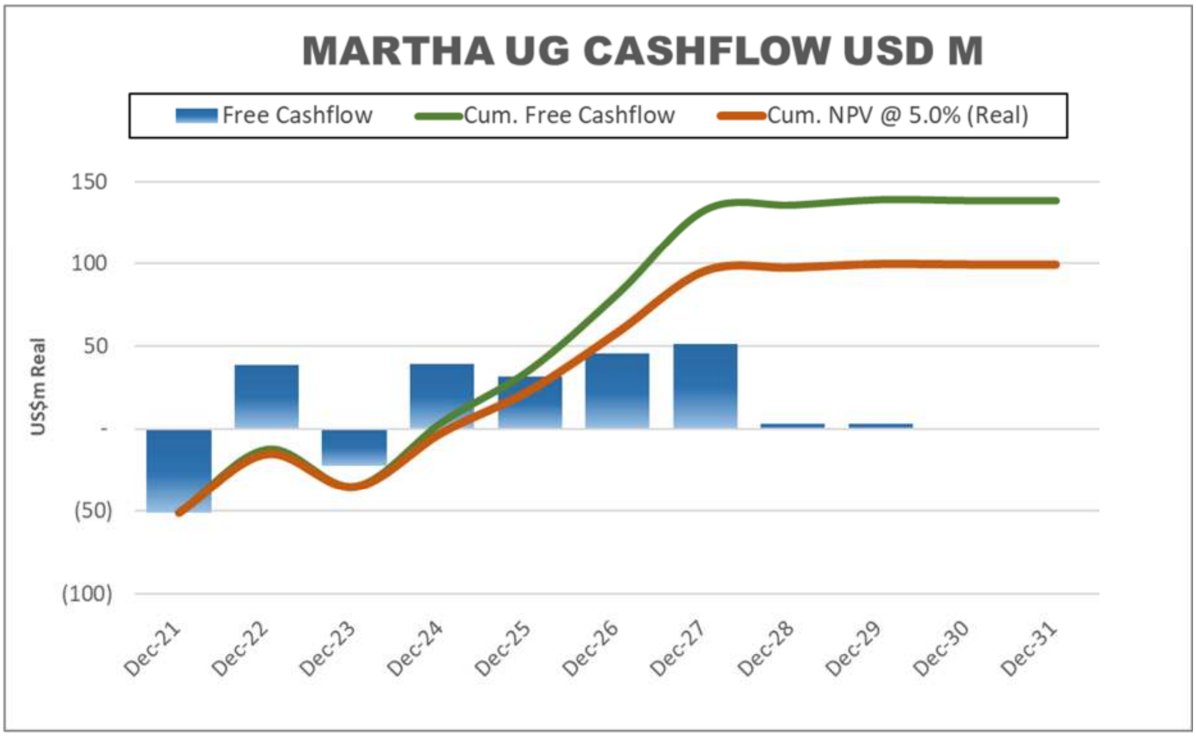
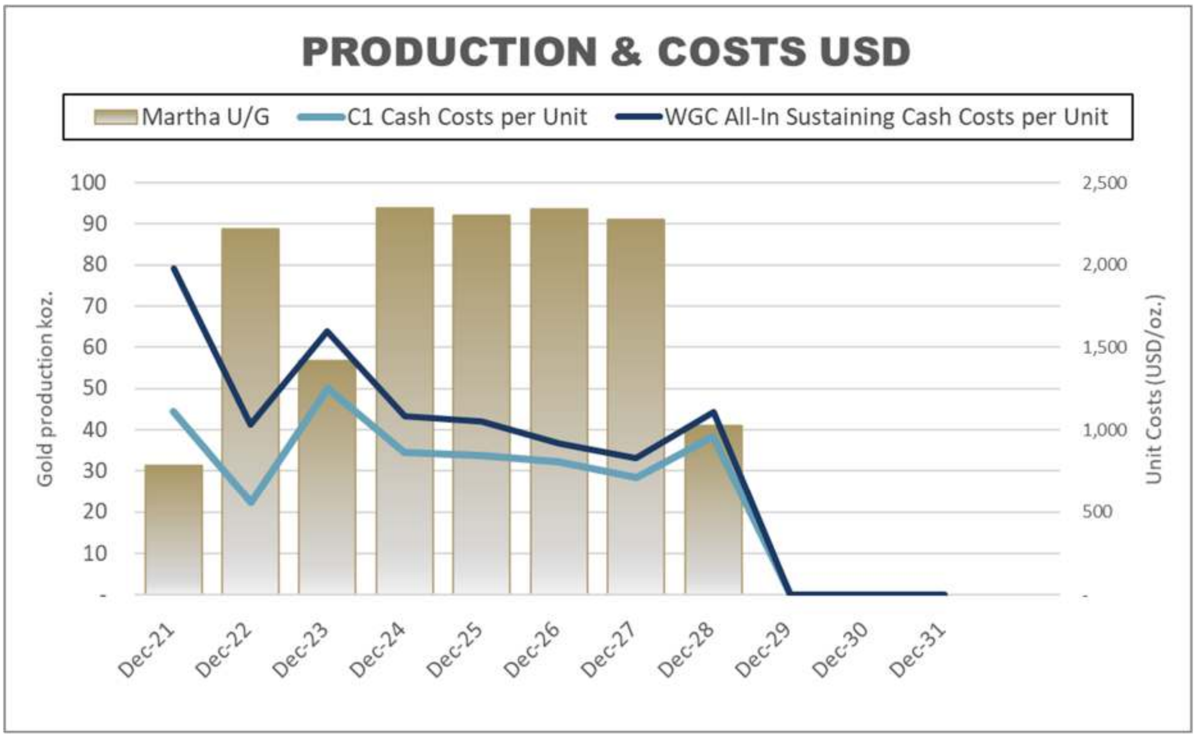


Figure 1-10: Cumulative Cash Flow Profile Reserve Case



The production and cost profile are shown in Figure 1-11.

Figure 1-11: Production & Costs Profile Reserve Case



The Reserve case represents production growth at all-in sustaining costs of $1107 per ounce and life of mine cash costs of $839 per ounce. These costs are based on mining unit costs that are higher than the historical mining costs. Processing costs are expected to be $30 per tonne processed. Figure 1-11 shows a sharp decrease in AISC and cash costs as Martha underground ramps-up, and production levels increase significantly.

In the Reserve case scenario with a long-term gold price of USD 1500/oz and discount rate of 5% an after-tax NPV of $100 million dollars and after-tax IRR over 36% is indicated. This is a robust project. The cumulative undiscounted free cash flows have been calculated at USD 139M pre-tax dollars while the average annual free cash flow is approximately USD 42 million from 2024 to 2027. The highest year for free cash flow is 2027 at USD 51 million.

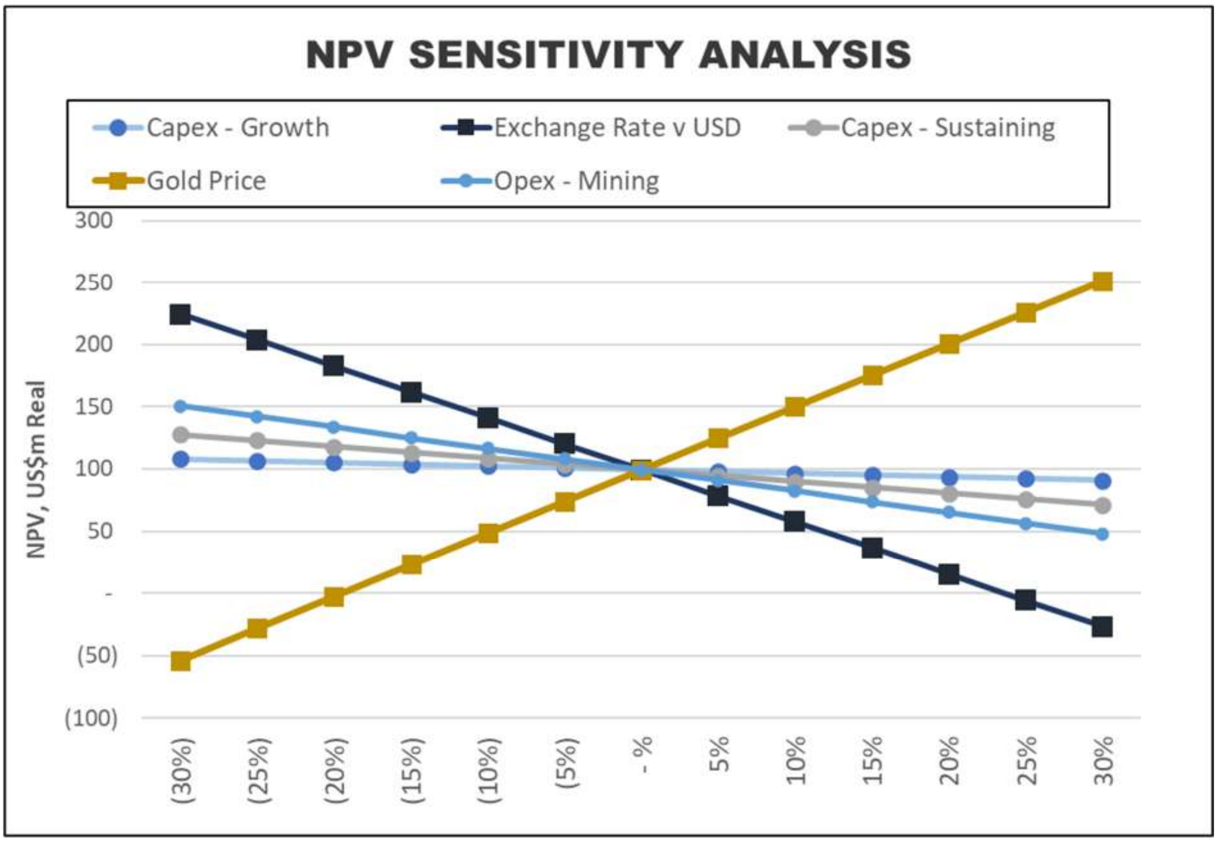
1.22.3 Sensitivity Analysis

The key economic risks were examined by running cash flow sensitivities:

- Gold price.
- Exchange rate.
- Gold head grade.
- Operating costs.
- Capital costs.

NPV sensitivity over the base case has been calculated for a range of variations. The after-tax sensitivities are shown in Figure 1-12.

Figure 1-12: NPV Sensitivity



1.23 Other Relevant Data and Information

The Waihi operation is in a high rainfall area, and heavy rain events are not unexpected. Procedures and costing are in place to deal with such events for the open pit operation and will not impact on the viability of extracting the Mineral Reserve.

New Zealand has an established framework that is well regulated and monitored by a range of regulatory bodies. OceanaGold has dedicated programs and personnel involved in monitoring consent compliance and works closely with authorities to promptly address additional requests for information. Risks associated with review and renewal of operating consents is, upon that basis, regarded as manageable within the ordinary course of business.

There is no material, unresolved matters dependent upon a third party on which extraction of the underground Mineral Reserve is contingent.

The risk management process is not static, and risks may change with time. The current study represents an understanding by the operations personnel and project team of significant risks associated with the Waihi operation, while recognising that the level of risk may change over time and that new risks may emerge. The risk register is considered a “live” document and forms part of the risk management plan which will be subject to regular review.

1.24 Conclusions & Recommendations

1.24.1 Conclusions

The following conclusion have been drawn from this Technical Report:

- Geologic controls for Martha ore bodies are well understood and documented by mapping and over 40,000 drill holes. The Mineral Resources and Mineral Reserves have been estimated in accordance with CIM. There are no open pit Mineral Reserves.
- A geotechnical field characterisation program has been undertaken to assess the expected rock quality which included logging core, laboratory strength testing, in-situ stress measurements and oriented core logging of jointing. A geotechnical assessment of the orebody shape and ground conditions has determined that long hole Avoca stoping is an appropriate mining method for virgin areas and selected remnant areas and a remnant mining method is applicable to mining skins on historic backfilled stopes and remnant pillars. Stopes have been sized to maintain stability once mucked empty.
- The design has been laid out using empirical design methods based on previous experience at the various Waihi mines. The stability of the design has been checked with 3D numerical stress-strain analyses of the workings which included consideration for mine-scale faulting. The modelling results confirm that stopes and access drifts are predicted to remain stable during active mining.
- The mine designs, mine dewatering designs, mining plans, and processing assumptions are based upon FS level evaluations. Mine planning work considered revenue for Au and a cut-off grade of 2.5 g/t to 3.2 was used depending on the mining methods. Stope optimisation was completed to identify economic mining areas. A 3D mine design was completed based on the stope optimisation results.
- Tonnage and grades presented in the Mineral Reserve include dilution and recovery and are benchmarked to the existing operation as well as other similar operations. Productivities were generated from first principles with inputs from site engineers, mining contractors, blasting suppliers, and equipment vendors where appropriate and benchmarked against existing Waihi operations. Equipment used in this study is standard equipment used worldwide with only standard package / automation features.
- The underground support infrastructure is largely in place and new infrastructure relatively straightforward.
- No fatal flaws have been identified through the investigations to date that would prevent the required permissions being obtained for the activities described in this assessment.

- Mining tenure held by OceanaGold in the areas for which Mineral Resources and Mineral Reserves are estimated is valid;
- OceanaGold holds sufficient surface rights to support mining operations over the planned life of mine that was developed based on the Mineral Reserves;
- Permits held by OceanaGold for the Project are sufficient to ensure that mining activities are conducted within the regulatory framework required by New Zealand law;
- Sufficient tailings storage facilities have been planned for the Mineral Reserve.
- The process route for precious metal extraction is effectively fixed for the Martha underground project. Metallurgical laboratory testwork supports this configuration;
- The existing process plant has sufficient capacity for the Martha underground and the process circuit is well suited to the metallurgy demonstrated over 30 years of processing similar ores;
- Closure provisions and allowances for additional activities have been appropriately considered. Monitoring programs are in place;
- The costs estimation is considered to be within ±15%, supported in part by the site's extensive and varied operational history;
- Adequate contingency has been allowed for in terms of costs and additional activities (mine development metres, ground support infrastructure, rehabilitation activities) to support the FS.
- An operational readiness plan is not required for this Project as the project team has already largely transitioned into the operations team and the technical services and operation team already have well developed operating procedures and processes.

1.24.2 Recommendations

Based on the conclusions of the technical report and the FS, the key following actions are recommended:

- Continue development of MUG in accordance with the current schedule to ramp up the mine production by Q2 2021.
- Continue drilling to improve the Resource classification to support and add future Mineral Reserves.
- Continue drilling to increase Mineral Resources around MUG.
- Continue to investigate opportunities for extraction of the MUG Mineral Resources not included in this FS.
- Continue to evaluate MUG as the other Waihi District Plan projects are further studied.

Specific recommendations in project areas follow.

Underground mining:

- Monitor and adjust as necessary development performance to ensure that rates achieved are in line with feasibility assumptions.
- Review opportunities to bring forward production in 2023 and address the current schedule dip; areas for investigation will include development scheduling and capacity, mine design and dewatering rates.
- Ensure the water table is lowered at the forecast rates, with a focus on adequacy of the monitoring system and performance of installed pumps; sufficient monitoring is to be in place for checking progress and sufficient dewatering capacity appears to be installed to meet the targets.

- On an ongoing basis, review opportunities for optimisation, remote operation and automation of the ventilation system.

Mine planning:

- Update the mine plan from time to time as appropriate, as updated information relating to geological resource, geotechnical conditions, mine dewatering levels and other relevant areas becomes available.

Geotechnical engineering:

- With management of voids a key issue for mining of the Martha underground project, continue to update and develop the Void Management Plan and formulate appropriate related procedures as experience in mining at Martha is gained;
- Undertake trial stoping in remnant mining areas, to build experience and identify factors for safe and effective recovery of targeted mineralisation
- Continue geotechnical assessments on an ongoing basis, including in-situ stress measurements, rock property testing and numerical modelling, to provide recommendations for mine design so as to enable updates and improvements from time to time of the mine plan.

Equipment selection:

- Monitor commercial development and availability of proven technologies that would potentially enable the introduction of battery electric (BEV) mine equipment, and facilitate the automation of mining processes.

Backfilling design and operation:

- Undertake further implementation level engineering work to optimise the backfill system which will include production and placement of Cemented Rock Fill (CRF) and Cemented Aggregate Fill (CAF).

Metallurgy:

- Undertake further variability and comminution testing as core becomes available to confirm recovery estimates, test assumptions for areas outside the current mine plan and to investigate potential alternative flowsheets that may further increase overall metallurgical performance.

Infrastructure:

- Progress engineering and other arrangements associated with upgrades to the existing electrical system to account for the underground power requirements.

2 INTRODUCTION

2.1 Terms of Reference

This report provides Mineral Resource and Mineral Reserve Estimates, and a classification of Mineral Resources and Reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines 10 May 2014 (CIM, 2014).

References in this report to "OceanaGold" include OceanaGold Corporation, Oceana Gold (New Zealand) Limited, Waihi Gold Company Limited and their subsidiaries and associates, as the context requires. This report has been prepared to satisfy OceanaGold obligations as a reporting issuer in Canada.

2.2 Principal Sources of Information

This Technical Report was prepared by OceanaGold. Information for the report was based on published material as well as the data, professional opinions and unpublished material obtained from work completed by OceanaGold, and materials provided by and discussions with, third-party contractors/consultants retained by OceanaGold. Reports and documents listed in Appendix A were also used to support preparation of the report. Additional information was sought from OceanaGold personnel where required to support preparation of this report.

Table 2-1: Specialist Consultants who provided information for the study

Consulting Company	Abbreviation	Consulting Package
PSM Consultants Pty Ltd	PSM	Geotechnical engineering - open pits
SRK Consulting Pty Ltd	SRK	Geotechnical engineering and mine engineering – WKP & MUG
GWS Ltd	GWS	Hydrogeology and groundwater
Engineering Geology Ltd	EGL	Waste rock disposal and tailings storage
Entech Pty Ltd	Entech	Geotechnical engineering, ventilation and mine engineering – MUG
GHD Pty Ltd	GHD	Materials handling - WKP
AECOM Pty Ltd	AECOM	Mining engineering - MUG
AMC Pty Ltd	AMC	Geotechnical engineering - MUG
Outotec Pty Ltd	Outotec	Paste backfill design
RSC Consulting Ltd	RSC	Sampling and geological modelling peer review
Beck Engineering Pty Ltd	Beck	Geotechnical numerical modelling - MUG.
Stantec	Stantec	Road design and costing
C&R Contracting, Redbull, DrillConnex	C&R	Methodology and costing for Martha pit operations.

2.3 Qualified Persons

The Qualified Persons (QP) for the report are OceanaGold employees engaged for the preparation of this Technical Report, as listed in Table 2-2. All the QPs except David Carr are based permanently on-site in Waihi. David Carr has been based in Dunedin, New Zealand in 2020 and has inspected the property several times throughout 2020.

Table 2-2: Qualified Persons who are responsible for preparing this Technical Report

QPs	Employer	Position	Technical Report Item(s) Contributed to or Reviewed
Trevor Maton (not Independent) BSc., M.Sc. MAusIMM (CP Mining), ARSM,	OceanaGold	Study Manager	Sections: 1.1 - 1.5, 1.16 - 1.22, 2, 5 & 16, 18 - 26
Peter Church (not Independent) BSc., MAusIMM (CP Geology)	OceanaGold	Principal Resource Development Geologist	Sections: 1.7 - 1.11, 1.13 & 1.21, 2 – 4, 6 – 12, 14, 23 - 26
David Carr (not Independent) BSc., MAusIMM (CP Metallurgy)	OceanaGold	Chief Metallurgist	Sections: 2, 3, 13, 17, 21.2, 23 - 26

2.4 Effective Dates

The effective date of this Technical Report is 31 March 2021.

2.5 Information Sources and References

OceanaGold has sourced information from appropriate reference documents as cited in the text and as summarised in Section 27 of this report. Additional information was provided by OceanaGold site personnel. The QPs have relied upon OceanaGold experts in the fields of mineral tenure, surface rights, permitting, social responsibility and environment.

2.6 Units of Measure

The metric system has been used throughout this report except for contained metal which is expressed in troy ounces. Tonnes are metric of 1,000 kg, or 2,204.6 lb. All currency is in New Zealand dollars (NZ$) unless otherwise stated.

3 RELIANCE ON OTHER EXPERTS

3.1 General

The Authors used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Authors do not consider them to be material. The Authors have relied upon OceanaGold for information regarding the surface land ownership/agreements as well as the mineral titles and their validity. Land titles and mineral rights for the project have not been independently reviewed by the Authors.

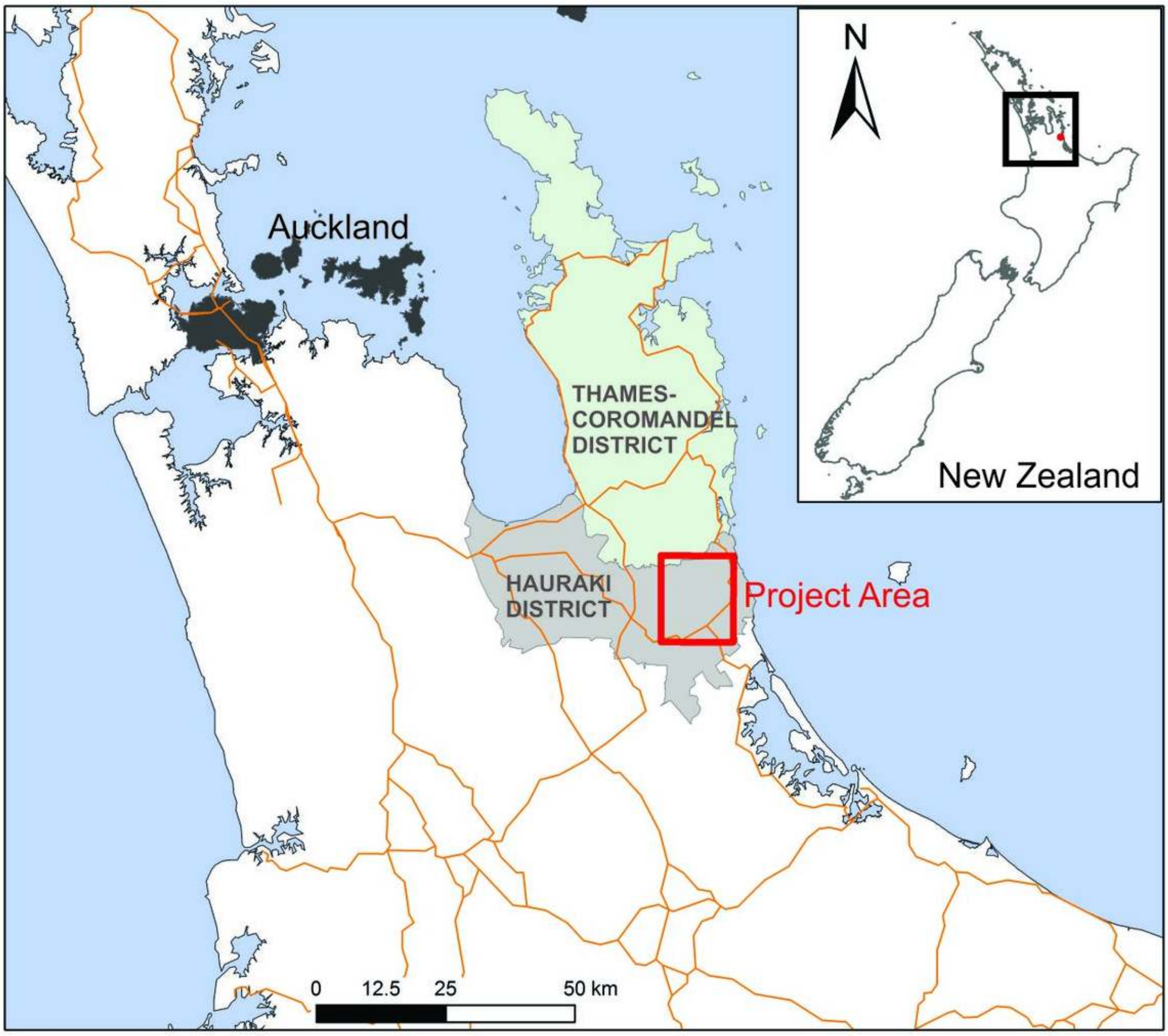
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The project area is located within the Hauraki District (Figure 4-1) and the Waikato region in the North Island of New Zealand. Martha operations are located within the world class gold mining town of Waihi.

Waihi is a small town situated approximately 142 km southeast of Auckland city. The MUG, MOP5 and GOP Projects are within the Waihi town. The WKP Project is located approximately 10 km north of Waihi.

Figure 4-1: Project Location Map



4.2 Mineral Tenure

All naturally occurring gold and silver minerals in New Zealand are owned by the Crown. Rights to prospect, explore or mine for these minerals are granted by permits issued under the Crown Minerals Act 1991 (CMA). Mineral EPs provide a permit holder the exclusive rights to explore for the specified minerals in an area. Minerals mining permits grant the holder to exclusive rights to mine for the specified minerals. The right to exchange a permit for a subsequent mining permit may occur provided certain criteria specified in the CMA are met, available at:

http://www.legislation.govt.nz/act/public/1991/0070/latest/whole.html#DLM246338

Figure 4-2 shows the location and extent of minerals permits held by OceanaGold within the Waihi WKP area as at 31 May 2020. The MUG, MOP5 and GOP Projects fall within Favona Mining Permit 41 808 (MP 41808). A small portion of the MUG Project (mainly the south-western extent of the Rex vein) falls within an area that was formally Waihi West Exploration permit (EP 40 767) and is currently under an application lodged with New Zealand Petroleum and Minerals (NZPAM) for an extension to MP 41 808 (Figure 4-2). The WKP Project falls within EP 40 598. Each of these permits are discussed in more detail below.

4.2.1 Favona Mining Permit MP 41 808

Of the projects outlined within this document, the MUG, MOP5 and GOP Projects lie within MP 41 808.

Favona MP 41 808 was granted to Welcome Gold Mines Limited and AUAG Resources Limited on the 22 March 2004 for the duration of 25 years, for gold and silver minerals. Work began on this permit with the development of the Favona decline in 2004 and the extraction of ore in late 2006. Two extensions of land to MP 41 808 were obtained in 2006 and 2017 firstly to include the Trio and potential extensions to Martha ore bodies and secondly to incorporate the land area previously covered by Mining Licence 32 2388 (ML 32 2388) prior to its expiry in July 2017.

MP 41 808 currently extends across an area of 1572.59 ha utilised by urban, rural and mining land use. All gold mining activities by OceanaGold in Waihi including the current underground mining operation, the ore Processing Plant, Tailings Facility and the inactive Martha Open Pit lie within the existing MP 41 808. The permit is 100% owned by OceanaGold (New Zealand) Limited. The permit will expire on 21 of March 2029. The Company has made an application for an extension of duration for MP41808, a decision on this application is currently pending (as at 1st March 2021).

On MP 41 808 the higher of a 1.0% royalty on net sales revenue from gold and silver or 5% accounting profits is payable to the Crown. 88 ha in the south western portion of MP 41 808 is subject to an additional 2% royalty payable to Osisko Royalties Ltd (acquired from Geoinformatics and BCKP) (Figure 4-2).

4.2.2 Hauraki Exploration Permit EP 40598

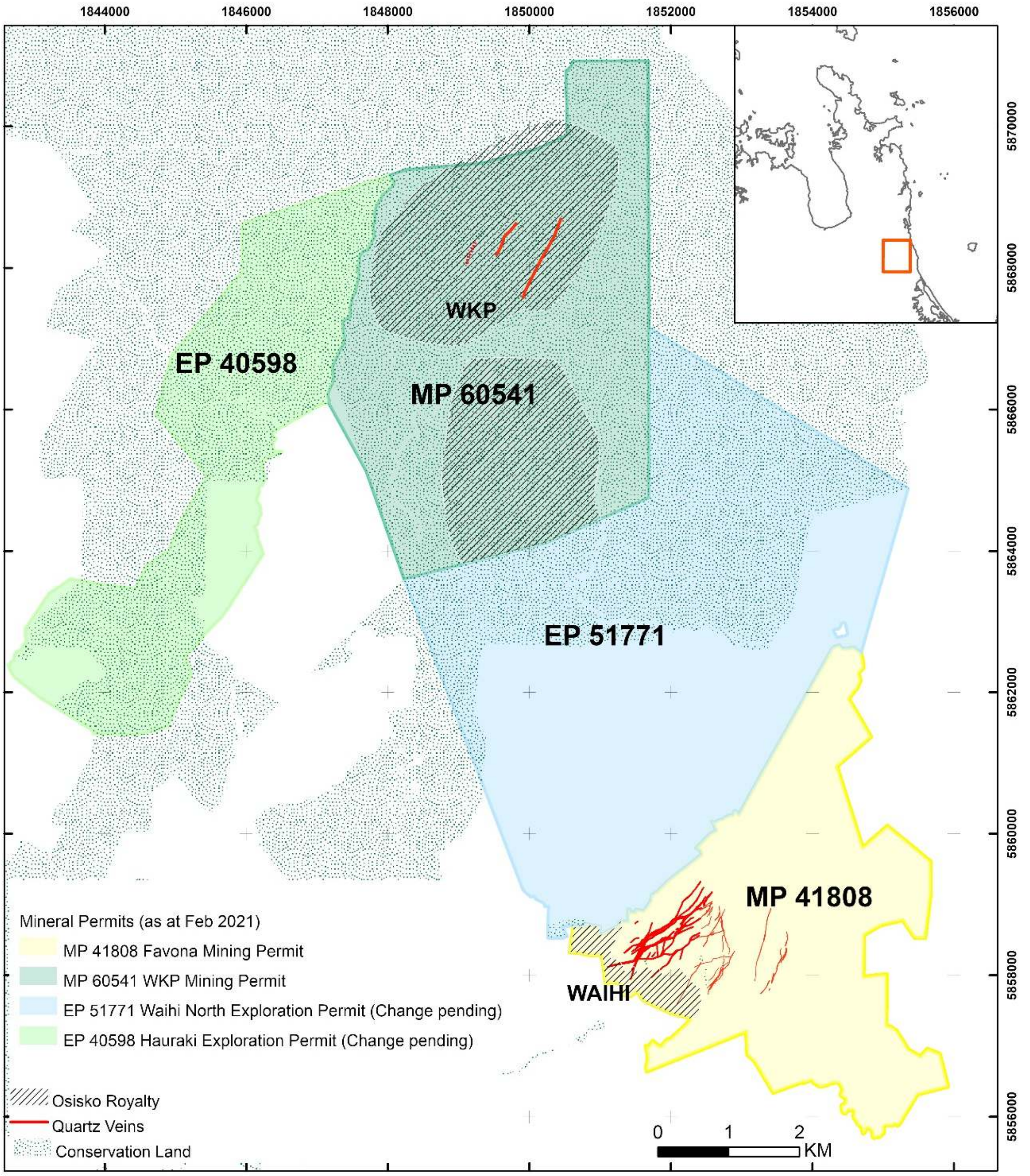
The WKP Project is situated within the MP 60541. This permit was granted on the 5th August 2020 for the duration of forty (40) years.

The permit currently covers an area of 2374.08 ha which is held (100%) by OceanaGold (New Zealand) Limited. Third-party rights to receive an interest in the project are confined to a Crown royalty of 1% of the turn over or 5% of the accounting profits whichever is higher and a 2% royalty payable to Osisko Royalties Ltd (acquired from Geoinformatics and BCKP) with respect to certain ‘target’ areas (Figure 4-2). In both cases the royalties are fixed and quantifiable for the purposes of inclusion in the business plan.

4.2.3 Other Exploration Permits

Table 4-1details the full set of permit interests held by OceanaGold (New Zealand) Limited on the North Island of New Zealand as at 28 February 2021 including rights to explore for minerals in the vicinity of the Waihi Mine and within the wider Hauraki and Thames Coromandel area.

Figure 4-2: Location of OceanaGold Tenements near the Waihi and WKP Projects



Source: NZPAM online permits webpage (http://data.nzpam.govt.nz/PermitWebMaps/?commodity=minerals)

Table 4-1: Tenement Status 28 February 2021

Permit	Location	Permit Type	Status	Granted	Term (years)	Expires	Area (ha)
41808	Favona	Mining	Active	22/03/2004	25	21/03/2029	1572.59
60541	WKP	Mining	Active	5/08/2020	40	4/08/2060	2374.08
51041	White Bluffs	Exploration	Active	15/10/2008	14	14/10/2022	450.973
51630	Ohui	Exploration	Active	22/06/2019	10	21/06/2023	1490.261
51771	Waihi North	Exploration	Active	28/04/2010	10	27/04/2020	3089.32
52804	Twin Hills	Exploration	Active	17/12/2010	10	16/12/2020	3223.786
40598	Hauraki	Exploration	Active	22/05/2003	18	21/05/2021	3762.94
40813	Glamorgan	Exploration	Active	7/09/2006	14	6/09/2020	2777.005
60149	Dome Field North	Exploration	Active	1/05/2017	5	30/04/2022	7287.262
60148	Dome Field South	Exploration	Active	1/05/2017	5	30/04/2022	10044.734
60528	Neavesville	Exploration	Active	31/07/2020	5	30/07/2025	2060.72

Permit	Application Pending	Permit Type
40813	Appraisal Extension of duration pending	Exploration
60149	Change of condition pending	Exploration
60148	Change of condition pending	Exploration
52804	Appraisal Extension of duration pending	Exploration
51771	Change pending	Exploration
41808	Change pending (extension of duration)	Mining

Source: NZPAM online permits webpage (http://data.nzpam.govt.nz/PermitWebMaps/?commodity=minerals)

4.3 Property Ownership and Access Arrangements

4.3.1 The MUG, MOP5 and GOP Projects

The operating MUG mine in Waihi is managed by OceanaGold (New Zealand) Limited, a 100% owned subsidiary of the OceanaGold Corporation. The locations of the MUG, MOP5 and GOP Projects are illustrated in Figure 4-3.

All gold mining activities in Waihi including the current underground mining operation, the Processing Plant, tailings facility and the inactive MOP are within the existing Favona MP 41 808. The land on which these activities take place is owned by various stakeholders including OceanaGold (Figure 4-3). In accordance with the requirements of the CMA, where mining activities involve surface disturbance on land not owned by OceanaGold an access arrangement with the landowner is required.

The MUG Project underlies land owned by various proprietors including the Crown (administered by Land Information New Zealand (LINZ)), Department of Conservation (DOC), the Hauraki District Council (HDC) and various private landowners. The portal to the underground mine is owned by OceanaGold which is accessed adjacent to the processing and water treatment plant.

The majority of the land covering the current Martha Open Pit is owned by the Crown and administered by LINZ. There are two parcels adjacent to the MOP that are administered for the Crown by DOC. OceanaGold has entered into an access arrangement with LINZ and a separate access arrangement with DOC providing an ongoing formal Authority to Enter and Operate on the various publicly owned land parcels for CMA purposes.

Land within the mining permit that hosts the conveyor belt corridor, the water treatment plant (and an associated pipeline for the discharge of the treated water into the Ohinemuri River), the process plant and the tailings storage facilities, is all owned by OceanaGold except for one parcel where the conveyor belt corridor runs through land adjoining the Union Hill area, which is in the name of the Commissioner of Crown Lands administered by LINZ, and portions of public roads, road reserve and river reserve. OceanaGold has entered into an access arrangement with LINZ providing an ongoing formal Authority to Enter and Operate on the conveyor belt corridor for CMA purposes.

The GOP, MOP4 and MOP (Phase 5) Projects occur on land owned by OceanaGold and government agencies (Figure 4-3). No access arrangements have been made for the MOP (phase 5) and GOP Projects at this stage.

The MUG Project requires mining beneath privately owned land. Section 57 of the CMA states that:

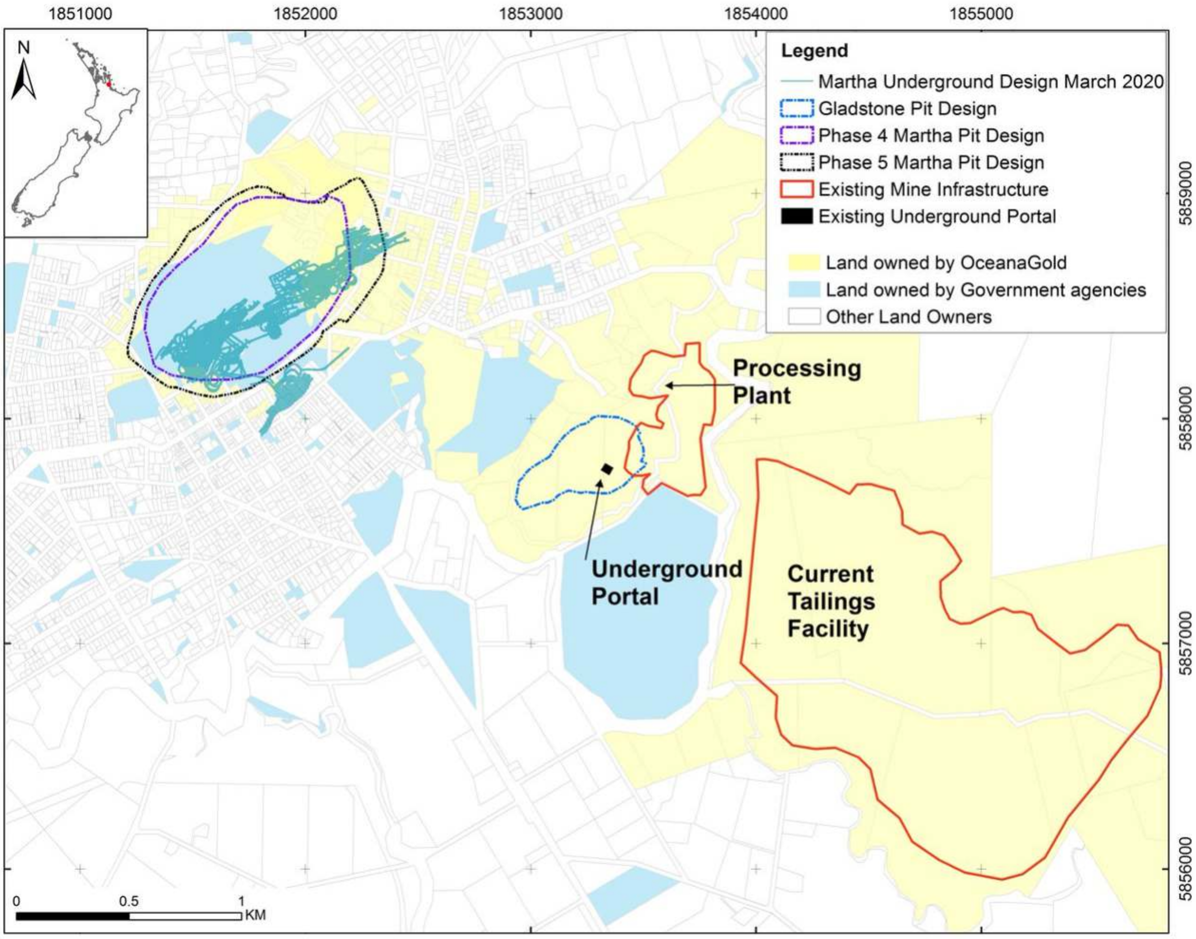
For the purposes of sections 53 to 54A, prospecting, exploration or mining carried out below the surface of any land shall not constitute prospecting, exploration or mining on or in land if it:

- Will not or is not likely to cause any damage to the surface of the land or any loss or damage to the owner or occupier of the land; or
- Will not or is not likely to have any prejudicial effect in respect of the use and enjoyment of the land by the owner or occupier of the land; or
- Will not or is not likely to have any prejudicial effect in respect of any possible future use of the surface of the land.

Resource consents currently in place set out a process (including arbitration) for gaining access to land above stopes and development drives. Specific conditions of the Principal Land Use resource consent for the MUG Project include the following:

- At least three months prior to the placement of the first explosives for any blasts immediately beneath any part of the legal title to a residential property overlying stopes for any mining provided for under this consent, the consent holder shall offer to purchase that property from the registered proprietor at market value this offer shall be set by reference to the two independent valuations required by condition 52); or if the registered proprietor prefers, to provide an ex gratia payment equal to 5% of the property's market value to the registered proprietor.
- Prior to the placement of the first explosives for any development blasts immediately beneath any part of the legal title to a residential property for any mining provided for under this consent, the consent holder shall offer to provide an ex gratia payment equal to 5% of the property's market value to the registered proprietor of that title.
- If the company's offer is not accepted, but the registered proprietor wishes to negotiate, the consent holder shall offer to commit to a binding arbitration process in relation to the property purchase or ex gratia payment referred to above, provided that the basis for determining the ex gratia payment is not amenable to further negotiation.

Figure 4-3: Location of the Projects within the Favona Mining Permit (NZTM grid)



Source: OceanaGold GIS database and LINZ online GIS database (https://data.linz.govt.nz/data/category/property-ownership-boundaries/)

4.3.2 The WKP Project

The WKP Project is located within MP 60541, on land owned by the Crown and administered by DOC as a conservation/forest park. An access arrangement between DOC and OceanaGold has been made to allow for exploration activities (including surface drilling) to take place within MP 60541. Known environmental liabilities are managed through stipulated conditions in the DOC access arrangement and Regional and District Council Consents and under conditions that protect the conservation (biodiversity and amenity) values of the land.

DOC access arrangement conditions include:

- Submission of an “Annual Work Program” to obtain an “Authority to Enter and Operate” for a 12 month period;
- Ecological surveys are to be undertaken by suitably qualified and experienced experts over areas requiring vegetation clearance such as drill sites, campsites, pump sites and helicopter landing sites;
- A DOC approved kauri dieback management plan must be in place and followed;
- Historical and cultural sites must be protected against damage; and
- All cleared sites are to be rehabilitated to the satisfaction of the DOC manager.

4.4 Consent and Permitting

The regulatory agencies primarily responsible for consents, permits, authorities and licences associated with the projects are as follows:

- NZPAM, which is the Crown agency responsible for administering rights to explore for and extract Crown-owned minerals, including gold and silver, under any Mining and Exploration Permits under the CMA;
- The Waikato Regional Council (WRC) formerly Environment Waikato is the regional government agency appointed under the Resource Management Act 1991 (RMA) responsible for air and water quality within the Waikato region (this includes Waihi and WKP). It manages activities including water extraction and discharge, vegetation removal and earthworks that can give rise to soil erosion;
- The HDC is the local government agency appointed under the RMA responsible for the management of land use, community issues and resource consents for the Hauraki District which includes the Waihi Project areas (i.e. MUG, MOP5, GOP and WKP Projects);
- The Thames Coromandel District Council (TCDC) is the local government agency appointed under the RMA responsible for the management of land use, community issues and resource consents within the Thames Coromandel District, which includes some of OceanaGold's regional exploration tenements;
- Heritage New Zealand Pouhere Taonga is the government agency appointed under the Heritage New Zealand Pouhere Taonga Act 2014 to grant authority to modify or destroy any archaeological site. Any impacts on old mine workings or old surface structures, where these pre-dates 1900 or are otherwise specifically protected by law, will require such an authority; and
- DOC is the government agency charged with conserving New Zealand's natural and historic heritage and managing public conservation land. To undertake mining activities that have a surface expression on public conservation land permission from the Department is required. This permission is in the form of a minimum impact arrangement or an access arrangement. Additionally, DOC is responsible for managing wildlife species under the Wildlife Act 1953 and for issuing Wildlife Act Authorities for activity that includes holding, catching, handling or release of wildlife.

An updated Land Use Consent (202.2018.00000857) for the MUG and the MOP4 Project was granted by HDC on the 12 December 2018 and after an appeal period expired, commenced on 1 February 2019. OceanaGold commenced development activity on the 27 of July 2019. This Land Use Consent allows for mining of the MUG resource and the remainder of the MOP4. In addition to the authorisations required by HDC, a suite of consents were obtained from WRC covering matters such as vegetation removal, water takes, diversions and discharges of water, discharges to air and construction of the tailing's storage facilities. Both HDC and WRC have conditions in place relating to mine closure, bonds and a post closure trust.

The MOP5, WKP and GOP Projects are in 'pre-planning' phase and have no consents in place for mining at this stage.

For exploration drilling at WKP a consent to abstract water for drilling purposes was obtained from WRC. Water take is metered, and maximum abstraction rates are stipulated within the granted permits. Restrictions apply if stream levels fall below specified thresholds during periods of reduced rainfall. Land use consents have also been obtained from HDC prior to vegetation clearance for drilling purposes and building consents are in place for drill platforms that have been erected to minimise ground disturbance.

Any development of the prospect for the purposes of advancing beyond exploration would require applications at that time under the RMA and (for surface impacts only) the CMA. Consent has not yet been sought for mining the WKP Project.

Changes to NZ government policy restricting access to mine on conservation land have been proposed, subject to a statutory consultation process that has not yet commenced. The precise nature of any proposal is not currently known.

4.5 Easements and Road Access

For MUG, public road access is provided to the OceanaGold underground amenities and Processing Plant site through Baxter's Road and to the open pit mine by Seddon Street. A number of paper roads exist within the mining area. No additional agreements are required except in the event that OceanaGold imports significant quantities of waste rock from the local quarry, in which event the Company is required to fund certain road upgrades.

For the GOP and MOP5 Resources a number of paper roads within the mining area will be stopped and/or realigned using processes under the Public Works Act 1981 and/or Local Government Act 1974.

5 ACCESSIBILITY, CLIMATE, PHYSIOGRAPHY, LOCAL RESOURCES AND INFRSTRUCTURE

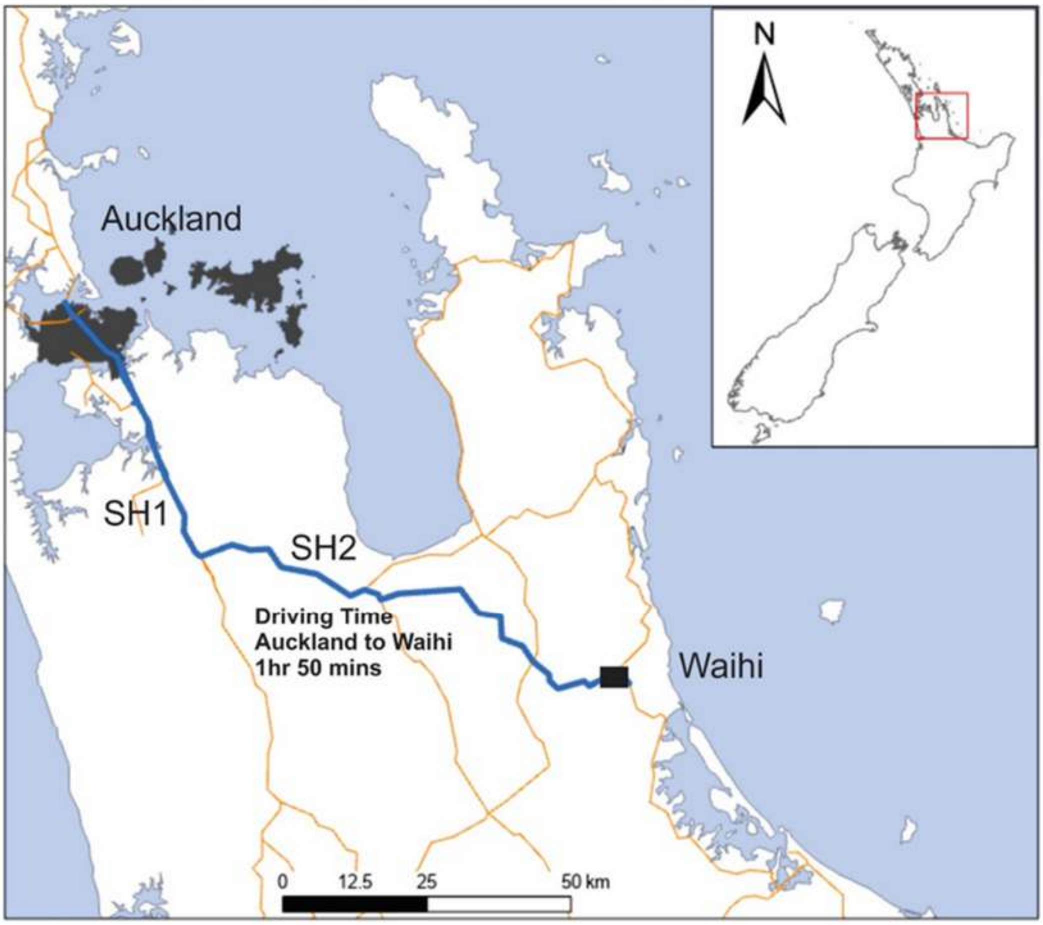
5.1 Accessibility

The Waihi site is located within the township of Waihi in the North Island of New Zealand and close to the major cities of Auckland (150 km north), Tauranga (60 km south) and Hamilton (100 km west). The project is located in North Island of New Zealand approximately 140 km southeast of Auckland, Figure 5-1.

Auckland (population approximately two (2) million) is the largest city in New Zealand. The project is close to the community of Waihi (population approximately 4,000) and Waihi Beach (population approximately 3,000). The communities have supported the mining industry in the area for well over 120 years with the history of mining in the area dating back to the late 1800s. Approximately 300 employees live in the area.

Road access between Waihi and the major centres of Auckland and Tauranga is good, along a dual carriage highway (State Highway 2). No rail access is available to the site.

Figure 5-1: Project Access



Source: Google Maps 2020 and LINZ online GIS database (https://data.linz.govt.nz/data/category/roads-and-addresses/)

5.2 Climate and Physiography

The Waihi township is at the foot of the Coromandel Peninsula, to the west are the hills of the Kaimai Ranges.

The climate is temperate. Mean temperatures range from 8 °C (46 °F) in the South Island to 16 °C (61 °F) in the North Island. January and February are the warmest months, July the coldest. New Zealand does not have a large temperature range, but the weather can change rapidly and unexpectedly. Winds in New Zealand are predominantly from the West and South-West, in winter, when the climate is dominated by regular depressions. In summer, winds are more variable with a northerly predominance associated with the regular large anti cyclones which cover all the country.

New Zealand is seismically active. In the Waihi region:

- Earthquakes are common, though usually not severe, averaging 3,000 per year. Mostly less than three on the Richter scale;
- Volcanic activity is most common on the central North Island Volcanic Plateau approximately 200 km to 300 km from Waihi;
- Tsunamis would not have any direct impact on Waihi;
- Droughts are not regular and occur less frequently over much of the North Island between January and April; and
- Flooding is the most regular natural hazard.

5.3 Local Resources and Infrastructure

Almost all of the employees reside in the nearby towns of Waihi, Waihi Beach, Katikati, Thames and Paeroa. Waihi is a relatively small community of approximately 5,403 people. Statistics New Zealand Census information shows that population numbers have grown since 2013 by 741 people (15.9%). The population pyramid from the 2018 Census shows a noticeable dip in the numbers of young people in the range from 20-30.

The largest sectors for employment in Waihi are the retail trade (16.5%), health care and social assistance (14%), education and training (10%) and manufacturing (10%). Mining is relatively high at 3.2% of the usual resident population compared to the Waikato region at 0.5% and New Zealand at 0.2%.

The employment status of those at least 15 was that 1,455 (32.5%) people were employed full-time, 690 (15.4%) were part-time, and 225 (5.0%) were unemployed. In addition, average wages in Waihi are lower than regional averages and are skewed towards lower income levels.

Community, health and education services are well established in Waihi with four primary schools, one secondary school, medical centres and various community health centres present. Most establishments are government funded.

A local service industry has established itself over the last 20 years to support gold mining in Waihi comprising engineering, cleaning, maintenance, drilling, rental and consumable suppliers, security, labour hire and other services. More technically advanced services are available from the regional centres in terms of heavy engineering, large equipment hire and other specialised services. Most suppliers are privately run and not affiliated with OceanaGold.

5.4 Mining Area Infrastructure

A shortage of backfill material exists for the MUG and this will be sourced from the Martha pit. It is expected that Mineral Reserve from the Martha underground mine will be processed relatively quickly with the ROM material stockpiled in the ROM stockpile located near the conveyor in the processing area.

All rock from the Martha pit is crushed and conveyed by an existing overland conveyor, either to the process plant area or the TSF's.

The maximum stockpile capacity at the Martha Pit is 200,000 t and a 1.2 Mt stockpile facility has been constructed adjacent to the Processing Plant.

Some stockpiling of material will be required to enable waste production to be scheduled in accordance with backfill requirements. The stockpile areas near the Favona portal will be used for the temporary storage of waste rock.

5.5 Comments on Accessibility, Climate, Local Resources, Infrastructure, and Physiography

In the opinion of the QPs:

- The existing and planned infrastructure, availability of staff, the existing power, water and communications facilities, the methods whereby goods could be transported to any proposed mine, and any planned modifications or supporting studies are well established and can support the declaration of Mineral Resources and Mineral Reserves;
- Within OceanaGold land holdings, there is sufficient area to allow construction of any required project infrastructure; and
- The QPs consider it a reasonable expectation that surface rights usages will continue to be granted for the projects with appropriate negotiation.

6 HISTORY

6.1 Waihi

The town of Waihi became established when the original Martha Mine opened as an underground operation in 1879. The mine was extremely productive producing approximately 1,056 tonnes of gold-silver bullion from about 12 million tonnes of ore by 1952. The historic mine extracted five main sub-parallel lodes (the Martha, Welcome, Empire, Edward and Royal) together with numerous branch and cross lodes. All lodes dip steeply and are fillings of extensional faults and fractures. Early stoping employed the cut and fill method, but this was phased out and largely replaced after 1914 by the shrink stoping method. Stopes were generally not backfilled after 1914 but left open. The workings reached a total depth of 600 m from surface on 16 levels. Seven (7) main shafts were used to access miners and supplies underground, and numerous other shafts were developed for ventilation and exploration. In 1894, the Waihi Gold Mining Company adopted the cyanide process for gold extraction, which was first trialled in the world at a nearby mine in Karangahake.

Exploration drilling between 1979 and 1984 by Waihi Mining and Development Ltd. and AMAX Exploration Ltd. identified large open pit reserves within the confines of the historic mining area. Following the granting of permits, the Martha Open Pit operation commenced operation in 1988 (under ML 32 2388) as an unincorporated joint venture between subsidiaries of Normandy Mining Limited Group and Otter Gold Mines Ltd. The Otter Gold holding was acquired by Normandy in 2002 and the Newmont Mining Corporation acquired full ownership of the Waihi Mine in 2002 through the acquisition of the Normandy Mining Group.

The Martha Mine Open Pit on ML 32 2388 produced 22 Mt@3.1 g/t Au (2.2 Moz.) between 1988 and April 2015 when a localised failure of the north wall undercut the main access ramp suspending open pit mining operations. There is no open pit mining currently active in Waihi. The ML 32 2388 expired in July 2017 and was amalgamated into the existing Favona MP 41 808.

MP 41 808 was granted in March 2004 for a duration of 25 years to mine the Favona ore body. Underground mining resumed at Waihi in 2004 with the development of the Favona mine located approximately two (2) km east of the Martha Pit. Mining of the Favona ore body led to further extensions of underground development towards the nearby Moonlight, Trio and Correnso deposits. The Correnso ore body has completed mining apart from a small amount of handheld narrow vein mining.

OceanaGold obtained full ownership of the Waihi property as an operating Open Pit Mine, Underground Mine and Processing Plant in October 2015. Resource consent for underground mining of the remnant mineralisation around the Martha Vein System and the MOP4 was granted on the 12 December 2018, increasing the current underground and pit mine life in Waihi.

Table 6-1 summarizes the annual production from Waihi since 1988. Figure 6-1 shows a map of Waihi illustrating the areas mined through time.

6.2 WKP

Early prospecting and mining at WKP were attempted between 1893-1897, but only 19 oz. of gold bullion was recovered from a 14-ton test parcel and mining was soon abandoned. Modern prospecting and exploration recommenced in 1978-1993 by Amoco, BP and others which included 5,500 m of drilling in 23 drill holes. Newmont acquired a controlling interest in the property in 2005 and started a reconnaissance geological mapping, sampling, CSAMT geophysics and drilling campaigns targeting high-grade underground minable veins. In 2010, hole WKP-24 intersected the main T-Stream Vein containing 156 m @ 1.6 g/t Au. Wide spaced follow up drilling confirmed the presence of three prospective vein zones each striking more than one (1) km in length, namely the Western Vein, the T-Stream Vein and the EG Vein. Newmont completed 7,000 m of DD in 15 holes intersecting locally high-grade gold mineralisation in each hole. Newmont ceased exploration in 2013 and the prospect remained idle until 2015 when OceanaGold acquired Newmont's New Zealand assets. Exploration then continued with additional geological mapping, sampling and geophysics leading up to further DD along the EG Vein (WKP40 to WKP 68). Drilling intersected significant Au mineralisation along the EG Vein including but not limited to 7.6 m (true width) averaging 10.84 g/t Au in the first hole (WKP40). Drilling has since intercepted significant gold mineralisation along this veined structure from which a resource estimate has been calculated.

6.3 Previous Studies and Resource Estimates

Resource estimates and exploration results have previously been publicly reported for the MUG, WKP and GOP Projects. These reports are available through the OceanaGold company website at https://www.oceanagold.com/investor-centre/filings/ and include the following:

- JORC Table 1 Waihi (March 2020)
- JORC Table 1 WKP (February 2020)
- JORC Table 1 WKP (November 2019)
- JORC Table 1 Waihi (September 2019)
- Material Information Summary and JORC Table 1 Waihi (March 2019)
- Material Information Summary and JORC Table 1 WKP (February 2019)

6.4 Historical Data

It is estimated that the historical Martha underground produced 4.9 Moz gold between 1883 and 1952 The underground workings extended over 15 vertical levels, 600 m deep and 1.6 km along strike. The Martha vein system was then mined from an open pit which produced approximately two (2) Moz of gold between 1988 and 2015.

Underground mine production recommenced in 2006 at the Favona vein system situated approximately two (2) km southeast of the Martha deposit followed by the Union-Trio-Amaranth vein system and Correnso Vein System. The Favona deposit produced approximately 400 koz of gold between 2006 and 2013, Trio produced approximately 200 koz of gold between 2013 and 2015 and Correnso approximately 557 koz between 2015 and 2019. Recent production figures from Waihi are presented in Table 6-1.

Mining of ore from the upper portions of the Correnso deposit continues at present using handheld airleg mining techniques and consequently ore extraction is insufficient to support regular ore processing. OceanaGold intends to campaign treat the mined ore in Q1, 2021 and resume regular ore processing by Q2, 2021.

Large scale mechanised underground mining has continued uninterrupted since 2008. Current mining is heavily focused towards the capital development associated with establishing the MUG Mining Project and productivity levels have been increased significantly in recent months to support this new project.

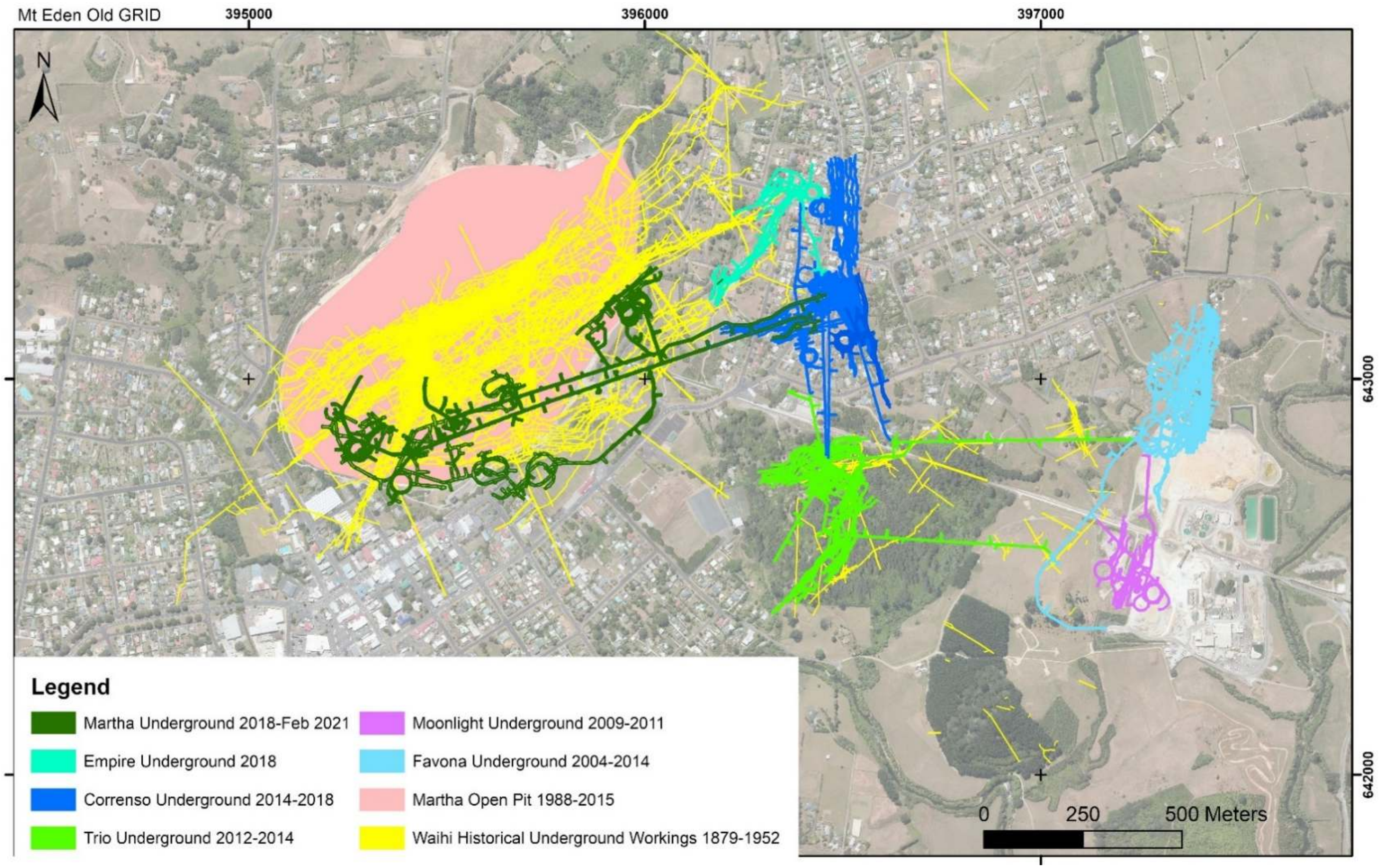
No significant gold production is recorded from WKP, apart from 19 oz Au recovered from a 14-ton test parcel in the late 1890s.

Table 6-1: Historic Production Post 1988

	Martha		Mined	Recovered	Favona		Mined	Recovered	Trio		Mined	Recovered	Correnso		Mined	Recovered	Martha UG		Mined	Recovered
Year End	Tonnes	Au(gpt)	Au(KOz)	Au(KOz)	Tonnes	Au(gpt)	Au(KOz)	Au(KOz)	Tonnes	Au(gpt)	Au(KOz)	Au(KOz)	Tonnes	Au(gpt)	Au(KOz)	Au(KOz)	Tonnes	Au(gpt)	Au(KOz)	Au(KOz)
30/06/1988	68,179	2.4	5.3	3.6																
30/06/1989	775,240	2.8	69.8	63.1																
30/06/1990	879,294	3.1	87.6	78.9																
30/06/1991	858,173	3.4	93.8	84.2																
30/06/1992	834,472	3.1	83.2	74.5																
30/06/1993	817,003	3.2	84.1	75.7																
30/06/1994	800,203	3.3	84.9	77.8																
30/06/1995	880,580	2.5	70.8	66.4																
30/06/1996	892,859	2.9	83.3	79.2																
30/06/1997	915,135	3.0	88.3	82.7																
30/06/1998	917,346	3.1	91.4	85.6																
30/06/1999	907,790	3.6	105.1	95.5																
30/06/2000	1,030,062	3.3	109.3	102.0																
30/06/2001	1,202,938	2.7	104.4	95.1																
30/06/2002	1,343,925	3.3	142.6	129.9																
31/12/2002	638,210	3.5	71.6	64.4																
31/12/2003	1,231,521	3.1	120.8	109.7																
31/12/2004	1,274,790	3.4	141.0	127.6																
31/12/2005	1,158,385	4.8	180.2	167.7																
31/12/2006	794,231	4.0	102.9	97.0	135,304	7.9	34.2	30.0												
31/12/2007	273,414	1.7	15.2	13.3	225,276	11.1	80.1	72.2												
31/12/2008	536,360	1.9	32.6	29.7	330,619	11.1	118.0	101.5												
31/12/2009	951,481	2.0	62.4	57.7	333,103	8.2	87.8	79.4												
31/12/2010	564,031	2.4	44.1	39.7	367,577	6.2	73.8	66.1												
31/12/2011	691,763	2.5	54.5	48.9	304,609	6.0	58.4	51.6												
31/12/2012	15,972	4.8	2.5	2.2	51,580	5.6	9.3	8.6	340,391	5.4	59.1	54.6								
31/12/2013	165,569	2.8	14.8	12.8	52,200	4.3	7.2	6.5	463,854	6.4	95.7	88.0								
31/12/2014	684,473	3.1	68.0	61.7	6,820	7.4	1.7	1.6	301,694	7.7	75.1	69.1	7,912	2.8	0.7	0.6				
31/12/2015	234,935	3.3	25.2	24.3									474,036	8.8	133.7	119.5				
31/12/2016													489,300	8.1	126.1	116.0				
31/12/2017													472,450	8.6	130.4	119.1				
31/12/2018													433,593	6.77	94.4	83.50				
31/12/2019													433,389	5.60	78.0	68.1				
31/12/2020													100,880	6.09	19.7	17.6	36,527	2.10	2.5	2.2
Total	**22,338,334**	**3.1**	**2,240**	**2,051**	**1,807,088**	**8.1**	**470**	**417**	**1,105,939**	**6.5**	**230**	**212**	**2,411,560**	**7.5**	**583**	**524**	**36,527**	**2.1**	**2**	**2**

Source: OceanaGold mine records and archives

Figure 6-1: Map Showing the Mined Areas in Waihi



Recent mining data is sourced from OceanaGold Internal mining database (February 2021), Historical mining data sourced from NZPAM mineral reports (https://data.nzpam.govt.nz/GOLD/system/mainframe.asp) and NZ mine plans (https://www.nzpam.govt.nz/maps-geoscience/nz-mine-plans/).

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

Both the Waihi and WKP projects are located within the Coromandel Peninsula which hosts over fifty gold and silver deposits that make up the Hauraki Goldfield. The peninsula is built up of Miocene to Quaternary volcanic rocks (the Coromandel Volcanic Zone) overlying a Mesozoic basement. It is bound to the west by the Hauraki Rift, a large graben filled with Quaternary and Tertiary sediments, and to the south by volcanics deposited by the presently active Taupo Volcanic Zone (TVZ).

A schematic geological map of the Coromandel Peninsula is shown in Figure 7-1. Jurassic greywacke basement and intruded granitic stocks and dykes of the Mania Hill Group are exposed in the northern part of Coromandel, becoming progressively down faulted to the south beneath younger volcanics. Coromandel geology is dominated by the CVZ, Miocene to Pliocene aged volcanics formed during three main phases of volcanism (Christie et al. 2007). The first phase constitutes the widespread andesites and dacites of the Coromandel group (18‑3 Ma). The second phase encompasses the predominantly rhyolitic units of the Whitianga Group (9.1-6 Ma) and the third phase is dominated by Strombolian volcanoes and dykes of the Mercury Bay Basalts (6.0-4.2 Ma) (Skinner 1986). Epithermal veins and hydrothermal alteration are observed within the Coromandel and Whitianga Groups.

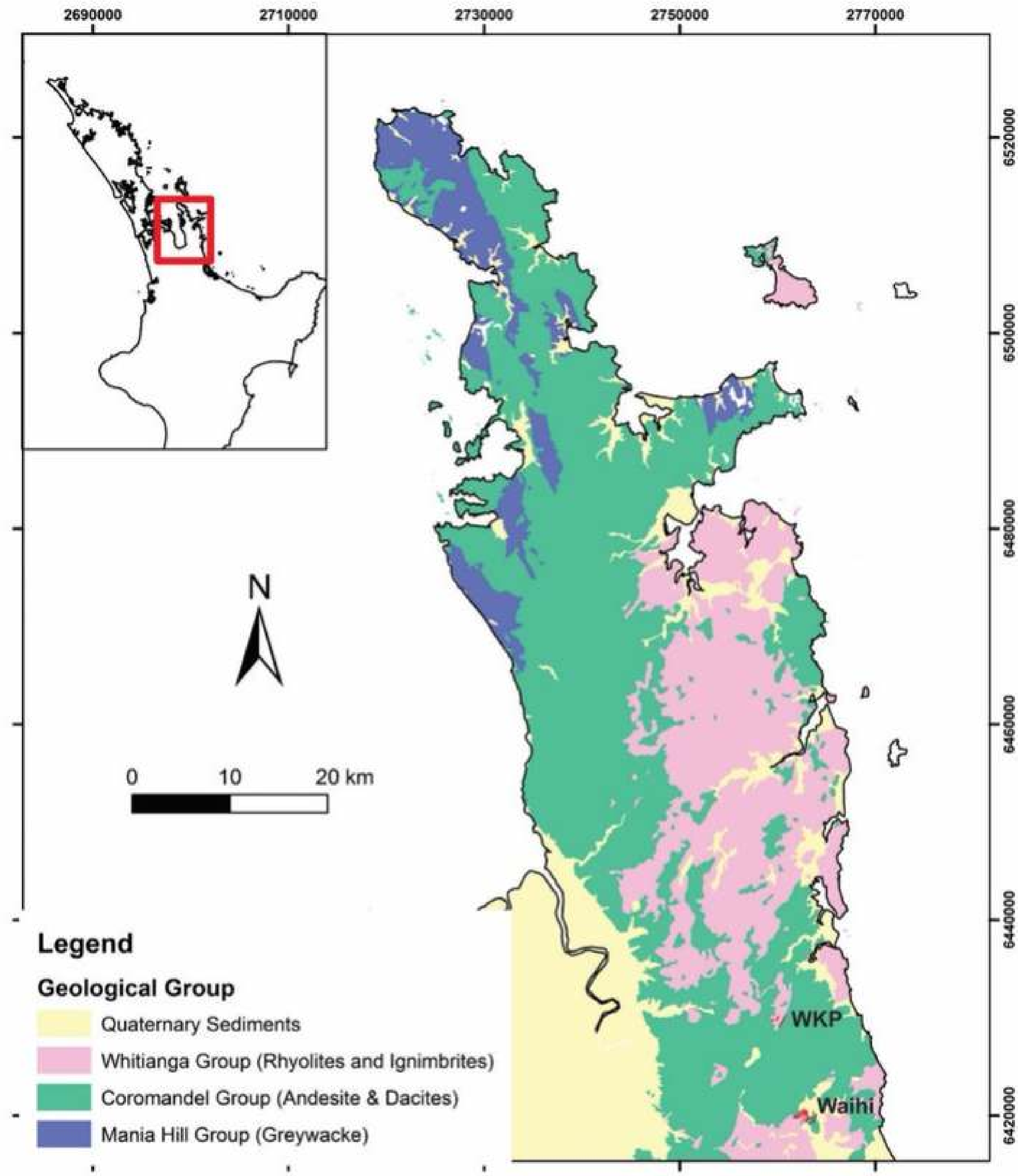
Coromandel group can be subdivided into the Kuaotunu Subgroup andesites, dacites and plutons, occurring in the northern region of the goldfield (ca. 18 to 11 Ma), the Waiwawa Subgroup andesites, dacites and rhyodacites in the south and east parts of the goldfield (ca. 10 to 5.6 Ma), and also the smaller Omahine (8.1 to 6.6 Ma) and Kaimai (5.6 to 3.8 Ma) andesite and dacite Subgroups in the southern parts of the goldfield (Edbrooke, 2001).

Mineralised sequences are overlain in places by post-mineral andesitic to dacitic flows of the Kaimai Subgroup, rhyolitic ignimbrites of the Ohinemuri Subgroup and more recent, Pleistocene age sediments and ash units. Although these post-mineral units do not blanket the mineralised units, they can be extensive and reach up to 1.5 km in thickness.

The CVZ hosts low- to medium-sulphidation epithermal Au-Ag and Cu porphyry deposits along its length (Christie et al. 2007). Porphyry Cu-Mo-Au deposits are associated with diorite-granodiorite composition intrusions and volcanic rocks dated between 18.1 Ma and 16.4 Ma. Epithermal deposits in the CVZ appear younger in age between 14-5 Ma.

The Au-Ag deposits of the Waihi District and WKP are classical low-sulphidation adularia-sericite epithermal quartz vein systems associated with north to northeast trending faults. The main ore minerals are electrum and silver sulphides developed within quartz veins. Other minerals present within the veins include ubiquitous pyrite and more localised adularia, calcite, illite, smectite, sphalerite, galena, chalcopyrite and rhodochrosite. Base metal sulphide content is low but generally increases with depth.

Figure 7-1: Regional Geological Map of the Coromandel Peninsula



Modified from IGNS 1:250 000 Geological Map (2001), (https://www.gns.cri.nz/Home/Our-Science/Land-and-Marine-Geoscience/Regional-Geology/Geological-Maps/1-250-000-Geological-Map-of-New-Zealand-QMAP)

7.2 Waihi Geology

The Waihi vein system is hosted within andesitic flows and pyroclastic units of the late Miocene (7.36-6.76 Ma) Waipupu Formation. The Waipupu Formation in Waihi can be subdivided into an upper quartz-phenocryst poor unit and a lower quartz-phenocryst rich unit which dip shallowly towards the SE. Some of the veining and gold mineralisation in Waihi appears to be better developed within the lower quartz-rich andesite flows, with the exception of the Favona, Moonlight and GOP deposits which are solely hosted within the upper andesite unit. Much of the mineralised andesites in Waihi are overlain by post-mineral rocks including dacite flows of the Uretara Formation (5.23 Ma), Pleistocene ignimbrites and recent ash deposits. Where veining is exposed close to the surface, the quartz-adularia altered andesites form resistant paleo-topo ‘highs’ that project through the post-mineral cover sequences.

A generalised map of the surface geology of Waihi and the location of veining at depth is illustrated in Figure 7-2. All known Au and Ag mineralisation in Waihi is confined to veining or vein fragment within hydrothermal eruption breccia. The major mineralised veins are typically coincident with dip-slip, normal faults believed to have formed in an extensional setting related to early, back-arc rifting of the TVZ dated at ca 6.1 Ma (Mauk et al 2011).

Some of the main mineralised veins within the Waihi area include the Martha Vein System (which includes the Martha, Empire, Welcome, Royal, Edward, Rex and Albert veins among many others) in the NW and the Correnso, Daybreak, Union, Trio, Amaranth, Favona, Moonlight and GOP veins progressively SE (Figure 7-2).

7.2.1 Martha Vein System

The Martha Vein System is the largest and most documented of the vein networks in Waihi. The veins are numerous and form a large network that extends for more than 1600 m along strike and 600 m below the surface. The vein network although complex in detail, simply comprises the dominant southeast-dipping Martha vein and several northwest-dipping hangingwall splays including the Empire, Welcome, Royal and Rex veins. The Martha vein is the largest vein structure reaching up to 30 m in thickness in places but averages 6-15 m wide. Increased vein widths are closely associated with the steepening of vein dips from an average of 65-70 ° to approximately 85 ° to the SE. Steeper portions of the vein tend to contain higher concentrations of Au and Ag. The vein itself comprises mainly intact brecciated quartz vein material evidence for vein emplacement during the late stages of dip-slip faulting. The quartz is characterised by multiphase brecciation and banding (colloform and crustiform) and quartz textures are highly variable from a fine, microcrystalline and chalcedonic character to more coarsely crystalline particularly at depth. Apart from the main Martha vein, the hanging wall splay veins are also significant mineralised structures reaching 18 m in width (e.g. the Empire Vein). The hangingwall splays closest to Martha link up with the Martha vein at depth often forming a higher-grade lode at the intersection. Hangingwall splays further away from Martha either thin out at depth or are not drilled deep enough to make out their relationship with Martha at depth (e.g. the Rex and Ulster Veins). Additional, smaller-scale splay veins are present linking the larger vein structures and form a valuable contribution to the mineralisation particularly in the Martha Open Pit. These splays typically comprise smaller veins between 5-50 cm in width infilling extensional structures with no fault displacement, dipping moderately towards the NW. Two steeply dipping, NNE-trending and well mineralised vein structures known as the Edward and Albert veins also form an important part of the overall Martha vein network.

The andesitic host rocks within proximity to veining have often undergone pervasive hydrothermal alteration, sometimes with complete replacement of the primary mineralogy. Characteristic alteration assemblages of the host rocks are dominated by argillic alteration (quartz+adularia+pyrite+illite) closest to veining and propylitic alteration (weak quartz+weak pyrite+ carbonate+ chlorite+ interlayered illite-smectite and chlorite-smectite clays) extending over tens of metres laterally from major veins. The degree of alteration within the Waihi District is variable and often dependent on the host rock lithology and the nearby veining. On rare occasions, some host rocks at or near the contact of large veins appears only weakly altered, for example the “hard bars” identified during the early historical mining of the Martha vein. Volcaniclastic units tend to have increased clay alteration compared to the flow units.

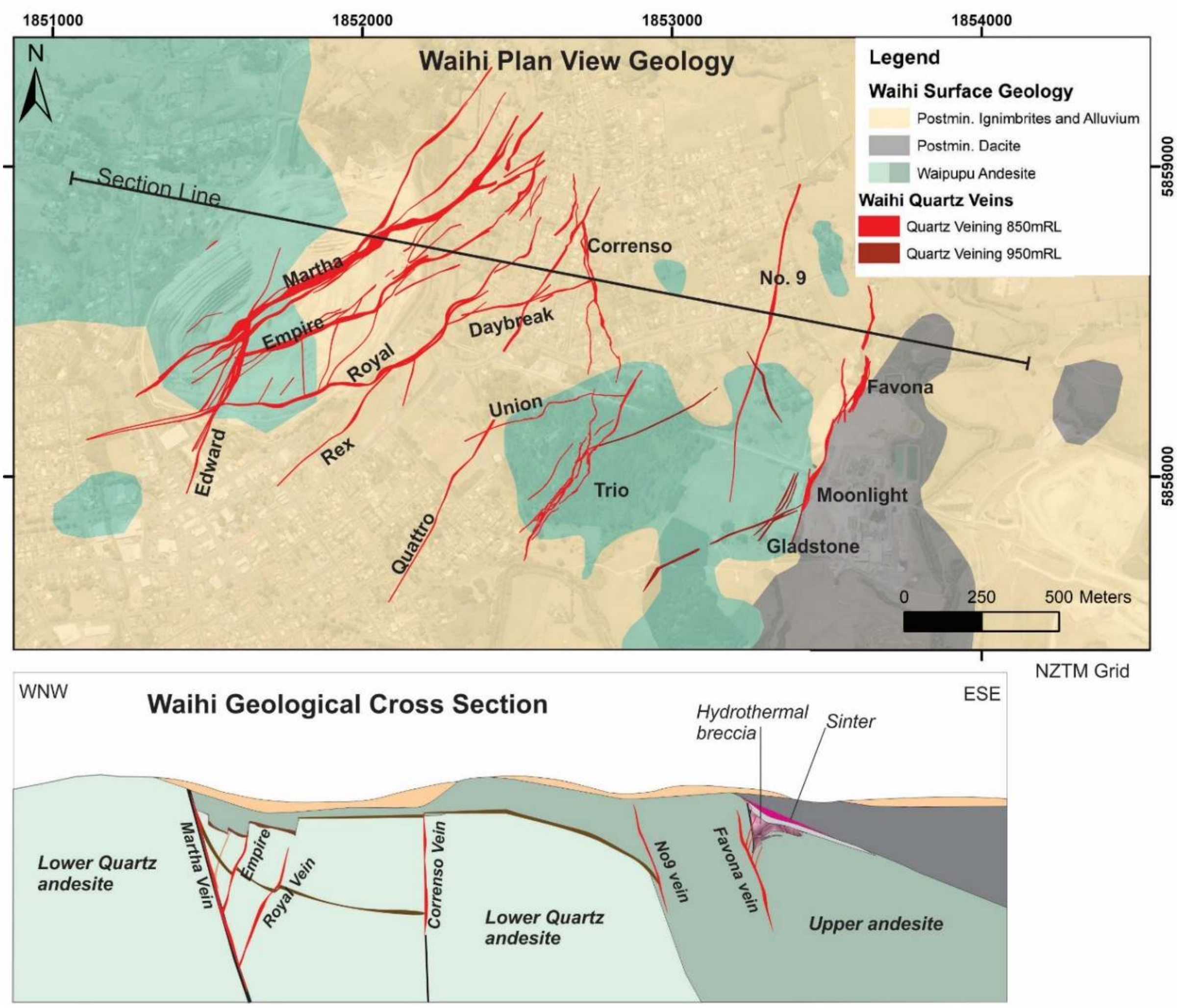
Gold occurs mostly as small inclusions of electrum (averaging 38% silver) occurring as both free grains in the quartz and as inclusions in sulphides such as pyrite, galena, sphalerite and less commonly chalcopyrite. Free gold is rarely observed. Acanthite associated with pyrite and galena is the main silver mineral.

Martha ore has silver to gold ratios of > 10:1, The Favona and Trio ores had silver to gold ratios of ~ 4:1, and Correnso ore had a silver to gold ratio of less than 2:1.

The base metal sulphide content is low but is observed to increase in concentration with depth within all the Waihi veins. Sphalerite and galena are the most abundant base metal sulphides while chalcopyrite is less common and pyrrhotite is rare. Correnso ore has higher base metal content than other Waihi veins. Oxidation extends down the vein margins to over 250 m below surface however, the andesite host rocks can appear only weakly weathered at or near the surface.

Much of the Martha Vein System has been mined from underground historically between 1883 and 1952. However, significant mineralised veined material remains intact adjacent to the historical workings that was not recoverable historically.

Figure 7-2: Geological Map and Section Across the Waihi Area



Modified from 1:50 000 scale IGNS Waihi Geological map (https://www.gns.cri.nz/Home/Products/Maps/Geological-Maps) using OceanaGold drilling data, mapping data and internal reports (as at Feb 2021).

7.3 GOP

The GOP deposit is part of the greater Waihi epithermal vein system located approximately two (2) km to the east of the Martha Open Pit. It is situated along the southern strike extent of the Favona and Moonlight deposits. Veining at GOP occurs within the upper 250 m below the surface, hosted within the upper andesite unit (devoid of quartz phenocrysts). The mineralisation is characterised by shallow-level, hydrothermal breccias and associated banded quartz veins interpreted to represent the top of the epithermal system. The uppermost mineralised quartz veins flare up into hydrothermal explosion breccias. The GOP veins are predominantly steeply dipping veins developed within the hanging wall of the Favona Fault that dips moderately towards the SE. GOP veining trends ENE to NNE between 010 ° and 070 ° and dips steeply towards the SE.

7.4 WKP

Low-sulphidation epithermal quartz veins at WKP are hosted in Whitianga Group rhyolites, typically rhyolite flow domes to sub-volcanic intrusions within polymict lapilli tuffs. Deep drilling to the west indicates the rhyolites are underlain by Coromandel group andesites. The mineralised sequences are partially overlain by strongly magnetic, fresh andesite flows, rhyolitic tuffs and recent ash deposits observed in drilling and regional mapping (Figure 7-3 and Figure 7-4).

Gold mineralisation occurs in association with quartz veining developed along two types of structurally controlled vein arrays. The principal veins, namely the EG-, T-Stream and Western Veins occupy laterally continuous, NE trending (025-47 °), moderately dipping (60-65 °) fault structures reaching up to 10 m in width. More subsidiary, extensional veins (1-100 cm wide) are developed between or adjacent to the principle fault hosted veins. These veins often form significant arrays that are moderate to steeply dipping with a more northerly to NNE strike and appear to lack lateral and vertical continuity compared to the principle veins.

The rhyolites have undergone pervasive hydrothermal alteration, often with complete replacement of primary mineralogy by quartz and adularia with minor illite and/or smectite clay alteration. Figure 7-3 illustrates the dominant veins at the WKP deposit in plan view.

The EG Vein is the largest and most continuous mineralised structure drilled at WKP to date. The vein strikes approximately NE (020 °) for over ~1000 m although the extent of veining to the north and south remains open due to limited drill data. Veining dips steeply to the west and is still considered to be open up-dip. To date the highest up-dip intersection on the EG Vein was WKP64 where abundant clay gouge was encountered in the place of veining which still carried significant Au values. The EG Vein has been well drilled to more than 400 m below the surface (WKP41, WKP39, WKP37 and WKP35). Veining and grade are seen to decrease at depth (at approximately -180 m RL) (Figure 7-4). Veining observed in drill core is characterised by multiphase white quartz/chalcedony with textures including colloform banding, brecciation, vein sediments and quartz replacing platey calcite.

Within the footwall of the EG Vein are a series of veins referred to as the East Graben footwall veins. These veins show unique characteristics to other WKP veins in that they appear more as sulphide-rich (pyrite-marcasite) vein breccias with slightly elevated As, Hg and Sb. The brecciated nature of these veins indicate they may be more fault controlled than extensional.

There are a series of sheeted hanging wall veins along the EG structure containing significant Au grade in places. These veins appear to have a more northerly strike with sub-vertical dips. These veins outcrop at surface and were the focus of minor historical workings (pre-1950s) and early DD in the 1980s.

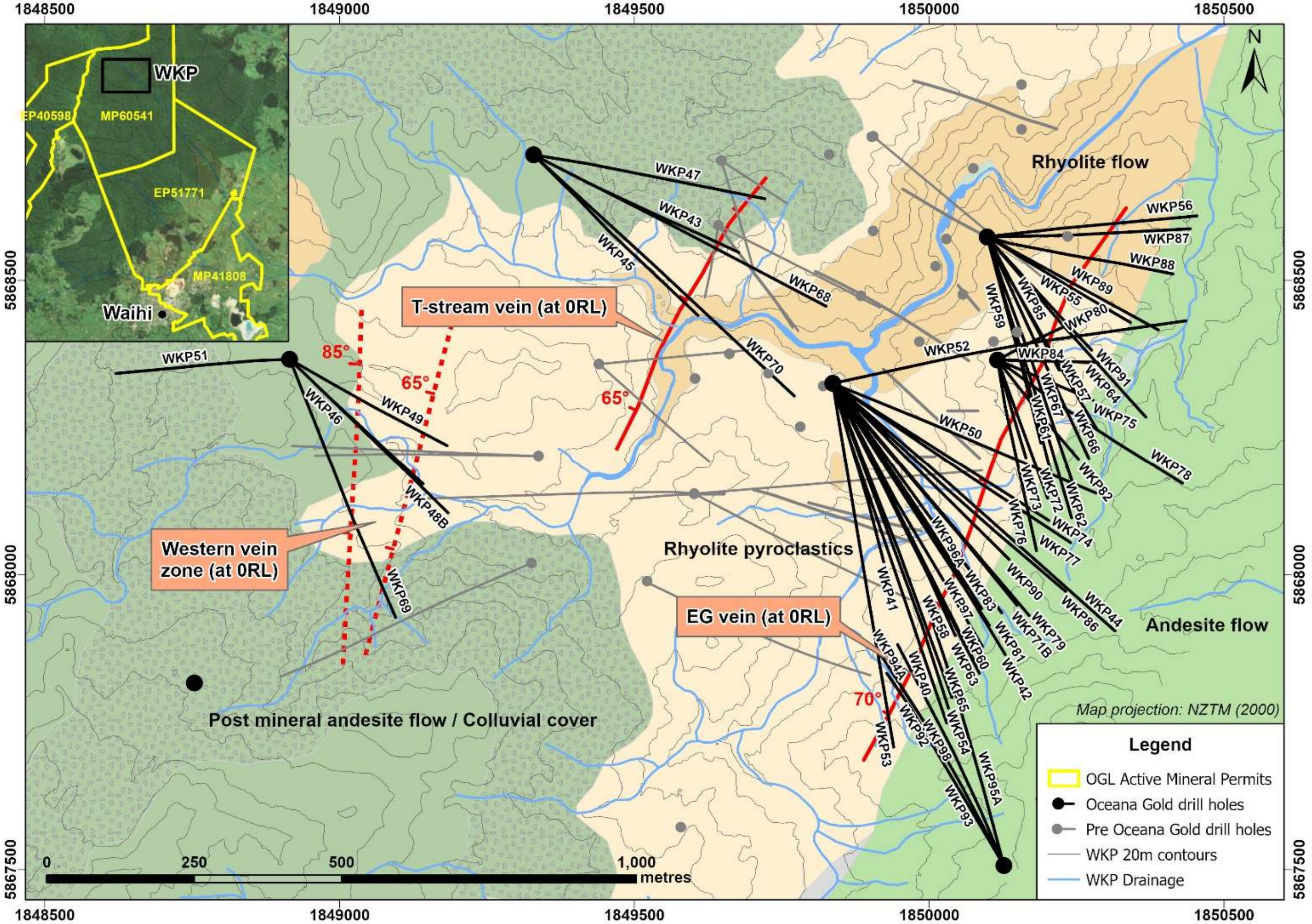
The T-Stream Vein is a breccia zone within rhyolite flows containing mineralised quartz veins located approximately 500 m to the west of the main EG Vein. This structure strikes approximately NE (020 °) and dips moderately (65 °) towards the west (Figure 7-3). The brecciated vein zone is exposed at the surface and appears oxidised and often broken at depth. Low-grade Au occurs over the entire width of the structure with narrow internal pockets of high Au grade veins. Drilling to date shows Au grade decreases below 0 m RL.

The Western Vein zone is located approximately one (1) km to the west of the EG Vein and is the least understood of the WKP veins. Drill data contains minimal orientation data due to poor ground conditions however, veining is believed to be N to NNE-trending and steeply dipping towards the west. The vein zones contain numerous individual veins not all of which carry anomalous Au. The dominant vein textures are quartz replacing platey calcite and minor chalcedonic quartz.

7.5 Comments on Geological Setting and Mineralisation

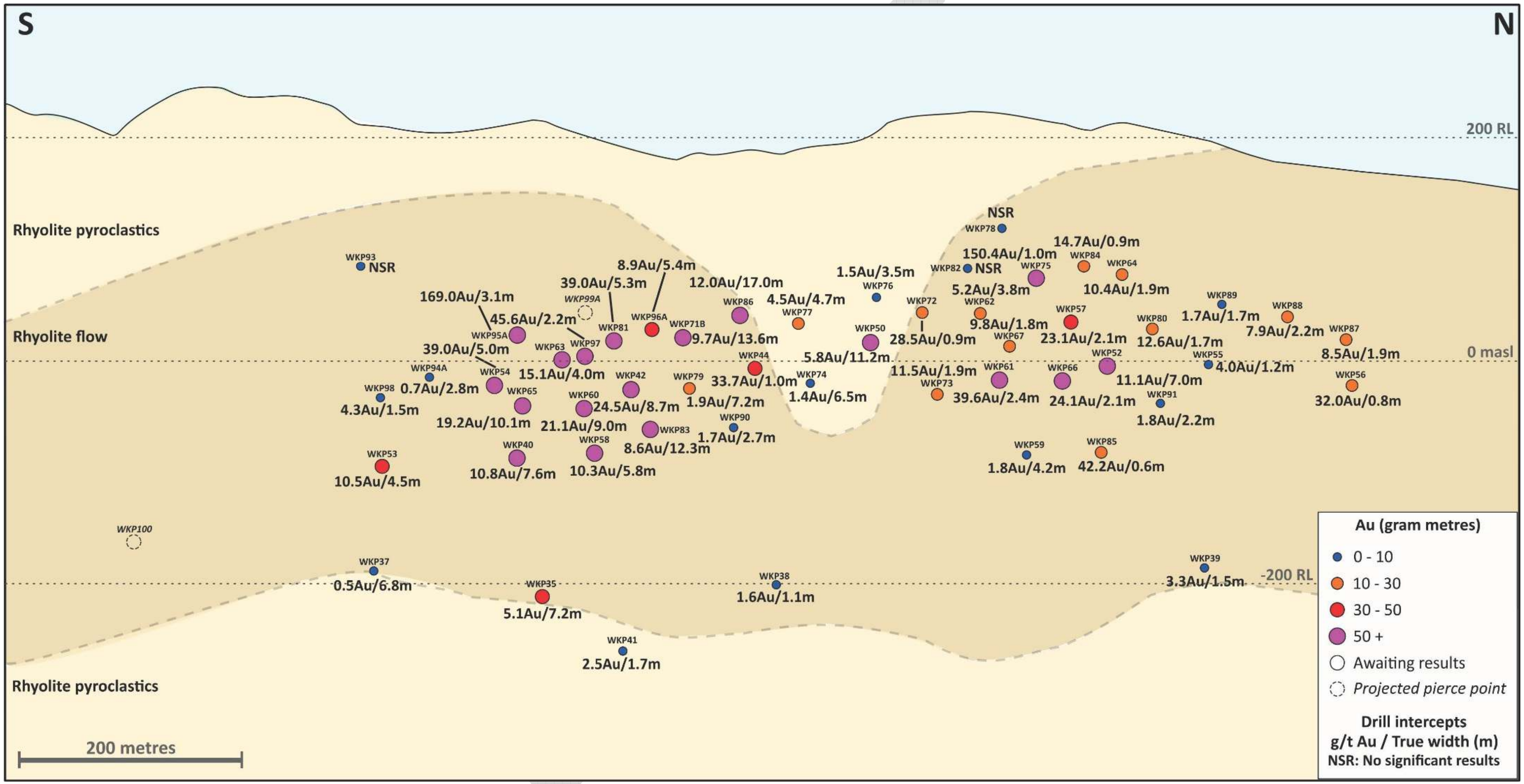
In the opinion of the QPs knowledge the geological control on mineralisation is well understood and is sufficient to support the estimation of Mineral Resources and Mineral Reserves. The current experience and geological knowledge of the area is also considered sufficiently acceptable to reliably inform mine planning.

Figure 7-3: Geological Map of the WKP Prospect



Modified from previous geological interpretation maps (https://data.nzpam.govt.nz/GOLD/system/mainframe.asp) using OceanaGold mapping data, and internal reports (as at Feb 2021). Permit layers were sourced from (http://data.nzpam.govt.nz/PermitWebMaps/?commodity=minerals).

Figure 7-4: Simplified Long Section along the EG Vein at WKP - February 2021



Sourced from OceanaGold drilling database and internal geological reports.

8 DEPOSIT TYPES

All the gold deposits outlined by OceanaGold to date in this report are considered to be typical of epithermal vein gold-silver deposits.

8.1 Comments on Deposit Types

In the opinion of the QPs, features observed in Waihi and WKP deposits display the following features that are typical of epithermal gold deposits elsewhere in the world:

- Gold-silver mineralisation is predominantly confined to localised bands within multiphase quartz veins;
- Host lithologies for veins are volcanic units of andesitic and/or rhyolitic composition;
- Sphalerite, galena and chalcopyrite commonly occur with gold-silver mineralisation within the Martha Underground deposit. This base metal content increases at depth with galena reaching up to +3% Pb and sphalerite in some localised areas exceeding 2.5% Zn;
- Host rock volcanics have undergone pervasive hydrothermal alteration, often with complete replacement of primary mineralogy. Characteristic alteration minerals include quartz, adularia, albite, carbonate, pyrite, illite, chlorite, interlayered illite-smectite and chlorite-smectite clays extending over tens of metres laterally from major veins;
- The upper portion of veining at the GOP deposit contains intact hydrothermal eruption breccias. The northern strike continuation of the GOP veins at Favona also contain hydrothermal breccias and an intact siliceous sinter sheet; and
- Mineralisation is structurally controlled.

9 EXPLORATION

Various companies have undergone exploration programs across the Waihi and WKP Project areas since gold was first discovered in the late 1900s. Both the Waihi and WKP Project areas have undergone geophysical and geochemical surveys, geological mapping, diamond drilling, geological interpretation and modelling. OceanaGold continues to drill in the project areas, with 25 km of diamond drilling planned for resource infill and reserve conversion for the Martha Underground Project for the rest of 2021. A further 6 km of diamond drilling is planned for the WKP Project in 2021.

Most exploration drilling was diamond core drilling done by triple tube wireline methods.

No exploration results are being presented in this report, rather this report is focused on advanced projects that have Mineral Resource estimates completed.

9.1 Grids and Surveys

All historic underground mine data in Waihi was recorded in terms of Mt Eden Old Cadastral grid (MEO). This is a local, Waihi specific grid this is still utilised for all underground and exploration activity within three (3) km of the Waihi Mine beyond which New Zealand Map Grid or New Zealand Transverse Mercator (NZTM) is utilised. The MEO grid is offset from NZTM by 5215389.166 (shift mN) and 1456198.997 (shift mE).

The MOP5 operation has historically used a local mine grid, referred to as 'Martha Mine Grid', derived from MEO grid but oriented perpendicular to the main veins. The grid origin is based at No.7 Shaft (1700 mE, 1600 mN) and rotated 23.98 ° west of MEO North. All open pit channel and open pit grade control drilling data has been converted to Mt Eden Old Cadastral for the resource estimation of the Martha Underground Resource.

Relative level (m RL) is calculated as sea level + 1000 m.

New Zealand Map Grid (NZMG) is used at WKP, which is in the NZGD1949 projection. False northing 6,023,150 m north; False easting 2,510,000 m east.

WKP topographic control is from high resolution aerial photography and LiDAR providing 0.5 m contour data.

WKP data is currently reported using NZMG. More detailed documentation on drilling is presented in Section 10 of this report.

9.2 Bulk Density

Bulk density determinations are discussed in Section 11.6.

9.3 Comments on Exploration

In the opinion of the QP:

- The exploration programs completed to date support the interpretations of the epithermal style of deposit and are appropriate to that style of mineralisation; and
- The projects retain exploration potential and additional work is ongoing.

10 DRILLING

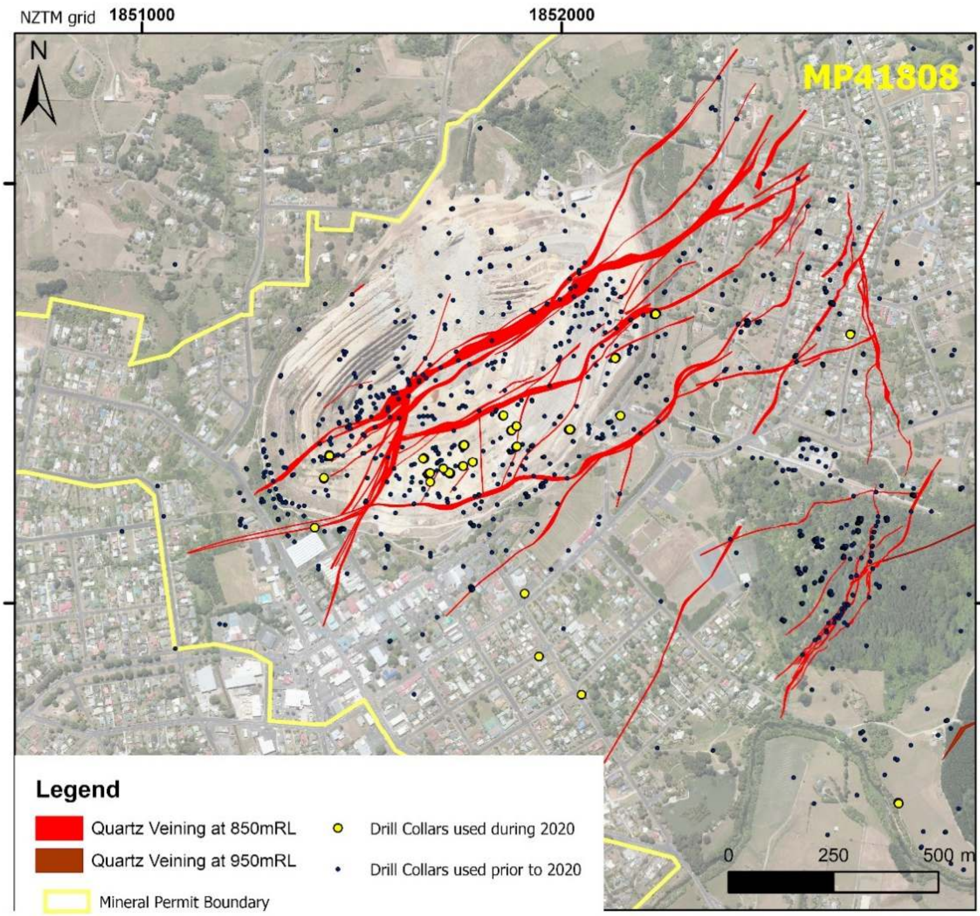
Approximately 268,685 m of diamond core has been drilled within the Martha and Gladstone projects since 1980 (as at February 2021). The WKP project has had 47,139 m of diamond drilling in 100 holes drilled since 1980 (as at February 2021).

Additionally, 86,074 m has been drilled in 4,445 Reverse Circulation Grade Control (RC) grade control holes during the Martha Open Pit Southern Stability Cut (SSC) and Eastern Layback (ELB) Projects between May 2007 and May 2015, using a 114 mm hole diameter and rig-mounted cyclone sampler. Details of the various drilling programs for Waihi are summarised in Table 10-1 by year. Locations of drill hole collars drilled in Waihi are shown in Figure 10-1.

10.1 Drill Methods

All surface drill holes were drilled by triple tube wireline diamond methods. Surface holes are collared using large-diameter PQ core, both as a means of improving core recovery and to provide greater opportunity to case off and reduce diameter when drilling through broken ground and historic stopes. Hole diameter is usually reduced to HQ at the base of the post mineral stratigraphy. All drill core was routinely oriented below the base of the post mineral stratigraphy, either by plasticine imprint or using the Ezimark or Reflex core orientation tool.

Figure 10-1: Drill Hole Collar Locations - Waihi



Sourced from OceanaGold drilling and GIS database

Drill hole location is recorded relative to the local mine grid Mt Eden Old Cadastral. Initial set-out and final survey of drill hole location for all recent drill holes (2004 onwards) have been carried out by mine surveyors using real time differential GPS. Downhole surveys were performed at 20 m intervals up to 60 m and then 30 m intervals to end of hole using a digital single shot camera. The accuracy of the downhole camera is checked monthly against a fixed camera stand. Magnetic field and gravity readings are taken for each camera shot and surveys are rejected if the readings are outside the normal ranges. Magnetic readings from downhole surveys are loaded to the database and converted to local grid north (Mt Eden Old Cadastral) or New Zealand Map Grid (for WKP holes) based on the current magnetic declination.

Core recovery is recorded for all drilled intervals and is typically greater than 95%. No grade versus recovery relationship is evident.

Table 10-1: Summary of Drilling by Year

Year	Waihi Near Mine				Grade Control		Other Districts	Drill hole series
	Surface exploration Diamond Drill meters	Surface exploration RC meters	Underground Diamond Drill meters	Total Drill meters	Underground Diamond Drill meters	Pit RC meters	Total Drill meters	
1980-1985	16,747			16,747				WHD, WE, WHD, WR, UW
1986	2631			2,631				WE, UW
1987	325			325				WE, WHD
1988	1095			1,095				FRC
1989	991			991				WE, WHD, WR
1990	2,273			2,273				UW
1991	3,567			3,567				WE, WHD, WR
1992	1,134			1,134				WE, WHD, WR
1993	975			975				WHD
1994	1,215			1,215				WHD
1995	90			90				WHD, WC
1996	3,768			3,768				WHD, WG, UW, WE
1997	3,052			3,052				WHD, UW, WG
1998	1,371			1,371				WHD, UW, WG
1999	3,064			3,064				UW, WHD
2000	1,442			1,442				UW, WHD
2001	12,084			12,084				UW, WHD
2002	18,893			18,893				UW, WHD
2003	12,427			12,427				UW, WHD
2004	20,434			20,434				UW
2005	23,389	330		23,719			4,118	UW, WHD, MRC
2006	15,464	1,149	*2,851*	19,464	1051		2,851	WHD, UW, MRC, FU, FD
2007	9,818	1,536	*6,129*	17,483	4353	16770	1,011	WHD, UW, MRC, MWRC, FU, FD
2008	7,572	1,972	6,892	16,436	5197	17192	2,250	WHD, UW, MRC, MWRC, MNDDH, FU, FD
2009	9,755	2,998	703	13,456	3963	13044	3,222	UW, FU, FD
2010	12,935		218	13,153	2828	7018	2,877	UW, FU, FD,
2011	15,516	4,709	3,464	23,689	13195	2479	3,243	UW, UG, MED, FU, FD, CGD
2012	8,186	1,170	4,947	14,303	15789	4186	5,016	UW, MED, CGD; TRIOUGDD
2013	0		5,290	5,290	9606	6750	1,400	TRIOUGDD and CORUGDD
2014	1,770		20,607	22,377	3200	14803		CORUGDD, DAYUGDD, GEMUGDD, UW
2015	2,681		20,035	22,716	14391	3831.8		CORUGDD, DAYUGDD, GEMUGDD, UW, WAIHIEXP
2016	14,227		21,693	35,920	6580			CORUGDD, DAYUGDD, GEMUGDD, UW, WAIHIEXP
2017	19,437		19,143	38,580	7994.7		3,750	CORUGDD, DAYUGDD, GEMUGDD, UW, WAIHIEXP, WAIHIRES, , WKP
2018	13,782		32,161	45,943	2659		12,002	CORUGDD, GEMUGDD, TRIOUGDD, UW, WAIHIEXP, WAIHIRES, WKP
2019	13,178		43,851	57,030	377.8		14,848	TRIOUGDD, WAIHIEXP, WAIHINORTH, WAIHIRES, WHITE BLUFF, WKP
2020	0		24,522	24,522	722.9		7,799	CORUGDD, GLAMORGAN, TWIN HILLS, WAIHIEXP, WAIHINORTH, WAIHIRES, WKP
TOTALS	275,288	13,864	212,506	501,658	91,907	86,074	64,387	

Sourced from OceanaGold drilling database

10.2 Geological Logging

Since October 2015, when OceanaGold took ownership of Waihi Gold all drill core has been logged into excel spreadsheets using validated templates.

Log intervals are based on geological boundaries or assigned a nominal length of one or two metres. The geological log incorporates geotechnical parameters, lithology, weathering, alteration and veining. A dropdown menu for each field allows the geologist to enter data by selecting from the available codes. Once logging is completed, the log is validated and then uploaded into an AcQuire database. A complete digital photographic record is maintained for all drill core.

RC grade control drilling in the open pit was sampled over 1.5 m intervals.

10.2.1 Lithology Codes

Lithology fields include three primary fields, composition, rock type and grain size. Secondary logging fields for lithological information (optional) include fields to record local or formal geological unit names, textural features, intensity of texture and composition of clasts.

With the increased drilling around historic workings, logging codes have been modified to sufficiently characterise material associated with the workings for example stope fill, open stope, collapsed stope and open holes such as drives.

10.2.2 Weathering Fields

Weathering is logged on a scale of one to five where five represents fresh rock and one represents intensely weathered material.

10.2.3 Alterations Fields

The primary fields for alteration use a 1-5 scale to record the intensity of hydrothermal alteration of the host rock. This includes fields for intensity of adularia, silicification, clay alteration, chlorite alteration, carbonate and hematite. A secondary field "alteration style" allows the style of alteration to be described, based on visual identification of the alteration mineralogy. The definitions for alteration style are based on mineral assemblages and associated temperature-pH charts from Corbett and Leach, 1998.

10.2.4 Structural Fields

Structural fields are used to record information about veins, secondary breccias (such as faults) and hydrothermal breccias. Fields used to record veining includes vein percentage, vein mineralogy, vein texture, vein style and sulphide content. Fields for secondary breccias includes breccia percentage, breccia type, matrix composition and clast composition.

Structural data is recorded in a separate excel spreadsheet with inbuilt formulas to convert measurements taken from the core to estimate the dip and dip direction of measured structures. The calculations take into account a 'top of core' reference line and the drilling direction and angle of the drillhole.

The orientation log is validated during logging and uploaded into the AcQuire database once logging is complete.

10.2.5 Geotechnical Logging

Geologists record standard geotechnical parameters, including rock quality designation, fractures per meter and hardness for all drill core. The geotechnical group then log selected mineralised and waste intervals in greater detail using geotechnical logging criteria.

10.3 Drill Core Recovery

Diamond drill core recovery is estimated by comparing the measured (recovered) core length against the drilled length (obtained from the drilling rig). Recovery data has been captured for all sample intervals for all diamond drill holes and there is no observed relationship between core recovery and grade. Core from the Martha Project is monitored for recovery daily to rationalise actual core loss against the intersection of historic mining voids with re-drilling actioned if necessary.

Core recovery within veined material (>40% vein in sample interval) varies between projects and is summarised as follows:

- 92.5% within MUG
- 92.5% within MOP5
- 89-90% for GOP
- 96.2% for WKP

RC drill sample recoveries were assessed using actual weights versus expected weights by the sampling technician and dispatching geologist. Samples were discarded where the recovered sample weight did not correlate well with the expected weights of the drilled interval.

Core recovery around historic workings have been increasingly difficult, so different methodologies have been trialled and adopted with relevance to the ground conditions that are encountered. Areas of core loss are broken out where possible so not to smear grade over disproportionate areas.

10.4 Collar Surveys

All historic (pre-1952) underground mine data in Waihi was recorded in terms of Mt Eden Old (MEO) grid. This grid has continued to be utilised for all underground and exploration activity within 3 km of the Waihi Mine beyond which NZMG. The MEO grid is offset from New Zealand Transverse Mercator (NZTM) Grid by 5215389.166 (shift mN) and 1456198.997 (shift mE).

The MOP5 operation has historically used a local mine grid, referred to as 'Martha Mine Grid', derived from MEO grid but oriented perpendicular to the main veins. The grid origin is based at No.7 Shaft (1700 mE, 1600 mN) and rotated 23.98 ° west of MEO North. All open pit channel and drilling data has been converted to MEO for the resource estimation of the MUG resource. Relative level (m RL) is calculated as sea level + 1000 m.

Drill collars at WKP are located using a total station in NZGD2000 Mt Eden. These coordinates are converted to New Zealand Map Grid using a LINZ coordinate convertor on the following webpage:https://www.geodesy.linz.govt.nz/concord/index.cgi?do_entry=Enter+more+coordinates&CI=&IC=H&ID=S&IH=-&IO=EN&IS=EDENTM2000&NEXT=&OC=H&OD=H&OH=NZVD2016&OO=EN&OP=2&OS=NZMG&PN=N&YEAR=now

All underground and surface drill collars used in resource and reserve estimates are surveyed using a total station by a registered professional land surveyor.

10.5 Downhole Surveys

At the start of the hole the driller lines up the mast (relative to true north azimuth) using a Gyrocompass Azimuth Aligner. Downhole surveys are taken on all drillholes at 20 m intervals up to 60 m and then 30 m intervals to the end of the hole using a digital single shot camera. Magnetic azimuth readings from downhole surveys are loaded to the drilling database. The drilling database automatically calculates local grid azimuths (MEO for Waihi and NZMG for WKP) based on the current magnetic declination. The magnetic declination has been validated for all holes used in resource estimations.

Where surveys are unable to be taken or inaccurate, they are replaced with an estimate. Where drillholes intercept old workings, surveys are estimated on either side of the workings to reduce the inaccuracy of the automatic curvature applied by Vulcan software irrespective of where the actual deviation occurred. All downhole surveys are validated by a geologist in the AcQuire database.

Downhole gyroscopic survey campaigns are used to validate the downhole survey data for exploration drillholes, most recently consisting of a suite of 16 holes surveyed across WKP, MUG & MOP5 Projects Q1 2019. The recorded hole dip is very similar for the two methods, and azimuth readings typically vary within a ~2 ° range.

10.6 Geotechnical Drilling

Geotechnical drilling has been carried out for GOP, MUG and WKP for the purposes of collecting samples for triaxial, uniaxial strength testing and other laboratory test work. All resource drilling has geotechnical components logged in detail which are analysed by a site-based geotechnical engineer.

Two geotechnical cover holes were drilled from the 920 level to primarily investigate ground conditions while mining the current drill drives for the Martha Project. Geotechnical data from these holes was also used to test the ground conditions associated with historic workings. Additional geotechnical cover holes included one investigating the conditions expected for the safe breakthrough into the MOP5 from the 920 level, and another covering a raise bore shaft between the 920 and 800 levels within the MUG development.

10.7 Current Drill Spacings

All resources under consideration in the integrated project have been adequately drilled to achieve an Indicated or Inferred Resource classification as summarised in Table 10-2. The MUG Project uses an average spacing to three drill holes of 60 m for Inferred and 40 m for Indicated Resource. The extensive mining history of Martha (>135 years+) has developed significant experience in assessing the continuity of mineralisation and mining the Martha Vein System and the adjacent deposits. The vein style mineralisation has a strong visual control, is well understood and has demonstrated continuity over significant ranges.

The WKP Project includes ~100 diamond drill holes (excluding re-drills and piezo holes) drilled up to February 2021. The bulk of recent drilling has targeted the EG Vein zone. The EG Vein zone has been intersected in drilling over a strike length of ~1 km. This structure is larger than those typically encountered in the Waihi Project area and on this basis the average drill hole spacing required for classification as an Inferred Resource has been increased to 70 m average distance to the three closest drill holes.

Table 10-2: Current Drill Spacing per Project

Project	Average distance to 3 closest holes	Average distance to 3 closest holes
	Indicated Resource	Inferred Resource
MOP5	<40 metres	40-80 metres
GOP	<40 metres	40-60 metres
MUG	<40 metres	40-60 metres
WKP	<45 metres	45-70 metres

10.8 Current Drill Orientations

Drill holes are designed to intersect known mineralised features in a nominally perpendicular orientation as much as practicable given the length of drillholes (often 250 m+) and availability of drill platforms.

10.9 Comments on Drilling

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation as follows:

- Geological logging of drill core (surface and underground) and RC chips meets industry standards for gold exploration within an epithermal vein gold setting;
- Collar and downhole surveys have been performed using industry standards;
- Recovery data from core drilling is acceptable;
- Geotechnical logging of drill core meets industry standards for underground and pit operations;
- Drill orientations are generally appropriate for the mineralisation style; and
- No material factors were identified with the data collection from the drill programs that could affect Mineral Resource estimation (refer to Section 14).

11 SAMPLE PREPERATION, ANALYSES, AND SECURITY

11.1 Sampling Methods and Preparation

11.1.1 Drill Core and RC Drilling

Once the core is logged, photographed and sample intervals allocated, it is cut in half length ways. If a vein is present, the cut line is preferentially aligned to intercept the downhole apex of the structure. Within each sample interval, one half of the core is bagged for sampling and the other is kept in storage. Whole core is sampled under the following conditions:

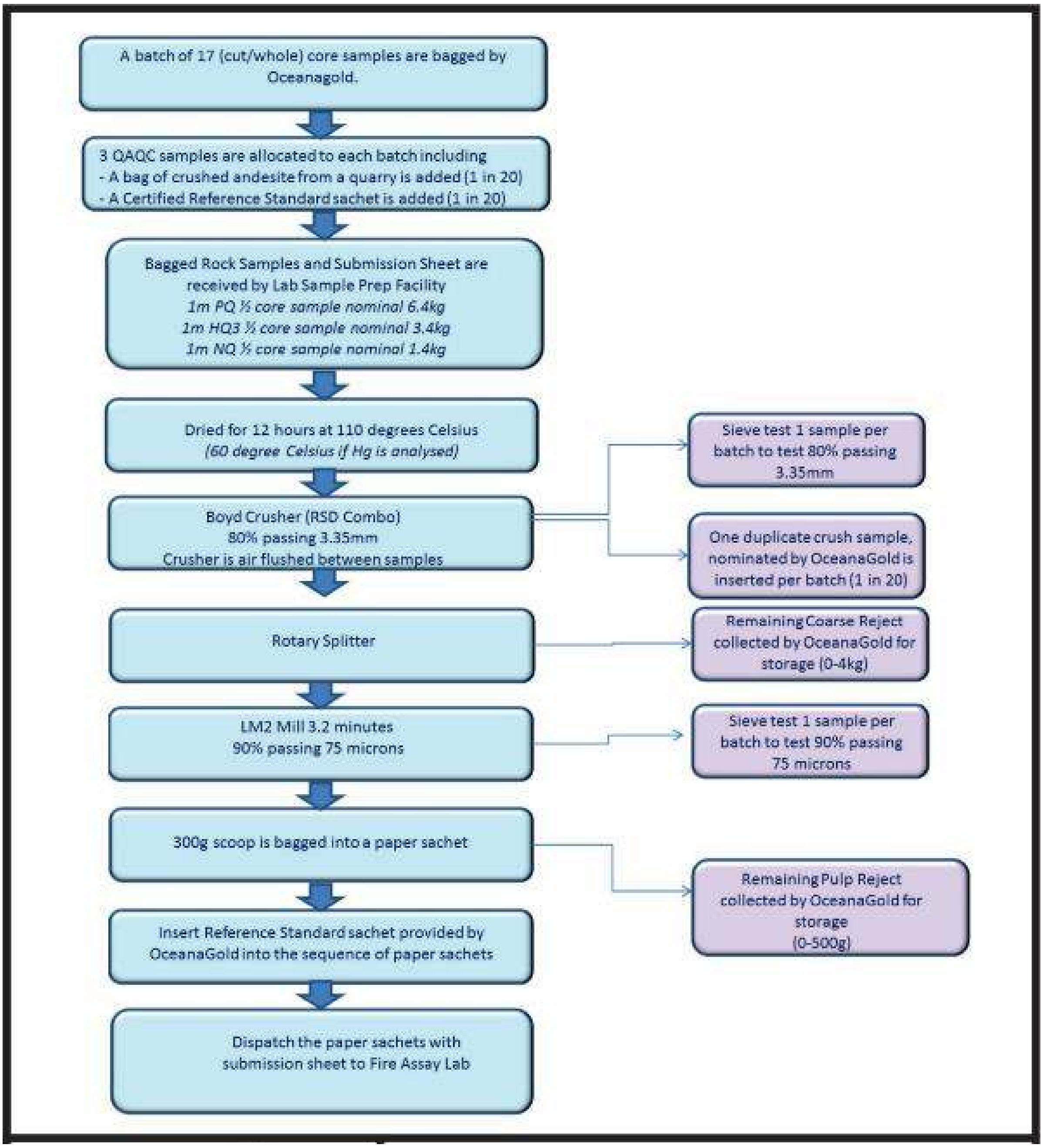
- Underground grade control drilling;
- Exploration drilling on occasion where there was significant core loss coupled with visible electrum; and
- Exploration drilling all BQ core is whole core sampled due to reduced sample volumes. BQ diameter core is only rarely drilled.

Labelled calico bags containing the cut core samples are routinely transported to the local Waihi SGS Laboratory for crushing and sample preparation. Refer to the sample preparation flow sheet illustrated in Figure 11-1.

Sample preparation has been carried out at the SGS Waihi laboratory since 2006. Prior to then the sample preparation facility was located at the Martha Mine site and operated by trained site employees. Some of the early WKP core (holes WKP40-45) was sent to the Westport SGS laboratory for crushing and sample preparation.

RC drill chips were sampled as part of the grade control process during the MOP5 operation but also on a minor scale for exploration purposes (approximately 4309 m used in MUG estimate). At the RC rig site, samples were collected in a bag attached to the cyclone at 1.5 m intervals from which a 3-5 kg sample was split using a cone splitter. Bags were then transported to the secure sample preparation facility. Sample preparation of RC chips is the same as drill core.

Figure 11-1: Sample Preparation Flow Sheet SGS, Waihi QA/QC



Source from OceanaGold internal reports

11.2 Quality Assurance and Quality Control

11.2.1 Exploration Drilling Samples

Analyses of drill sample pulps from exploration core was undertaken predominantly at the SGS Laboratory in Waihi but also at the ALS laboratory in Brisbane and Townsville. The quality of exploration assay results has been monitored by:

- Sieving of the jaw crush and pulp products at the laboratory;

- Monitoring of assay precision through routine generation of duplicate samples (one (1) every batch of 17 samples) from a second split of the jaw crush and calculation of the fundamental error; and
- Monitoring of accuracy of the results through insertion of CRM and blanks into each batch of 17 samples.

Blank, duplicate and CRM results are reviewed prior to uploading assay results in the AcQuire database and again on a weekly basis. The Waihi protocol requires CRMs to be reported to within two (2) standard deviations of the certified value. The criterion for preparation duplicates is that they have a relative difference (R-R1/mean RR1) of no greater than 10%. Blanks should not exceed more than four times the lower detection method of the assay method. Failure in any of these thresholds triggers an investigation and re-assay.

11.2.2 Underground Face Samples

Routine grade control underground face channel sampling protocols ensure a CRM standard, a blank, a crush and field duplicate were submitted within the sample sequence of every face. A blank sample was entered into the sample sequence preferably after what appears to be the highest-grade sample in the face. A field and crush duplicate of the sample preceding the blank, was entered at the end of the sample sequence, followed by the CRM standard.

11.2.3 RC Grade Control Data

Assay quality control procedures for grade control RC data are set out in the site MOP5 grade control procedures updated in 2015. These procedures were designed to detect any poor sampling and sample preparation practices and ensure that results are within acceptable ranges of accuracy and precision. The QAQC protocols implemented for RC grade control sampling in the Martha Pit are summarised in Table 11-1.

Table 11-1: Grade Control QAQC Samples for RC Sampling

Check	Description	Frequency
Blanks	Coarse Post-Mineral Andesite (Tirohia Quarry); submitted blind to the lab	1 per drillhole
Standards	Currently using Rocklabs standards; submitted as pulp to the lab	1 per drillhole
Field Duplicates	Additional RC sample taken from reject material from drill rig split	1 every fifth drillhole
Crush Duplicates	Split of crush residue repeat assayed by 50g Aqua Regia Assay	1 every 50 samples
Fire Assay	Repeat assay of pulp by 30g Fire Assay	30 per month

11.3 Laboratory Analyses

The standard suite of elements analysed at SGS in Waihi for all exploration drill and RC samples are gold and silver, although a significant proportion of core is also analysed for copper, arsenic, lead, zinc and antimony. Gold is assayed using a 30 g charge for fire assay with AAS finish. Between May 2007 and September 2014 pulps were assayed by SGS for gold and silver by 30 g Aqua Regia digest. From September 2014 fire assay analyses were conducted on gold only. Over range gold results of >100 g/t are re-assayed using an increase in dilution for the acid digest prior to instrument finish. Silver is analysed using a 0.3 g charge and AAS or ICP-MS instrument finish. For all other elements, the samples undergo a 0.3 g Aqua Regia digest followed by an ICP-MS instrument finish.

Generally, elements including mercury, arsenic, selenium and antimony increase at shallow levels within epithermal deposits. The presence of sinter and high-level quartz vein textures in the GOP area indicate that the resource is at the top of an epithermal system. As a result, multielement data with an extended suite of elements (Au, Bi, Hg, Sb, Se, Sn, Te, Th, Ti, U, W, Ag, Al, As, B, Ba, Be, Ca, Cd, Ce, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sc, Sc, Sr, Ti, V, Zn) has been undertaken at ALS Laboratories in Brisbane. Sample preparation was conducted at SGS Waihi following standardised procedures with a variation to sample drying temperature. A reduced temperature of 60 °C has been used to limit mercury volatilisation.

A selection of WKP and Waihi holes have undergone additional 42 element ICP-MS geochemical analyses at the ALS laboratory in Brisbane.

Comparison of the Ultratrace data with routine multielement data produced by SGS Laboratory in Waihi showed good correlation between the parent (SGS) and umpire (Ultratrace) data sets for silver, lead, zinc and arsenic, which gives confidence in the accuracy of SGS data for these elements. For samples with over range silver and lead, these elements are found to be extracted more efficiently by using a more dilute Aqua Regia digest (one (1) g sample weight rather than the standard 10 g per 50 ml). Antimony is not efficiently extracted by the current Aqua Digest method at SGS and consideration should be given to using the Peroxide Fusion extraction if more accurate antimony results are required.

11.4 Database

All QAQC data is managed in an AcQuire database. Blanks and CRM standards are reviewed on a weekly basis using AcQuire QAQC objects. Any patterns or concerns regarding sample preparation or assay quality are discussed directly with the laboratory.

11.5 Sample Security

All drill core is logged at a facility owned by OceanaGold and access onto site is strictly controlled. Core boxes consist of plastic which provides good protection to the core provided it is stored under cover. All core is stored in secure designated core sheds in Waihi. OceanaGold employees transport sample bags containing core samples to the Waihi SGS Laboratory on a daily basis. This laboratory is a secure facility.

11.6 Density Determinations

An updated assessment of density determinations was completed in May 2018. Weight measurements are routinely collected for representative core samples in air and in water during the logging process. The AcQuire database is set-up to automatically calculate the specific gravity (SG) from the measured weights using the formula:

$$\text{Specific Gravity (SG)} = \frac{\text{Weight in Air}}{(\text{Weight in Air- Weight in Water})}$$

11.6.1 MUG

Table 11-2: Specific Gravities used in the MUG Estimation

Domain	Sample Count	Mean SG	Standard Deviation
Quartz Andesite	1,361	2.52	0.15
Vein	634	2.53	0.09
Base Metal content logged (some overlap in Vein above)	426	2.56	0.08
Global Average	2153	2.5	0.16

The SG of the host rocks and vein structures in MUG are slightly variable. The andesitic host rocks average 2.52 g/cm^3 with the maximum recorded at 2.8 g/cm^3. Weathering and hydrothermal clay alteration generally decreases the SG while pyrite alteration increases the SG. The SG of 'vein' material in the MUG model is mostly influenced by weathering proximal to historical workings rather than surface weathering. Minerals present in large percentages within veins are the main factors contributing to variations in SG of vein material, particularly base metals, calcite and clay content.

'Stope fill' is assigned a SG of 1.8 within the resource estimate. Collapse zones associated with the 'Milking Cow' subsidence zone and the MOP5 wall failure have been assigned a SG of 1.9. Fill is captured in the model via the 'mined' variable summarised in Table 11-3.

Table 11-3: Mined Variable Values used Around Historical Workings

Mean Variable Value	Material Type	Modifying Factor
0	In-situ	As estimated
1	Backfilled Stopes	Density and grade modified
2	Subsidence	Density and grade modified
5	Open Stope	Density set to zero, grade removed
6	Open Development	Density set to zero, grade removed

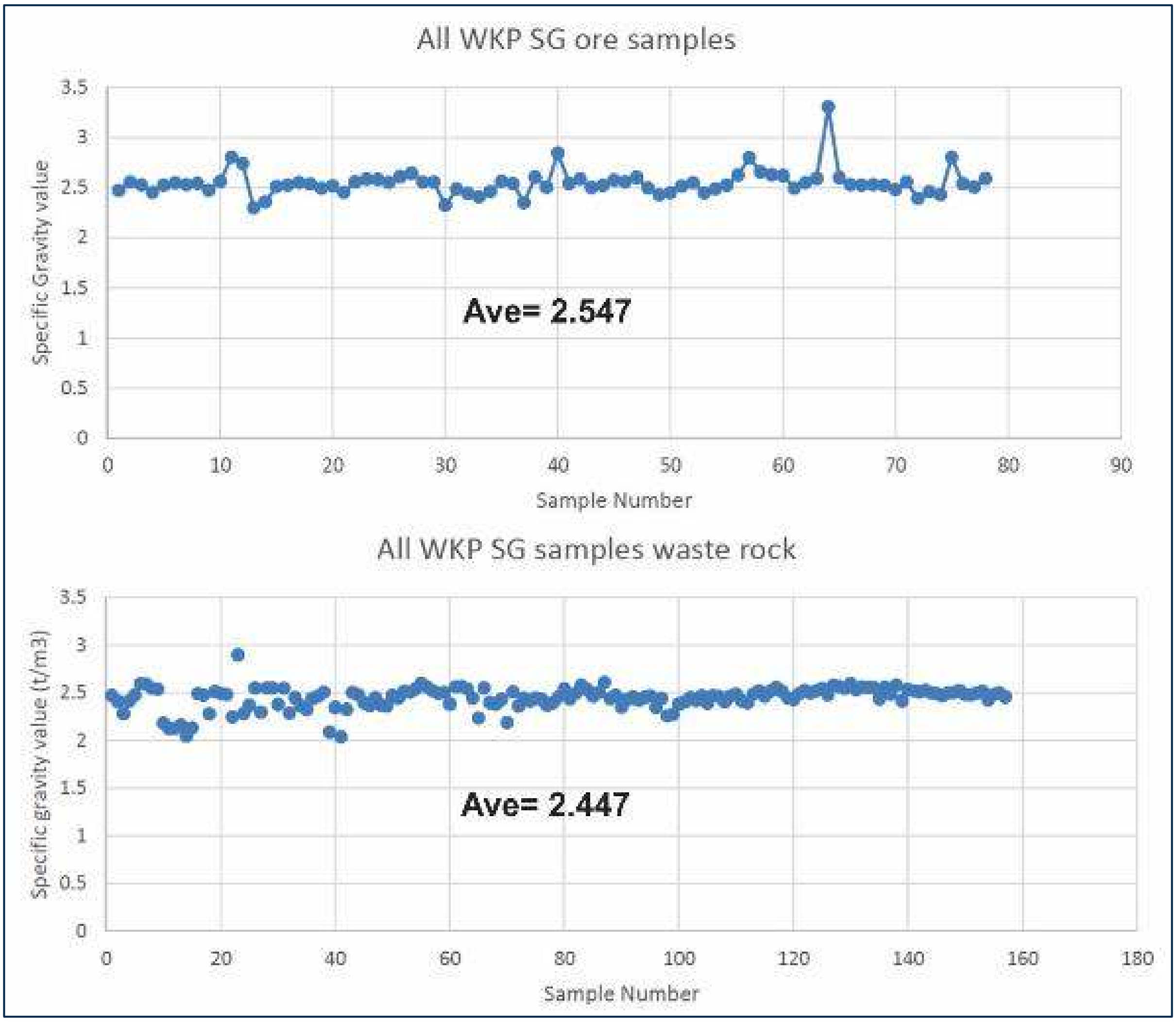
11.6.2 WKP

SG data is routinely collected during the logging of the WKP drill core. SG measurements taken from diamond core drilled at WKP is summarised in Table 11-4 and Figure 11-2. In general, the rhyolitic host rock is the least dense with an average SG of 2.45 (waste rock) compared to the quartz vein material with an average SG of 2.54.

Table 11-4: Specific Gravities used in the WKP Estimation

Domain	Sample Count	Mean SG
Waste Rock	156	2.45
Vein	79	2.54
Global Average	235	2.50

Figure 11-2: Calculated SG Values for Mineralisation and Waste Rock Samples at WKP



11.6.3 GOP

The GOP resource estimate assigns a SG to material by geologic unit and oxidation level. The SG categories assigned to the model are shown in Table 11-5.

Table 11-5: Specific Gravities used in the GOP Estimation

Zone	Area	Oxide Density	Primary Density
1	Black Hill Dacite	2.2	2.2
2	Rhyolite Tuff	2.1	2.3
3	Andesite	2.0	2.2
4	Volcaniclastics	2.0	2.0
5	Hydrothermal Breccias	2.2	2.2
9	Quartz Veins	2.3	2.5
Mined 1	Mined Development	0	0
Mined 2	Avoca Stopes	1.8	1.8

11.6.4 MOP5

The MOP5 model is the only model that currently has variable densities assigned to lithological units where surface hardness and weathering influence the model. Table 11-6 lists the density values assigned to the current estimation.

Table 11-6: Specific Gravities used in the Current Models

Project Martha	Model Density
Dig	1.60
Rip	2.00
Blast	2.30
Very Hard	2.47
Stope Fill	1.80

11.7 Comments on Sample Preparation, Analyses and Security

In the opinion of the QPs, sample collection, preparation, analysis and security for all OceanaGold drill programs are in line with industry-standard methods for gold deposits and provide data that are sufficiently bias and error free to support Mineral Resource estimation.

12 DATA VERIFICATION

Drill hole data is entered into an AcQuire database interface which includes protocols for validation. All drill collars, traces and surveys are checked for accuracy in 3D using Vulcan while holes are being drilled. Once the hole has completed drilling the collar position is picked up by a qualified surveyor and updated in AcQuire.

All geological logging is checked and validated by a second geologist on the core bench prior to core cutting. Logging data is entered into AcQuire by a geologist and then checked for completeness and errors once it is loaded into AcQuire. Laboratory results are uploaded into an AcQuire database using the files emailed directly from the laboratory.

Assay results are only successfully uploaded to AcQuire if they pass the QAQC verification. If any of the QAQC fail the verification, then further investigation is required by the geologist before the results can be uploaded to the database. Each drillhole has a checklist that needs to be completed before the hole can be classified as 'closed out' in the database.

Geology personnel are well trained and regularly monitored for consistency. Below level detection limit assay results are stored in the database as (negative) half the detection limit. No other modification of the assay results is undertaken. Monthly QAQC reporting and review is undertaken on all laboratory assay results. CRMs performance is regularly scrutinised, and the database QAQC function thresholds are reviewed bi-annually. CRMs are currently assigned to batches on a rotational roster in a 'pigeon pair' system.

A limited number of twinned holes were completed during the initial investigations for Correnso. These indicated that there is some short-range variability in gold mineralisation. No twinned holes have been drilled for the other projects. Geologists can recognise strong visual indicators for high-grade mineralisation observed both in drill core and in underground development.

All intercepts are reviewed during the construction of the geological wireframes prior to grade estimation, this review involves visual comparison of core photography, assay and logging data and spatial relationships to adjacent data. Significant intercepts are reported internally on a weekly basis for peer review purposes.

Check assay programs have been undertaken for projects previously as a part of the project advancing past milestones such as feasibility level studies.

12.1 Internal Reviews

A number of internal reviews have taken place to verify data collected for Mineral Resource purposes. A list of some of these reviews are provided below:

12.1.1 Geology & Wireframing

Rhys, DA. 2009 Observations and exploration recommendations at Newmont exploration properties Hauraki Goldfield. Unpublished Memo to Newmont Waihi Gold.

Rhys, DA. 2010 WKP prospect: review of exploration results with recommendations. Unpublished Memo to Newmont Waihi Gold.

Rhys, DA. 2011 Observations of selected drill core from the WKP prospect (with WKP-30 information added). Unpublished Memo to Newmont Waihi Gold.

Rhys, DA. 2011. Review of the Structural Setting of the Correnso Vein System, Waihi, New Zealand. Unpublished Report to Newmont Waihi Gold.

Rhys, DA. 2017. Waihi District geology: continuing contributions to understanding structural setting and zonation as applied to exploration and mining. Unpublished Memo to OceanaGold.

Rhys, DA. 2020. Review of the structural controls of the WKP prospect. Unpublished Memo to OceanaGold.

Rhys, DA. 2020. Review of the core logging template for the Martha Underground Project. Unpublished correspondence to OceanaGold.

Richards, SD. 2019. Review of the WKP vein model. Unpublished internal validation.

12.1.2 Density

White, T. 2012 Correnso Dry Bulk Density Study. Unpublished Internal Report, Newmont Waihi Gold.

Mcarthur, F. 2019 WKP SG Data Memo. Unpublished Internal Report. OceanaGold.

Vigour-Brown, W. 2019 Martha Underground SG Memo, Unpublished Internal Report. OceanaGold.

12.1.3 Assay QAQC and Multielement Geochemistry

Inglis R. 2013. Heterogeneity Study. Unpublished Internal Report, Newmont Waihi Gold.

Barker, S., Hood, S., Hughes, R., Richards, S. 2019. The Lithogeochemical signatures of hydrothermal alteration in the Waihi epithermal District, New Zealand. New Zealand Journal of Geology and Geophysics, Vol 62, Issue 4.

Biggalow, J. 2015. Review of multielement geochemistry of Waihi drill data. Unpublished Internal Review. Newmont.

12.1.4 Static and Kinetic Testwork

Kirk, A. 2012. Geochemistry of Ore, Tailings and Waste Rock Assessment by URS New Zealand for the Correnso Underground Mine (Newmont Waihi Gold).

12.1.5 Mineralogy

Mauk J. 2009. Petrographic Examination of Samples from the Reptile North and Number Nine Veins, Waihi. Unpublished Report to Newmont Waihi Gold.

Mauk, J.L., Hall, C.M., Barra, F., and Chesley, J.T., 2011, Punctuated evolution of a large epithermal province: The Hauraki goldfield, New Zealand. Economic Geology, v. 106, p. 921–943.

Ross, KV. and Rhys, DA. 2011. Petrographic Study of Representative Samples from the Correnso Vein System, Waihi District, New Zealand. Unpublished Report to Newmont Waihi Gold.

Menzies A. 2013 QEMSCAN Analysis of Samples from the Waihi District, New Zealand: Correnso. Unpublished report. Universidad Catolica del Norte, Antofagasta, Chile.

Coote, A. 2011 Petrological Studies of Diamond Core from WKP029 and WKP030, of the WKP South Project, Coromandel, New Zealand. Unpublished Report to Newmont Waihi Gold

Coote, A. 2012 Petrological Studies of Diamond Core from WKP024 and WKP031, of the WKP Epithermal Deposit, Coromandel, New Zealand. Unpublished Report to Newmont Waihi Gold.

Simpson, M. 2012 SWIR report for drill holes WKP-24, WKP-27 and WKP-30, Wharekirauponga, Southern Hauraki Goldfield. Unpublished Report to Newmont Waihi Gold.

12.1.6 Hydrology

GWS Limited 2012. Proposed Underground Mining Extensions ‑ Waihi. Assessment of Groundwater Inflows and Throughflows. Prepared for Newmont Waihi Gold.

12.2 Comments on Data Verification

The QP has reviewed the appropriate reports and is of the opinion that the data verification programs undertaken on the data collected from the project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation.

Database audits confirm the data are acceptable for use in estimation with no significant database errors identified. No bias corrections were considered warranted on drill and analytical data.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The testwork programs completed in 2018, 2019 and 2020 are listed in Table 13-1, Table 13-2 and Table 13-3.

Table 13-1: Testwork Program 2018

Project	Testwork Program
GOP	metallurgical composites variability Multiple Liberation Analysis (MLA) Comminution and water treatment plant (WTP) testing
MUG	Metallurgical composites
Flotation & Ultra-Fine Grind (FUFG)	Process Engineering Pre-feasibility Study

Table 13-2: Testwork Program 2019

Project	Testwork Program
MUG	Metallurgical variability Comminution and WTP testing
FUFG	Variability Locked cycle Diagnostic leach testwork Signature plot testwork
WKP	Metallurgical variability Comminution and WTP testing

Table 13-3: Testwork Program 2020

Project	Testwork Program
MUG	Metallurgical variability Comminution and WTP testing Feasibility Study
FUFG	Variability testwork for Martha and WKP
WKP	Metallurgical variability Comminution and WTP testing Pre-feasibility Study

13.1 Introduction

The Waihi mill has treated ore sourced from the Martha open pit as well as several underground ore bodies over the last 30 years. Considerable operating experience and data has been accumulated on the vein structures running through the Martha pit that has contributed the majority of the ore over this time.

With the additional targets that have been discovered the approach has been to develop a series of Bingo charts in conjunction with the geology team to identify the main gold bearing domains and to indicate the minimum number of composites to be targeted for metallurgical testing to generate recovery and throughput estimations.

Competency testing on selected composites comprises measuring Bond Rod and Ball Mill work indices, abrasion indices and SMC ® testing on core samples for competency estimates.

13.2 MUG

The testwork programs completed in 2018, 2019 and 2020 are listed in Table 13-1, Table 13-2 and Table 13-3.

Table 13-4: Testwork Program 2018

Project	Testwork Program
MUG	Metallurgical composites
Flotation & Ultra-Fine Grind (FUFG)	Process Engineering Pre-feasibility Study

Table 13-5: Testwork Program 2019

Project	Testwork Program
MUG	Metallurgical variability Comminution and Water Treatment Plant (WTP) testing
Flotation & Ultra-Fine Grind	Variability Locked cycle Diagnostic leach testwork Signature plot testwork

Table 13-6: Testwork Program 2020

Project	Testwork Program
MUG	Metallurgical variability Comminution testing

13.2.1 Flotation and Leaching

Prior to 2018, metallurgical test work was completed on 30 composite samples of intercepts from the various vein structures in the MUG resource. Twenty-three samples were submitted to the Newmont Inverness testing facility. Six samples representing the Edward vein were submitted to Ammtec Laboratory in Perth. Samples were mostly submitted both as quarter core and as jaw crush reject material (95% <7 mm), if both were available.

In 2019, 18 composites from intercepts were submitted to AMML Laboratories in Australia for testing direct leach performance and 6 composites samples were sent to JKTech for comminution testing.

In 2020, 25 composites samples from intercepts were sent to the OceanaGold Macraes Metallurgical Laboratory for testing direct leach performance, and 22 composites samples were sent to JKTech for comminution testing.

Separately, flotation testing was done on 27 samples (Phase 1 - 9 samples, Phase 2 - 18 samples) at a grind size of 75 µm. Results from this testwork indicated that there is little to no recovery benefit at 1% sulphur grade. At an overall grade of 1.11% sulphur recovery benefits is less than 1%. Sensitivity analysis on the incremental financial model shows a 3.4% improvement benefit is needed to generate an economic return. At this recovery benefit the net present value (NPV) is neutral. To generate an overall benefit of 3.4% recovery the sulphur model indicates that the overall grade should be 1.7% total sulphur. For MUG there is insufficient material from underground to make FUFG economically beneficial as the overall average sulphur grade is only 1.1%. Further testwork is planned in 2021 to confirm preliminary results and expand sulphur model for the mineralised domains from MUG.

A summary of the vein location and leach testing campaign is shown in Table 13-7.

Table 13-7: Summary of MUG Composite Samples Tested

Vein Structure	Historical (2011)	2019	2020	Total
Edward	18	3	9	30
Empire East	2	4	10	16
Martha	9	7	-	16
Grace	1	-	-	1
Royal	-	4	5	9
Rex	-	4*	1	5
Total	**30**	**22**	**25**	**77**

* 4 Rex samples tested at 75 microns

A review of the composite sample locations relative to the defined resource and preliminary stope design identified 50 of the total 77 composites lay in or within 20 m of expected mined areas. Bottle roll CIL tests results for the historical samples, 2019 and 2020 samples are summarised in Table 13-8.

Table 13-9 summarises gold recovery data at the grind sizes tested for the 2019 samples. The metallurgical samples tested in 2019 were processed at AMML Laboratories in Australia. The Rex samples were tested at 75 microns as part of the Variability Leach and Flotation and Ultra fined grind (FUFG) testwork program.

A total of 13 historical samples and 16 samples selected in 2019, and 21 samples selected in 2021 have been located based on the stopes (within a 20 m halo) to be mined over the LOM for the MUG deposit. Bottle roll tests were completed at three different grinds (38 µm, 53µm and 75µm).

Table 13-8: Gold Extraction Results for Historical Composites

Domain	Hole ID	Calculated gold Grade, Au g/t	As, ppm	Au Extraction (%)		
				38 µm	53 µm	75 µm
Edward	UW388-1000	3.75	42	96.50	95.57	94.98
Edward	UW388-1000 (Dup)	4.14	50	96.52	96.20	95.17
Edward	UW388-1001	4.64	34	97.57	96.87	96.06
Edward	UW388-1001 (Dup)	4.78	33	97.34	97.02	95.83
Edward	UW395-1000	20.47	112	97.73	97.46	96.26
Edward	UW395 1000/1001	14.83	95	97.05	96.49	95.37
Edward	UW395-1001	10.39	67	96.58	95.78	93.88
Edward	UW407-1000	5.54	30	97.20	95.60	94.70
Edward	UW407-1001	3.34	20	98.40	99.00	93.10
Edward	UW409-1000	12.30	60	98.00	95.50	93.40
Edward	UW409-1001	7.72	60	97.90	94.80	93.50
Edward	UW411-1001	4.95	40	97.90	97.70	97.80
Edward	UW412-1000	1.76	30	89.06	88.08	84.83

Table 13-9 summarises gold recovery data at the grind sizes tested for the 2019 samples. The metallurgical samples tested in 2019 were processed at AMML Laboratories in Australia. The Rex samples were tested at 75 microns as part of the Variability Leach and Flotation and Ultra fined grind (FUFG) testwork program.

Table 13-9: Gold Extraction Results for 2019 Composites

Domain	Hole ID	Assay Head Grade, Au g/t	As, ppm	Au Extraction (%)		
				38 µm	53 µm	75 µm
Edward	920SP9MR1318	6.54	11.9	98.9	98.5	-
Edward	920SP9MR1264 920SP9MR1320	5.89	36.1	96.1	95.8	-
Empire	920SP7MN1303	6.82	189	87.4	89.0	-
Empire	920SP7MN1290	4.66	111.5	92.5	91.1	-
Empire	800SP1MR1224	5.75	48.5	97.4	96.0	-
Empire	800SP1MN1095	5.73	185	86.4	84.0	-
Martha	800SP3MR1227 800SP3MN1188 800SP3MR1300	5.00	301	86.2	79.4	-
Martha	800SP1MN1100 800SP1MN1109 800SP1MN1118 800SP1MN1100 800SP1MR1224	6.13	45.6	94.0	95.4	-

Martha	800SP1MN1127 800SP1MR1214	5.09	139.5	83.9	80.5	-
Martha	800SP1MR1317 800SP1MR1280 800SP2MN1191 800SP1MR1317	5.76	259	82.60	78.9	-
Royal	920SP9MN1281 920SP9MN1297 920SP9MN1301 920SP9MN1276	5.45	178	91.9	89.5	-
Royal	800DC1RN1246 800DC1RN1240 800DC1RN1255	4.35	79.2	91.3	88.5	
Rex	UW715 UW725 UW721 920RCCRN1256 920RCCRN1259 920RCCRN1266 UW671	3.18	10.5			93.5
Rex	UW718 UW712 UW706	3.13	14.9			92.7
Rex	UW719 UW717 UW679	4.79	15.9			93.6
Rex	UW667 UW708 UW711	4.84	46.3			91.6

Table 13-10 summarises gold recovery data at the grind sizes tested for the 2020 samples. The metallurgical samples tested in 2020 were processed in-house at Macraes Metallurgical Laboratories in New Zealand.

Table 13-10 Gold Extraction Results for 2020 Composites

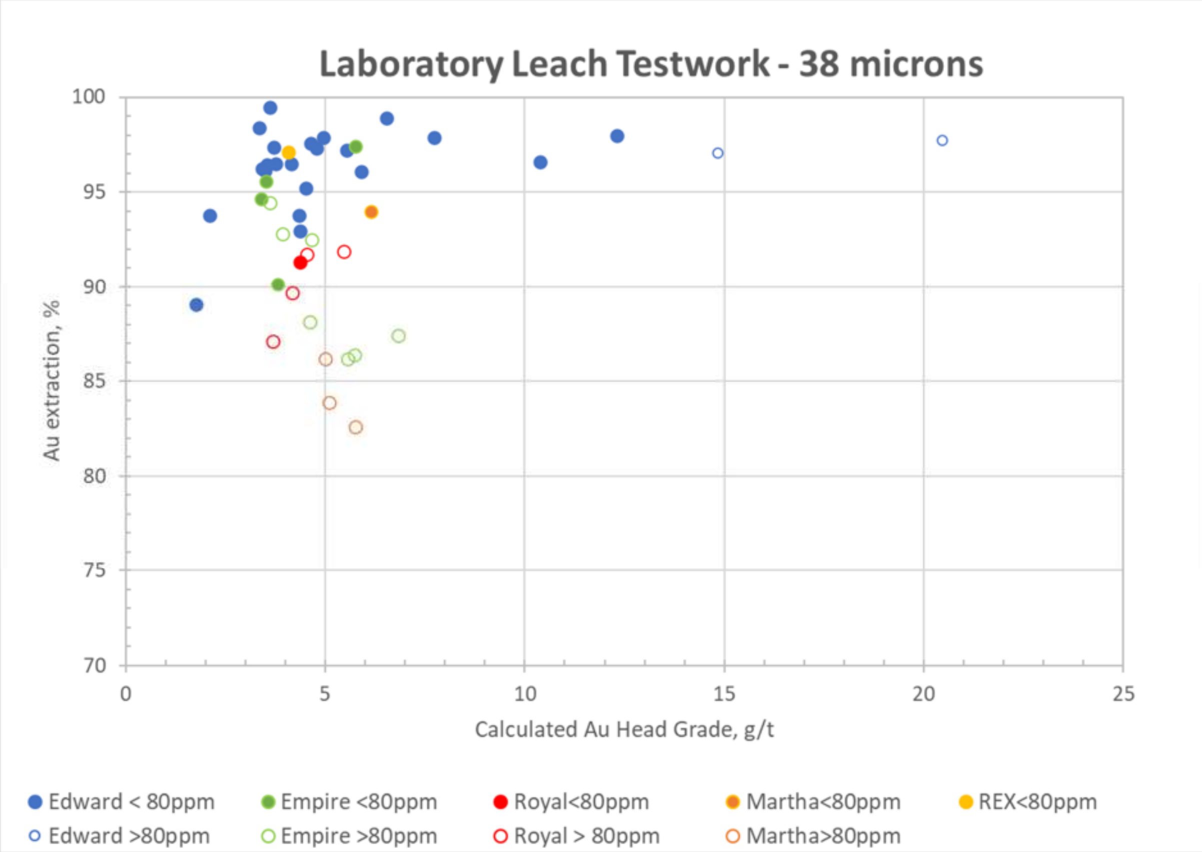
Domain	Hole ID	Assay Head grade, Au g/t	As, ppm	Au Extraction (%)		
				38 µm	53 µm	75 µm
Edward	800DC8MR1476	2.09	62	93.8	92.7	91.7
Edward	920SP8MR1363	3.48	75	96.1	95.8	95.4
Edward	800PC2MR1453	3.41	33	96.3	95.9	94.2
Edward	800DC8MR1436	4.34	35	93.8	93.8	93.7
Edward	920SP9MR1366	4.35	70	93.0	92.2	91.0
Empire	800DC7MN1337	3.50	39	95.6	93.3	90.3
Empire	800DC5MN1345	3.80	79	90.1	89.4	86.8

Domain	Hole ID	Assay Head grade, Au g/t	As, ppm	Au Extraction (%)		
				38 µm	53 µm	75 µm
Empire	920SP6MN1432	3.38	51	94.6	92.2	88.7
Empire	920SP4MR1413	3.79	76	91.0	90.9	92.3
Empire	920SP2MR1326	5.55	176	86.2	86.2	83.3
Empire	800DC2MR1477	3.93	95	92.8	92.8	90.2
Empire	920SP7MN1396	2.74	76	91.4	91.7	87.1
Empire	920SP6MN1446	4.61	201	88.2	87.2	84.2
Royal	800DC3RN1375	3.68	91	87.1	88.3	86.5
Rex	UW722	4.06	8	97.1	96.2	95.1
Edward	800RC3MR1442	3.6	19	99.4	99.3	99.1
Edward	800DC8MR1503	3.71	22	97.4	96.9	95.6
Edward HW	920SP8MR1315	4.51	31	95.2	95.6	93.3
Edward HW	920SP9MR1358	3.53	16	96.5	95.8	94.0
Empire	800DC4MN1540	6.19	140	93.9	92.2	89.8
Royal	14EMPRN1576	4.52	223	91.7	85.3	79.7
Royal	800RC3RN1438	4.18	125	89.7	88.2	83.2

Gold extraction results for historical, 2019 and 2020 samples at different grind sizes indicate that a 38 µm grind size provides the best gold extraction in the laboratory. In average for all metallurgical samples, gold recovery improvement between 38 µm and 53 µm is 0.70% for Edward, 0.90% for Empire, 3.10% for Martha, 2.4% for Royal and 0.90% for Rex. Plant operating experience has shown that an equivalent laboratory gold recovery at a P_{80} of 38 µm is equivalent to a grind size P_{80} of 53 µm in the plant. This relationship is due to the laboratory grind testwork being in open circuit, whereas in the plant the grinding circuit is in closed-circuit. This results in the higher density sulphides being preferentially ground finer and hence liberating more gold particles that are disseminated within the sulphides.

Figure 13-1 shows gold extraction (recovery) for the historical, 2019 and 2020 samples tested at a grind size of 38 µm against calculated gold feed grades indicate a range of recoveries from 89% to 99% for the Edward samples, 83% to 94% for Martha samples, 86% to 97% for Empire, 87% to 92% for Royal, and 92% to 94% for Rex samples. Arsenic grades in the composites ranged from 11ppm to 301ppm whilst the reserve grades in the mine schedule are generally in the 30-50ppm range. The chart highlights the composites tested above and below an 80ppm grade.

Figure 13-1: Gold Extraction as a Function of Feed Grade



13.2.2 Comminution

Ore characterisation on historical samples from the Martha underground in 2011 indicated that the MUG ore is considered medium competency for SAG milling with Axb of 41.9, and a high Bond Ball Work Index (BBWI) of 19.2 kWh/t. Table 13-11 shows historical results from testwork conducted in 2011.

Table 13-11: Historical comminution results on ore from Martha Underground -2011

Sample	Sample ID	DWI	Axb	SG	BBWI (kWh/t)
Edward	UW240-1003	6.29	40.8	2.55	17.3
Edward	UW367-1000	5.35	48.4	2.60	17.0
Edward	UW407-1000	5.83	42.8	2.48	18.7
Edward	UW407-1001	5.31	47.1	2.49	19.5
Edward	UW409-1000	5.87	42.9	2.50	17.7
Edward	UW409-1001	5.66	44.3	2.50	19.2
Edward	UW411-1001	4.81	53.7	2.57	19.7
Edward	UW412-1000	7.36	33.9	2.46	18.8
Empire East	UW240	4.39	60.7	2.67	14.8
Martha UG	WHD188	6.03	41.9	2.54	17.7
Grace	UW210	4.81	55.8	2.67	16.4
75th percentile		6.03	41.9	2.60	19.2

Six ore samples from Martha Underground were submitted in 2019 for comminution testing to the JKTech testing facilities in Brisbane, Australia. Comminution testing consisted of SMC drop weight, Bond Rod Mill and Bond Ball Mill work indices, and bond abrasion index. The selected samples represent mineralisation to be mined from four vein structures at Martha Underground. Samples were submitted as quarter core (1/2 HQ). The comminution test results are summarised in Table 13-12. The characterisation conducted on the Waihi mill feed sources has indicated that MUG mineralisation is very competent for SAG milling (75th percentile Axb 33.2) and hard to grind in ball mill (BBWI 21.0 - 25.2 kWh/t).

Table 13-12: Summary of Comminution Testing of 2019 MUG Mineralisation Samples

Sample	DWI	Axb	SG	BBWI	BRWI	BAI
Edward	7.67	34.1	2.58	22.5	15.4	0.3966
Empire	8.55	30.7	2.61	21.0	16.1	0.3458
Martha 1	6.70	37.1	2.51	23.5		
Martha 2	6.75	38.2	2.59	25.2	14.0	0.3537
Royal 1	6.60	36.9	2.43	22.2		
Royal 2	6.66	37.9	2.53	23.4	15.7	0.2418
75th percentile	7.89	33.2	2.60	23.9	16.0	0.3859

Twenty-two samples were selected in 2020 from Martha Underground for comminution variability testing. Samples comprised of quartered HQ drill core from various domains of the MUG deposit. Samples were selected based on Bingo charts in conjunction with the geology team to identify the main gold bearing domains and to indicate the minimum number of composites to be targeted for metallurgical testing. Sample selection was based on core availability for the MUG ore, spatial distribution within each of the MUG vein structures, and representativeness of average gold grades over the LOM. Table 13-13 presents ore characteristics for the 2020 comminution samples. The Axb value (75th percentile) for the 2020 samples is 39.4.

Table 13-13: Summary of Comminution Testing of 2020 MUG Mineralisation Samples

Sample	DWI	Axb	SG	BBWI	BRWI	BAI
Edward 1	4.45	57.5	2.56	16.6	12.9	0.5856
Edward 2	5.58	46.0	2.57	16.5	14.6	0.5353
Edward 3	5.51	46.6	2.58	15.7	14.0	0.7033
Edward 4	5.97	43.4	2.58	17.0	14.5	0.6846
Edward 5	6.17	41.7	2.57	15.6	15.3	0.6923
Edward 6	6.24	41.0	2.57	16.8	14.4	0.428
Edward 7	6.34	40.7	2.59	17.2	14.9	0.5585
Edward 8	4.99	51.2	2.55	16.4	12.7	0.6122
Edward 9	7.74	33.0	2.59	20.0	15.2	0.5234
Empire 1	5.69	45.1	2.55	17.1	14.2	0.6829
Empire 2	5.18	49.7	2.59	16.0	13.9	0.4983
Empire 3	6.63	39.1	2.6	17.0	14.4	0.5865
Empire 4	6.95	36.2	2.53	17.4	15.5	0.4159
Empire 5	5.81	44.3	2.59	17.5	14.3	0.5225
Empire 6	5.55	46.3	2.56	16.6	14.4	0.4056
Empire 7	7.29	35.7	2.6	16.2	14.8	0.626
Empire 8	8.40	34.9	2.92	16.7	15.3	0.6562
Empire 9	6.08	42.1	2.58	18.4	14.0	0.4833
Empire 10	6.30	41.4	2.6	20.3	14.6	0.3723
Empire 11	5.93	42.3	2.5	17.1	15.5	0.715
Rex 1	6.55	39.6	2.58	16.2	14.6	0.6149
Royal 1	6.38	39.7	2.52	17.2	13.3	0.5993
75th percentile	6.57	39.4	2.59	17.2	15.0	0.6629

The comminution results from 2020 indicate that the MUG mineralisation is moderately competent for SAG milling (Axb 39.4 - 75th percentile) and that the ore is hard with a BBWI of 17.2 kWh/t.

Table 13-14 shows the combined results from the 2019 and 2020 comminution testwork programmes. These results indicate that the MUG mineralisation is moderately competent (Axb 36.9) and hard for ball milling with a bond ball work index of 20.2 kWh/t. The tested samples also showed to be highly abrasive with a Bond Abrasion index of 0.63.

Table 13-14: Combined Comminution Testing results 2019 and 2020

Sample	DWI	Axb	SG	BBWI	BRWI	BAI
75th percentile	6.74	36.9	2.59	20.2	15.3	0.6336

The SMC test results can be used to estimate specific power requirements for the MUG ore using the comminution parameters derived from these tests. Based on the power-based model developed by Morrell, the SAG mill specific power is estimated at 10.4 kWh/t (75th percentile) and the Ball mill specific power is 22.1 kWh/t (75th percentile).

The 75th percentile comminution results from 2019 and 2020 were used in the Waihi comminution circuit, a primary grind size of 80% passing 53 µm was utilised for the primary grind for Martha underground feed. The existing grinding circuit is capable of processing Martha underground ores over the LOM.

13.2.3 Recovery Estimates & Assumptions

The recovery models developed for each of the vein structures provided below are based on the reviewed leach testwork results conducted on the historical, 2019 and 2020 samples. Multiple Linear Regression (MLR) was used to predict gold recovery with the explanatory variables being gold head grade and arsenic content in the feed. Table 13-15 provides the recovery models developed for Edward, Empire, Martha, Royal and Rex domains.

Table 13-15: MUG Recovery Models

Domain	Recovery
Edward	Recovery (%) = 96.69 + (0.51 * Au ppm) – (0.077 * As ppm), r^2=0.38
Empire	Recovery (%) = 93.74 + (1.33 * Au ppm) – (0.081 * As ppm),, r^2=0.90
Martha	Recovery (%) = 76.41 + (2.68 * Au ppm) – (0.024 * As ppm), r^2=0.55
Royal	Recovery (%) = 80.25 + (1.41 * Au ppm) + (0.023 * As ppm), r^2=0.96
Rex [3]	Recovery (%) = 91.92 + (0.78 * Au ppm) - (0.092 * As ppm), r^2=0.87

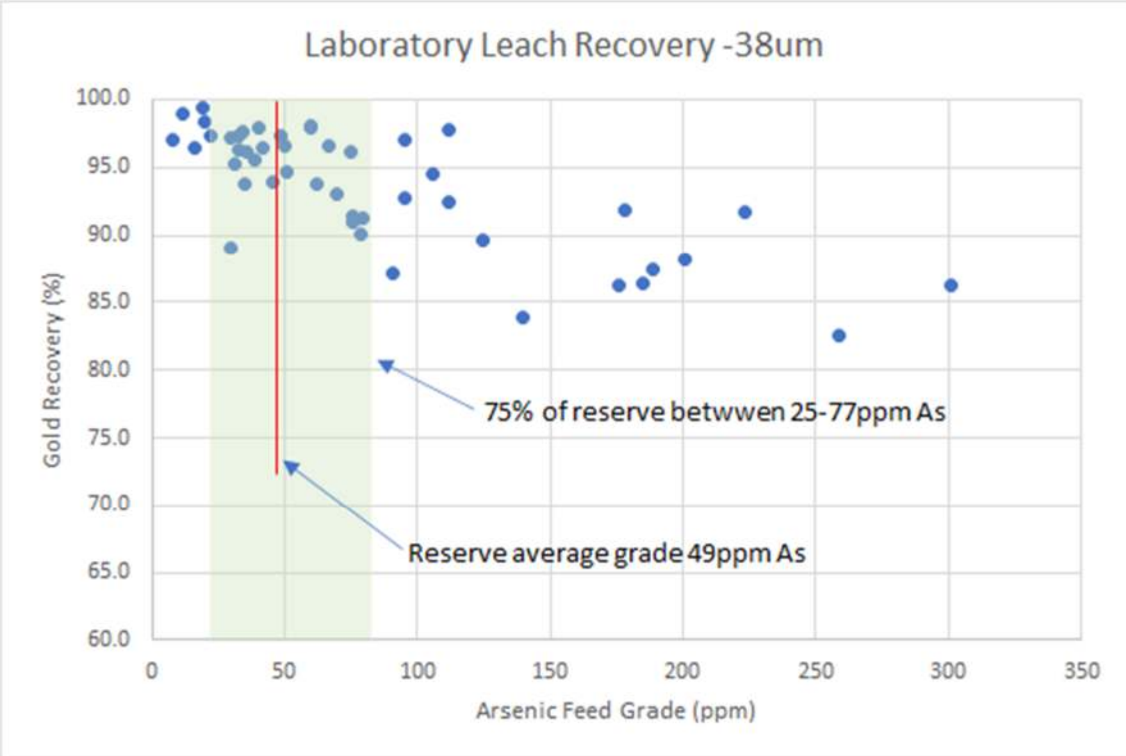
The gold recovery models developed for Martha underground deposit are used to forecast gold recovery in the mine schedule on a yearly basis. Applying the recovery models to the mine schedule indicates the gold recovery for MUG Mineral Reserve is 94.9%.

Recovery is correlated to both gold and arsenic head grades with gold present in arsenopyrite identified previously as being finer grain size than the majority present in ore. High arsenic levels yield a lower recovery in the tested composites over the range from 8 to 301ppm and highlighted in Figure 13-1 related to the arsenic presence.

The block model Gold and Arsenic generated grades in the mine schedule are used to model recovery based on ore source as provided in Table 17-1, with the average grade of the reserve being 49ppm Arsenic and approximately 75% of the reserve between 25 and 77ppm. The impact of arsenic grade on gold recovery for the ore composites tested is shown in Figure 13-2 below along with the average and modelled reserve grade ranges.

[3] The Au recovery model developed for Rex was based on leach testwork data at a grind size P80 75 µm as there were no tests conducted at 38 µm.

Figure 13-2: Arsenic Grade / Recovery Relationship



Further variability and comminution testing will be undertaken on Martha Underground deposit as core becomes available to increase the confidence in the recovery estimates and to investigate potential alternative flowsheets that may further increase overall metallurgical performance. Further Flotation and Ultra fine grind testwork is planned in 2021 to confirm preliminary results and expand sulphur model for the mineralised domains from MUG.

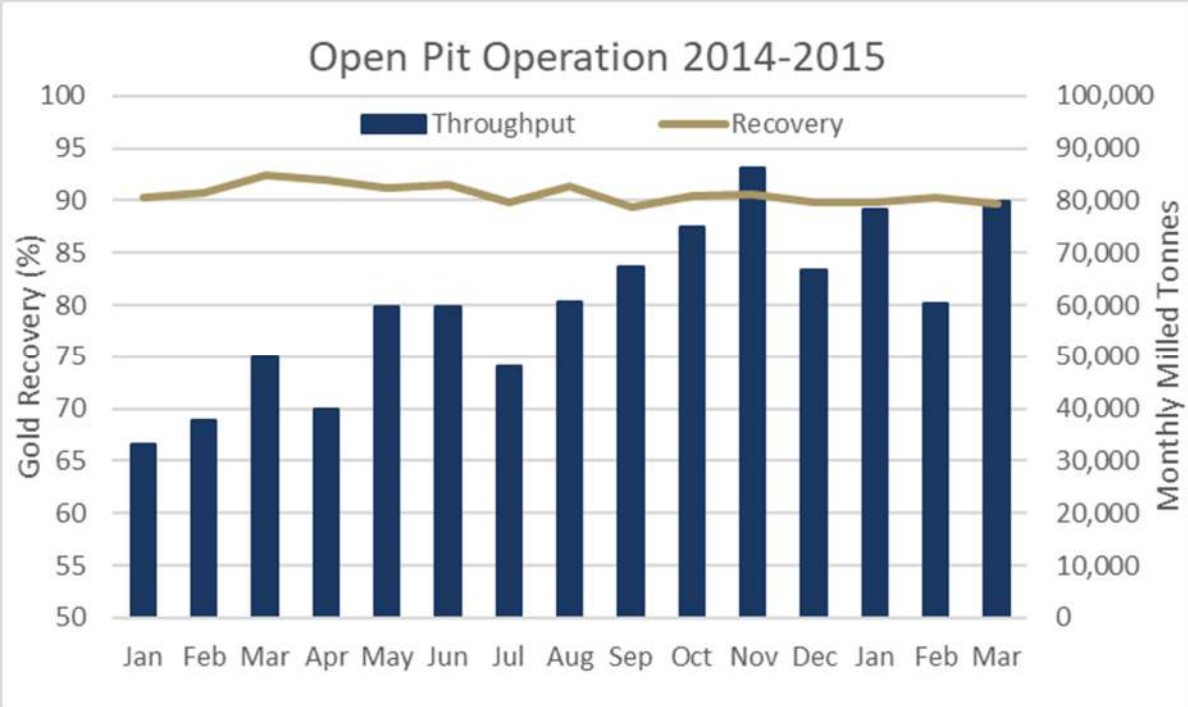
A review of the methodology used to estimate the metallurgical recoveries and testwork was undertaken by G Butcher Consulting Pty Ltd which endorsed the laboratory testing and mathematical modelling methods used to develop the recovery algorithms and that the selection of sampling locations and the representivity of the ore domains appears to have been undertaken with diligence, although additional sampling and testing of the Rex and Royal domains is recommended to improve the confidence of the models developed to date.

13.3 MOP5

MOP5 metallurgical recovery of gold is estimated at 90% and silver recovery is estimated at 63% based on the process plant performance and reconciliations over the last 30 years of operation. Throughput and gold recovery data from the last open pit campaign through the Martha mill in 2014-15 is shown below in Figure 13-3 with the monthly reconciled recovery of 90% achieved or exceeded. The proposed cutback will expose mineralisation at similar or higher levels during the first five (5) years of open pit operation.

As infill drilling of the open pit resource is conducted composites will be prepared for confirmatory tests for ore competency and metal recovery to de-risk the production schedule.

Figure 13-3: Historical Open Pit Performance



13.4 GOP

Laboratory scale test work has been conducted on the drill hole samples obtained for the GOP Mineral Resource. The key focus of the metallurgical work has been to derive gold recovery, throughput rates, reagent consumption and to confirm the suitability of current plant configuration. This test work has shown the GOP mineralisation to be amenable for processing via the existing Waihi treatment plant flowsheet.

Recovery is shown to vary with the weathering extent of the GOP mineralisation.

The weathered domain achieves higher recoveries than the primary un-weathered domain. Separate recovery relationships have been determined for the weathered and un-weathered domains. A small separate metallurgical domain characterised by the hydrothermal breccia host rock was also identified.

A grind size of P80 of 90 µm has been selected, as plant operating experience has shown that this is equivalent to a laboratory gold recovery at a P80 of 75 µm. The gold and arsenic relationship identified in Correnso resource is not observed in the GOP resource. The statistically significant drivers of recovery within the GOP resource are weathering and gold head grade.

The recovery estimate from the test work is calculated at a P80 of 75 µm.

Table 13-16: Recovery Estimate

Testwork	Recovery
Weathered	Recovery % = 100 * (0.902 - (0.049 / Head grade Au))
Un-weathered	Recovery % = 100 * (0.85 - (0.452 / Head grade Au))
Hydrothermal Breccia	Recovery % = 74%

This relationship predicts an average recovery for the GOP resource of 71% based on the average Mineral Resource grade of 1.49 g/t Au. An average process recovery of 71% has been used for GOP based on leaching testwork.

Four samples were submitted for comminution testing to the JKTech testing facilities in Brisbane, Australia. Comminution testing consisted of SMC drop weight and Bond Ball mill work index. The selected samples represent mineralisation to be mined from four domains in GOP. Samples were submitted as quarter core (1/2 HQ). The comminution test results are summarised in Table 13-17. The characterisation conducted on the GOP mineralisation has indicated that the material is classified as 'moderately soft' (i.e. Breccia) to "medium" in terms of competency (SAG milling ‐ average Axb 47.1). The weathered material had the softest BBWI of 17.2 kWh/t and it is categorised as 'hard', the three remaining samples are considered very hard (BBWI 20.9 ‐ 22.6 kWh/t).

Table 13-17: Comminution Testing of GOP Mineralisation Samples

Parameter	HBX	Un-weathered A	Un-weathered B	Weathered	Average
Axb	56.6	47.8	41.2	42.9	47.1
SG	2.49	2.69	2.58	2.54	2.58
DWI	4.39	5.62	6.26	5.93	5.55
BBWI	20.9	22.6	21.3	17.2	20.5

The comminution results provided in Table 13-17 were applied to a comminution circuit similar to current Waihi circuit, and for mill sizing. A primary grind size of 80% passing 75 µm was utilised for the primary grind design for GOP mineralisation.

13.5 WKP

During 2017 and 2018 a series of ten composite samples were generated from drill core obtained from the WKP EG Vein across the long section and at varying depths in several test programs. Eight of these composites represent material in the main EG Vein with the other two testing the footwall and hanging wall structures adjacent. The composite samples were subjected to a standard suite of tests to characterise the recovery of gold from the samples via conventional mineral processing flowsheets similar to that employed at the Processing Plant.

Composite six (6) lies below the main high-grade mineralisation but was included in the test program due to the slightly higher gram-metre result and to test performance of the deeper and higher sulphur mineralisation. From a geo-metallurgical perspective, it is not regarded as representative of the main vein mineralisation and is not included in calculating aggregated results but shows the potential to process material at the extremities of the system. Composites four (4) and six (6) lie in adjacent structures and are not part of the main EG Vein. A further six composites were generated from additional drilling and tested during 2019 from both the EG Vein and EG Footwall Vein. The source of each composite is outlined in Table 13-18 and Table 13-19 and relative location is displayed on the long section diagram of the EG Vein in Figure 13-4.

Testing on the composites was completed by ALS Metallurgy in Perth, Australia and included:

- Head assay and screen fire assay;
- Gravity gold recovery at 106 µm grind size;

- Cyanide leach of both gravity concentrate and gravity tails; and
- Sulphide flotation and leaching of flotation products.

Head grade analysis is outlined in Table 13-20 below and indicate a gold head grade ranging from 4.2 g/t to 50.6 g/t for the main EG Vein samples. Total sulphur head grades range up to 1.82% sulphur and arsenic grades range up to 580ppm, similar ranges to the Correnso north deposit processed at Waihi.

Table 13-18: 2018 WKP Composite Locations

Composite #	Metallurgical Samples		
	Hole ID	Sample No	Vein Structure
2018 Samples			
1	WKP40	WKP40-0492-0500.8	EG Vein
2	WKP42	WKP42-0430.5-0440	EG Vein
3	WKP50	WKP50-0403-0406	EG Vein
		WKP50-0413-0415	
4	WKP52	WKP52-0550	EG FW Vein
	WKP55	WKP55-0363-0364	EG FW Vein
	WKP55	WKP55-0307	EG FW Vein
5	WKP50	WKP0087, WKP50-0093,0094	EG HW Vein
6	WKP35	WKP35-576.4-587.2	EG Vein
7	WKP44	WKP44-0410-0422	EG Vein
8	WKP53	WKP53-0677-0689	EG Vein
9	WKP56	WKP56-0348-0356	EG Vein
10	WKP57	WKP57-0341-0349	EG Vein

Table 13-19: 2019 WKP Composite Locations

Composite #	Metallurgical Samples		
	Hole ID	Sample No	Vein Structure
2019 Samples			
11	WKP54	WKP054-0582-0596	EG Vein
12	WKP60-1	WKP60-0465-0475	EG Vein
13	WKP60-2	WKP60-0570-0577	EG FW Vein
14	WKP61	WKP61-0387-0394	EG Vein
15	WKP63	WKP63-0527-0545	EG FW Vein
16	WKP65	WKP65-0491-0505	EG Vein

Figure 13-4: Long Section along the EG Vein at WKP

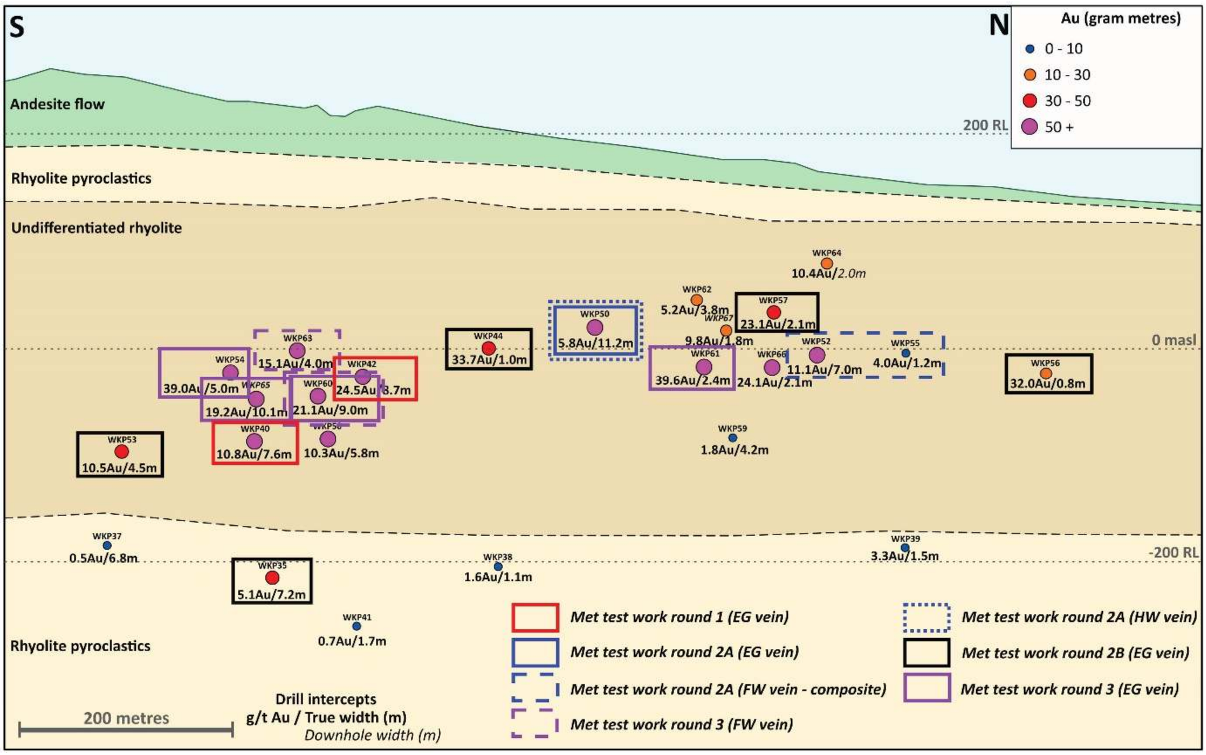


Table 13-20: WKP Composite Head Assay Results

Composite #	Au g/t FA	Ag g/t	As ppm	Hg ppm	SiO2, %	S Total, %
1	7.53	10	15	<0.1	-	-
2	26.0	35	325	0.8	-	-
3	9.47	8	100	<0.1	88.4	0.42
4	4.83	4	270	2.9	82.0	1.34
5	4.54	16	30	0.1	89.2	<0.02
6	4.20	11.4	580	0.4	80.8	1.82
7	4.60	5.4	350	0.1	84.6	0.52
8	7.00	4.5	80	<0.1	89.0	0.26
9	5.21	6.9	390	0.5	80.4	1.74
10	7.67	12.9	110	0.2	81.6	0.86
11	50.6	98	230	<0.1	82.0	0.36
12	19.4	26	80	<0.1	90.2	0.28
13	13.1	24	540	1	86.4	2.06
14	17.7	62	140	0	82.8	0.74
15	62.8	88	30	<0.1	87.6	0.04
16	22.6	24	170	<0.1	84.8	0.62

Gravity concentrates were produced using a laboratory gravity concentrate with the concentrate subject to intensive cyanide leach conditions and the gravity tail subject to standard leach conditions. The combined leach recoveries are indicative of that expected from a conventional gold processing flowsheet.

Table 13-21 shows results from the 2018 composites indicating that gravity gold recovery ranged from 8.1% to 41% averaging 18.4% for the EG Vein samples at either 53 µm or 106 µm grind size. The relatively low gravity recovery results and screen fire assay results suggest the majority of the gold is present as fine particles.

The average gold recovery from leaching on the main EG Vein samples (composites 1,2,3,7,8,9 & 10) averages 90.7% and suggests the majority of the EG Vein material can be regarded as free milling. The lower recovery experienced in composites four (4) and six (6) may be attributable to the higher sulphur feed grade and likely partially refractory locked in sulphides.

Table 13-21: 2018 Composite Gold Recovery Results

Composite #	Calculated Au Grade, g/t	Au/Ag ratio	P80, µm	Gravity Au Recovery, %	Total Au Recovery (%)	
					53 µm	106 µm
1	7.96	1.0/1.2	106	35.1		95.5
2	28.7	1.0/1.2	53	15.1	89.5	
3	9.78	1.0/1.4	53	25.0	89.3	
4	5.08	1.0/1.6	53	8.1	66.4	
5	4.46	1.0/1.4	53	12.5	80.9	
6	3.78	1.0/2.7	106	11.5		68.8
7	5.35	1.0/1.2	106	10.9		91.2
8	6.65	1.0/0.6	106	41.0		95.8
9	5.72	1.0/1.3	106	9.7		84.3
10	7.58	1.0/1.7	106	15.5		89.1
Average					**90.7**	

The 2019 composites examined the effect of grind size on overall recovery with average recovery increasing to 94.3% at a 38 µm grind in the laboratory. In Waihi ores typically higher recoveries are achieved with decreasing grind size from liberation of fine gold present in sulphide particles. The recovery results for these composites are shown below in Table 13-22 indicating a 1.4% improvement in overall gold recovery from grinding from 53 µm down to 38 µm, yielding approximately NZD15/tonne higher revenue based on a 15 g/t Au assumed feed grade.

Table 13-22: 2019 Composite Gold Recovery Results

Composite #	Calculated Au Grade, g/t	Au/Ag ratio	Total Au Recovery (%)				
			38 µm	53 µm	75 µm	90 µm	106 µm
11	50.7	1.0/1.9	95.3	92.6	91.1		
12	19.1	1.0/1.3	96.6	94.7	93.6	91.8	90.6
13	13.2	1.0/1.8	85.9	86.1			
14	18.9	1.0/2.8	96.1	96.2	96.5	95.0	
15	59.7	1.0/1.5	95.5	93.4	93.4	91.6	
16	23.1	1.0/1.0	96.2	94.6	92.3	91.0	
Average			**94.3**	**92.9**	**93.4**	**92.4**	**90.6**

Processing Plant operating experience has shown that an equivalent laboratory gold recovery at a P_{80} of 38 µm is equivalent to a grind size P80 of 53 µm in the plant. This relationship is due to the laboratory grind test work being in open circuit, whereas in the plant the grinding circuit is in closed-circuit. This results in the higher density sulphides being preferentially ground finer from the cyclone classification and hence liberating more gold particles that are disseminated within the sulphides.

Diagnostic leach tests were completed on direct leach tailings samples for 10 of the composites from the EG Vein. The results show there is little free milling gold remaining in the tails (6%) that would be recoverable with longer leach residence time. Up to 32% of the unleached gold appears to be silica locked and given the high silica head grade is unlikely to be recoverable via leaching or flotation without further grinding to liberate the locked gold. Unleached gold locked with sulphide minerals represents 61% of the total gold lost to tailings. The sulphide minerals may be recovered through sulphide flotation. Preliminary flotation testwork conducted at 75 µm on the EG Vein has indicated no significant recovery benefits when compared to direct cyanidation at a grind size of 38 µm (i.e. 92% (flotation) vs 96% (direct cyanidation)). Further FUFG testwork is planned in 2020 on WKP samples to confirm if there are recovery benefits.

The recovery model developed for the EG Vein structure provided below is based on the leach testwork results for the high-grade samples at a grind size of 38 µm.

MLR was used to predict gold recovery with the explanatory variables being Au head grade and arsenic content in the feed. Below is the recovery model developed for EG veins at 38 µm.

EG Vein Recovery (%) = 97.14 - (0.024 * Au ppm) – (0.0028 * As ppm), r^2=0.93.

The gold recovery model developed for WKP deposit applies for the high-gold grades i.e. >13 g/t Au.

The testwork completed to date supports the adoption of a direct leach flowsheet for gold recovery at a laboratory primary grind size of 38 µm and an expected recovery of 94.9% or higher is a reasonable assumption at gold grades >13 g/t Au. Further leaching testwork is planned in 2020 over a wider range of Au head grades (i.e. gold head grades <13 g/t Au) to validate model recovery for WKP. A process recovery of 90% has been used for WKP in this Technical Report based on 2018 leaching testwork.

Six (6) samples were submitted for comminution testing to the JKTech testing facilities in Brisbane, Australia. Comminution testing consisted of SMC drop weight, Bond Rod Mill and Bond Ball Mill work indices, and bond abrasion index. The selected samples represent mineralisation to be mined from two vein structures at WKP. Samples were submitted as quarter core (1/2 HQ). The comminution test results are summarised in Table 13-23. The characterisation conducted on the Waihi mineralisation sources has indicated that WKP mineralisation is very competent (SAG milling – average Axb 37.2) and hard to grind in ball mill (BBWI 18.7 – 22.1 kWh/t). Further comminution testing on WKP mineralisation is planned in 2020.

Table 13-23: Summary of Comminution Testing of WKP Mineralisation Samples

Parameter	EG Comp 1	EG Comp 2	EG Comp 3	FW Comp 1	FW Comp 2	FW Comp 3	Average
Axb	38.8	34.0	39.2	39.4	36.8	34.7	37.2
SG	2.49	2.53	2.53	2.59	2.62	2.59	2.56
DWI	6.43	7.51	6.45	6.55	7.07	7.38	6.90
BRWI	16.1			16.0			16.1
BBWI	18.7	22.1	19.1	19.3	20.0	20.3	19.9
BAI	0.70			0.70			0.70

The comminution results provided in Table 13-23 and Table 13-12 were applied to a comminution circuit similar to current Waihi circuit, and for mill sizing. A primary grind size of 80% passing 53 µm was utilised for the primary grind design for WKP feed.

13.6 Further Testwork

Further variability and comminution testing will be undertaken on MUG and WKP deposits as core becomes available to increase the confidence in the recovery estimates and to investigate potential alternative flowsheets that may further increase overall metallurgical performance.

When drill core becomes available for the MOP5 infill drilling additional direct leach variability testing will be undertaken to confirm the recovery assumptions from historical plant performance.

13.7 Comments on Section 13

In the opinion of the QP, the following conclusions are appropriate:

- Metallurgical test work and associated analytical procedures were performed by recognised testing facilities, and inhouse facilities and the tests performed were appropriate to the mineralisation type;
- Samples selected for testing were representative of the various types and styles of mineralisation within the Waihi and WKP areas. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on adequate sample mass;
- Average weighted recoveries have been assumed based on test work completed and source proportion in the inventory. These recoveries are appropriate to be used in support of Mineral Resource and Mineral Reserve estimation, based on the drill hole spacing and sample selection;

- Metallurgical testwork conducted on the composites to date from the MUG deposit supports an expected gold recovery assumption of 95% for treatment through the existing process plant flowsheet based on targeting a primary grind size of 53 µm used in the mine optimisation;
- Metallurgical testwork on the main EG Vein samples tested from the WKP deposit supports an expected gold recovery assumption of 90% for treatment through the existing process plant flowsheet based on a similar 53 µm grind size, and
- Historical metallurgical results on the Martha open pit deposit supports an expected gold recovery assumption of 90% for treatment through the existing process plant flowsheet based on a 90 µm grind size.

14 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions / Basis of Estimate

Mineral Resource Estimates for four projects have been prepared with close out dates for the databases used in estimation as shown in Table 14-1. Data used to support the estimates include surface and underground diamond drill core, RC chips and underground grade control channel sample chips.

Table 14-1: Model Close Out Dates

Project	Close out Date
MUG	30 November 2020
MOP5	1 November 2019
WKP	31 December 2019
GOP	February 2018

14.2 Geological Models

Open pit and underground mining since 1988 have provided a large database of mapping and grade control sampling, which has confirmed the geological interpretation to date.

The geological interpretation process routinely utilises all available drill logging data, core photography, drill assay data and oriented core measurements, all of which are systematically collected and validated. Often additional data may be available to contribute to the final geological model including surface mapping, underground face and backs mapping, Grade control channel sampling, historical underground quartz vein mapping and channel sampling.

Geological modelling of the MOP5, MUG, WKP and GOP Projects was performed in Leapfrog using the interval selection and vein systems tools. Drilling data in Leapfrog was linked directly to the ADMWAIHIEXP AcQuire database.

Geological models and geological concepts have been routinely reviewed by internal and external reviewers.

14.2.1 MUG

A MUG geological model update and reserve estimate was completed in November 2020.

The comprehensive Martha dataset includes DD, in-pit mapping, Grade control channel and RC data, backs mapping from modern development, historic cross cuts, historic mapping, digitised historic mining wireframes.

Comparative assessment of the November 2020 model and previous iterations indicates only modest localised vein geometry changes relative to the previous estimate and an increase in resource confidence classifications within the deposit as a consequence of the infill drilling completed in 2020.

Updates to November 2020 model include:

- New drilling undertaken from a number of underground drill platforms, with significant infill drilling having occurred in the central and western area of the MUG project to allow infill between two previously drilled areas of the resource following the completion of drill drive development in 2019;

- Some additional veins defined within the geologic model;
- Backs mapping from underground development;
- Historical workings were updated including development headings, open stope and filled stope triangulations; and

Outputs from the November model includes:

  - 81 vein triangulations (Figure 14-1). The fundamental architecture remains consistent. Additional DD has allowed for the extrapolation of more minor structures clearly present in open pit grade control data;
  - 10 lithology triangulations; and
  - 3 oxide triangulations.

Historical workings were updated including development headings, open stope and filled stope triangulations.

The model was built with underground mining economics in mind, and delineation of consistently narrow or low-grade structures was not necessary.

Wireframes were created using Leapfrog Geo software. Geological logging fields of drilling data such Vein Textures, Vein Mineralogy, Vein Percentage, Breccia Type and Historical Voids were initially used to create representative wireframes of vein structures. These initial wireframes were then modified on a vein-by-vein basis and compared to Au and Ag grade, core photography and structural measurements to establish geological consistency between veins.

Veins defined by pit grade control data but without supporting drilling information to substantiate vein extrapolation beyond the pit boundary were not included in the wireframes.

Individual veins were validated at various stages throughout the modelling process. Upon completion of the modelling process, additional validation includes:

- A visual review in three axis sliced planes viewing Au grade, historical voids and logged geology;
- Drill hole review following domain flagging and filtering for Au immediately outside of vein boundaries;
- Peer review within the Waihi geology team; and
- Review against historic mining. Note that in instances where mined voids had no drill data, relative position of stoping panels was determined using vein wireframes. This ensures a conservative approach was taken to depletion.

Figure 14-1: Martha Vein Domain Triangulations used in the November 2020 Estimation

1100_Martha.00t
1109_Martha_East.00t
1110_Nth_Branch.00t
1111_Loop_No1.00t
1112_Martha_Sth_Sect.00t
1113_Loop_No2.00t
1120_Mary.00t
1130_No2.00t
1131_No2_West.00t
1132_No2_South.00t
1140_Flat_A.00t
1141_Flat_B.00t
1143_Ella.00t
1144_Flat_C.00t
1201_Albert.00t
1220_Edward.00t
1221_Edward_Link.00t
1222_Edward_B.00t
1223_Edward_C.00t
1224_Edward_D.00t
1225_Edward_A.00t
1226_Edward_F.00t
1227_Edward_E.00t
1228_Edward_B1.00t
1229_Edward_B2.00t
1230_Edward_FW1.00t
1302_Victoria.00t
1304_Magazine.00t
1305_Welcome.00t
1306_Welcome_Back.00t
1307_Welcome_C.00t
1308_Alexandra.00t
1309_Welcome_D.00t
1310_Welcome_E.00t
1311_Welcome_F.00t
1312_Welcome_G.00t
1313_Welcome_H.00t
1320_Grace.00t
1331_Vic_FW3.00t
1332_Vic_FW9.00t
1333_Vic_FW7.00t
1334_Vic_FW10.00t
1335_Welcome_HW1.00t
1336_Welcome_HW2.00t
1400_Empire.00t
1401_Letter_H.00t
1402_Letter_J.00t
1403_Letter_C.00t
1404_Gordon.00t
1405_Letter_D.00t
1406_Letter_X.00t
1407_Letter_L.00t
1408_Letter_Y.00t
1409_Emp_HW1.00t
1410_Letter_E.00t
1411_Letter_M.00t
1420_Nth_Sect_Empire.00t
1421_State_Branch.00t
1422_State_Reef.00t
1423_Dominion.00t
1424_Republic.00t
1425_Wowser.00t
1426_Harry.00t
1427_Boxall.00t
1500_Royal.00t
1501_Royal_FW_A.00t
1508_Archie.00t
1510_Rex.00t
1511_Princess.00t
1512_Royal_Nth_Branch.00t
1513_Dreadnought.00t
1514_Dreadnought_Sth.00t
1515_Meghan.00t
1516_Royal_HW2.00t
1517_Royal_HW3.00t
1520_George.00t
1530_Ulster.00t
1551_Louis.00t
1552_Emp_F1.00t
1553_Roy_HW1.00t

14.2.2 Historical Workings Model

Given the mining history of the MUG Project the accurate treatment of historic mine workings is recognised as being of high importance to the project.

A 3D model of historic workings was constructed as part of ongoing geotechnical studies and captures the extent of known stopes within the major lode structures. A 3D model of the historical stopes was initially constructed by draping digitised historical long sections of stoping blocks on to the footwall of the vein wireframe to form 3D polygons. These 3D polygons were then extruded, towards the vein hanging wall, the average width of the block as determined from historic data to form a solid wireframe. Using the historical long sections, the stope wireframes were then attributed with stoping type, to determine if filled or void. Unknown types were assumed to be voids, unless verified by current mining. A review of the historic development and stope models of the Martha and Grand Junction workings in 2009 found the original interpretation of most historic workings were modelled between 2 m and 15 m lower than surveyed intercepts with workings in the pit.

Recently, new data has provided an additional source from which the historical void model can be updated and remodelled. This new data includes the ongoing Martha DD program, recent underground mine development (2017 to present) and additional historical mine plans made available through the Auckland War Memorial Museum. Underground surveyors provided the geology team survey pickups of all old workings intercepted during mining.

Significant updates were applied to the historical mine workings. All the workings were separated into individual wireframes and assessed for position, orientation and width against all the newly available data. Where required, the following adjustment techniques were applied using Vulcan software:

- **Translation:** workings were shifted to match drill hole intercepts and/or vein wireframes. Translations used a 'reference point to destination point vector';
- **Rotation:** whole wireframes were rotated either in cross section or plan view to match vein wireframe orientations. Stopes that required partial or incremental rotation were sectioned into polygons then each polygon was rotated individually in cross section before the solid wireframe was recreated;
- **Reshaped:** stopes that required width change to match drill hole intercepts were also sectioned into polygons, moved and reshaped to match drill hole logging before the solid wireframe was recreated; and
- **Reclassified:** stopes were reclassified if the recent data (drilling/development) contradicted with the void/fill classification in the original stope model.

Stope shapes and levels were validated for closure, consistency and crossing triangles to ensure they could be evaluated for volume, then re-merged into a complete set of development levels, filled stopes and open stopes. All remodelled historical workings were peer reviewed and validated against previous models. All updates are recorded in a 'stope adjustment register'. The updated model contains wireframes for development levels, open stopes, filled stopes, shafts, passes and the Milking Cow caved zone.

14.2.3 MOP5

The MOP5 Project is a cutback on the existing open pit on which the Waihi Mining District commenced modern mining operations in 1988. Mining in the open pit ceased in April 2015 as a consequence of a localised wall failure that compromised access to the active working area. Subsequent to the loss of access, the open pit Mineral Reserve of 0.08Moz. was reclassified as Mineral Resource.

The Martha Open Pit mineralisation is very well understood. The open pit had operated continuously for a period from 1988 until 2015, with a number of cutbacks completed successfully during the operations history. The proposed MOP5 cutback leverages off this knowledge base and utilises knowledge gained through the successful exploration of MUG, which sits directly beneath the proposed cutback and changes in economic assumptions.

Gold was first discovered at Martha in 1878 and was historically mined by underground methods between 1882 and 1952, producing some 4.9 Moz of gold and 29 Moz of silver from 12 Mt of ore. The deposit was worked over 1600 m in strike and to a depth of 600 m.

The modern open pit operation began in 1988 mining remnant material adjacent to lodes and backfill material from the workings. All material produced in the pit was crushed and conveyed 2.5 km to the Processing Plant and TSF.

The quartz vein system at Martha is hosted by hydrothermally altered quartz bearing andesite flows and flow breccias inter-bedded with thin tuffaceous sediments, dipping south-east at about 40°. These are unconformably overlain by a post-mineral sequence of late Pliocene to Quaternary ignimbrite and alluvial units. These units thicken to the south and east and are Inferred to infill a caldera-like structure. Oxidation extends down the vein margins to over 250 m below surface.

Gold-Silver mineralisation within the deposit is contained in quartz veins within a low-sulphidation epithermal vein system hosted by Miocene calc-alkaline volcanics of the CVZ. The system comprises of four main northeast trending veins (Martha, Welcome, Empire and Royal) and a two north trending cross-cutting vein structures, the Edward and Albert. The main veins are enveloped by a stockwork of subsidiary veins. Mineralisation extends for 1600 m along strike with a width of 500 m and was historically mined to over 600 m below surface.

Management of historic voids within the Martha open pit resource is as per the process described in section 14.2.2 of this report.

14.2.4 WKP

The most recent WKP Mineral Resource was reported to the market in February 2020. Veins were modelled using Leapfrog Geo software. Geological logging of drill core such as Vein Textures, Vein Mineralogy, Vein Percentage and Breccia Type were initially used to create representative wireframes of vein structures. These initial wireframes were then modified on a vein-by-vein basis and compared to Au and Ag grade, core photography and structural measurements to establish geological consistency between veins. Some small, sporadic high-gold grade intercepts that cannot be corelated with neighbouring drillholes have been excluded from the vein modelling.

In September 2019, geological consultant David Rhys from Panterra Geoservices Inc. completed an independent review of the geological setting, style and structural controls on mineralisation at the WKP Project based on all available surface mapping and drill hole data.

14.2.5 GOP

GOP is based on open pit's around the Gladstone Hill and Winner Hill area. The resource model describes the mineralisation within Gladstone and Winner Hills and includes part of the Moonlight orebody, depleted for underground mining.

GOP mineralisation is characterised by shallow-level, hydrothermal breccias and associated banded quartz veins interpreted to represent the top of the epithermal system. The uppermost mineralised quartz veins flare up into hydrothermal explosion breccias. The GOP veins are predominantly steeply dipping veins developed within the hanging wall of the Favona Fault that dips moderately towards the SE. The vein trend ENE to NE between 035 ° and 075 ° and dips steeply towards the SE.

14.3 Exploratory Data Analysis

14.3.1 MUG

The MUG Project has an extensive mining history, consequently there is abundant data collected over many years that requires assessment in construction of grade estimates for the deposit. The model update incorporates all available data including exploration DD data, in-pit grade control channel data and in-pit grade control RC drill data to build of the geologic model and undertake the grade estimate.

There are three datasets from within the open pit that have been utilised in the development of this model:

- A large set of grade control channel data from the open pit collected between 1988 and 2008;
- RC drilling data collected between 2008 and 2015; and
- Exploration drilling data collected over the life of the project.

Table 14-2: Summary Statistics for MUG Major Domains (2 m composites)

Lode	Martha	Albert	Edward	Victoria	Magazine	Welcome	Empire
Statistic	Domain 1100	Domain 1201	Domain 1220	Domain 1302	Domain 1304	Domain 1305	Domain 1400
Samples	3043	94	944	251	333	845	1376
Minimum	0.005	0	0.005	0.025	0	0.005	0
Maximum	208.99	20.45	131.32	18.31	114.06	88.66	345.76
Mean	3.28	2.41	5.13	2.92	3.31	3.36	5.43
Standard deviation	7.42	4.18	9.75	3.06	8.13	5.48	13.77
CV	2.26	1.73	1.90	1.05	2.46	1.63	2.53
Variance	55.00	17.45	95.08	9.34	66.15	30.08	189.59
Skewness	11.18	2.50	6.00	1.75	10.51	6.63	14.12
Log samples	3043.00	93.00	944.00	251.00	332.00	845.00	1372.00
Log mean	-0.15	-0.83	0.45	0.41	-0.04	0.17	0.37
Log variance	3.41	4.79	3.21	1.83	3.42	2.96	3.67
Geometric mean	0.86	0.44	1.56	1.50	0.96	1.18	1.45
95%	13.24	13.52	19.83	9.00	11.44	12.26	20.55
97.50%	18.89	13.70	26.04	11.10	16.35	16.00	32.53
99%	29.80	18.54	38.18	13.48	21.10	22.22	46.56

The channel and RC data are spatially distinct from each other and cover those portions of the deposit that have already been mined or are immediately adjacent to the mined portion of the deposit whereas the exploration drilling data covers the full extent of the area being modelled.

14.3.2 WKP

Statistical assessment is undertaken on independent domains for all Waihi deposits, the domains are defined by vein wireframes. All domains in Waihi/WKP District have hard boundaries. Summary statistics by domain are tabulated using the Snowden “Supervisor” package. Outputs from the analysis are presented in Table 14-3 to Figure 14-3.

Table 14-3: Summary Statistics for WKP Major Domains

	T Stream	EG	EG FW1	Sth FW Spla
Statistic	Domain 400	Domain 410	Domain 420	Domain 425
Samples	184	230	197	104
Minimum	0.025	0.09	0.09	0.19
Maximum	57.9	239	39.207	151.086
Mean	2.06045	12.7132	2.86772	19.9816
Standard deviatior	5.51032	21.4893	5.64885	30.8568
CV	2.67432	1.69031	1.96981	1.54426
Variance	30.3636	461.791	31.9095	952.145
Skewness	6.79037	5.28429	4.09362	2.10757
Log samples	184	230	197	104
Log mean	-0.410525	1.5366	0.272306	1.69018
Log variance	1.75567	2.46686	1.17736	3.19996
Geometric mean	0.663302	4.64878	1.31299	5.42047
50%	0.479	5.66	1.09	4.574
95%	10.555	47.855	14.171	90.197
97.50%	13.99	58.85	21.97	108.747
99%	20.653	101.199	25.008	111

Figure 14-2: WKP Log Probability Plot of the EG Vein

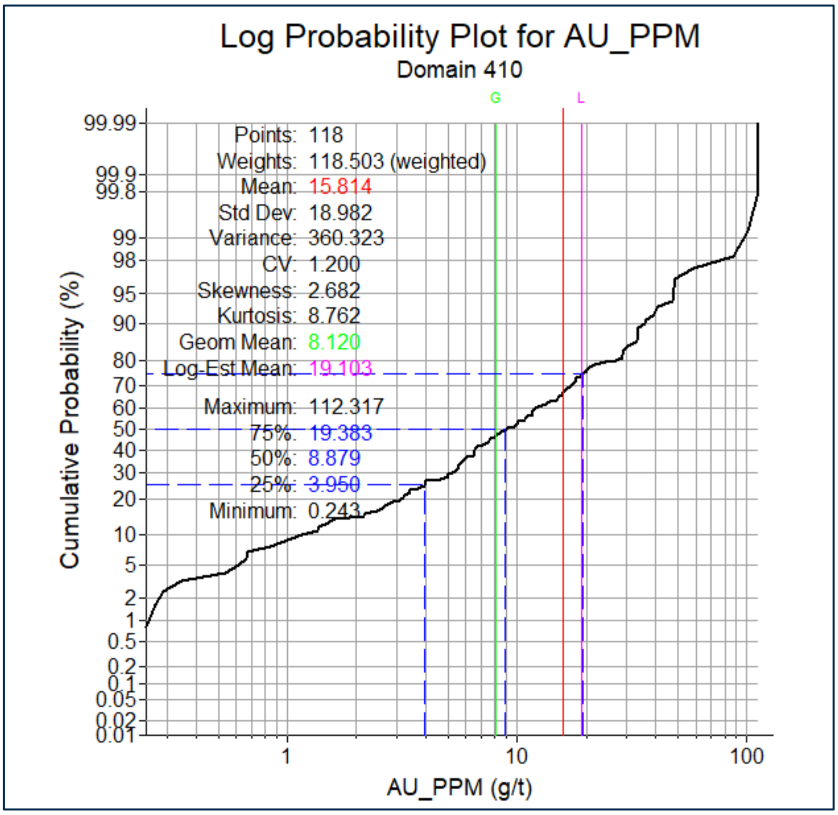
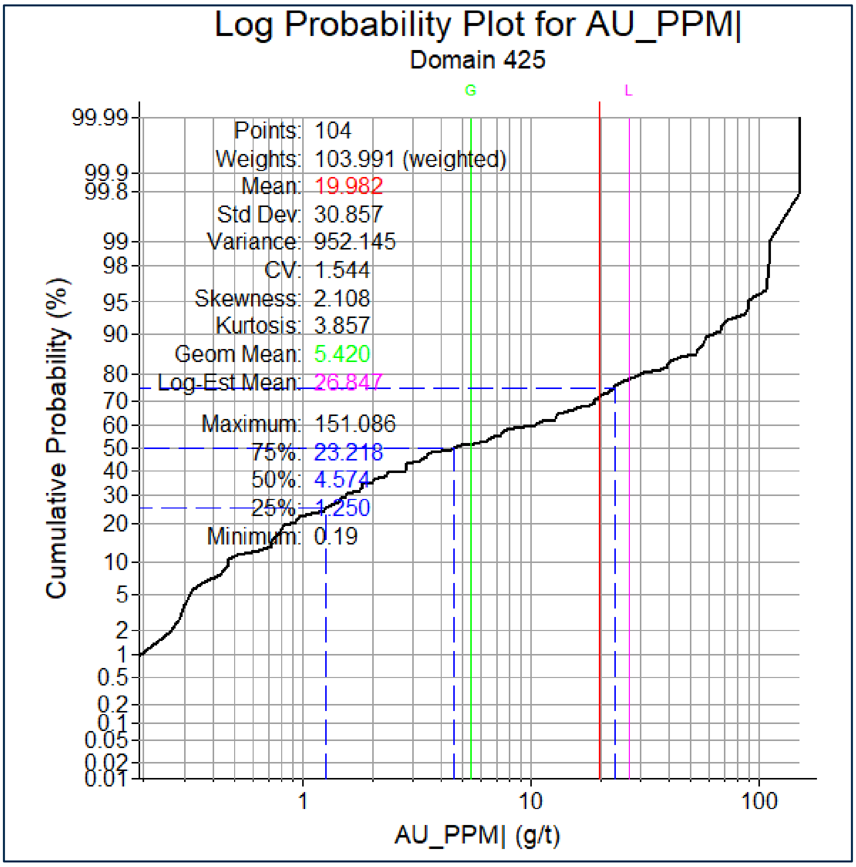


Figure 14-3: WKP Log Probability Plot of the South FW Splay



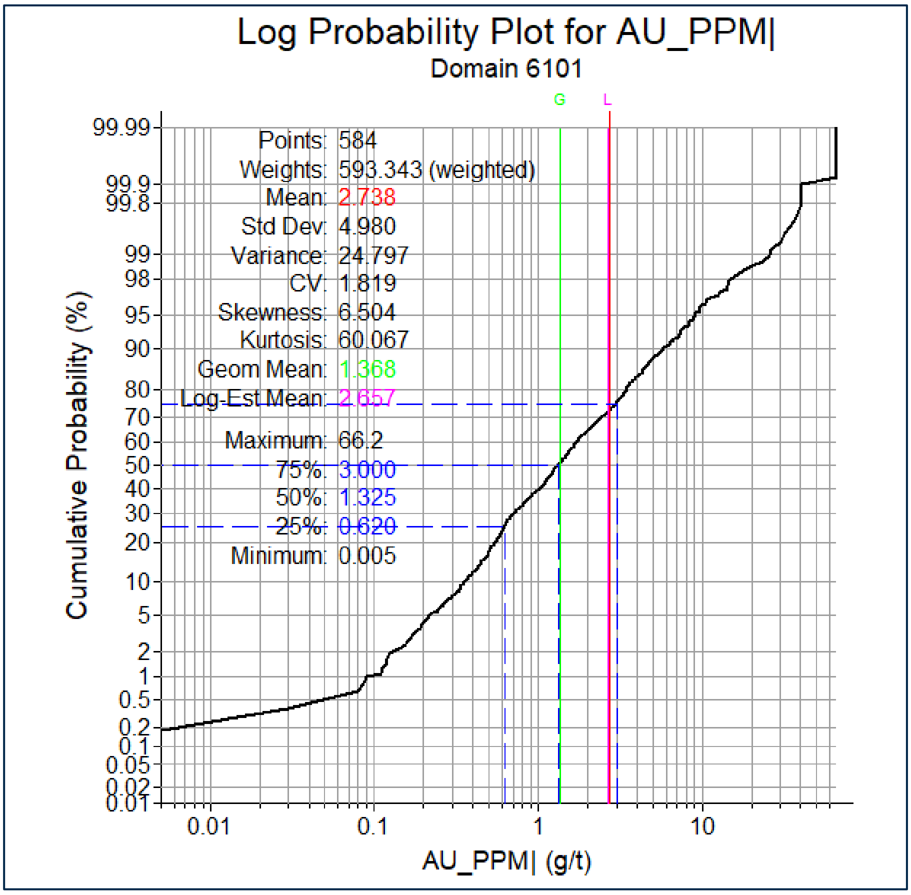
14.3.3 GOP

Summary statistics for the major GOP domains are presented in Table 14-4. As with all estimates at Waihi utilising an Inverse Distance based estimation scheme the top-cut is set to the 98th percentile of the cumulative distribution as illustrated in the cumulative probability plot for the primary GOP vein shown below in Figure 14-4.

Table 14-4: Summary Statistics for GOP Major Domains

Statistic	Domain 6101	Domain 6102	Domain 6103	Domain 6105	Domain 6109	Domain 6201	Domain 6204
Samples	1138	736	418	815	296	2441	26093
Minimum	0.03	0.01	0.03	0.01	0.01	0.01	0.01
Maximum	66.2	26.363	25.7	35.137	23.2	25.88	57.637
Mean	2.74	1.31	1.55	1.56	1.46	0.38	0.31
Standard deviation	5.43	1.91	2.60	3.27	2.22	0.78	0.93
CV	1.98	1.46	1.67	2.10	1.52	2.02	3.05
Variance	29.49	3.65	6.74	10.71	4.92	0.61	0.87
Skewness	6.65	5.64	5.84	6.04	5.31	16.80	25.57
Log samples	1138	736	418	815	296	2441	26093
Log mean	0.23	-0.41	-0.15	-0.52	-0.24	-1.67	-1.98
Log variance	1.44	1.73	1.09	2.07	1.30	1.47	1.41
Geometric mean	1.26	0.66	0.86	0.59	0.79	0.19	0.14
95%	9.23	3.84	4.24	5.22	4.75	1.27	0.94
97.50%	14.20	6.24	7.05	8.87	6.59	1.73	1.49
99%	25.60	9.83	13.53	17.30	10.86	2.92	2.67

Figure 14-4: GOP Log Probability Plot for the 6101 Domain



14.3.4 MOP5

Summary statistics for the major MOP5 domains are presented in Table 14-5. As with all estimates at Waihi utilising an Inverse Distance based estimation scheme the top-cut is set to the 98th percentile of the cumulative distribution as illustrated in the cumulative probability plot for the primary Martha vein shown below in Figure 14-5.

The open pit channel and RC data has been utilised in the construction of the Martha model, these datasets are spatially distinct from each other and cover those portions of the deposit that have already been mined or are immediately adjacent to the mined portion of the deposit whereas the exploration drilling data covers the full extent of the area being modelled. Data analysis is completed for each domain and each data type as a routine process in the construction of the Martha grade estimates. Differing composite lengths are utilised for differing styles of mineralisation within the Martha deposit. To this end data analysis is also conducted on 1.5 m and 3 m composites for each data type and each domain.

Table 14-5: MOP5 Bulk Domains 1.5 m Au Composite Statistics by Domain DD Dataset

Lode	Martha	Edward	Victoria	Welcome	Empire	Royal
Statistic	Domain 1100	Domain 1220	Domain 1302	Domain 1305	Domain 1400	Domain 1500
Samples	1967	410	265	716	558	147
Minimum	0.005	0.005	0.02	0.01	0.005	0.005
Maximum	283.9	151.0	20.7	85.0	117.5	44.9
Mean	3.3	6.3	2.9	3.3	5.0	4.5
Standard deviation	9.5	14.7	3.5	6.0	9.3	7.3
CV	2.9	2.3	1.2	1.8	1.9	1.6
Variance	89.4	214.6	12.5	35.7	86.7	52.8
Skewness	17.1	5.6	2.1	6.2	5.7	2.8
Log samples	1967	410	265	716	558	147
Log mean	-0.4	0.1	0.2	0.0	0.5	0.0
Log variance	3.9	4.6	2.3	3.1	2.9	4.7
Geometric mean	0.7	1.1	1.2	1.0	1.6	1.0
95%	14.1	25.0	10.2	11.8	17.8	16.7
97.50%	19.9	39.7	12.4	17.5	28.0	26.8
99%	26.4	80.8	16.5	26.1	47.2	35.0

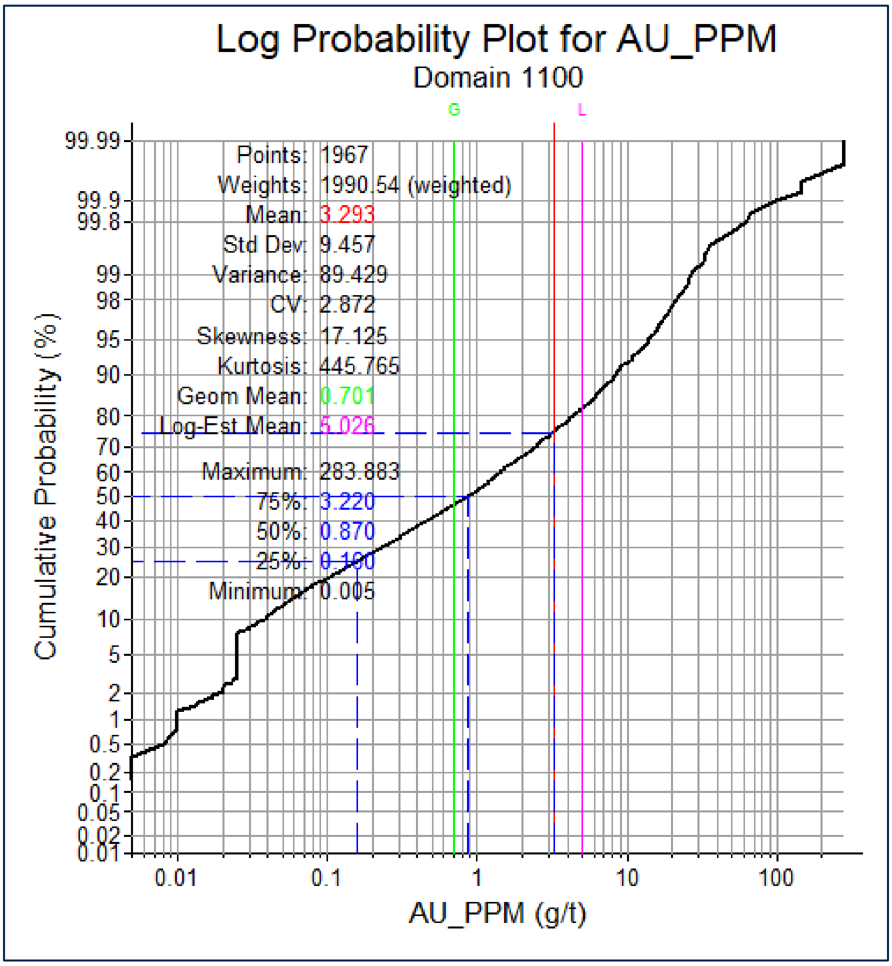
Table 14-6: Bulk Domains 3m Au Composite Statistics by Domain DD

Statistic	Domain 1900	Domain 1901	Domain 1902	Domain 1903	Domain 1904	Domain 1905
Samples	1892	1797	398	672	3346	1783
Minimum	0.005	0.002	0.005	0.007	0.005	0.005
Maximum	88.38	77.94	14.07	33.97	58.01	35.20
Mean	0.69	0.70	0.60	1.34	0.35	0.95
Standard deviation	3.27	3.17	1.54	3.12	1.87	2.20
CV	4.76	4.56	2.57	2.33	5.29	2.32
Variance	10.70	10.07	2.36	9.73	3.50	4.85
Skewness	16.60	14.61	4.76	5.40	17.75	7.39
Log samples	1892	1797	398	672	3346	1783
Log mean	-2.75	-2.24	-2.46	-1.17	-2.83	-1.24
Log variance	4.25	2.88	3.86	3.10	2.50	2.43
Geometric mean	0.06	0.11	0.09	0.31	0.06	0.29
95%	2.99	3.03	3.26	6.70	1.39	4.08
97.50%	5.28	5.45	5.14	9.55	2.62	6.34
99%	10.63	9.24	7.28	13.53	4.74	9.78

Table 14-7: Bulk Domains 3 m Au Composite Statistics by Domain DD

Statistic	Domain 1906	Domain 1908	Domain 1909	Domain 1910	Domain 1911	Domain 1912
Samples	1840	783	305	940	3363	2221
Minimum	0.005	0.005	0.007	0.002	0.005	0.005
Maximum	15.16	18.50	26.47	4.21	38.14	61.90
Mean	0.18	0.64	1.56	0.06	0.33	0.31
andard deviati	0.81	1.57	2.82	0.23	1.74	1.10
CV	4.56	2.44	1.81	3.56	5.24	3.61
Variance	0.66	2.47	7.96	0.05	3.01	1.22
Skewness	10.23	5.59	5.10	13.65	12.55	18.63
Log samples	1840	783	305	940	3363	2221
Log mean	-3.53	-1.81	-0.41	-3.53	-3.40	-2.83
Log variance	2.12	2.51	1.87	0.98	2.98	2.40
eometric mea	0.03	0.16	0.66	0.03	0.03	0.06
95%	0.68	3.02	5.46	0.17	1.24	1.51
97.50%	1.68	4.82	6.99	0.35	2.76	2.63
99%	3.13	8.05	17.60	0.53	6.32	4.57

Figure 14-5: Martha Vein Log Probability Plot 1.5 m Au Composites



14.4 Composites

Composite weighting by length was applied during estimation to avoid bias from very small, high-grade composites. There has been no change to the compositing method for any Waihi Projects used since May 2010.

The standard method used to define composites for all resource estimates was to flag the raw data in the local drilling database for the project against the geology solids. The Vulcan compositing program (run length) was run to generate a length composited database at the required sample length. Compositing was by fixed length, honouring the domain boundaries. 1m fixed length composites are routinely generated for the narrow veins across all deposits. There are five vein-based domains in the MUG Project that have a vein width of greater than 10 m, these broader domains are composited to a 2 m fixed length interval.

For narrow domains across all underground deposit the drilling data is composited to a 1 m composite length using the distributed technique, this methodology is consistent with the techniques applied for the Waihi deposits. Composite weighting by length is applied during estimation to avoid bias from small, high-grade composites.

Open pit models are estimated using larger composites. Veins domains are composited to a 1.5 m length and bulk domains to 3 m, this being representative of the mining bench height and therefore the implied mining selectivity inherent in the model.

14.5 Grade Capping / Outlier Restrictions

14.5.1 MUG

Reconciliation history for the Waihi Project has demonstrated that some level of high-grade restriction is necessary to limit the influence of outliers on grade estimates for the epithermal veins that have been mined during the operations history;

Statistical assessment of the input data is undertaken by domain, typical top-cut selection is based on the assessment of the population distribution characteristics and for Inverse Distance estimates cutting at the 98th percentile on the log probability distribution has been a long-standing methodology that has produced acceptable results. Estimates using an OK estimation scheme utilise a 99th percentile threshold;

The use of this method in determining top caps has resulted in good reconciliation historically. Typically, different data types are assessed independently in the capping analysis process; and

MUG estimate is based on an OK estimation plan and based on comparative assessment of the OK outputs a top-cut % of 99% has been adopted for kriged estimates.

- Number of samples above the cap
- Percentage of samples above the cap
- Minimum, maximum, mean and variance of samples above the cap
- Mean and variance of uncapped data
- Mean and variance of capped data
- Capped % difference:

$$\frac{(\text{uncapped mean} - \text{capped mean})}{\text{uncapped mean}} \times 100\%$$

o Contribution of the samples above the cap to the uncapped variance:

$$(\text{mean above the cap} - \text{uncapped mean})^2 \times \frac{\%\text{ of data above the cap}}{\text{uncapped variance}}$$

- Contribution of the samples above the cap to the total metal:

$$(\%\text{ of data above the cap}) \times \frac{\text{mean of data above cap}}{\text{uncapped mean}}$$

Statistical assessment was undertaken independently on the different data types as it is recognised that the data is spatially zoned.

Top-cut assessment was undertaken on each of the fixed length composited datasets generated in the compositing stage, top cuts were then assigned by domain to the individual datasets for the composite databases through the addition of a 'Au_cut' field to the composites database.

14.5.2 WKP

Reconciliation history for the Waihi Project has demonstrated that some level of high-grade restriction is necessary to limit the influence of outliers on grade estimates for the epithermal veins that have been mined during the operations history. Statistical assessment of the input data is undertaken by domain, typical top-cut selection is based on the assessment of the population distribution characteristics and for Inverse Distance estimates cutting at the 98th percentile on the log probability distribution has been a long-standing methodology that has produced acceptable results. For the WKP deposit, MOP5 and the GOP the approach of cutting to the 98th percentile is considered appropriate.

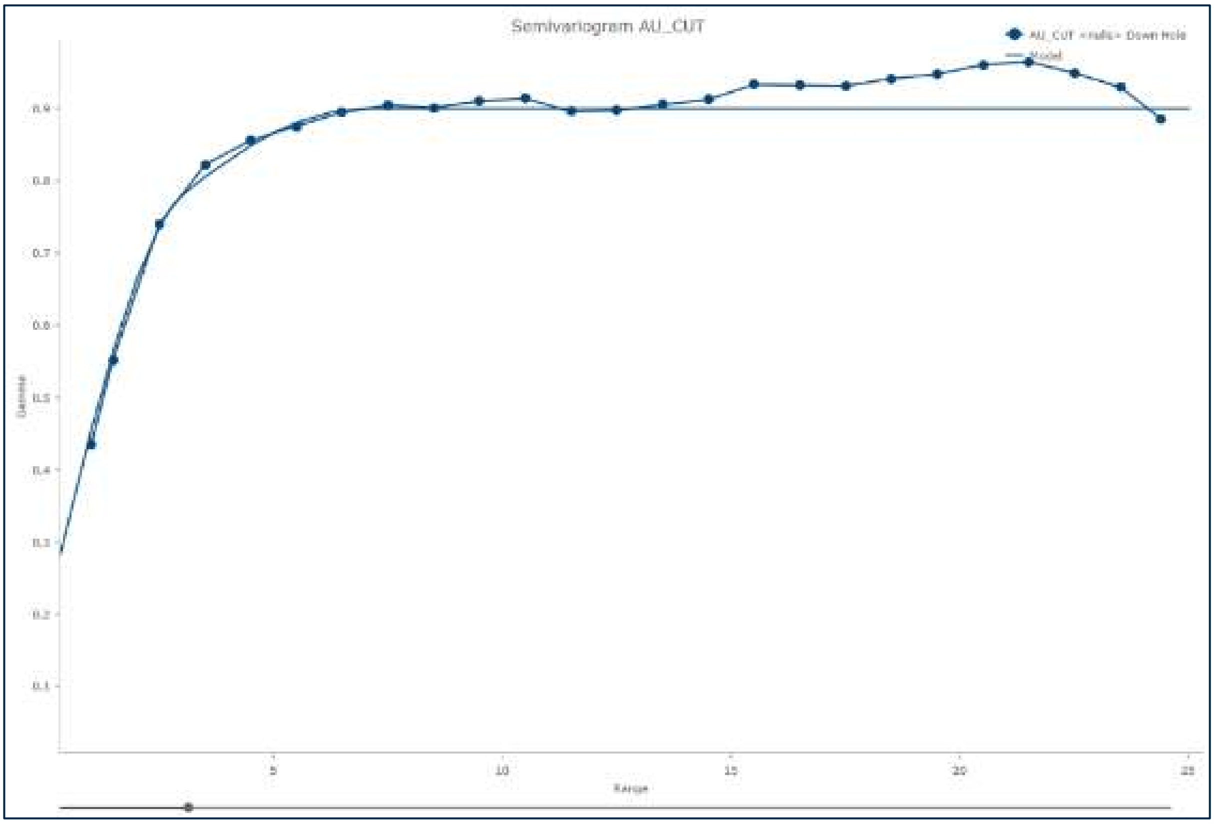
14.6 Variography

Downhole and directional variography are typically run using Snowden Supervisor v7 software. Variograms are run to test spatial continuity within the selected geological domains. Due to the scarcity of data, variogram models often are not easily obtained so in this instance anisotropic ratios are based on geological observation rather than on fitting data to the variogram models. Dominant mineral continuity is set along the strike of the modelled veins.

14.6.1 MUG

Variograms were modelled using Vulcans data analysis tools. In Waihi the generation of variograms is only successful on un-domained data, as the domaining process removes the variance necessary to model a robust variogram. Orientations of the omni-direction variogram are defined by the orientation of the vein. The MUG omnidirectional semi-variogram model is presented in Figure 14-6 below.

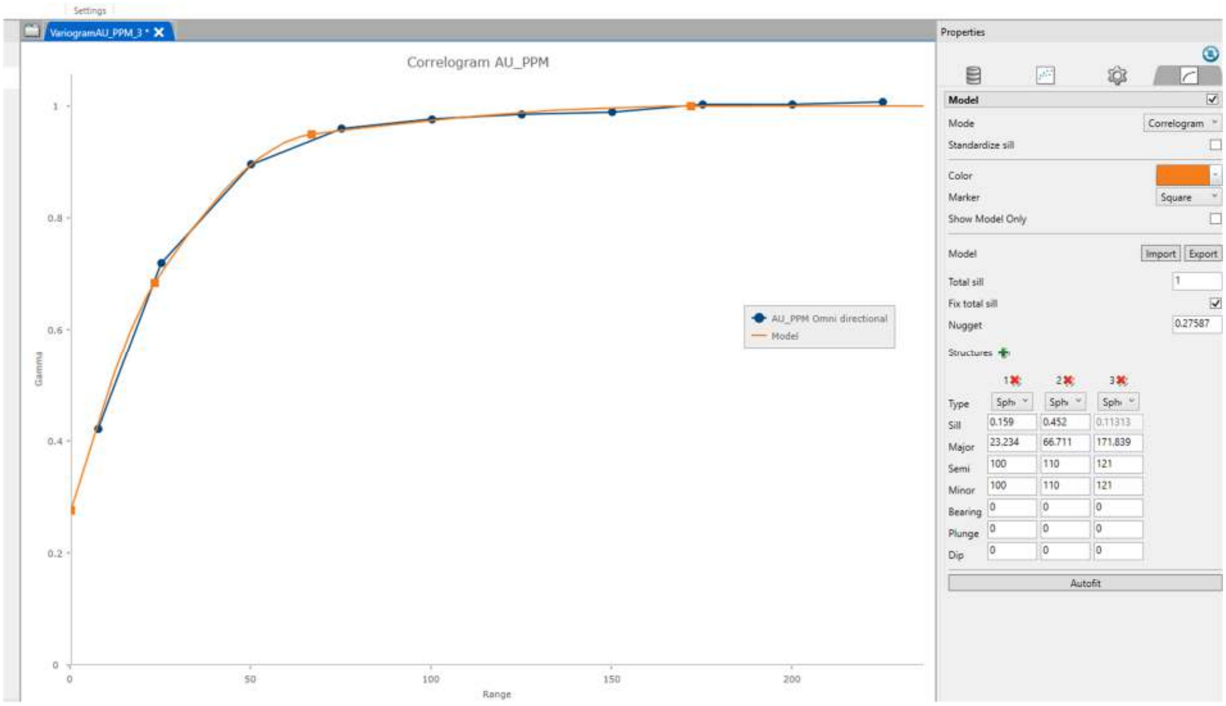
Figure 14-6: MUG Omni Directional Semi-Variogram for the DD Data



14.6.2 WKP

The best model of the variability for this project is the vein interpretation. Given the variable drilling density across the WKP Project area not considered appropriate to develop any form of kriged estimate at this time. A robust omni-direction variogram has been modelled based upon the un-domained drilling data as shown in Figure 14-7. Given the challenges faced in modelling variograms of domained data in a narrow vein setting there has been no attempt to generate a kriged estimate of grade for this deposit. This is not regarded as a risk to this project as the site has significant experience reconciling Inverse Distance grade estimates to mill production and consequently has an established estimation methodology that can be demonstrated to be appropriate for the epithermal veins encountered at the site.

Figure 14-7: WKP Omnidirectional Variogram - All Data - Un-Domained



14.6.3 GOP

GOP has been estimated utilising an Inverse Distance estimation method due to challenges faced in generating robust variograms for this deposit. This is again not regarded as a risk to this project as the site has significant experience reconciling Inverse Distance grade estimates to mill production.

14.7 Estimation / Interpolation Methods

The modelling process employed in the grade estimation for all the Waihi Projects is performed using numerous Vulcan and Leapfrog processes summarised in the steps outlined below:

1. Input data Validation;
2. Update lithological domains, geologic model construction;
3. Data selection, Drill hole data selection from the site AcQuire database;
4. Exclusion of unwanted drill holes by data type;
5. Flag data files by lithology;
6. Composite drill holes to fixed length composites within defined geological boundaries, typically 1 m using length weighting;
7. Exploratory data analysis by domain, generation of domain and data type summary statistics;
8. Variography;
9. Assign top caps by domain and data type to input data files;
10. Block Model construction based upon lithological wireframes;
11. Run estimation for all domains for Au, Ag, As, Resource Classification;
12. Assign density, mining depletions, back fill grade, stripping of negative values from non-estimated blocks, assignment of grade to dilution domains; and
13. Classify model.

The model is estimated in Vulcan. Estimations were performed in individual lithological domains using length weighted downhole composites.

Vulcan software version 12.0 has been used to construct the MUG, MOP5, WKP and GOP estimation models. MineSight software has historically been used to construct resource estimates for the MOP5 during past operations.

Sub-blocking with either OK or Inverse Distance weighting to the second power (ID2) is used for all underground models. Ordinary kriging in conjunction with tetra unfolding, has repeatedly produced outputs that are consistent with those achieved using ID2 and also produce acceptable reconciliation between resource and mill in the case of the underground projects that have been in production over the mine's recent history. The method of unfolding was adopted for the estimation of vein models as a way of dealing with the sinuous character of the veins.

The underground block models are rotated in bearing to align with the dominant strike of the veins and they are run using Vulcan software. Sub-blocking is used to define narrow veins and to maintain volume integrity with the geology solids. The grade estimation for all models is strictly controlled by the geology, with both sample selection and estimation of blocks limited to domains defined by the geology interpretation solids. Gold is estimated using one of the following methods either:

- a single pass with a combined channel and drilling dataset; or
- two-pass estimation using a combined dataset with short search range first.

Then be followed by a second pass using drillhole data only with longer search ranges to estimate blocks not estimated in the first pass.

14.7.1 MUG

The MUG block model dimensions, origin and cell size are provided in Table 14-8.

Table 14-8: MUG Block Model Dimensions

Variable	X	Y	Z
Origin	395200	642200	500
Extents (m)	1600	1200	700
Block Size (Parent)	10	10	10
No. of Blocks (Parent)	340	190	140
Sub-Block Size	1.0	1.0	1.0
Orientation	+65 degrees	X axis around Z	

For this model the vein domains were estimated using OK, tetra unfolding was employed for all domains to improve estimation locally in areas with complex vein geometries and to aid in resolution of the sample selection for the estimation.

Models are created using a standard block variable schema to enable capture of all relevant grade fields, Resource Classification evaluation data and geologic information. Parent block model variables captured are presented in Table 14-9. Mining evaluations are performed on a stripped-down version of the parent model, with all non-essential variables removed from the engineering model edition to assist in processing requirements.

Dilution domains where created based on a 5 m halo around the veins and grades where assigned into these domains using mean grades for the dilutant domain.

An octant search was applied to all domains. Example estimation parameters used for the major domains are presented in

Table 14-10.

Table 14-9: List of Fields in the MUG Model

Model Field	Type	Default Value	Description
code	Short (Integer * 2)	-99.0	vein code
sg	Float (Real * 4)	2.5	density value =2.5
rescat	Byte (Integer * 1)	4	4=MI (Mineral Inventory); 3=Inferred; 2=Indicated; 1=measured
rescat_nsamps	Short (Integer * 2)	0	Resource Classification # of samples
rescat_avedist	Float (Real * 4)	-99	Resource Classification average distance
rescat_nholes	Short (Integer * 2)	-99.0	Resource Classification # of holes
rescat_id	Float (Real * 4)	4	
mined	Byte (Integer * 1)	0	historic workings and subsidence
hdns	Byte (Integer * 1)	3	hardness code for pit
pit	Byte (Integer * 1)	99	pit phase
oxide	Byte (Integer * 1)	2	oxide surface
est_id	Integer (Integer * 4)	-99.0	estimation id
au_id_nsamps	Short (Integer * 2)	-99.0	number of samples used in id2 estimate
au_id_nholes	Short (Integer * 2)	-99.0	No of drillholes used to calculate block grade id2
au_id_avedist	Float (Real * 4)	-99.0	average distance to samples
au_id_ndist	Float (Real * 4)	-99.0	distance to nearest sample
au_nn_c1	Float (Real * 4)	-99.0	nearest neighbour estimate cut
au_nn_ndist	Short (Integer * 2)	-99.0	distance to nearest sample nearest neighbour
au_pref	Float (Real * 4)	-99.0	preferred au
au_ok_nholes	Short (Integer * 2)	-99.0	No of drillholes used to calculate block grade id2
au_ok_nsamps	Short (Integer * 2)	-99.0	No of samples
au_ok_avedist	Float (Real * 4)	-99.0	average distance to samples
au_ok_ndist	Float (Real * 4)	-99.0	distance to nearest sample
au_ok_k_var	Integer (Integer * 4)	-99.0	kriging variance
ag	Float (Real * 4)	-99.0	
as	Float (Real * 4)	-99.0	
geol	Name (Translation Table)	none	

Table 14-10: MUG Estimation Parameters used in Estimate – Major Veins

	1100	1201	1220	1221	1302	1305	1308	1400
Major Axis X	120	120	120	120	120	120	120	120
Semi-major Z	100	100	100	100	100	100	100	100
Minor - X (1)	1	1	1	1	1	1	1	1
Bearing	235	200	200	25	235	50	250	235
Plunge	0	0	0	0	0	0	0	0
Dip	-75	-75	82	50	-83	-88	-70	80
Discretisation XYZ	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3
min Samp	6	6	6	6	6	6	6	6
Max Samp	18	18	18	18	18	18	18	18
Samp per DH	3	3	3	3	3	3	3	3
Max samp/octant	9	9	9	9	9	9	9	9
High-Grade Restraining	na	na	na	na	na	na	na	na
Method	OK	OK	OK	OK	OK	OK	OK	OK
Tetra Model	1100.tetra	1201.tetra	1220.tetra	1221.tetra	1302.tetra	1305.tetra	1308.tetra	1400.tetra

(1) Tetra unfolding ranges for the across strike range are expressed as a relative proportion of the vein thickness

14.7.2 WKP

For the WKP deposit the grade was estimated into the sub-celled blocks, future estimates will utilise parent cells of 10 m x 10 m x 10 m, with minimum sub-block dimensions of 0.5 m in each direction.

The raw assays are composited to one metre fixed lengths and “distributed” (1MD) across the vein width to eliminate very small remnant composites. The distributed method divides the vein interval into several equal length samples as close to the desired sample composite length as possible given the intercept width, this is an option available in the Vulcan software.

The general approach to estimation for WKP is consistent with approaches used for other epithermal deposits in the Waihi Project area. Veins for the WKP Underground model were interpreted using Leapfrog software. Vein and geology wireframes were then utilised to construct a block model within Vulcan. Compositing of data for grade estimation is within distinct geological boundaries. For this model the vein domains were estimated using Inverse Distance estimation techniques. All domains are estimated using hard geologic boundaries, tera unfolding of the search ellipse and estimation parameters that have been calibrated based on the long operating history of the Waihi Project.

The WKP block model is rotated in bearing to align with the dominant strike of the veins. Sub-blocking is used to define narrow veins and to maintain volume integrity with the geology solids.

The grade estimation for all models is strictly controlled by the geology, with both sample selection and estimation of blocks limited to domains defined by the geological interpretation solids. Gold is estimated using a single estimation pass. The specific details for the WKP resource estimate such as block model dimensions, origin and cell size are provided in the tables below.

Table 14-11: WKP Block Model Dimensions

Variable	X	Y	Z
Origin	2759700	6429325	-345
Extents (m)	900	1000	620
Block Size (Parent)	10	10	10
No. of Blocks (Parent)	280	164	62
Sub-Block Size	0.5	0.5	0.5
Orientation	+100 degrees	X axis around Z	

Table 14-12: List of Fields in WKP Model

Model Field	Type	Default Value	Description
code	Integer (Integer * 4)	0	vein code
sg	Float (Real * 4)	2.5	density value = 2.5
au_id	Float (Real * 4)	-99.0	au ID2 estimated value cut
ag_id	Float (Real * 4)	-99.0	ag OK estimated value cut
au_id_u	Float (Real * 4)	-99.0	au ID2 estimated value uncut
res_cat	Byte (Integer * 1)	4	classification 2= Ind; 3 = Inf, 4=mineral inventory
au_samps_id	Integer (Integer * 4)	-99.0	number of samples used in estimate ID2
au_dist_id	Float (Real * 4)	-99.0	Weighted average distance of samples for ID2
rescat_id	Float (Real * 4)	-99.0	
rescat_avedist	Float (Real * 4)	-99.0	
rescat_nsamp	Float (Real * 4)	-99.0	
rescat_nholes	Float (Real * 4)	-99.0	
au_id_nn	Float (Real * 4)	-99.0	nearest neighbour Au

14.7.3 GOP

The GOP deposit is estimated in Vulcan, the model is constructed using a regularised block size of 2.5 m in all dimensions. The grade is estimated into the regularised blocks with Inverse Distance interpolation. Estimations were performed in individual lithological domains using 1 m length weighted downhole composites for all vein-based domains and 2m length weighted composites for the lithological domains, (Andesite and handing wall breccia).

The raw assays are composited to one metre fixed lengths and "distributed" (1MD) across the vein width to eliminate very small remnant composites, a separate compositing file is generated using a 2 m composite length for estimation of the lithological domains. Vein boundaries are treated as hard contacts in compositing, model construction and in grade estimation. The distributed method divides the vein interval into several equal length samples as close to the desired sample composite length as possible given the intercept width, an option available in the Vulcan software.

14.7.4 MOP5

MOP5 deposit is estimated in Vulcan, the model is constructed using a sub-blocked model of 2.5 m dimensions in all directions. The model is however, regularised into a 5 m cell size prior to pit optimisation assessment. The grade is estimated using OK and parent cell estimation.

Estimations were performed in individual vein domains using 1.5 m length weighted downhole composites for all vein-based domains and 3 m length weighted composites for the lithological (bulk) domains.

The raw assays are composited to 1.5 m and 3 m fixed lengths and "distributed" (1MD) across the vein width to eliminate very small remnant composites. Vein boundaries are treated as hard contacts in compositing, model construction and in grade estimation. The distributed method divides the vein interval into several equal length samples as close to the desired sample composite length as possible given the intercept width, this is an option available in the Vulcan software.

Table 14-13: MOP5 Block Model Dimensions

Variable	X	Y	Z
Origin	395150	642330	500
Extents (m)	1700	950	700
Block Size (Parent)	5	5	5
Sub-Block Size	1.25	1.2	1.25
Orientation	+65 degrees	X axis around Z	

Table 14-14: MOP5 Estimation Parameters used in Estimate - Major Veins

	1100	1201	1220	1221	1302	1305	1308	1400
Major Axis X	120	120	120	120	120	120	120	120
Semi-major Z	100	100	100	100	100	100	100	100
Minor - X (1)	1	1	1	1	1	1	1	1
Bearing	56	18	25	28	55	55	70	57
Plunge	0	0	0	0	0	0	0	0
Dip	-75	-85	82	50	-83	-88	-70	80
Discretisation XYZ	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3	3,2,3
min Samp	4	4	4	4	4	4	4	4
Max Samp	16	16	16	16	16	16	16	16
Samp per DH	3	3	3	3	3	3	3	3
Max samp/octant	6	6	6	6	6	6	6	6
High-Grade Restraining	na	na	na	na	na	na	na	na
Method	OK	OK	OK	OK	OK	OK	OK	OK
Tetra Model	1100.tetra	1201.tetra	1220.tetra	1221.tetra	1302.tetra	1305.tetra	1308.tetra	1400.tetra

(1) Tetra unfolding ranges for the across strike range are expressed as a relative proportion of the vein thickness

14.8 Classification of Mineral Resources

The Resource Classification is based on an assessment of average drilling density.

There is significant experience in mining and assessing the continuity of mineralisation in the Waihi District epithermal vein setting. This vein style mineralisation has a strong visual control, is well understood and has demonstrated continuity over significant ranges.

An estimation run is undertaken utilising the three closest drill holes intersecting the domain of interest, the average distance to the three closest drill holes used to estimate the block is then stored to form the basis for classification.

The MUG Project uses an average spacing to three drill holes of 60 m for Inferred and 40 m for Indicated Resources. Any mineralised backfill is not classified or inclusion within the Mineral Resource due to uncertainty in both the continuity and the distribution of grade within the back filled stopes.

The East Graben Vein zone of the WKP Project has been intersected in drilling over a strike length of ~1 km, this structure is larger than those typically encountered in the Waihi Project area and on this basis the average drill hole spacing required for classification as an Inferred Resource on the EG vein structure has been increased to 70 m average distance to the three closest drill holes. An average drill spacing of three holes within 50 m was used as the basis for classification as Indicated Resource for the EG structure. All other WKP mineralisation has been classified using a distance threshold of 60 m to the three closest drill holes for classification as Inferred.

For MOP5 an average drill hole spacing of 60 m to the three closest drillholes on the major mineralised veins for classification as Inferred and a spacing of 35 m for classification as Indicated Resource. A tighter spacing of 22.5 m has been implemented for classification as Indicated Resource for the non-vein-based domains, typically these are more complicated zones exhibiting strong brecciation and/or stockwork veining. As with MUG any mineralised backfill is not classified or inclusion within the Mineral Resource due to uncertainty.

The Gladstone deposit is classified using an average drill hole spacing of 60 m to the three closest drillholes on the major mineralised veins for classification as Inferred Resource and a spacing of 35 m for classification as Indicated Resource. A tighter spacing of 22.5 m has been implemented for classification as Indicated Resource for the non-vein-based domains, typically these are more complicated zones exhibiting strong brecciation and/or stockwork veining.

The ranges are determined based on the observed continuity of the veining for a given project

Table 14-15: Classification Criteria

Project	Drill Spacing for Measured Resource	Drill Spacing for Indicated Resource	Drill Spacing for Inferred Resource
Martha Open Pit	20 metres	50 metres	100 metres
Gladstone Open Pit	15 metres	30 metres	52.5 metres
Martha Underground	20 metres	40 metres	60 metres
WKP	15 metres	40 metres	80 metres

The classification of Open pit Mineral Resources takes into account geologic, mining, processing and economic considerations, and have been confined within appropriate LG pit shells, and therefore are classified in accordance with the 2014 CIM Definition Standards for Mineral Resources.

For underground the Measured material is classified on the basis of proximity to drilling and sill drive development. Blocks are classified as Measured if they are within an average distance of 10 metres of three separate sampled locations, either drill holes or lateral ore drive development channel sample locations

The resource estimate outlined in this document appropriately reflects the Competent Person's view of the deposit.

14.9 Block Model Validation

14.9.1 MUG

Numerous methods have been used to validate the r1120 MUG resource model, including the following:

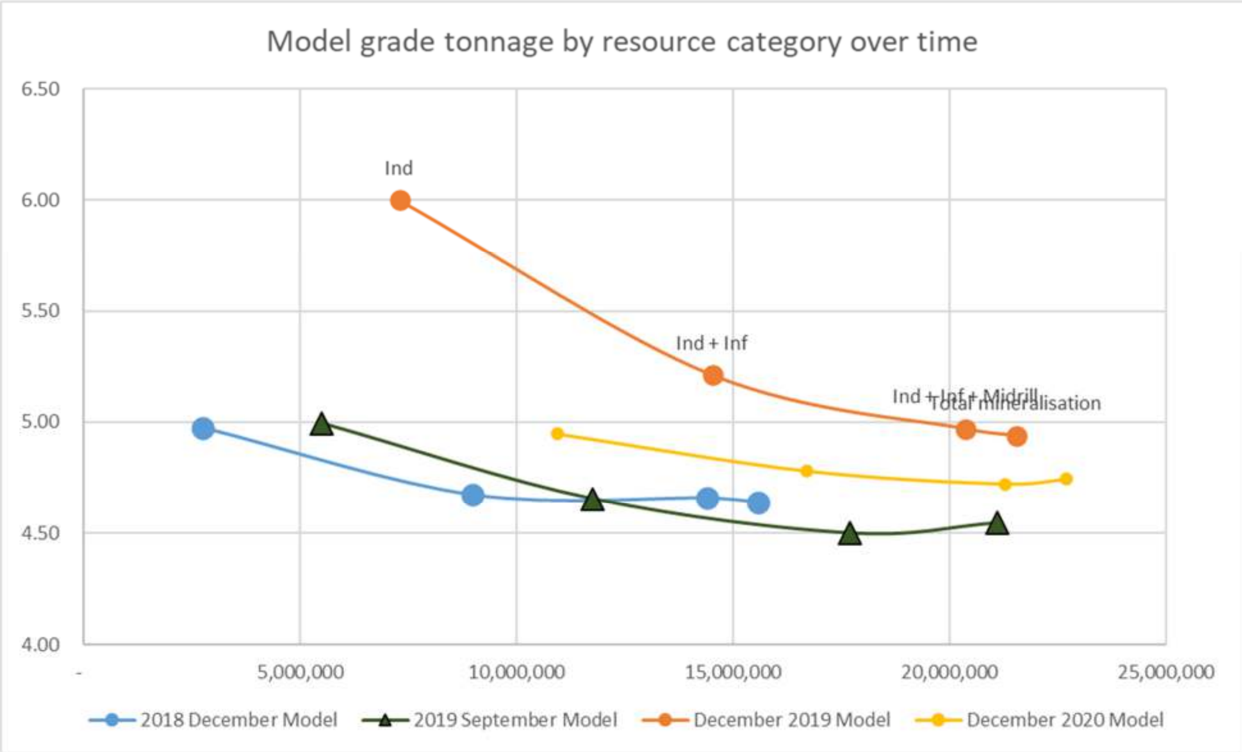
- Validation of the new data;

- A review of the interpretation, including classification shapes;
- A review of the methodology;
- A review of the Exploration Data Analysis (EDA) work, including variography and search neighbourhoods;
- Global grade and tonnage comparisons with the previous model;
- A visual sectional validation of the block model with interpretation and drilling;
- A comparison of tonnes and grade of the LOM shapes and upcoming pit/stope designs will be provided upon completion of preliminary design work; and
- Swath plots are generated, blocks selected for comparative evaluation relative to the samples database are limited to blocks estimated in the first pass, blocks below the Phase 4 consent pit and exclude historic mined blocks. Axis values are relative to the model origin, 395150 x, 642330 y and 500 z. The variance in observed output for the drift analysis is a result of not estimating blocks within the depleted portion of the open pit and thereby restricting the block the model data.

Comparative assessment of the model relative to previous estimates again indicates only modest changes between estimates in the total in-situ resource. The influence of the infill drilling undertaken in 2020 has resulted in a substantial increase in Indicated Resource confidence category material as shown in Figure 14-8.

The model has been compared to several previous models. The grade tonnage curve (Figure 14-8) provides a view of the increase in confidence of the in-situ mineralisation as a consequence of drilling completed into the project in 2020. The increase in grade observed for the Indicated component of the resource in the December 2019 model is a function of the conversion of the high-grade Rex vein to Indicated status in December 2019. This is prior to an assessment of the prospect of economic extraction and is therefore, illustrative of the increase in confidence of the insitu material contained within the model only.

Figure 14-8: Relative Confidence for In-situ Mineralisation by Model Iteration



14.9.2 WKP

Numerous methods have been used to validate the WKP0120 resource estimate, including the following:

- Validation of the new data;
- A review of the interpretation, including classification shapes;
- A review of the methodology;
- A review of the EDA work, including variography and search neighbourhoods;
- Global grade and tonnage comparisons with the previous models; and
- A visual sectional validation of the block model with interpretation and drilling.

Drilling during 2019 at the WKP Project focused on infill drilling on the primary EG Vein structure. As seen in the global grade tonnage curve, Figure 14-9, there has been relatively little change in the overall mineralisation defined for this project. When confidence in the resource is considered (Figure 14-10) , however, it is apparent that there has been a significant upgrade in resource confidence on the back of the work completed on this project in 2019.

Figure 14-9: In-situ Grade Tonnage Curves - All Material - December 2019 vs February 2019

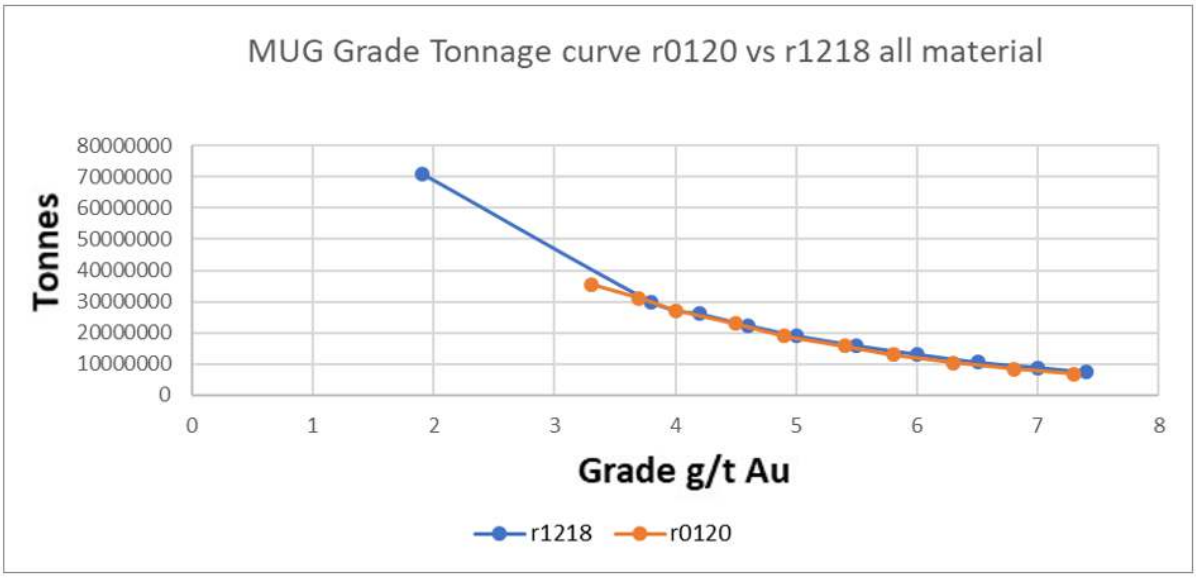
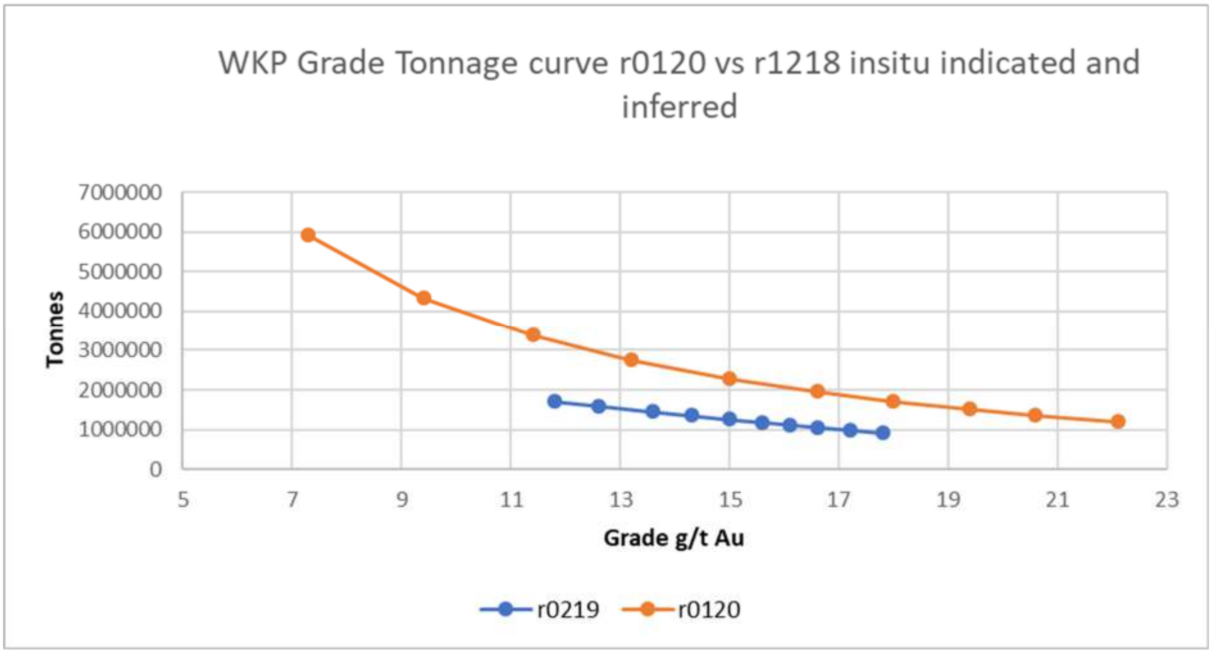


Figure 14-10: In-situ Grade Tonnage Curves - Indicated and Inferred Combined - December 2019 vs February 2019



14.9.3 Grade Control and Definition Strategies

Ongoing resource definition drilling is planned to be undertaken throughout 2021 and beyond for all projects on an as required and as appropriate basis.

Ongoing drilling is being undertaken at MUG. This drilling is planned as a combination of infill and extensional resource delineation. There is an integrated planning process that involves assessment of resource risk, economic considerations and mine development sequencing to ensure areas with elevated resource risk are identified and de-risked through drilling in a sustained and appropriate manner.

Grade Control strategies for MUG will be consistent with those used for previous underground mine developments in Waihi including:

- Monitoring and identification of local high-risk geological area, infill DD of these risk areas as required; and
- In-cycle mine development monitoring, mapping, sampling, geologic data capture and model development on individual lodes as they are developed to ensure appropriate risk management.

Planned WKP drilling has been impacted in 2021 however, there are planned extension and infill drilling activities for 2021 and beyond. Grade control activity will be undertaken via infill drilling upon attaining underground access to the project area. In-cycle grade control processes will continue to be similar to those used for other underground mine projects developed at Waihi.

Open pit grade control practices will be based on protocols used previously at Waihi with RC drill sampling of open pit deposits at appropriate drill spacing to guide ore selection at a mining scale.

14.10 Cut –off Grade Estimates

All Mineral Resource cut-off grades have been estimated using a long-term gold price of USD1,700 and exchange rate of USD 0.71:NZD (NZD2,394/oz.) and silver price of USD17/oz. as advised by OGC. Estimated cut-off grades for the various Resources are shown below in Table 14-16.

Table 14-16: Resource Cut-off Grade Estimates

	Price NZD/oz.	Metal Recovery	Process Cost NZD/t	G&A NZD/t	Sustain Capex NZD/t	Mining NZD/t	Royalty %	Cut-off Grade g/t
MOP5	2,394	90%	28	10		-4	2%	0.50
MUG	2,394	94%	33	20	9	90	2%	2.15
GOP	2,394	74%	25	10		-4	2%	0.56
WKP	2,394	90%	33	10	8	115	4%	2.50

14.10.1 Martha Open Pit

The cut-off grade used to determine the Mineral resource was 0.5g/t Au. Inputs to the calculation of cut-off grades for MOP5 include mining costs, metallurgical recoveries, treatment and refining costs, general and administrative costs, royalties. A metallurgical recovery of 90% been used for the mill feed cut-off grade calculation.

No mining dilution or mining recovery factors have been applied to the geological model due to:

- The mineralised zones are broad on each mining bench, and the overall dilution edge effects are minimal, with the result that there is little difference between the overall in-situ and diluted tonnes and grade;
- The Mineral Resource block model has a block dimension which is larger than the optimum selective mining unit (SMU), and
- Historical reconciliation between the mill, grade control and exploration data has shown good correlation over the last 30 years.

14.10.2 Martha Underground

A cut-off grade of 2.15g/t has been used for the Martha underground Mineral Resource. Cut-off grades are based on projected processing costs of NZD 33/tonne, general and administration costs of NZD 20/tonne, mining costs of NZD 90/tonne. Additional sustaining capital costs of NZD 9/tonne have been allowed for to cater for fixed and mobile plant.

14.10.3 Gladstone Open Pit

Inputs to the calculation of cut-off grades for Gladstone Open Pit include mining costs, metallurgical recoveries, treatment and refining costs, royalties. A metallurgical recovery of 74% been used for the mill feed cut-off grade calculation.

The cut-off grade used to determine the mill feed for the GOP was 0.56 g/t Au based on an empty mill and a waste rock disposal credit (i.e. less cost to haul to process plant than place and compact in waste rock stacks). This is an adjustment over the 2019 Mineral Resource estimate.

No mining losses were applied. It is considered that the resource estimation technique applied to the broad Mineral Resource zones provides an adequate estimate of the mill feed tonnes and grades.

14.10.4 WKP

The WKP Resource is calculated above a cut-off grade of 2.5 g/t Au. Parameters used to calculate the cut-off grade were derived from the nearby Waihi operation with additional costs allowed for surface and underground haulage of the Resource to the Waihi process plant. WKP attracts an additional third party royalty.

14.11 Mining Factors or Assumptions

14.11.1 Martha Pit

The MOP5 cutback was developed from a Whittle optimisation carried out in 2016 and further validated in 2017. Inputs comprised a maximum 7 Mt per annum operation and 1.5 Mt per annum processing throughput. Open pit slopes were generated for separate rectangular sub-regions based on different rock units calibrated with existing pit slopes and with allowance for haul roads. Processing and administration costs were estimated from the existing Waihi Operation. Mining costs were based on actual mining costs from 2006 to 2007 when the Martha Pit was operating at moderate production rates escalated by the Consumer Price Index (CPI).

The Whittle optimisation and the optimum pit selected considered the proximity of the pit to the Waihi township, social and environmental constraints and the need for high geotechnical factors of safety and limits on encroachment.

The design slopes for the MOP5 cutback are shown in Table 14-17. Berm intervals are generally 20 m below 1090 m RL and 15 m above 1090 m RL. In the past slopes to the south and south-west have been flatter due to effect of historic workings on the rock mass quality, the proximity of the town and presence of argillic andesite. Slopes to the east are the shallowest slopes due to presence of the post-mineral sediments comprising tuffs and alluvial layers as well as a weaker andesite unit.

Table 14-17: MOP5 Pit Slope Design Criteria

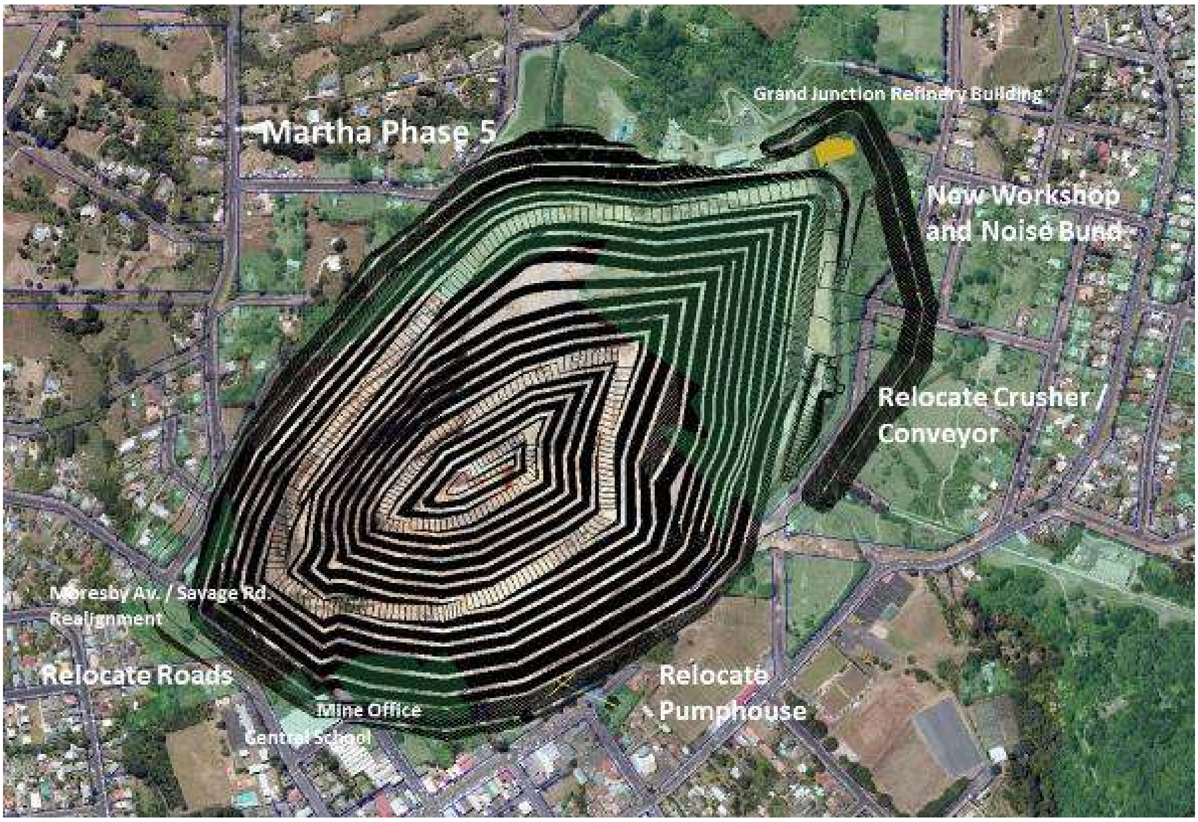
Bench (m RL)	Berm width	Face Height	South / West Walls	North-West Walls	North-East / South-East Walls	East Wall
1135 to 1150	5	15			35	
1120 to 1135	5	15	25		35	
1103.5 to 1120	5	16	30	35	35	30
1090 to 1103.5	7	14	45	55	60	35
1070 to 1090	7	20	45	65	60	30
1050 to 1070	7	20	50	65	55	45
1030 to 1050	7	20	55	65	55	55
1010 to 1030	7	20	55	70	55	45
990 to 1010	7	20	55	70	55	50
970 to 990	7	20	55	70	55	50
950 to 970	7	20	55	70	55	50
930 to 950	7	20	55	70	55	55
910 to 930	7	20	60	70	55	60
Below 890	7	20	60	70	60	60

The pit encroaches towards the town centre, residential and low-density residential zones, and for this reason a plan change to the Hauraki District Plan will be required to provide for this component of the project. Key points of note are:

- A constraint has been placed on the cutback of the western pit wall so as not to encroach into the Moresby Avenue reserve and the Central School grounds;
- Mining provides rock to backfill the underground mine and to construct tailings storage facilities;
- Operations require relocation of the existing crusher and belt conveyors from the existing crusher slot to a new crusher slot, 70 m to the east and installation of a new crushing facility;
- Relocation and enlargement of the noise bund beyond Grey Street into Slevin Park and construction of a noise bund along the remaining MOP5 pit rim is required;
- Relocation of the historical Cornish pumphouse.
- Partial realignment of the Eastern Stream is required; and
- Relocation and re-establishment of the open pit office block, fuel bowser, substation, workshop, wheel wash and magazine are required.

The pit configuration is shown in Figure 14-11.

Figure 14-11: Plan of Martha Phase 5 Pit



The final dimensions of MOP5 are:

- Pit area - approximately 66 Ha;
- Pit depth - approximately 316 m;
- Pit floor level - approximately 840 m RL;
- Pit length x breadth - approximately 1,115 x 830 m; and
- Total volume - approximately 59 Mm^3.

14.11.2 Martha Underground

Mining method selection work for the Martha underground project was undertaken by SRK in 2011, 2016 and 2017 and confirmed by Entech Pty Ltd in 2020. The Mineral Resource estimate has applied the same recommended mining methods recommended by SRK and Entech. Much of the deposit can be extracted using Avoca which has been the predominant mining method at Waihi since 2004. A proportion of the Mineral Resource inventory will involve the extraction of remnant ore skins in the footwall or hanging wall of previously mined stopes, or the extraction of both remnant ore skins and historical backfill.

No mining recovery or dilution were applied to the Mineral Resource estimate. OceanaGold has estimated the Mineral Resource using the Deswik Stope Optimiser (SO). The Mineral Resource is reported within the SO shapes above the 2.15 g/t cut-off grade.

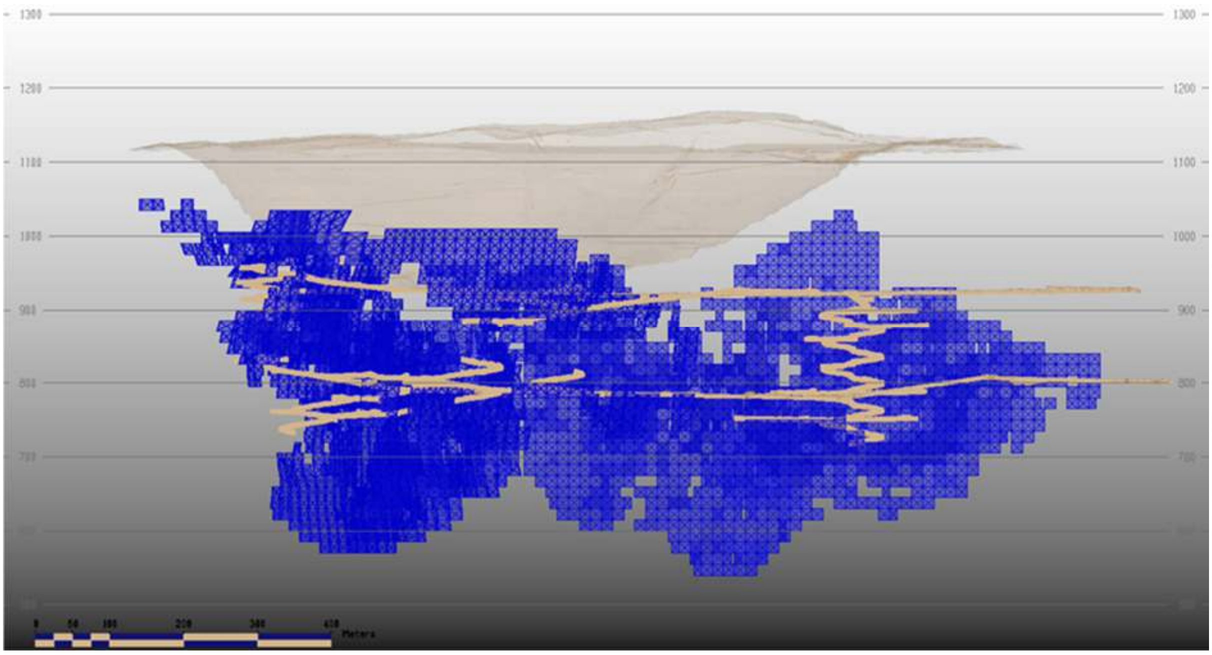
No unclassified material contained within the SO shapes is reported. Nominal stope dimensions of 15 m high by 10 m in length were selected for the design. Stope widths vary, depending on the thickness of the mineralisation. A minimum stope width of 0.5 m was used and 0.5 m of dilution was applied to both the footwall and hanging wall resulting in a minimum stope width of 1.5 m . A maximum stope width of 15 m was used with a minimum pillar width between stopes of 8 m. A maximum percentage of historical stoping of 10% was allowed in each SO shape.

The method of specifying the strike and dip angles for the initial stope-seed-shapes in SO was to apply a stope control surface wireframe over the full extent of the orebody where stope shapes are to be generated. The following stope shapes were manually excluded from the Mineral Resource estimate:

- Isolated stope shapes either showing lack of continuity or distant from the main concentrations of shapes.
- Stopes closer than 50 m from the surface.
- Within a solid created as an exclusion solid around the historical "Milking Cow" zone by projecting the cave zone outwards by 20 m.
- All stopes intersecting the base of the Martha Phase 5 pit.

Figure 14.12 presents the SO shapes after exclusion based on geotechnical and economic assessment.

Figure 14.12: Martha Underground Mineral Resource Long Section



14.11.3 Gladstone Pit

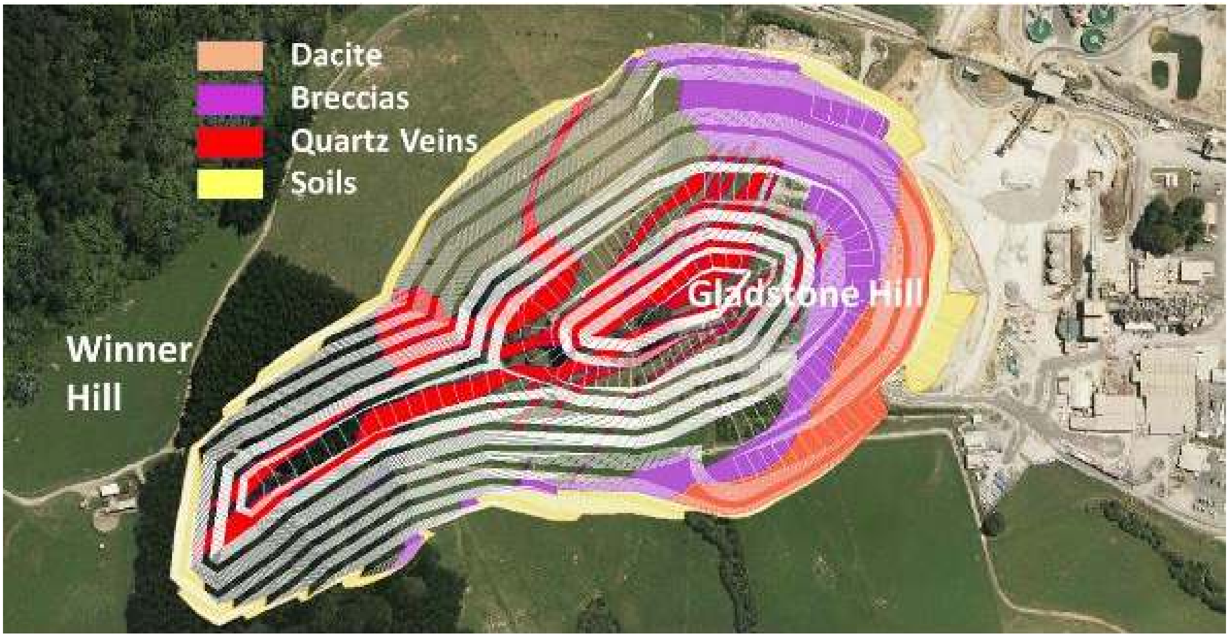
Geotechnical studies during 2017 on preliminary design concepts including geotechnical drilling, rock/ soil testing and detailed core logging showed that the slopes in the Winner Hill pit and the northern slopes in Gladstone Hill were generally satisfactory under fully saturated or partially drained conditions. However, the southern and eastern upper slopes were shown to be marginally stable under fully or partially saturated conditions particularly where there was a significant depth of the surficial deposits.

The geological model shows the north-western wall will comprise andesite, overlain by a thin band of hydrothermal breccia and a relatively thin sheet of rhyolitic tuff/ignimbrite thickening to the south. The south-eastern wall has a thicker band of rhyolitic tuff/ignimbrite and hydrothermal breccia overlying andesite; and the east wall has the greatest thicknesses of dacite and volcaniclastics. Design pit slopes were modified based on a geotechnical study completed by PSM in early 2018 including three additional geotechnical holes and geotechnical modelling to achieve satisfactory factors of safety. Geotechnical domains were re-defined based on drilling. The design criteria used to support calculation of Mineral Resource are reported in Table 14-18 and the geotechnical domains are presented in Figure 14-13.

Table 14-18: GOP Pit Slopes

Pit Design Parameter	Bench Height (m)	Face Slope (degrees)	Berm Width (m)
GOP Pit			
1040 to 1100 m RL	15	60	5
1100 to 1140 m RL	10	40	5
Breccias	10	40	5
Surface to 6 m depth		35	
Haul Road Width	20 m wide @1 in 10, surface to 1070, 12 m wide @ 1 in 9 to 1040		
Goodbye Cut		5 m deep	
Winner Pit			
1060 to 1085 m RL	15	60	5
1085 to 1100 m RL	15	55	5
1100 to 1130 m RL	10	55	5
Surface to 8 m depth		30	
Haul Road Width		18 m wide 1 in 10	
Goodbye Cut		None	

Figure 14-13: GOP Pit Geotechnical Domains



14.11.4 WKP

SRK assessed the geotechnical data to establish the geotechnical characteristics and conceptual design elements for the underground mine. The assessment entailed:

- Understanding the geological setting of the gold deposit;
- Creation and population of an interpretable geotechnical property database based on the limited geotechnical core logging available;
- Collection and recording of suitable core samples for rock property testing in a laboratory, supported by field estimates (point loads) of rock strengths;
- Graphical representation, interpretation and reporting of recorded data, culminating that describes the geotechnical environment, and
- Transformation of data into Barton's Q' value.

SRK recommended that the hydraulic radii shown in Table 14-19 be used for initial stope sizing by area and depth.

Table 14-19: Preliminary WKP geotechnical Parameters

	Eastern Graben EG Rhyolite		Central Area Lapilli Tuff		Western T stream Rhyolite	
	HR min	HR max	HR min	HR max	HR min	HR max
80-160 m	5.5	5.5	5.1	5.1	6.8	6.8
160-240 m	4.8	5.5	4.5	5.1	6.8	6.8
260-320 m	4.2	5.5	4.0	5.1	6.7	6.8

Mining method selection work for the WKP Project was undertaken by SRK in 2019, SRK state both pillar and artificially supported methods are suitable for the WKP deposit. The deposit will not be able to be supplied an engineered fill such as paste or cemented hydraulic fill because the location of the processing plant is 10 km distance from the mine. Backfill for the mine could be either cemented rock fill or rock fill.

The use of in-situ pillars was not considered by SRK due to the high-grade of the Mineral Resource, as such if pillars are required these could be cemented fill rather than in-situ pillars. The existing OceanaGold operation Waihi use the Avoca mining method and SRK considers that Avoca mining method is also suitable for WKP. SRK recommended a sub-level height of 20 m and stope strike length of 15 m be adopted for stope optimisation which is within the preliminary geotechnical parameters with a HR of 4.3.

OceanaGold has estimated the Mineral Resource using the Deswik® Stope Optimiser (SO). The Mineral Resource is reported within the SO shapes above the 2.5 g/t cut-off grade. No unclassified material contained within the SO shapes is reported. Nominal stope dimensions of 15 m high by 15 m in length were selected for the SO. Stope widths vary, depending on the thickness of the mineralisation. A minimum mining width of 0.5 m was used and 0.5 m of dilution was applied to both the footwall and hangingwall resulting in a minimum stope width of 1.5 m . A maximum stope width of 15 m was used with a minimum pillar width between stopes of 8 m. The method of specifying the strike and dip angles for the initial stope-seed-shapes in SO was to apply a stope control surface wireframe over the full extent of the orebody where stope shapes are to be generated. All shapes within 50 m of the surface topography were excluded from the estimate. Figure 14.14 and Figure 14.15 present the SO shapes. No mining recovery or dilution were applied to the Mineral Resource estimate.

Figure 14.14: WKP Mineral Resource Long Section

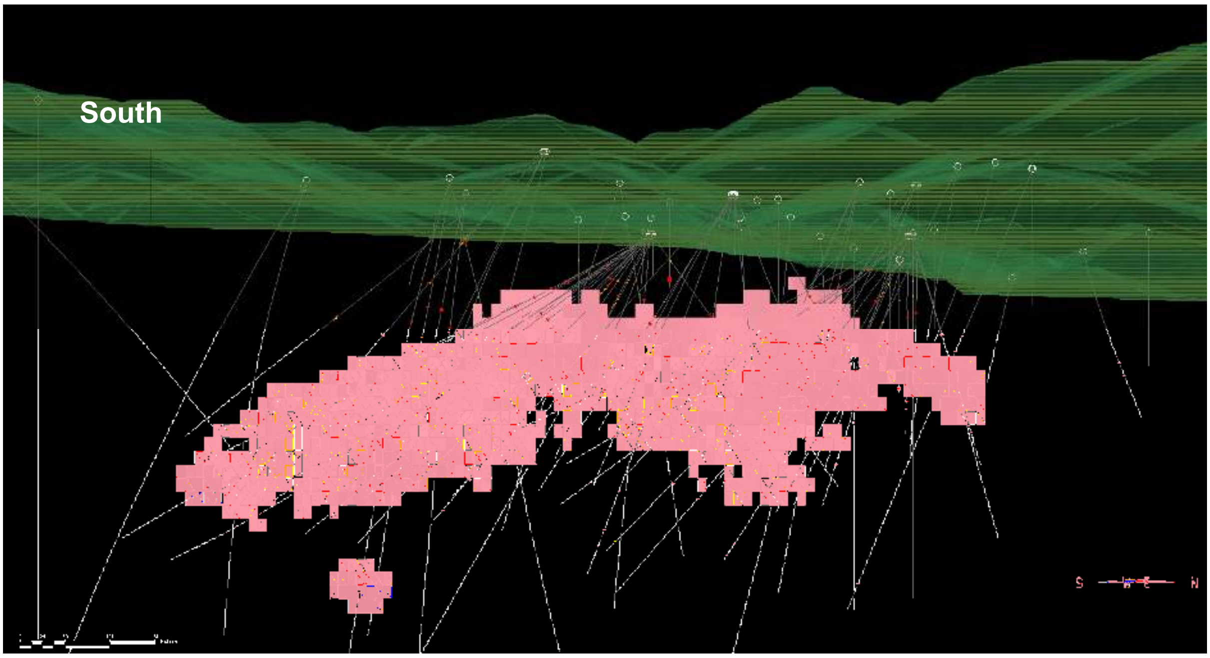
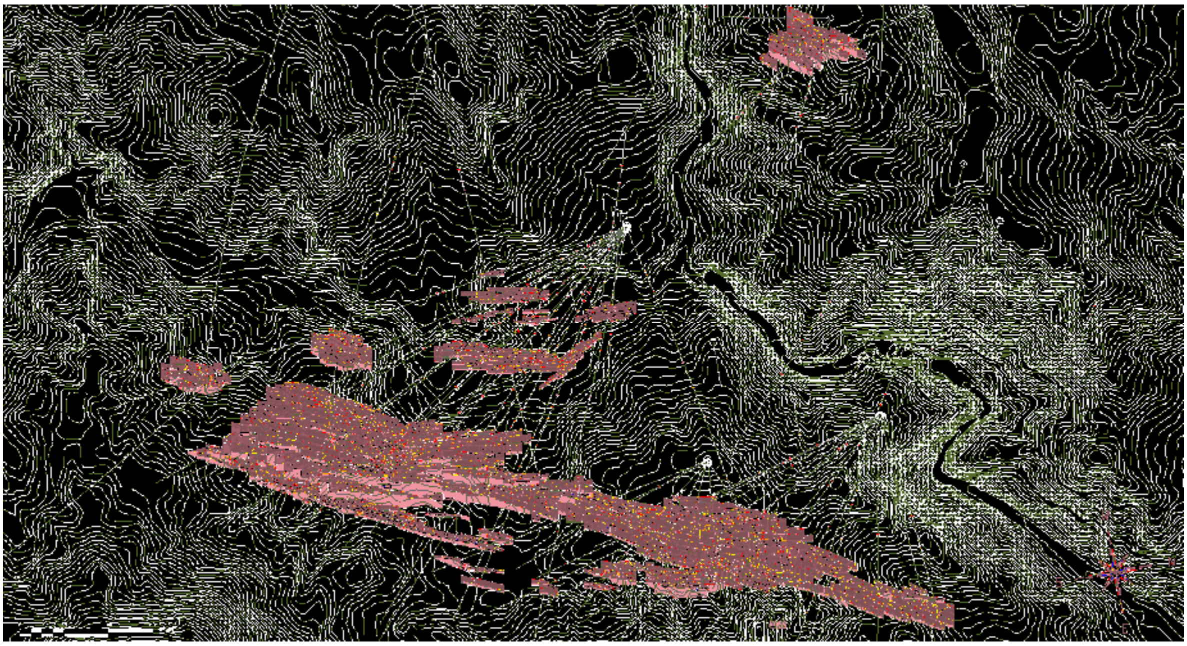


Figure 14.15: WKP Mineral Resource Plan View



14.12 Resource Estimate

The OceanaGold Waihi resource estimates, as at 31 December 2020, are presented in Tables 14-25, 14-26, and 14-27 and are classified in accordance with CIM and JORC 2012. Mineral Resources are inclusive of Mineral Reserves and are reported at a commodity price of NZ$2,394/oz gold.

The resource estimate is sub-divided for reporting purposes: an open cut resource that includes material within the limits of the conceptual Martha pit and Gladstone pit; and underground resources within the Correnso area, the Martha Underground project area and for the WKP project. The resources are depleted for mining as at 31 December 2020.

Information relating to Geology, Sampling, Data Verification and Mineral Resources in this document was prepared by or under the supervision of Peter Church. Peter Church is a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy and is the Qualified Person for those topics. Mr Church is a full-time employee of OceanaGold Limited and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Qualified Person. A summary of the Mineral Resource Estimates is provided in Table 14-20.

Table 14-20: Summary of Mineral Resources as at 1 January 2020

	Indicated			Inferred		
	Inventory (Mt)	Grade (g/t Au)	Cont., Metal (Moz)	Inventory (Mt)	Grade (g/t Au)	Cont., Metal (Moz)
Correnso	0.05	5.2	0.01	-	-	-
MUG	6.00	5.2	1.01	2.5	4.7	0.4
MOP5	3.98	2.0	0.25	4.9	1.9	0.3
GOP	2.77	1.6	0.14	0.6	1.1	0.0
WKP	0.98	13.0	0.42	1.9	11.6	0.7
Total Resources	**13.78**	**4.1**	**1.83**	**9.9**	**4.4**	**1.4**

Note

- MUG Resources are reported below the MOP5 design and are constrained to within a conceptual underground designed based upon the incremental cut-off grade of 2.15 g/t which is defined at a gold price of NZ$2,394/ oz (US$1,700/ oz @ USD:NZD 0.71);
- WKP Resources are constrained to within a conceptual underground design based upon the cut-off grade of 2.5 g/t Au which is defined at a gold price of NZ$2,394/ oz (US$1,700/ oz @ USD:NZD 0.71);
- MOP5 and GOP resources are based upon conceptual pit designs, incremental cut-off grades of 0.5g/t and 0.56g/t respectively and a gold price of NZ$2,394/ oz (US$1,700/ oz @ USD:NZD 0.71);
- The tabulated Resources are estimates of metal contained as troy ounces;
- No dilution is included in the reported figures and no allowances for processing or mining recoveries have been made;
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly; and
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.

14.13 Risks

The Mineral Resource Estimates that form the basis of this report are based on a number of assumptions and are subject to a variety of risks and uncertainties which could cause actual results to differ from those reflected in this report. Potential geologic risks include unusual or unexpected geological complexities, variation in estimation and modelling of grade, tonnes, geologic continuity of mineral deposits, the possibility that future exploration, development or mining results will not be consistent with expectations and the potential for historic mine workings to be materially different to that assumed in these studies.

Indicated and Inferred Mineral Resources both have inherent risk, The term "Inferred Mineral Resource" refers to that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.

The term "Indicated Mineral Resource" refers to that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The geological and grade continuity to be reasonably assumed.

The orebodies within the Waihi Project area are geologically complex and in some instances are difficult to fully drill test. WKP has limited consented drill platforms, deeper portions of the Martha system are at challenging drill intersection angles and the town infrastructure and pit highwalls overlying the MOP5 Project place limitations upon the ability to fully drill test the deposit, Notwithstanding this, Waihi has a strong operational history and the risks are considered to be in line with those dealt with throughout the operations past history, these risks are thought to be well understood and actively managed.

15 MINERAL RESERVE ESTIMATES

15.1 Reporting Standard

The reserves were compiled with reference to the NI 43-101 and JORC. This section summarises the main considerations in relation to the preparation of Mineral Reserves and provides references to the sections of the study where more detailed discussions of particular aspects are covered. The basis for the estimation of Mineral Reserves is a metal price of NZD 2,112 per oz (USD 1,500 per ounce) for gold.

15.2 Reporting Date

Mineral Reserves for Martha and Correnso underground are reported as at December 31, 2020. There are no open pit Mineral Reserves.

15.3 Qualified Person

Information relating to underground Mineral Reserves, Mine Planning, Project Infrastructure, Capital and Operating Costs, and Economic Analysis in this document was prepared by or under the supervision of Trevor Maton. Trevor Maton is a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy and is the Qualified Person for those topics. Mr Maton is a fulltime employee of OceanaGold Company Limited and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Qualified Person as defined by NI 43-101 and is employed at the Waihi operation.

15.4 Mineral Reserves

15.5 Underground Reserves

Mineral Reserves are being declared for Martha underground based on the recent completion of the Feasibility Study. The Mineral Reserves were classified using the 2014 CIM Definition standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate Modifying Factors, as described herein, to potential mining shapes created during the mine design process.

Mineral Reserves is supported by studies which meet the definition of a Feasibility Study (FS). All permits and consents are in place for the extraction of the Mineral Reserve.

The Mineral Reserves for Martha underground and Correnso are summarised in Table 15-1.

Table 15-1: Reserve Statement for OceanaGold's MUG and Correnso Mine

Reserve Area	Class	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
MUG	Proven	-	-	-	-	-
	Probable	4.46	4.33	13.5	0.62	1.94
Total MUG		**4.46**	**4.33**	**13.5**	**0.62**	**1.94**
Correnso	Proven	0.04	4.92	9.2	0.01	0.01
	Probable	0.02	6.01	10.2	0.00	0.01
Total Correnso		**0.06**	**5.25**	**9.50**	**0.01**	**0.02**
Total Mineral Reserve		**4.52**	**4.34**	**13.5**	**0.63**	**1.95**

Notes to Accompany Mineral Reserve Table:

- Mineral Reserves are reported on a 100% basis;
- Mineral Reserves are reported to a gold price of NZD 2,112/oz;
- Tonnages include allowances for losses resulting from mining methods. Tonnages are rounded to the nearest 10,000 tonnes;
- Ounces are estimates of metal contained in the Mineral Reserves and do not include allowances for processing losses. Ounces are rounded to the nearest thousand ounces;
- Rounding of tonnes as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content;
- Tonnage and grade measurements are in metric units. Gold ounces are reported as troy ounces.
- Underground Mineral Reserves are stated using 2,4 to 3.3 g/t Au cut-off. Mining recovery ranges from 67% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated within a dilution zone. The dilution ranges from 0% to 7% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash flow model.
- The underground Mineral Reserves were estimated by Trevor Maton, MAusIMM (CP), QP.

15.6 Open Pit Reserve

There are no Open Pit Mineral Reserves.

15.7 Comments on Section 15

The QPs are of the opinion that the Mineral Reserves for the Project conform to the requirements of CIM (2014).

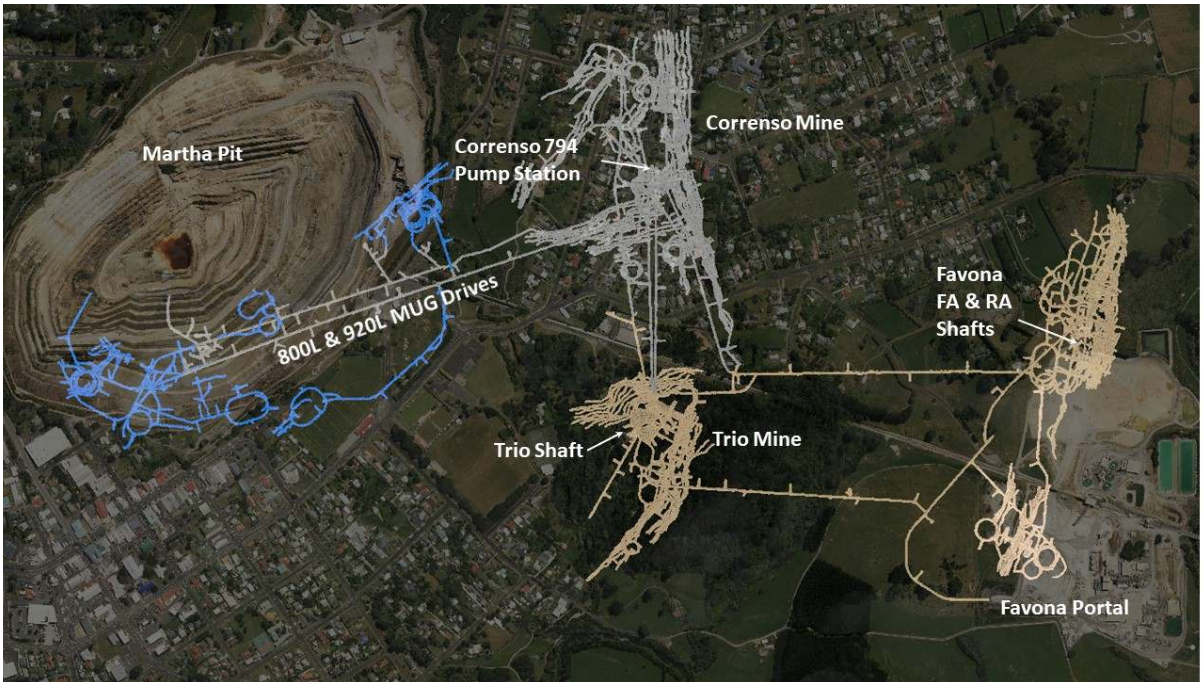
16 MINING METHODS

Permits and consents have been granted for Martha underground and all selected mining methods are to be accordance with the license, permit and consent conditions, principally related to placement of backfill, blast vibration limits, methods of working and hydrogeological controls.

16.1 Status of Current Mine Development

The Martha underground is accessed via the existing Favona portal through the existing Trio and Correnso workings and shares the ventilation development and shafts as well as the Correnso workshop, Trio cribroom and dewatering systems, refer Figure 16-1.

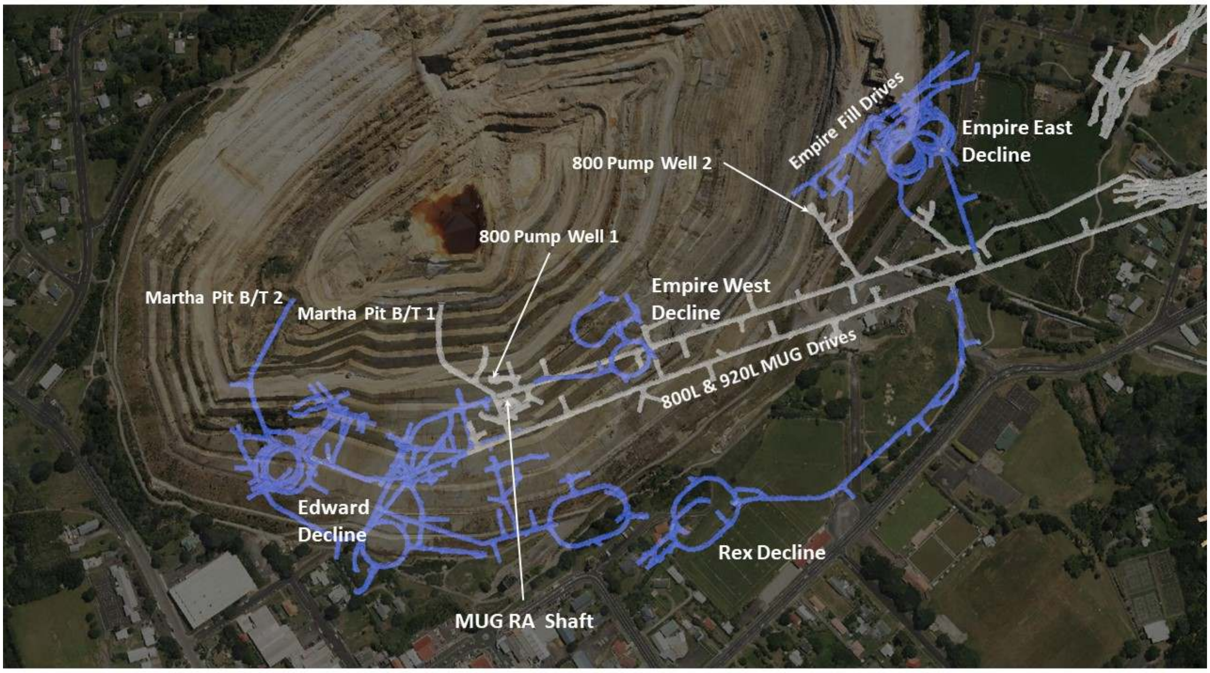
Figure 16-1: MUG Mine Access Development



Exploration drives were completed on 800[4] m RL and 920 m RL in 2018. Development of Martha underground commenced in mid-2019 and 2,169 m of lateral development and a 120 m ventilation raise were completed by the end of 2019 and a further 7,554 m of lateral development completed in 2020. Two breakthrough openings into the pit for ventilation and escape were also completed. The extent of development is shown in Figure 16-2. Development up to end 2020 has been focussed on ramp accesses for Edward, Empire, Rex and Royal mine zones, ventilation connections, pumping well access drives, drilling platforms and back fill drives.

[4] Note that the RL used for the underground mine is based on the Mt. Eden Grid with 1000 m added to the mean sea level so as to avoid the need for negative levels.

Figure 16-2: MUG Mine Development to End 2020



16.1.1 Historical Workings

The "Historical Workings" consists of the mining performed between 1878 and 1952, prior to the recommencement of mining activities in 1986. The origins of the current digital Historical Workings data were converted to a digital format from tracing historic mine plans available at the Moresby Street Office, in 2002. The Historic plans available were not a complete set. The historical plans were produced on several different medium (paper, linen, film) and in varying conditions from excellent quality to water damaged and torn and creased. This work is also covered in a report by GNS.

A focus on the information contained in the scanned mine plans and reports, has been used to compare and add information to the digital historical model. This process is usually captured during the void assessment process and outlined in the Company's Void Management Plan. Areas in the major lodes have required the inclusion of numerous sublevels not previously captured in the initial plan digitising project. The data is confirmed against the scanned copies of the long section and cross section plans available for the major ore lodes, and the probe drill holes and scanning programs in place.

The scope of the probe drilling program has increased to include inspection (camera stills and videos) of the holes, drill reports and scans of voids intersected where this has been possible.

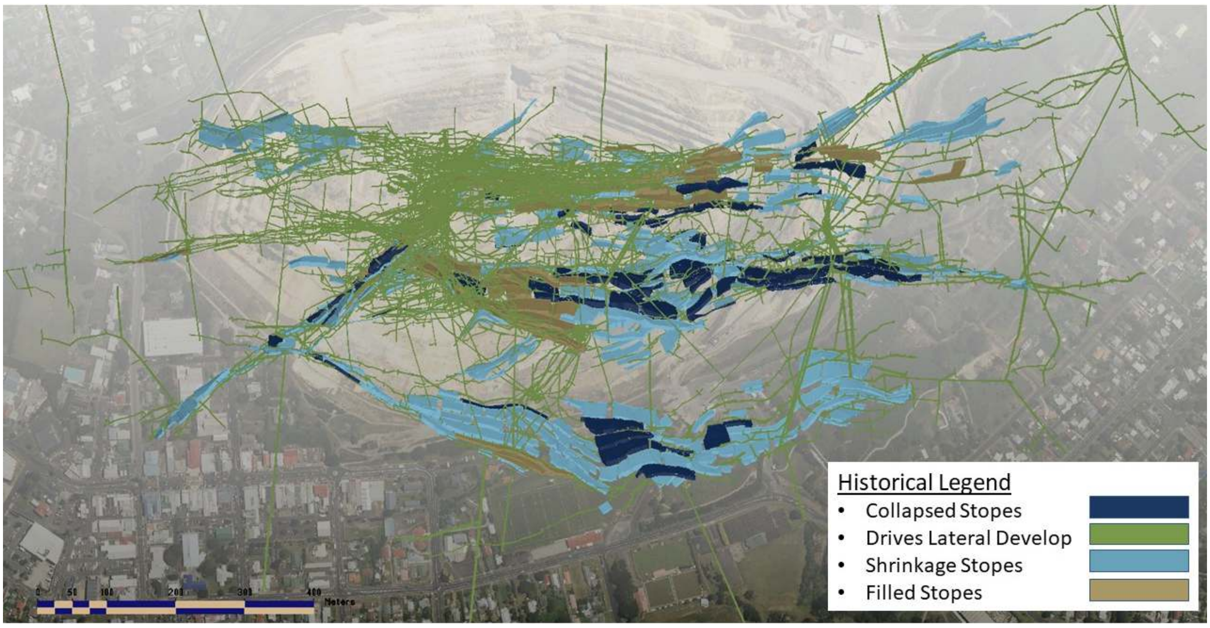
The understanding of the historical workings has increased as more probe hole, scan information and interrogation of historical plans has been performed. Open void extents of several stopes have been defined as well as increasing the understanding of the fill status and stope width dimensions from scanned historic long section and cross section plans.

The position of the historical workings that has been located, are generally within 10 m of there expected location. On occasion over broken/collapsed historic workings have been identified in the historic workings. The cause of the over break is potentially associated with historical ore extraction unidentified in the historic plans or due to poor ground conditions.

During 2020 the historic data was transferred from its Surpac format to a Deswik CAD model. This has consolidated the information and allowed more accessibility to the data. Currently the model is in the process of combining the geology historical stope model and the survey positioned historical stope model. An audit of the two data sets is required to consolidate the information and this is planned for 2021.

A centralised library of Deswik CAD projects has been generated, containing the scanned plans, imported and translated to best fit the current mine digital model. This allows the plans to be underlaid with the digital model as a reference for checking the position and accuracy of the digital data. The most recent digital model is shown below in Figure 16-3.

Figure 16-3: Historical Stope Mining Database



16.1.2 Mine Levels

Mine design references the historic mining levels as shown below in Table 16-1. Level spacing is approximately 18 m vertically and aligns approximately with the naming system of the historical mine.

Table 16-1: MUG Level & RL Nomenclature

MUG Level	Edward Area m RL	Empire East Area m RL	MUG Level	Edward Area m RL	Empire East Area m RL
5 level	980	950	16 level	782	752
6 level	962	932	17 level	764	734
7 level	944	914	18 level	746	716
8 level	926	896	19 level	726	696
9 level	908	878	20 level	708	678
10 level	890	860	21 level	690	660
11 level	872	842	22 level	672	642

12 level	854	824	23 level	654	624
13 level	836	806	24 level	636	606
14 level	818	788	25 level	618	588
15 level	800	770	26 level	600	570

16.2 Cut-off Grade Calculation

16.2.1 Underground Mining Costs

Previous operating costs for the Favona, Trio and Correnso underground mines have been reviewed from 2009 to 2019. Operating costs per tonne of ore mined vary depending on:

- the ore tonnage mined,
- the proportion of lateral development to stoping,
- the average width of stoping,
- any floor benching or overhand cut and fill, and
- the operating development in waste.

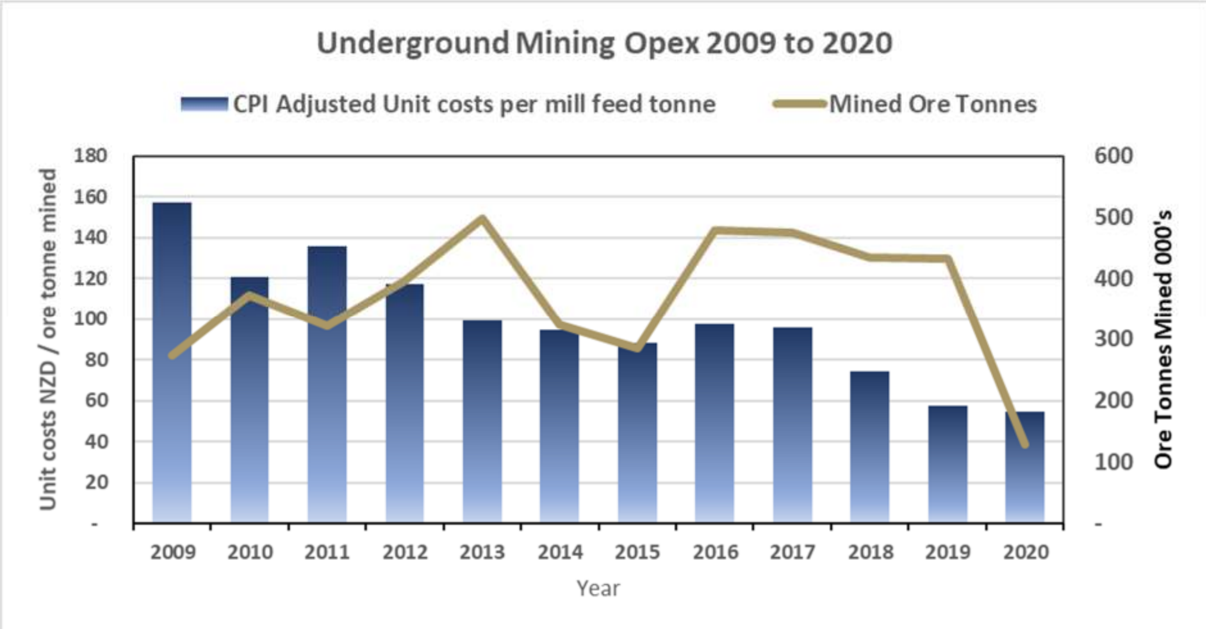
Unit operating costs have shown a general trend of decreasing operating costs as the mill feed tonnage increases and a strong trend in decreasing costs over time due to improvements in efficiency, improvements in ground conditions and the transition to owner mining. Table 16-2 shows the feed tonnage mined by year with actual operating costs and costs adjusted for Consumer Price Index (CPI).

Table 16-2: Underground Mining Operating Costs 2009 to 2019

Year	Mill Feed Tonnes	UG Operating Costs NZD M	Unit costs per tonne mill feed	New Zealand CPI Adjustment	CPI Adjusted Unit costs per tonne mill feed
2009	274,534	36.2	132	19.4%	157
2010	371,755	38.2	103	17.5%	121
2011	322,836	38.0	118	15.0%	136
2012	395,822	41.1	104	12.7%	117
2013	498,822	44.6	89	11.1%	99
2014	324,640	28.0	86	9.3%	94
2015	286,000	23.2	81	9.2%	88
2016	479,000	43.4	91	7.8%	98
2017	474,000	42.9	90	5.6%	96
2018	434,000	31.3	72	3.2%	75
2019	433,000	24.6	57	1.4%	58
2020	128,449	6.9	54	1.4%	55

This same information is shown in Figure 16-4 as the CPI adjusted unit costs and the mill feed tonnes over time.

Figure 16-4: Underground Mining Operating Costs 2009 to 2019



16.2.2 Cut-off Grade and Modifying Factors

Martha underground cut-off grades were reviewed in 2019 with the objective to provide cut-off grades for the various mining methods to be used. Three mining methods are proposed, viz:

- In areas where backfilled historic stopes are present the design stope would target recovery of the backfill and either a footwall or hangingwall skin where grade is present. A top down, side ring method was proposed in these areas by SRK and endorsed by Entech.
- In areas where unfilled historic stopes are present, Entech proposed to backfill these stopes with a cemented fill and apply conventional Avoca mining adjacent to the cemented fill for new stoping to recover footwall or hanging wall skins or remnant pillars.
- In areas previously unmined by the historic workings, conventional modified Avoca with rock fill would be used.

Parameters used to calculate the cut-off grades were derived from the nearby Waihi operation and the study work carried out by Entech and are summarised in Table 16-3. The cut-off grades determined for the various mining methods and used in the mine design are shown below in Table 16-4.

Table 16-3: MUG Cut-off Grade Parameters

Area	Units	Cut-off grade Input
Metal recovery	%	94%
Gold price	USD/oz	1,500
Royalty	%	2%
Mining cost Avoca	NZD/t	79.28
Mining cost remnants	NZD/t	107.64
Mining cost backfilled remnant	NZD/t	134.72
Backfill	NZD/t	5.42
Sustaining capital	NZD/t	1.77
TSF growth	NZD/t	2.18
Processing & refining	NZD/t	39.59
G & A	NZD/t	25

Cut-off grades also take into account silver as a credit at a 3.1:1 ratio to gold and process recovery of 60%. Mining costs include:

- finance leases on mobile equipment,
- supply and placement of rockfill and CAF,
- additional mine development for placing fill in historic workings, and
- footwall and crosscut development, additional ring drilling and higher proportions of remote mucking for the backfill remnant areas.

Mining recoveries vary based on the mining method.

Table 16-4: MUG Cut-off Grade by Mining Method

Area	Cut-off grade (g/t Au)
Virgin Avoca mining	2.4
Avoca mining in remnant areas with CRF	2.8
Backfill remnant areas, top down, side ring mining method	3.3

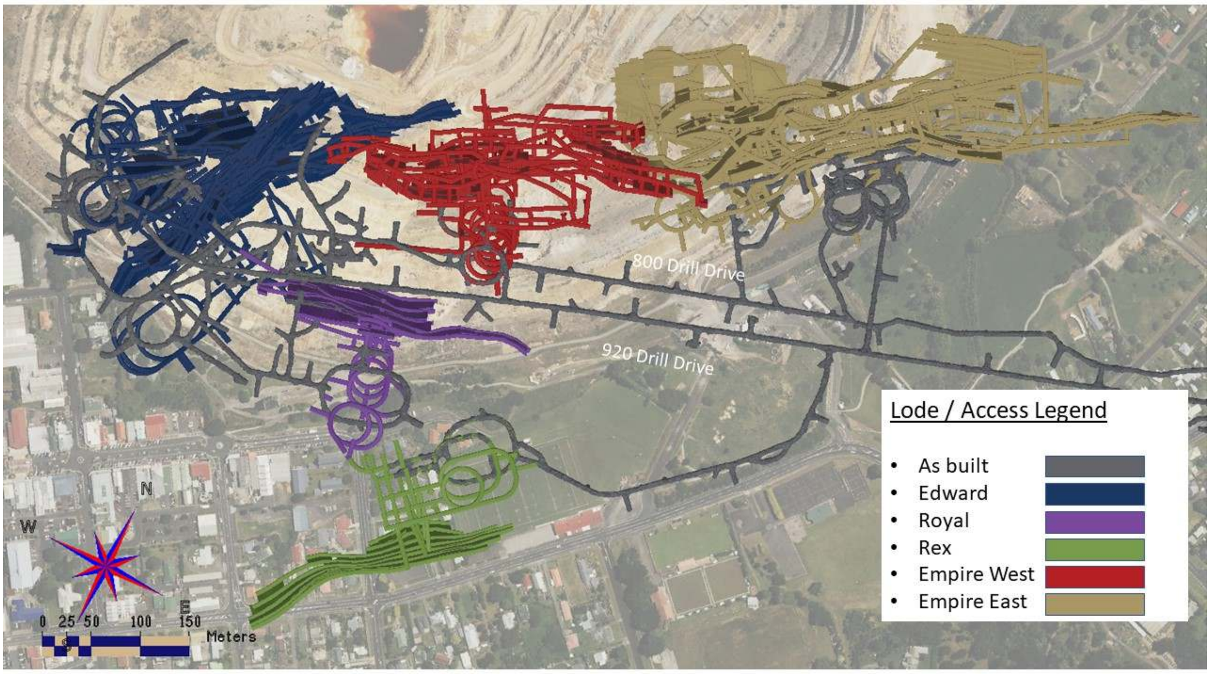
16.3 Mine Design

Stope optimisation results were used as a basis for the underground mine design. The top of the mineralisation is approximately 120 m below surface and extends to a depth of approximately 500 m below surface.

Mineral Resources extend below and outside of the existing open pit mine. There are multiple orebodies within the MUG Mine including Edward, Empire, Martha, Rex, and Royal. All mining areas share similar stoping methods and have very similar modifying factors and assumptions and design criteria applied. For simplicity all these areas are hereafter collectively referred to as Martha underground.

The study work undertaken for Martha meets FS level standard. Mining studies have been conducted for mine design, mine planning, ventilation, cut-off grade, detailed cost estimation and economic evaluation. The site has had a 14 year operating experience with Mineral Reserve reconciliation and metallurgical recovery performance. Actual costs for underground mining, ore processing, general and administration and selling costs are well known. A mine plan has been developed which is technically achievable and economically viable. All modifying factors have been duly considered.

Figure 16-5: General Lode Layout



16.4 Hydrogeology

The hydrology of the Waihi area is generally well understood and well documented with ongoing studies taking place to further improve understanding. It is understood that there is good / direct hydraulic connectivity between the Martha Pit and historical underground (Russell, 2010, GWS 2018) and with the nearby Correnso mine. The majority of groundwater within the Martha area is contained within major structures and historical voids. Perched water and water stored within historical voids and structures may cause higher instantaneous water inflows during mining at Martha and will need to be carefully managed. GWS (2019) consider that three groundwater flow systems exist in the Waihi area: A shallow system within the alluvium and recent volcanic ashes (water table aquifer); a transitional system within unweathered, young volcanic rocks; and a deep system within the andesite rocks.

Groundwater monitoring wells and piezometers have been installed in shallow and transitional (upper levels of the andesite rock mass) groundwater systems. Groundwater levels in veins were monitored in pumped shafts or pumping wells and currently, with underground piezometers installed in the pumping wells.

Andesite rock forms the local basement rock body for the area and hosts the mineralisation. The andesite rocks were formerly exposed at the surface where the weathering profile developed that perches groundwater in the overlying younger materials and acts as the transition zone between groundwater in the andesite and the shallower groundwater systems. This transition zone is indicated to extend up to 24 m or more in thickness. Within the upper part of the unweathered andesite (above about 700 mRL to approximately 1000 mRL), fracturing is more widespread due to relaxation of the rock mass. With increasing depth, the density and aperture of the fractures reduces, resulting in lower permeability.

Overall, the andesite rock mass is of low permeability due to the low-density of fractures in most areas. Exceptions to this occur where post mineralisation faulting has taken place, known to result in fracture zones of higher permeability. Vein systems and associated fracturing provide planar zones of higher permeabilities within the rock mass. The contrast between rock mass permeability and vein zone permeability is marked by full and rapid water losses from investigation drill holes when vein and associated fracture zones are intercepted.

Groundwater levels in the andesite rocks are influenced by current mine dewatering. Groundwater levels and directions are controlled by the presence and interconnections (where they occur) of the workings, vein systems and post mineralisation structures (faults and fracture zones).

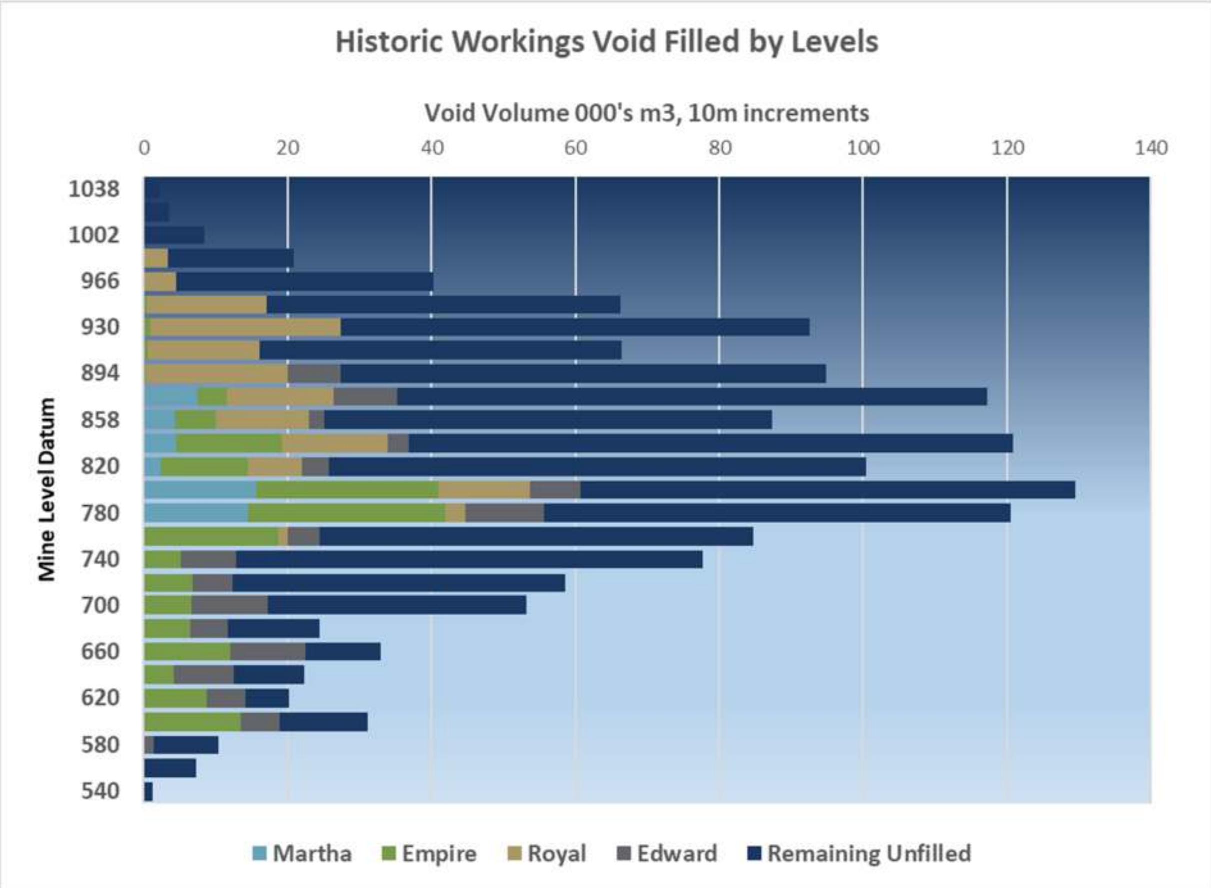
The extent of dewatering in the andesite rock mass is limited in spatial extent due to low rock mass permeability. To the north and west the extent of the dewatering effect is understood to be limited by faulting which act as hydraulic barriers to groundwater flow. A restricted area of dewatering is observed to the south west that is considered to be due to drainage along north-south oriented vein structures

16.4.1 Historical Dewatering

Mining of the Martha ore bodies commenced in 1878 with a series of open cut operations at the elevated location of Martha Hill where gold could be extracted above the water table. Dewatering began once the water table was reached (1110 mRL; 1893) and progressed at a relatively steady rate of approximately 18 m/year depth. Later mining of the Martha veining and other prospects to the east of the Martha Mine required the sinking of shafts to access the ore body at depth. In the Martha ore bodies the deepest documented shaft was the Waihi No five (5) shaft that extended to a depth of 557 mRL. The other workings along the Martha veins to the east were recorded to have reached a depth of 541 mRL. The original Martha Mine was closed in 1952.

The historical mining operations resulted in a series of shafts being sunk across the Martha ore bodies, with drives along the length of the veins at various levels to enable stoping of the ore bodies, thereby increasing interconnection. The openness of the old workings after mining is reasonably well documented as shown in Figure 16-6 which is the void volume by 10 m increments; although consolidation and adjustment of rock mass into the stoped areas is likely to have reduced void space.

Figure 16-6: Historical Workings Void Filled by Levels



Source: GWS Ltd, Martha Groundwater Effects Assessment, 2019, updated OGC 2021.

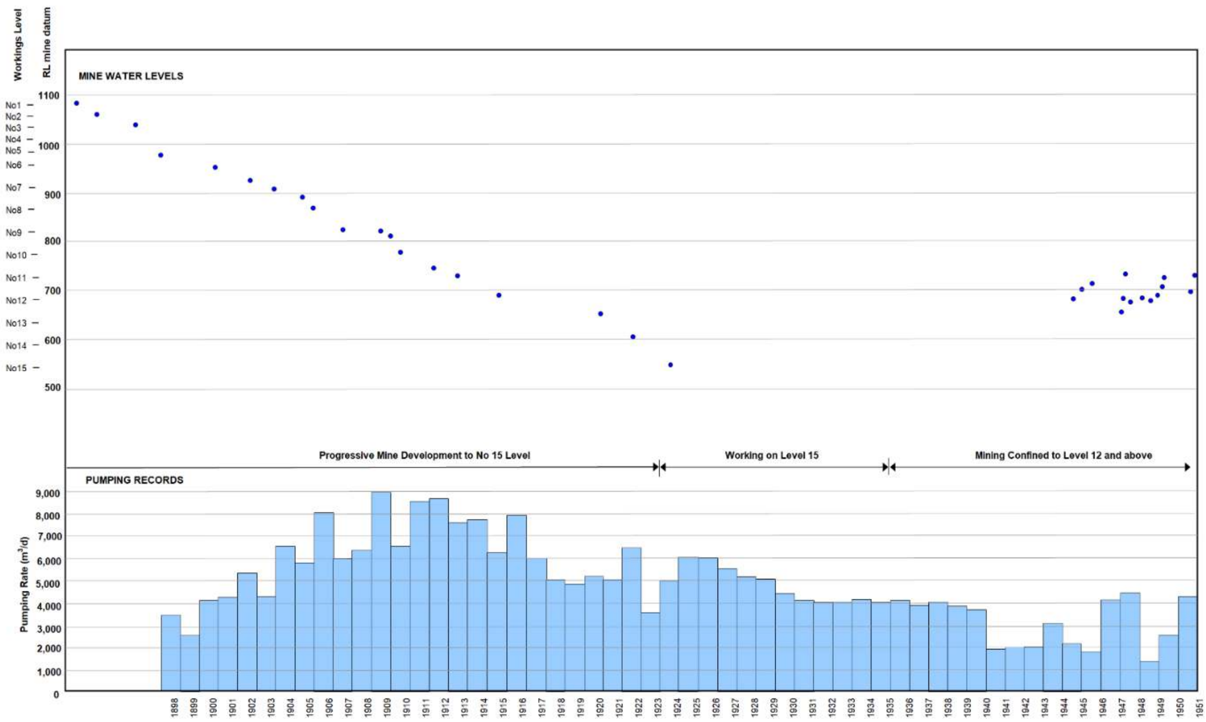
It should be noted that Figure 16-6 does not include any estimates of the void volumes available in historically non-mined veins and fault systems or void reduction by closure by ground adjustment or collapse.

Dewatering of the Martha ore bodies was problematic during historical mining. Limitations in pumping technology meant that groundwater inflows impeded the access to reserves at increasing depth. Relatively good records of pumped volumes of water were kept after 1900 and the average daily inflows relative to shaft depth are included in Figure 16-7. As shaft depths increased, periods of high initial groundwater inflows were experienced reaching some 9000 m^3/d. The water levels in the mine and shafts between Level 6 and Level 11 (950 to 730 mRL) were stated to have been pulled down after several months of pumping at a rate of 13,000 m^3/d, which reduced to between 4,900 and 6,500 m^3/d over time. Peak inflows were noted to occur immediately after accessing the veins at a new level. It was reported by the Chief Engineer in 1912 (McAra J.B, 1988) that inflows at Level 10 (780 mRL) were estimated to be 26,000 m^3/d when first cut. In the 1920's and 1930's the mine was expanded along strike between Level 13 and Level 15 (650 to 557 mRL) and groundwater inflows stabilised to 4,200 m^3/d. This rate of pumping was considered "normal" for the Martha Mine. It was noted that the inflows reduced from Level 11 to Level 15 (730 to 550 mRL) and was thought to reflect the narrower and less cavernous lodes and tighter country rock.

There were a number of occasions, historically, where pump failures or shutdowns occurred that resulted in recovery of the water levels within the mines. In 1912, as a result of a miners' strike, the water level rose 45 m over a period of 68 days from 728 mRL to 773 m RL. And in 1945 and 1949 the water level rose 30 to 40 m from 678 m RL to 708-718 m RL over the period of one year.

After the closure of the historical mines the old workings were collapsed locally and allowed to fill with water, thereby forming the depression of what became the Martha Lake. The water level in the lake recovered to an elevation of approximately 1110 m RL which reflected the groundwater level through the interconnected vein system.

Figure 16-7 Historical Martha Annualised Average Pumping Rates and Levels



Source: GWS Ltd, Martha Groundwater Effects Assessment, 2019

16.4.2 Recent Mine Dewatering

Since the re-commencement of mining, dewatering has been a necessary part of the operation since 1989. The MOP operation required initial dewatering of the Martha lake and then progressively from the workings as excavation advanced. Dewatering was undertaken until 2011 from the old No.7 shaft which is within the footprint of the mine. In 2011 three bores constructed from a bench within the pit at 940 m RL (mine datum) were used for dewatering. In May 2015, pumping from Correnso mine commenced with the establishment of a major pump station at 794 m RL. Correnso water is pumped to the Trio decline and exits the underground at the Favona portal to discharge at the WTP. Favona water is now pumped up the Trio incline, lifted to the Trio decline and, thence, to the Favona portal.

The water pumped from the underground comprises:

- Water sourced from storage within the interconnected old workings, vein systems, and where present, from post mineralisation faulting.
- Water released from groundwater storage in the surrounding country rock.
- Rainfall within Martha Pit catchment which enters the historical workings from the pit floor.

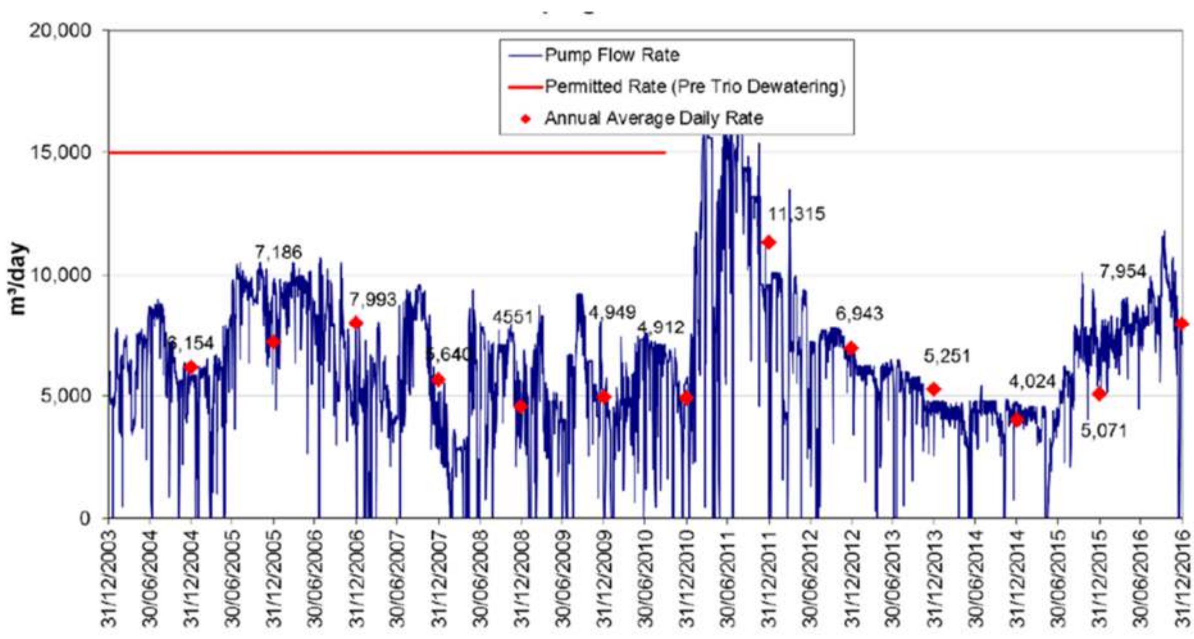
The groundwater response to dewatering at the Martha Pit and the underground mines has been observed at a network of piezometers located around the Waihi area. It has been shown, over time, that the vein systems of the Martha ore bodies (Martha, Empire, Royal, Trio and Correnso) are all highly interconnected and are separated from the Waihi East ore bodies (Favona, Moonlight). As parallel development of the underground mines and at the Martha Pit continued, it became apparent that dewatering at Martha had the effect of dewatering the other vein systems. This was also known and utilised historically.

Pumping records for the combined water take from the Martha and Correnso operations are included in Figure 16-8. It can be seen that from:

- 2003 to 2008 show high initial pumping rates (>7,000 m^3/d) as deepening of the mine took place and water levels were rapidly pumped down.
- 2008 to 2010 a stable volume of dewatering from 4,000 to 5,000 m^3/d was reached, with these volumes consistent with historical inflows.
- 2010 to 2011 the Trio underground mining operation commenced and, due to the connectivity with other vein systems and the rate of mine development, the pumping rate increased to greater than 15,000 m^3/d as water levels were rapidly drawn down.
- 2011 to 2015 these volumes diminished until pumping rate stabilised between 4,000 and 5,000 m^3/d, reflecting inflow from the rock mass plus rainfall infiltration via Martha Pit.
- 2015 to 2016 a gradual increase in pumped volumes is noted, and this is interpreted to be a result of an increased rate of development to deeper level resulting in further release of water from storage, bringing the water level down from the 795 m RL.

The water levels within the vein systems in all of the ore bodies are held approximately constant at the pumped level from wherever dewatering takes place. As there is no direct hydraulic connection, groundwater from the Martha East (Trio/Correnso) and Waihi East (Favona / Moonlight) underground operations are pumped separately to the Favona portal.

Figure 16-8: Pumping Rates for Martha and Correnso from 2003 to 2016



Source: GWS Ltd, Martha Groundwater Effects Assessment, 2019

16.4.3 Groundwater Inflow Estimates

Dewatering of the veins and workings will be required to reach the target mine depth. Inflows will be from storage within the workings, veins and the rock mass. In addition, rainfall infiltrating the base of the pit will report to MUG during mining. Small pockets of perched groundwater may also be encountered in old mine workings. Below 550 m RL previously unmined veins are expected, so only water from the veins, rock mass and Martha Pit will be removed.

GWS modelled the groundwater inflows to MUG using an analytical model that is based on the historical dewatering data set for the historical Martha Mine. Discharge from the historical mine was sourced from storage in interconnected vein systems and groundwater inflow from the surrounding rock mass. Vein water storage was greatest where the veins were widest. These were also the areas of maximum gold yield. Groundwater sourced from the rock mass was considered to be a minor contributor to the historical discharge. Previous mining took place to a depth of approximately 541 m RL. Backfilling of stopes was variable and collapse and compression of stopes will have modified the void space following historical mining. It is assumed that the sum of these void increases and decreases has not substantially altered the original groundwater storage capacity.

The spreadsheet model developed by GWS to calculate groundwater inflows was calibrated against the historical inflows down to the 541 m RL level where mining ceased. The observed mine elevation and inflow relationships have then enabled predictions of inflows for the remaining 41 m of unmined vein to the proposed level of mining at 500 m RL.

GWS used two methods to consider the rate of lowering to compare the modelled vs observed drawdowns at this time. The first is to simply alter the rate of pumping in the predictive model to that actually occurring. Doing this suggests it would take 5 years (circa 2025) from the commencement of dewatering to lower groundwater to an elevation of 500 m RL. An alternate approach to using the model is to project the observed rate of dewatering forward, doing this results in a lowering of the groundwater level to 500 m RL in 3 years (circa 2023).

GWS conclude that given the uncertainties involved the actual time at which lowering to 500 m RL is achieved is likely to fall into this range. Table 16-5 provides the dewatering schedule and calculated average pumping rates.

Table 16-5. Predicted Pumping Rates

Target Depth m RL	Average Pumping Rate m3/day	Method 1 Target Date	Method 2 Target Date
690 to 650	12,500	Feb-22	Sep-21
650 to 600	12,500	May-23	Jun-22
600 to 550	12,500	Aug-24	Mar-23

Source: GWS, memorandum, Martha Underground – Review of Dewatering Rates and Effects on Groundwater, February 2021

The current average daily dewatering rate is approximately 14,000 m^3/d.

16.4.4 Martha Underground Dewatering – Drawdown Effects

Dewatering from 700 m RL to approximately 500 m RL will be achieved by pumping water inflows from vein systems collected in underground sumps. Lowering water level in the veins will result in a corresponding pressure response in the andesite country rock. Groundwater monitoring to date has shown that this pressure change in the vein systems has had little effect on groundwater levels in the overlying younger volcanic rocks during mine dewatering.

16.5 Geotechnical

Geotechnical studies related to MUG were undertaken by SRK between 2011 and 2017 and by Entech, AMC and PSM during 2018 and 2019 and by Entech 2020 and 2021.

PSM engineering consultants reported on the effect of MUG on the Martha Pit wall stability for the smaller MOP4 pit and concluded that MUG will run in parallel with the Martha Pit and this will have several benefits.

AMC investigated the stability of the underground workings AMC reported that based on the current understanding of ground conditions, the planned ongoing investigation of conditions as suitable drilling positions become available, and the proposed cautious approach to development using close ground control techniques where required, AMC is confident that the proposed Martha underground mine can be developed and brought into production without any compromise to underground or surface stability.

AMC reported that the ground conditions influence the mining method, the means of access, and the design of stopes and access tunnels. A critical aspect of the Martha underground is to undertake investigations to understand those conditions so that a safe and efficient mining method and well-informed approach to developing the mine is used.

AMC concluded that stable stope strike spans of up to 30 m can be mined. Caving and surface subsidence potential has been assessed for development and stoping with the risk being low if recommendations for ground support, allowable spans, and management techniques are followed.

AMC also reported on a geotechnically significant zone, known as the Milking Cow zone, is a cave zone located below and to the east of the MOP5 pit. Historical mining of this zone commenced prior to 1907 and it was noted in historical documents that the surface subsidence crater was “constantly moving”. Although production records are not available, anecdotal evidence suggest that +/- 1.5 Mt of ore was extracted over a 20-year period. No Mineral Resource is extracted from this area by the underground mine.

Entech were commissioned in 2020 to undertake a FS level geotechnical assessment of the mining plan which also included advanced numerical modelling. The following sections refer to their report, dated March 2021, Martha Underground Mine - Geotechnical Feasibility, Ent 0721.

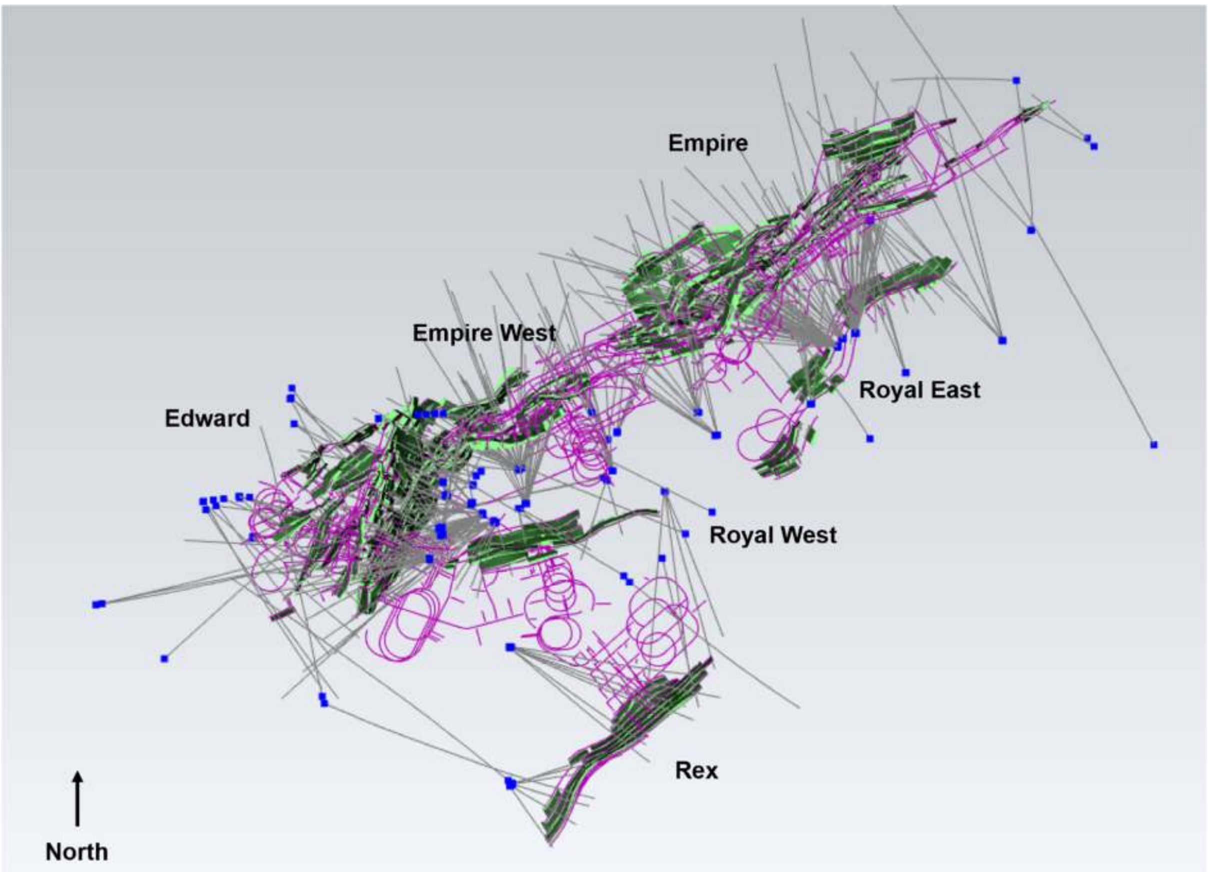
16.5.1 Rock Mass Characterisation

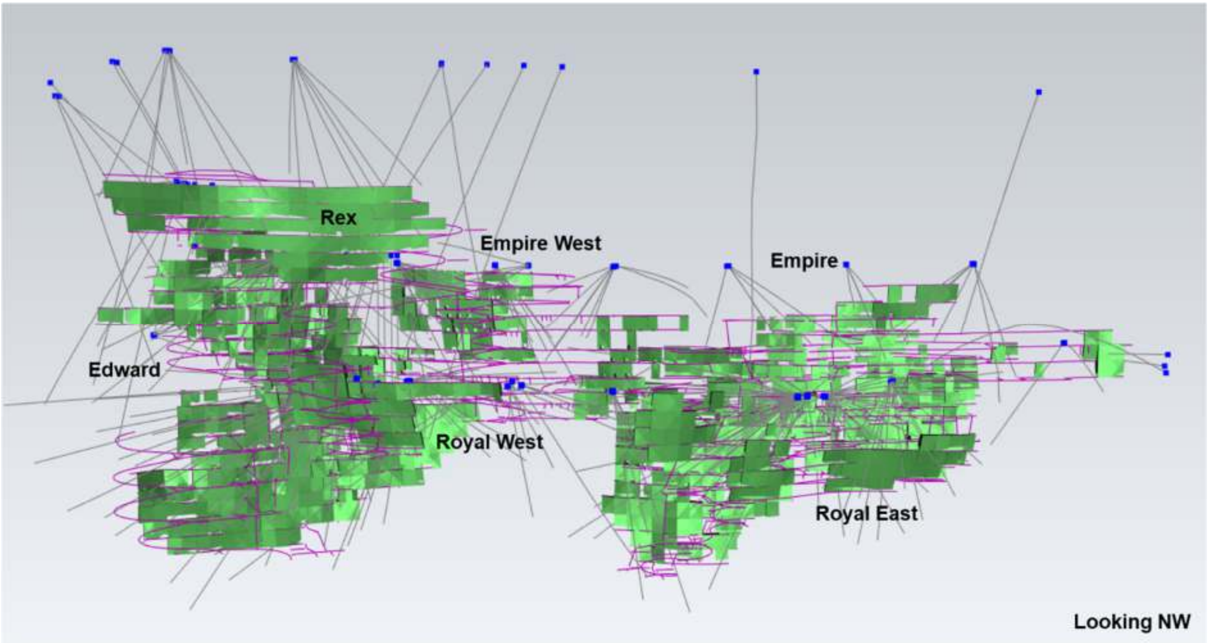
A total of 84,168 m of drill core was provided by the OGC exploration geology department for assessment. The majority of this was geotechnically logged, mainly by the onsite geologists under the direction of the underground Senior Geotechnical Engineer. Comments on the coverage of the drilling are provided:

- Entech considered the extent of drill hole coverage with domain geotechnical logging is satisfactory for the Martha Underground Mine, and is generally considered sufficient for FS level geotechnical studies (Figure 16-9).
- The Rex orebody had the least coverage and more data is required to increase the confidence of the rock mass at Rex, especially the lower levels. There is currently a drilling program underway to drill further extents of Rex.

- Further on-going geotechnical logging (either photo or on-site) of holes as they become available from resource definition drilling to gain more data would be beneficial to building a more detailed dataset for MUG, especially for deeper section of the orebodies. This will also help to validate the data collected by the site geologists.
- Decline and development infrastructure is partially covered with geotechnical drilling and logging (see Figure 16-9).
- No structural data was collected directly from core for joints, geologists only record structural measurements for veining and faults.

Figure 16-9. Plan & Long Section of Geotechnically Logged Drill Holes.



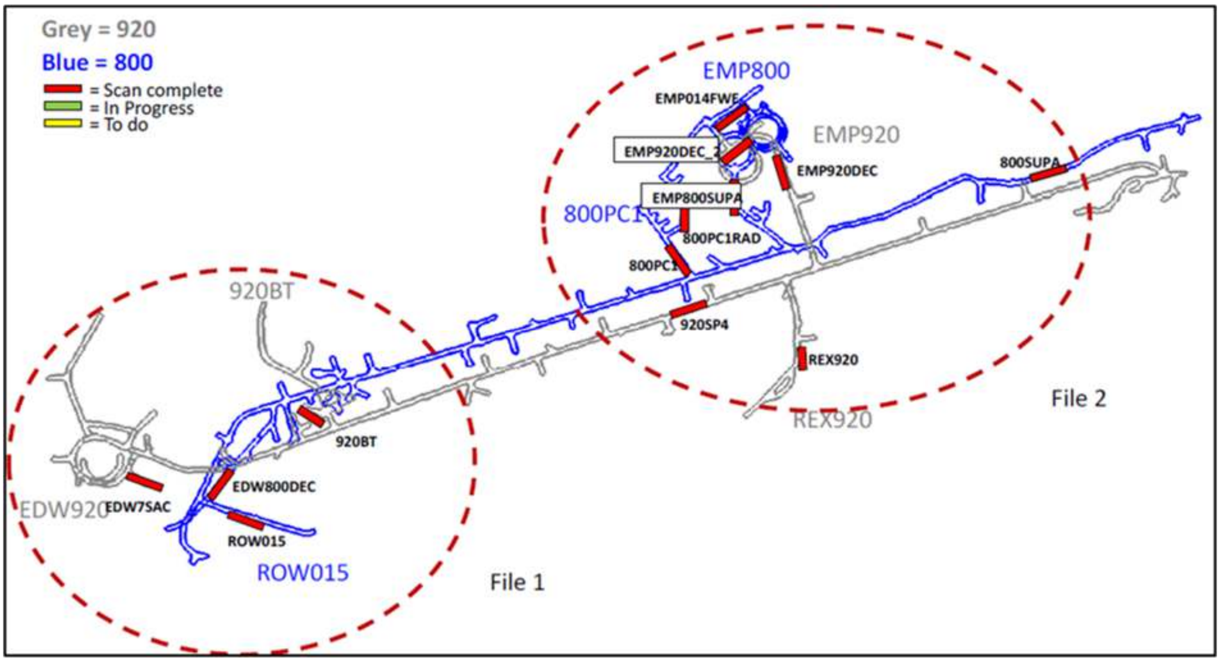


Source: Entech 2021 FS report, ENT 0721

16.5.2 Structure

A total of 807 orientation measurements were recorded during the underground data collection campaign from scanline mapping, refer Figure 16-10.

Figure 16-10. Plan View of Scanline Mapping Locations.



Source: Entech 2021 FS report, ENT 0721

Mapping of development headings and key structures within MUG was also recorded by the site geotechnical engineers and this data was validated and analysed to compare with the scanline mapping. A total of 114 data points were collected for this data set.

The combined data was used as the initial base of structural interpretation for Martha, and Entech recommend on-going scanline mapping within the MUG development. Data analysis for the ground support review suggested that a single structural domain exists for Martha. This will need to be further explored with the comparison of underground mapping data from each mining area.

All data was grouped into a single domain and the data filtered and examined in the Rocscience software package DIPS to determine structural sets present throughout the Andesite at Martha.

The legend/key for structures and percentage makeup of total population in the mapping database are as follows:

- jointing (65.5 %)
- vein (27.6 %)
- shear (4.5 %)
- fault (2.2 %)
- bedding (0.2 %)

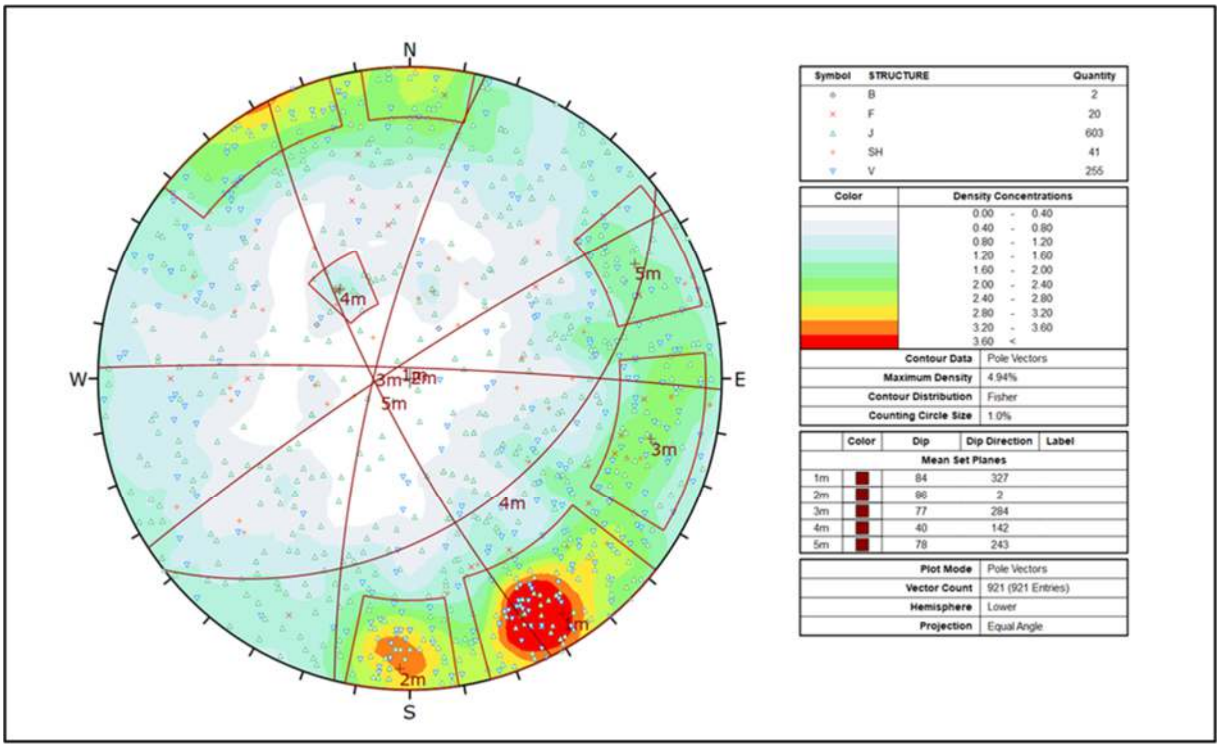
Jointing and Veins make up the majority of the structure types recorded in the mapping at MUG, with Shears and Faults also making up a reasonable proportion of recorded structures. This is expected and typical of the Waihi Mine due to the structural nature of the deposit.

A stereonet plot of the structures recorded is given in Figure 16-11 and the major identified sets are as follows:

- Set 1: 64° / 327° - moderately dipping, most dominant joint set.
- Set 2: 66° / 2° - moderately dipping set, sub parallel to set 1.
- Set 3: 77° / 284° - Steep dipping set.
- Set 4: 40° / 142° - moderately dipping set, weak set with few measurements.
- Set 5: 78° / 243° - Steep dipping set.

Data was then further filtered into different structure types to determine any other structural relationships. The same sets are seen across the joints and veins with faults and shears very dispersed but one small concentration which lines up with the Edward Lode trend where a lot of the development and mapping was undertaken.

Figure 16-11: DIPS Plot of all relevant drill hole structure data for the Martha underground Andesite unit.



Source: Entech 2021 FS report, ENT 0721

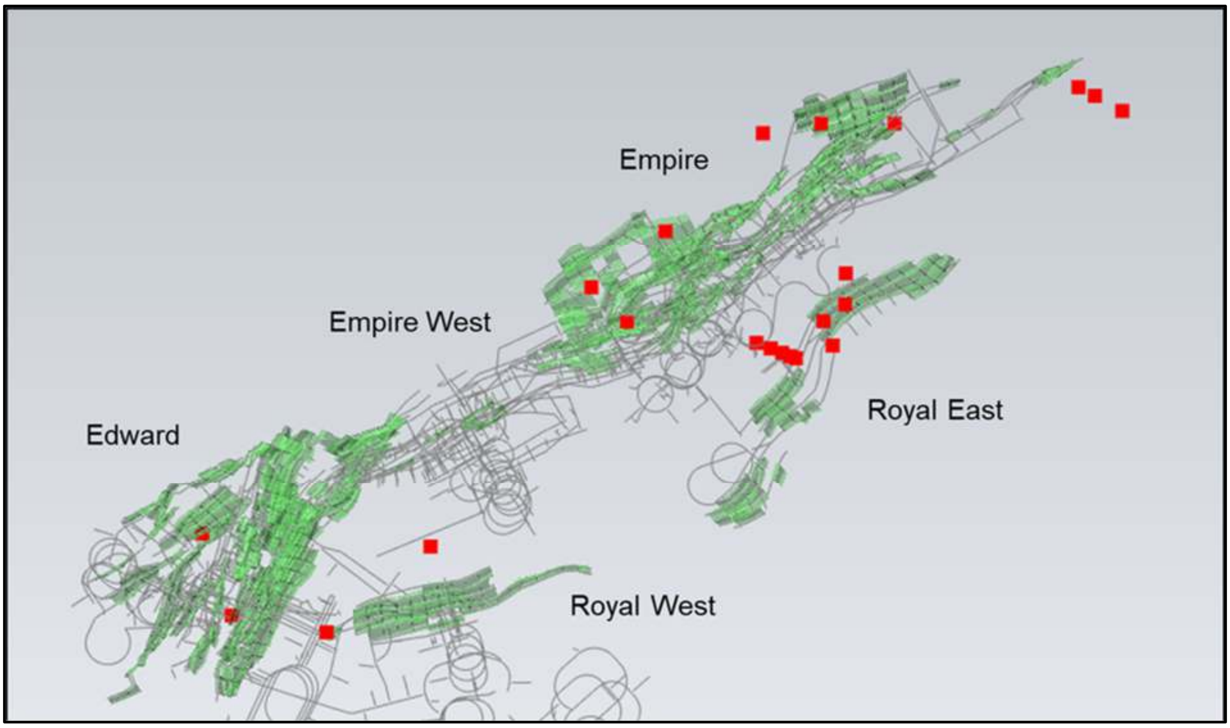
16.5.3 Rock Testing

A number of rock properties testing has been undertaken at the Waihi Underground mine since 2006. A specific testing program from MUG was undertaken in 2020 undertaken on drill core samples taken during site visits in late 2019. The results of all testing at Waihi has been summarised within this section for the benefit of compiling a complete database for the site geotechnical engineers to have moving forward and to allow comparisons of MUG to the past underground mines. The types and number of tests conducted consisted:

- Uniaxial compressive strength x 65 (36 at MUG)
- Indirect Brazilian tensile strength x 101 (45 at MUG)
- Youngs Modulus x 41 (24 at MUG)
- Poisson's Ratio x 31 (24 at MUG)
- Point Load strength x 16 (16 at MUG)
- Triaxial Single Stage x 18 (18 at MUG)
- Bulk Density x 103 (53 at MUG)

The rock properties testing undertaken to date is quite unevenly distributed across the planned MUG mining layout. No testing has been undertaken on the rock mass in or around Rex or Empire West and only limited testing around Edward and Royal West. Empire and Royal East have an adequate distribution of testing. The spatial distribution of samples throughout the planned MUG mine can be seen in Figure 16-12.

Figure 16-12: Plan View of Samples Collected for Geotechnical Testwork.



Source: Entech 2021 FS report, ENT 0721

The type, number and distribution of samples collected for testing is considered adequate for the feasibility level study apart from the lack of data from the ore zones. This is mainly due to the single host rock type and its consistency across the mine. From early development the Andesite in Rex appears to be softer and slightly different, so testing of rock samples from Rex is highly recommended. Further testing of the rock mass at depth is also recommended and testing campaigns should be carried out on all underrepresented areas as mining progresses. Table 16-6 provides a summary of results of each test for rock type as well as the number of tests undertaken.

Table 16-6. Summary of Laboratory Tests

Rock Sample Test	AQ - Correnso	AQ - Martha	Quartz	PO AQ - Correnso	AF - Trio	AQ - Trio
Uniaxial Compressive Strength (MPa)						
Mean	83	106	24	-	21	78
Max	179	255	24	-	21	180
Min	17	9	24	-	20	23
Sample Size	18	36	1	-	2	8
Uniaxial Tensile Strength (MPa)						
Mean	8.0	8.0	8.1	7.6	3.9	6.3
Max	16.3	20.6	14.3	9.4	12.0	11.0
Min	1.9	1.9	3.5	5.7	1.0	2.0
Sample Size	29	45	3	3	11	10
Young's Modulus (GPa) Secant (0-50%)						

Mean	35	36	-	-	13	24
Max	68	47	-	-	13	29
Min	6	19	-	-	13	20
Sample Size	14	24	-	-	1	2
Poisson's Ratio Secant (0-50%)						
Mean	0.3	0.2	-	-	0.3	0.2
Max	0.7	0.3	-	-	0.3	0.2
Min	0.0	0.2	-	-	0.3	0.1
Sample Size	4	24	-	-	1	2

Source: Entech 2021 FS report, ENT 0721

The data was then further refined to exclude all results where the sample failed in shear or along defects and that were moderately to highly weathered. This is undertaken to determine the true intact strength of the rock types without the effect of the defects causing premature failure.

16.5.4 Weathering

Weathering around the Waihi deposits typically extends down the vein margins to over 250 m below surface however, the andesite host rocks can appear only weakly weathered at or near the surface. All mining will be undertaken in fresh rock, but weathering and oxidation in the veins will vary from fresh to very weathered.

16.5.5 Rock Mass Characterisation

Rock mass characterisation used the Q system developed by Barton et al (1974). The Q-system classifies a rock mass based on several factors including When determining worst, most likely and best case rock mass conditions, each of the individual input parameters were assessed by lode and by geotechnical domain. In general, most likely has been determined as the mean RQD, and the in the case of Jn, Jr and Ja as the most frequently occurring value/number. Worst and best case values represent the upper and lower values encountered from the logging. A summary of rock mass conditions for each geotechnical mining domain (hangingwall, ore and footwall assessed is shown in Table 16-7.

Table 16-7. Summary of Rock Mass Characterisation

Edward Combined		RQD	Jn	Jr	Ja	Jw	SRF	Av Q'	Av Q	UCS	Q rating
HW	Worst	10	15	1	15	1	10	0.044	0.004	0.5	Exceptionally Poor
	Best	100	1	4	0.75	1	2.5	533	213	250	Extremely Good
	Expected	48	6	3	6	1	2.5	4	1.6	75	Poor
ORE	Worst	10	15	1	15	1	10	0.044	0.004	0.5	Exceptionally Poor
	Best	100	1	4	0.75	1	2.5	533	213	250	Extremely Good
	Expected	36	6	3	6	1	2.5	3	1.2	75	Poor
FW	Worst	10	15	1	15	1	10	0.044	0.004	0.5	Exceptionally Poor
	Best	100	1	3	0.75	1	2.5	400	160	250	Extremely Good

		RQD	Jn	Jr	Ja	Jw	SRF	Av Q'	Av Q	UCS	Q rating
	Expected	52	6	3	6	1	2.5	4.3	1.7	75	Poor
Empire Combined		**RQD**	**Jn**	**Jr**	**Ja**	**Jw**	**SRF**	**Av Q'**	**Av Q**	**UCS**	**Q rating**
HW	Worst	10	15	0.5	15	1	10	0.022	0.002	3	Exceptionally Poor
	Best	100	1	4	0.75	1	2.5	533	213	175	Extremely Good
	Expected	66	6	3	6	1	2.5	5.5	2.2	75	Poor
ORE	Worst	10	15	0.5	15	1	10	0.022	0.002	0.5	Exceptionally Poor
	Best	100	1	4	0.75	1	2.5	533	213	175	Extremely Good
	Expected	60	6	3	6	1	2.5	5	2	75	Poor
FW	Worst	10	15	1	15	1	10	0.044	0.004	3	Exceptionally Poor
	Best	100	1	3	0.75	1	2.5	400	160	175	Extremely Good
	Expected	74	6	3	6	1	2.5	6.2	2.5	75	Poor
Royal West		**RQD**	**Jn**	**Jr**	**Ja**	**Jw**	**SRF**	**Av Q'**	**Av Q**	**UCS**	**Q rating**
HW	Worst	10	12	1	15	1	10	0.06	0.01	15	Extremely Poor
	Best	100	1	4	2	1	2.5	200	80	75	Very Good
	Expected	65	4	3	6	1	2.5	8.1	3.3	75	Poor
ORE	Worst	10	15	1	15	1	10	0.04	0.004	3	Exceptionally Poor
	Best	100	1	3	2	1	2.5	150	60	75	Very Good
	Expected	62	4	3	8	1	2.5	5.8	2.3	75	Poor
FW	Worst	11	9	1	15	1	5	0.08	0.02	3	Extremely Poor
	Best	100	1	4	0.75	1	2.5	533	213	75	Extremely Good
	Expected	80	3	3	6	1	2.5	13.3	5.3	37.5	Fair
REX		**RQD**	**Jn**	**Jr**	**Ja**	**Jw**	**SRF**	**Av Q'**	**Av Q**	**UCS**	**Q rating**
HW	Worst	10	15	1	12	1	7.5	0.06	0.008	15	Exceptionally Poor
	Best	100	1	3	1	1	2	392	157	75	Extremely Good
	Expected	43	9	3	6	1	5	3	0.76	75	Very Poor
ORE	Worst	10	15	1	15	1	7.5	0.04	0.006	15	Exceptionally Poor
	Best	86	3	3	2	1	2.5	14.3	5.7	75	Fair
	Expected	14	15	3	6	1	5	0.3	0.07	37.5	Extremely Poor
FW	Worst	10	15	1	15	1	7.5	0.04	0.006	37.5	Exceptionally Poor
	Best	100	1	3	1	1	2.5	196	78	75	Very Good
	Expected	54	6	3	6	1	2.5	5.4	1.9	75	Poor

Source: Entech 2021 FS report, ENT 0721

Entech also commented on the rock mass characterisation:

- The rock mass in Edward is expected to be ‘Poor’ in most areas and ‘Very Poor’ around the historic fill and collapsed stopes. This indicates that ore development and stoping conditions may be challenging in some areas, as has already been experienced with development in the upper Edward Levels. From analysis of the drilling data, conditions are not expected to improve at depth.

- The rock mass in Empire is expected to be ‘Poor’ in most areas. This indicates that ore development and stoping conditions may be challenging in some areas, especially where veins/planned stoping interact and pillars are narrower.
- Initial mining in Empire West has encountered a mix of good and poor ground conditions. This analysis has not looked at the waste development locations but from observations underground and this data, infrastructure development is expected to be in poor to good ground.
- Initial mining in Royal West has encountered a mix of very good in waste and poor ground conditions in ore development, with clay filled shears noted in some ore development. This analysis has not looked at the waste development locations but from observations underground, infrastructure development is expected to generally be in fair to good ground throughout.
- Initial mining in Rex has encountered a mix of mainly fair to good ground conditions. The Rex andesite is notably different from other mining areas containing softer feldspars. The analysis did not look at the waste development locations but as noted, infrastructure development is expected to be in fair to good ground, although a much softer rock and one that weathers out to clays quicker than the rest of MUG. An increased level of geotechnical attention should be paid to Rex development and to collecting data to get the best understanding of the rock mass.

16.5.6 Insitu Stress Field

Stress measurement tests were conducted at Waihi for the Trio Underground Mine Project by the West Australian School of Mines (WASM) in 2010 using the Acoustic Emission (AE) method (Villaescusa and Machuca, 2010). Measurements were taken from two separate drill holes (FU163 and FU178). Results from the two holes were broadly in agreement and are given in Table 16-8.

Table 16-8. Summary of Acoustic Emissions Testing Results for the Trio Project

Vertical Depth (m)	Sample location ID	Stress Component	Magnitude (MPa)	Bearing (°)	Plunge (°)
174	FU 363	σ1	12	018	03
		σ2	8	287	07
		σ3	5	133	82
420	FU 178	σ1	23	019	00
		σ2	15	109	05
		σ3	11	285	85

Source: Entech 2021 FS report, ENT 0721

Review of the Acoustic Emissions results indicates that:

- Samples for testing were taken from two drill holes at the Trio Deposit. Due to the close proximity of Trio to MUG, results are determined to be relevant to all mining zones MUG. It is possible that some rotation of the stress field may occur from deposit to deposit due to the faulted nature of the MUG mining area. Further testing will be required to confirm this and is recommended to ensure a good understanding of the stress conditions at MUG are understood before production reaches below the 700 m RL.

- Horizontal: vertical stress ratio ranges from 2.1 - 2.4.
- The major principal stress orientation (sigma 1 = 02/019°) is shallow dipping and near horizontal in nature so will have differing effects on each of the orebodies at MUG:
    - The orientation is sub-perpendicular to the Empire and Empire West orebody strike.
    - The orientation is sub-perpendicular to the Royal and Royal West orebody strike.
    - The orientation is sub-parallel to the Rex orebody strike.
    - The orientation is parallel to sub-parallel to the Edward orebody strike.

16.5.7 Seismicity

New Zealand is located within an active tectonic subduction zone where the Australia Plate is interacting with the Pacific plate. Consequently, New Zealand experiences frequent large scale earthquakes. There is limited information which details the effects of large scale earthquakes on underground mining operations. In fact, there is more documentation on the effects on civil structures such as tunnels and caverns. This is in part due to the fact that there is little evidence of regional earthquakes causing damage to underground mines.

Laurie Richards (2012) has summarised the following points:

- There is an extensive body of literature available on earthquake damage to tunnels. This indicates that underground structures suffer minor damage only compared to above ground structures. Damage to deep tunnels in rock is generally restricted to lining distress. Tunnel portals and near surface tunnels in soils are prone to greater damage.
- Large earthquakes have occurred relatively close to the large underground openings in the Manapouri Power Station and have caused no significant damage. The recent Christchurch earthquakes were extremely destructive to surface buildings but also caused little damage to underground facilities such as the Cashmere Cavern and the Lyttelton Road and Rail Tunnels.
- Laurie Richards is not aware of any catastrophic failure of a large open cavity at depth in rock as a direct result of a natural earthquake.

Therefore, the potential for an earthquake on the North Island of New Zealand to cause damage to the Martha project is considered low.

Due to the shallow mining nature (~150-600 m depth) and given local knowledge from mining underground at nearby projects in similar rock types and at a similar depth, issues due to mining induced seismicity are considered unlikely.

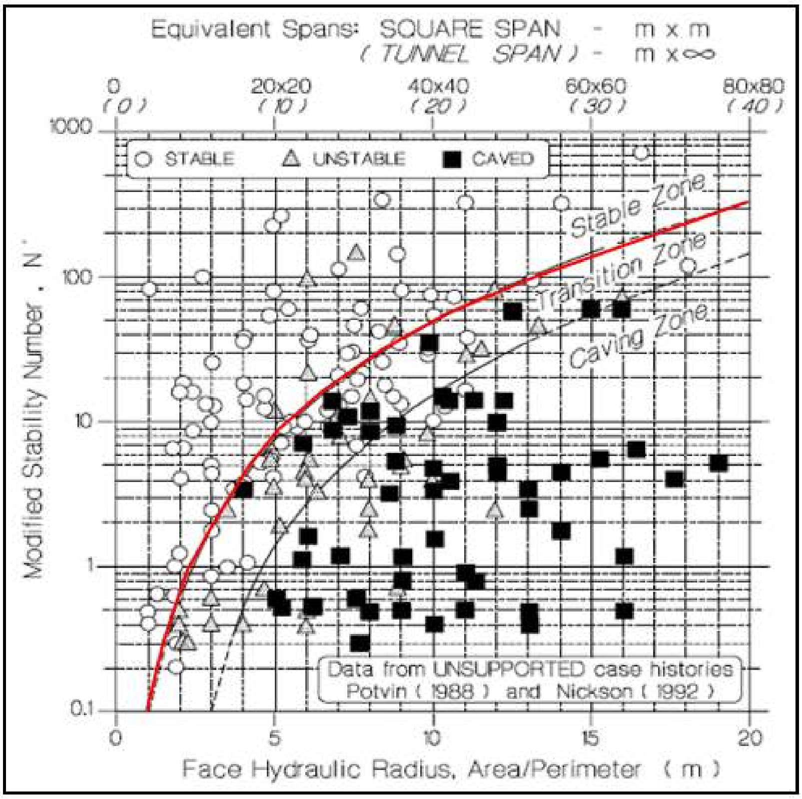
16.5.8 Mine Design Parameters

Avoca long hole mining has been selected as the mining method for much of the orebody. This method requires upper and lower access drifts to be developed with a proposed 15 m to 20 m vertical spacing between levels. The overall open stope dimensions are 15 m wide x 20 m high x 15 m long (maximum length).

Empirical methods of stope design have been employed to evaluate stability conditions. The Stability Graph Method (Mathews et al., 1981), as modified by Potvin et al. (2001), is based on more than 480 case histories worldwide and has been used to size stopes so they remain stable during mining. The Stability Graph Method plots the stability number (N') on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope wall or back on the x axis. The stability number is calculated based on the rock mass quality (Q' system), geologic structure and induced stress conditions. The stability of the stope walls has been checked using Matthews method to ensure that remote mining equipment can extract all the ore in an unsupported stope without being buried by falling ground.

The approach chosen to determine appropriate Q' values for use in stope span design was *expected* (average), *Minimum* and *Maximum.* This approach was chosen to examine the range of possible stope spans for the expected stability / instability of principally hangingwall spans for a given hydraulic radius. *Expected* values will be used for span determination for maximum unsupported stable span according to the Mathews Potvin Stability Graph (see Figure 16-13). *Expected* values are generally close in value to the arithmetic *average.*

Figure 16-13. Potvin Stability Graph for Stable Unsupported Stope Spans.



Rock mass drill hole data was separated by orebody (i.e. Martha, Empire, Edward, Rex) and then by domains (immediate hanging wall, immediate footwall and Ore zones) for each of the mining areas. For the purpose of this study, the *immediate* hot water (HW) and feed water (FW) is the rock mass that occurs within 5 m either side of the orebody (or mineable stope shape). This is the volume of rock most likely to HW and FW performance and therefor stope dilution.

The hydraulic radius (HR) was then determined for each mining area. Re-arranging the HR equation allows for the calculation of maximum allowable stope span. Allowable stope spans have been calculated for the footwall (west), Hangingwall (east) and backs exposures for the collective stoping areas at MUG and are summarised in Table 16-9 for each of the hangingwall and footwall exposures of the different veins.

Table 16-9. Calculated Maximum Allowable Strike Lengths

Mining Area	Parameters	HW	ORE	FW
Edward - All	Q' / N'	4.0 / 2.7	3.0 / 2.0	4.3 / 2.9
	HR	3.4	3.1	3.4
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**11 m**	**9 m**	**11 m**
Edward - Upper	Q' / N'	5.1 / 3.9	2.5 / 2.0	4.5 / 3.5
	HR	3.8	2.7	3.8
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**13 m**	**8 m**	**13 m**
Edward - Lower	Q' / N'	3.8 / 2.5	3.3 / 2.3	4.2 / 2.9
	HR	3.4	3.1	3.4
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**11 m**	**9 m**	**11 m**
Empire - All	Q' / N'	5.5 / 2.6	5.0 / 2.4	6.2 / 3.0
	HR	3.4	3.1	3.4
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**11 m**	**9 m**	**11 m**
1400 Empire	Q' / N'	6.2 / 4.0	5.0 / 3.3	10.4 / 6.7
	HR	4	3.7	4.8
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**14 m**	**13 m**	**21 m**
1100 Martha	Q' / N'	5.1 / 9.2	3.3 / 6.0	4.3 / 7.7
	HR	5.3	4.5	5
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**26 m**	**18 m**	**23 m**
Empire West	Q' / N'	3.5 / 2.1	0.9 / 0.5	3.2 / 1.9
	HR	3.1	1.9	2.7
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**9 m**	**5 m**	**8 m**
Royal East	Q' / N'	11.3 / 3.8	4.5 / 1.6	13.2 / 4.6
	HR	3.8	2.7	4.2
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**13 m**	**8 m**	**16 m**
Royal West	Q' / N'	8.1 / 3.6	5.8 / 2.6	13.3 / 5.9
	HR	3.8	3.4	4.5
	Known Dimension (m)	18 m height	18 m height	18 m height
	Allowable Strike Length	**13 m**	**11 m**	**18 m**
Rex	Q' / N'	3.0 / 1.8	0.3 / 0.2	5.4 / 3.2
	HR	2.7	1	3.7
	Known Dimension (m)	14 m height	14 m height	14 m height
	Allowable Strike Length	**9 m**	**2 m**	**16 m**

Source: Entech 2021 FS report, ENT 0721

Entech report some important points and observations to note regarding 'Allowable Strike Spans' in Table 16-9:

- Initial stoping in each mining area should be treated as "trial stoping" with a period of review and refinement of stoping parameters for the domains. This is a normal practice and does not diminish the confidence within the available dataset, but rather allows for confirmation of assumptions and refinements where required.
- Overall, allowable strike lengths have been assessed as fairly similar across the mining areas with the exceptions being poorer conditions being in Rex and Empire West. For Edward, Empire, Royal West and Royal East allowable spans are around 10-14 m. This is lower than the estimates made in the scoping study; and are conservative in nature so further analysis of the data and discussions will be had with OGC to work with these spans.
- Allowable spans around the historic collapsed and filled stopes are less than the virgin areas as expected, although they are overall not too different.
- Poor ground conditions resulting from features such as closely spaced fractures, oxidised quartz calcite, and faulting and clay filled structures around the veins dominate stope wall and crown behaviour, with these zones being a main source of any overbreak/unravelling issues. It is hard to model accurately from the drilling data, but further analysis can be undertaken by the site geotechnical engineers and along with adequate ore drive mapping, key areas of concern for stability can be identified and stoping extraction and extra support planned accordingly.
- The rock mass conditions in the Rex stoping areas are expected to be very poor or worse in some areas, and this is currently restricting the allowable strike length. Further geotechnical data is required for Rex from upcoming drilling and underground mapping of ore drives to confirm or hopefully upgrade this assessment.
- There is variation within the rock mass across the domains which needs to be given consideration when planning stoping on a level-by-level basis. It is expected that geotechnical mapping of ore drives will take place routinely once development is in place to enable proper characterisation of the rock mass. This will allow optimisation of stoping design dimensions based on the work and guidance completed here.
- Work by Stewart (2005) has indicated limited applicability of the Potvin Stability Graph Method when used in assessing narrow vein orebodies. However, this approach has been used with reasonable success at other narrow vein mine sites as an initial guideline and has been used successfully with a high degree of confidence at Waihi since mining of Favona and Moonlight was undertaken.

16.5.9 Dilution Estimates and Recommended Stope Spans

Dilution was estimated using the method developed by Clark and Pakalnis (1997) based on an extensive set of case histories for open stopes. The method predicts the quantity of unstable wall rock for a given rock mass quality from a given stope size. The thickness of external dilution is estimated as equivalent linear overbreak / slough (ELOS). summarises the ELOS dilution assumptions used for mine planning purposes. Similar estimates were made for open stopes in other rock types and stress conditions. Refer Table 16-10. Dilution will be heavily influenced by pre-conditioning of the rock mass which has occurred as a result of extensive historical stoping.

Table 16-10. Summary of Expected Stope Dilution for Various Mining Areas

Orebody	Mining Method	Stope Span (m) Recommended	Dilution / Wall (m)
Martha, Edward, Royal, Rex & Edward	Backfill	15	0.3
	Remnant Skins	15	0.3
	Virgin	15-20	0.3

Source: Entech 2021 FS report, ENT 0721

16.5.10 Backfill

Backfilling of all stoping voids forms a major component of the mining consent condition for underground mining in Waihi. Mining of stopes in virgin ground will utilise loose rockfill for backfilling of stope voids. This is in line with current practices on-site.

A cemented fill type will be required for mining of remnant and near-remnant mineralisation. CRF has been proposed for use and costed in this study. Some loose rockfill may be used in selected historical voids or remnant mining where further exposure of the fill mass is not expected.

CRF will be required to enable exposure of the fill mass during the life cycle of the mining front. Due to the predominantly mining sequence of the remnant mining (particularly within the Edward and Empire lodes), the CRF may need to be exposed due to adjacent stoping, as well as exposed due to underhand mining or for tunnelling through. To this end, it is expected that, on the basis of stopes being mined typically in 15 m panels, with an average width of eight (8) m, cement addition rates of ~6-7% are anticipated. Dilution of 0.5 - 1 m or more from undercutting can be expected, and less for sidewall exposure.

The site has some experience in mixing and placing CRF for specific projects however, studies and testing will be required to refine cement design requirements based upon type of exposure, exposure size, and waste rock characterisation.

16.5.11 Void Management

Historical mining took place at Martha over a period of almost 70 years, from the 1880s until early 1950s. The Martha project will be dealing with the issue of void interaction on a daily basis, and it is therefore considered the single biggest challenge to the project. Historical stopes (unfilled and filled), development and raises / shafts are present throughout the entire project and will need to be carefully monitored and managed.

Operations presently maintain a void management document specific to the MUG project area. The void management plan (VMP) is currently based upon set rules, standoff distances and guidelines around managing the interaction with historical voids. The benefit of such an approach is that rules are set, and the operation follows them. The approach the VMP is:

1. Risk-based.
2. Team-approach.
3. Controlled by ideally a single responsible role on-site such as a Voids Management Office.

16.5.12 Stope Support Requirements

Stoping conditions of the hangingwall, footwall and crown are generally expected to be very poor to poor, as described in Section 4.6, across the mining areas. This will vary based on the structures present, vein alteration and composition and degree of weathering. There are no major concerns about global stability with the current mining layout and the use of backfill in all stopes and historic stopes where required, will aid with this. The numerical modelling completed by Beck gives more detail and outlines expected global stability changes during and after mining.

The requirement for additional stope support on a stope by stope basis, will only be required in poorer ground conditions as assessed during stope design and development. The main failure modes that could be expected are:

- Sliding and slabbing failure along the intersecting joint sets present in the blocky Andesite, confined to HW and Crown.
- Unravelling and gravity driven failure in the HW and Crown, possibly FW, of stopes associated with the very poor ground and heavily structure zones.

The exact designs and requirements for stope support will be assessed on a case-by-case basis and will likely include cable bolts into the HW and Crown, as well as the use of strapping (OSRO and mesh) and shotcreting. The application of stope support is expected to be required in some areas of poor ground, and may include:

- Crown cables and other surface support from within ore drives, particularly relevant to top levels of a sloping area, stopes with development drives above and/or below any sill pillars that may be left.
- Hanging wall cables and other surface support from within an ore drive either top or bottom drives to prevent failure of HW blocks and hold conjugate structures to the orebody. Further comment on the placement of these are given below.
- Resin injection/pressure grouting into historic fill and slender pillars between historical voids and planned stopes.

In most locations limited access for installation of effective cable bolts may occur as specific cable bolting development is not planned. That is, access above the HW and crown of the stoping area to drill downhole cable bolt holes to gain the most benefit in stabilising the rock mass cannot be achieved. This limits the effectiveness of any designed cable bolting array at arresting and stabilising the entire HW of a stope with stability concerns. Rather drilling and installing cable bolts are described above will be undertaken from the ore drives. They will still be effective in arresting potential failures at the drive level and around brows, preventing further unravelling to subsequent stopes. The alternative method for controlling instability is limiting open strike spans of stopes.

Provisions for cable bolting and other additional support (straps, shotcrete etc.) for stope brows should be made and cable bolting standard developed in higher risk areas including but not limited to:

- where ground conditions are assessed as “Very Poor” or worse;
- where the width of the ore drive exceeds 6 m;
- mining below the 700 m RL due to modelled increase in stress at depth;
- extracting stopes in pillars between historic voids and adjacent to historic voids, due to the potential for the rock mass may become unconfined.

The requirement for cable bolting of stope brows, and potential standard cable support patterns, should to be assessed once development is in place and trial stoping has been reviewed using the different methods. A conservative approach needs to be taken for the initial stopes outside the current modified AVOCA method used at Waihi.

16.5.13 Pillars

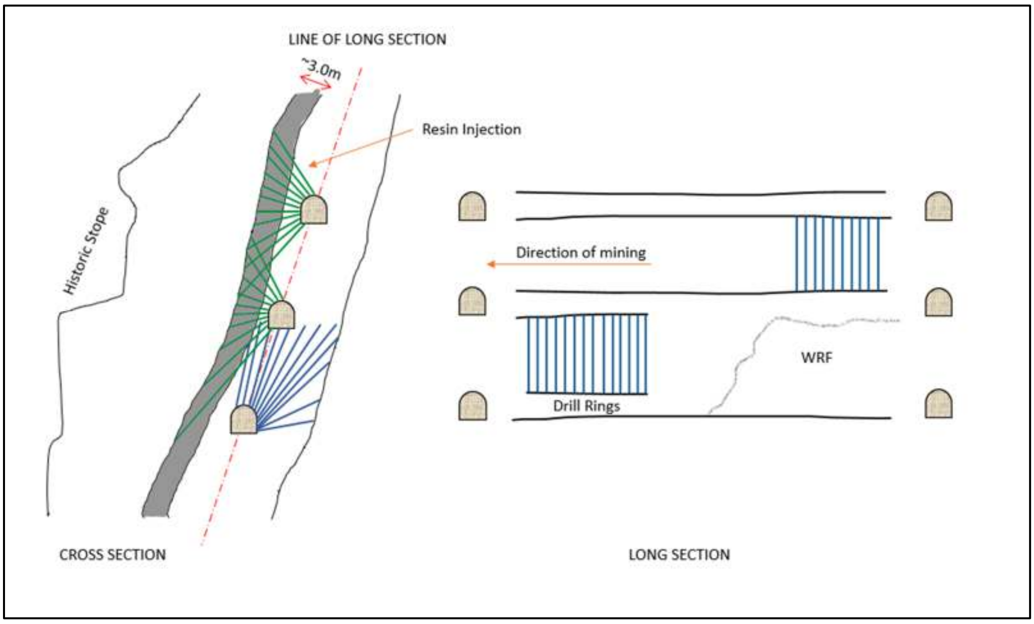
At this stage, the following pillars are or will potentially be designed at MUG:

- Rib pillars in stoping levels between stope panels are only planned in ore that is uneconomical to stope. Further rib pillars will be left to cater for unexpected mining conditions such as excessive dilution and control overbreak from precious firings as has been done in the past at Waihi. The location and size of these will need to be assessed on a case-by-case basis in this instance.
- Crown pillars between stopes and overlying development or historic voids to control unravelling and potential chimney caving above.
- Standoff pillars (a sort of rib pillar) between planned MUG stopes and historic collapse or backfilled stopes. These are stability pillars to allow extraction of the ore in place adjacent to the historic voids (further explained below in Section 5.9.1). At this stage a 3.0 m metre pillar width is being proposed, further refinement of this will be on-going as more rock mass data becomes available from development and drilling, and back analysis of pillar performance is undertaken.

If any pillars were required a detailed assessment would have to be undertaken on a case-by-case basis to determine the appropriate pillar dimensions.

In locations where stoping will occur adjacent to a historic stope which is either open or filled with previous collapse material a standoff pillar is to be left between the footwall of the historic and new stope void, see Figure 16-14.

Figure 16-14. Example of Standoff Distance between Historic & MUG Stopes.



Source: Entech 2021 FS report, ENT 0721

An approximate 3.5 m standoff pillar will be left in place along the footwall of the stope to provide stability. Prior to stoping, resin injection to consolidate the rock mass in the pillar will occur along with observation holes drilled through the pillar to determine the in-situ condition of the stope. The development drive will be orientated along the Hangingwall to allow for parallel drilling and remove the need for stab holes. A review will be completed to assess the requirement of reinforcement in the pillar (theoretically installed after the completion of resin injection).

16.5.14 Ground Support and Stability

During 2020 Entech was requested to undertake a geotechnical analysis from rock mass characterization data collected from a number of representative locations at the early stages of development of the Martha Underground Mine. The full report should be referred to for all analysis and justification of the ground support system recommended, this chapter will only summarise this work.

The ground support design was based upon a reinforcement pattern using friction stabilizers and weld mesh. A detailed reinforced block analysis was undertaken considering a potentially worse-case unstable scenario for two critical excavation orientations. The failure mechanism in consideration was static loading and the reinforced block analysis indicates that stable conditions are very likely at Waihi.

From the analysis the calculated out of balance forces were less than 20 KPa for all potentially unstable block geometries. As a comparison, friction stabilizers are capable of providing 40-50 KPa of radial pressure at the boundary of the excavations. Similarly, the apex height (which can be linked to a maximum depth of failure) defined by the potentially unstable wedges was limited to 2.5 m .

The following ground support recommendations were made from the study:

- A 1.2 m x 1.2 m pattern of 2.4 m long. 47 mm diameter galvanized friction stabilizer in conjunction with galvanized weld mesh is considered to be stable for the mining development excavations at Waihi. The assumption is that excavations are not subjected to excessive corrosion.
- Furthermore, a pattern of 1.1 x 1.4 m pattern of 2.4 m long, 47 mm diameter friction stabilizer in conjunction with 50 mm in-cycle sprayed fibrecrete is considered to be stable for the mining development excavations at Waihi.
- In order to minimize instability, the back of the excavations be designed with a semi-circular shape. This would minimize the risk of unravelling, as the analysis also detected a large number of small wedge geometries.
- Although the mapping did not indicate alteration or weathering, the size of the blocks suggests than unravelling may be more likely than large wedge failures. Therefore, shotcrete support may also be indicated for surface support at Waihi.
- On-going geotechnical inspections are required to determine any instability from large scale geological features that may either fail by unravelling or form larges blocks which may require cable bolting.

16.5.15 Geotechnical Numerical Modelling

Numerical modelling using Abacus software was undertaken by Beck Engineering in 2020 and 2021 with geotechnical data supplied by OGC and Entech. The study was broken down into three main areas, MUG above and below the 700 m RL and the Rex area. In addition comments were provided on rock damage forecasts, groundwater and main infrastructure. These are summarised below.

<u>Upper levels (above ~700 m RL)</u>

- Generally, favourably mining conditions with low to moderate stress conditions during the early stages of production for 2021 to 2023 in the upper levels:

- Isolated areas of moderate rock mass damage in proximity to existing stopes.
- Mining in proximity to historic stopes and old mine workings is a hazard that will require careful management. This includes identification of voids (stopes and airleg workings), perched water and wet rockfill, backfill type in historic stopes and potential for unknown voids.
- The stoping front should be ~45 degrees to minimise the number of stopped which future access must traverse as well as stress management. The current sequence generally targets a 45-degree stoping front.
- Converging mining fronts and diminishing central pillars generally do not result in significant ground control problems in the model forecasts above ~700 m RL.

Lower levels (below ~700 m RL)

- Mining is forecast to become difficult below ~700 m RL. Increasing deformation and some rock mass damage in working areas is caused by increasing stress due to depth, the increasing excavation ratio of the mine, interaction between veins at depth, as well as unfavourable features of the current mine plan. These features includes:
    - Diminishing pillars, including central close out pillars between stoping fronts, and diminishing sill pillars with previously mined stoped panels above.
    - Deformation and stress concentration caused by small (15 m and 20 m) sublevel spacing is considered a vulnerability in the current mine design.

Rex Lode Specific

- No significant issues identified
- The shallow depth, low stress and rock mass conditions for this region are favourable.
- The central retreat with the diminishing central pillar is considered unfavourable, however the planned sequence does not cause any significant adverse effects for stability or deformation in the model forecasts due to the otherwise favourable conditions

Overall Damage Forecasts

- Most problems will be manageable with additional ground support, redrilling or rehabilitation as required. However, this rework may to result in production delays which has the potential to necessitate mining of other stopes out of sequence to meet ounce profiles and business targets. Developing a sound mining sequence to mitigate ground control problems and delays will ultimately lead to a safe, reliable and schedulable mine plan.
- The highest level of predicted damage is in the drive backs. This is due to stress concentration between drives due to the locally high extraction ratio between the closely spaced levels
- The Empire, State and Martha lodes confluence below ~650 m RL. Yielding and some rock mass damage in stope pillars between lodes and in areas of future stoping is forecast in the model.
- Given the orebody and host rock conditions, it is expected that squeezing and deformation will be the most problematic ground control problems for active workings.

Infrastructure Modelling Results

- No significant damage forecast.
- Minor stress and damage concentrates in the pillar between the stacked level accesses due to the short level spacing

Groundwater impacts due to mining

- Overall, the model forecasts match the measured data, including magnitude of measured subsidence as well as the patterns of movement.

Numerical Modelling Recommendations

- Review the current mine design, including:
    - Avoid stacked level accesses
    - Re-position the lower declines to enable end access, where possible
- Review the current mine sequence including:
    - Remove diminishing pillars/converging mining fronts where possible, particularly below 700 m RL
    - Consideration of a top-down and centre-out sequence (or end access) below ~700 m RL to avoid diminishing pillars. This requires careful consideration with the backfill system and may not be possible.
    - Target continuous stoping retreat on each level, without re-accessing previously mined levels for remnant stoping. A lead-lag of 1-2 stopes between neighbouring lenses is favourable
    - Revise the sequence to eliminate remnant mining and re-accessing previously stopped areas
    - Maintain 45-degree stoping fronts
- Continue development of the comprehensive void management plan. The mine must carefully consider the proximity of future mining with historic mining, including old airleg workings. This includes identification and management of voids, potential for perched water (inc. wet rockfill) and backfill type in historic stopes
- Ongoing monitoring to confirm forecasts.

16.6 Mine Access

The mine will be accessed via the Favona Mine portal, Trio and Correnso mines and the 920 and 800 RL exploration drives as shown in Figure 16-1. Two portal breakthroughs have been completed in the south western corner of the MOP and are being used for ventilation and secondary egress purposes, refer Figure 16-15. Ground support to secure the pit highwall above the portal has been completed and consists of shotcrete with chain link mesh secured to the highwall face to contain minor scats from falling into the portal area.

Figure 16-15: Martha Underground Overview Plan View

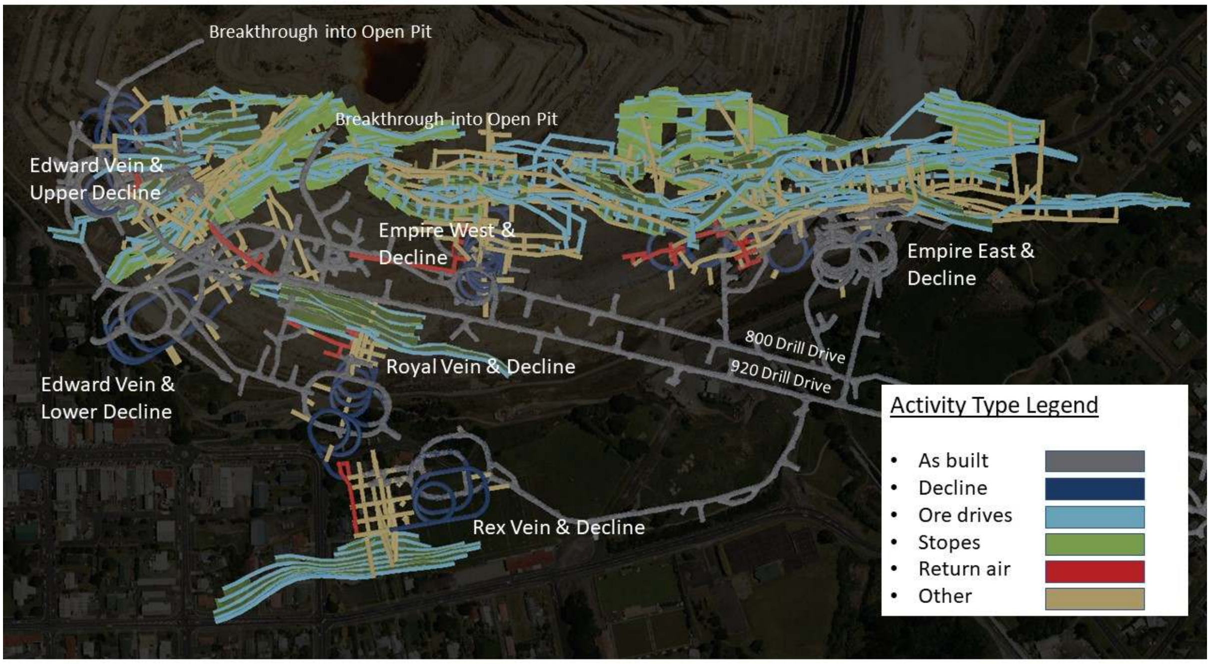
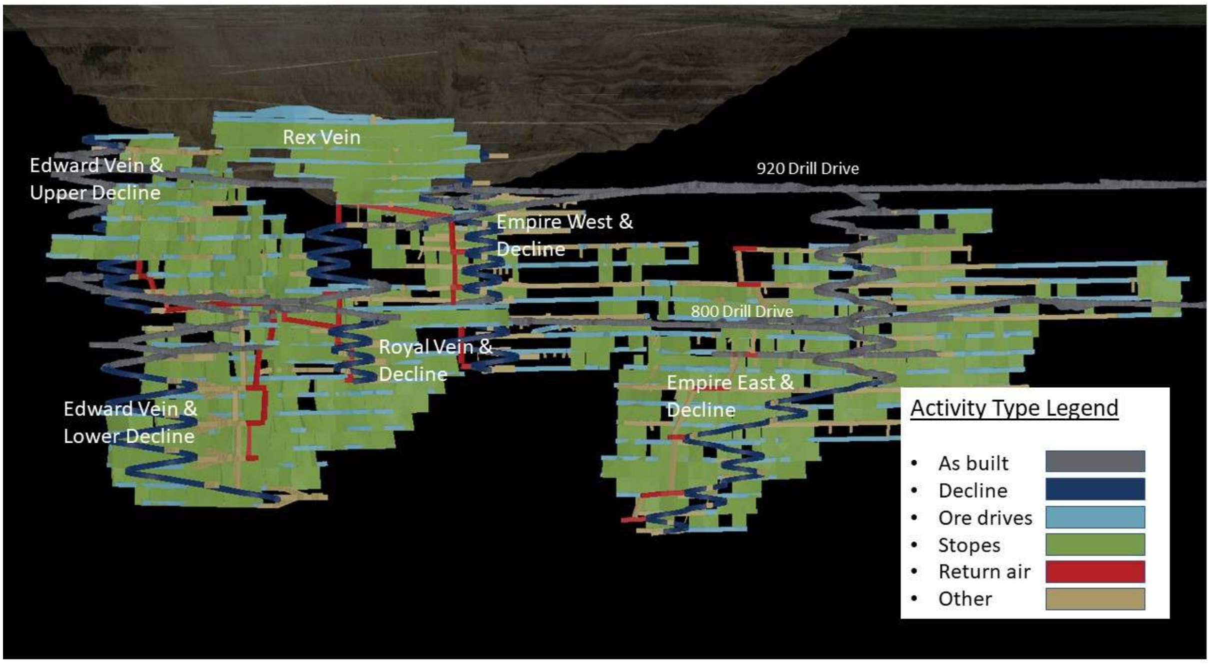


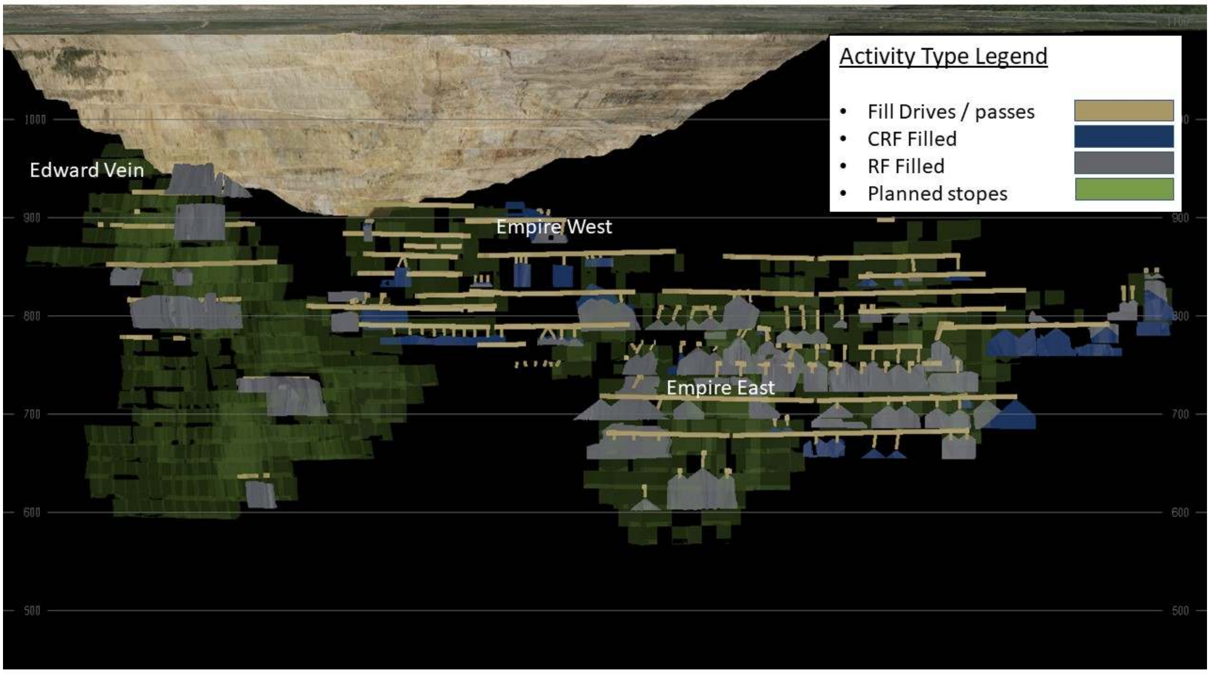
Figure 16-16: Martha Underground Overview Long Section



Based on the proposed mining method and equipment, historical experience and orebody geometries, the development strategy for all underground operations involves mining of declines for access to five main stoping blocks. Access drives will be mined to develop drilling and loading levels, generally intersecting the orebodies centrally. Access drives will be spaced at 18 m vertically over the height of the mine. Ore drives will be developed in both directions along strike from the access drives. Stockpiles will be mined off the decline and in levels for truck loading. The development design used for the Feasibility Study ties in with current operating practices at Waihi.

Key differences with current operating practices involves the development of footwall drives, crosscuts and a pass systems in selected locations mainly confined to Edward, Empire east and west to backfill the historical workings with CRF or RF. Cross cut spacing is generally at 20 m to 25 m spacing. Historical stopes are backfilled to provide both regional and local stability. The extent of this planned development is shown in Figure 16-17.

Figure 16-17: Martha Underground Overview – Fill Development Long Section



16.7 Stoping

2016 and 2017 and confirmed by Entech in 2018 and by OceanaGold in 2020. Four mining methods are proposed for the mine:

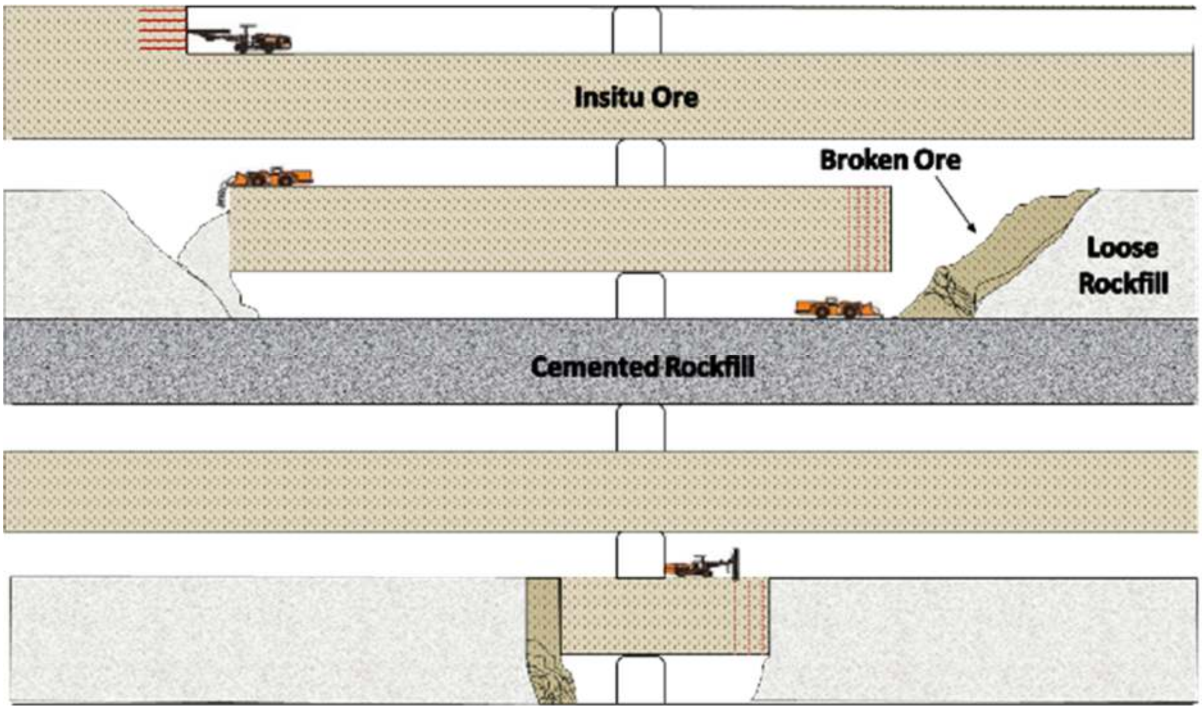
1. Modified Avoca with rockfill in virgin (previously unmined) areas.
2. Modified Avoca with rockfill in remnant areas adjacent to collapsed stopes separated by an intermediate pillar.
3. Modified Avoca with rockfill in remnant areas adjacent historical stopes filled with engineered fill (CRF / CAF)
4. Bottom up side ring method with CRF/CAF/RF where skins adjacent to historical backfill are extracted.

This is discussed further below:

16.7.1 Virgin Areas

Much of the Mineral Reserve can be extracted using the modified Avoca mining method, refer Figure 16-18, similar to the methods employed at Favona, Trio and Correnso. The modified Avoca method with RF is a semi-selective and productive underground mining method, and well suited for steeply dipping deposits of moderate thickness. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralisation. Access is required centrally within stope panel to allow for mining to progress longitudinally. Down holes are drilled and loaded with explosives and the stope is blasted, with broken material falling to the bottom drive for extraction. Conventional bogging of the broken material can occur until the brow is exposed. At this point, remote controlled LHD's are employed required to remove the blasted material from the stope. Stope structural support is provided through a combination of cable bolting and uncemented RF. It is not planned to leave rib pillars unless there is limited access to the sub-level or recommended to maintain overall mine stability.

Figure 16-18: Modified Avoca Mining Method



Source SRK Consulting Ltd

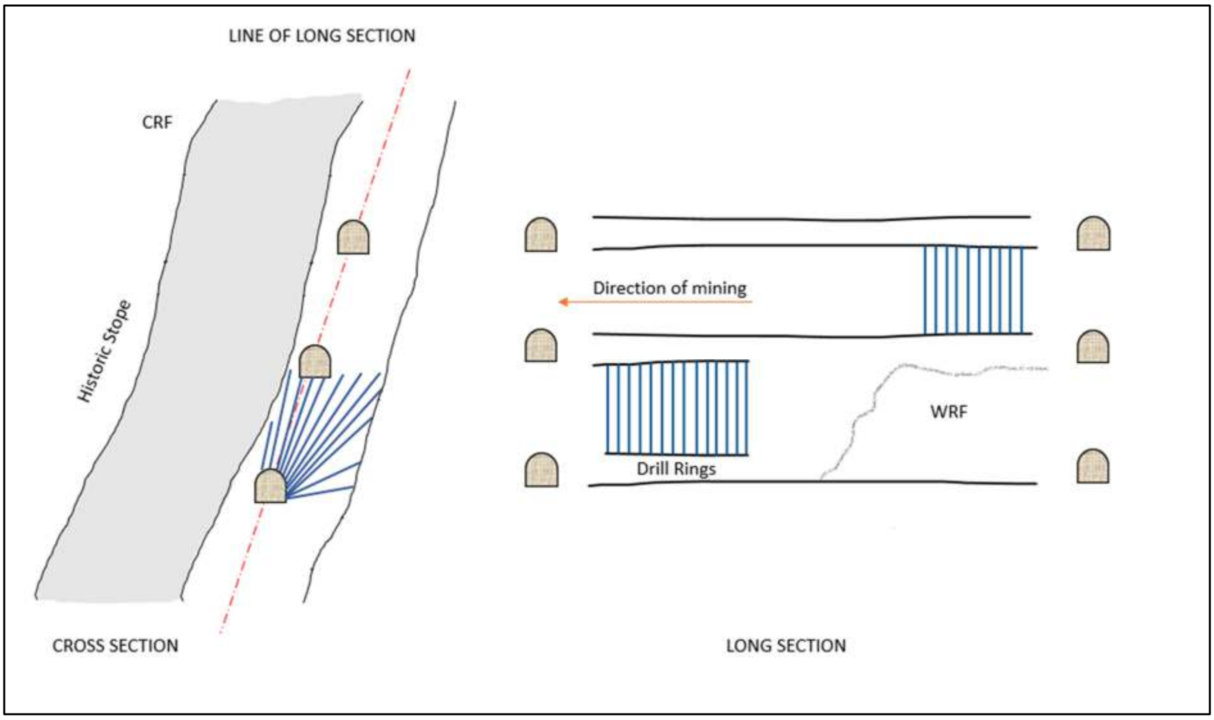
16.7.2 Remnant Areas

A small proportion of the Mineral Reserve will involve the extraction of remnant skins in the footwall or hangingwall of previously mined (historical) stopes, or the extraction of both remnant skins. Historical backfill may also be mined and experience with OP mining shows this material may be above the cut-off. However, as it is currently classified as Inferred Resource it is not included as Mineral Reserve. Following detailed studies over the last nine years, three methods are proposed for the extraction of remnant areas, adjacent to historic workings.

<u>Modified Avoca against CRF</u>

Whereby the historic stope is backfilled with CRF prior to stoping and the remnant skin is extracted by conventional modified Avoca using RF in a bottom up sequence that exposes the CRF in one of the stope walls. Historic open stopes will be first backfilled with CRF, then Modified Avoca (bottom up) stoping is completed against the CRF to allow complete extraction of the skin of ore against the historic stope boundary. A shorter open strike will be required for stability purposes when compared to Modified Avoca in virgin ground. And will involve development situated against the hangingwall of the historical stope, parallel drilled holes to CRF contact, no stab holes into Hangingwall and stepping hangingwall hole in and allow stope to fire back to CRF contact. This is shown in Figure 16-19.

Figure 16-19: . Modified Avoca against Historical CRF Filled Stope



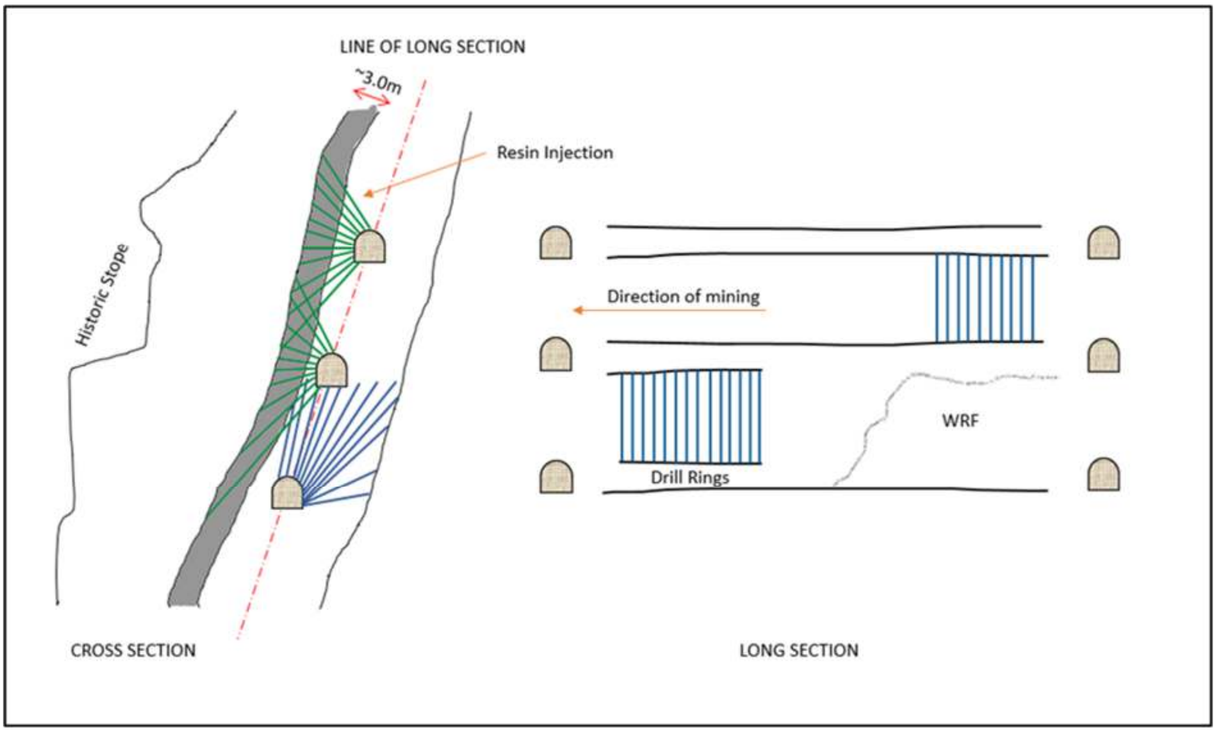
Source: Entech Consulting Ltd

Modified Avoca adjacent collapsed historical stope

Where backfill of the historical stope with CRF is not feasible and a stand off from the historic wall of 3.5 m is maintained with lower estimated recoveries, higher dilutions.

The intent of this proposal is to detail the situation in which mining will occur adjacent to a historic stope which is either open or filled with previous collapse material (Figure 16-20). An approximate 3.5 m standoff pillar will be left in place along the footwall of the stope to provide stability. Prior to stoping, resin injection will be assessed to consolidate the rock mass in the pillar will occur along with observation holes drilled through the pillar to determine the in-situ condition of the stope. The development drive will be orientated along the hangingwall to allow for parallel drilling and remove the need for stab holes. A review will be completed to assess the requirement of reinforcement in the pillar.

Figure 16-20: Modified Avoca against Collapsed Historical Stope



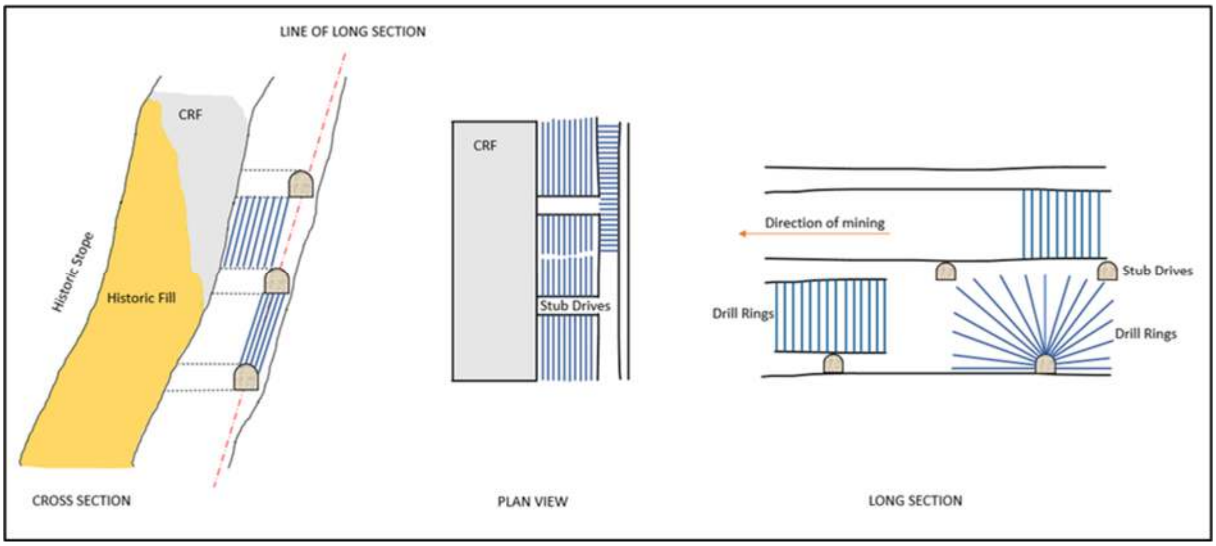
Modified Avoca against historical filled stope

A modified Avoca method remote side ring method where the remnant wall rock is extracted against the historical backfill. Lower recoveries and higher dilutions apply to this method which is described below.

In the situation where the historical stope has been filled, the intent is to mine the skin of ore adjacent to the footwall of the stope and have the original backfill remain in-situ using modified Avoca (bottom up). Development will be planned along the footwall with Stub drives perpendicular. Observation holes will be drilled from these Stub drives to confirm the presence of fill in the historic stope.

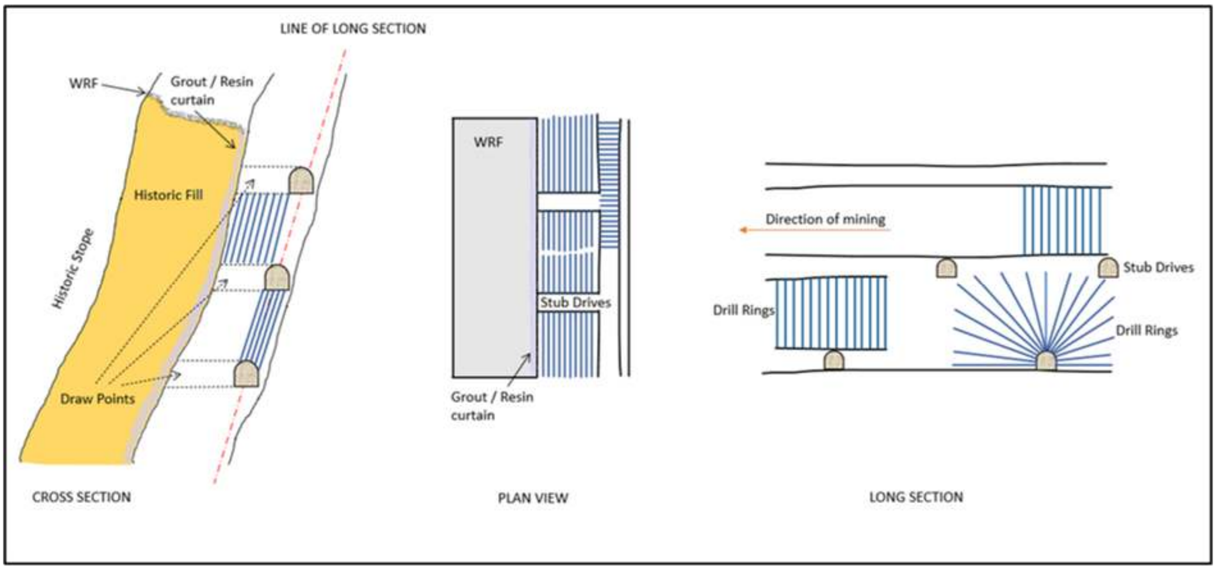
If only partially filled (and practical and safe to do so), the void is to be filled with CRF (Figure 16-21) to stabilize the existing material. If CRF is not achievable, the installation of a resin curtain along the footwall to consolidate the fill material is recommended (Figure 16-22). In both situations, the extraction of very short strike lengths (to be determined) are recommended to ensure the stability of the historic fill and the requirement of planned rib pillars for stability purposes should be considered. This method is largely employed in the Empire west area and comprises a very small proportion of the Mineral Reserve.

Figure 16-21: V1 Modified Avoca against Historical Filled Stope



Source Entech Consulting Ltd

Figure 16-22: . V2 Modified Avoca against Historical Filled Stope



Source Entech Consulting Ltd

With all stopes voids being filled after extraction, the number of open stopes at any one time is constrained by the maturity of the mining front and sequencing on the stacked levels.

For this method, mining of mineralised historical backfill will be challenging and is a rare occurrence within mining globally. Mining spans will be limited to reduce the likelihood of instability, and mining will occur remotely to remove personnel and equipment from exposure to poor ground. It is recommended that this mining method be treated on a trial basis initially, with a rigorous period of review before broader implementation. Mining may occur in a top down approach, so that the stope backs can be controlled with an engineered fill (i.e. cemented rockfill). Trial mining of mineralised backfill should occur before wider implementation takes place.

16.8 Lateral Development

Lateral development includes interlevel ramps, level accesses, stope accesses, and short connecting drifts for ventilation. The interlevel ramp system will be a continuation of the trio and Correnso access decline and will have the same dimension (5.0 m wide by 5.5 m high with an arched back) and the same maximum gradient (14%). Level accesses and stope accesses will be 5 m wide by 5 m high with a flat back and will be mined higher at the muck bays to allow the haul trucks to be loaded by the LHD. The short connecting drifts for ventilation will be 4.5 m wide by 4.5 high with a flat back.

Interlevel ramps and level accesses will be located in the footwall and have been designed to avoid crossing fault zones to the maximum extent possible. Stope accesses are oriented perpendicular to the strike of the orebody.

The lateral development is sized for the operation of the mining equipment fleet that has been selected for the operation. The stope accesses are wide enough to allow 50 t haul trucks to reverse into the stopes for direct tipping of backfill, provided that the back is mined higher (> 6.5 m) at the tipping point to allow the truck bed to be raised to the full height. The development profiles include allowances for ventilation ducting and services.

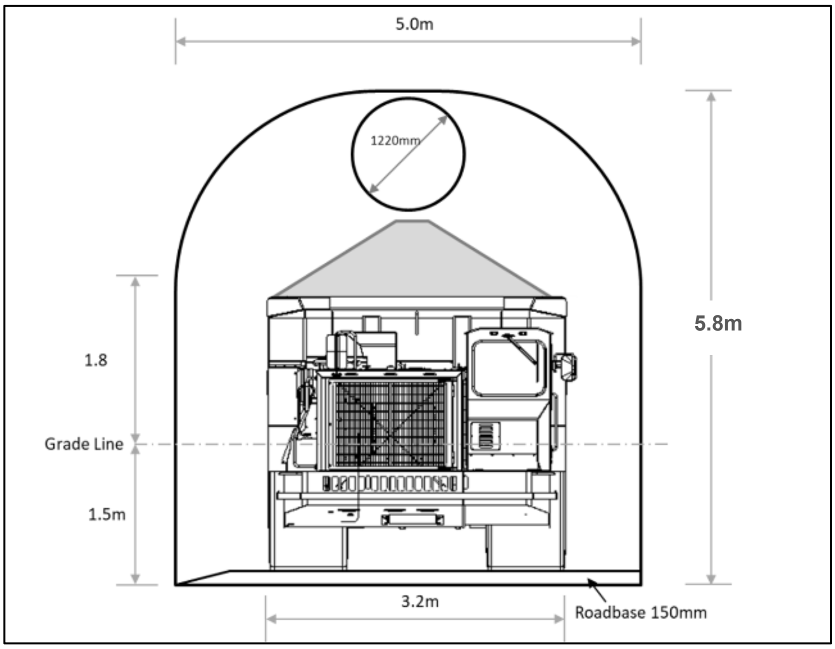
The development profiles outlined match the current development as-builts being constructed at Martha. The development types and their profiles are listed in Table 16-11.

Table 16-11: Martha Underground Development Profiles

Development Type	Size (mW x mH)
Ramp Profile	5 x 5.8 (m) Arch
Escapeway Profile	4.5 x 5.5 (m) Arch
Sump Profile	5.0 x 4.5 (m) Arch
Bogger Access Profile	5.0 x 5.0 (m) Arch
Ore Drive Profile	5.0 x 5.0 (m) Arch
Truck Access Profile	5.0 x 5.6 (m) Arch
Truck Tipple Profile	5.0 x 7.2 (m) Arch
Stockpile Profile	6.0 x 6.0 (m) Arch
Narrow Vein Profile	3.0 x 3.2 (m) Arch
Cut And Fill Profile	4.5 x 4.5 (m) Square
Fill Drive	4.5 x 5.0 (m) Arch
Workshop Bay	8.0 x 6.0 (m) Flat Arch

All ramp development was designed at a maximum 1-in-7 gradient. Decline and capital development requiring truck access were planned to be developed at a 5.0 m wide x 5.3 m high profile to allow the use of 50 t-sized underground trucks while matching the current as-built development used for access to the deposit. As shown in Figure 16-23.

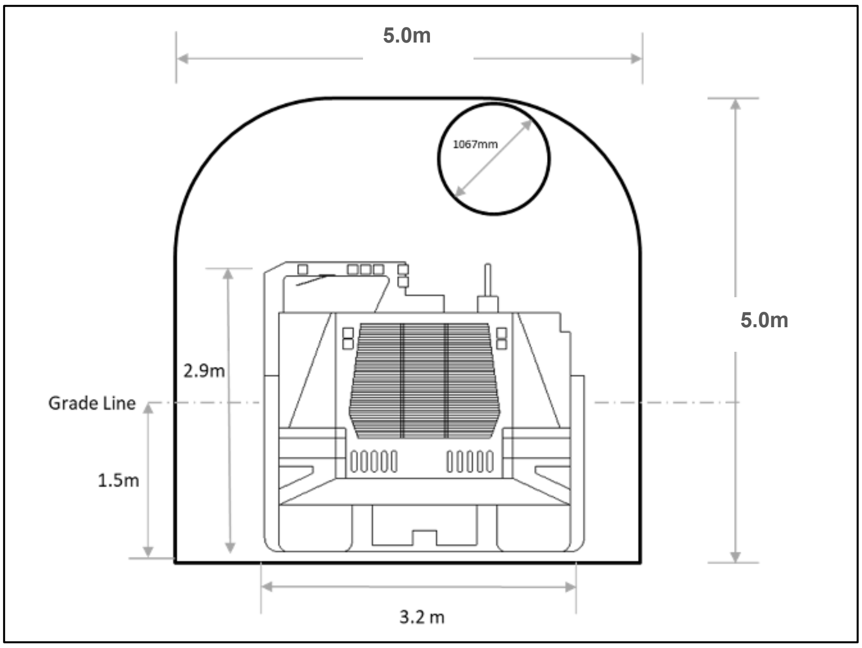
Figure 16-23: Schematic Cross-Sectional Profile of Decline with TH551 Truck



Source: Entech Pty Ltd.

Level access beyond truck access areas (i.e. in-level from truck loading stockpiles) and ore drive development have been designed at 5 mW x 5 mH with one 1 m radius rounded shoulders. This profile allows the efficient use of 7 m^3 sized loaders (e.g. Sandvik LH517i or equivalent), as well as providing suitable room for adequate installation of ground support with twin boom jumbos using split feed booms. A schematic representation of this profile with the Sandvik LH517i loader is included in Figure 16-24.

Figure 16-24: Schematic Cross-Sectional Profile of Ore Drive with LH517 Loader



Source: Entech Pty Ltd

16.9 Vertical Development

Longholing will be used for the main ventilation openings and escape raises. There are no vertical raisebores planned to surface, with lateral development into the pit acting as the primary ventilation links. Consents do provide for a single raisebored shaft to the pit if required. A 4,5 m diameter raisebored shaft between 800 and 920 m RL was completed in 2020.

Conventional drop raising will be used to establish ventilation connections between level access drifts. The ventilation drop raises will be 4.5 m wide by 4.5 m long by approximately 20 m high.

16.10 Backfilling

16.10.1 Waste Rock

Waste rock from the underground will be used as non-cemented fill in Avoca stopes when possible. If a stope is not available at the time the waste is mined, it will be hauled to the surface and placed in the UG stockpiling area where it will subsequently be either hauled via OP trucks to the Polishing Pond waste rock stockpile or used to make CRF. The waste rock is hauled to the surface primarily during the pre-production period prior to stopes being developed and will be reclaimed at a later date for rockfill for the Avoca stopes or to make CRF.

16.10.2 Paste Fill Trade-off Study

Introduction of paste fill into the mine plan was investigated by Outotec and AMC and was found to be technically feasible to manufacture.

AMC reviewed the Outotec report in 2019 and were commissioned to look at the practicalities and logistics of placing paste fill and concluded that:

- Supply and production of paste fill prepared from Waihi tailings is technically feasible but the loss of paste into old workings was not addressed.
- Waihi tailings respond better to slag cement binders than GP grade cement, but slag binders are not available domestically in New Zealand.
- Paste fill has the lowest operating cost but the loss of excess paste fill into old workings could negate that benefit.
- Paste fill has a significantly higher capital cost being nearly double that of an engineered Cemented Aggregate Fill (CAF) system.
- All backfill operating costs are higher than usual due to the need to pre-fill old voids prior to access, drilling and extraction of new stopes followed by filling of those new stopes. The total backfill operating cost is spread over a lower tonnage than comparable green field mining operations.
- Outotec have demonstrated that paste fill is technically feasible to produce but the risk of uncontrolled loss of paste fill during placement in remnant stopes represents a potential fatal flaw to the method.

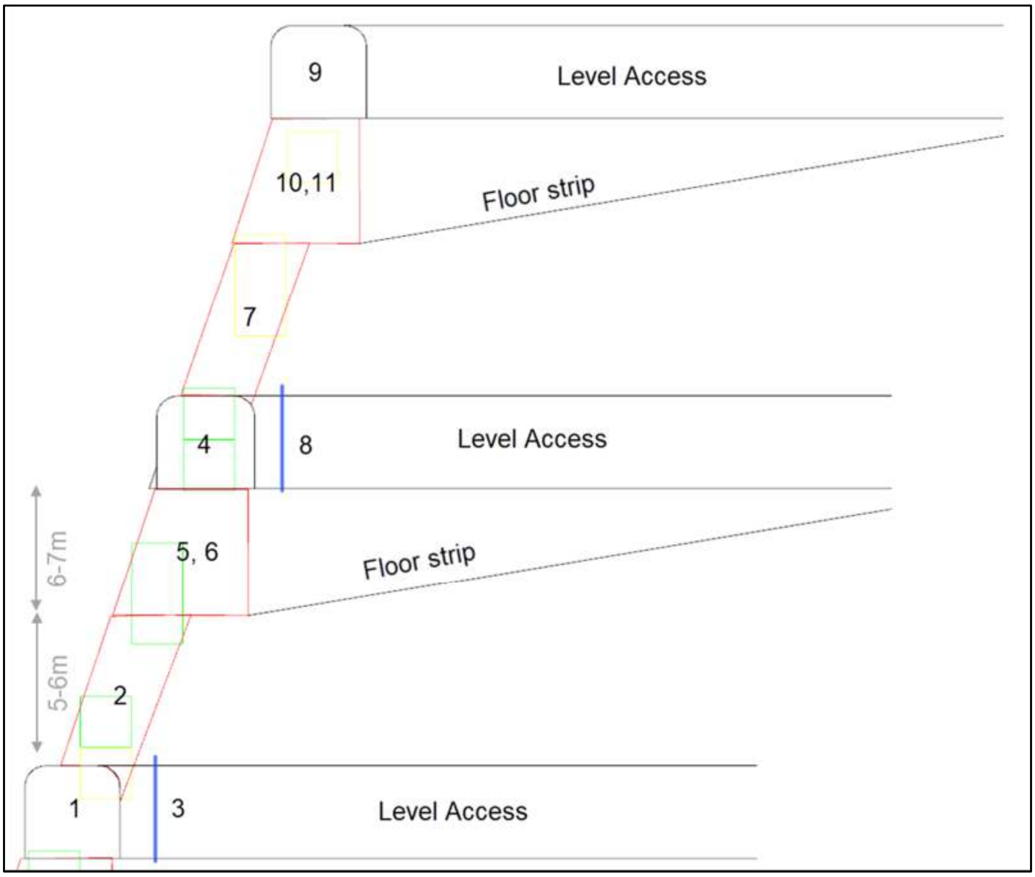
OGC studies were carried out in September 2019 and reviewed the options for Avoca style mining and other benching fill using a paste fill method for mining of MUG primary stopes (stoping areas unaffected by remnants). The studies concluded that:

- Traditional approach at Waihi is bottom up modified Avoca with waste fill, method suits orebody and generally performs well

- Paste fill reviewed as potential replacement for waste fill in stopes due to potential lack of tailings storage area on the surface
- Modified Avoca method utilising a paste identified as unsuitable due to scheduling constraints and costs (bulkhead construction and paste cure times)

An Alternative benching method was proposed to extract the primary stopes, this method schematic and sequencing is shown in Figure 16-25. The benching method aims to utilise similar sub level spacing as modified Avoca to keep development costs down via a combination of upholes and downholes. The method (bottom up) can utilise a lower paste cement content as there is no crown or sidewall exposure (floor only) and the method focuses on extraction of entire orebody along strike (at a reduced stope height) to mitigate against delays such as cure time and bulkhead construction (1 bulkhead per level vs 1 every 15-20 m).

Figure 16-25: Alternative benching Method to Use Paste Fill



The study concluded that for benching:

- costs were lower in wider sections of orebody due to reduction in planned dilution,
- the revenue was lower than Avoca due to lower mining recoveries,
- profitability varied significantly based on vein width and angle,
- near vertical/near drive width veins are slightly more profitable than Avoca but as vein width and angle decreases, benching becomes unfeasible.
- Overall viability of benching is driven by orebody width and dip. Too narrow and flat = introduction of dilution. Too wide = underbreak likely due to inability to strip out blind holes to width (low void ratios etc). The conclusion was that the disadvantages from benching outweighed any advantages and the method was discounted.

Opportunities were investigated to use paste for backfilling historical and remnant workings. Advantages of using a paste fill were considered to be:

- Quality controlled, engineered fill, not reliant upon UG operators to mix correctly to ensure adequate strengths etc
- Less variation in flow and strength properties versus other flowable mixes
- Higher quality cured fill resulting in ability to use lower factors of safety and lower target design strengths
- Reduction in required surface tailings storage capacity
- Reduction in heavy equipment required to complete fill cycle, no boggers/trucks required to transport and place fill in stopes, machinery can be allocated elsewhere or alternatively not required resulting in reduction in diesel particulate matter emissions
- Eliminates the need to expose personnel and equipment to hazards associated with working around open holes
- Low beach angle allows for tight filling

The disadvantages of using a paste fill were considered to be:

- liquid nature means containment of material requires walls to be constructed, in areas that are inaccessible or costly/timely to access
- paste filling in and around old workings is not a common practice within the mining industry, and
- high upfront capital costs.

Cash flow analysis was conducted on a paste system and CAF / CRF system to determine viability of using paste. Various methods of placing paste into the historical workings were evaluated including winzes into the historical stope, staging of the paste filling and uncontrolled filling each with an assumption of losses of paste into the historical workings. The study concluded that:

- CRF is cheaper than both paste fill options (total cost and cost per m^3 placed);
- capital cost for CRF consists largely of development required for fill access on levels;
- development of fill drives into remnant voids also provides contingency for stope inspection and remediation of potential failure over time (access for teleremote boggers);
- if remnants have had large amount of failure over time, paste costs will increase as development will be required into voids regardless to remove material;
- paste will require a thorough remote inspection system via inspection/observation holes into old development workings to monitor flow of material along drives, with particular attention to drives connected to remnant shafts and the open pit;
- remnant areas are narrow and generally long strike distances (>200 m). Compared to a traditional paste setup (one fill point in a transverse stope), numerous fill points are required along strike for remnants at MUG to ensure tight filling;
- industry advantages of paste fill in terms of recovery versus other forms of backfill is not particularly relevant to MUG due to the mining method/layout;
- a low beach angle of paste allows for tight filling of wider/flatter zones however the transverse nature of the CRF fill drives and tipping from two levels above will allow for CRF to achieve the same result in terms of tight fill in remnant areas, and
- Historically, lower dilution from paste vs CRF has been achieved, particularly in crown exposure areas. For MUG, fill exposure will be primarily in the sidewalls. however if properly managed, CRF can achieve similar results

Paste fill has been discounted as a method of backfill for MUG.

16.10.3 CRF, CAF and RF

Backfilling is expected to be required for the MUG in the following circumstances:

- RF for the Avoca bench mining method;
- CRF filling of the historical voids which are expected to be encountered; and
- Both RF and CRF are required for remnant mining areas.
- CRF or CAF for crown pillar recoveries.

The use of CRF is important for the remnant mining areas, since in some cases backfill will be exposed adjacent to, or above, stopes being mined. In other cases, CRF will be required to fill portions of historical voids to allow mining through the CRF at a later stage for access, and to stabilise voids below the active mining horizon to reduce the risk of sudden and unexpected backfill subsidence into historical voids at depth.

Rockfill will consist of MUG development waste, which can be stockpiled on surface and re-used as backfill, or (in some cases) moved from the development face directly to a void which requires filling.

AMC were commissioned to review the use of CRF and CAF in the underground mine which included laboratory testwork on large scale samples. The following is a summary of the AMC report which is included in the Appendices.

AMC evaluated four cemented rock and aggregate fill options as shown in Table 16-12.

Table 16-12: Evaluated Martha Underground Backfill Options

Label	Description	Cement delivery
CRF Agi	Waste rock trucked from development headings and mixed by a loader with cement slurry and tipped into a stope.	Surface prepared cement slurry and delivered by agitator to mixing sumps underground.
CRF Slick	Waste rock trucked from development headings to spray bar system and tipped by loader into a stope.	Surface prepared cement slurry delivered by slick line to central spray bars underground and sprayed onto trucks.
CAF Trucked	Graded aggregate produced on surface from Waihi waste rock and hauled underground to be tipped by loader into a stope.	Surface prepared cement slurry sprayed on to back of trucks on surface.
CAF Rock Pass	Graded aggregate produced on surface from Waihi waste rock and delivered UG via choke fed 1.0 m diameter raise to fill station underground.	Surface prepared cement slurry delivered by slick line to spray bars underground and sprayed onto trucks.

Source: AMC, 720018 MUG RF Study

Where old stopes have economic margins, they will be filled with cemented fill and the margin extracted by long hole mining using the Avoca method. This will result in hangingwall or footwall exposures with dimensions of either 18 m or 36 m high and typically 25 m along strike. The exposure will either be in the footwall, vertical or in some cases overhanging from the hangingwall. Vertical fill exposures may also occur during longitudinal retreat mining. For orebody widths of up to 15 m, 400 kPa will provide sufficient strength to maintain stability.

The design fill strengths for the expected range of stope lengths and heights exposed by Avoca mining are shown in Table ES.2.

Table 16-13: Design Strengths for Range of Spans & Heights (kPa)

	Exposure width (m)							
Exposure Height (m)	**5 m**	**10 m**	**15 m**	**20 m**	**25 m**	**30 m**	**35 m**	**40 m**
18 m	200	300	300	300	400	400	400	400
36 m	200	300	400	400	400	500	500	500

Source: AMC, 720018 MUG RF Study

Where sill pillars are to be removed or mining in a top down sequence, high strength cemented fill will be placed to a specified thickness. For access with tele remote bogging equipment a Factor of Safety (FoS) value of 2 is recommended to ensure stability. If there is any possibility of requiring direct person access under cemented fill sills (e.g. – mining through) then the FoS must be raised to 3. The design sill strengths required are listed shown in Table 16-14.

Table 16-14: Cemented fill sill dimensions and design strengths (kPa)

Width of Sill (m)	5	6	7	8	9	10	11	12	13	14	15
Minimum depth of sill (m)	5.0	5.0	5.0	5.0	5.0	5.0	5.5	6	6.5	7	7.5
Crushing Stability Limit (kPa)	500	500	500								
Caving Stability Limit (kPa)				527	593	659	725	791	857	923	989
Design strength at FoS 2 (Remote entry only)	1,000	1,000	1,000	1,100	1,200	1,400	1,500	1,600	1,800	1,900	2,000
Design strength at FoS 3 (Person entry)	1,500	1,500	1,500	1,600	1,800	2,000	2,200	2,400	2,600	2,800	3,000

Source: AMC, 720018 MUG RF Study

Test work

AMC concluded that the test work shows similar average strength results for both CRF and CAF but the CRF samples show more variability due to sample segregation. This effect is expected to be more prominent in placement in stopes. The test work shows that the majority of the strength of CAF and CRF samples is developed by seven days curing with minor strength gains at 14 days. This will assist with stope cycle times.

Batch plant

AMC noted that the current batch plant specification can produce cement slurry only when combined with a mobile agitator or transmixer. If cement slurry is required to be sprayed into backhaul ore trucks containing aggregate to make CAF, then an additional separate slurry mixing unit and spray arrangement will be required.

Costings

AMC has carried out a high-level comparison of capital and operating costs for the four main options available to Waihi at a CRF rate of 1.2Mt of CRF and 2.1Mt of CRF. Cost calculations are summarized below and all costs are presented in USD for the 1.2Mt case. Table 16-15 summarizes the operating cost values for the four options expressed in USD/m^3.

Table 16-15: Backfill options operating cost estimation

Opex USD / m3	CRF Agi	CRF Slick	CAF (Trucked)	CAF (Rock Pass)
Power	0.42	0.42	0.42	0.42
Barricades/Boreholes/Consumables	0.54	1.08	0.54	1.08
Equipment	6.58	4.52	6.97	4.91
Labour	18.44	15.27	18.44	14.98
Binder	15.66	15.66	15.66	15.66
Total Opex	**41.63**	**36.94**	**42.02**	**37.04**

Table 16-16 summarizes the capital costs for the four options expressed in USDM.

Table 16-16: Backfill options capital cost estimation

Capex USD M	CRF Agi	CRF Slick	CAF (Trucked)	CAF (Rock Pass)
Plant	0.90	0.90	2.14	2.69
Equipment	2.61	2.41	3.41	2.41
UG Infrastructure	3.09	3.45	3.09	4.16
Total Capex	**6.60**	**6.76**	**8.64**	**9.26**

Economics

AMC evaluated the capital and operating costs of the four options and the results are summarized below in Table 16-17. Over the life of mine (LOM), 1.2 million tonnes of fill is required in 7 active years of cemented filling up to 2027 with final MUG mine production completing in 2028.

Table 16-17: Backfill options economic comparison

	CRF Agi	CRF Slick	CAF Trucked	CAF Rock Pass
Operating cost ($/m3)	41.63	36.94	42.02	37.04
Capital Cost ($)	6.60	6.76	8.64	9.26
Net Present cost @ 7.5% discount rate	29.75	27.34	32.02	25.56

For the CRF options, the provision of a cement slurry slick line to a centralized backfill spraying station provides a lower LOM cost than the operating cost of eight active years of agitator transport. The practical downside of this option is increased haulage distance for all trucks hauling development waste for CRF purposes.

A similar benefit is seen with CAF and the rock pass option with a slick line also from reduced haulage distances from surface to underground.

Based on the test work carried out to date, the results show similar cement dosing rates for both types. AMC generally recommends CAF with better control of grading sizes and based on other operations a reduction of 1% cement compared to CRF could provide a project opex upside of up to NZ$3.50/m3.

Project Conclusions

The CRF Agitator option is the simplest and has the lowest capital cost. However, the use of a cement slick line and spray bar to apply cement slurry directly on to waste trucks could save over $4 million over the life of mine from reduced agitator activities and labour savings.

Trucked CAF is the simplest of the graded rockfill options but the use of a rock pass with a cement slurry slick line would reduce the overall truck haulage requirements and would potentially save $4 million over the life of mine from the surface truck option.

AMC considers that the CRF with slick line option and CAF with a rock pass and slick line provide the best options, but both carry risks to the overall success of the MUG operations.

AMC understands from the discussions during the study that a number of other factors that influence the relative risk of the backfill options. Two of these are:

- The state of old mined voids in various states of stability, partially filled with economic mineralization, or collapsed stope walls. Each of these factors could influence the overall cemented fill requirements.
- The success of safely placing cemented fill into the old voids and the strength and quality of the resulting cured fill exposures resulting from segregation and placement geometry.

Project Recommendations

AMC recommends a staged approach to introducing cemented fill at MUG. This will enable full scale trails to test the operating risks addressed above prior to commitment of major capital.

- Using existing mine equipment, set up a trial stope using the CRF agitator method to test overall mixing and placement activities.
  - During Avoca mining of the economic stope margins, carefully monitor the CRF exposure (using progressive CMS surveys after each blast) to determine dilution and overall fill exposure performance.
  - If successful, then do a trade-off study to compare LOM cement delivery by slick line compared to agitator.
  - Continue with CRF method.
- If the performance of the CRF is not acceptable, then do a second trial stope with graded aggregate to make CAF.
  - Monitor CAF exposure performance with CMS surveys.
  - If successful, continue with CAF method.
  - Evaluate potential for a rock pass with CAF method.

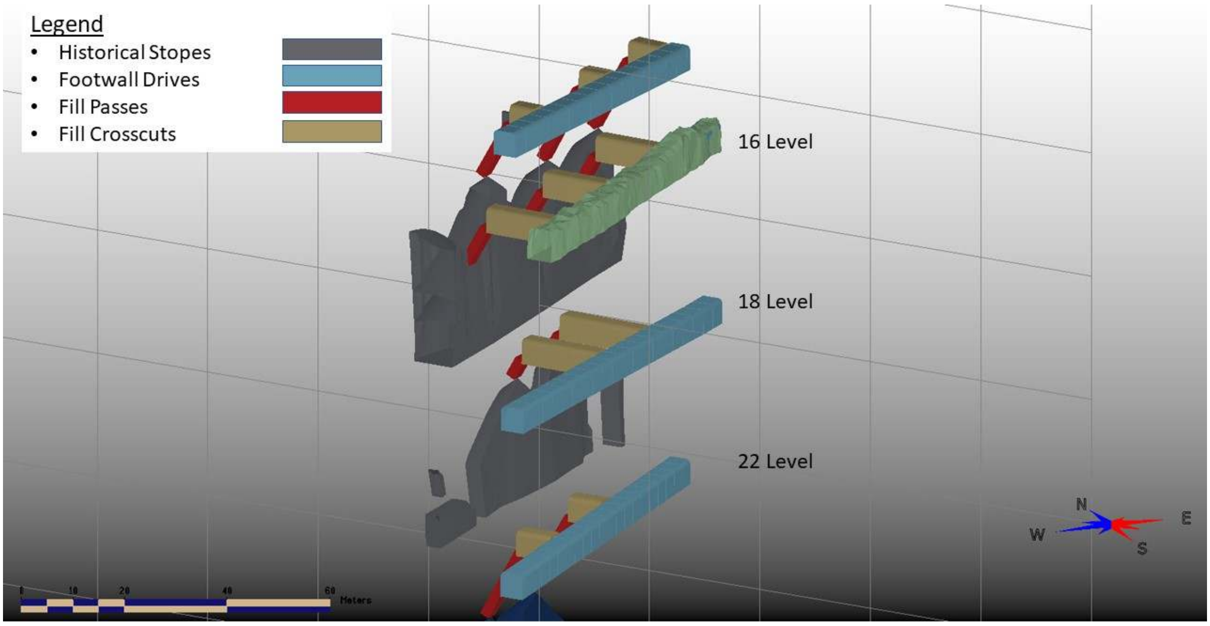
- Do further investigations on overall materials handling and rehandling of cemented fill.
- Fine tune the cemented fill recipes to optimize (minimize cement consumption).

It is assumed that all fill material if on surface will be loaded onto trucks and hauled underground via the decline. The trucks will enter the level and dump the backfill material into designated stockpiles. From here, LHD equipment will load, tram and dump the backfill into the void.

Placement Methodology

The general layout for the placement of CRF and RF into historical stopes involves the development of fill drives, crosscuts and passes intersecting the stopes as shown in Figure 16-26.

Figure 16-26: Fill Placement General Layout



AMC commented on placement the placement methodology for CRF into historical stopes and noted that:

- Current access to orebodies for initial backfilling is on 36 m level intervals. i.e., every second mining level.
- Intermediate access at 18 m vertical intervals will be established for production and backfilling.
- Most mining will take place in a conventional overhand bottom up sequence with the exception of remnant mining in Empire West.
- Cross cuts will be driven from the footwall drive at approximately 17 m centres to access old historical voids for backfilling with cemented fill. Longhole winzes will be fired into the historical voids as fill passes.
- If old historical filled voids contain economic grade and the voids remain stable, the previously placed backfill will be drawn out.
- CAF will be placed in open and emptied old historical voids with strengths as required for future exposure dimensions.

- Stopes will be filled from the crosscuts up to that elevation with cemented fill. Filling operations then move to the next crosscut and repeated until the stope is filled.
- Following completion of filling to crosscut elevation, cross cuts will be advanced to the next void or orebody accesses.
- Any remaining voids will be filled with either cemented rock fill or loose rockfill where required.
- Production stoping will commence using long hole methods extracting economic material around historical voids in section and along strike.
- Cemented fill will be exposed in vertical and undercut exposures and, in some cases, the hangingwall of the stope.

Cemented fill strength empirical design methods are limited to vertical and horizontal exposure geometries. At MUG, the mining is likely to expose a small number of overhanging fill masses where the footwall ore is removed beside and below a steeply dipping fill mass. AMC recommends that OCG identify the range of stopes that may have this orientation and conduct a numerical modelling study to develop the appropriate fill strength requirements. It should be noted there is very little experience with overhanging fill performance other than horizontal sills and appropriate design cautions should be undertaken with monitoring of actual performance whenever possible.

Other backfill placement options could include the use of boreholes and / or fill raises to reduce trucking requirements however, the extensive strike of the orebody and the timing of filling does not make it a simple decision and it is recommended that these backfill placement options be evaluated in more detail in future studies.

CRF Batch Plant

Cement slurry and a dry mix cement will be generated in a surface plant, refer Figure 16-27 which is a render of the proposed plant without the cement silos or aggregate storage bins located at the polishing pond stockpile area. Pricing was received from several qualified contractors to provide the following plant configuration:

- 4 material storage bunkers of reinforced concrete construction,
- 20 m x 20 m metal framed building with concrete floor,
- Integrated weigh hopper, complete with load cells, belt conveyor and digital display,
- 2 x 70t vertical or horizontal cement silos complete with load cells, filter system, instrumentation, fill pipe system, valves and screw conveyors,
- 3 m x 3 m prefabricated control room, complete with internal wiring, lights GPO’s etc.,
- drive in clean out sump, suitable for a L120 Loader, or equivalent,
- concrete foundations (or hard stand) for all equipment,
- 2 x 20kL plastic water tanks,
- air compressor, reticulation pipe work and valves,
- pavement slabs, with containment bund walls, with drainage reporting to the drive in sump,
- sump pump and two water supply pumps,
- batch plant control system program, and
- piping and valves, instrumentation and electrical installation and area lighting.

Figure 16-27: Dry Mix Cement Backfill Plant

Source: OceanaGold, 2021

The cement will be provided by a local manufacturer that will truck the cement in 25 t tanker transports to the site at the rate of approximately two trucks per week. Table 16-18 shows the total LOM volume breakdown of the RF (low strength) and CRF (high strength).

Table 16-18: LOM Backfill Quantities of CRF and Non-Cemented Waste Rock

Backfill Summary	(tonnes)
Total Backfill	3.8M
Rockfill	2.5M
CRF	1.3M

16.10.4 Backfill Deficit

There is a deficit of waste rock for backfilling the underground voids. Waste rock will be sourced from the Martha open pit (MOP) under the current consent conditions. Waste rock is also available from materials stockpiled at the western end of the Martha Pit and mined from the upper benches of the north wall under the consent.

The OP mining process at Martha has been determined largely by the consents granted to the company for the previous pits. Waste rock will be mined by conventional drill, blast, load and haul methods from the OP. Strict controls on blast vibration will determine the blast hole spacing and the maximum allowable charge weight per delay. Material will generally be blasted in 5 m vertical intervals (two flitches), but blast vibration limitations may require some blast holes to be drilled at 2.5 m vertical intervals. Electronic detonators will be used in all holes to ensure detonation of charges occur as per the design sequence. The company will be required to monitor each blast vibration for conformance.

Initial mining operations will likely use small equipment until the northern pit perimeter is established and mining operations are below the crest. Small equipment will likely comprise 50 t excavators and 50 t articulated trucks.

Once the pit rim is established, all mined material will be loaded via 110 t or 190 t backhoe excavators into 85 tonne rear dump trucks and trucked via a 1 in 10 ramp and direct tipped to a jaw crusher via a bin and apron feeder. Small quantities of waste rock may be temporarily stockpiled close to the crusher.

The presence of historic workings in the OP will require probe drilling to identify voids or weak pillars which create both a safety hazard and an operating constraint. Underground voids will, be either bunded off or marked with hazard tape. Excavators and trucks will be required to operate around the void working in towards the void. This process can at times influence the bench extraction sequence.

Except for selected waste rock that is sent direct to MUG to backfill stopes and historical voids via the southern pit portal or fill pass all waste rock will be crushed. Waste rock will be conveyed either 2.0 km to the rock and tailings storage area loadout for construction of the tailings storage facilities or conveyed 1.5 km to ROM stockpile area and backhauled via the Favona portal.

It is anticipated that the working hours within the OP, adjacent service facilities and conveyor corridor will be restricted to:

- Monday-Friday 0700-2100
- Saturday 0700-1200

The pit will not operate on public holidays or Sundays. Blasting will generally be between 10 am and 3 pm on working days.

16.11 Grade Control

The characterisation of ore versus waste will be completed through diamond core drilling of the stope prior to mining. Diamond drilling will be undertaken from decline crosscut development or from footwall drives. It is planned to decrease the Indicated spacing from 30 m centres to 20 m centres through infill drilling In general drill holes will be drilled beyond the planned length of the stope access. The core will be logged, sampled and analysed to provide grade control information. Geologic and block models will be updated with this information and ore/waste grade boundaries will be pre-determined prior to mining the lateral ore development.

Approximately 20km of grade control drilling is required to further delineate the various veins. Empire and Empire west mine areas has a significant portion of proposed stope skins to be mined. These skins will be accessed by footwall and hanging wall drives in around these veins and also the old working voids. Grade control drilling will be required from these drives to define economic skins and void positioning. The rig will need to drill in a drive 4.5 m wide, and to 90 degrees to the drive. Approximately one hole every 9 metres is planned to be drilled. The planned grade control drilling programme is shown in below in Table 16-19.

Table 16-19: Grade Control Drilling Programme

Grade control	2021	2022	2023	2024	2025	Total
Planned drill metres	7,700	3,000	5,400	4,100	1,000	21,200

The existing Waihi SGS laboratory analyses Au, Ag and As with an expected turnaround time of 18 to 26 hours.

Additionally, as the stope level development is being mined, a mine geologist will observe, map and channel sample the exposed rock on each face. Typically, the veining will display an inclined sub-vertical orientation on the face and the channel samples will need to be oriented normal to the vein.

The mine geologist will identify the presence of silicification and use this alteration to determine where sample breaks will occur. Using the inclined channel sample orientation, sample lengths for each cut will be 4 to 5 m in length. Samples will be collected as continuous channels at nominal 1 to 2 m lengths depending on the observations of the geologist. Channels can be sawn or chipped depending on the nature of the rock being collected. Sample breaks will be made at lithologic or alteration contacts.

Once mining has progressed to a point where sufficient testing has shown that estimated grades based on core samples reconcile to channel sampling during actual mining, a portion of the diamond drill or channel sampling may be eliminated.

Ore trucked from the stopes is routinely sampled at the RoM to assist in reconciliation of mine to mill production.

16.12 Mine Control & Technology

October 2019 through to December 2019 saw the MUG Smart Centre commissioning phase. The Smart Centre has been in operation since January 2020. Key successful components of implementation include:

- Construction and installation of Smart Centre, and dedicated Server Room for Waihi UG operations;
- Hiring and basic training of Smart Centre Operators;
- Purchase, training, and implementation of FEWZION planning software, for UG mine scheduling;
- Development and publication of 24hr shift plan from digital sources, alongside hard-copies;
- Significant Uptake in use of reporting tools and platforms, making more use of available data including:
- Microsoft Power BI – Report Development Platform
- Microsoft Teams – Collaboration Platform
- OSISoft PI Vision – Real-time and Historical Data Analysis Tool and Dashboard for monitoring key infrastructure e.g. pumps, fans, stench gas;
- Commit Works – FEWZION – Pre-shift and In-shift Mine Scheduling
- MineDash – UG Personnel and Vehicle Tracking Tool.

The room operates 24/7, manned by a smart centre operator working the same roster as each crew. Handovers between personnel occur between 0615 and 0630, or 1815 and 1830 and cover a range of topics, including:

- Location of mobile plant and status of stopes / headings;

- Technical issues relating to software, systems etc;
- Machine breakdowns passing over to following shift;
- Comments on data quality in current shift;
- Expected personnel absentees;
- General comments and feedback on shift performance;

Data, and job status is communicated by underground operators via radio call-ups to smart centre operators, to give near-live updates of shift progress against plan. This information is largely captured within FEWZION, or a production database. End-of-shift data validation is conducted by referring hard-copy information against in-shift call-ups. Similarly, key plant and infrastructure breakdowns, and maintenance tasks are communicated via radio call-ups to surface, giving near-live status of equipment.

Work has been conducted to develop live dashboards within PI Vision for monitoring the following:

- dewatering pump station and submersible pump flow status
- stench gas release status
- compressor status
- power usage
- gas monitoring
- Tagboard and portal status

OGC's long-term objective of the smart centre is to allow centralised decision making, short term planning and response, semi-automated dispatch of work and improved schedule adherence.

16.13 Stope Optimisation

The Martha deposit ranges in thickness from approximately 0.5 m to 18 m. Stope optimisation within Deswik software was used to determine potentially economically minable material. Stope sizes used in the initial optimisation were 15 m wide and a minimum of 10 m long and alternate stope heights were evaluated. Figure 16-28 shows the raw MSO shapes colour coded by gold grade and Figure 16-31 shows the same shapes colour coded by average stope width.

Figure 16-28 Stope Optimisation Shapes Grade

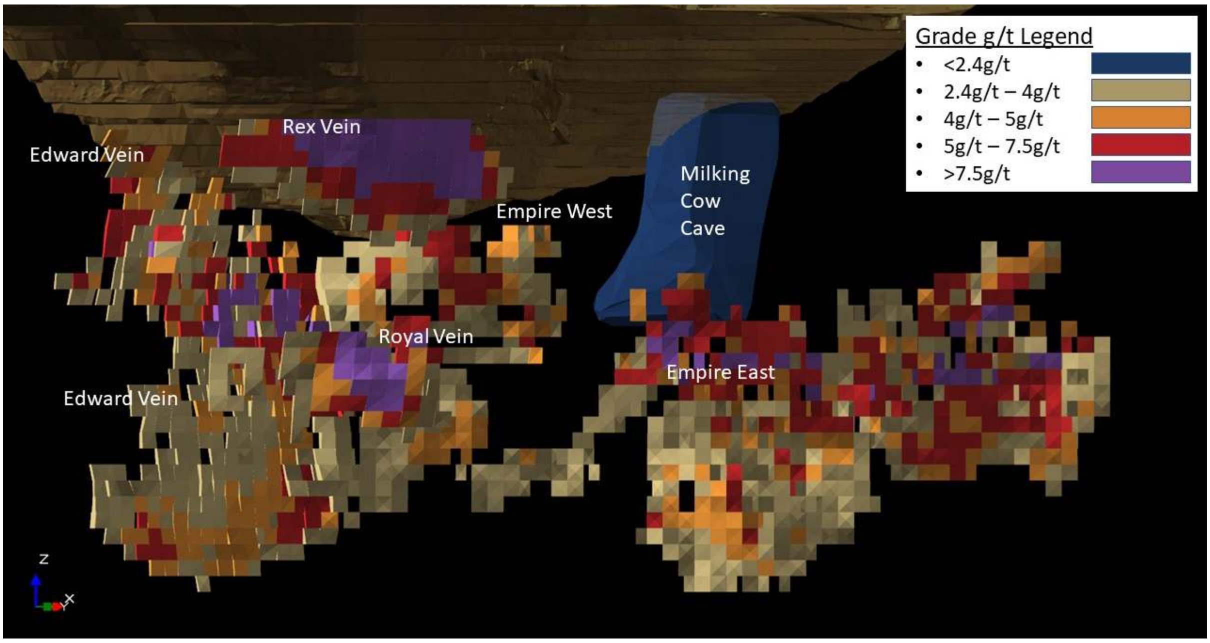
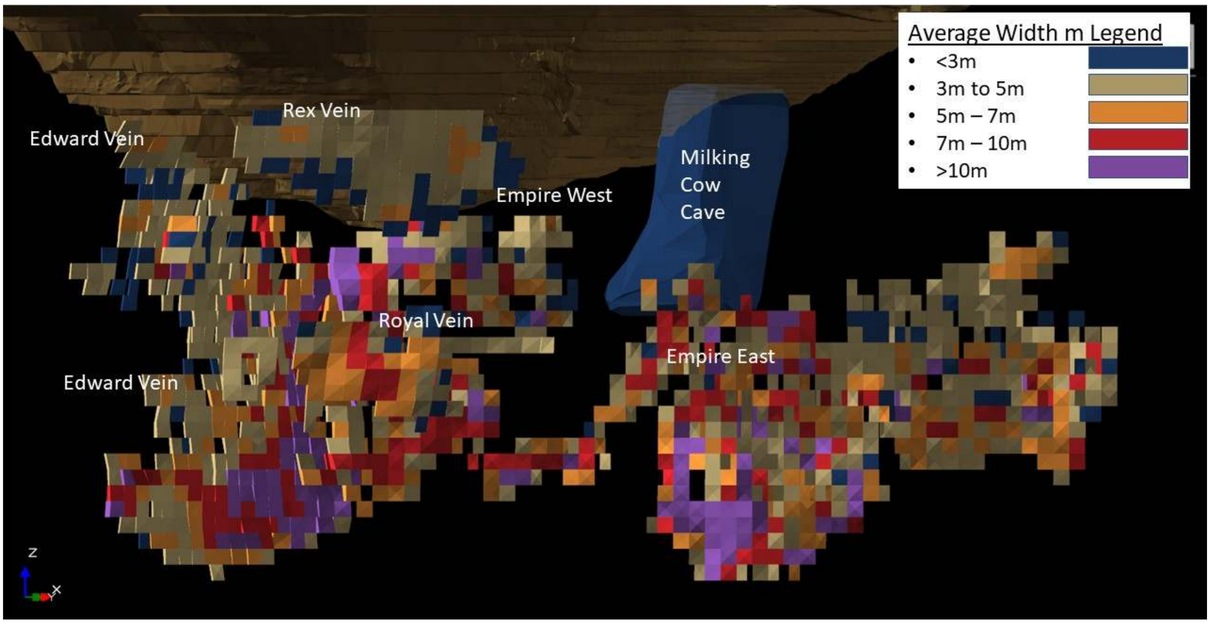


Figure 16-29 Stope Optimisation Shapes Width



The economic benefit of various sill pillar locations was evaluated considering both material left in-situ or the use of a CRF crown. Numerous stope optimisation runs were completed to determine optimal level locations.

The following observations have been made from the stope optimisations:

1. 70% of the mineable stope material sits within stopes greater than 4.7 m in width.
2. 80% of the mineable metal sit within stopes greater than 4.1 m in width.
3. The majority of stopes less than 3.0 m in width lie on the extremities of the Orebody.
4. Grade distribution between stope widths 1.9 m to 15.0 m is relatively uniform with some outliers of high-grade in the 3.5 m to 4.5 m range.

5. The internal dilution percentage is highest in the narrower stopes showing a marked decline in the widths above 6 m.

The distribution of grade, tonnages, metal and dilution by the average stope width is shown below in Figure 16-30 to Figure 16-32 inclusive.

Figure 16-30: MUG MSO Tonnes, Metal by Width

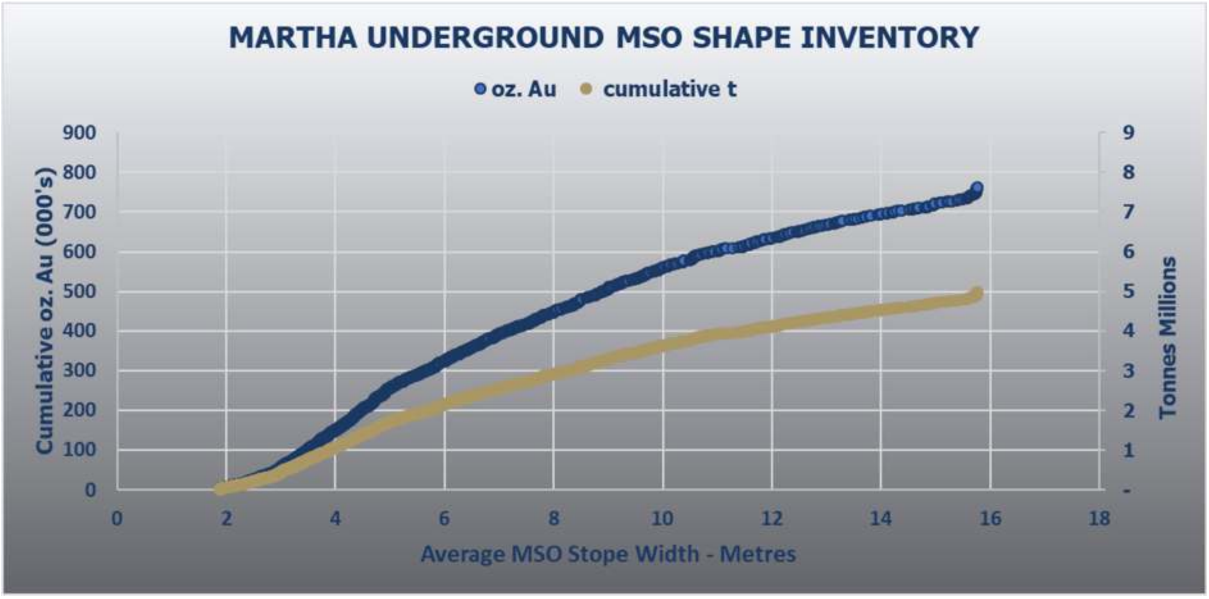


Figure 16-31: MUG MSO Grade by Width

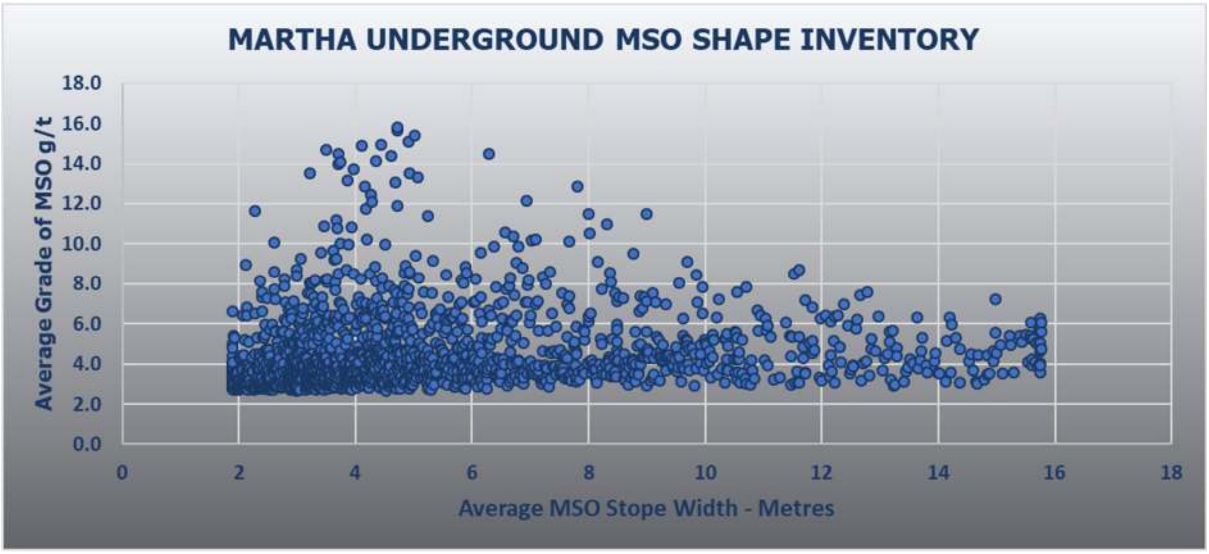
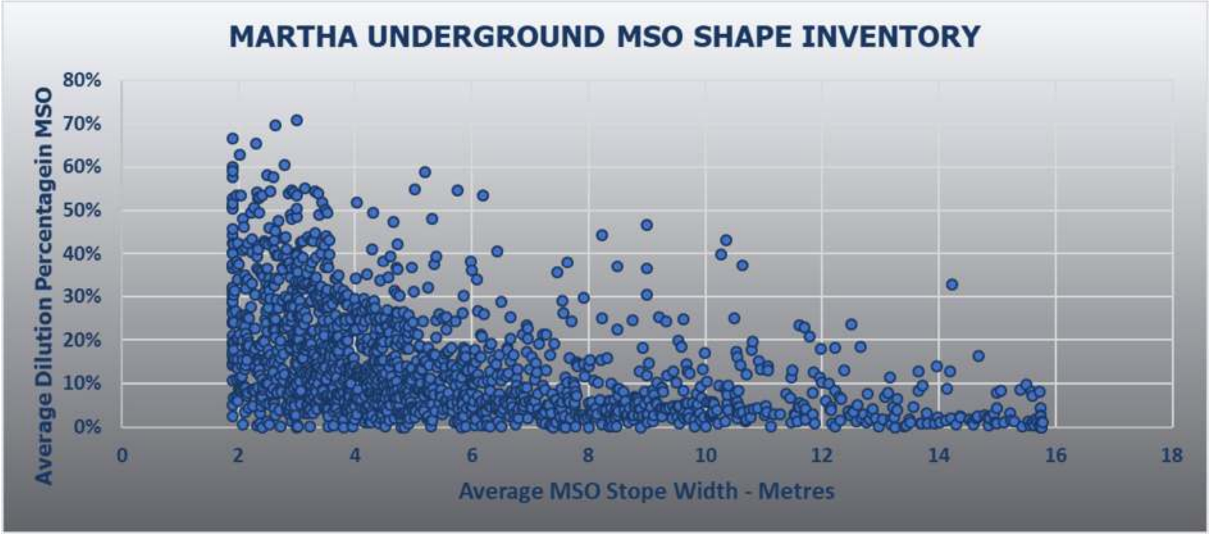


Figure 16-32: MUG MSO Dilution by Width



Given the majority of stopes with widths less than 3.0 m lie on the extremities of the orebody, and that 70% of the mineable tonnes sit within stopes greater than 3.0 m design principals should be applied for the optimum extraction of stopes greater than 3.0 m (inclusive of dilution). Using these design principals will still allow for reasonable extraction of stopes down to 1.9 m in widths.

Subsequent optimisations used a level interval of 18 m which aligned reasonably well with the historical level spacing. Vertical intervals of the optimisation were set to match as best as practicable the historical level intervals. The minimum mining width applied was 1.3 m with 0.3 m dilution applied to both footwall and hangingwall making the minimum mining width 1.9 m.

16.14 Stope Design

16.14.1 Geotechnical Design Parameters

It has been proven that stable stope strike spans of up to 30 m can be mined. Caving and surface subsidence potential has been assessed for development and stoping with the risk being low if recommendations for ground support, allowable spans, and management techniques are followed.

The following geotechnical parameters have been used within the MUG Mine design:

- Development ground support regimes with bolting and mesh required in all areas, fibrecreting as required in poorer ground areas and cable bolting of drive intersections and wider excavations.
- Minimum 1:1 pillar width separating development openings.
- 15 m to 18 m vertical level spacing provides a good basis for stable stoping and manageable blast vibration.

16.14.2 Dilution

The mining dilution estimate for the FS is based on the ELOS methodology (Clark, 1997). ELOS is an empirical design method that is used to estimate the amount of overbreak / slough that will occur in an underground opening based on rock quality and the HR of the opening. For the FS design, ELOS was applied to in-situ rock exposures. In addition to ELOS allowances, a 0.5% additional dilution allowance was used to account for other potential sources of dilution (e.g., dilution from the floor when mucking a stope). ELOS assumptions are shown in Table 16-20.

Table 16-20: Underground Mining Dilution Factors

Activity	ELOS m
Sidewalls (rock)	0.3 m
Bottom (backfill)	0.05 m

Stopes were designed with 0.4 m dilution on both the footwall and the hangingwall which when applied with the stope recovery factors reconciles with the performance of stopes in Favona, Trio and Correnso mines.

The rock sidewall dilution material will contain low-grade mineralisation. To determine the grade that should be used for dilution material, 3D triangulations were generated for the dilution for through the entire deposit. The grade for the diluting material was calculated from the block model and averaged 0.3 g/t Au. A zero grade is used for back fill dilution. The ELOS and additional dilution factor of 0.5% gives a total dilution of 7% at a grade of 0.3 g/t Au from in-situ rock that is exposed in the stopes.

16.14.3 Recovery

A development recovery factor of 100% was used for all horizontal development. Overbreak is included in the capital and operating lateral waste development dimensions so that no additional overbreak is assigned. No overbreak is assumed for operating lateral ore development as the overbreak tonnes are generally ore which are included in the stope tonnes. Assuming zero overbreak in the ore drives removes the risk of either double counting or under calling ore tonnes and metal.

The mining recovery factors applied for MUG are summarised in Table 16-21. The stope recovery factor was varied as a function of mining method and took into account wall and rill dilutions (additional tonnages), underbreak from stope firings 92% was used (reduced gold oz.) and the recovery of the material from the stope by the remote bogging operations (70 to 96%) . The following items were used to calculate this factor:

- Material loss to side and end walls of 0.3 m;
- Material loss to stabilising pillars (limited application, only as needed);
- Material loss to mucking along the sides and in blind corners of the stopes; and
- Additional loss factor due to rockfalls, misdirected loads, and other geotechnical reasons.

Tonnage recovery factors shown in the table below for stoping include in-situ ore plus dilution material. Metal recovery factors consider the difficulties associated with recovering all ore from a stope, particularly under remote control operations. Additionally, the factors allow for the potential loss of metal due to excess dilution burying ore and limiting recovering of all of the ore.

Table 16-21: Underground Mining Dilution & Recovery Factors

Mining Method - Modifying Factors	Wall / Rill Dilution	Under-break	Bogging Recovery	Modifying factors	
Virgin Avoca & Mining against Cemented Fill					
Tonnes	5%		96%	1	Based on Correnso
oz. Au		3%	96%	0.93	
Proximal to Collapse (+3 m pillar)					
Tonnes	7%		93%	1	Increase – shorter panels
oz. Au		3%	93%	0.9	Allow Some panels fail
Adjacent to Collapse or Historic Fill					
Tonnes	25%		70%	0.88	Corners cannot be bogged out
oz. Au		5%	70%	0.67	High dilution in historic fill

No Inferred Resource metal has been included in the Mineral Reserve. Each individual design item was interrogated to report against each Mineral Resource category, and the average grade of each design item assessed allowing only contribution of metal from Measured and Indicated Mineral Resource categories. As such, any Inferred Resource material was effectively included as diluting material at zero grade.

16.14.4 Additional Allowance Factors

Additional ramp allowance factors were used to account for additional excavations not included in the FS design. These items will be designed at the detailed planning stage. Items are summarised in Table 16-22 for the main ramps. These factors are applied to the mining schedule as additional activities and the volume / tonnage of rock is increased similarly. This captures the development time required to excavate these openings, cost to do so, and reports the expected amount of waste volumes for tracking purposes.

Table 16-22: Underground Mining Additional Allowance Factors

Activity	m
Electrical - 4.5 m x 4.5 m x 4 m long, 1 per level	100
Muck Bays - high backs for loading - 7 m tall	150
Additional allowance for main ramps:	100
Workshops, service bays, bypass development	1,000
Additional allowances for other development	650
Total additional development metres	**2,000**

All planned maintenance will be undertaken on the existing surface and underground shop facilities and are not included in the design. The CRF facilities are also located on the surface and no additional infrastructure is required underground. Where possible accesses/ramps have been designed to be located in the away from known faults. Where ramps must cross a fault, the crossing is designed perpendicular to the structure to minimise the length of development through these structures. Figure 16-33 and Figure 16-34 show the completed mine design coloured by activity type and Au grade respectively. Table 16-23 summarises the mine design by activity type.

Figure 16-33: MUG Long Section Showing Activity Type

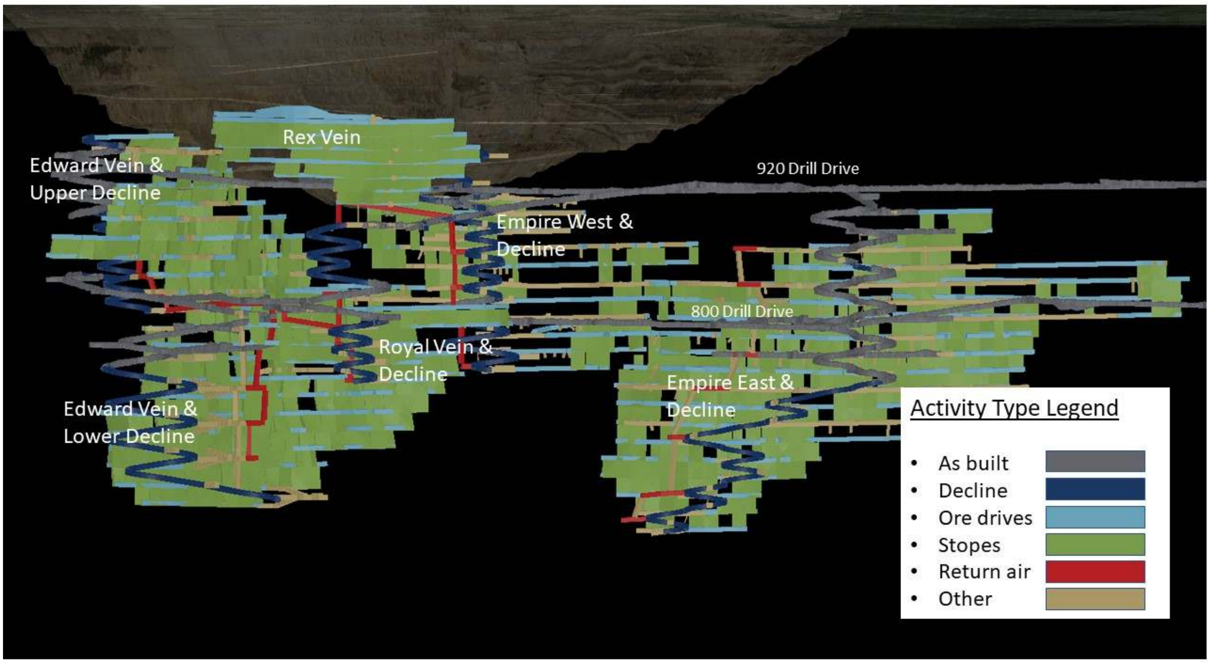


Figure 16-34: MUG Long Section Showing Grades (g/t)

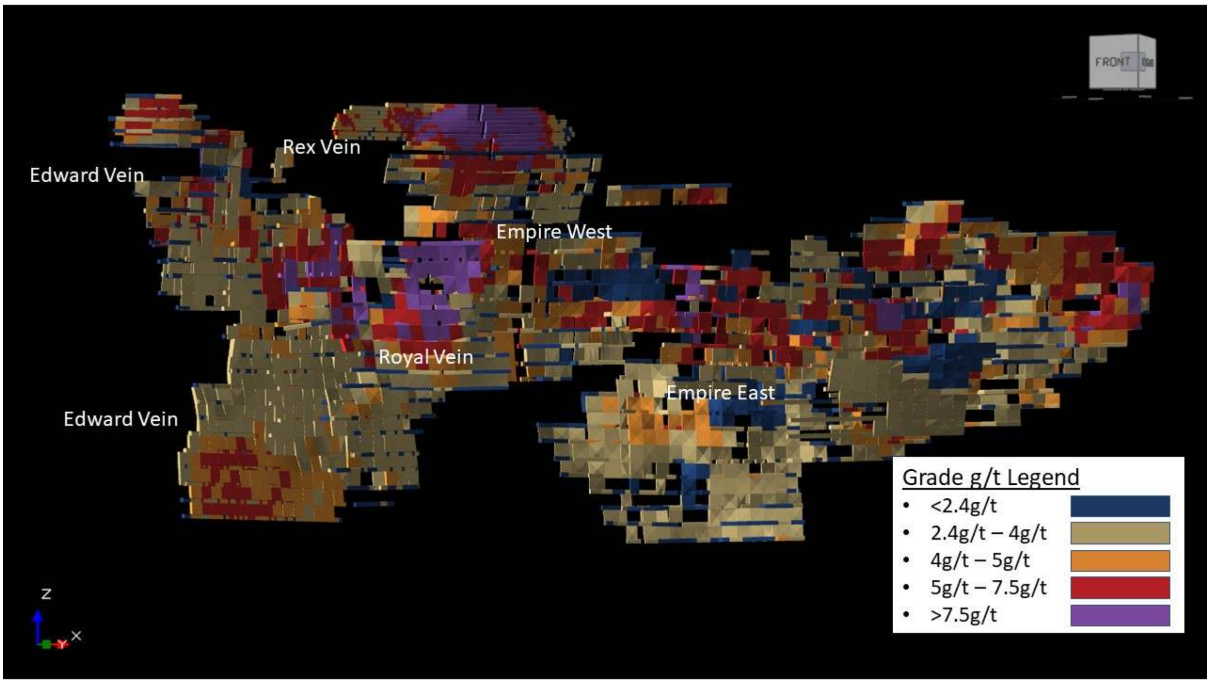


Table 16-23: Mine Design Summary by Activity Type

General Summary	Metric	Value
Ore Tonnes	(kt)	4,458
Ore Au	(g/t)	4.3
Waste Tonnes	(kt)	2,336
Development Ore Tonnes	(kt)	1,085
Stope Production Tonnes	(kt)	3,370
Lateral		
Ramp Profile	5 x 5.8 (m) Arch	5,263
Escapeway Profile	4.5 x 5.5 (m) Arch	496
Sump Profile	5.0 x 4.5 (m) Arch	145
Bogger Access Profile	5.0 x 5.0 (m) Arch	32,610
Ore Drive Profile	5.0 x 5.0 (m) Arch	5,630
Truck Access Profile	5.0 x 5.6 (m) Arch	3,257
Truck Tipple Profile	5.0 x 7.2 (m) Arch	1,151
Stockpile Profile	6.0 x 6.0 (m) Arch	257
Narrow Vein Profile	3.0 x 3.2 (m) Arch	45
Cut And Fill Profile	4.5 x 4.5 (m) Square	556
Fill Drive	4.5 x 5.0 (m) Arch	2,046
Workshop Bay	8.0 x 6.0 (m) Flat Arch	55
Vertical		
Escape Raise Profile	SQR_1.2 X 1.2	584
Return Air Large Profile	SQR_4.5x4.5	873
Fill Pass Profile	SQR_2.0X3.0	1,176

16.15 Conversion of Indicated Resources to Probable Reserves

Figure 16-35 and Figure 16-36 show the location of Indicated Resource converted to Reserve. The study has converted approximately 60% of the Indicated Resource to Probable Reserve. The reasons for this conversion rate are described below.

- A lower cut-off grade is used for Resources compared to Reserves (2.2g/t vs. 2.5 to 3.3g/t).
- Historical stope backfill is classified as Inferred and cannot be included in Reserves, and either the skin adjacent to the historical backfill cannot support the mining costs by itself or only one skin can be extracted.
- A significant Indicated Resource has been reported within the Martha lode. The Martha lode was intensively mined historically with a mixture of cut and fill, square set stoping, shrinkage stoping and caving of pillars and recovery of floor pillars. This area was largely excluded from the Mineral Reserve estimate pending further work around

defining the extent and condition of the historic workings, the intervening pillars, the historic backfill, geotechnical conditions and confirmation of a suitable method of access, mining methods and sequencing. Further work will also be focused on deriving appropriate Modifying Factors to account for mineralization within intermediate pillars and on the footwall and hangingwall of the main veins.

- Areas requiring mine development have insufficient Indicated Resource to support the development of the level or panel.

Figure 16-35: MUG Long Section Showing Location of Indicated Resource & Reserves

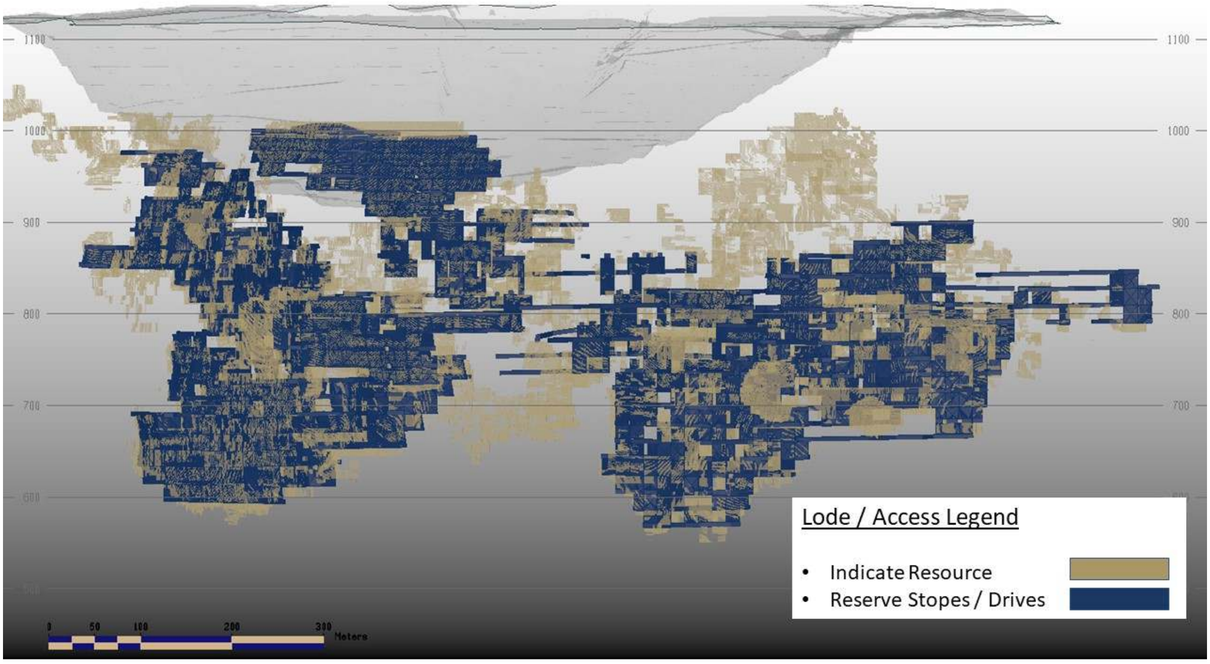
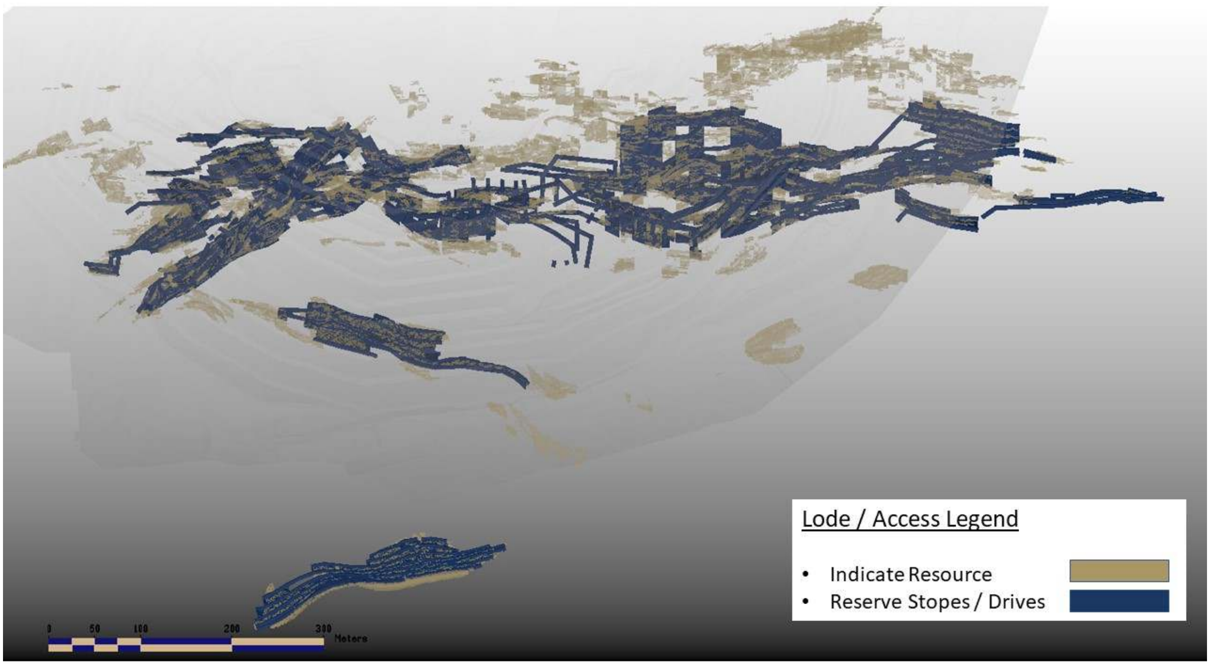


Figure 16-36: MUG Plan Showing Location of Indicated Resource & Reserves



16.16 Production Schedule

An integrated LOM design was prepared using Deswik software, specifically developed for mining industry, that allows for mine physicals and fixed and variable costs to be scheduled. The design software incorporates functionality to export all design and block model interrogation data to the scheduler, including volumes, tonnes, grades, and segment lengths. Furthermore, graphical sequencing is exported for the critical links between all development and production activities.

Martha underground mine production criteria were calculated from benchmarked rates. Table 16-24 lists the productivity rates and activity durations used in the mine development and production schedule.

Table 16-24: Underground Mining Rates

Activity	Martha Underground Rate
Critical access and ventilation development	18m / week max
General rate level waste development	9-12m / week
High priority ore drive development	15m / week
General rate ore drive development	9m / week
Stope production – peak individual stope	650t / day - 800 t / day
Stope production – total mine peak	2,000 t / day
Standard backfilling rate	850 t / day
Long hole drilling rate per rig	160 m / day

Production scheduling was undertaken in Deswik. Checks were made to ensure development and stopes were sequenced correctly with bottom up development or top down development where appropriate, development completed on the level prior to stoping commencing and adequate separation between the stoping fronts on the various levels. Checks were also made to ensure stoping, drilling and backfilling activities on a single level could be carried out independently of each other.

The MUG schedule allows for a dewatering rate of 40 vertical metres per year. Crown pillars were located at strategic horizons to enable production targets to be met and to ensure that a tail of low production towards the end of the mine schedule could be minimised. All crown pillars allowed for cemented backfill and recovery. The mining sequence is shown in Figure 16-37 viewed from the southwest.

The mining schedule was split into the following working areas:

- Rex,
- Edward upper levels,
- Empire east and Martha,
- Empire west,
- Royal east,
- Royal west,

Each working area had independent decline access and independent escape and ventilation development.

The steps involved in creating the mining schedule were:

1. Identify inputs into SO (stope optimisation) runs = minimum mining width, stope dilution/recovery, development dilution/recovery
2. Perform multiple SO runs at varying cut-off grades.
3. Exclude shapes within or close to the existing pit, within a 110 m from surface (Rex consent condition), Milking Cow cave exclusion zone and isolated shapes that would not support development.
4. Investigate areas where stopes may have a concentration of grade less than cut-off. This can generally be excluded from the mine plan.
5. Select the MSO shapes to bring into the first schedule run. Interrogate shapes and check grade and classification and risk.
6. Import latest historic wireframes from survey.
7. Identify historic stopes requiring backfill in order to extract proposed stopes.
8. Mine design including capital decline access, lateral waste development, escape and vent raises, fill pass development and ore drives in Deswik. Define strings and then attach wireframes to the strings. Mine design takes into account diamond drilling priorities.
9. Cut stope shapes with lateral ore drives to ensure contents are not double counted.
10. Break development into manageable lengths for scheduling ~ 15 m .
11. Assign activities and codes to the solids / wireframes for identification, activity type, mine area, resources.
12. Connect all mining solids in a logical sequence i.e. development, drill, blast, muck and fill. (Create dependencies done in Deswik, often rule based (auto) with some manual tweaks to ensure tasks are completed in the correct order i.e. ensure stopes are not going before development is complete, often a visual check and correct as required)
13. Interrogate all shapes for tonnes grade volume etc.
14. Set up the dilution logic, (i.e. mined oz. = 93% insitu ounces), and reference reconciliation data to confirm Modifying Factors.
15. Run the mining schedule over all of the solids and check logic. Mining schedule will report all physicals and costs by any timeframe depending on the selected report format. The schedule also creates the 3D visualisation tool.
16. If the mining schedule has peaks ‐ invariably this is the case, run resource levelling such that equipment numbers is optimised. i.e. 5 jumbos, 6 loaders and 5 trucks and assign priorities to the development in order for the resource levelling to optimise.
17. Minor re-schedule based on classification, diamond drill priorities, primary stoping versus remnant stoping, prioritise higher-grade, easily accessible ore in short term.
18. Review economics for each level / area within the Deswik- a report is provided on the profitability of each level to see if it meets the criteria, often positive cash flow.
19. Remove any areas that are demonstrably loss making from the plan and re-run the mining schedule.
20. Check the ventilation circuits comply with the mining sequence and that two methods of egress are available prior to committing to stoping. Amend any links to ensure this.
21. Finalise the mining schedule and import into the cost model.

The mine is planned to produce at a rate of 750,000 t/yr. To ensure overall mine production rates are achievable, activities associated with the mining works were scheduled in a logical sequence using the rates discussed above.

The major constraints on the underground scheduling were as follows:

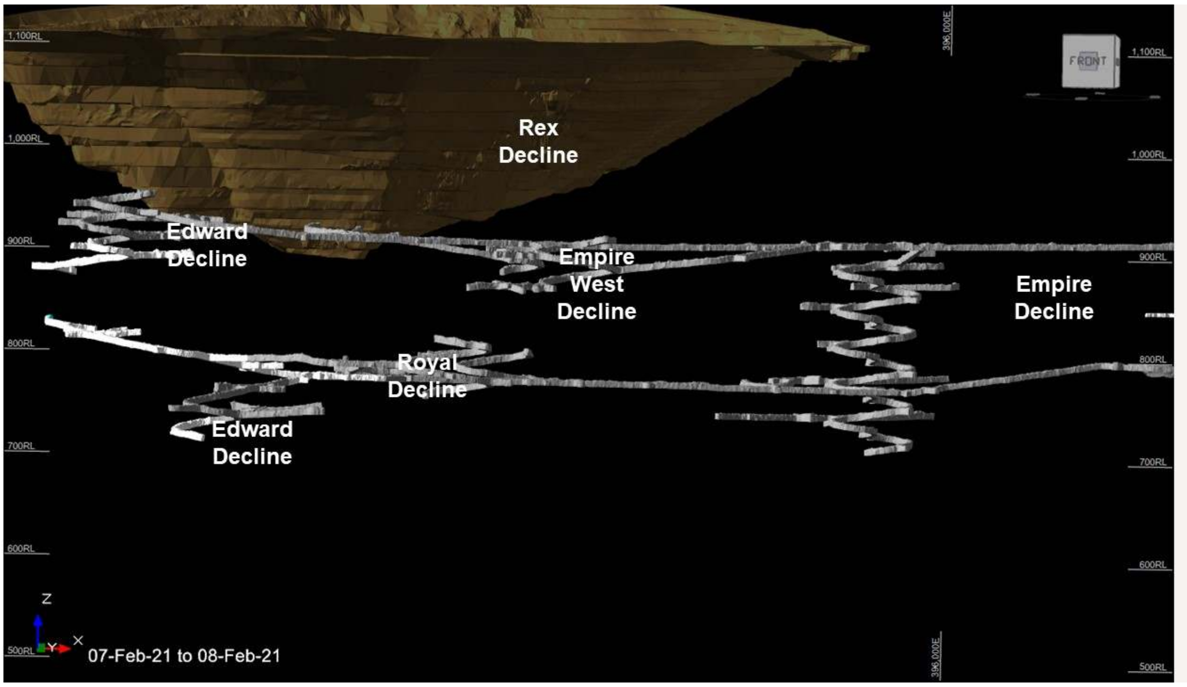
- Ensure a smooth ramp up to steady ore production;
- Minimise variations in development rates and production to avoid additional project costs due to under-utilisation of the mobile equipment;

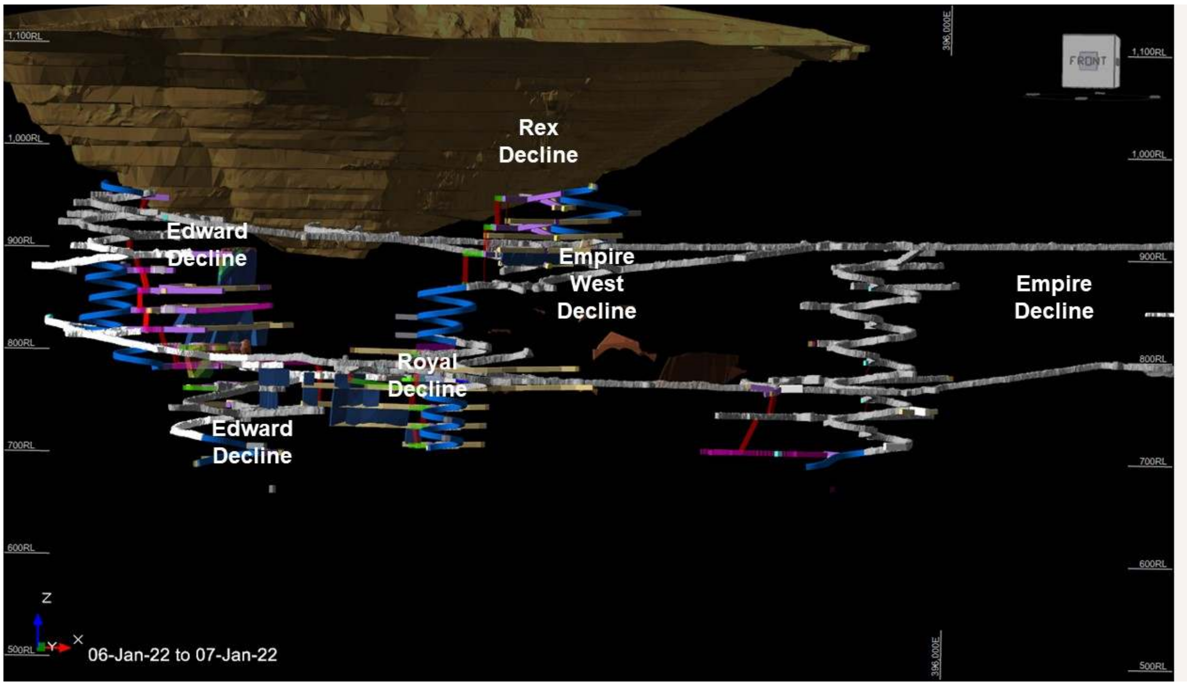
- Maintain capital development approximately one full stope block ahead of production to enable capital infrastructure to be established “just in-time”; and
- Stope production can only commence once the main return airway and second egress is established.

The decline was divided into individual 14 m sections to become separate activities. Crosscuts were linked to the corresponding decline activity using a “finish-to-start” constraint. In this manner, development, stoping and backfilling activities were linked to subsequent activities. The ability to carry out the linking visually in Deswik provides a schedule that is easy to check and audit..

The schedule comprises six independent ramp accesses each with separate escape and return air systems which are gathered at the key 800L and 920L haulage drives. From each decline a crosscut is developed into the orebody and sill dives developed along the orebody, each level also includes minor infrastructure for stockpiling, loading, electrical and pumping. On some levels additional development including footwall drives, crosscuts and passes are developed for backfilling the historical stopes. Figure 16-37, shows the mining schedule sequence and activities by year viewed from the south looking north. A summary of the mining physicals and schedule are detailed in Table 16-25.

Figure 16-37: Martha Underground Mining Sequence

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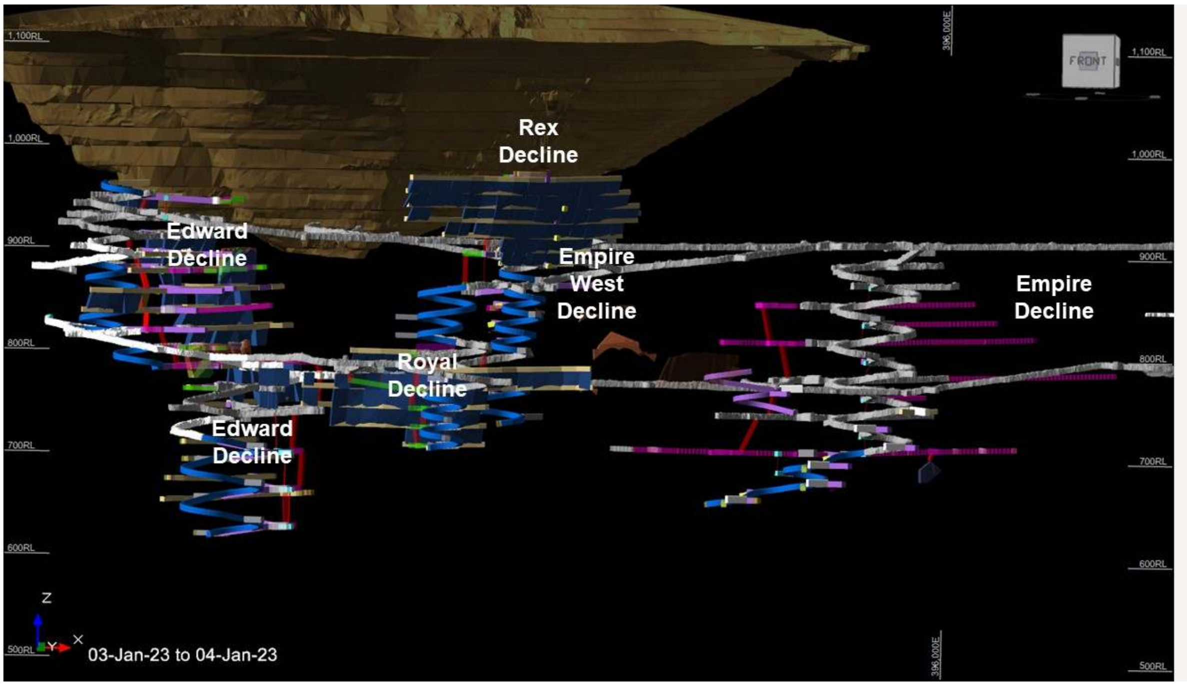
FRONT
06-Jan-22 to 07-Jan-22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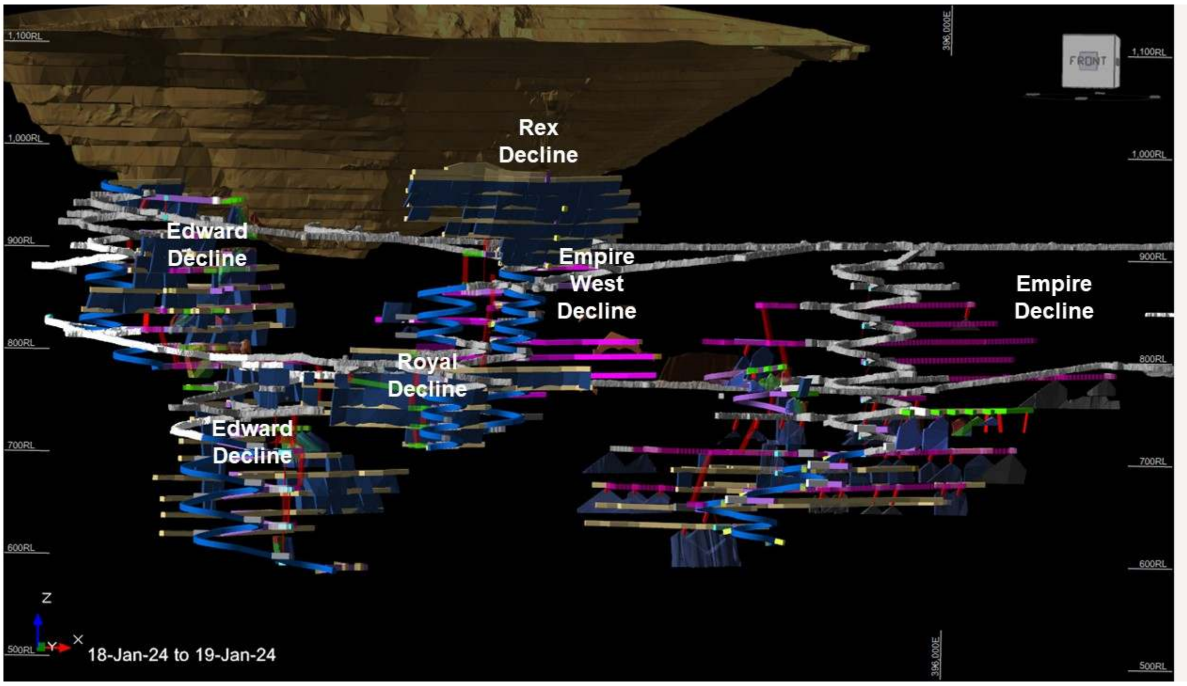
03-Jan-23 to 04-Jan-23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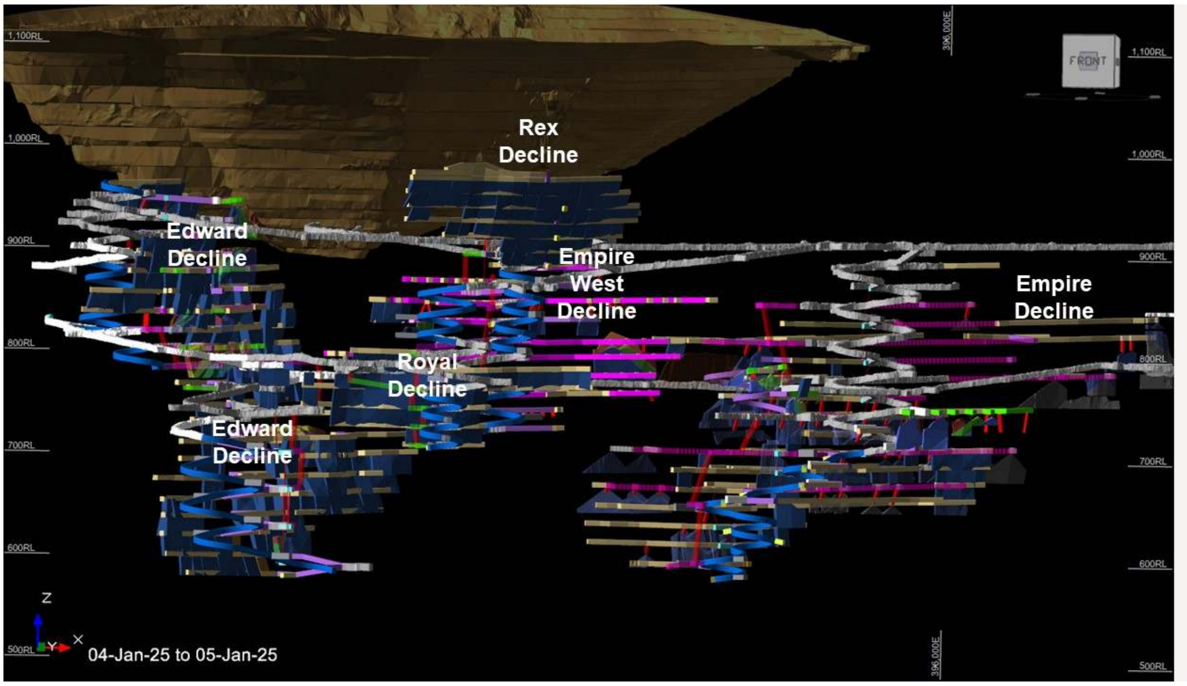
18-Jan-24 to 19-Jan-24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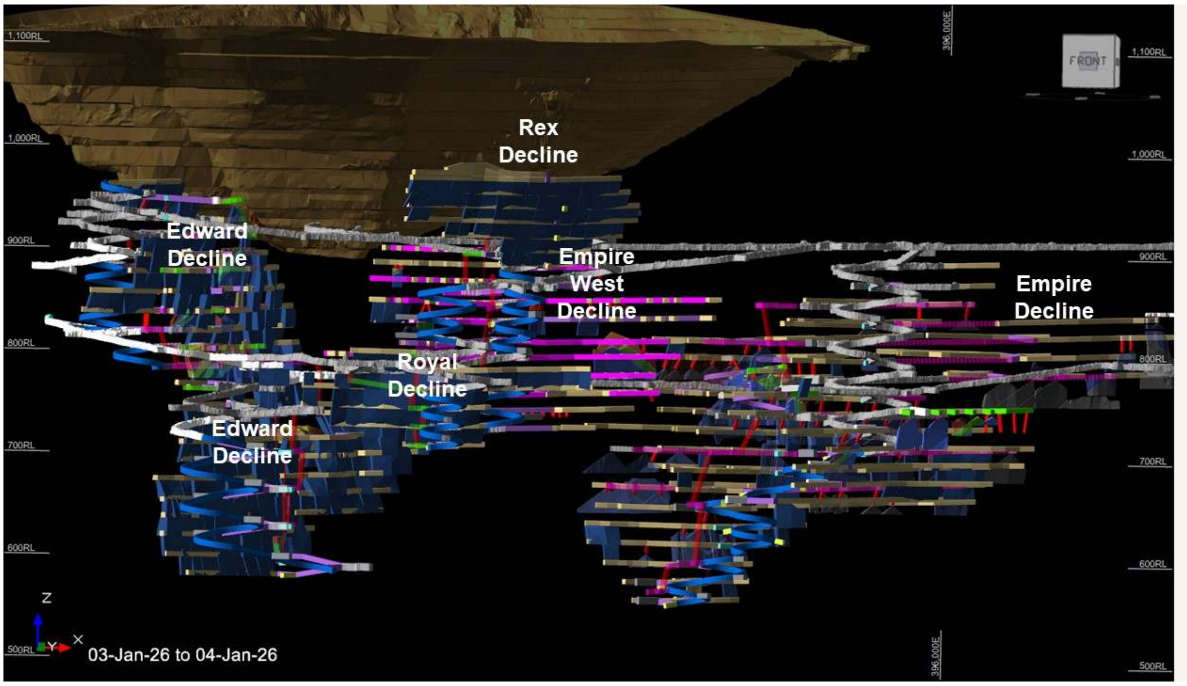
04-Jan-25 to 05-Jan-25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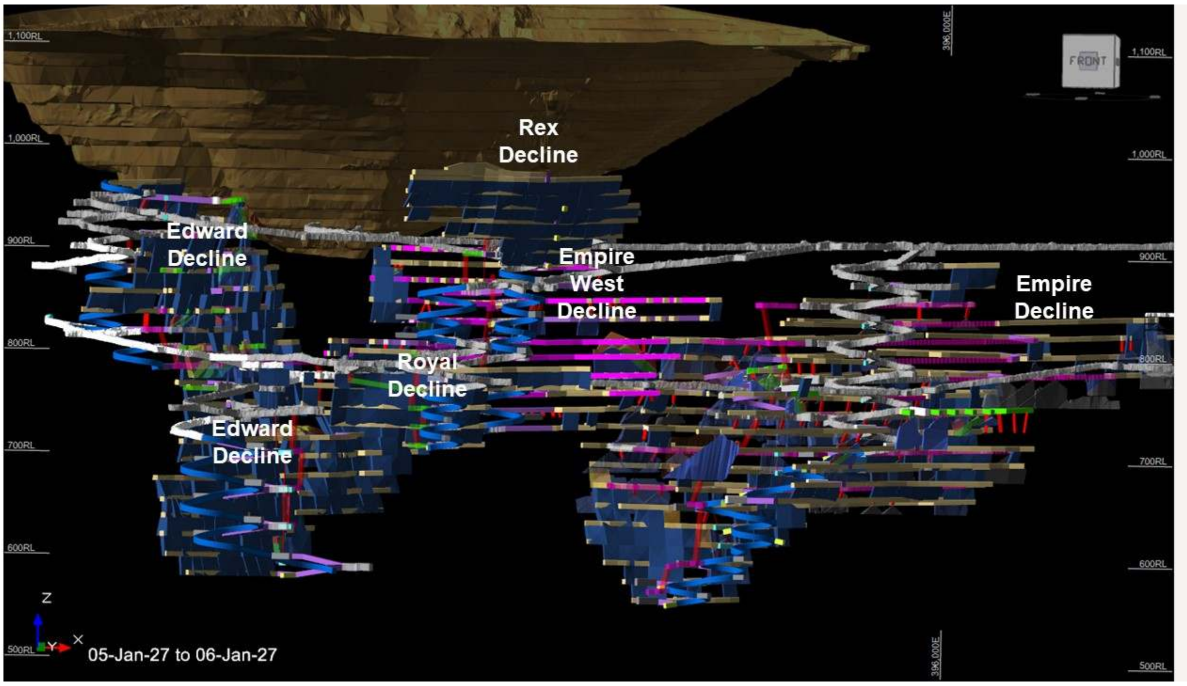
03-Jan-26 to 04-Jan-26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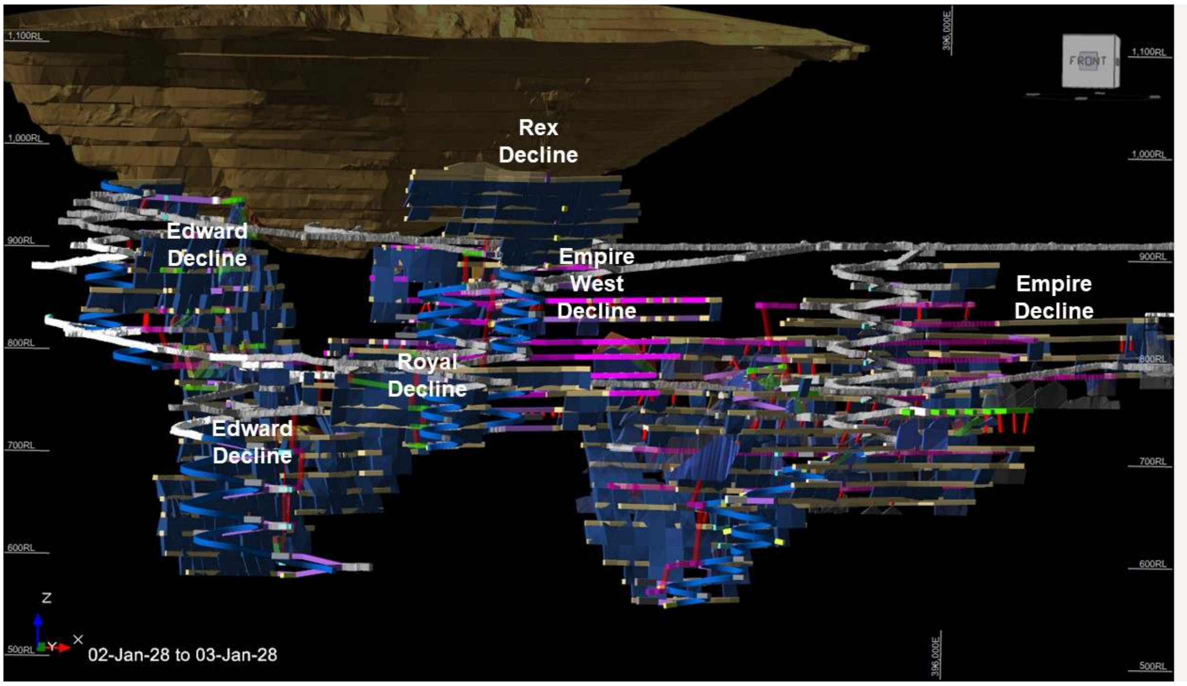
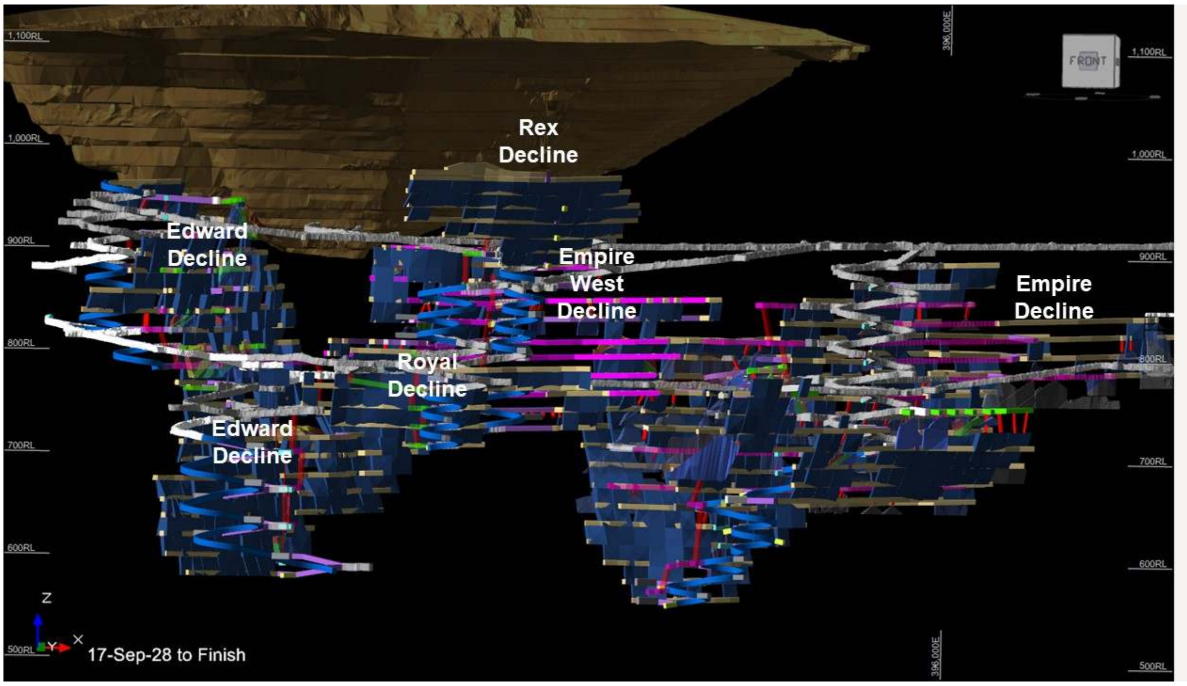
05-Jan-27 to 06-Jan-27





2021

- development of Edward to decline to the target around 690 m RL;
- development of the upper Edward crosscuts and ore drives and commencing stoping on the 800 m RL;
- development of the Royal west decline, ventilation network, ore drives and commencing of stoping;
- connection of Empire decline 920 m RL to 800 m RL, development of 2 levels below 800 m RL;
- development of Rex incline, ventilation infrastructure and commence ore drives and stoping on lowest levels, and

- connection of Edward decline 920RL to 800RL, crosscuts, ore development and backfilling historical stopes.

2022

- Edward decline advanced to 640 m RL, development of return air system, development of footwall drives
- stoping of Rex largely completed to top level prior to minor cut and fill;
- completion of stoping in the Royal west area;
- stoping above a crown pillar in the upper Edward decline are above 780 m RL and completion of infrastructure above this level;
- development of the lower Edward to decline to dewatered depth, around 600 m RL, and
- Empire decline advance to 650 m RL, development of return air system, development of footwall drives and commencing of filling historical workings.

2023

- Rex cut and fill commences and is completed;
- stoping and backfilling on upper western Edward levels above the crown pillar is largely completed;
- decline on lower Edward completed to target depth of 590 m RL, ore drives developed, ventilation system commissioned and stoping commences on lowest levels;
- Empire decline advance to 610 m RL, development of return air system completed, development of footwall drives continues and extensive filling of the historical workings is underway from passes and ore drives developed on lowest four levels;
- footwall drives commenced in Empire west, and
- Lowest levels of Edward deeps completed.

2024

- stoping and backfilling on upper western Edward levels above the crown pillar is largely completed;
- decline on lower Edward completed to target depth of 590 m RL, ore drives developed, ventilation system commissioned and stoping commences on lowest levels;
- Empire decline advance to 610 m RL, development of return air system completed, development of footwall drives continues and extensive filling of the historical workings is underway from passes and ore drives developed on lowest four levels;
- footwall drives continued in Empire west and first stope extracted using the side-ring method, and
- Stoping of the lowest levels at Edward completed.
- Steady state 750tpa achieved.

2025

- All capital mine development completed.
- Empire decline reaches target depth of 580 mL, backfilling of historical workings continues, ore drives largely completed and stoping commences at peripheries of the orebody and on lowest level;
- development of Royal east commences including decline, ventilation connections and ore drives;

- stoping of the Edward vein completed;
- ore development of the Empire east area completed, all historical stopes backfilled and stoping commences on the lowest levels, and
- footwall drives completed in Empire west and stoping progresses using the side-ring method.

2026

- all mine development (ore drives) completed;
- All vertical development completed.
- stoping concentrated in Empire east, Royal east and Empire west, and
- Stoping completed in Empire west.

2027 to finish

- stoping concentrated in Empire east and Royal east.

Table 16-25: Underground Annual Mining Schedule

Key Physicals		TOTAL	2021	2022	2023	2024	2025	2026	2027	2028
Jumbo metres										
Capital waste	**km**	**16.2**	**6.8**	**6.1**	**2.0**	**0.8**	**0.4**	**-**	**-**	**-**
Operating waste	**km**	**17.2**	**0.9**	**1.3**	**3.7**	**4.9**	**3.4**	**2.6**	**0.3**	**-**
Ore development	**km**	**18.0**	**2.0**	**2.7**	**4.4**	**4.0**	**3.3**	**1.5**	**-**	**-**
Total	**km**	**51.5**	**9.8**	**10.1**	**10.1**	**9.7**	**7.2**	**4.2**	**0.3**	**-**
Longhole drilling	**km**	**1,185**	**108**	**147**	**177**	**194**	**167**	**166**	**163**	**63**
Lateral Development tonnes	**kt**	**1,088**	**126**	**173**	**264**	**236**	**197**	**92**	**-**	**-**
Lateral Development grade	**g/t**	**4.4**	**4.6**	**5.0**	**4.1**	**4.4**	**4.2**	**4.0**	**-**	**-**
Stoping tonnes	**kt**	**3,370**	**95**	**319**	**200**	**453**	**559**	**663**	**760**	**321**
Stoping Grade	**g/t**	**4.3**	**4.7**	**6.2**	**3.6**	**4.4**	**3.9**	**4.1**	**4.1**	**4.3**
By Area										
Edward tonnes	kt	1,801	117	259	324	445	533	125	-	-
Edward grade	g/t	4.2	4.8	4.3	3.8	4.6	3.9	4.3	-	-
Empire tonnes	kt	2,104	1	4	114	236	196	613	751	189
Empire grade	g/t	4.1	2.7	2.8	3.7	4.0	4.1	4.1	4.1	4.5
Royal tonnes	kt	367	76	88	9	8	27	18	10	132
Royal grade	g/t	4.8	4.7	6.8	3.3	4.8	3.8	3.9	4.0	4.1
Rex tonnes	kt	186	27	141	17	-	-	-	-	-
Rex grade	g/t	7.32	4.36	7.94	6.85	-	-	-	-	-
Mineral Reserve tonnes	**kt**	**4,458**	**221**	**492**	**464**	**689**	**756**	**755**	**760**	**321**
Mineral Reserve grade	**g/t**	**4.33**	**4.68**	**5.79**	**3.90**	**4.38**	**3.93**	**4.11**	**4.09**	**4.33**
Mineral Reserve oz.	**koz.**	**620**	**33**	**92**	**58**	**97**	**96**	**100**	**100**	**45**
Development waste	**kt**	**2,353**	**569**	**530**	**404**	**389**	**263**	**176**	**21**	**-**
CRF / CAF	**kt**	**1,242**	**69**	**62**	**359**	**348**	**110**	**110**	**136**	**47**
Rockfill	**kt**	**2,504**	**75**	**262**	**152**	**288**	**417**	**514**	**552**	**244**

16.16.1 Mining Schedule Risk Profile

Ore tonnes have been grouped within the schedule by mining method and proximity to historic workings. Four mining methods are proposed for the mine:

- AVO - Modified Avoca with rockfill in virgin (previously unmined) areas.
- RMO - Modified Avoca with rockfill in remnant areas adjacent historical stopes filled with an engineered fill (CRF / CAF)
- RMC - Modified Avoca with rockfill in remnant areas adjacent to collapsed stopes separated by an intermediate pillar.
- RMF - Bottom up side ring method with CRF/CAF/RF where skins adjacent to historical backfill are extracted.

A small proportion of the Mineral Reserve will involve the extraction of remnant skins in the footwall or hangingwall of previously mined (historical) stopes, or the extraction of both remnant skins. The mining methods have been explained in Chapter 16 of this report.

The mine production schedule has been reviewed in terms of the mining methods and the risk rating as discussed below. Risk is defined in terms of the recovery and dilution of the resource, and not HSLP risks.

- Modified Avoca in virgin areas (AVO) and lateral ore development pose very low risk in terms of ore extraction as has been demonstrated over the last fifteen years at Favona. Lateral ore development represents 25% of the mined oz. and 24% of the mined tonnes. Avoca in virgin areas represents 54% of the mined oz. and 55% of the mined tonnes
- Where the historic stope can be backfilled with an engineered CRF (RMO) than the wall can be considered geotechnically stable and again the risk is considered to be low in terms of recovery of the resource and dilution although slightly higher risk than mining in virgin areas. Avoca next to filled historical stopes represents 11% of the tonnes and oz.
- Higher risk areas are defined when mining adjacent to collapsed historical stopes (RMC) which because they have partially caved cannot be filled with an engineered fill. Avoca next to collapsed historical stopes represents 5% of the tonnes and oz.
- The highest risk area is expected when recovering wall rock adjacent to historical backfill (RMF) where there is a high likelihood of some historical fill material reporting to the stope and either high dilution is taken or the stope is abandoned. Avoca next to historical backfill represents 5% of the tonnes and oz.

Appropriate modifying factors have been applied to the mining methods as detailed in Table 16-21

In terms of the schedule in the first 4 years 90% of the production comes from lateral development and stoping in virgin areas and 72% in the following four years which is considered low risk.

In terms of the higher risk mining, this represents 5% of the production in the first four years and 12% in the following four years. This is shown in Figure 16-38 and Figure 16-39

Figure 16-38: FS Stoping Method by Year – Contained oz. Au

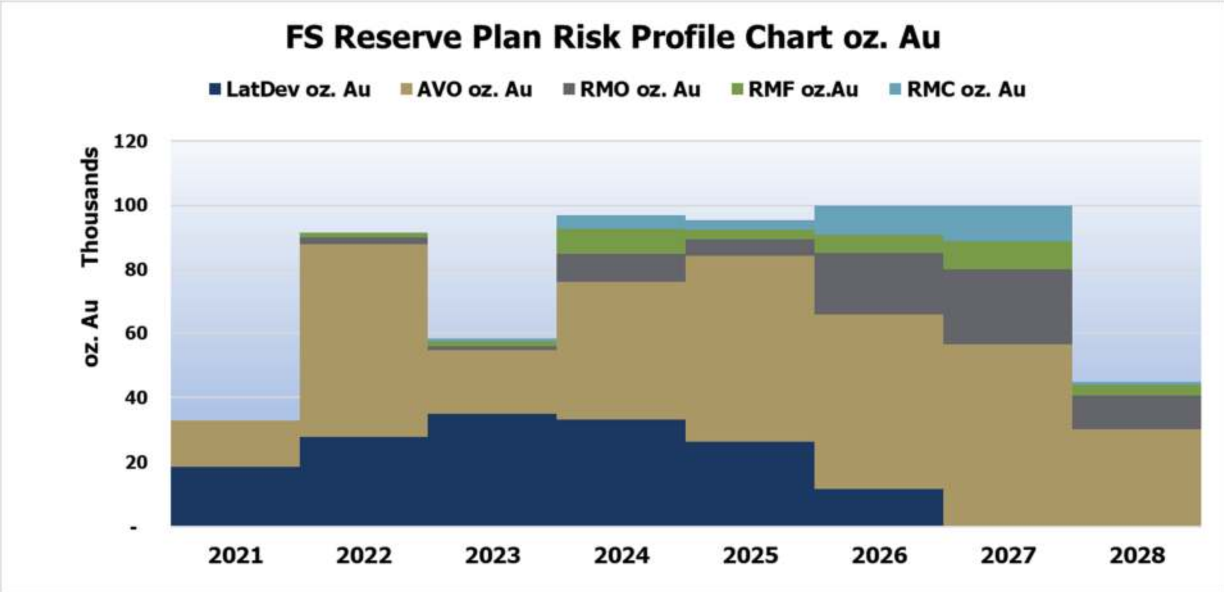
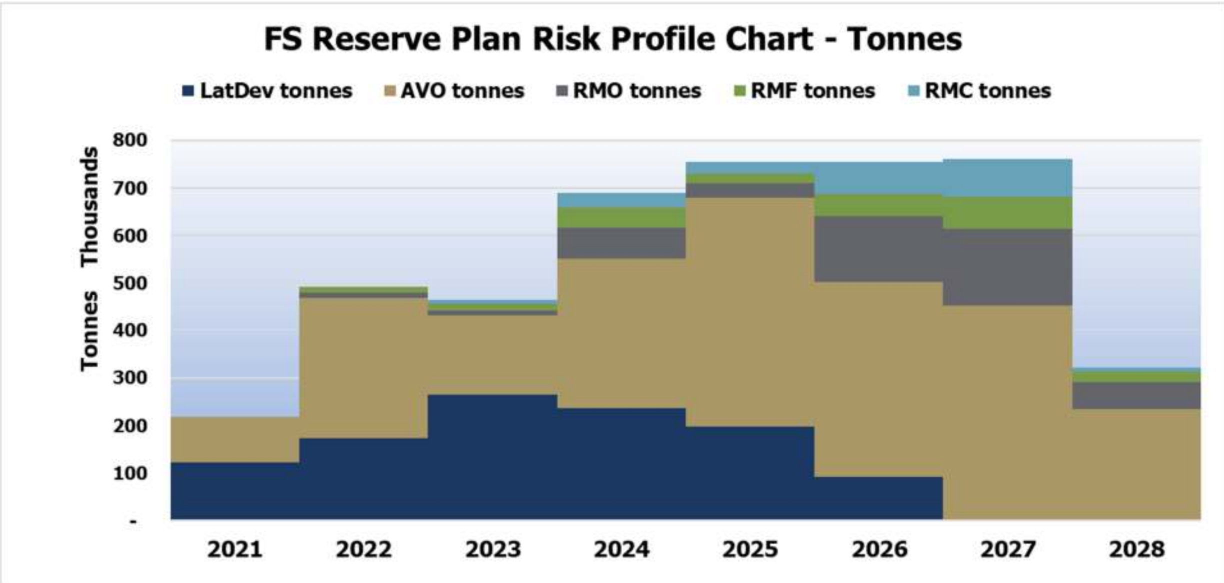


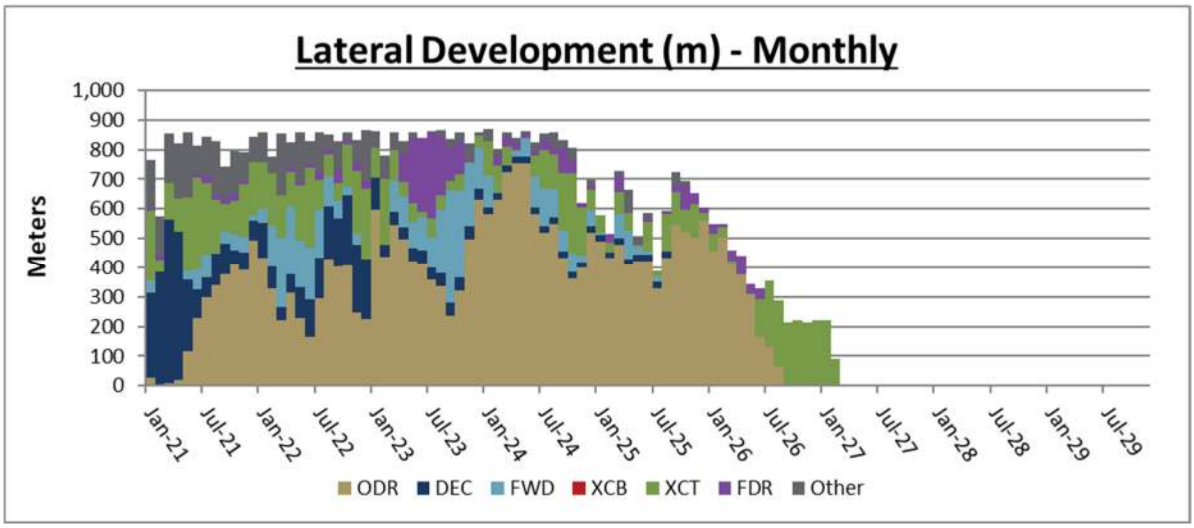
Figure 16-39: FS Stoping Method by Year – Tonnes



16.16.2 Jumbo Development

The maximum instantaneous decline advance rate was assumed to be 120 m/month, equivalent to nine 3 m rounds per week. The instantaneous rate for all headings reduces as the heading approaches any known void. The schedule has a total of five separate jumbos planned for mining of the separate lodes. The LOM horizontal development schedule is shown in Figure 16-40.

Figure 16-40: Monthly Jumbo Development Advance (m)



The development of the return air drives was scheduled to occur "as soon as possible" as it is important to gain access to the primary ventilation circuit as the mine progressed. Other priority headings to be established were the DD platforms and declines. Most other development activities such as ore driving, waste development and vertical development are generally scheduled "as late as possible" to ensure a smooth workload for the development jumbos.

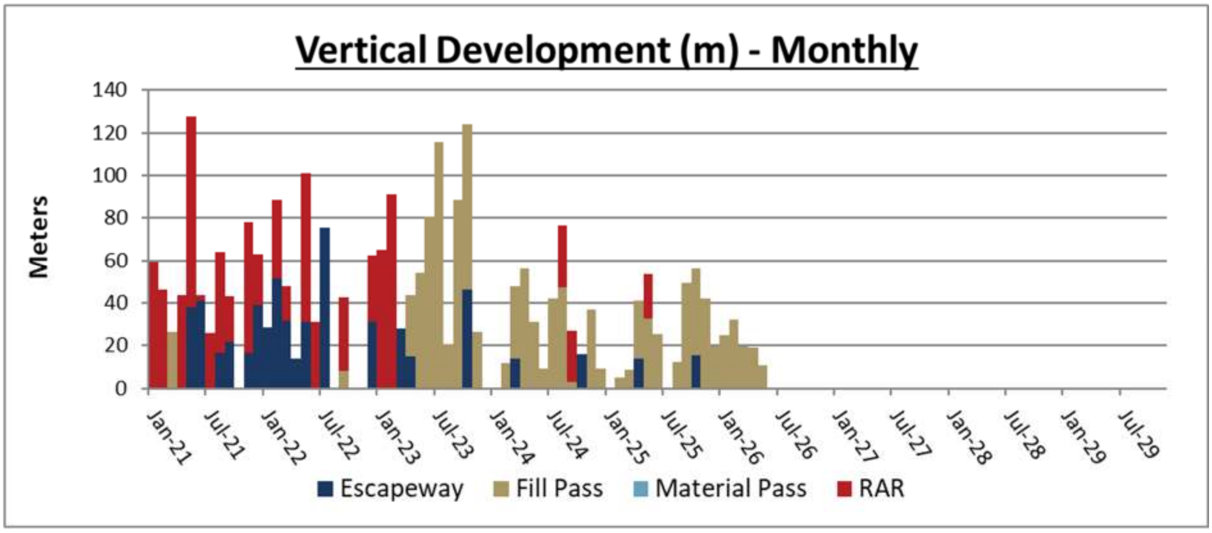
Advance rates have been assigned to development drives according to priority with the remainder of a jumbos available metres being distributed according to mining sequence. Task priorities followed were:

1. Establish primary ventilation drives;
2. Develop initial diamond drill platforms;
3. Establish capital level access and primary ventilation drives;
4. Push decline to establish multiple production horizons; and
5. Develop ore drives as fast as possible (working bottom up).

16.16.3 Vertical Development

Vertical development is by longholing and comprises escapeways, fill passes, material passes and return air rises. The LOM vertical development schedule is shown in Figure 16-41. Return air and escapeway development is prioritised in the early years with fill pass development for filling historical stopes in Empire scheduled later as part of the stoping sequence.

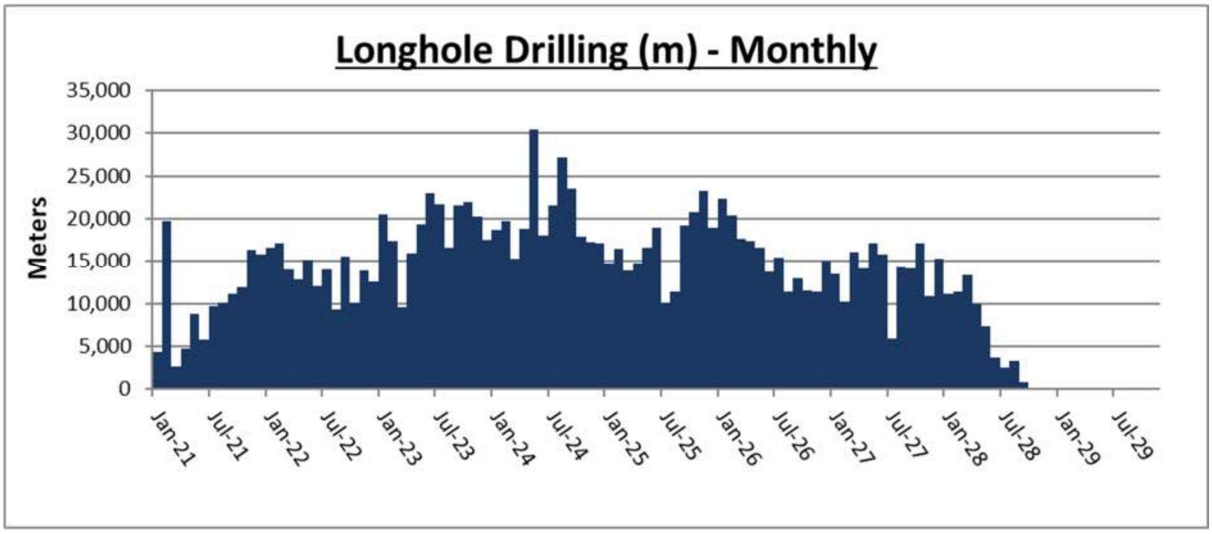
Figure 16-41: Vertical Development (m)



16.16.4 Production Drilling

Longhole drilling will be carried out by a single Sandvik DL431 - 7C (or similar) drill scheduled at a maximum rate of 160 m/d. The drill hole diameter is 64 mm and average length 20 m. Drilling requirements are illustrated in Figure 16-42 including cable bolting and probe drilling.

Figure 16-42: Monthly Stope Drilling (m)



Production drilling requirements ramp up over the first 24 months with the first side ring drilling commencing in month 18. Production drilling has been scheduled "as soon as possible". Drilling of a stope only commences on the completion of the entire stope cycle of the prior stope i.e. a stope must be drilled bogged and filled before the drilling of the next stope can commence. The mine carries very little drill stocks (stopes that have been drilled but not yet fired).

Longhole drilling comprises 971km stope production drilling, 113km, vertical development drilling, 48km cable bolt drilling and 230km probe drilling.

16.16.5 Production Charging

Production charging has not been scheduled as a separate activity however, the drilling and bogging rates applied account for the time required to charge each stope.

16.16.6 Stope Bogging

A production activity was created for each stope, scheduled within Deswik using the following targets:

- for all stopes, all level development, vertical development to be completed prior to stoping;
- maximise grade as early as practical, and
- a sustainable underground ore production rate of 750,000 t/y.

Stoping commences in end 2021 from the eastern end of Edward lode on the 854 m RL and ramps up to full production over a 16-month period as shown in Figure 16-43.

Figure 16-43: Monthly Ore Production (t)

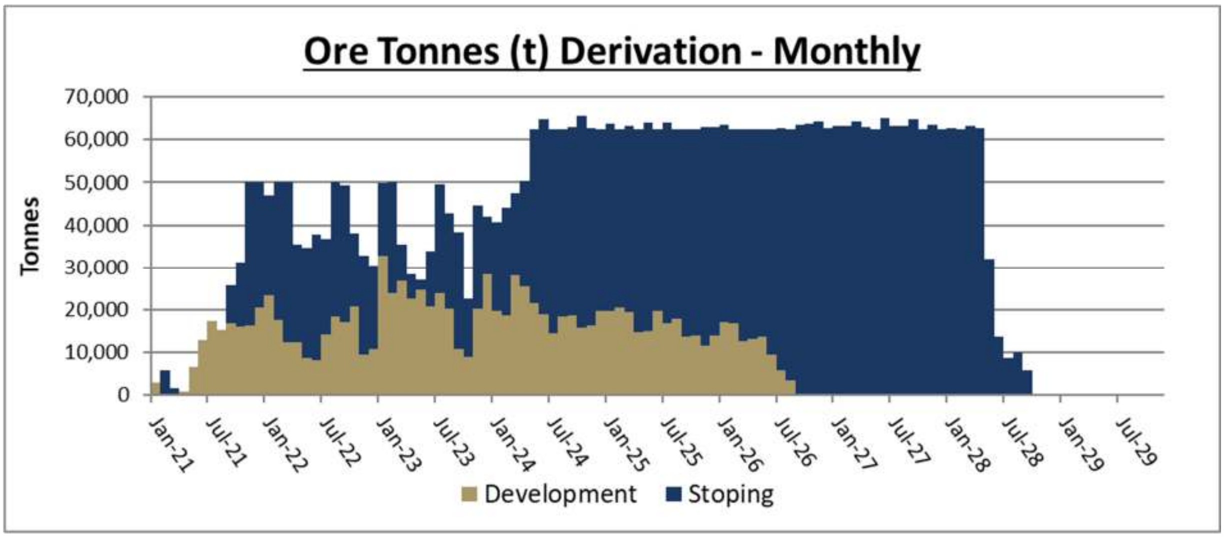
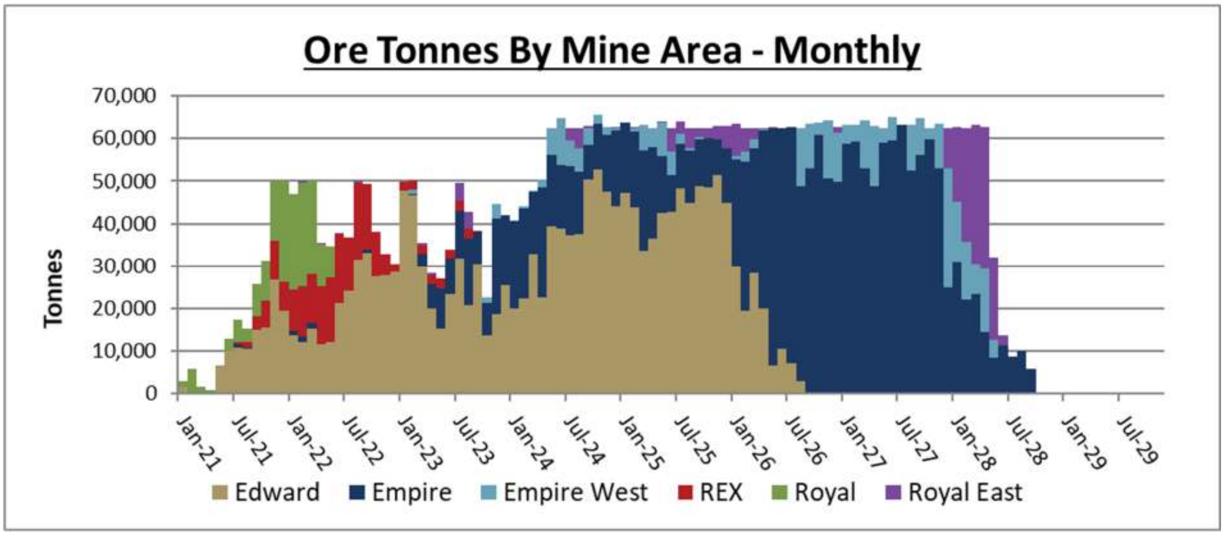


Figure 16-44: Monthly Ore Production (t) by Area

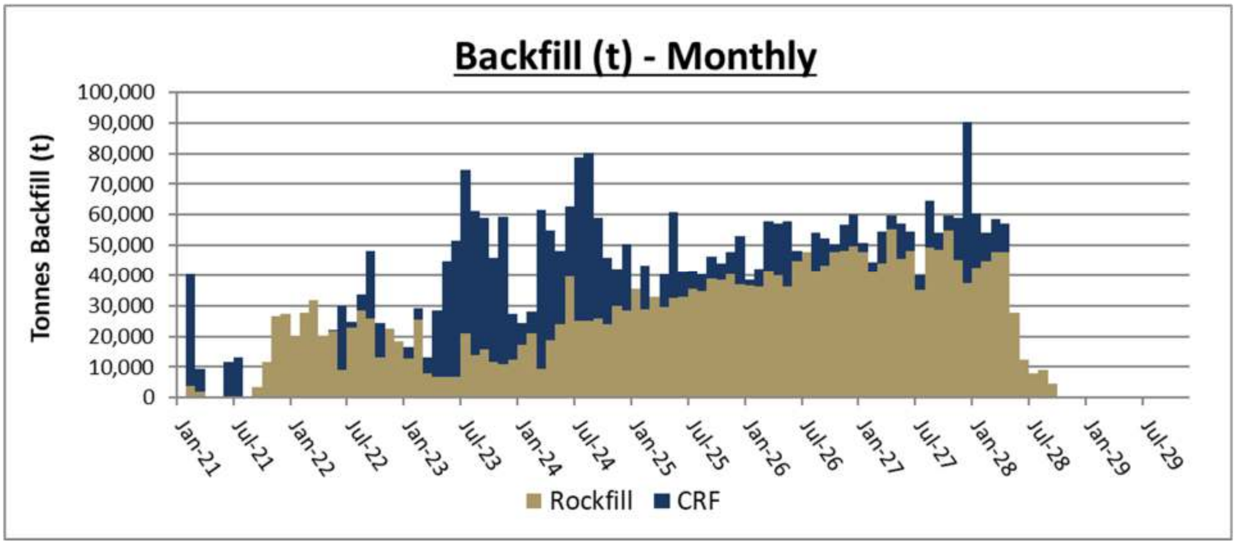


16.16.7 Backfilling

Backfilling of stopes is carried out with a mixture of unconsolidated RF and CRF. All filling activities have been scheduled to be completed using the stope loader. Filling of stopes has been scheduled at rates outlined in and commences after a 24-hr delay upon the completion of stope bogging.

It has been assumed that filling of all historical voids is completed with CRF whilst the filling of the stoping voids will be completed with a mixture of CRF and unconsolidated rockfill (RF).

Figure 16-45: Monthly Backfill (t)



16.16.8 Haulage

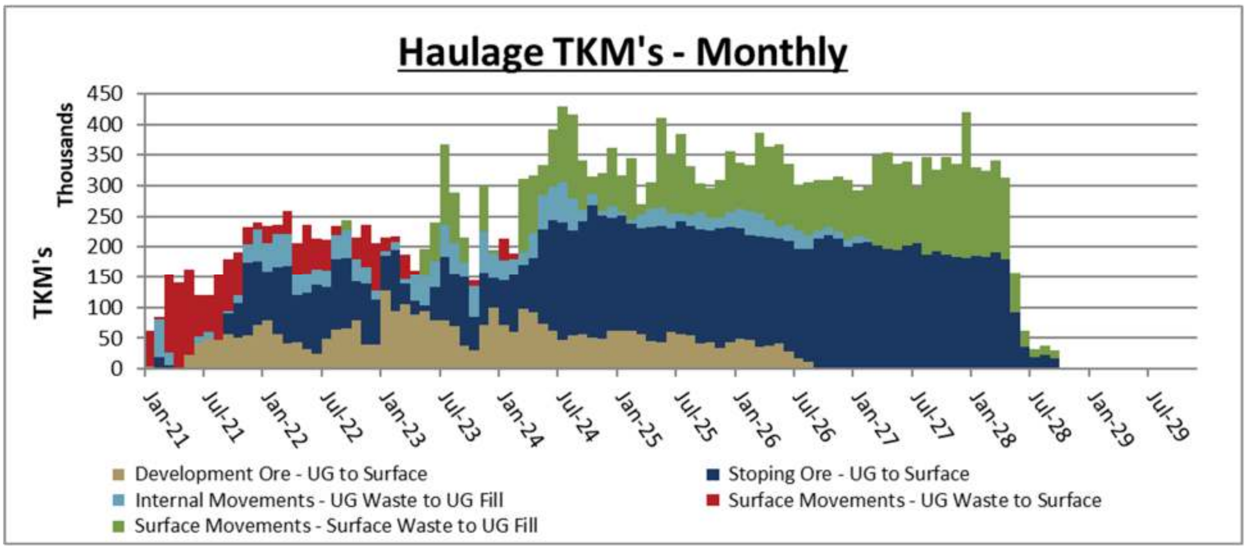
The haulage requirement from the mine, inclusive of all ore and waste material is illustrated in Figure 16-45 and is stated on a tonne-kilometre (tkm) basis.

The planned material movement requires a maximum of six trucks, with trucks being required to be mobilised to achieve the plan on average every five to six months as the mine ramps up to full production. The number of trucks is based upon haulage capacity for a 50 t haul truck (approximately 50,000 tkm/month).

The haulage calculation has assumed that all material is hauled from the level access to the surface through the current portal position. It is unlikely that all waste will be hauled through the portal as some material will be used for stope fill as the schedule permits without being hauled the extra distance to the portal.

Figure 16-46 shows the monthly haulage profiles for the various movement categories, internal movements and underground to surface and surface to underground for ore and waste.

Figure 16-46: Monthly Haulage Requirements (tkm)

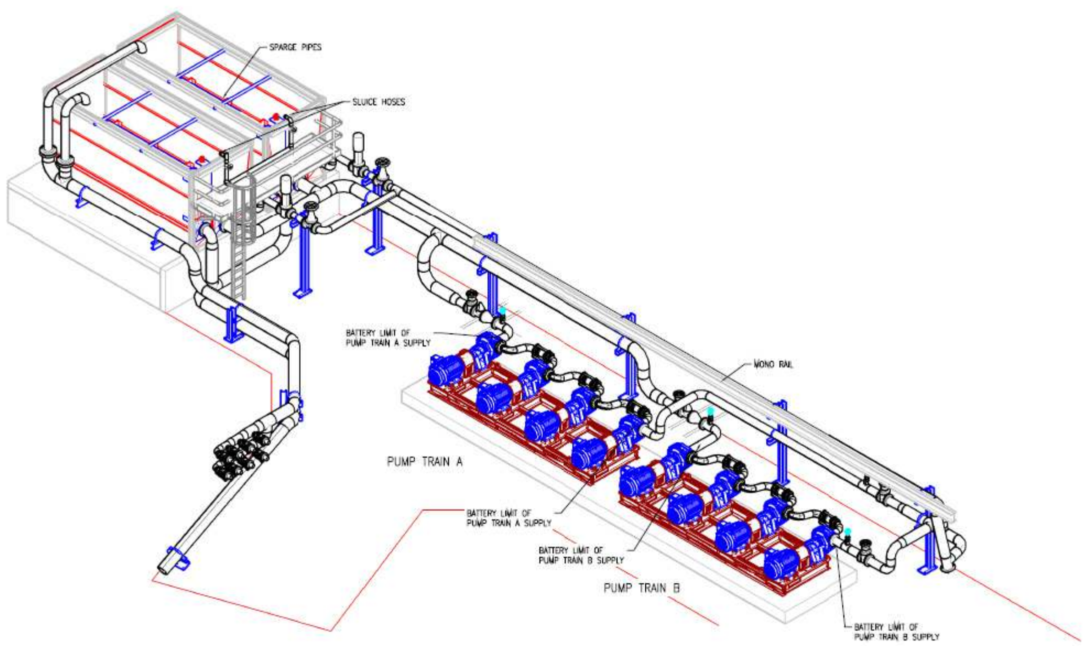


16.17 Mine Services

16.17.1 Primary Pumping Station

The mine primary pump station is installed in the upper levels of Correnso at 794 m RL and consists of two trains of four Wier 6/4 AHPP-08 centrifugal multiflow pumps long coupled, horizontal shaft type, and mounted on a 2.4x 6 meter skid. The pumps are each powered by Teco 132kW motors with independent variable speed drives. The pump chamber includes two modified sea containers equipped with level float switches and sparging for temporary storage of water. The general layout is shown in Figure 16-47

Figure 16-47: Martha Primary Pumping Station Layout



Source: AECOM Pump Station Design Review Report, 2015

Two 250NB carbon steel 126.6 meter rinsing mains are installed from the pump chamber (RL 794) in a nearly vertical shaft rising to RL 912. A single 300 NB carbon steel rising main is installed from the top of the vertical riser (RL 912), and along the Correnso access tunnel until RL1010, where water pressure is less than 16 bar. At RL1010 the pipe is split to into two runs of High Density Poly Ethylene (HDPE) piping rated to a nominal pressure of 16 bar(g) (PN 16) and has an outer diameter of 250 mm. These pipes run to the mine portal surface 1124RL then onto the water treatment plant. Flow monitors are installed at various points along the routes.

16.17.2 Borehole Pumps

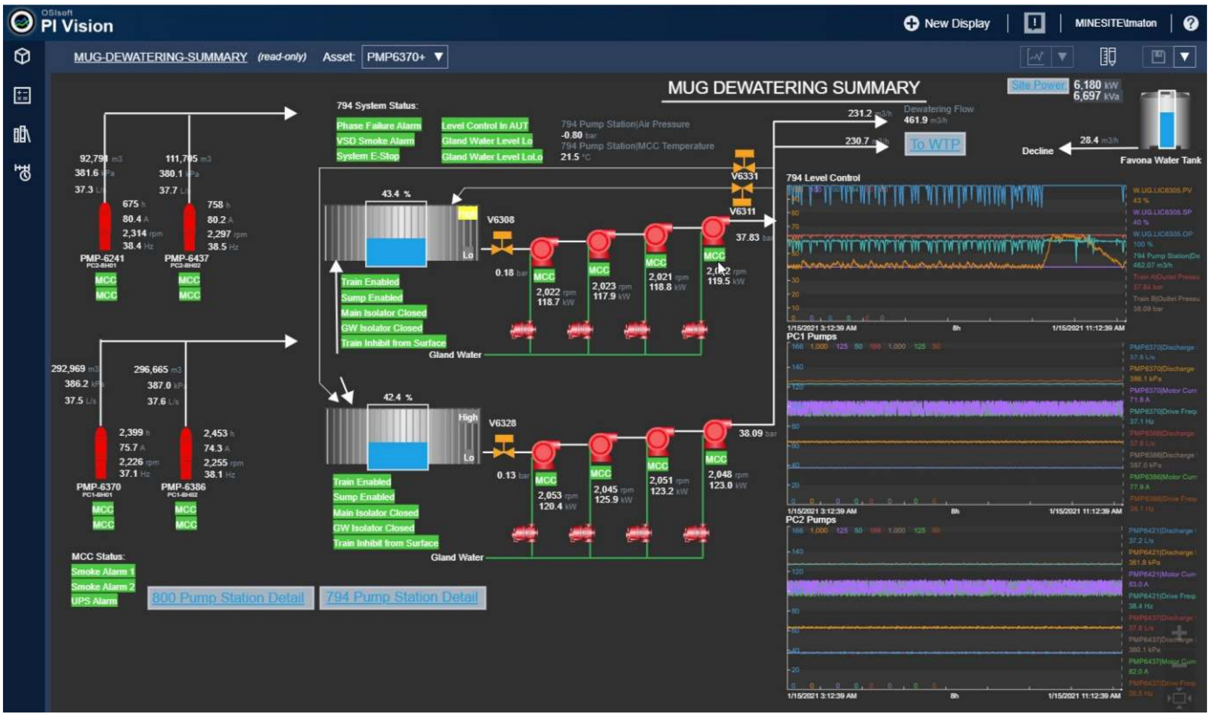
Four dewatering bores were commissioned in 2020, namely PC01-BH01, PC01-BH02, PC2-BH03 and PC02-BH04, targeting old mine workings and were drilled from 2 access caddies along the 800 drill drive. It is expected that these bores will reduce groundwater levels from the original 730 m RL down to 620 m RL (110 m head).

Each borehole pump comprises of Franklin multistage units, model VSC156 submersible pump complete with VSD enabled electric motor. The boreholes are inclined (sub-vertical), and as such the pumps will be fitted with ‘centralisers’ to ensure they do not rest on the casing, which can cause performance reduction and potential issues with cooling of the pump motor

The pumps are installed inside a borehole liner, which is a steel pipe that has apertures in the lower section of the pipe to allow the borehole to be flooded with groundwater inflows. The apertures are sized for the system target flow range, and also to prevent solids entering. The pumps are coupled and supported by a Riser. The Riser is connected into a steel headworks piping arrangement at the borehole collar. From the headworks, HDPE piping is installed to connect the bore pumps with the 794 Pump Station reservoirs. The HDPE piping is a common manifold, which all bore pumps discharge into and tied into the reservoir inlet manifold at 794 Pump Station.

Water is pumped from the borehole pumps in parallel to the 794 Pump Station reservoirs. The flow rate from each borehole pump can be manually set by an operator using the PLC HMI. The system has a target flowrate of 37.5L/s from each borehole pump (150L/s combined inflow to the 794 Pump Station). The system can operate continuously (all pumps are duty pumps) unless there is unavailability of the 794 Pump Station (reservoirs are full or if the 794 Pump Station is isolated). The primary pumping layout is shown schematically in Figure 16-48.

Figure 16-48: Martha Primary Pumping Layout



Source: OGC OSIsoft PI vision

16.17.3 Pumping Below 610 m RL

At this stage, the dewatering bores installed at PC01 and PC02 are expected to drop the groundwater level to 610 m RL and further dewatering will be required to reach 560 m RL. Below are possible dewatering options from 620 to 560 m RL.

- Passive dewatering: whereby you mine into the water (wet mining) with no prior dewatering, the groundwater seeps onto the floor or walls through structures, is diverted to drains and dedicated sumps and then pumped out. This approach can slow development/production rates, create unsafe working conditions, extra road maintenance and require dirty water management and treatment.
- Dewatering bores through active dewatering: this approach is when dewatering occurs ahead of mining, as currently PC01 and PC02 are doing. The option to install additional dewatering bores targeting the old workings down to 560 m RL should be investigated as would provide dry and safe mining conditions (cost benefit analysis). Success will depend on the extent of old stopes and the extent to which they are still open and connectivity with the vein systems.
- Combination of active and passive dewatering: this method is practiced at Correnso, Trio and Favona mine. When high inflows are intersected, diamond drilled holes are drilled into the area to dewatering the area faster and allow dryer working conditions in developments going forward. The water from the drill holes is dropped onto the floor, directed through drains to dedicated sumps and pump stations.

The preferred approach to dewater levels from 620 to 560 m RL, would be active dewatering as it ensures dry and safe mining conditions, reduces pump maintenance and lessens amount of water treatment required. This approach is not always feasible with the cost and time to install dewatering bores. The subsequent best option is to do a combination of passive and active dewatering approach as already practiced and familiar with at Correnso, Trio and Favona but with targeted dewatering using diamond drill holes (cover drilling or from jumbo water intersection). An improvement option would be to plumb the diamond drill holes and pipe water direct to pump stations, which would reduce dirty water management and road/drain maintenance. Clean water from the diamond drill holes could also be used as back up mine water supply for equipment.

16.17.4 Secondary Pumping

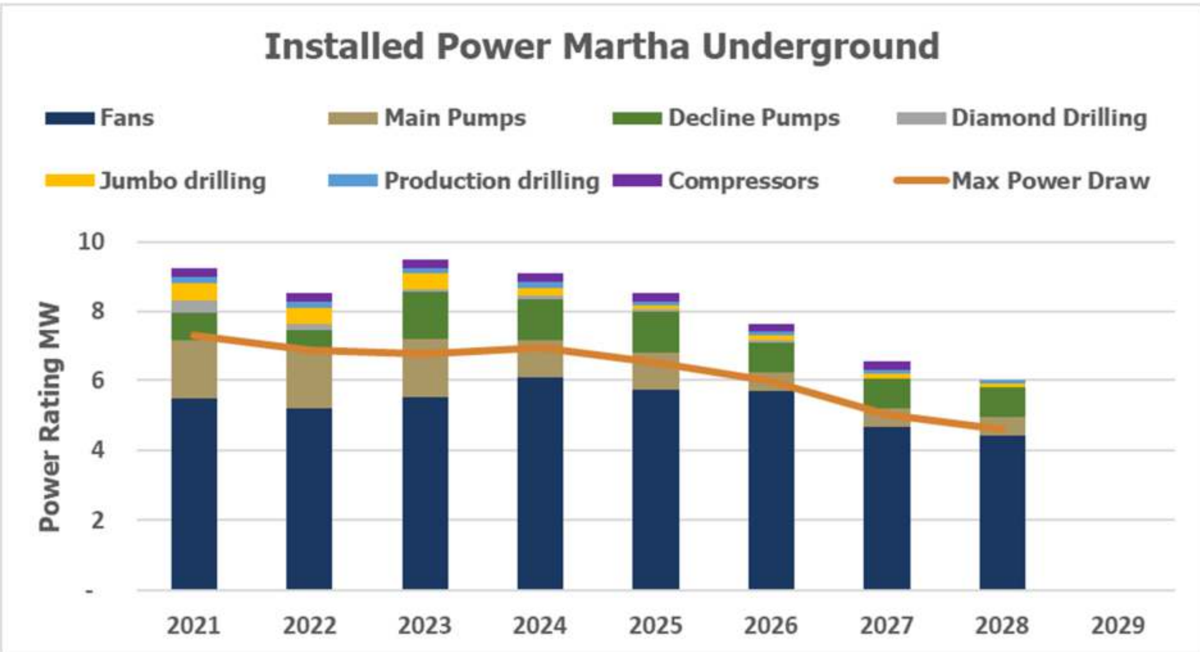
The secondary dewatering system will typically include 8 kW and 20 kW electric submersible pumps (e.g. Flygt pumps) and a small number of re-locatable helical-rotor pumps (often referred to as “travelling mono’s”). These secondary pumps will transfer water from sumps located in the active headings to the primary pumping system and will be supplied by the underground mining contractor.

The secondary pumping strategy for will be to pump or in most cases gravity FW to sumps located in the level access development. This water is then gravity fed through drain holes linking level sumps and pumped or directed to the primary pumping stations.

16.17.5 Electrical

Power to the underground is primarily required for the main and secondary fans and also the primary and secondary pumps with minor usages for jumbo, production and diamond drilling. The installed power over time is shown below in Figure 16-49.

Figure 16-49: Martha Underground Installed Power



Initially the main power feed will need to come from the current workings in Correnso however, it would be ideal for a new High Voltage (HV) feed to the underground mine will via the MOP breakthrough. The long-term HV feed will be reticulated throughout the mine via a system of service holes ideally drilled from cuddies to avoid long cable runs down the decline. Removing the cable from the decline also removes it as a hazard as it is less likely to be hit by heavy equipment.

It is assumed the underground feed will be distributed through the mine at 11 kV to four number 1.5 MVA 11 kV - 1 kV step down transformers. Power supply to the primary underground fans will via a single 2 MVA 11 kV - 415 V transformer.

The ability to have a ring main system by installing the electrical infrastructure in this manor may be beneficial enabling power to be supplied to the eastern areas such as Correnso and Trio from Martha and vice versa in the event of power loss from one of the feeds.

Various substations will be located within the mining areas associated with each decline. From these substations power will be distributed via cable holes or in the decline depending of the ability to drill cable holes in suitable locations.

It is assumed that throughout the underground 4 x 1.5 MVA 11 kV / 1000 V transformers will be located at approx. 150 m vertical intervals to ensure sufficient power supply to all sections of the mine. One 2 MVA 11 kV / 415 V transformer will be located in close proximity to the primary underground fans.

The underground mine connected load including all equipment at full capacity is approximately 7 MW. The LOM average usage is 43 GWh per annum with a peak usage of 63GWh.

16.17.6 Health & Safety

The mine design incorporates New Zealand Health & Safety Act safety standards. 12 person and eight (8) person mobile refuge chambers are included and will be located in active working areas over the LOM.

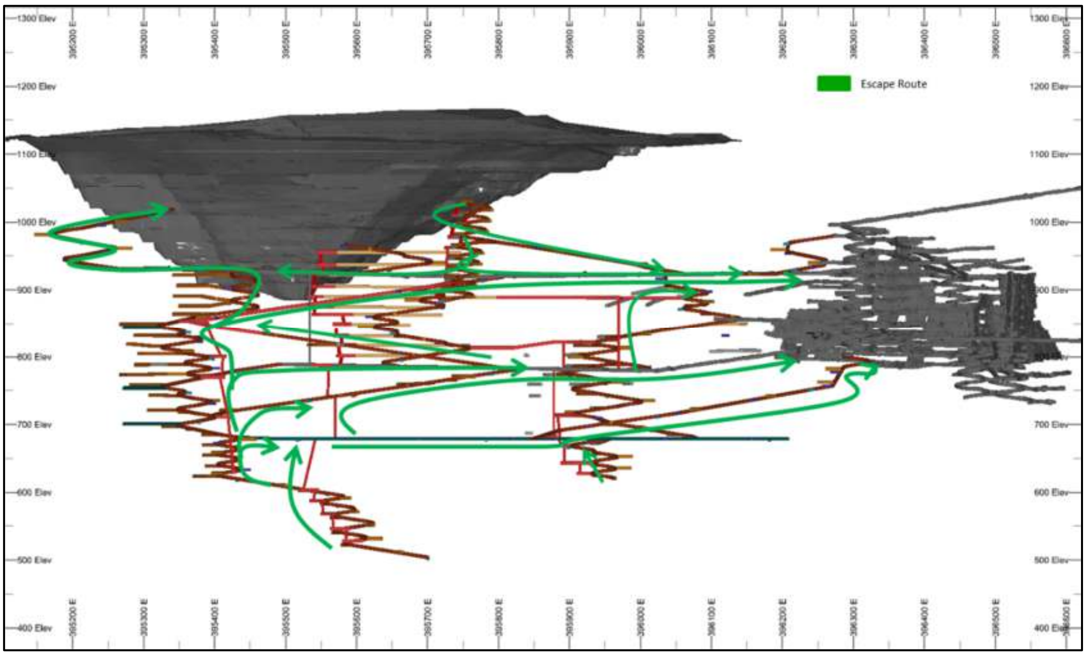
The mine will have a communications system that has both mine phones and wireless communication through a leaky feeder system. A mine rescue team will support the operation. The mine safety program will integrate with local providers in case of any mine emergency. A stench gas emergency warning system will be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Second means of egress from the underground mine will be established in accordance with health & safety regulations to enable personnel to exit the mine in the case that one egress (e.g. decline) becomes blocked.

To ensure that all personnel can reach a connection drive, ladderways will be installed in the return air way rises. The ladderways have been assumed to be constructed from galvanised steel in a modular form and will include protective caging and rest platforms.

There are only two areas of the mine that will be dependent on ladderways alone, those being below the 680 m RL drill drive in the Edward decline and Empire decline. These areas have no lateral connection between them hence the only way to develop a secondary means of egress is via the ventilation rises. All other areas of the mine have multiple connections between them via drill drives or access declines.

Figure 16-50: Martha Escapeways



Access to the ladderways will be through man-doors built into the shotcrete walls. In certain cases, escaping the mine via second means of egress in an emergency may not be possible or may not be safe. Some examples would be a mobile equipment fire that creates large amounts of smoke which would prevent the use of the escape ladderways or blockage of a single entry heading preventing personnel in that heading from escaping the drive.

To provide refuge for underground personnel in such circumstances, re-locatable refuge chambers will be installed in the mine. The chambers will vary in size from 4 to 20-man capacity. Two four (4) man portable, battery powered chambers will be used throughout the mine development as these chambers are easily moved by an I.T. into single entry headings where there is the risk of entrapment (for example when a jumbo operator is working at the decline face with trucks being loaded between the jumbo and the nearest access to an escapeway, thereby creating an entrapment situation). The chambers will be purchased and installed progressively as the mine is developed deeper.

The actual locations of the chambers will change over time and will be dependent on the numbers of personnel typically working in each part of the mine.

The total number of personnel working in each part of the mine will be limited to the refuge chamber capacity and should be controlled by tag board procedures.

16.17.7 Manpower

Manpower levels are estimated based on the production schedule and associated equipment operating requirements. The estimate is based on owner mining using a mine operating schedule consisting of 12 hours per shift, two shifts per day, and 7 days / week. Each 12 hr shift is supported by a four-crew rotation working a 4 x 3 roster (i.e., 26 weeks of the year the crew will work 4 days and 26 weeks of the year the crew will work three (3) days). The 4 x 3 roster results in 1,976 hours per year being worked at regular wage rate and 208 hours per year being worked at the overtime wage rate (i.e., time and a half). The management and technical team will work five 8 hr days per week. The table below shows the required workforce.

Table 16-26: Typical Mining Labour by Day Shift

Day Shift	Qty	
Underground Mine Manager	1	
Site Admin	1	
UG Engineering Superintendent	1	
Senior Engineer	2	
Principal Geotechnical Engineer	1	
Geotech Engineer	2	
Planning Engineer (Long-Term)	1	
Development Engineer	1	
Production Engineer	2	
Ventilation Engineer	1	
Senior Mine Geologist	1	
Project Geologist	1	
Mine Geologist	2	
Mine Tech	2	
Senior Surveyor	1	
Mine Surveyor	3	
Survey Tech	2	
Smart Centre Operator	4	
Shift Coordinator	2	
Mine Systems Tech	1	
Mine Foreman	2	
Maintenance Superintendent	1	
Maintenance Foreman	2	
Maintenance Admin	1	
Maintenance Planner	1	
Dayshift Fitter	2	
Dayshift Auto Electrician	1	
UG Pump Fitter	1	
UG Maintenance Trade Assistant	1	
Total	**44**	

Table 16-27: Typical Mining Labour by Rotating Shift

Rotating Shift	Per Shift (Qty)	Total (Qty)
Shift Supervisor	1	4
Jumbo Operator 1	2	8
Jumbo Operator 2	1	4
Jumbo Operator 3	1	4
Prod Driller 1	1	4
Prod Driller 2	1	4
Air Legger	2	8

Senior Charge-Up - Development	1	4
Senior Charge-Up - Production	1	4
Charge-Up - Development	1	4
Charge-Up - Production	1	4
Remote Bogger	2	8
Bogger Operator 1	1	4
Bogger Operator 2	2	8
Bogger Operator 2	1	4
Service Lead Hand	1	4
Service Crew	3	12
Truck Operator 1	1	4
Nipper 1	1	4
Truck Operator 2	2	8
Truck Operator 3	3	12
Trainee Miner	1	4
UG Stores Operator	2	2
Mine Infrastructure	3	3
Shotcreter/Sprayer	3	3
Agi Driver	1	1
Grader	2	2
Diesel Fitter Leading Hand	1	4
UG Diesel Fitter 1	1	4
UG Diesel Fitter 2	2	8
UG Auto Electrician	1	4
UG maintenance Serviceman	1	4
Senior Light Vehicle Fitter	1	1
UG Light Vehicle Fitter	1	1
UG Elect Superintendent	3	3
Project Electrician	1	1
Senior UG Electrician	1	2
UG Electrician 1	1	4
UG Electrician 2	1	2
Comms Technician	2	2
Total	**58**	**171**

The wages workforce is unionised and hourly pay rates and benefits are determined from annual negotiations with the union. The estimated labour rates are consistent with the existing underground wage scale.

16.17.8 Equipment

The underground mobile equipment requirements as summarised in Table 16-28 are based on the production schedule and an assumed LOM mechanical availability of 85%.

The yearly fixed equipment list is shown in Table 16-28. This information is located in the cost model spreadsheet (OGC_Cost_Model_UG_FS_Waihi_Indicated_final.xlsm).

Mine equipment requirements have derived from first principles' calculations. The listed equipment types have been determined following a review of the mine design and schedule. Table 16-28 provides a summary of the peak number of units required for mine development and production. These resources are planned to perform the following duties:

- excavate the lateral and decline development in both ore and waste;
- install all ground support including rockbolting and surface support;
- maintain the underground road surfaces;
- drill, charge and bog (including remote bogging) all stoping ore material;
- drill slot rises for production stoping;
- install all underground services for development and production.

Table 16-28: Underground Peak Equipment Requirements

Equipment	Model	Max Number
Development Drills	Sandvik DD421-60C	5
LH Drills	Sandvik DL421-7C	3
Loaders	Sandvik LH517i	5
Trucks	Sandvik TH551	5
Spray mech	Spraymec 6050WP	1
Agi		1
Wheel loader large	Cat 962H	2
Wheel loader small	Cat 938K	1
12K Grader	Cat 12K	1
Charge-up	Charmec MC 605	2
Scissor Lift	Utimec Scissor	1

The Martha underground mining equipment fleet that is currently in place comprises:

- Lateral Development face drilling and bolt installation with five number two (2) Boom Jumbo (Axera and Sandvik DD421).
- Loading using Cat 1700 or 2900 underground loaders, fitted with remote mucking capability.
- Trucking using Atlas MT5020 50t underground trucks.
- Solo 5V long hole drill rigs capable of drilling in the diameter range of 51 mm to 76 mm and up to 25 m hole length.
- Ancillary equipment (explosive charge vehicles, shotcrete equipment, integrated tool carries, grader).

It is planned to transition the Martha underground fleet to a largely Sandvik manufactured fleet replacing the existing fleet once it reaches its economic life. The loaders fleet will be transitioning to Sandvik 515i loaders by 2023 and trucks will be transitioning to Sandvik TH551i trucks by 2023. Fleet sizes and mobile plant replacement schedule are shown in Table 16-29.

Table 16-29: Underground Equipment Schedule

	2021	2022	2023	2024	2025	2026	2027	2028
Development Drills	5	5	5	5	5	3	1	-
LH Drills	2	2	3	3	3	3	2	1
Loaders	3	4	5	5	5	5	5	2
Trucks	3	3	4	5	5	5	5	2
Spray mech	1	1	1	1	1	1	1	1
Agi	1	1	1	1	1	1	1	1
962H	2	2	2	2	2	2	1	1
938K	1	1	1	1	-	-	-	-
12K Grader	1	1	1	1	1	1	1	1
Scissor Lift	1	1	1	1	1	1	1	1
Charge-up	2	2	2	2	2	2	1	1

16.18 Ventilation

Entech assessed the FS design regarding the mine ventilation strategy based on the mining schedule completed in February of 2021. The study was carried out through both desktop analysis and confirmation of the ventilation design using Ventsim Design© software. Entech's report provides details for the ventilation analysis along with subsequent recommendations.

16.18.1 Primary Airflow Requirements

Entech used parameters prescribed in the Western Australian Mine Safety and Inspection Regulations 1995 and industry best practice. Entech previously compared design parameters from other jurisdictions in the 'Entech_13072018_OGC_Waihi_Ventilation_NFR_ISSUED' letter report. Entech concluded that OGC adheres to the minimum requirements in Western Australia as well as New Zealand. Table 16-30 outlines the parameters used in this assessment. Mine design options, regarding a combination of diesel operated and battery electrified vehicles (BEV's), were also assessed as part of the airflow requirement analysis associated with this FS.

Table 16-30: Ventilation Design Parameters

Parameter Description	Design Parameter	Comments
Maximum recommended air speed in declines.	<6.0 m/s	Best practice recommendation to reduce dust generation during vehicle movements in the decline.
Minimum air velocity workplaces with flow-through ventilation.	0.5 m/s	Prescribed by Western Australian Mine Safety and Inspection Regulations 1995 for underground mines where the wet bulb temperature exceeds 25°C. A minimum of 0.3 m/s for temperatures less than 25°C.
Intake shafts.	<22 m/s	A guide to optimise power input by primary fans.
Exhaust shafts.	0-7;12-20 m/s	Industry best practice to avoid water blankets inside the shaft. 7-12 m/s could be used in short-length shafts.

Escapeway ladders.	<10 m/s	A guide for preventing eye injuries from dust generated by others using ladders.
Stop-work wet bulb temperature.	32 °C	Industry best practice.
Reject wet bulb temperature for design purposes.	30 °C	A limit for the main return airway of a production district. A 2 °C increase is typically expected between the reject temperature and the working face.
Diesel dilution factor	0.05 m³/s/kW	Prescribed by Western Australian Mine Safety and Inspection Regulations 1995.

Source Entech, 2021, Martha UG Ventilation Design, Report Number: ENT_702_OGC

Table 16-31: Ventilation Design Assumptions

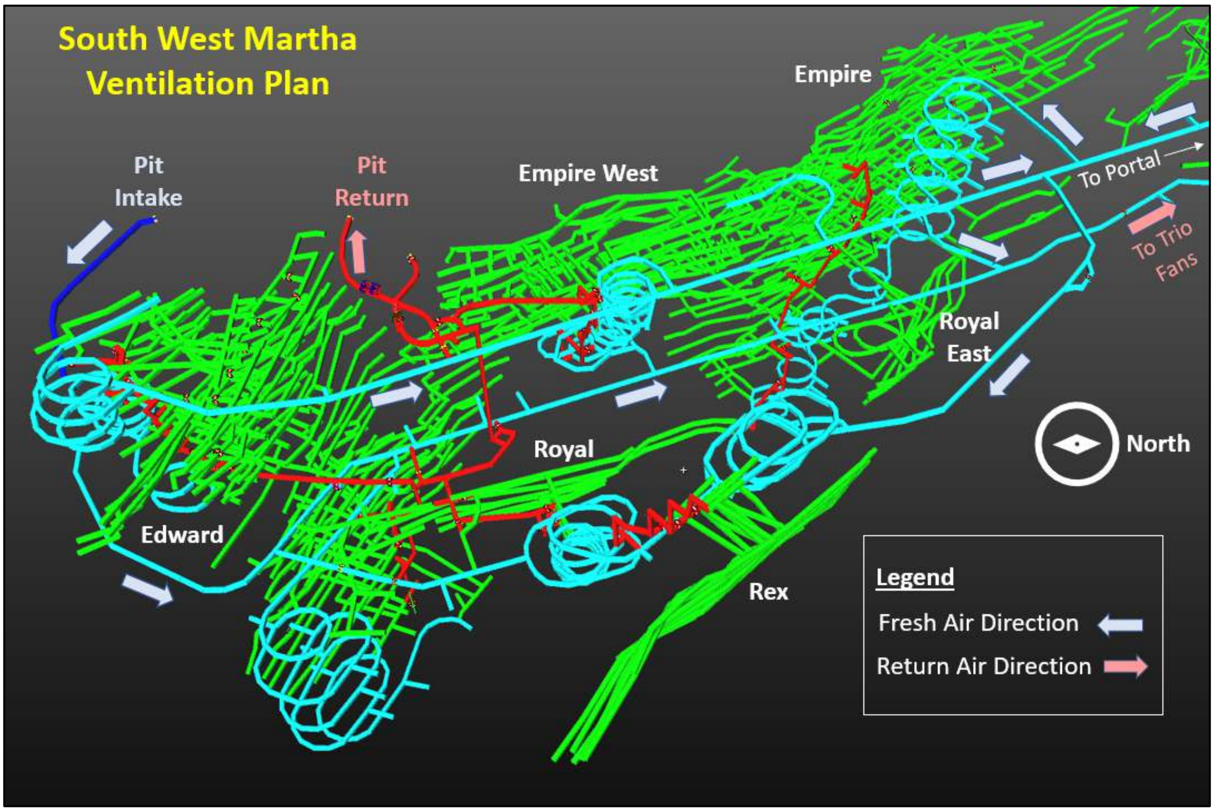
Assumption	Comments
Heavy diesel equipment utilisation factor: • efficiency – 35 %. • availability – 85 %. • utilisation – 80 %. • loading – 50 %.	Typical allowance for production equipment in calculating heat loads.
Ancillary diesel equipment utilisation factor: • efficiency – 35 %. • availability – 85 %. • utilisation – 50 %. • loading – 50 %.	Typical allowance for ancillary equipment in calculating heat loads. The assumption is that ancillary equipment operates a third of the time due to intermittent usage.
Power cost of NZD 0.11 per kWh.	Used as a guide for grid power.
Secondary fan power utilisation factor: • utilisation - 80%. • load - 90%.	Utilisation and load factors are applied under the assumption that fans will not operate 100% of the time.
Series ventilation allowed.	Series ventilation causing the reuse of air is appropriate when additional airflow is not available to prevent fans from recirculating downstream. However, for diesel equipment it is advisable that contaminated air be exhausted at the nearest return airway.

16.18.2 Primary Airway plan

The Martha underground primary ventilation plan is shown in Figure 16-51. Fresh air enters MUG's south western district via the northern haulage portal and the pit intake above the Edward lode area and also from the Correnso mine via the 920 m RL and 800 m RL drives. Return air exits the mine via the pit exhaust drive and the existing 4.5 m diameter 920 to 800 m RL raisebored shaft. The existing Trio primary fans only need run when accessing remnant areas in the north otherwise they can be switched off. The secondary means of egress is via the pit intake.

Having a complex system of connecting airways between ventilation districts, with multiple mining fronts running concurrently, will require a good understanding of the ventilation circuit. Opening and closing louvred regulators as well as ventilation gates in strategic places may help to manage airflow distribution but can also have unexpected consequences. Reversal of primary airflow has been seen in the Ventsim© simulation. This may result in the secondary fans recirculating and may cause fogging in the decline. Vent on demand (VOD) systems, along with variable speed drives on fans (VSD) will simplify the process of airflow balancing. Intuitive programmable systems now exist and will be mentioned in this report.

Figure 16-51: Martha Primary Ventilation Layout



Source Entech, 2021, Martha UG Ventilation Design, Report Number: ENT_702_OGC

16.18.3 Airflow Quantity Definition by Diesel Exhaust Dilution

Table 16-32 lists the mobile equipment as for MUG in March of 2021. It shows the number of vehicles anticipated in the peak production year of 2024, along with their subsequent airflow requirements for dilution of diesel exhaust. Air requirement for heavy battery electric vehicles was also compared.

Table 16-32: Mobile Diesel Fleet

Full Diesel Fleet	Assumed Model	Engine power rating (kW)	Airflow requirement*	Qty	Total Airflow (m^3/s)
Truck	MT 5020	485	24	5	121
Loader	R1700G	260	13	3	39
loader	R2900G	305	15	2	31
Grader	Cat 12H	83	4	1	4
Charge-up	Charmec 1610B	110	6	1	6
Agitator	LF700	170	9	1	9
Development Drill	Axera 7-240	110	6	5	28
Production Drill	Solo DD330	55	3	3	8
IT	L120G	200	10	1	10
LV	Toyota Landcruiser	151	8	3	23
Total (m^3/s)					**277**

Source Entech, 2021, Martha UG Ventilation Design, Report Number: ENT_702_OGC

*Airflow calculated for diesel equipment at 0.05 m^3/s per kW engine power.

Table 3-2 offers a comparison, with heavy vehicles replaced by fully battery-operated electric vehicles for loading and haulage. In this example six 40 tonne BEV's are required to match the five 50 tonne diesel trucks. These Epiroc vehicles are solely operated by battery power. Hybridised vehicles, which utilise a combination of electric power and diesel, should still use the dilution factor for fully diesel-powered vehicles.

Table 16-33: Mobile Diesel & BEV Fleet

Mixed Fleet, Diesel & BEV's Fleet	Assumed Model	Engine power rating (kW)	Airflow requirement*	Qty	Total Airflow (m^3/s)
Truck	MT42 Battery**	N/A	0	6	0
Loader	ST14 Battery**	N/A	0	3	0
Grader	Cat 12H	83	4	1	4
Charge-up	Charmec 1610B	110	6	1	6
Agitator	LF700	170	9	1	9
Development Drill	Axera 7-240	110	6	5	28
Production Drill	Solo DD330	55	3	3	8
IT	L120G	200	10	1	10
LV	Toyota Landcruiser	151	8	3	23
Total (m^3/s)					**87**

Source Entech, 2021, Martha UG Ventilation Design, Report Number: ENT_702_OGC

*Airflow calculated for diesel equipment at 0.05 m³/s per kW engine power.

**Epiroc BEV's used as an example of a fully battery powered vehicle.

As a direct comparison, 160 m³/s can be saved across the ventilation circuit during peak production by replacing the heavy vehicles with fully electric ones. Regardless of this saving design parameters must still be maintained in flow-through areas according to Table 16-30, even though a dilution factor is not applicable for these heavy vehicles. For example, production levels require a minimum of ~10 m³/s to satisfy the 0.5 m/s requirement for managing heat and dust. A high-level heat analysis determined that heat in the mine will not be an issue for MUG. The hottest area of the mine had a return wet bulb temperature of 26°C.Airflow capacity must also be evaluated to determine required flow rates.

16.18.4 Airway Capacity

Airway capacity is compared with airflow requirements in Table 16-34 below.

Table 16-34: Mine Airway Capacity

Airway	Cross-sectional Area of Airway (m²)	Ideal Air Speed for Airway (m/s)*	Maximum Capacity of Airway According to Ideal Air Speed (m³/s)	Air Speed According to Airflow Requirements (m/s)**	Air Quantity According to Airflow Requirements (m³/s)	Spare Capacity (m³/s)
Intake						
Portal Decline	27.2	6	163	3.5	95	68
Pit Fresh Air Intake	25.8	6	155	9.5	245	-90
					Total	-22
Exhaust						
Pit Return Drive	23.4	22	515	16	374	140

Source Entech, 2021, Martha UG Ventilation Design, Report Number: ENT_702_OGC
*ideal air speeds according to design parameters in section 2.1.
**air speed according to Ventsim© simulation.

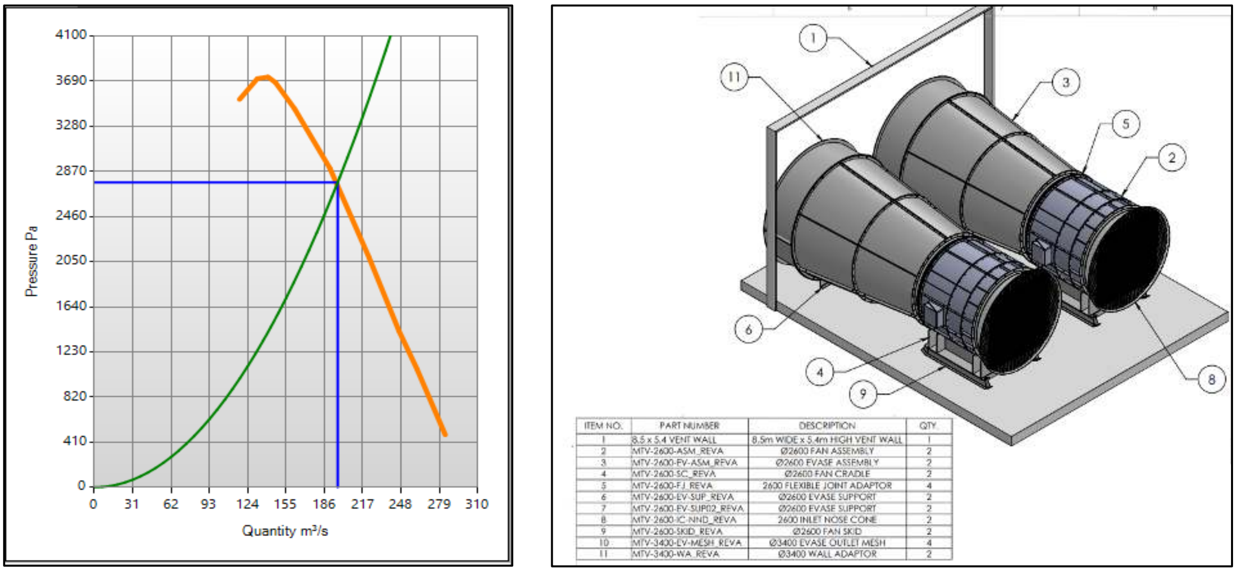
The above summary indicates that the mine is intake constrained due to air velocities in drives, connecting pit and portal, need to be managed for dust generation.

16.18.5 Primary Fans

The current primary fan arrangement for south-west MUG is comprised of four 110 kW Clemcorp CC1400 axial flow fans. These fans will be replaced by two 700 kW fans from MTV later this year. The new fans consist of two axial flow fans in parallel located underground in the pit exhaust airway. At 100 % of their speed these fans will draw up to ~390 m³/s for full depth of mine. Changing fan speed to meet airflow requirements in ventilation districts will be achieved using variable speed drives (VSD’s). At full speed and full mine resistance these fans will perform well according to Ventsim© modelling, refer Figure 16-52.

The fan curve suggests that there is enough flexibility for unexpected changes in mine resistance, making these fans suitable for further mine expansions. Below is the general layout for two MTV axial flow fans in parallel.

Figure 16-52: MTV Primary Fan Performance Curves & Assembly



16.18.6 Secondary Fans

The declines will be fitted out with 1400 mm diameter ducting lines for maximising airflow in the work areas. Fan selection is based on the distance to be developed before flow-through ventilation can be established, and the subsequent truck loading location if conducted under secondary air.

Consider two fans in parallel for areas with multiple mining activities. One fan will typically support truck loading only, causing other activities on the level, e.g. production drilling, to stop. Primary flow adjustments to the circuit must consider parallel fans in declines to prevent recirculation. Two 90 kW fans required a minimum of 80 m³/s, for instance.

If truck loading is to be carried out in flow-through air, using the level's ventilation regulator, then allow enough air to exhaust without contaminating work areas downstream. If decline flows are reduced significantly then fans downstream will recirculate.

Single-staged 90 kW (1 x 90 kW) or twin-staged 90 kW (2 x 90 kW) fans are already in use at MUG and these fans will allow truck loading at ~150 m. The single 55 kW fans in use throughout the mine will not provide the dilution required for truck loading on levels unless two fans are used in parallel.

16.18.7 Power

Table 16-35 provides a summary of the estimated power supply for primary and secondary fans using the equipment fleet fuel burn estimates.

Table 16-35: Power Supply Summary Based on Full Mine Resistance

	Current Mine to	2022	2023	2024	2025	2026

	Fan Upgrade					
Duration (Months)	7	12	12	12	12	12
Primary Fans (kW)	870	1,050	1,137	1,147	1,147	1,153
Secondary Fans (kW)	567	648	648	648	648	486
Total Estimated Electrical Energy Consumption (kW)	1,437	1,698	1,785	1,795	1,795	1,639
Total for duration (kWh)	7,242,480	14,670,720	15,422,400	15,508,800	15,508,800	14,160,960
Total Power Cost @ $0.11 per kWh	**$796,673**	**$1,613,779**	**$1,696,464**	**$1,705,968**	**$1,705,968**	**$1,557,706**

Primary fan input power was calculated using the input power to achieve the full 390 m^3/s for the maximum mine depth, resulting in a factor of 3.326 $kW/m^3/s$.

16.18.8 Ventilation on Demand

OGC has proposed to automate the functioning of its primary and secondary fans through VOD systems and the benefits have been highlighted in the Entech, Ventilation on Demand Scoping Study 2020. There are different levels of sophistication with VOD, which affords the client a means to integrate complexity over time to avoid upfront systems issues.

VOD systems range from a simple remote on-off functionality to an ecosystem of automated ventilation controls that talk to environmental monitoring devices and adapt to changes in the mine plan using algorithms. Controlling changes to the ventilation circuit can be done in real time and some products allow the user to control the underground circuit using Ventsim Control© without having to purchase a Howden system. Those familiar with Ventsim© will find this useful without much extra training.

The OGC proposal to turn secondary fans on and off via a SCADA system has been around for some time and has proven to work well in achieving short re-entry times amongst other benefits.

The best systems in operation overseas, use an electronic fob device that sends a signal to the auxiliary fan’s control switch to change the ventilation flow in an area according to the dilution requirements of the vehicle.

Primary fans talking to automatic louvred regulators, and decline ventilation gates, can adjust ventilation flows in districts according to the needs of the fleet, and help balance the airflow, thus avoiding unnecessary over ventilation and consequently saving on power costs.

Even though VOD systems have yet to be fully realized in Australia, any level of technology that optimises power consumption is recommended. For example, the Clemcorp dual speed fan mentioned in this study is a good option. Minetek fans that use a variable inlet vane to dampen airflow can be integrated into a VOD system without the need of a VSD.

With a two-speed fan control on auxiliary fans, running auxiliary fans on half speed for half the time will save NZD1.8M over six years.

16.18.9 Battery Electric Vehicles

Fully electric underground mining vehicles for loading and haulage are a new technology yet to make an impact on the mining industry.

For example, Epiroc's battery powered trucks and loaders have been operating in North America for about 12 months. It is estimated that it will be 12-18 months before the product is available on the Australian market.

The loader has about 4 hours battery life, enough to bog a decline heading before the battery will require switching-out.

The 40-tonne truck can travel about 6km up a decline fully loaded before requiring a battery switch-out.

Battery life is key to the success of the BEV and each piece of equipment will require multiple batteries ready to go at a charging station. Things to note with batteries:

- Charging stations run off a jumbo box and can be easily moved about to be closer to activity.
- Batteries can be speed charged over 15 minutes, however this generates heat, and the cabling requires artificial cooling or alternatively, the battery can be trickle charged over an hour to avoid the heat issue.

16.18.10 Ventilation Control Devices

Ventilation airlock doors large enough for mobile equipment such as trucks will be required in various locations in the mine to help control the airflow balance. For each airlock, the doors are to be spaced at least 30 m apart to ensure that a large truck can fit between the doors without affecting the door operation. Each airlock must have provision for personnel access doors. The airlock ventilation doors and the required operating system must be designed to withstand the static pressure in their locations.

An airlock will be required in the mid access to Empire to ensure primary air reaches the bottom of the decline. Another airlock will be required in the top access to Rex to ensure primary air makes it to the top of the incline. Other doors may be required in the remnant areas of the mine to ensure that primary flow is directed to the new areas, see the Ventsim model that accompanies this report for details.

For active ventilation management of the primary airflow on the levels for flow and power optimisation, Entech recommends that the louvre regulators on each level are designed to be adjusted both manually and through automation. These devices are highlighted in the VOD study by OGC.

16.18.11 Recommendations

Entech have made the following recommendations with respect to the ventilation plan for the mine.

- As the mine is intake constrained decline air velocities will be above design conditions at various stages of mine life according to the airflow distribution in the mine. Careful airflow balancing will be required to overcome this, and it is recommended that an integrated VOD system is incorporated into the management of this balance. The VOD will allow for the remote operation of ventilation controls, such as automatic louvres and ventilation doors in travel ways as well as managing the primary fan speeds.
- Careful consideration of how ventilation balancing will affect the direction of airflow in travel ways so that secondary fans do not recirculate.

- There is a need for thorough risk management around emergencies particularly when VOD systems are constantly changing the direction and force of ventilation circuits.

17 RECOVERY METHODS

The metallurgical process at Waihi is well-tested and proven technology, having been in operation for 30 continuous years.

17.1 Actual Plant Performance

Mill production tonnes processed and plant utilisation for the 2018 -2019 years are shown in Figure 17.1.

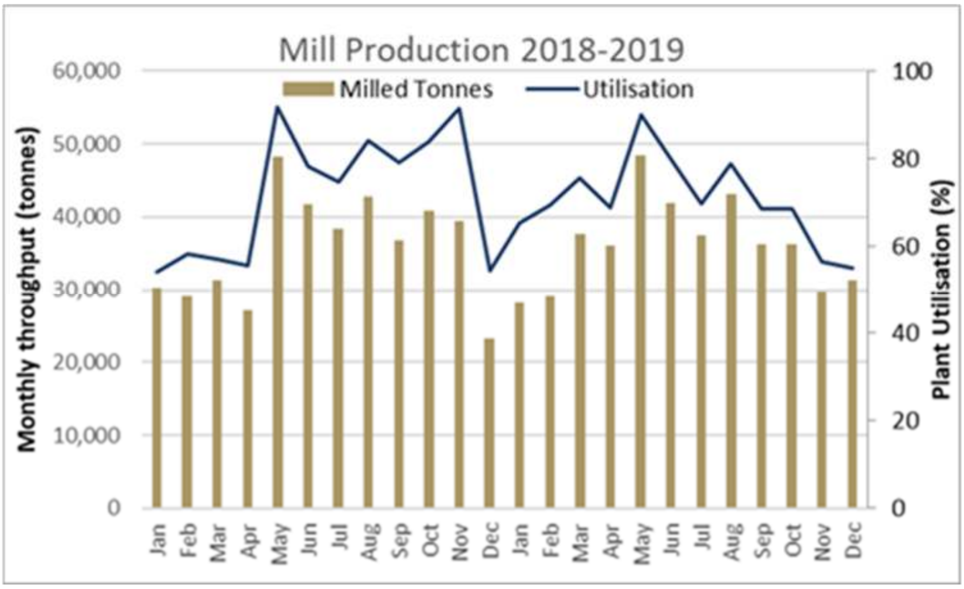


Figure 17.1: Underground Mill Feed Tonnes and Utilisation 2018-2019

Mill throughput has been limited with a single ore source from the Correnso mine at reduced milling rates and plant utilisation.

Process unit costs are dependent to a large degree on ore availability with initiatives implemented since 2015 to reduce costs in the mill given the limited supply of ore. Processing costs have ranged from NZD 30-40/tonne milled when more than 40,000 tonnes per month of ore was available and trends down at higher throughput rates as the fixed cost component is allocated over a larger tonnage. Unit cost history for the Waihi mill is shown in Figure 17-2 below.

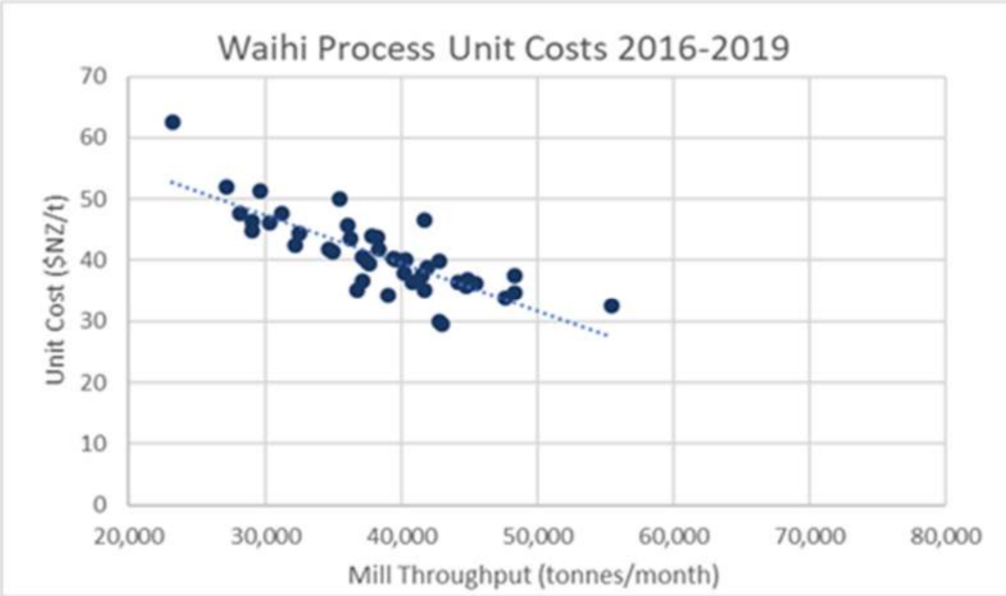


Figure 17.2: Actual Process Unit Costs 2016-2019

17.2 Metallurgical Accounting

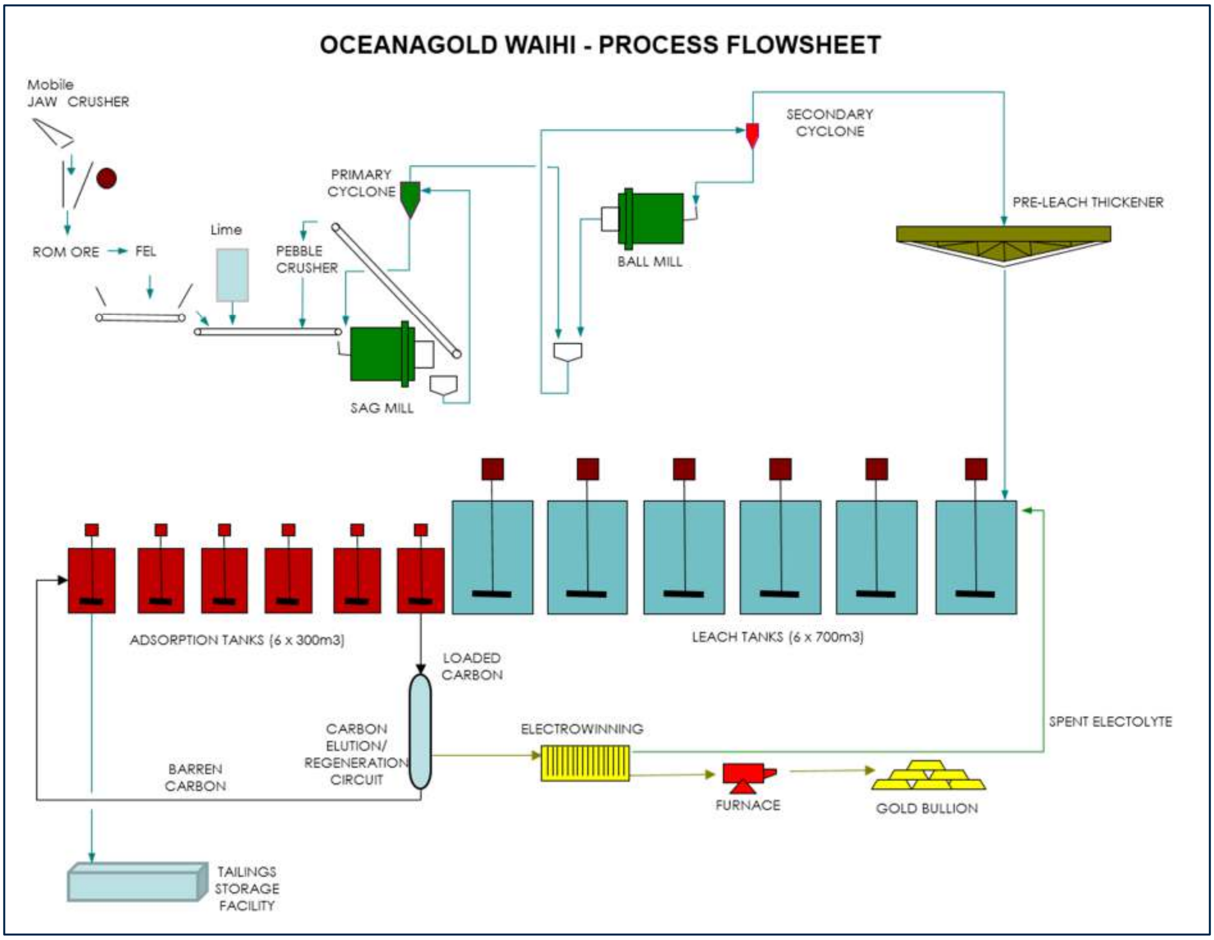
Metallurgical accounting at Waihi is primarily based on the tonnage of wet ore processed through the comminution circuit, as totalised on a conveyor weightometer and gold receipts from the Mint. Wet tonnes are converted to dry tonnes by using a moisture factor, the moisture factor is derived from samples taken from the conveyor. Gold production is based on gold receipts from the Mint and the changes to the gold stocks in circuit. Gold stock takes are taken monthly.

Samples are taken at strategic points in the processing stream to measure gold concentrations in those streams to determine plant efficiencies on a day to day basis. All information is entered into a data base which then performs the metallurgical accounting.

17.3 Ore Processing

Ore processing consists of five stages: comminution, leaching/adsorption, elution, electro-winning and smelting as shown in Figure 17.3.

Figure 17.3: Process Flow Sheet



17.4 Comminution

Underground stockpile ore is reclaimed at 80 tonnes per hour by front end loader and fed onto a static grizzly with an aperture of 150 mm. The final conveyor from the ore handling circuit transports the ore into the grinding circuit.

Prior to entry into the feed chute of the semi autogenous (SAG) mill, the ore is further reduced in feed size via a jaw crusher to a P_{80} of 110 – 130 mm. The SAG mill-ball size is 125 mm and the mill will operate typically with a 10% ball load. The SAG mill draws between 2.1 and 2.5 MW of power.

The SAG mill discharge is sized using a trommel attached to the SAG. The +12 mm oversize material is conveyed to a 30kW cone crusher and is recycled back to the SAG mill. The undersize slurry from the SAG trommel is pumped to two 0.5 m diameter inclined Weir Warman Cavex cyclones. The cyclone underflow reports to the SAG mill feed chute. The cyclone overflow gravitates to the ball mill discharge hopper, whereby the slurry is pumped to a cyclone distributor, which consists of fourteen 250 mm diameter Weir Warman Cavex cyclones. The cyclone underflow reports back to the ball mill for further grinding and the cyclone overflow reports to the pre-leach thickener.

Normally, the comminution process is set-up as a closed-circuit, but the plant has an ability to operate to an open circuit system if required.

17.5 Leaching and Adsorption

The pre-leach thickener increases slurry density from approximately 15% solids to approximately 37 to 40% solids prior to the leach/adsorption circuit, which comprises six leach and six carbon in pulp (CIP) adsorption tanks. The leaching tanks capacity are 700 m^3 and the adsorption tanks have 300 m^3, providing a total residence leach/adsorption time of 24 hours for Martha ore and 48 hours for Correnso ore. Wedge wire cylindrical inter-stage screens with mechanical wipers are installed in each adsorption tank. The inter-stage screens retain carbon in the tank but let the slurry pass through to the next stage. A bleed stream is pumped from an adsorption tank to the previous tank in the circuit, the carbon contained in the bleed stream is retained in the previous adsorption tank in the circuit, this provides counter current flow whereby the slurry flows from adsorption tank 1 to 6 while the carbon flows from adsorption tank 6 to 1. This allows for maximum carbon loading in adsorption tank 1 and maximum scavenging of gold solution in adsorption tank 6. From adsorption tank 6 the slurry passes over a carbon safety screen to collect any carbon that may have leaked from the adsorption circuit, the barren tailings slurry is then pumped to the tailings storage facility.

Cyanide is delivered and mixed on-site, via a sparging system to a concentration of 21 % wt./vol. The cyanide is dosed into the first leach tank and the concentration is maintained at 280 ppm for Martha and 240 ppm for Correnso. Oxygen is added to the first leach tank by a shear reactor to enhance the leach kinetics and reduce cyanide consumption.

17.6 Elution, Electrowinning and Smelting

Loaded carbon from the adsorption circuit is fed into an elution column where the carbon is washed at high temperature and pressure to remove the gold and silver from the carbon and into a pregnant eluate. The pregnant eluant is then passed through electrowinning cells where gold and silver are electroplated onto stainless steel cathodes. Following elution, the barren carbon is reactivated and recycled to the adsorption tanks.

The cathodes are periodically harvested and rinsed to yield a gold and silver bearing sludge which is dried, mixed with fluxes and put into a furnace at 1200^{O}C. Once the sludge is molten it is poured as bars of doré (alloy of gold and silver) ready for shipment to the Mint.

17.7 Other

The Waihi processing plant has a SCADA control system. Equipment protection and P&ID control loops to optimise the control of the major streams/processing parameters within each process circuit are actively in use within the process plant.

The Processing Plant has the capacity to treat either 1.25Mt of Martha open pit Resource or 0.9Mt of Martha underground ore per annum.

17.8 Processing Plan

The processing plan for the Mineral Reserve is shown in Table 17-1**Error! Reference source not found.**. An average processing recovery of 94.9% is estimated based on the proportions of the various veins being mined.

Table 17-1: Mineral Reserve Processing Plan

	Units	Total	2021	2022	2023	2024	2025	2026	2027	2028
Processed ore	kt	4,458	221	492	464	689	756	755	760	321
Processed grade Au	g/t	4.33	4.68	5.79	3.90	4.38	3.93	4.11	4.09	4.33
Processed grade Ag	g/t	13.5	15.9	16.0	9.9	12.3	11.7	15.0	14.9	13.3
Processed grade As	g/t	49	43	39	13	26	25	65	98	77
Recovery	%	94.9%	94.8%	97.1%	97.4%	96.8%	96.5%	93.7%	91.2%	91.7%
Ounces recovered Au	oz.	589	31	89	57	94	92	94	91	41
Ounces recovered Ag	oz.	1,223	71	160	94	172	179	230	231	87

18 PROJECT INFRASTRUCTURE

18.1 Site Access Infrastructure and Logistics

The project is an active mining project with the majority of the infrastructure required for its ongoing operation already in place. Site access from major ports, international and domestic airports and roads are well established at the Waihi site. Supplies, equipment, and materials are trucked to the sites via the paved roads. As this is a gold project there are no concentrate shipping constraints. There are no material logistic limitations impact the project.

18.2 Existing Mine Site Surface Infrastructure

MUG will use the facilities in place established at Waihi operations in 1988 and upgraded in the late 1990's and from 2004 for the Favona mine. The location of the existing infrastructure is shown in Figure 18-1.

Figure 18-1: Waihi Existing Infrastructure

Existing facilities comprise:

- two separate tailings storage facilities,
- numerous silt and collection ponds,
- stockpile facilities,
- mine access roads,
- water treatment facilities,
- Favona underground administration and change house,
- Martha pit surface conveying and loadout facilities,
- Favona surface and underground workshops,
- Trio underground cribroom,
- surface explosives magazines, and,

- existing process plant.
- Access via the Favona, Trio and Correnso mines and existing ventilation shafts.

A cement batch plant is proposed to be constructed close to the existing polishing pond stockpile entry.

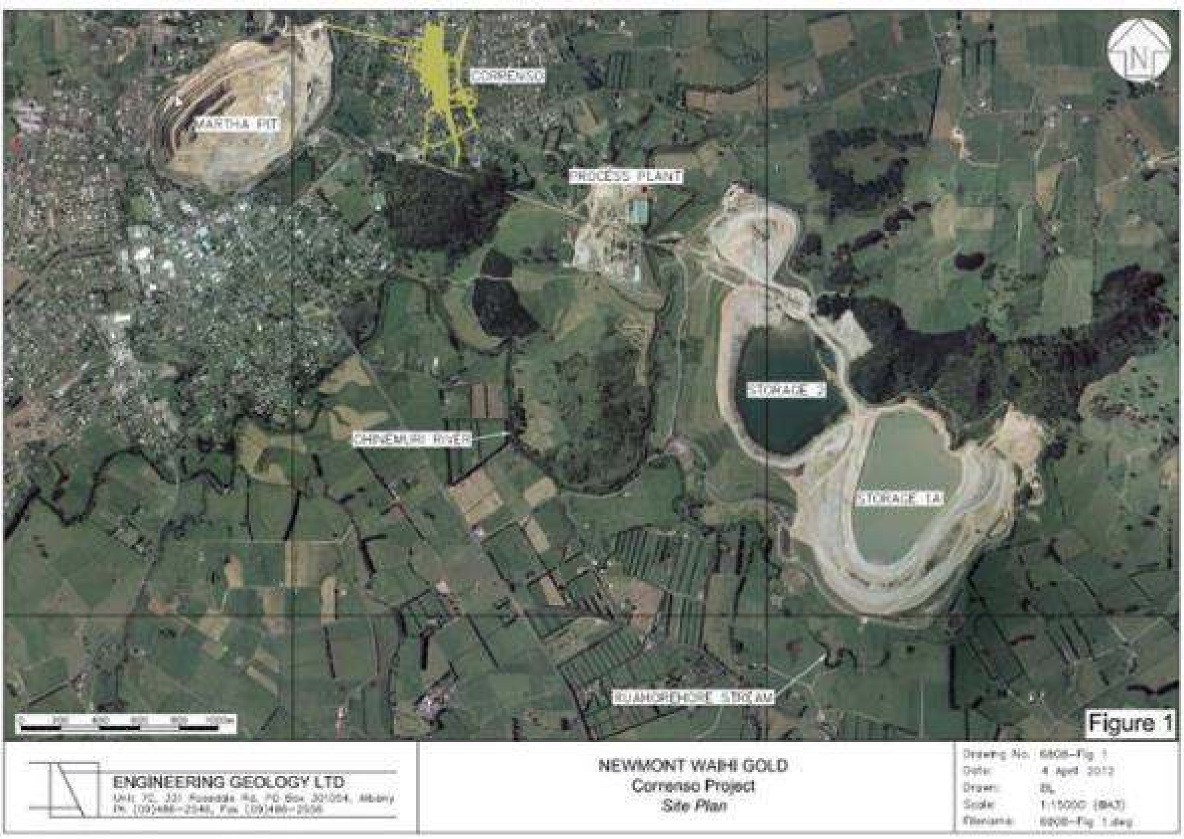
18.3 Open Pit Mine

The Martha pit has not operated since April 2015 when a wall failure on the north wall closed the main ramp. In order to provide backfill, the open pit must recommence operations which will require refurbishment of the existing crusher and conveyor system and pit rim works.

18.4 Tailings Storage Facility

Waihi has two Tailings Storage facilities (TSF's) known as TSF2 and TSF1A. Both are located SE of the WPP and Martha Pit as shown in Figure 18-2. The TSF's are formed by downstream constructed embankments that abut elevated ground to the east of TSF2 and north of TSF1A.

Figure 18-2: Location of Tailings Storage Facilities



TSF2 has a planned finished crest elevation of 159.5 m RL and the planned crest of TSF1A is 182 m RL. The embankments have both been constructed from overburden material obtained from mining Martha pit. TSF2 was constructed first and provided tailings storage from 1989 to 2000. TSF1A has since provided tailings storage. TSF1A and TSF2 are permitted by the Mining Licence, TSF1A has a Building Consent allowing it to be constructed to 177.25 m RL. TSF2 has a Building Consent allowing it to be raised to 161 m RL. A further lift on TSF1A to 182 m RL is planned through a combination of downstream and centreline construction techniques.

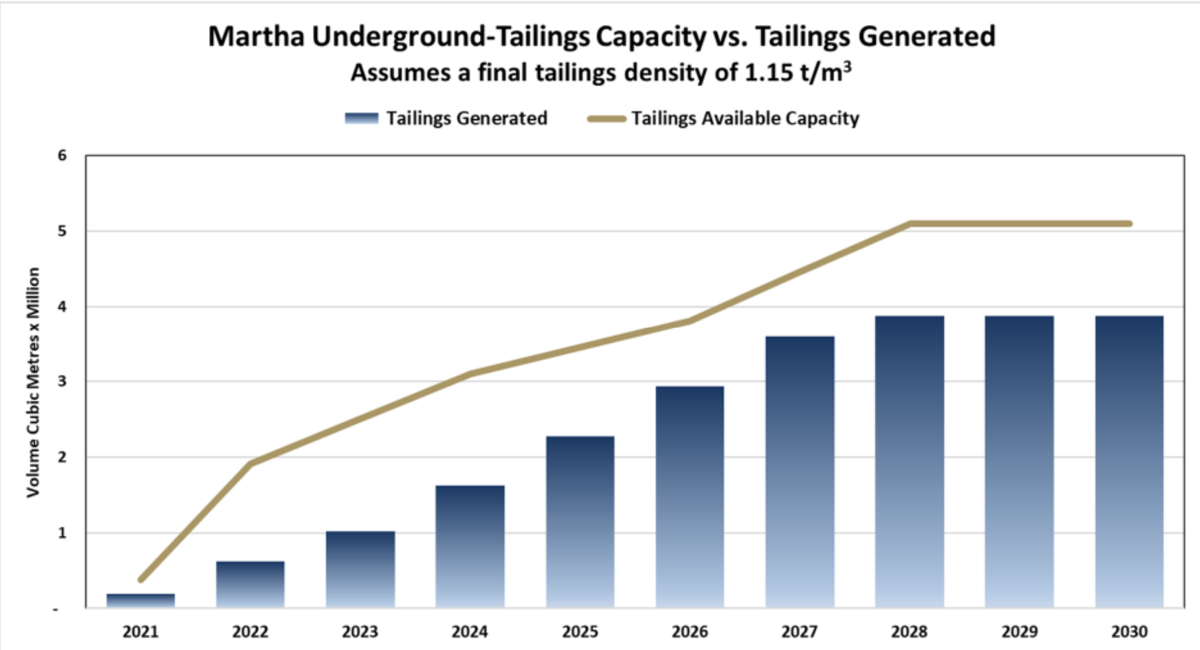
The project requires facilities for the disposal of up to 6.4 Mm^3 of tailings and will use the tailings disposal facilities shown in Table 18-1. For the FS an insitu tailings dry density of 1.15 t/m^3 is estimated based on the current measured density of 1.2t/m^3.

Table 18-1: Concept Tailings Storage Plan

Area	Tailings Storage			Fill required		
	Storage Mm^3	Cumulative storage Mm3	Cumulative storage Mt	Fill required Mm3	Cumulative fill Mm3	Cumulative fill Mt
TSF1A 173.3	0.38	0.38	0.43	0.05	0.05	0.09
TSF1A RL177.25	1.54	1.92	2.20	0.21	0.26	0.47
TSF1A RL180.25	1.19	3.10	3.57	0.27	0.53	0.96
TSF1A RL182	0.70	3.81	4.38	0.36	0.90	1.62
TSF2 RL159.5	1.30	5.10	5.87	0.16	1.06	1.91

Construction of the tailings facilities has been scheduled to ensure the TSF's meet the minimum freeboard conditions and provide adequate tailings capacity throughout the LOM Figure 18-3 below.

Figure 18-3: Tailings Storage and TSF Construction Chart



The deposition of tailings into the TSF will be via a high density polyethylene pipeline located around the perimeter of embankment crest. Deposition will occur from multiple spigots inserted along the tailing's distribution line. The deposition locations will be moved progressively along the distribution line, as required, to maintain slightly graded deposition of tailings towards the decant pond that is located in the southeast corner of the facility. Water from the decant pond will be recycled back to the mill for makeup water and will be reclaimed by utilising either vertical turbine pumps mounted on a floating barge above the decant pond or skid mounted pumps to be located on the ramp within the southern end of the decant pond.

The TSF will continue to be designed as a zero discharge facility. In addition to the anticipated tailing storage and operating pool requirements, the facility will be designed to contain the probable maximum precipitation storm event and an additional one metre of freeboard at all times.

18.5 Waste Rock Usage

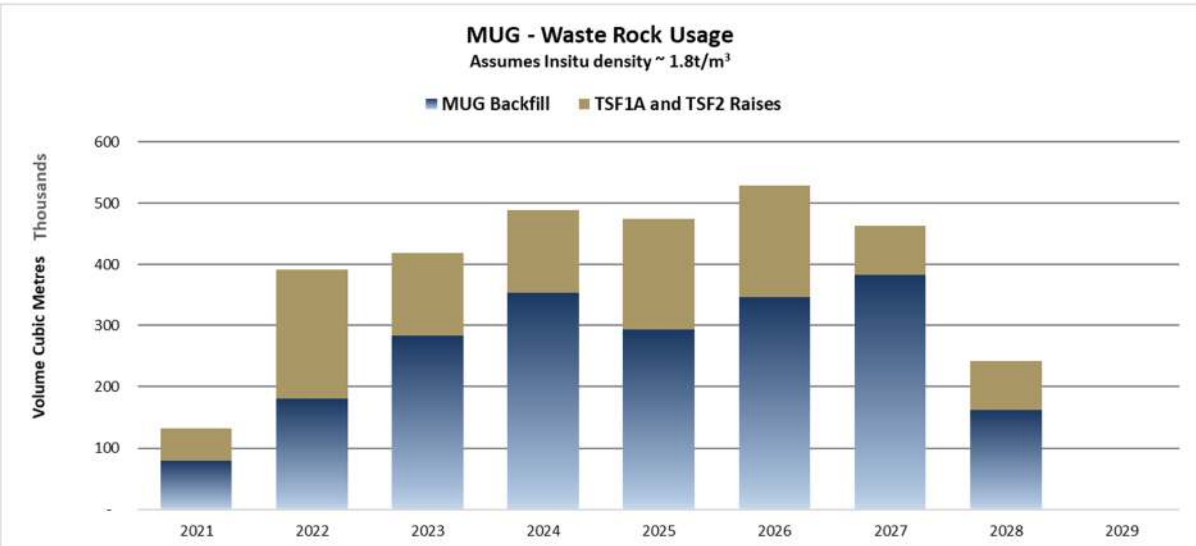
Waste rock is required to backfill the Martha underground and selected historical workings and to construct the crest raises for TSF1A and TSF2 as shown in Figure 18-4. Waste rock sufficient for the small raises on the TSF's is located close to the TSF's in the northern and eastern stockpile areas.

All waste rock produced from the underground mine is classified as potentially acid forming and is returned underground as stope backfill.

Some stockpiling of waste rock from the underground will be required to enable waste production to be scheduled in accordance with backfill requirements. The stockpile areas is already established near the Favona portal and will be used for the temporary storage of waste rock. A surge stockpile is available at the polishing pond stockpile up to 1Mt capacity.

Waste from the Martha pit will be conveyed 2.5 km to the waste development loadout site where it will be either directly loaded into 100 t trucks and transported a further 1 km to the tailings embankment or stockpiled for future use. At the waste development site, the waste will be selectively placed in accordance with a quality control program to form an engineered dam for the tailing impoundment.

Figure 18-4: Waste Rock Usage



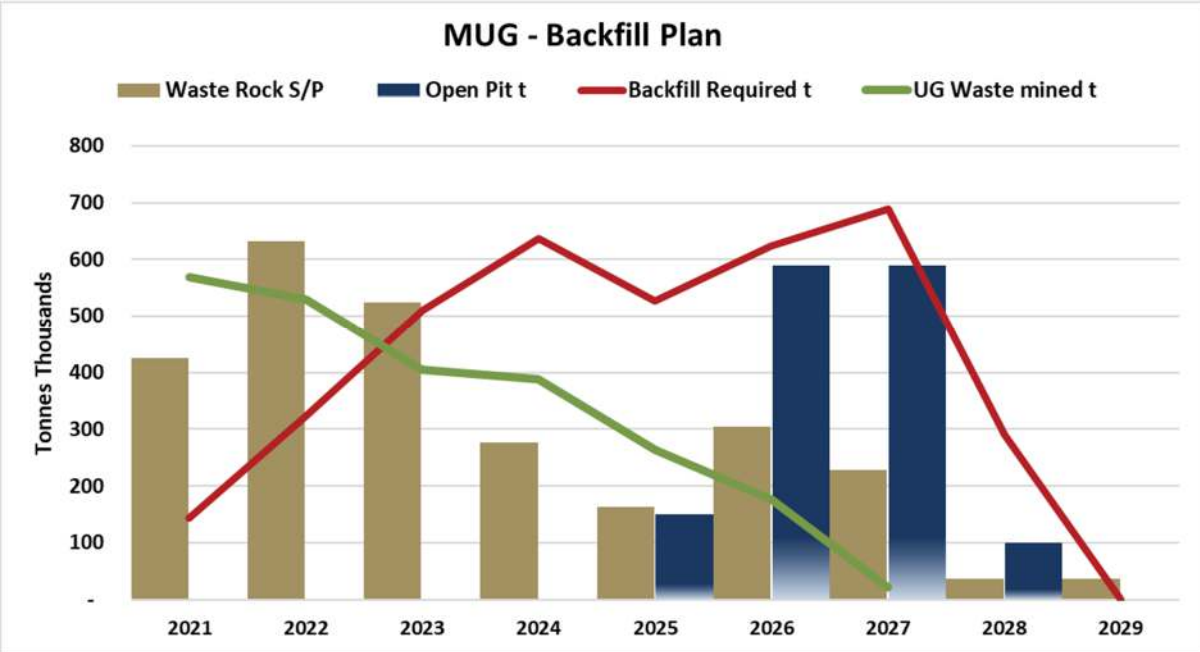
18.6 Backfill Supply

As noted in Section 16.10.4, there is a deficit of waste rock for backfilling the underground voids. Waste rock will be sourced from the Martha open pit (MOP) under the current consent conditions. Waste rock is also available from materials stockpiled at the western end of the Martha Pit and mined from the upper benches of the north wall under the consent.

C&R contracting, Red Bull explosives and DrillConnex were approached to provide a cost estimate and methodology to mine waste rock from the north wall of Martha pit and supply to the crusher and conveyor system. All Company's have had operating experience at the Martha pit and understand the operating environment. OGC estimated the operating costs for the crusher and conveyor system. C&R contracting are currently involved in constructing the downstream lift on TSF1A.

The MUG backfill plan is shown in Figure 18-5. This shows a stockpile being built up whilst initially development waste produced is higher than stope voids to be filled and then depleted as waste development rates decrease and then material mined form the open pit to supplement the stockpile and address the deficit.

Figure 18-5: MUG Backfill Plan



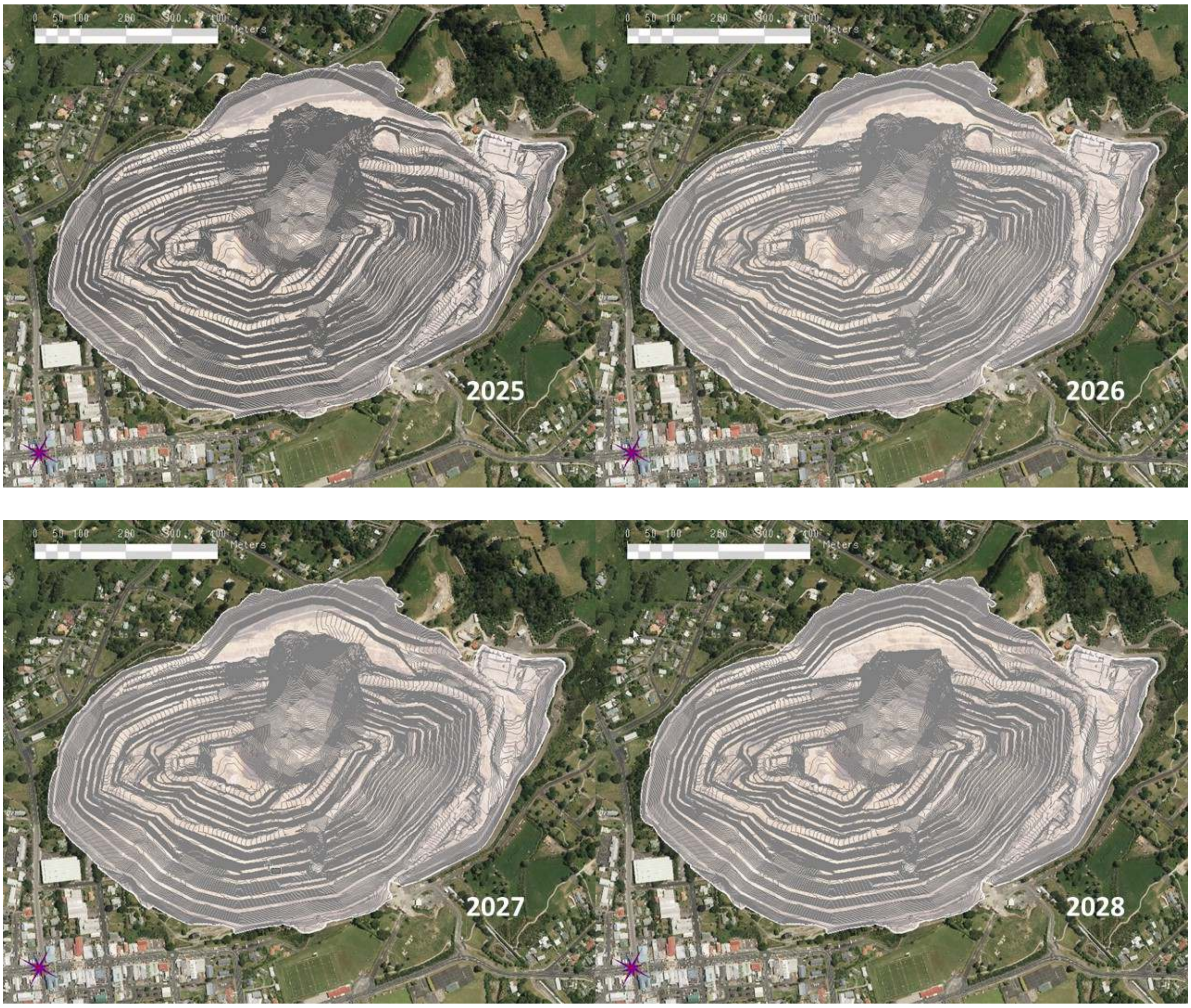
The mining schedule indicates the shortfall in waste rock for backfill as shown in Table 18-2 to be supplied from the Martha pit..

Table 18-2: Backfill Supply Schedule

BACKFILL SUPPLY SCHEDULE	2025	2026	2027	2028
Rock Fill tonnes (000's)	200	590	590	100
Target m RL	1157 RL	1137 RL	1122 RL	1110 RL
Drilling metres (000's)	42	122	122	21
Holes blasted No.	8,386	24,739	24,739	4,193
Kg Explosives	54,545	160,909	160,909	27,273

The pit development by year to provide the backfill is shown in Figure 18-6. Ramp access has been established on multiple levels for the cutback and the cutback progresses as a single cut on each bench to supply the backfill. All established ramps can access the crusher conveyor system and the workshops, diesel bowser, wheel wash and other facilities.

Figure 18-6: Martha Pit Development by Year

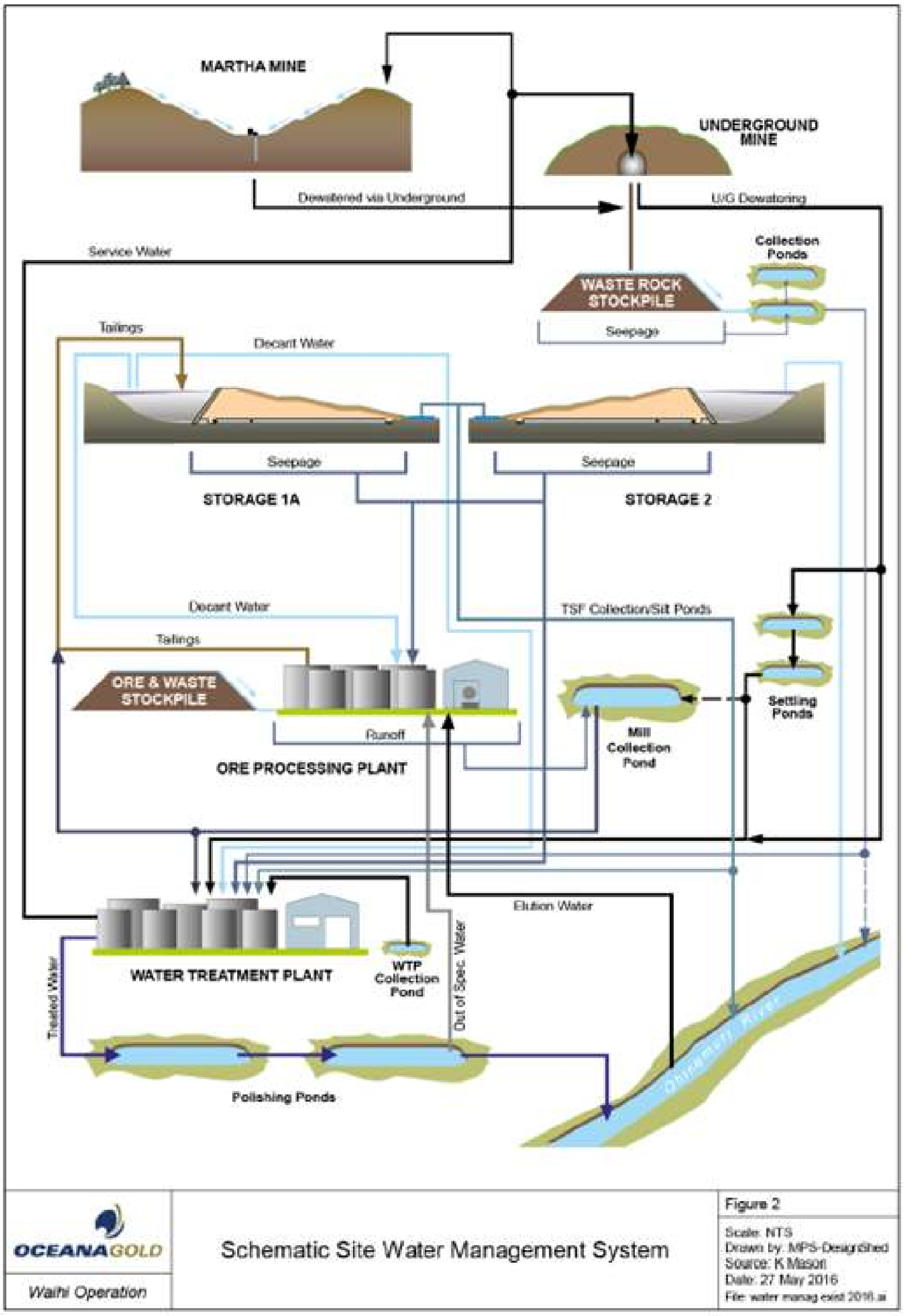


18.7 Site Wide Water Management

The site wide water management system is shown in Figure 18-7 and provides a schematic representation of the current water management system with TSF2 decant water overflowing to the Ohinemuri River.

Waihi requires approximately 1,750ML of water to process a nominal 1 MT of ore annually. Most of this water is sourced from the pumping from the underground mine and from water contained within the TSF. The high rainfall in the area means there is a net surplus of water and water is treated from the tailings impoundments, collection ponds and process plant and after treatment is either recycled or reused or discharged to the Ohinemuri River. The volume of water treated in 2019 was 7.8Mm3 and water discharged from site was 6.7 Mm3.

Figure 18-7: Site Wide Water Management



The current water management system is designed to capture and treat all water impacted by mining activity; and divert clean water where practicable. While some water is re-used as process water there is always a net gain of water on-site due to the high rainfall experienced in Waihi. The basic rules applied to site water management that have been effective in nearly 30 years of operation to date include:

- natural water is diverted away from areas disturbed by mining activities wherever practicable in order to reduce the volumes of water affected by the mining activities.
- all water from areas disturbed by mining activities is directed to appropriate collection and treatment facilities prior to discharge off-site.

- where practicable, OGNZL endeavours to reduce the volumes of water requiring treatment. An extensive programme of water quality monitoring is key to checking what water sources do require treatment.
- Disturbed areas are progressively rehabilitated at the earliest practicable time to minimise silt losses and improve runoff water quality.

The volume of water that can discharged on any given day is limited to an allowable discharge; which forms part of a suite of resource consents and is related to both the flow in the river and the treatment regime in operation. There are some sources of water requiring treatment that are relatively constant and need to be treated at all times. This applies in particular to water pumped from underground workings, seepage collected from the TSF’s underdrains, toe drains and other sources also requires ongoing treatment since it cannot be stored.

The site WTP operators manage the system such that sufficient freeboard is maintained in collection ponds and the active TSF to provide buffer storage over periods where the allowable discharge is less than the volume of water requiring treatment.

18.7.1 Water Treatment

As the Waihi processing plant operates with a positive water balance, a water treatment plant (WTP) and a reverse osmosis (RO) plant treat various water sources before discharging into the local Ohinemuri River.

The WTP has been in operation since 1988 and has been subject to upgrades in 1999 and 2011. A reverse osmosis plant was built and commissioned in 2008 to provide an additional treatment option for metals removal. The WTP has performed consistently well with no recorded non compliances with consent conditions.

The WTP incorporates four parallel streams with three of these dedicated to soluble metals removal only. The fourth stream has two phases of treatment; oxidation of cyanide to destroy the cyanide complexes followed by metals precipitation and removal.

- Cyanide oxidation is achieved using a combination of hydrogen peroxide, copper sulphate and lime. A series of tanks are used for reagent mixing followed by retention to provide time for chemical reaction. Hydrogen peroxide in the presence of copper destroys all free cyanide through chemical oxidation. Weak acid dissociable (WAD) cyanide is also oxidised during the process. On oxidation, cyanide yields simple carbon and nitrogen compounds.
- Lime and ferric chloride are added to all four water streams to facilitate metals precipitation and removal. Metals tend to occur in a soluble form when the pH of water is low and raising the pH with lime in the presence of ferric chloride creates insoluble hydroxides and carbonates to form. Following mixing and retention a polyelectrolyte (flocculant) is added along with more lime to form flocs that can be settled out.

Clarifiers at the end of the treatment process allow the suspended solids and metals to be removed from the water. The suspended solids and metals fall to the bottom of the clarifiers forming a slurry. The slurry is pumped to the tailings pond via a thickener. Carbon dioxide is added to the clean water overflow from the clarifier to reduce the pH of the water to meet the compliance limits.

There are two polishing ponds that hold the treated water for approximately 16 hours prior to discharge to the river. This provides time for the treated water to be tested, and the results to be received and interpreted prior to the water discharging to the Ohinemuri River.

Water that meets the discharge criteria is discharged to the Ohinemuri River. If the water does not meet the discharge criteria, it is recycled back through the plant, used in processing, or piped to the tailings storage facility.

There are four operating regimes and each provides for a different combination of water requiring cyanide destruction versus metals removal only. These operating regimes recognise that the proportion of water being treated for cyanide destruction impacts the treated water quality.

18.7.2 Forecast Water Treatment

GHD were commissioned to model the site hydrology for the Martha underground mine including operating TSF1A and TSF2. GHD used a water balance model (WBM) to assess how water gains change over the life of the mine and to check that proposed infrastructure for conveyance, storage and treatment will be adequate. This was carried out using the Goldsim (refer www.goldsim.com) software package which is designed to run Monte Carlo simulations for probabilistic analysis of dynamic systems. The objective in building the model was to have a tool to forecast storage requirements in the TSF's and as an ongoing check that the site water management infrastructure as a whole had capacity for ongoing mine development. The model was also used to predict water treatment requirements post closure as a component of annual bond calculations.

The WBM aims to capture all significant water movements across the site affected by mine operations. The model is run as a probabilistic analysis based on 100 years of measured rainfall data, corresponding Ohinemuri River flow rates and the Martha underground mine plan. GHD consider the model to provide a good representation of site conditions and based on the calibration is conservative. Based on the water balance analysis the existing water treatment plant is sufficient for the Martha underground dewatering, as predictions for the life of mine indicate that suitable capacity is available and that water can be treated to meet existing discharge consent conditions.

18.8 Site Services Infrastructure

18.8.1 Fuel Supply and Storage

Fuel is supplied directly to the mine, by local venders who contract supply from Tauranga, 60 km south of Waihi.

18.8.2 Water

Process water is sourced from the TSF, via a pump on a pontoon collecting clear water, which is conveyed to the mill via a pipeline. Water to supply the buildings, as well as for the fire suppression distribution system will be provided by the town water supply.

Mine water is supplied from the WTP. Used water and sewage are handled by a septic tank system.

18.8.3 Security

The current security office and guardhouse are provided at the public entrance to the Martha site and Process / TSF site will be staffed continuously.

18.8.4 Communications

The Waihi site has an established telecommunications system including radio and telephone installations, these will be extended to cater for the project. Technologies installed at MUG is expected to include:

- Mine control employing Fewzion technology.
- Person to person communications consists of hand-held and vehicle mounted radios transmitting via leaky feeder cable reticulated through the mine.
- Proximity detection (equipment to equipment and equipment to person).
- Electronic tagging and tracking and proximity detection using proprietary cap lamps.

18.9 Power and Electrical

Power is supplied through the local utility. The power supply is provided from the national grid and supplied to the company substation at the mill location and mine locations. The company has backup generation available to support the main lines if needed. The mine is currently allocated 12 MW but during peak holiday seasons, the mine is currently restricted in its power draw to 9 MW and some areas of the operations are shut down during these times.

The Waihi Mine Site power is supplied from the Waihi Substation via dual 11 kV Powerlines. Normally the maximum site load is limited to 12 MW but during public holidays the site is restricted to 5 MW and the WPP is usually shutdown during these times. The current load for surface and underground is between 7.6 MW and 7.9 MW but the WPP is not fully loaded due to low feed tonnages from the underground. With the MOP coming online to provide backfill the site maximum demand has been estimated to increase to approximately 12.4MW as shown below:

Project power demands average and peak are shown below in Table 18-3.

Table 18-3: Waihi Power Demands

Activity	Peak Power (MW)	Average Power (MW)
Ball and SAG mill	3.4	2.4
Other processing and water treatment	0.9	0.9
MUG	7.3	7.0
OP crushing and conveying	2.5	2.5
Total Power Draw	**14.1**	**12.4**

As the existing infrastructure at the Waihi Town Substation is not capable of handling this additional load, PowerCo was requested to investigate what is required in the way of new infrastructure to cater for the expected load increases. PowerCo issued a preliminary high-level options report based on the installation of a new 33 kV Powerline from the Waikino Substation (11 km's north of Waihi Township) to the Waihi Town Substation and from there to a proposed new 33 kV/11 KV Substation at the WPP. Edison Consulting Group were chosen by PowerCo to carry out concept design report. Various options were tabled with and without connection to the Waihi Substation and within road reserve or private land. Cost estimates ranged from NZ$10 M to NZ$13 M.

18.10 Infrastructure Manning

General administration staff include management, administration, human resources, safety, community support and environmental technicians and number 38 over the project life. The planned administration structure is shown below in Table 18-4.

Table 18-4: General & Administration Manning

Position	No.	Comment
General Manager	1	
Executive Assistant / Receptionist	1	
Health, Safety, Environmental Manager	2	
Safety Superintendent Mines Rescue	3	
HSLP Administrator	1	
Commercial Supt	1	
Mine Accountant	1	
Accounts Payable Officer	1	
Payroll Officer	-	Covered in Corporate allocated costs
Purchasing Officer	1	
Surface Storeman	2	
HR Manager	1	
HR Assistant	2	
Senior Environmental Officer	3	
Environmental Technician	5	
Security Guard	-	Independent contractors
Occupational Hygienist / Nurse	1	
Education Officer	1	
Community Relations Manager	1	
Community Liaison Officer	2	
External Affairs Coordinator	1	
Site Services	5	
Total Labour	**38**	

18.11 Comments on Infrastructure

In the opinion of the QP, the existing infrastructure is appropriate to support the Mineral Reserve.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The mine has been operational continuously for the last 30 years and has current contracts in place for doré refining and other goods and services required to operate an underground mine and open pit mine.

19.2 Commodity Price Projections

Metal price assumptions are provided by OceanaGold Corporation. Prices used for the December 2020 Mineral Reserve estimates:

- Gold: USD 1,500/oz
- Exchange Rate 1 NZD = USD 0.71.

The metal price assumptions provided by OceanaGold Corporation for the December 2020 Mineral Resource estimates:

- Gold USD 1,700/oz
- Exchange Rate 1 NZD = USD 0.71.

19.3 Contracts

OceanaGold has agreements at typical industry benchmark terms for metal payables and refining charges for doré produced from the Waihi operations. Gold and silver bearing doré is shipped to an Australian refinery for further processing under a toll refining agreement.

Contracts are in place covering underground mining, transportation and refining of bullion, and the purchase and delivery of fuel, electricity supply, explosives and other commodities. These agreements conform to industry norms.

19.4 Comments on Market Studies and Contracts

In the opinion of the QPs:

- OceanaGold is able to market the doré products produced from the Project.
- The terms contained within the sales contracts are typical and consistent with standard industry practice and are similar to contracts for the supply of doré elsewhere in the world.
- Metal prices are set by OceanaGold Corporation management and are appropriate to the commodity and mine life projections.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Statutory Requirements for Environmental Consents

All consents, permits and licences are in place for the Martha underground. Additional consents under the RMA 1991 will be required for the raising of TSF 1A above the consented 177.25 m RL.

20.2 Regulatory Context

The regulatory agencies responsible for consents, permits and licenses associated with the Martha project are:

- Crown Minerals for the Mining License (ML) under the Mining Act 1971 and Mining Permit under the Crown Minerals Act 1991.
- The Waikato Regional Council takes responsibility for air and water quality issues, including vegetation removal and earthworks activities that can give rise to erosion of soils, for activities that affect any of these values in respect of both the ML and resource consents under the RMA 1991.
- The HDC is responsible for the management of land use and community issues in respect of both the ML and resource consents. It is also responsible for Building Permits under the Building Act 1991.
- The Historic Places Trust for an “Authority to occupy or modify any archaeological site”. Any use or rehabilitation of old mine workings or preservation or modification of old surface structures will require such an Authority.

20.2.1 Favona Mining Permit MP 41 808

The provisions of the CMA cover the allocation of the minerals, including royalties. Under the CMA, Favona MP 41 808 was granted on the 22nd March 2004 for the duration of 25 years over the Favona Mine. Subsequently an extension of land was sought on Favona MP 41 808 to cover the Trio project and potential resource extensions on the Martha vein system and was granted on the 14th March 2006. This permit provides for mining the Martha underground project.

20.3 Permitting Process and Schedule

All land use, water discharge and take and air discharge permits are in place for the MUG project. Additional consents under the RMA 1991 will be required for the raising of TSF 1A above 177.25.

Permitting of the MUG project involved submitting applications under the RMA 1991 with two regulatory authorities, the Regional and Local councils. Permitting is a fully public process involving lodgement of applications, notification to the public requesting submissions, and a review of the project by independent commissioners who hand down a decision. This decision can be objected to by any submitter and referred to the Environment Court which will hear the submissions and make a binding ruling.

OGC has a well established environmental and external affairs team that monitors and responds to changes in license, permit and agreement conditions. The MUG project does add complexity and will change conditions and reporting requirement for the operating site due to changes in consent conditions related to blast vibration, ground settlement, property and social compensation, cultural commitments, reporting and monitoring.

OGC has in place sophisticated monitoring procedures as well as a peer review process for noise, vibration, water quality and dust emissions. The regulatory authorities also have in place monitoring facilities. As part of the existing permit conditions, OGC also supplies to the regulators monitoring plans and responses to changes in conditions. It is up to the permit holder to comply with the conditions.

20.3.1 Hauraki District Council

Land use consent has been obtained from HDC for MUG. The land use consent covers and places controls on, the majority of activities associated with the project. The key consent conditions relate to blast vibration, ground settlement, property compensation and social / cultural activities. In accordance with the consent and physical works commencing OGC has provided:

- management plans for vibration and noise,
- additional vibration monitors,
- structural surveys of selected residences / school, and
- ex gratia payments to properties overlying development.

20.3.2 Waikato Regional Council

The following consents have been obtained from the WRC and are in place:

- Discharge permit in respect of all air emissions, (dust; CO2; blasting fumes; vehicle fumes; emissions from the ventilation shaft).
- Water permit to pump groundwater for dewatering the underground mine.
- Discharge permit to place waste underground as backfill.
- Discharge permit flood the underground post-closure with treated water from the WTP.
- Discharge permit to allow degraded-quality groundwater to discharge from the flooded workings into the surrounding ground post-closure.

20.3.3 Crown Minerals Act

The required permit Favona MP 41 808 is in place.

20.3.4 Historic Places Trust

Consent to modify or destroy or reinforce any early relics on surface or underground will not be required.

20.4 Environmental Baseline Assessment and Data Acquisition

Environmental data has been collected over the last 32 years of Waihi operations and baseline data was collected prior to the start of operations and reported in the original ML application. Data is routinely collected for noise levels, blast vibration traces, air quality, discharge water quality from various sources, ground settlement and ground water levels. All data collected is peer reviewed on an annual basis by independent reviewers.

20.5 Environmental Design Criteria

Project environment design criteria is focused on limiting noise, blast vibration, ground differential settlement, air quality and water quality to as a minimum comply with the permit conditions. Key environmental permit limits are:

1. Noise**:** limits at the nearest privately owned residence is 55 dB (L10) measured at the boundary during daytime hours and 40 dB (L10) during night time. Noise from the MUG Mine is required to be measured cumulatively along with all operations within the processing plant, waste and tailings area, and the conveyor and associated activities.
2. Blast vibration**:** conditions require that there shall be no more than three blast events per day, from Monday to Saturday and between 0700 and 2000, and no blasting shall be undertaken at night (2000 to 0700 the following day), on Sundays or on public holidays. In addition:

   The peak particle velocity (vector sum) shall be no more than:

   For development blasts;
   - 5 mm/s for 95% of the monitored events.
   - 2 mm/s on average.

   For production blasts;
   - 5 mm/s for 95% of the monitored events.
   - 3 mm/s on average.

   Compliance: with the 95% and average limits shall be measured over a six-month rolling period and determined separately for development blast events and for production blast events,
3. Air quality**:** Dust arising from operations shall not exceed the following levels more than twice a year:
   - Dust deposition: 5 grams per square metre per month;
   - Total particulate matter concentration: 100 micrograms per cubic metre;
   - Particulate matter of less than 10 microns: 55 micrograms per cubic metre.
4. Underground mining within the Martha underground mine shall be conducted to ensure ground surface stability. This shall include adoption of the following measures:
   a) Mining methods shall be restricted to those that require stope voids created or enlarged as a result of this consent to be backfilled;
   b) Historical open voids formed from caving or stoping shall be identified to be backfilled to ensure that these do not cause localised disturbance or displacement as a result of interaction with future stoping. This is to ensure short-term and long-term stability;
   c) No stoping in the Rex Orebody shall occur above a depth of at least 40 m below the top of the andesite, unless investigations reported to the Council demonstrate to its satisfaction that a lesser depth will ensure surface stability. Any such investigation report is to include, at least, results from groundwater monitoring above the Rex workings, results from extensometers installed from the surface above the Rex workings, and surface settlement results from markers in the area above the Rex Orebody;
   d) No stoping shall occur within 20 m of the mapped extent of the Milking Cow Zone.
   e) Backfilling of any other underground workings that overlap with the MUG Mine where geotechnical conditions require backfilling to ensure long-term stability;
5. Lighting: any lighting installed in the project area shall not exceed 8lux at the boundary of the site.
6. Storage of hazardous materials is limited to approved currently quantities for diesel, ammonium nitrate, packaged explosives and detonators.
7. Water quality: the discharges authorised by the permits shall not cause a significant adverse environmental effect on the receiving water, or on users of that resource, or in the case of surface water, on aquatic biota.

8. Differential ground settlement: when measured between two permanent survey markers is to be less than 1 in 1000 so as to ensure no damage to structures by a wide margin.

20.6 Environmental Assessment

Environmental studies were conducted to support the permit application for Martha underground. The environment effects based reports were all independently reviewed by consultants employed by the Regulators (permit issuers). Studies include, air quality, water quality and ecology, noise, blast vibration effects, traffic, potential for subsidence, ground settlement in response to dewatering, property values, dewatering and geochemistry of tailings, waste and groundwater. These are discussed briefly below.

20.6.1 Air Quality

BECA was engaged by OGC to undertake an assessment of the air quality implications. BECA's report concludes that the effects can be managed to comply with the conditions of existing discharge permits, which have proven to be effective in avoiding or mitigating effects on the Waihi environment. The main conclusions and recommendation from the report are:

a) All above ground activities associated with Martha underground are located within the Martha extended project, the Favona, or the Trio project areas and therefore no additional consent for discharges to air is required. The location of the CAF batching plant at the Favona portal stockpile is covered by the current permits issued for the Favona Underground Mine.
b) Stockpiling associated with Martha underground is not expected to be an air quality issue, but any dust issues that may arise will need to be addressed in a proactive way.
c) Operation of the processing plant for the ore from the Martha underground will be similar to that for the Favona and Trio underground mines and the Martha Mine, and so the air quality significance of those activities will continue to be in full compliance with Discharge Permit 971281.

20.6.2 Noise

Marshall Day acoustic consultants were commissioned to undertake noise studies in support of the Martha permit application and to review the potential effects of truck movements, fill plant operation and ventilation fan noise on the environment. Marshall Day report noted that the following activities have the potential to generate noise:

- Construction and operation of the vent shafts;
- Operation of the CAF batching plant to be located in the current stockpile area near the Favona portal (now excluded from WCP); and
- The use of existing stockpile areas for the temporary storage of ore, waste rock and the crushed rock and aggregate to be used for backfilling.

The noise conditions for MUG are as set out in Table 20-1 below.

Table 20-1: Noise Limits MUG Mine

Time Period	Noise Level	
	L10	Lmax
Monday – Friday 0700 – 2100	55	N/A
Saturday 0700 – 1200	55	N/A
At all other times	40	70

Marshall Day concluded that stockpiling the waste rock in the processing plant and stockpile area will continue as at present and there will be no change in the noise generated, which has been demonstrated in practice to be in full compliance with consent limits.

20.6.3 Blast Vibration

Heilig & Partners were commissioned to undertake blast vibration studies in support of the MUG permit application to review the potential effects of blasting activities on the environment and residents amenity.

The Hauraki District Plan has a vibration limit of 5 mm/s for impulsive vibration. Compliance with the 5 mm/s value ensures there is no superficial damage or structural damage to the Waihi properties, as it is significantly less than the vibration levels necessary to cause damage.

Following a comprehensive review of the conditions contained in the mining licences and resource consents which authorise mining activities at Waihi, Hauraki District Plan vibration performance standards and international guidelines from Australia, Britain, Germany, a suite of conditions were developed to ensure the Martha project will retain the amenity values of Waihi's town centre and residential environment.

These conditions are among the most rigorous applied to blasting. The permit conditions require compliance at average and maximum levels at each installed vibration monitor over six-monthly intervals. Specific conditions providing design criteria are:

- No more than three blast events per day, no blasting shall at night (2000 to 0700 hrs the following day), on Sundays or on public holidays and the peak particle velocity (vector sum) limits. Other significant criterion is blast duration, i.e.
- For all blast events, including those involving a combination of production and development blasts (95% compliance at 5mm/sec):
    - Production blasts shall have a total duration of not more than 9 seconds, 5 mm/sec for 95% monitored events and 3 mm/sec on average;
    - Development blasts shall have a duration of not more than 12 seconds, 5 mm/sec for 95% monitored events and 2 mm/sec on average; and
    - A combination of production and development blasts shall have a duration of not more than 12 seconds.
- No blast event shall have duration of more than 18 seconds.

In addition the consent imposes a cost penalty on the Company for blasts through a payment to nearby residents on the basis of blast magnitude, Clause 36.

> *The consent holder shall use the recorded data from the vibration compliance monitoring network to estimate the vibration received at occupied residences from blasting associated with the Martha Pit and the Martha underground mine, and shall*

make payments to the occupiers of those residences in accordance with the table and criteria below:

Table 20-2: Vibration Payments

Vibration Magnitude	Payment per Blast Event
>1.5mm/sec	18.68
>3.5mm/sec	55.92
>5mm/sec	186.75
>6mm/sec	371.69

The stated payment rates are those existing at 1 January 2018. The rates will be adjusted for the start of each calendar year by the CPI.

20.6.4 Hydrogeology

GWS Ltd was commissioned to undertake studies in relation to the groundwater lowering required to access MUG. The studies also provided information to determine the amount of ground surface settlement that would occur as a result of the depressurisation. A description of their studies and conclusions are provided in section 16 of this report.

20.6.5 Settlement and Dewatering

Engineering Geology Ltd was commissioned to assess the magnitude of settlement due to dewatering of Martha underground. Engineering Geology Ltd concluded that:

- The groundwater level in the Martha project will have to be lowered by up to 200 m below the existing consented dewatering level to RL700 m of the Trio and Correnso underground mine. Dewatering will be achieved by the installation of stage pumping chambers and wells within the mine that pump into the Correnso dewatering system at RL780 m.
- Estimated settlements arising due to the proposed dewatering from RL700 m to RL500 m are considered acceptable and expected to occur within the andesite rock mass below the ground surface.
- Differential settlements, which are normally the concern for buildings and shallow buried services, if they occur, are expected to be very small (less than 1 in 1,000). This is because settlements are associated with the andesite rock mass and not the overlying younger volcanic deposit or surficial soils, and so damage potential is considered negligible.
- Groundwater drawdown associated with dewatering of the Martha project area is expected to be confined to the andesite bedrock. Groundwater levels in the overlying younger volcanic deposits are generally not expected to be affected.

To address this issue the permit conditions require that:

- Clause 73: The VMP shall include a procedure describing preventative and mitigation actions that would be implemented to ensure that the mining in the Rex Orebody does not drain the strata overlying the andesite via existing drill holes and structures. Preventative and mitigation actions may include grouting drill holes from underground where underground development intercepts holes into which water flows or geotechnical defects with significant and sustained water flows.

20.6.6 Geochemistry

OGC engaged AECOM to carry out a geochemical study of ore, tailings and waste rock expected to be recovered from Martha and assess the potential influence these materials may have on the environment. AECOM conducted analytical and leach (static and kinetic) tests on ore and waste material from Martha drill core and concluded:

- Overburden and ore material excavated from MOP4 and the MOP mines will comprise similar geology to that mined from the Martha Pit over the previous three decades and as such no significant difference to the overburden management or philosophy from past practices is expected.
- For overburden placed within temporary stockpiles, it is proposed to amend with limestone as a rate of 12 kg $CaCO_3$ per tonne of overburden or at an amendment rate of approximately 1.2% in order to introduce a lag period of 30 weeks, which should allow sufficient time for the overburden to be placed outside of the zone of oxidation. Additional dosing is recommended where the exposure period is likely to extend beyond the 30 weeks period.
- Overburden backfilled into the underground workings is unlikely to impact groundwater based on a limited potential for oxidation once the materials are at their final destinations. The current oversaturation of groundwater in respect to sulphate and a likely high degree of attenuation on trace elements via the sorption to ion-hydroxide minerals will ensure that any impacts of groundwater quality within the vicinity of the workings as a result of oxidised overburden should be minimal.

20.7 Stakeholder Engagement

OceanaGold (formally Waihi Gold) has undertaken community consultation since the start of operations and the role of the Community Liaison Person is established in the Mining License and subsequent permits.

Consultation has been undertaken with the following parties:

- Land owners and occupiers in the immediate vicinity and neighbouring the permit area
- The rest of Waihi
- Iwi groups - Ngati Hako, Ngati Tamatera, Ngati Maru, Ngati Koi, Ngati Tara Tokanui, Ngati Maru, Ngati Whanaunga and Ngati Pu
- Government departments including Ministry of Energy and Resources, Ministry for Economic Development, Department of Conservation.
- Waikato Regional Council and Hauraki District Council and Thames Coromandel District Council.
- Consultation has taken the form of neighbourhood and community meetings, mail outs of information brochures and questionnaires, local medial articles, a dedicated web-site linking to results of technical monitoring, home visits and individual meetings.

20.7.1 Property Program

The consent specifies that for stoping or intensive mine development below a residential property, the consent holder shall offer to:

a) Purchase that property from the registered proprietor at market value. This offer shall be set by reference to the two independent valuations required by Condition 90; or
b) If the registered proprietor prefers, to provide an ex gratia payment equal to 5% of the property's market value to the registered proprietor.

A process for market valuation, additional payments to cover legal, moving and inconvenience are defined as well as an arbitration process.

20.7.2 Project Closure and Reclamation

Rehabilitation proposals and concept plans were developed well before the commencement of construction for OP mining in 1987, and those plans are revised annually. In preparing these plans, the advice and skill of a large range of experts, including soil scientists, hydrologists, engineers, aquatic biology and water quality specialists has been sought. Where possible, OGC progressively rehabilitates areas of disturbed land.

20.7.3 Rehabilitation and Closure Plan

Rehabilitation proposals and concept plans were developed well before the commencement of construction for open pit mining in 1987, and those plans are revised annually. In preparing these plans, the advice and skill of a large range of experts, including soil scientists, hydrologists, engineers, aquatic biology and water quality specialists has been sought. Where possible, OceanaGold progressively rehabilitates areas of disturbed land.

Closure of the Martha underground Mine will involve the removal of the underground infrastructure. Backfilling of the stopes will occur as a part of stoping and is required by the consents. The shafts will be filled with rock and capped with concrete and the portal will be plugged or otherwise blocked off. Rehabilitation of other facilities such as the processing mill, water treatment plan and tailings storage facilities are already provided for under the resource consents.

Re-flooding of the underground workings will occur naturally from groundwater recharge once dewatering required for underground mining has ceased and will also occur as part of the consented pit lake formation which is provided for under the Company's Rehabilitation and Closure Plan. River and treated water may be used to supplement the natural groundwater inflows and accelerate re-flooding.

Hauraki District Council and Waikato Regional Council hold both cash and bank bonds over the Company for the quantum of the closure works. The purpose of the rehabilitation bond is to provide the Councils with unencumbered access to a source of funds to close and rehabilitate the current mine site in the unlikely event that OceanaGold fails to meet its closure obligations. The quantum of this bond is assessed annually, calculated on the basis of the cost to close the site at the end of each 12-month bond period.

Each year, a Rehabilitation and Closure Plan is prepared to describe the proposed method of rehabilitation and closure of the site. The overall objective of this plan is to ensure rehabilitation and closure of the site in such a manner that in the long-term the site, and any structures on it, will remain stable; and any water discharging from the site, and any groundwater under the site, will be of a quality such that it will not adversely affect aquatic life, or other users of the water resource.

20.8 Social & Cultural Impacts

Waihi is a town in Hauraki District, located at the base of the Coromandel Peninsula. Approximately 4,500 people make up the community of Waihi. Gold mining has been a feature of the region for three centuries with modern mining practices being present since 1987.

The Waihi community has experienced a history of mining operations both open pit and underground and new projects during this time. With the close proximity of a community to these operations, the need to understand and manage how mining affects the community and society at large is integral to successfully operate within a town. This has led to mining becoming part of the social fabric and identity of Waihi today, creating strong community connections built through trust by sharing information.

To identify and analyse how mining affects the Waihi community, reliable information is gathered through various monitoring methods across a broad section of disciplines. Part of this suite of monitoring information has been the extensive use of community surveys and polls, alongside multiple Social Impact Assessment conducted by credible third-party experts. Since 2014 the company has been required by consent condition to conduct an annual Social Impact Monitoring Plan report. This report monitors and reviews community perception and social impacts. It provides the opportunity for the local regulatory authority (HDC) to make comment or suggest modifications to activities and requires the company to suggest mitigating actions to address any negative issues or trends that are identified.

These monitoring methods help to inform strategies to alleviate any concerns raised by the community. In order to build trust and relationships unconstrained communication with the company is important as this ensures the best possible opportunity for community members to provide feedback, an example is the Complaints Hotline and procedure. It provides the community an opportunity to provide continual and unfettered feedback whenever they feel appropriate, meaning the community has access to communicate with the company whenever they need. Another example of providing access, is the appointment of a Community Liaison Officer, someone to act as the main point of contact between the community and the company.

Mitigation strategies have been implemented through innovation and use of policy. With technology continuing to evolve new and innovative ideas become more viable and effective. The Amenity Effects Program (AEP) arose from mediation with residents during a consent application and is a condition of consent. The program allows for a scientific method to be applied in identifying homes that are affected by vibration. An objective formula calculates the amount of mitigation the company needs to pay to the affected resident. Another example of innovation can be seen through the Blast Notification Program, where an alert device notifies residents' (who participate in the scheme) before a blast is initiated. By utilising innovative technology, proprietary hardware and public information sessions a process was developed to provide predictability and a sense of control around blasting. These issues were identified as needing solutions through the various monitoring and work done with the community.

Not all innovative ideas require technology but instead rely on a new perspective and understanding. Maori have a close spiritual connection with the earth and waterways. The physical and spiritual survival of all things dependent on the maintenance of the life force (Mauri), spirit, power and sacredness of Papatuanuku (Earth Mother). Maori regard land, soil and water as taonga (treasures) and they consider themselves to be the kaitiaki (guardians) of these taonga. Pukewa is the Maori name for the hill that once stood at the centre of Waihi town. Pukewa was mined. The mining of Pukewa had significant negative impact on the spiritual connection of local Iwi with the land. To assist in mitigating this impact, solutions are sought through collaborative means and robust engagement practices with an Iwi liaison group. A group that invites all interested Iwi groups to be involved in discussions on how to address the cultural significance of the area and provide up to date information about the operations as well as the progress of projects. The development of a cultural balance plan in partnership with local Iwi is in progress. The aim of the plan is to identify the Mauri of the whenua (land) and to work collaboratively to ensure the life force is restored at the conclusion of mining. Iwi are one example of how the company strives to build mutual trust through shared values, multiple opportunities are continually sought with key stakeholder groups that are identified through various methodologies of understanding the context.

Stakeholder identification is used to assist with effective engagement practices. Extensive work to map stakeholders using the technical studies, social risk assessment and local knowledge allows for a targeted approach to ensure relevant information is provided to the appropriate groups.

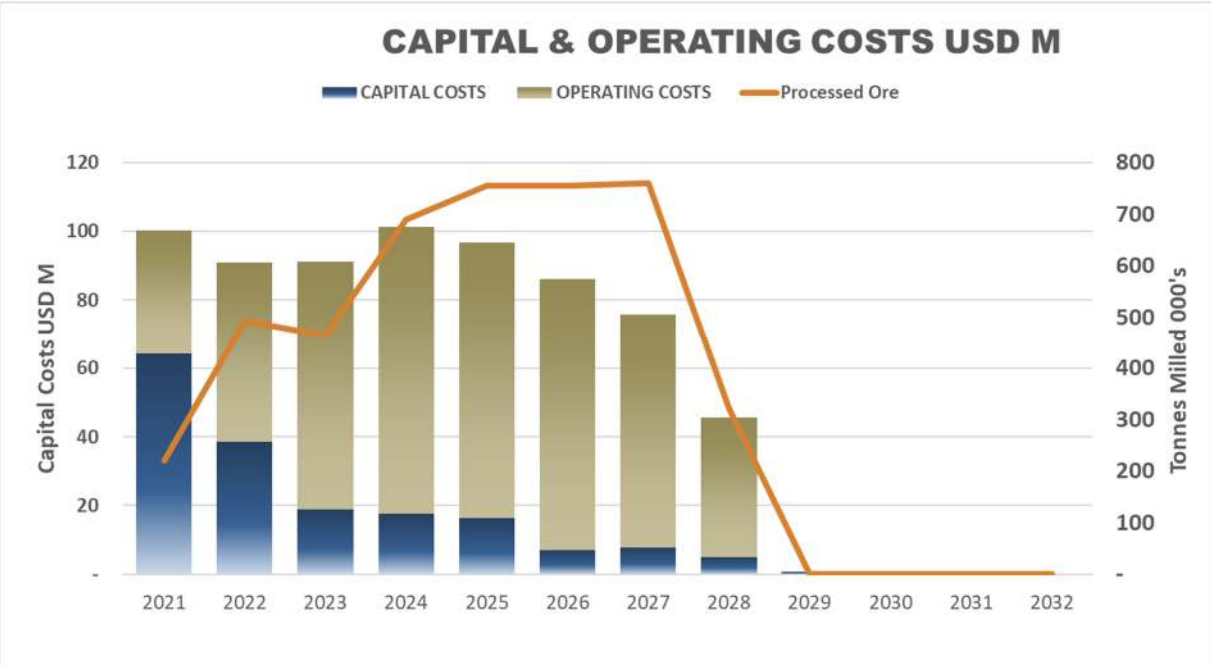
The way we communicate is an important aspect of how we listen to the various perspectives and expectations that can lead to conflicting interests within the project and decision making processes. Due to the diverse range of perspectives and experiences a participatory approach is required.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

For the MUG Feasibility Study, the project's technical team together with specialist consultants prepared estimates of both capital and operating costs associated with the LOM production schedule used to determine the Mineral Reserve. This section of the report presents and details the basis of the Capital (CapEx) and Operating (OpEx) costs estimates. All estimates are based on annual inputs of physicals and all financial data is first quarter CY 2021, all currency is in US dollars (USD), unless otherwise stated. An exchange rate of USD 0.71 / NZD has been used throughout. Estimated capital and operating costs for Martha underground are shown in Figure 21-1.

Figure 21-1: Capital & Operating Cost Estimate



21.2 Martha Underground Mining Costs

Underground mine operating costs have been estimated from first principles (but referencing current mining costs), derived from OEM, contractor and supplier quotations, firm quotations and in place contracts or estimated internally by OceanaGold based on the sites operating history. Unit costs for materials and supplies were sourced mainly from firm quotations, contracts and recent vendor input where available. This information is located in the cost model spreadsheet:

- (STU058_Cost_Model_UG_FS_MUG_Indicated_final_2021031.xlsm).

A constant LOM diesel price of US$0.67 /litre was assumed, along with an electricity cost of US$0.078 /kWh. The mine services cost category includes a miscellaneous supplies allowance that is equal to 10% of the total cost of the itemised consumables for each of the key mining activities.

Salary, hourly wage rate, and fringe benefit information was provided by OceanaGold based on the current wage and benefit / overheads structure that is in place for the Martha underground operation.

Total mine services costs and total labour costs for each period were estimated and then allocated on a pro rata basis across the key mining activities based on the equipment operating costs and consumables costs associated with each key mining activity. The estimate includes a provision for the staffing and operation of a diamond core drill for stope definition drilling.

21.2.1 Activity Driven Costs

Operating costs are based on the underground mine ramping up to and then stabilising at a nominal mining rate of 1,924 t/d. Costs are estimated on a period-by-period monthly basis in accordance with the production schedule. The operating costs were estimated from the most recent information available, and in all instances the cost inputs were sourced from either 2020 or 2021 cost data. OGC considers the operating cost estimate to be at a FS level of accuracy (± 15%). The underground mine cost summary is shown in Table 21-1 and Table 21-2.

Table 21-1: Underground Mine Cost Summary by Expense

Item	USD 000's	US$/t-ore	%
Labour	125,674	28.19	36%
Backfill	-	-	0%
Fuel	10,636	2.39	3%
Power	37,823	8.48	11%
Equipment Operating	35,283	7.91	10%
Equipment Maintenance	57,808	12.97	17%
Explosives	21,130	4.74	6%
Ground Support	38,390	8.61	11%
Grout/Shotcrete	8,766	1.97	3%
Services	7,337	1.65	2%
Contracts	6,523	1.46	2%
TOTAL	**349,371**	**78.37**	**100%**

Table 21-2: Underground Mine Cost Summary by Activity

Item	US$ 000's	US$/t-ore	%
Supervision & Control	77,092	17.29	22%
Lateral Development	101,604	22.79	29%
Vertical Development	1,406	0.32	0%
Production	53,134	11.92	15%
Materials Handling	31,155	6.99	9%
Backfill	20,837	4.67	6%
Mine Services	64,142	14.39	18%
TOTAL Cost	**349,371**	**78.37**	**100%**

Table 21-3 through to

Table 21-9 show the unit costs for key mining activities (with itemised detail provided for each of the main inputs). Note that for each of the key mining activities, the cost of loading muck into haul trucks at a remuck and then hauling the muck to the surface (or to an internal waste dump as the case may be) is a separate cost item that is not included in the "Equipment Operation" category. Also note that in the case of medium-term and long-term development openings (main ramp, footwall accesses, ventilation drifts, and ventilation drop raises) an operating cost is estimated but this cost will be treated as either an initial or sustaining capital expenditure depending on when the development mining occurs.

Table 21-3: Supervision and Control

Item	US$ 000's	US$/t-ore	%
Labour	60,720	13.62	79%
Fill	-	-	0%
Fuel	595	0.13	1%
Power	279	0.06	0%
Equipment Operating	6,371	1.43	8%
Equipment Maintenance	2,604	0.58	3%
Explosives	-	-	0%
Ground Support	-	-	0%
Grout/Shotcrete	-	-	0%
Services	-	-	0%
Contracts	6,523	1.46	8%
TOTAL	**77,092**	**17.29**	**100%**

Table 21-4: Lateral Development

Item	US$ 000's	US$/m	%
Labour	17,749	344.88	17%
Fill	-	-	0%
Fuel	2,089	40.58	2%
Power	360	6.99	0%
Equipment Operating	16,228	315.31	16%
Equipment Maintenance	13,290	258.23	13%

Explosives	14,554	282.79	14%
Ground Support	23,256	451.87	23%
Grout/Shotcrete	8,713	169.29	9%
Services	5,366	104.27	5%
Contracts	-	-	0%
TOTAL	**101,604**	**1,974.22**	**100%**

Table 21-5: Material Handling

Item	US$ 000's	US$/t-ore	%
Labour	10,160	2.28	33%
Fill	-	-	0%
Fuel	4,834	1.08	16%
Power	-	-	0%
Equipment Operating	2,451	0.55	8%
Equipment Maintenance	11,740	2.63	38%
Explosives	-	-	0%
Ground Support	-	-	0%
Grout/Shotcrete	-	-	0%
Services	1,971	0.44	6%
Contracts	-	-	0%
TOTAL	**31,155**	**6.99**	**100%**

Table 21-6: Backfill

Item	US$ 000's	US$/m^3	%
Labour	-	-	0%
Fill	-	-	0%
Fuel	850	-	4%
Power	-	-	0%
Equipment Operating	727	-	3%
Equipment Maintenance	4,574	-	22%
Explosives	-	-	0%
Ground Support	14,686	-	70%
Grout/Shotcrete	-	-	0%
Services	-	-	0%
Contracts	-	-	0%

TOTAL	**20,837**	**-**	**100%**

Table 21-7: Production

Item	US$ 000's	US$/t-ore	%
Labour	22,701	5.09	43%
Fill	-	-	0%
Fuel	432	0.10	1%
Power	323	0.07	1%
Equipment Operating	8,154	1.83	15%
Equipment Maintenance	14,782	3.32	28%
Explosives	6,240	1.40	12%
Ground Support	449	0.10	1%
Grout/Shotcrete	53	0.01	0%
Services	-	-	0%
Contracts	-	-	0%
TOTAL	**53,134**	**11.92**	**100%**

Table 21-8: Mine Services

Item	US$ 000's	US$/t-ore	%
Labour	14,344	3.22	22%
Fill	-	-	0%
Fuel	1,772	0.40	3%
Power	36,813	8.26	57%
Equipment Operating	684	0.15	1%
Equipment Maintenance	10,528	2.36	16%
Explosives	-	-	0%
Ground Support	-	-	0%
Grout/Shotcrete	-	-	0%
Services	-	-	0%

Contracts	-	-	0%
TOTAL	**64,142**	**14.39**	**100%**

Table 21-9: Vertical Development

Item	US$ 000's	US$/t-ore	%
Labour	-	-	0%
Fill	-	-	0%
Fuel	63	24.61	5%
Power	48	18.80	3%
Equipment Operating	668	259.53	48%
Equipment Maintenance	291	112.90	21%
Explosives	336	130.48	24%
Ground Support	-	-	0%
Grout/Shotcrete	-	-	0%
Services	-	-	0%
Contracts	-	-	0%
TOTAL	**1,406**	**546.31**	**100%**

The above costs are split into operating costs and capital costs based on the time period to achieve an ore production rate of 1,400tpd except for the mine development that is capitalised over the life of the mine.

21.2.2 Other Costs

Other costs include the finance leases on mobile equipment, major rebuilds on mobile plant, fixed underground and surface infrastructure and light vehicle purchases.

These costs together with the activity driven costs are shown below by year. Table 21-10 shows the total cost, Table 21-11 shows the capital cost estimate and Table 21-12 shows the operating costs.

Finance leases are calculated on a four year lease with no residual and 4.5% interest rate. Finance leases are applied to all major mobile plant but not to light vehicles.

Table 21-10: Summary of Costs by Activity USD 000's

UNDERGROUND MINING	2021	2022	2023	2024	2025	2026	2027	2028	TOTAL
Grade Control Drilling	765	2,094	1,103	846	397	553	-	-	5,758
Supervision and Control	10,125	11,662	10,625	10,261	9,740	9,599	8,804	6,279	77,093
Lateral Development	19,032	19,451	19,325	18,107	14,334	9,286	1,533	536	101,605
Vertical Development	455	270	354	163	124	40	-	-	1,406
Production	5,063	6,701	7,712	8,636	7,689	7,659	6,476	3,198	53,134
Material Handling	3,377	4,744	4,976	5,313	4,655	3,548	3,028	1,515	31,155
Backfill	1,599	1,149	5,327	5,294	1,983	2,077	2,492	917	20,837
Mine Services	7,012	7,653	8,054	9,962	9,913	8,677	7,569	5,301	64,140
Diamond Drilling Pre-prod	-	-	-	-	-	-	-	-	-
Finance Leases - Mobile Plant	5,384	5,531	6,687	7,426	3,072	2,925	2,712	4,473	38,210
Rebuilds	-	-	-	-	2,509	2,537	-	-	5,046
Mine Infrastructure - Underground	3,484	1,403	142	931	213	337	-	-	6,510
Mine Infrastructure - Surface	414	-	-	-	-	-	-	-	414
Mine - Light Vehicles	323	369	-	85	85	-	170	-	1,032
Rex Properties	2,840	-	-	-	-	-	-	-	2,840
Capital Other	840	-	-	-	-	-	-	-	840
Finance Lease Interest	868	650	587	405	338	203	245	161	3,457
Total Cost	**60,815**	**59,583**	**63,789**	**66,583**	**54,655**	**46,887**	**33,028**	**22,379**	**407,720**
Unit Cost ($/t ore)	**275.79**	**121.06**	**137.40**	**96.64**	**72.33**	**62.11**	**43.43**	**69.69**	**91.46**

Table 21-11: Summary of Capital Costs by Activity USD 000's

UNDERGROUND MINING	2021	2022	2023	2024	2025	2026	2027	2028	TOTAL
Supervision and Control	7,810	7,401	3,048	1,513	720	553	-	-	21,045
Lateral Development	13,551	11,682	3,851	1,590	1,898	-	-	-	32,572
Vertical Development	341	176	86	12	17	-	-	-	632
Production	3,183	2,971	1,350	530	245	-	-	-	8,279
Material Handling	2,216	2,102	900	324	159	-	-	-	5,701
Backfill	1,351	480	805	296	68	-	-	-	3,000
Mine Services	4,723	3,409	1,426	616	328	-	-	-	10,502
Diamond Drilling Pre-prod	-	-	-	-	-	-	-	-	-
Finance Leases - Mobile Plant	5,384	5,531	6,687	7,426	3,072	2,925	2,712	4,473	38,210
Rebuilds	-	-	-	-	2,509	2,537	-	-	5,046
Mine Infrastructure - Underground	3,484	1,403	142	931	213	337	-	-	6,510
Mine Infrastructure - Surface	414	-	-	-	-	-	-	-	414
Mine - Light Vehicles	323	369	-	85	85	-	170	-	1,032
Rex Properties	2,840	-	-	-	-	-	-	-	2,840
Capital Other	840	-	-	-	-	-	-	-	840
Total Capital Cost	**46,460**	**35,524**	**18,295**	**13,323**	**9,314**	**6,352**	**2,882**	**4,473**	**136,623**
Unit Cost ($/t ore)	**210.69**	**72.18**	**39.41**	**19.34**	**12.33**	**8.41**	**3.79**	**13.93**	**30.65**

Table 21-12: Summary of Operating Costs by Activity USD 000's

OPERATING	2021	2022	2023	2024	2025	2026	2027	2028	TOTAL
Grade Control Drilling	765	2,094	1,103	846	397	553	-	-	5,758
Supervision and Control	2,315	4,261	7,577	8,748	9,020	9,046	8,804	6,279	56,048
Lateral Development	5,481	7,769	15,474	16,517	12,436	9,286	1,533	536	69,033
Vertical Development	114	94	268	151	107	40	-	-	774
Production	1,880	3,730	6,362	8,106	7,444	7,659	6,476	3,198	44,855
Material Handling	1,161	2,642	4,076	4,989	4,496	3,548	3,028	1,515	25,454
Backfill	248	669	4,522	4,998	1,915	2,077	2,492	917	17,837
Mine Services	2,289	4,244	6,628	9,346	9,585	8,677	7,569	5,301	53,638
Finance Lease Interest	868	650	587	405	338	203	245	161	3,457
Total Capital Cost	**15,120**	**26,153**	**46,597**	**54,106**	**45,738**	**41,089**	**30,146**	**17,907**	**276,855**
Unit Cost ($/t ore)	**68.57**	**53.14**	**100.37**	**78.53**	**60.53**	**54.43**	**39.64**	**55.76**	**62.10**

21.3 Processing Costs

Processing operating costs include maintenance and operating labour, consumables, maintenance consumables, power, grinding media and water treatment plant operations, which is well understood from 30 years of almost continuous plant operations in its current configurations. Unit costs per tonne of ore processed are shown below in Table 21-13

Table 21-13: Operating Costs Estimate Processing & WTP Operations

Item	Unit	Unit Costs
Labour Supervision	USD / tonne	1.06
Labour wages	USD / tonne	6.84
Power	USD / tonne	5.30
Crushing and Grinding Operations		
UG Grinding Media Balls Ball Mill	USD / tonne	1.30
UG Grinding Media Balls SAG Mill	USD / tonne	1.74
Primary Mill Liner Set Cost	USD / tonne	0.32
Regrind Mill Liner Set Cost	USD / tonne	0.08
Outside Contractors	USD / tonne	0.39
Fasteners	USD / tonne	0.08
Cyclone Liners	USD / tonne	0.02
Screens	USD / tonne	0.53
CIL and Elution Operations		
UG Flocculent	USD / tonne	0.12
UG Cyanide	USD / tonne	2.08
UG Lime	USD / tonne	0.24
UG Oxygen	USD / tonne	0.48
UG Carbon	USD / tonne	0.17
UG Acids Hydrochloric	USD / tonne	0.24
UG Caustic soda	USD / tonne	0.43
UG Propane	USD / tonne	0.80
Contracted Services	USD / tonne	0.63
Pumps and Other Parts	USD / tonne	0.33
Rubber Liners	USD / tonne	0.20
Electrical	USD / tonne	0.13
Water treatment		
Acids Ferric Chloride	USD / tonne	0.94
Flocculent	USD / tonne	0.02
Hydrogen Peroxide	USD / tonne	0.58
Lime	USD / tonne	1.08
Other Chemicals reagents	USD / tonne	0.01
pH modifiers CO2	USD / tonne	0.13
Services, laboratory and assay	USD / tonne	3.80

The estimated labour force required at the processing plant and water treatment plant to operate at a capacity of 900ktpa of underground ores is shown in Table 21-14 below. Current operations work a four panel even time roster.

Table 21-14: Staffing Estimate Processing & WTP Operations

Management and Technical	No.
Process Plant Manager	1
Process Operations Supt.	1
Senior Metallurgist	1
Metallurgist Trainee	2
Operations	
Plant Supervisor	1
Plant Operators	24
Gold Room Operators	4
Maintenance	
Mech Maintenance Supt	1
Leading Hands	1
Tradesperson Mechanical	7
Tradesperson E & I	2
Trades Assistant	4
Planner	1

21.4 Capital Costs

21.4.1 Basis of Estimate

Capital costs are developed for growth and sustaining capital. Growth capital represents pre-production underground mining and capital required to increase production. OGC developed the sustaining capital cost estimate to account for underground mine development, mine equipment and TSF construction capital costs through the LOM, by applying the same estimating methodology as for growth capital.

The capital cost estimate for the FS has an expected accuracy of ± 15%. Underground capital mine development costs are well known through the sites operating history as is the costs of salaries, wages, ground support, drilling, blasting and mobile plant consumables.

The estimate includes direct and indirect costs (such as engineering, procurement, construction and start-up of facilities) as well as owner's costs and contingency associated with mine and process facilities and on-site infrastructure.

The following areas are included in the estimate:

- Mine (underground mine development, equipment fleet finance leases, backfill plant and supporting infrastructure and services).
- Process plant replacement of existing Waihi SAG Mill shell (currently being fabricated with known costs).
- Tailings Storage Facility raises to TSF1A and TSF2 were estimated by independent consultants.
- On-site infrastructure (water treatment and distribution, electrical substation and distribution, and other general facilities).

- Pit rim works including relocation of public roads, estimated by independent consultant.
- Refurbishment of the overland crusher and conveyor system, estimated by OGC.
- Property purchases above the Rex orebody.
- Duplication of the 33kV line from Waikino to Waihi with a buried cable and new substation.
- Incremental mine site rehabilitation[5].

Engineering work, being in the range of 25‑30% of total engineering for the project, was carried out to support the estimate. The estimate was based on the following project-specific information:

- Mine design,
- TSF raise design.
- Preliminary major mechanical equipment list for mining.
- Equipment fleet and infrastructure for mining.
- Preliminary general site layout.
- Electrical supply trade-off study.
- Earthworks quantities derived from preliminary sketches (sections).

Where possible costs were estimated from similar constructions at Waihi, sourced from OEM or otherwise factored, end-product units and physical dimensions methods were used to estimate costs. The following assumptions were considered:

- All equipment and materials will be new.
- The main equipment will be purchased and manufactured in appropriate sizes to be transported by the existing main roads from Tauranga to the project site.
- The execution work will be continuous without interruptions or stoppages.
- Concrete will be produced at the Firth facility in Waihi.
- Contractors will be contracted under unit price contracts.
- All mining surface and underground mobile equipment is costed as a finance lease.
- The project will be executed by OceanaGold.

The following are excluded from the capital cost estimate:

- Finance costs and interests during construction, except for mobile plant leases.
- Costs due to fluctuations in exchange rates.
- Cost of working capital.
- Changes in the design criteria.
- Changes in scope or accelerated schedule.
- Changes in New Zealand legislation.
- Provisions for force majeure.
- Wrap-up insurance.

[5] Note the site currently has a NZD 57M rehabilitation cost estimate to complete rehabilitation and closure of the mine. The Company also holds approximately NZD 30M of property which will be divested to partially cover the rehabilitation costs. Only the incremental rehabilitation cost is considered i.e. that additional cost solely attributed to MUG.

21.4.2 Labour Assumptions

For mine development current salaries, award wages and overheads in place at Waihi are applied. The construction labour and equipment costs were included in the factors that were used in the estimation to account for installation costs or in the unit costs when applied.

21.4.3 Material Costs

All materials required for facilities refurbishment, mine development or construction are included in the capital cost estimate. Material costs include freight to the site.

Material cost related to the TSF and road relocation were determined by material-take off quantities from sketches/drawings and installation unit costs and supplied by independent consultants.

All earthworks quantities were assumed to be insitu volumes, with allowance for swell, waste or compaction of materials. Industry-standard allowances for swell and compaction were incorporated into the unit rates.

21.4.4 Underground Mining

The underground mine capital cost estimate is comprised of the following main elements.

- **Capitalised operating costs:** This category includes development mining in waste for main ramps, footwall accesses, ventilation connections, and ventilation drop raises.
- **Mine mobile equipment purchases:** This category includes the purchase of new mobile equipment units during the pre-production period to establish the initial mining fleet and later during the production period, to increase the size of the fleet to account for longer truck cycle times as the mine deepens. Mobile equipment costs are calculated as a finance lease with a four year term and at a 4% interest rate. The mine is currently carrying approximately USD1.4M of lease costs.
- **Mine mobile equipment rebuilds and replacements:** The cost of rebuilding mine mobile equipment in accordance with manufacturer-recommended rebuild intervals is allowed for at a cost of 60% of the original purchase price. Replacement of high hour equipment is allowed for in accordance with manufacturer recommendations.
- **Fixed and other equipment:** This category includes fixed equipment such as the cement backfill plant, primary and secondary ventilation fans, dewatering pumps, electrical substations and power centres, refuge chambers, and mine office building upgrades, technical services instruments, and minor plant. It also includes allowances for communications infrastructure and tool allowances.
- **Rex property purchases:** This category allows for purchase of all private residential property overlying stopes and main development in the Rex area. Costs are estimated in terms of the consent conditions allowing for legal, relocation and inconvenience payments as well as the market value of the residences.
- **33kV power upgrade:** This category allows for duplication of the 33kV line with a buried cable from Waikino to Waihi and a new substation. It is required prior to recommencing operation of the open pit crushing and conveying system. This cost is included as growth capital under infrastructure.

OGC has relied upon written quotes from equipment vendors and mining contractors to support the capital cost estimate. Budgetary quotes for mine mobile equipment were obtained from Sandvik, Normet, Atlas Copco, and Caterpillar, with appropriate allowances for freight and set up. Fixed equipment cost estimates were either supported by vendor quotes or were sourced from the Company's database.

The capital costs were estimated from the most recent information available, and in all instances the cost inputs were sourced from either 2019 or 2020 cost data. OGC considers the capital cost estimate to be at a FS level of accuracy (±15%). This information is located in the cost model spreadsheet:

- (STU058_Cost_Model_UG_FS_MUG_Indicated_final_2021031.xlsm).

Table 21-15 shows the capital cost summary.

Table 21-15: Capital Cost Summary for Underground Mining

Description	Initial (US$000's)	Sustaining (US$000's)	LOM (US$000's)
Supervision and Control	3,905	17,139	21,044
Lateral Development	6,775	25,796	32,572
Vertical Development	171	462	632
Production	1,591	6,687	8,279
Material Handling	1,108	4,593	5,701
Backfill	676	2,324	3,000
Mine Services	2,362	8,142	10,503
Subtotal UG Development Capital	16,588	65,144	81,732
Diamond Drilling Pre-prod	0	0	0
Finance Leases - Mobile Plant	2,854	35,356	38,210
Rebuilds	0	5,046	5,046
Mine Infrastructure - Underground	1,949	4,561	6,510
Mine Infrastructure - Surface	0	414	414
Mine - Light Vehicles	0	1,033	1,033
Rex Properties	2,840	0	2,840
Capital Other	839	1	840
Contingency	435	2,818	3,253
Total Capital	**25,504**	**114,373**	**139,877**

Table 21-16 and Table 21-17 show the LOM requirement for mobile and fixed equipment along with the estimated capital cost.

Table 21-16: LOM Estimated Capital Cost for Mobile Equipment

Item	Capital Cost (US$000's/ea.)	Type	Total LOM Requirement	Total Capital Cost (US$000's)
DD421-60C Jumbo	1,373	Mobile	6	8,239
DL421-7C Longhole	1,696	Mobile	5	8,480
LH517 Loader	1,344	Mobile	5	6,722
TH551 Truck	1,583	Mobile	5	7,914
Spraymec 6050WP	755	Mobile	2	1,510
UTIMEC MF 500	599	Mobile	2	1,198
962H Wheel Loader	381	Mobile	3	1,143
938K Wheel Loader	265	Mobile	1	265
12K Grader	311	Mobile	1	311
Charmec MC 605	613	Mobile	3	1,839
UTIMEC SCISSOR	589	Mobile	1	589
Total				**38,210**
Light Vehicles (4x4 Pickup Trucks)	80	Mobile	13	1,033
Total				**39,243**

Table 21-17: LOM Estimated Capital Cost for Fixed Equipment

Item	Capital Cost (US$000's/ea.)	Type	Total LOM Requirement	Total Capital Cost (US$000's)
Surface Batch Plant	840	Fixed	1	840
Rex Properties	237	Fixed	12	2,840
33kV Powerline PFS & Consenting				
Refuge Chambers	85	Fixed	1	85
Escapeways	142	Fixed	12	1,633
Primary Fans	927	Fixed	1	927
Magazine	107	Fixed	1	107
Secondary Fans - 110kW & starter	57	Fixed	10	582
Fan Starter - 110kW Twin	79	Fixed	4	282
Fan Starter - 8kW	8	Fixed	4	31
Fan Starter - 20Kw	11	Fixed	6	66
Fan Starter - 37Kw	17	Fixed	8	129
Pumping Station #2	71	Fixed	4	284
Pumps - 37/20kW	0	Fixed	0	0
Jumbo Starters - Single, Double	21	Fixed	3	58
Jumbo Pump Starter	13	Fixed	1	13
Distribution Boards - 3,4,6 Ways	63	Fixed	1	63
HD Pump Starter	43	Fixed	3	128
HV Cable 600 m Drum	43	Fixed	1	43
Starter Refurbishments	39	Fixed	5	209

Underground HV backbone supply and install	0	Fixed	0	0
Primary Fan Substations	142	Fixed	1	142
Substation 9,10	0	Fixed	0	0
Tele-Remote Stations & equipment	355	Fixed	1	355
Toilets	0	Fixed	0	0
UG Service Bay Infrastructure	71	Fixed	5	337
Capping Shaft #2 in pit pumping	71	Fixed	5	355
Baskets and Implements	43	Fixed	2	85
VSD control for Trio Primary Fans	103	Fixed	1	103
DBR Walls	17	Fixed	7	121
C-ALS Scanner OGC-1064	85	Fixed	1	85
CMS and Scanner	57	Fixed	1	57
Total Station replacement	36	Fixed	1	36
Extensometers for Rex	142	Fixed	1	142
Centralised UG Firing System	53	Fixed	1	53
UG Offices	142	Fixed	1	142
Batch Plant	130	Fixed	1	130
Relocation of Workshop Lubricant and Door Installation	71	Fixed	1	71
Nifty Lift	71	Fixed	1	71
Total				**10,603**

21.4.5 Mine Capital Costs – Open Pit

The Martha pit has not operated since April 2015 when a wall failure on the north wall closed the main ramp. In order to provide backfill, the open pit must recommence operations which will require refurbishment of the existing crusher and conveyor system.

This item accounts for the capital costs associated with the surface mine development. It also includes surface preparation which includes re-routing a public road and refurbishment of the crusher / conveyor system, building upgrades. The initial capital costs for surface mining are shown in Table 21-18.

The crusher refurbishment costs were estimated by OGC based on maintenance and services, OEM pricing for supply and commissioning, steelwork weights and pricing for installations from similar works in 2010 and the ongoing maintenance costs. For estimating purposes, it was assumed that the open pit mine will use a contractor overseen by the owner.

The cost associated with realigning the and relocating the public road were provided by an experience external consultants and based on earthworks quantities estimated from the preliminary general site layout and sketched sections and unit costs sourced from the consultants internal database.

Table 21-18: Surface Mining Capital Costs

Surface Mining Capital Expenditure	Growth	Sustaining
	MUG LOM Estimate USD M	MUG LOM Estimate USD M

Project management and permits	0.00	0.00
Road relocation and minor earthworks	0.00	1.22
Crusher refurbishment	0.00	0.76
Conveyor refurbishment	0.00	0.96
Mine Workshop Refurbishment	0.00	0.50
Mining equipment fleet / LV's	0.00	0.43
Other	0.00	0.00
Contingency	0.00	0.33
Resource drilling	0.00	0.00
Total	**0.00**	**4.20**

21.4.6 Process Capital Costs

This item accounts for the capital costs associated with the process plant, which is sustaining capital. No major changes to the plant is required for the MUG FS and hence capital costs are largely sustaining costs which are well known after 30 years of operations. The only significant capital expenditure is replacement of the existing SAG mill shell. The capital costs for the process plant are shown in Table 21-19.

Table 21-19: Process Plant Capital Costs

Processing Summary Capital Expenditure	Growth MUG LOM Estimate USD M	Sustaining MUG LOM Estimate USD M
Crushing	0.00	0.00
SAG Mill Shell Replacement	0.00	1.46
Leaching	0.00	0.75
Elution	0.00	0.40
Buildings	0.00	0.21
Mobile plant	0.00	0.14
Telemetry and control systems	0.00	0.45
WTP filtration	0.00	0.50
Contingency	0.00	0.59
Total	**0.00**	**4.51**

21.4.7 TSF Construction Costs

The TSF costs were estimated by an independent consultant based on quantities obtained from detailed designs, contractor estimates, material specification and drawings of the TSF and associated infrastructure and unit costs obtained from OGC's internal database.

TSF Constructions	MUG LOM Estimate USD M	MUG LOM Estimate USD M
TSF 1A Lift to 173RL	0.00	0.49
TSF 1A Lift to 177.25RL	0.00	1.45
TSF 1A Lift to 180.25RL	0.00	1.91
TSF 1A Lift to 182RL	0.00	3.08
TSF 2 Lift to 159.5RL	0.00	3.78
Contingency	0.00	1.48
Total	**0.00**	**12.19**

21.4.8 Other Infrastructure

Infrastructure areas include:

- Off-site electrical substation and distribution.
- Power supply (including the relocation of a section of the two existing power lines on-site).

The cost associated with the site electrical substation and on-site distribution was estimated based on a conceptual system design and benchmarked costs for the major components.

The costs for upgrading the total power supply to site was obtained from the regional power supplier, Power Co. The power supply cost includes costs associated with a new transmission line from the Waikino substation to the existing on-site electrical substation, as well as the costs associated with the relocation of a section of the existing transmission lines on-site. These costs were estimated based on sketched routes and benchmark costs sourced from PowerCo's internal database. Infrastructure capital costs are summarised in Table 21-20.

Table 21-20: Infrastructure Capital Costs

Other Infrastructure	Growth	Sustaining
	MUG LOM Estimate USD M	MUG LOM Estimate USD M
33kV Powerline PFS & Consenting	0.12	0.00
Site 33kV Power Supply Upgrade	11.15	0.00
33kV Contingency 10%	1.12	0.00
Infrastructure	**12.39**	**0.00**

21.4.9 General and Administrative Capital Costs

The sustaining capital costs for General and Administrative functions have been estimated based on previous years expenditures, see Table 21-21. Sustaining capital included costs associated with items such as the general office building, IT and warehouses.

Table 21-21: G&A Capital Costs

General & Administration Summary Capital Expenditure	Growth	Sustaining
	MUG LOM Estimate USD M	MUG LOM Estimate USD M
Warehouse upgrade	0.00	0.32
Communications and CCTV	0.00	0.41
HR	0.00	0.14
IT	0.00	0.28
Commercial and Admin	0.00	0.28
Environment	0.00	0.28
HSLP and Mines Rescue	0.00	0.28
Contingency	0.00	0.19
Total	**0.00**	**2.20**

21.4.10 Property Divestment and Rehabilitation

The company currently has USD 25 M of commercial and residential property that it plans to divest over the next 15 years. There is also approximately USD 1.1 M of annual rental income from these properties and payment of rates of USD 0.2 M which will decrease over this time. The project requires the purchase of property above the Rex stopes with subsequent divestment at the end of the project valued at USD 3 M. Rehabilitation and closure costs for the current site has been estimated at USD 42M.

Except for the purchase costs of the Rex property, these costs are not included in the economic assessment as these costs will be incurred whether or not the Martha underground goes ahead. This is a conservative assumption as the deferral of rehabilitation and closure has a favourable impact on the net present value of the site.

It has been estimated that the Martha underground will incur an additional rehabilitation cost of USD1.37M as shown in Table 21-22. These costs are included in the economic assessment.

Table 21-22: Incremental Rehabilitation Costs

Incremental Rehabilitation Summary Capital Expenditure	Growth	Sustaining
	MUG LOM Estimate USD M	MUG LOM Estimate USD M
Cyanided water treatment	0.00	0.25
Non-cyanided water treatment	0.00	0.66
Backfill portals and shafts	0.00	0.09
Remove backfill plant and rehabilitate	0.00	0.14
Contingency 20%	0.00	0.23
Total	**0.00**	**1.37**

21.4.11 Capital Cost Summary

Capital costs include the direct costs for project execution, as well as the indirect costs associated with design, construction and commissioning.

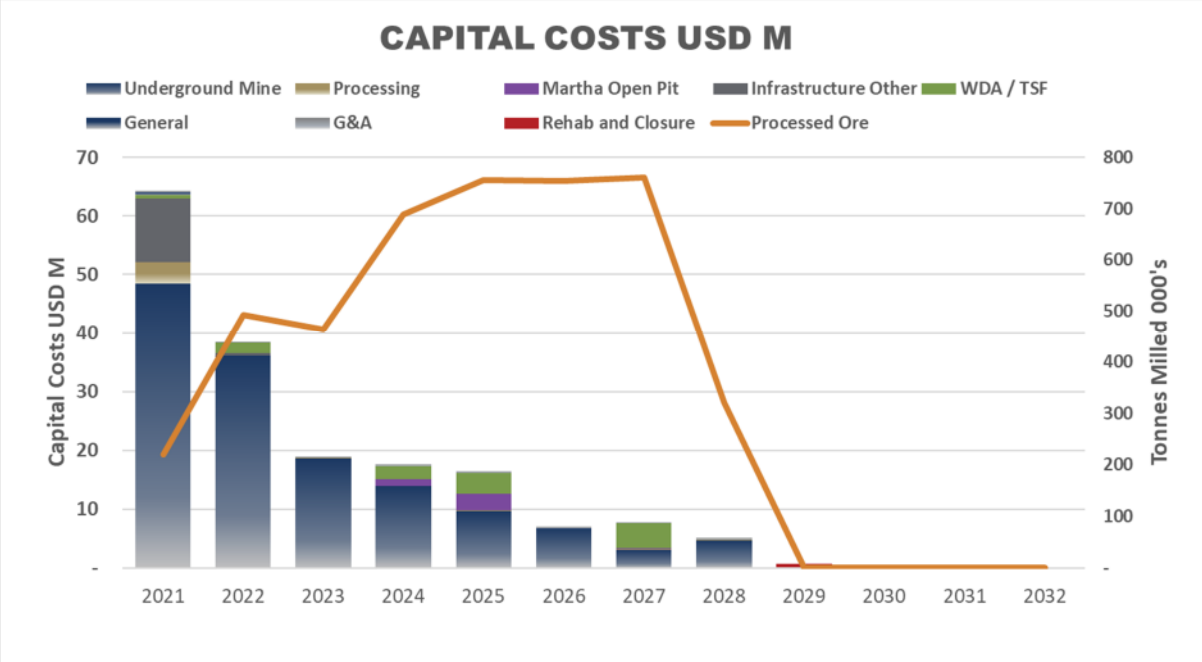
Indirect project capital costs include third-party consultants, construction facilities and services, equipment freight, vendor support, first fill and spares (for the first year of operation). Percentage factors based on OGC's experience with similar projects were used to determine indirect project costs, based on the project direct cost.

The capital costs including sustaining capital is outlined in Table 1-6 and shown by year in Figure 1-7. The range of accuracy for the capital cost estimate is ± 15%.

Table 21-23: Capital Costs Initial and Sustaining

Summary Capital Expenditure Schedule	Growth	Sustaining
	MUG LOM Estimate USD M	MUG LOM Estimate USD M
General and Administration Costs	0.00	2.20
Processing	0.00	4.51
Open Pit Mining Martha Pit	0.00	4.20
Underground Mining Martha	25.50	114.37
TSF Constructions	0.00	12.19
Infrastructure	12.39	0.00
Rehabilitation	0.00	1.37
Total	**37.89**	**138.84**

Figure 21-2: Capital Cost Estimate



21.5 Operating Costs

21.5.1 Basis of Estimate

The operating cost estimate has an expected accuracy range of ±15% and is expressed in Quarter 1 CY 2021 USD. The estimate includes the underground mining, OP mining, processing and G&A operating costs. It excludes costs associated with escalation beyond Quarter 1 CY 2021, currency fluctuations, off-site costs, interest charges and taxes. No contingency has been included in the operating costs.

21.5.2 Mine Operating Costs

Cost models were developed for underground mining which build up costs from first principles using physical inputs as drivers and unit rates sourced from the Waihi site and suppliers.

21.5.3 Underground Mining

The underground mine operating costs were developed from first principles by OGC, based on assumptions of a combination of remnant vein and pillar recovery methods and virgin Avoca mining at MUG. Costs were benchmarked from the previous projects at Waihi for the planned ±2,000 t/d production rate. All costs assume a continuation of owner mining.

The underground mine operating cost estimate includes the costs associated with stope preparation, drilling, blasting, ground support, backfill, underground loading and hauling and material transport to the primary crusher on surface, as well as support and ancillary equipment operations and maintenance, power, direct labour and mine operations supervision staff.

Mine operating costs include the sustaining mobile equipment finance leases and the interest payments on the leases.

Mine staff wages and salaries were included as part of benchmark costs based on the current operation at Waihi. A diesel cost of USD $0.67 /l and power cost of USD 0.078 /kWh was assumed.

The overall average underground mine operating cost was estimated USD 60 /t of ore tonnes mined (which equals the mill feed processed), and the total LOM mine operating cost is USD 271 M, excluding pre-development costs (capitalised mine development costs during the pre-operational period of the process plant).

The underground mine operating cost summary is shown in Table 21-24.

Table 21-24: Underground Operating Mine Cost Summary by Expense

Underground Mining Operating Cost	LOM Estimate USD M	LOM Estimate USD / tonne
Grade Control Drilling	5.76	1.29
Supervision and Control	56.05	12.57
Lateral Development	69.03	15.48
Vertical Development	0.77	0.17
Production	44.86	10.06
Material Handling	25.45	5.71
Backfill	17.84	4.00
Mine Services	53.64	12.03
Finance Lease Interest	3.46	0.78
Total	**276.86**	**62.10**

21.5.4 Surface Mining (Backfill)

The OP mine operating costs to supply backfill were sourced from a number of local contractors with an association with the Martha pit. These costs were benchmarked against past operations at Waihi and assumptions of mining 2.5 m high benches, use of electronic detonators, compliance with strict blast vibration limits and restricted work hours to comply with night time noise limits. OP mining costs include crushing and conveying of all Martha waste rock to either the RoM area as backfill or to the loadout for construction of the proposed TSF lifts. Mine operating costs include the contractor costs, operation of the crusher conveyor system, geotechnical monitoring, engineering support and supervision.

The overall average OP mine operating cost was estimated USD 14.89 /t of material mined. Table 21-25 presents the surface mine operating cost summary and the unit cost per open pit tonne mined.

Table 21-25: Surface Mining Operating Costs

Open Pit Mining Operating Cost	LOM Estimate USD M	LOM Estimate USD / tonne
Contractor Mobilisation / Demob	0.03	0.02
Drilling	1.51	1.06
Blasting	3.34	2.34
Loading & Haul	2.67	1.87
Dayworks	0.00	0.00
RoM Operations	1.66	1.16
Contractors Supervisors / OH's	2.16	1.51

Crusher & Conveyor	1.49	1.04
Owners costs	8.42	5.89
Total	**21.29**	**14.89**

21.5.5 Process Operating Costs

The process operating cost estimate accounts for the operating and maintenance costs associated with the current 0.9 Mtpa process plant operation, water treatment, supporting services infrastructure, and tailings disposal to the various TSF's.

Process plant operating costs were estimated using the following cost categories: power, labour, reagents and consumables, maintenance supplies and services. In general, the process operating cost estimate is based on the following preliminary documentation: conceptual process flowsheet, conceptual mass balance, mechanical equipment list, list of reagents and consumables, and a referential staffing plan.

Power consumption was estimated based on the power requirements by the major and secondary processing plant equipment (excluding stand-by equipment) and adjusted using benchmark factors to account for auxiliary and minor equipment power demand. Assumptions included:

- 95% annual availability.
- A unit power cost of USD 0.078 /kWh.

Reagent consumptions and crushing / grinding consumables were estimated based on the results of metallurgical testwork and previous operating experience at the Waihi plant on Martha ores.

General consumables for the process plant (personnel protective equipment, metallurgical laboratory, chemical laboratories, maintenance, office supplies and others) were estimated from the total consumable and reagent costs.

Labour costs were estimated based on a preliminary staffing plan estimate for the operation and maintenance of the process plant based on OceanaGold's experience at the site. The estimate accounts for management personnel, plant operators and supervisors, as well as WTP operators and maintenance personnel.

Services costs include the following areas: chemical assays, maintenance services by contractors, personnel mobilisation, as well as water and compressed air supply and distribution and other general services.

The chemical assay costs were estimated based on a preliminary testwork program for control of the process plant and unit costs for laboratory tests.

The maintenance services costs associated with the replacement of mill liners and grinding media were estimated based on previous experience at the Waihi process plant. The costs associated with the personnel mobilisation, scheduled maintenance services for plant shutdowns (carried out by contractor companies) and other general services were assumed as the 2% of the total direct capital process plant cost, while the water and compressed air supply and distribution costs were assumed as the 4% of the direct capital cost of these systems. Table 21-26 is a summary of the estimated process plant operating costs as well as the estimated LOM process operating cost is USD134 M or USD30.12 /t of processed mill feed. The range of accuracy for the unit operating cost estimate is ± 15%.

Table 21-26: Processing Operating Costs

Processing Operating Cost	LOM Estimate USD M	LOM Estimate USD / tonne
Power	23.85	5.35
Labour	36.88	8.27
Grinding Media	13.57	3.04
Reagents	34.11	7.65
Crusher Liners	3.76	0.84
Mill Liners	1.79	0.40
Maintenance Materials	15.12	3.39
Lab Assay Costs	4.67	1.05
Tailings Management	0.54	0.12
Total Processing Costs	**134.29**	**30.12**

21.5.6 Infrastructure Operating Costs

General on-site infrastructure operating costs are included in the G&A operating costs.

21.5.7 General and Administrative Operating Costs

The G&A operating cost was estimated at a total LOM of USD 79 M or an equivalent of USD 17.85 /t of Mineral Reserve processed, based on previous costs at the Waihi operation and as outlined in Table 21-27.

Table 21-27: General and Administration Operating Costs

General and Administration Operating Cost	LOM Estimate USD M	LOM Estimate USD / tonne
Labour	29.20	6.55
Consulting fees and audits	14.04	3.15
Contracted services & security	13.21	2.96
Property payments	6.87	1.54
Waihi township,	5.94	1.33
Insurances	5.20	1.17
Donations	2.83	0.64
Mines rescue & PPE & training	2.83	0.63
Total General and Administration Costs	**80.11**	**17.97**

21.5.8 Operating Cost Summary

The cost estimate is +/- 15%. This level of accuracy is attributed to the site operating history over a range of conditions. Table 1-7 summarizes the estimated operating costs and is approximately USD 115 /t for the Mineral Reserve.

Table 21-28: Operating Costs

Summary Operating Expenditure Schedule	LOM Estimate USD M	LOM Estimate USD / tonne
General and Administration Costs	80.11	17.97

Processing	134.29	30.12
Open Pit Mining Martha Pit	21.29	4.78
Underground Mining Martha Underground	276.86	62.10
Other & stockpile	0.00	0.00
Total	**512.54**	**114.97**

The estimation of operating costs by year together with the Mineral Reserve processed are shown in Figure 21-3.

Figure 21-3: Operating Costs for Project

21.6 Unit Costs & Comments

Table 21-29: Operating Unit Costs

Operating Costs NZD		USD / t	USD / oz.
General and Administration Costs	Processed t / oz.	17.97	129.19
Processing	Processed t / oz.	30.12	216.56
Open Pit Mining MOP	Mined t / oz.	4.78	34.33
Underground Mining MUG	Mined t / oz.	87.47	628.83
Capital Costs NZD		**USD / t**	**USD / oz.**
General and Administration Costs	Processed t / oz.	0.49	3.74
Martha Open Pit	Mined t / oz.	0.94	6.78
Martha Underground Fixed and DMD	Mined t / oz.	31.63	227.37
Processing	Processed t / oz.	1.01	7.66
WDA / TSF / Other Infrastructure	Processed t / oz.	5.26	39.84
Rehabilitation	Processed t / oz.	0.31	2.33

The following contingencies were applied to the capital estimate.

- capitalised mine development, additional 2km of development and 10% on development costs for rehabilitation.

- TSF construction – 15% contingency.
- mobile equipment capital costs – 5% contingency.
- underground mining and surface infrastructure costs – 15% contingency.
- process plant sustaining capital 15% contingency.
- rehabilitation and closure work 25% contingency.
- road realignment 30% contingency.

No contingency was applied to the operating costs estimate. The capital cost is for the project is USD 177 M. Operating cost per tonne of Mineral Reserve is USD 115 /t. The consolidated project capital and operating cost is shown below in Table 21-30.

Table 21-30: Consolidated Operating and Capital Costs (USD M) Mineral Reserves

OPERATING COSTS USD M	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	TOTAL
General and Administration Costs	10.3	10.3	10.3	10.3	10.3	10.3	10.3	7.8				80.1
Processing	10.4	15.8	15.3	19.4	20.6	20.6	20.7	11.5				134.3
Open Pit Mining MOP	0.0	0.0	0.0	0.0	3.7	7.1	7.0	3.4				21.3
Underground Mining MUG	15.1	26.2	46.6	54.1	45.7	41.1	30.1	17.9				276.9
Other & stockpile	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0				0.0
Total	35.9	52.3	72.3	83.8	80.4	79.1	68.2	40.6				512.5

CAPITAL COSTS USD M												
General	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0				0.7
Martha Open Pit	0.0	0.0	0.0	1.2	2.9	0.0	0.1	0.0				4.2
Martha Underground Fixed and DMD	48.4	36.1	18.7	13.8	9.6	6.7	3.0	4.7				141.0
Processing	3.8	0.1	0.1	0.1	0.1	0.1	0.2	0.1				4.5
WDA / TSF	0.6	1.7	0.0	2.2	3.5	0.0	4.2	0.0				12.2
General and Administration	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2				1.5
Rehab and Closure	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.7	0.5	0.1	1.4
Total	64.3	38.5	19.0	17.5	16.4	7.0	7.8	5.0	0.7	0.5	0.1	176.7
CAPITAL & OPERATING COSTS USD M												
Total	100.2	90.8	91.2	101.3	96.7	86.0	75.9	45.7	0.7	0.5	0.1	689.3
Cost / oz. Au sold USD /oz.	3,187	1,021	1,608	1,080	1,049	919	832	1,113	-	-	-	1,170

22 ECONOMIC ANALYSIS

22.1.1 Revenue

The Revenue assumptions are based on the assumptions in Section 19 and further summarised below.

- Processing plant production rate of 0.9 Mtpa has been scheduled.
- Gold Price ‐ NZ$1,500 /oz.
- Exchange Rate – 1 NZD = USD: 0.71.
- Metallurgical recovery varies as a function of gold and arsenic grades and by lode but averages 94.9% for Martha underground.
- Royalty payments include higher of 1% of net sale revenue or 5% accounting profit to the Crown plus royalty payable to Osisko.
- Revenue is recognised at the time of production.

22.1.2 Cost

The basis of the Capital and Operating Cost assumptions are described in Chapter 21. Capital and operating costs are well known from the 30 years of operations and have been appropriately applied to develop cut-off grades and inputs into economic analysis.

- The operating cost estimate has an expected accuracy range of ±15% and is expressed in Quarter 1 CY 2021 USD.
- The estimate includes the underground mining, open pit mining (for backfill), processing, G&A operating and rehabilitation costs derived from historical data and forward cost estimates.
- The capital cost estimates have an expected accuracy of ±15% that includes direct and indirect costs and, owner’s costs associated with the mine and process facilities and on-site infrastructure.

The main items in included in the capital cost estimate are listed below, Engineering work, in the range of 20‐30% of total engineering for the project, was carried out to support the estimate. Where possible costs were estimated from similar constructions at Waihi, sourced from OEM or otherwise factored, end-product units and physical dimensions methods were used to estimate costs.

- Underground mine development, mobile equipment fleet, fixed infrastructure and services.
- Tailings Storage Facility, small engineered raises to TSF1A and TSF2.
- On-site infrastructure (water treatment and distribution, electrical substation and distribution, and other general facilities).
- Pit rim works - relocation of a public roads.
- Mine site rehabilitation.
- Project management.

The following have been excluded from the economic analysis:

- Finance costs and interests during construction.
- Costs due to fluctuations in exchange rates.
- Cost of working capital.
- Changes in the design criteria.

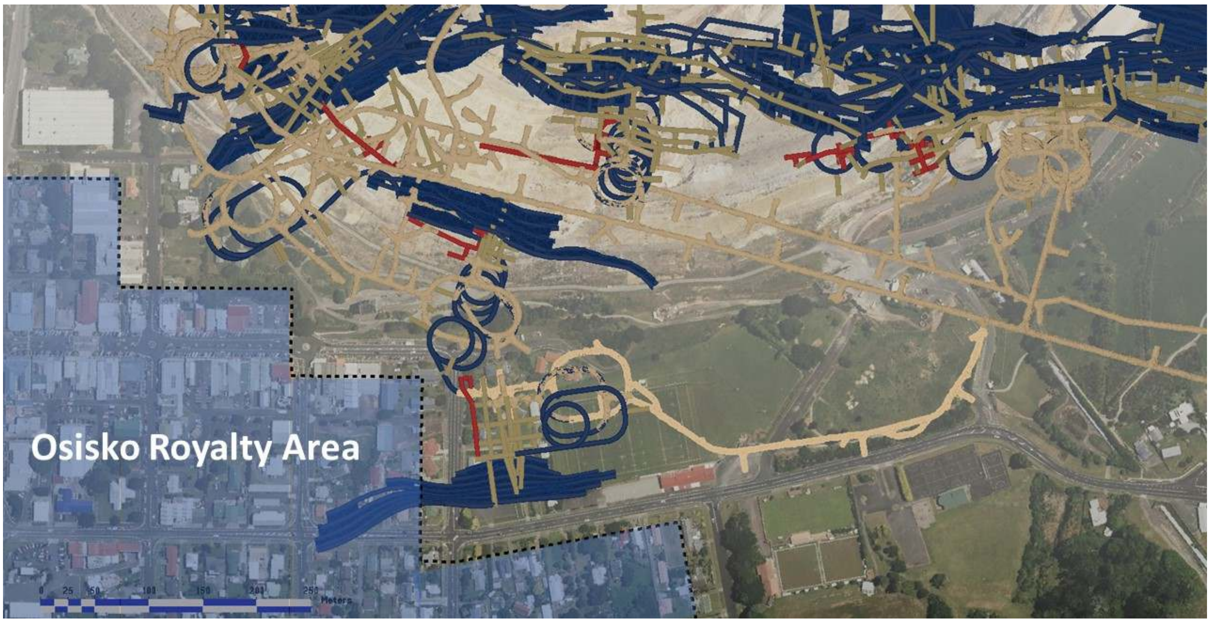
- Changes in scope or accelerated schedule.
- Changes in New Zealand legislation.
- Provisions for force majeure.
- Wrap-up insurance.
- Lack of geotechnical and environmental definitions.

22.2 Taxation and Royalties

OceanaGold hold a 100% interest on the Favona MP 41 808 that includes MOP, MUG and GOP. Royalties payable include the higher of a 1% royalty on net sales revenue from gold and silver or 5% accounting profits is payable to the Crown.

There is a royalty held by a third party Osisko which covers the western end of Rex where it crosses into the Waihi West tenement. The royalty is 2% ad velorum (and is payable over 31,100t at 8.6g/t for 8,600oz. in the Rex mine design for the Reserve case). The royalty boundary is shown below in Figure 22-1.

Figure 22-1: Osisko Royalty Area



The corporate taxation rate included in the analysis is 28%.

22.3 Cash Flow Analysis

Base case assumptions for economic projections include gold price US$1500 /oz and USD 0.71 exchange rate. The key economic results are as presented in Table 22-1, using 1 January 2021 as the reference commencement date. The LOM projections for the free cashflow, are shown in Figure 22-2 and Figure 22-3. The cash flow summary is presented in Table 22-2.

Table 22-1: Key Economic Metrics

Financial Metric	Unit	Reserve Case
Gold Price	$/oz	1500
Exchange Rate	USD:NZD	0.71

Before-Tax		
$NPV_{5\%}$	USD M	143
Internal Rate of Return	%	47
LOM Cumulative Free Cash Flow	USD M	193
$NPV_{5\%}$	USD M	99.4
Internal Rate of Return	%	36
LOM Cumulative Free Cash Flow	USD M	139
Payback Period	years	3.9
Cash Costs C1	USD/oz.	839
AISC	USD/oz.	1107

Figure 22-2: Cash Flow Profile

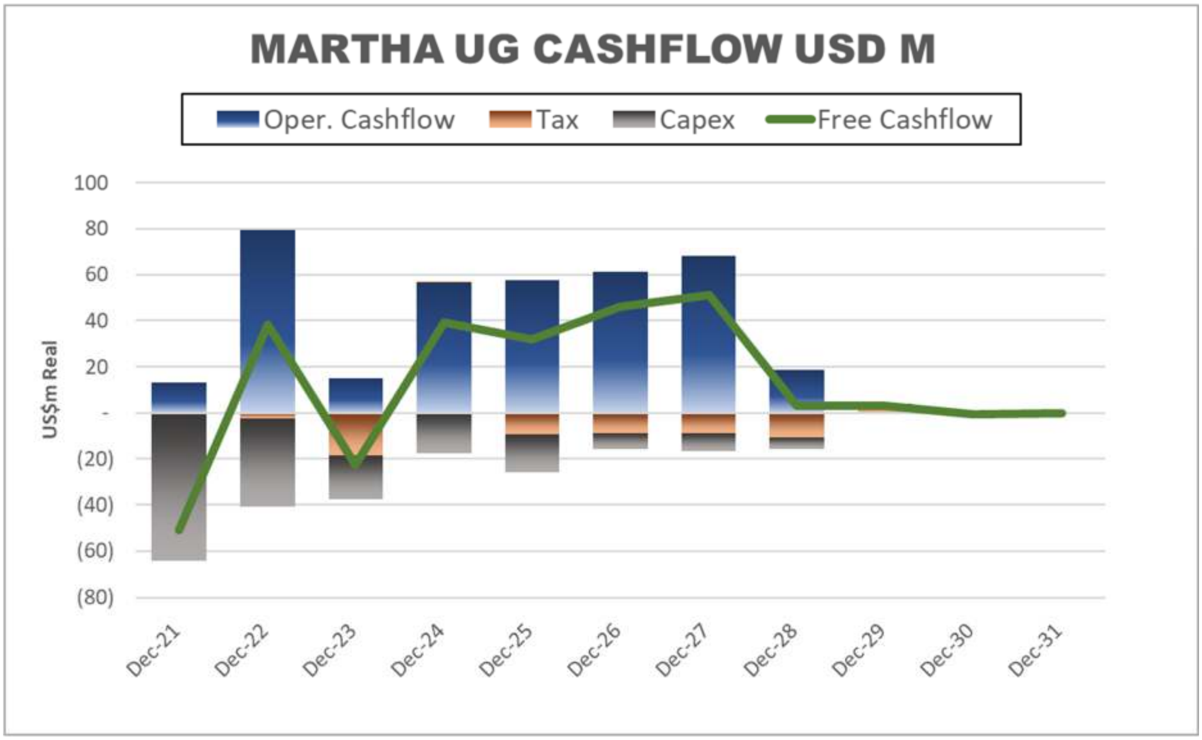
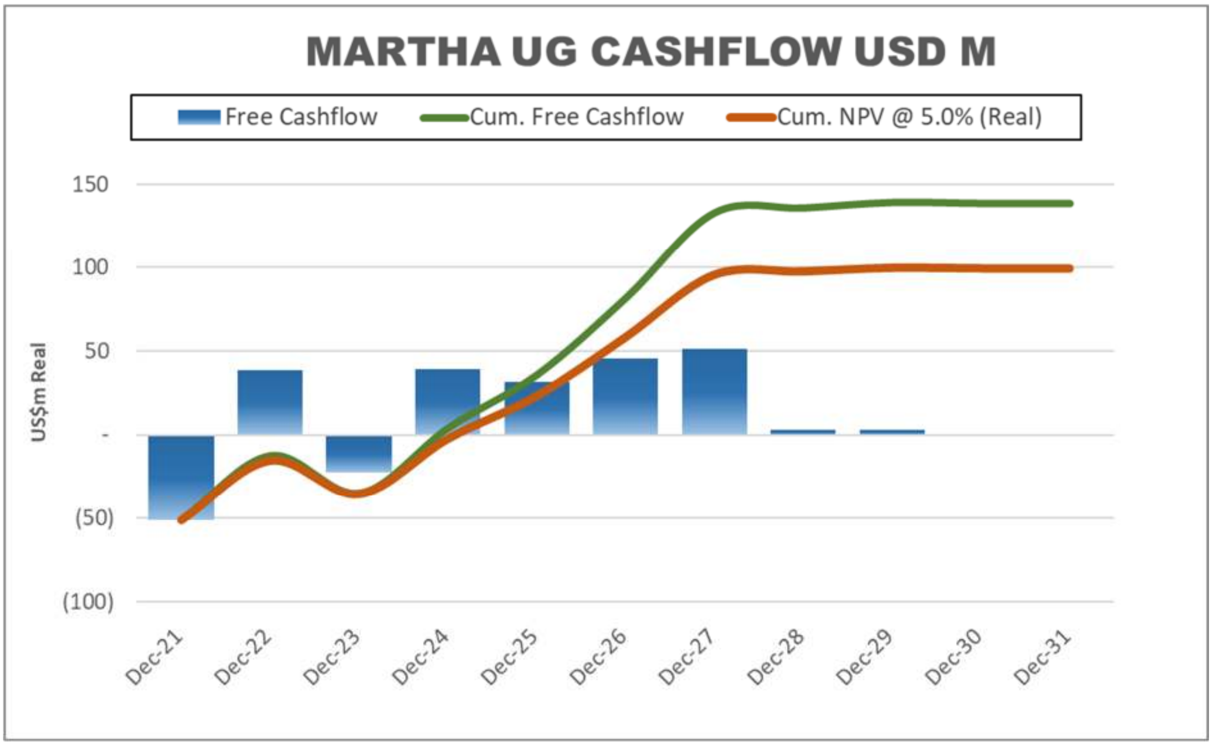
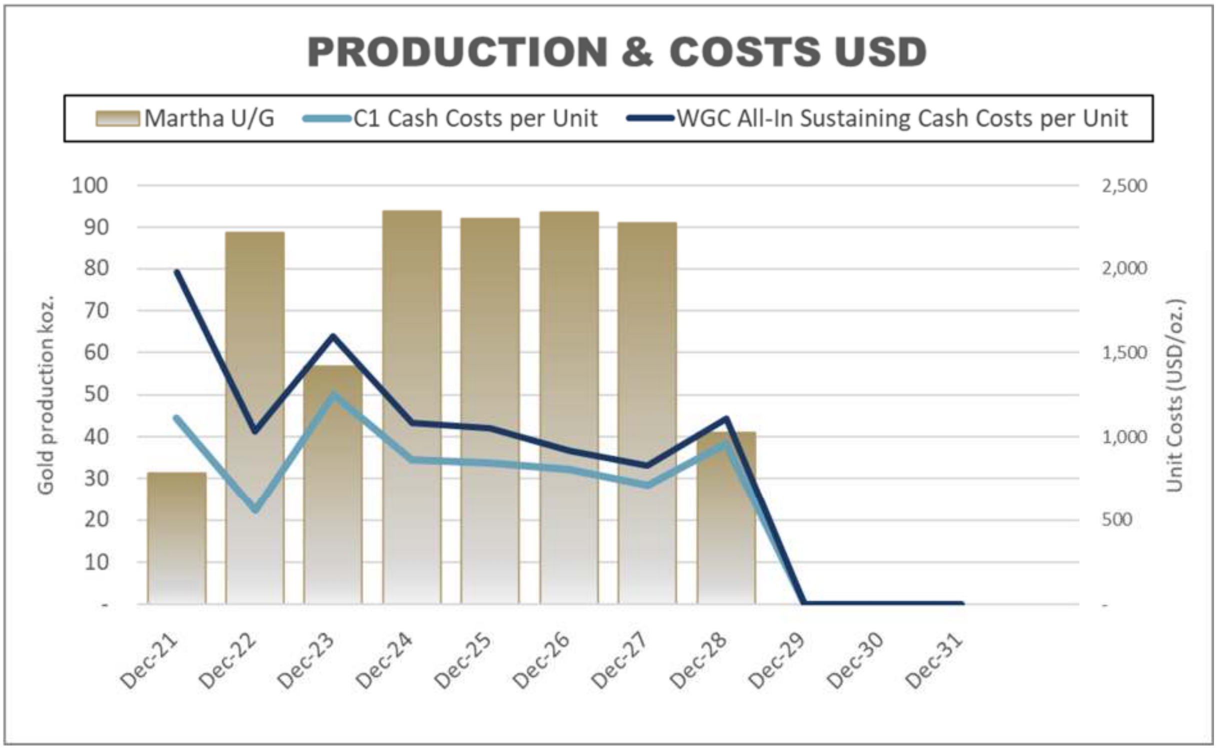


Figure 22-3: Cumulative Cash Flow Profile



The production and cost profile are shown in Figure 22-4 and the cash flow summary in Table 22-2.

Figure 22-4: Production & Costs Profile



Martha underground represents production growth at all-in sustaining costs of USD 1107 per ounce and life of mine cash costs of USD 839 per ounce. These costs are based on mining unit costs that are higher than the historical mining costs. Processing costs are expected to be $30 per tonne processed. There is a sharp decrease in AISC and cash costs as Martha underground ramps-up, and production levels increase significantly.

In the Reserve case scenario with a long-term gold price of USD 1500/oz and discount rate of 5% an after-tax NPV of ~$100 million dollars and after-tax IRR over 36% is indicated. This is a robust project. The cumulative undiscounted free cash flows have been calculated at USD 139M pre-tax dollars while the average annual free cash flow is approximately USD 42 million from 2024 to 2027. The highest year for free cash flow is 2027 at USD 51 million .

Table 22-2: Cash Flow Summary Reserve Case

Cashflow Statement USD 000's	TOTAL	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Sales Receipts												
Gold Dore	900,780	48,157	135,655	86,390	143,163	140,797	143,684	140,157	62,776	-	-	-
Other Revenue	-											
Total Sales Receipts	900,780	48,157	135,655	86,390	143,163	140,797	143,684	140,157	62,776	-	-	-
Operating Costs												
Direct Operating Costs												
Mining	(298,145)	(15,120)	(26,153)	(46,597)	(54,106)	(49,477)	(48,183)	(37,164)	(21,344)	-	-	-
Processing	(134,290)	(10,449)	(15,826)	(15,323)	(19,370)	(20,571)	(20,559)	(20,658)	(11,534)	-	-	-
G&A	(80,108)	(10,337)	(10,337)	(10,337)	(10,337)	(10,337)	(10,337)	(10,337)	(7,752)	-	-	-
Other	-	-	-	-	-	-	-	-	-	-	-	-
Rehab & Closure	-	-	-	-	-	-	-	-	-	-	-	-
Sub Total Direct Operating Costs	(512,543)	(35,906)	(52,316)	(72,256)	(83,813)	(80,385)	(79,079)	(68,158)	(40,631)	-	-	-
Royalty	(18,734)	(591)	(4,043)	(864)	(2,828)	(2,880)	(3,081)	(3,446)	(1,001)	-	-	-
Total Operating Costs	(531,278)	(36,496)	(56,359)	(73,120)	(86,641)	(83,265)	(82,160)	(71,604)	(41,632)	-	-	-
Movement in Balance Sheet Items												
(Inc) / Dec in Debtors, Prepayments	-	(396)	(719)	405	(463)	16	(24)	29	1,152	-	-	-
Inc / (Dec) in Creditors	-	1,967	899	1,093	621	(175)	(72)	(598)	(3,735)	-	-	-
Sub Total	-	1,572	180	1,498	157	(159)	(95)	(569)	(2,583)	-	-	-
Operational Cashflow	**369,503**	**13,232**	**79,477**	**14,768**	**56,679**	**57,373**	**61,429**	**67,983**	**18,561**	**-**	**-**	**-**
Interest Received	-											
Interest Paid	-											
Income Tax Paid	(54,357)	-	(2,562)	(18,444)	102	(9,270)	(8,582)	(8,993)	(10,655)	4,046	-	-
Capital Expenditure	(176,741)	(64,260)	(38,498)	(18,973)	(17,534)	(16,352)	(6,956)	(7,771)	(5,025)	(735)	(520)	(117)
Asset Sales and Other Income	-	-	-	-	-	-	-	-	-	-	-	-
Terminal Value and Salvage Value	-											
Free Cashflow	**138,404**	**(51,027)**	**38,417**	**(22,649)**	**39,247**	**31,751**	**45,891**	**51,219**	**2,881**	**3,311**	**(520)**	**(117)**

22.4 Sensitivity Analysis

Sensitivities are reported for the Mineral Reserve case. The key economic risks were examined by running cash flow sensitivities over:

- Gold price.
- Exchange rate.
- Gold head grade.
- Operating costs.
- Capital costs.

$NPV_{5\%}$ and IRR sensitivity over the Reserve case has been calculated for a range of variations. The after-tax sensitivities are shown in Figure 22-5 to Figure 22-6.

Figure 22-5: NPV Sensitivity Analysis

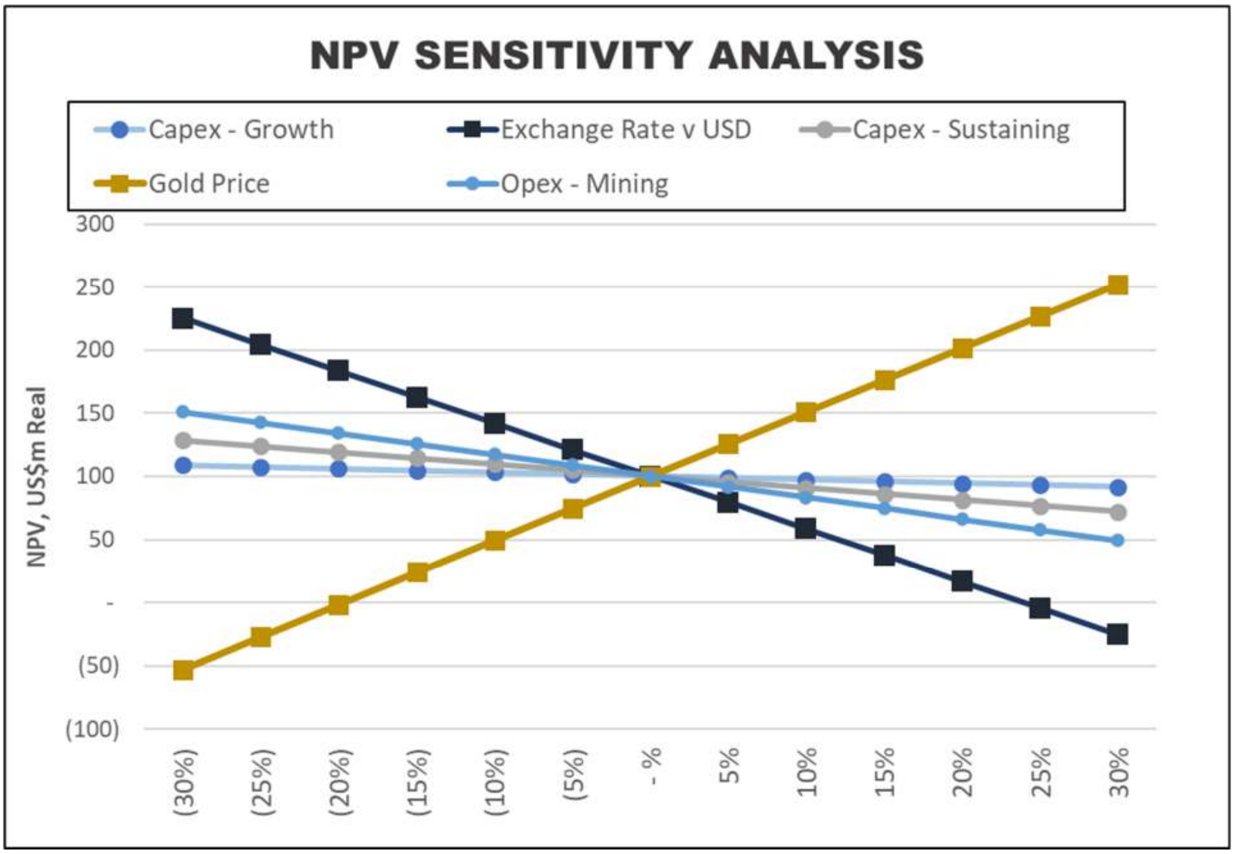
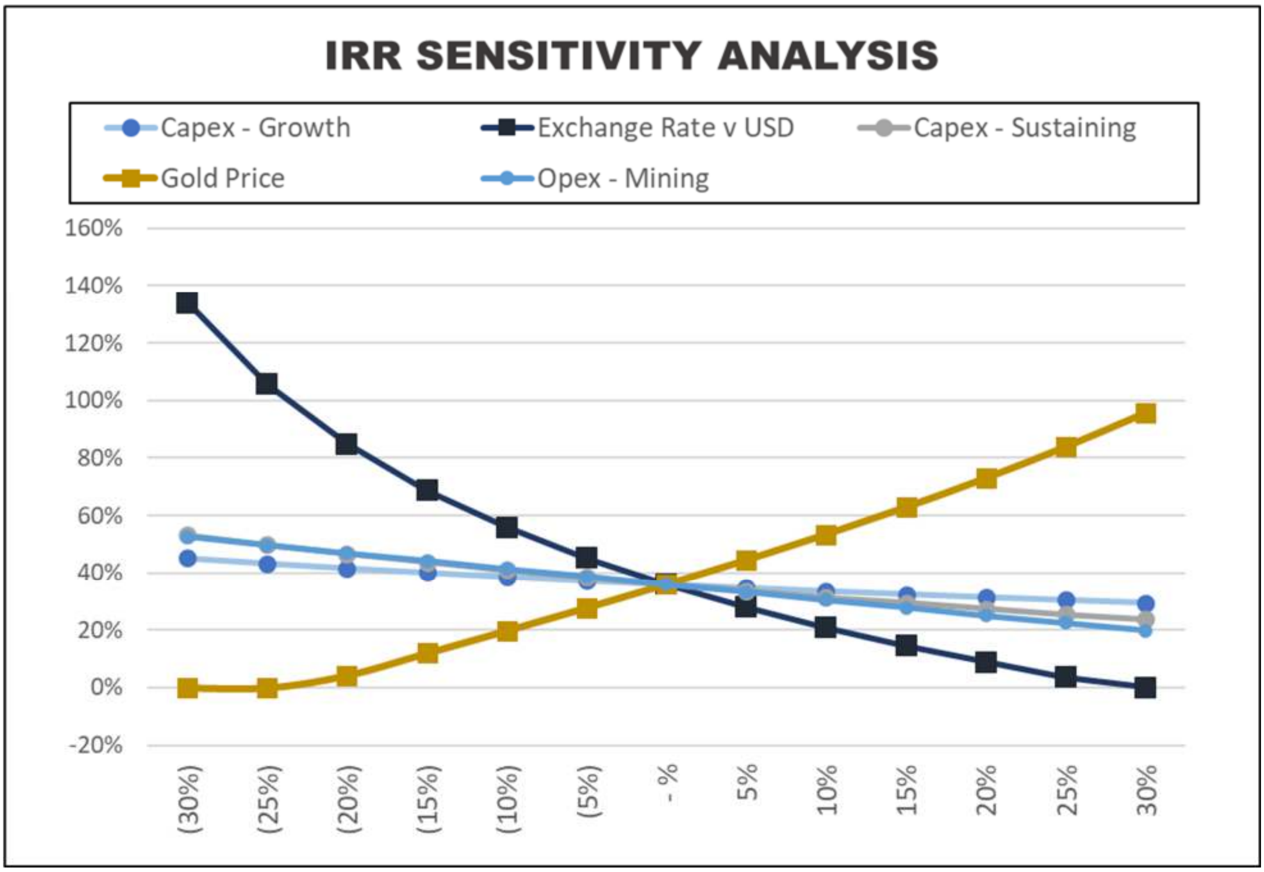


Figure 22-6: IRR Sensitivity Analysis



The most significant drivers of value are gold price and exchange rate as the project is levered to both. The base case is USD 1500 gold with an exchange rate of 1 NZD = USD 0.71 and that with an increase in gold price the value of the MUG significantly increases as it does with a weakening New Zealand dollar. Changes to capital investments do not move the value too much demonstrating the robustness of the deposits.

23 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to this report.

24 OTHER RELEVANT DATA AND INFORMATION

New Zealand has an established framework that is well regulated and monitored by a range of regulatory bodies. OceanaGold has dedicated programs and personnel involved in monitoring consent compliance and works closely with authorities to promptly address additional requests for information. Risks associated with review and renewal of operating consents is, upon that basis, regarded as manageable within the ordinary course of business.

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

Following review of the data available on the Waihi District Study, the QPs have reached the following interpretations and conclusions.

25.1 Mineral Resources & Reserves

- The Mineral Resources and Mineral Reserves have been estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standard Definitions for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM definitions).

25.2 Mineral Tenure, Surface Rights, Royalties, Environment, Social and Permits

- Mining tenure held by OceanaGold in the areas for which Mineral Resources and Mineral Reserves are estimated is valid.
- OceanaGold holds sufficient surface rights to support mining operations over the planned life of mine that was developed based on the Mineral Reserves.
- Permits held by OceanaGold for the Project are sufficient to ensure that mining activities are conducted within the regulatory framework required by New Zealand law;
- Sufficient tailings storage facilities have been planned for.
- OceanaGold has sufficiently addressed the environmental impact of the operation, and subsequent closure and remediation requirements that Mineral Resources and Mineral Reserves can be declared, and that the mine plan is appropriate and achievable. Closure provisions are appropriately considered. Monitoring programs are in place.
- The existing infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods are transported to the mine, and any planned modifications or supporting studies are sufficiently well-established, or the requirements to establish such, are well understood by OceanaGold, and can support the declaration of Mineral Resources and Mineral Reserves and the current mine plan.
- The mine currently holds the appropriate social licenses to operate.
- OceanaGold has developed a communities relations plan to identify and ensure an understanding of the needs of the surrounding communities and to determine appropriate programs for filling those needs. The company monitors socio-economic trends, community perceptions and mining impacts.

25.3 Geology and Mineralisation

- The geological understanding of the setting, lithologies, and structural and alteration controls on mineralization is sufficient to support estimation of Mineral Resources and Mineral Reserves. The geological knowledge of the area is also considered sufficiently acceptable to reliably inform mine planning.
- The mineralization style and setting are well understood and can support declaration of Mineral Resources and Mineral Reserves. The deposit displays classic features that are typical of volcanic-hosted epithermal Au deposits. The QPs consider the model and interpreted deposit genesis to be appropriate to support exploration activities.

25.4 Exploration Drilling and Data Analysis

- Exploration activities since 1986 comprised surface reconnaissance exploration, geological and structural mapping, geochemical sampling, airborne, ground and downhole geophysical surveys, surface and underground drilling, engineering studies and mine development.
- The exploration programs completed to date are appropriate to the style of the deposit and prospects. The research work supports the genetic interpretation of the Waihi vein deposits.
- The majority of surface drilling was by triple tube wireline diamond methods. Surface holes are collared using large-diameter PQ core, both as a means of improving core recovery and to provide greater opportunity to case off and reduce diameter when drilling through broken ground and historic stopes. Drill hole diameter is usually reduced to HQ at the base of the post mineral stratigraphy. RC drilling is mostly confined to the immediate pit vicinity, or isolated first pass exploration drill holes.
- The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration, delineation, underground, and grade control drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation;
- Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation and mine planning purposes;
- The quality of the analytical data is reliable and sample preparation, analysis, and security are performed in accordance with industry standards;

25.5 Metallurgical Testwork

- Metallurgical test work and associated analytical procedures are appropriate to the mineralization type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralization styles found within the Project;
- Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass. As mining progresses deeper and/or new mining zones are identified, additional variability tests will be undertaken as required;
- Mill process recovery factors are based on production data or from ore composite test work and are considered appropriate to support Mineral Resource and Mineral Reserve estimation, and mine planning.

25.6 Mineral Resource and Mineral Reserve Estimates

- Mineral Resources and Mineral Reserves for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM Definition Standards on Mineral Resources and Reserves (2014). The Mineral Reserves are acceptable to support mine planning;
- Reviews of the environmental, permitting, legal, title, taxation, socio-economic, and marketing factors and constraints for the Project support the declaration of Mineral Reserves using the set of assumptions outlined;
- Resource and Reserve estimates have been based on assumptions, which are considered appropriate, for commodity prices, metallurgical recovery, changes to the

geotechnical and hydrogeological parameters used for stope and open pit mine design, dilution and changes to capital and operating costs.

25.7 LOM Plan

- The LOM plan has been developed in a logical and appropriate sequence targeting the high-grade Rex lode, previously unmined Royal west lode and the Edward lode early in the sequence pushing the remnant Empire areas towards the back of the schedule allowing sufficient time for probing and backfilling the historic stopes.
- Access development to each of the five main working areas together with ventilation and services infrastructure is fit for purpose.
- Tonnage and grades presented in the Mineral Reserve include dilution and recovery and are benchmarked to the existing Correnso operation as well as other similar operations. Productivities were generated from first principles with inputs from site engineers, mining contractors, blasting suppliers, and equipment vendors where appropriate and also benchmarked against the Correnso operation.
- Equipment selected is well proven in the mining industry. As well, considerable experience with much of the equipment assumed exists through its employment at the Correnso operation. The establishment of the mine control and Smart centre will facilitate the progressive introduction of automated and battery electric (BEV) equipment and operations, as these become commercially available and operationally proven.
- A monthly / quarterly / yearly production schedule was generated using Deswik software. The schedule targeted a production rate of 1,924 t/d.
- A workable ventilation plan was developed to support the mining fleet that satisfied the NZ Mining Regulations. The capacity of the circuit is expected to be 380 m3/sec.
- The underground support infrastructure is largely in place and new infrastructure relatively straightforward including construction of a new underground service bay, underground magazine and cribroom. In terms of the surface facilities this will comprise small upgrades to the existing changehouse, administration building, an upgrade and addition to the power distribution system and installation of a dry cement plant at the polishing pond stockpile area.
- Key factors in delivering the LOM plan are:
  - achieving the forecast development rates,
  - acquisition of the equipment fleet to undertake required mining activities, and
  - ensuring the water table is lowered at the forecast rates; it is considered that sufficient dewatering capacity is installed to meet the targets and appropriate monitoring is in place.

25.8 Infrastructure

- Sufficient tailings storage facilities have been planned for the Mineral Reserve, involving lifts on existing TSFs. Material to construct the TSF raises is available in the northern and eastern stockpiles and does not require additional material to be mined from the open pit.
- The existing process plant has sufficient capacity for the ore scheduled and the process circuit is well suited to the metallurgy demonstrated over 30 years of processing similar ores.
- Closure provisions and allowances for additional activities have been appropriately considered. Monitoring programs are in place.

- An upgrade of the 33kV power supply to the operation has been allowed for.
- Supporting infrastructure, including buildings, services and communications facilities and transport and delivery means are well established and well understood, and can be supplemented as required to support the mine plan.
- Availability of people with sufficient skills and capabilities is considered sufficient to support the mine plan, in conjunction with recruitment and training programs that will be progressed.

25.9 Operational Readiness Plan

An operational readiness plan is not required for this Project as the project team has already largely transitioned into the operations team and the technical services and operation team already have well developed operating procedures and processes.

25.10 Economic Analysis

- The Mineral Reserves case are presented using underground Mineral Resources. Mineral reserves are declared for the first time.
- Total project capital costs are USD177M, including USD38M initial capital. Total project operating costs, including mining, processing and G&A, are USD 513M, or USD 115 per tonne of ore. All-In Sustaining Cost (AISC) for the project averages USD 834/oz, with life of mine cash cost USD 834/oz.
- The cumulative undiscounted free cash flows have been calculated at USD 140M pre-tax dollars while the average annual free cash flow is approximately USD 42 million from 2024 to 2027. The highest year for free cash flow is 2027 at USD 52 million
- For USD 1500/oz gold price and 5% discount rate, pre-tax NPV is USD 99M and after-tax NPV USD 143M. Pre-tax IRR is 47% and after-tax IRR is 36%.

25.11 Risk

- Key project risks lie in defining the geological resource and grade estimation, project staffing, geotechnical conditions and dewatering rates. Mitigation plans have been developed to address these risks.

25.12 Conclusions

In the opinion of the QPs, Mineral Resources and Mineral Reserves have been appropriately estimated for the Waihi Mine and WKP. Mining and milling operations are performing as expected. This indicates the data supporting the Mineral Resource and Mineral Reserve estimates were appropriately collected, evaluated and estimated.

26 RECOMMENDATIONS

26.1.1 Key Project Recommendations

Based on the conclusions of the technical report and the FS, the key following actions are recommended:

- Continue development of MUG in accordance with the current schedule to bring the mine into production by Q2 2021.
- Continue drilling to improve the Resource classification to support and add future Mineral Reserves.
- Continue drilling to increase Mineral Resources around MUG.
- Continue to investigate opportunities for extraction of the MUG Mineral Resources not included in this FS.
- Continue to evaluate MUG as the other Waihi District Plan projects are further studied.

26.1.2 Underground Mining

- Monitor and adjust as necessary development performance to ensure that rates achieved are in line with feasibility assumptions.
- Review opportunities to bring forward production in 2023 and address the current schedule dip; areas for investigation will include development scheduling and capacity, mine design and dewatering rates.
- Ensure the water table is lowered at the forecast rates, with a focus on adequacy of the monitoring system and performance of installed pumps; sufficient monitoring is to be in place for checking progress and sufficient dewatering capacity appears to be installed to meet the targets.

26.1.3 Ventilation

- On an ongoing basis, review opportunities for optimisation, remote operation and automation of the ventilation system.

26.1.4 Mine Planning

- Update the mine plan from time to time as appropriate, as updated information relating to geological resource, geotechnical conditions, mine dewatering levels and other relevant areas becomes available.

26.1.5 Geotechnical

- With management of voids a key issue for mining of the Martha underground project, continue to update and develop the Void Management Plan and formulate appropriate related procedures as experience in mining at Martha is gained;
- Undertake trial stoping in remnant mining areas, to build experience and identify factors for safe and effective recovery of targeted mineralisation
- Continue geotechnical assessments on an ongoing basis, including in-situ stress measurements, rock property testing and numerical modelling, to provide recommendations for mine design so as to enable updates and improvements from time to time of the mine plan.

26.1.6 Equipment Selection

- Monitor commercial development and availability of proven technologies that would potentially enable the introduction of battery electric (BEV) mine equipment, and facilitate the automation of mining processes.

26.1.7 Infrastructure

- Progress engineering and other arrangements associated with upgrades to the existing electrical system to account for the underground power requirements.

26.1.8 Backfilling Operations

- Undertake further implementation level engineering work to optimise the backfill system which will include production and placement of Cemented Rock Fill (CRF) and Cemented Aggregate Fill (CAF).

26.1.9 Metallurgy

- Undertake further variability and comminution testing as core becomes available to confirm recovery estimates, test assumptions for areas outside the current mine plan and to investigate potential alternative flowsheets that may further increase overall metallurgical performance.

27 REFERENCES

Refer to Appendix A.

APPENDIX A - REFERENCES

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APPENDIX B - TECHNICAL GLOSSARY AND ABBREVIATIONS

Abbreviation	Definition
AAS	Atomic absorption spectroscopy
AECOM	AECOM Pty Ltd
AEP	Amenity Effects Program
AEPEP	Annual Environmental Protection and Enhancement Programs
Ag	silver
AMC	AMC Consultants
Analabs	Analabs Propriety Limited
ANCOLD	Means the Australian National Committee on Large Dams Inc., which is an Australian based non-governments, non-profit association of professional practitioners and corporations with a profession interest in dams. ANCOLD is a member of the International Commission on Large Dams (ICOLD) and publishes international recognised guidelines for the sustainable development and management of dams and water resources.
ASX	Australian Securities Exchange
ATV	Acoustic Televiewer
Au	gold
AuEq	gold equivalent
bcm	bank cubic metre(s)
BFA	bench face angles
Black model	is a computer based representation of a deposit in which geological zones are defined and filled with block which are assigned estimated values of grade and other attributes. The purpose of the block model is to associate grades with the volume model.
BQ	is a reference to the ~ 60 mm diameter drill rods used to recover diamond drill core.
Bulk density	is the dry in-situ tonnage factor used to convert volumes to tonnage.
CIL	carbon in leach
CIM	the Canadian Institute of Mining, Metallurgy and Petroleum

CIM Standards	are the CIM Definitions Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on 27th December 2010, for the reporting of Mineral Resource, Mineral Reserve and Mining Studies used in Canada. The Mineral Resource, Mineral Reserve and Mining Study definitions are incorporated, by reference, into the NI 43-101, and from the basis for the reporting of Reserves and Resources in the Technical Report. With triple listing on the TSX, ASX and NZX. OceanaGold also reports in accordance with the JORC Code and where necessary reconciles its reporting to ensure compliance with both the CIM Standards and the JORC Code.
CIP	carbon in pulp
CMA	Crown Minerals Act 1991
cm	centimetre(s)
CRM	Certified Reference Material
CSR	corporate social responsibility
Cu	copper
cut-off grade	is the lowest grade value that is included in a Mineral Resource Statement, being the lowest grade, or quality of mineralised material that has reasonable prospect for eventual economic extraction.
CVZ	Coromandel Volcanic Zone
DH	drill hole
diamond drilling or DD	is a rotary drilling technique using diamond set or impregnated bits, to cut a solid, continuous core sample of the rock.
DOC	Department of Conservation
E	East
EDA	Exploration Data Analysis
EG	East Graben
EIS	Environmental Impact Assessment
EISS	Means the Environmental Impact Statement System, established under the Mining Act, for classifying projects in terms of their potential impact on the environment. A project that is classified as environmentally critical or located in an environmentally area requires an ECC from the DENR, certifying that the operator will not cause a significant negative environmental impact and has compiled with all the requirements of the EISS.
ELB	Eastern Layback
Entech	Entech Consultants
EOM	end of month
EOY	end of year
EPCM	Engineering, Procurement and Construction Management
EPRMP	Environmental Performance Report and Management Plan

ERA	the Environmental Risk Assessment conducted under the conditions of the ECC
ESE	East South East
ESIA	Environmental and Social Impact Assessment
ETF	the Environmental Trust Find established for the Didipio operation under the conditions of the ECC.
FAR	fresh air rise
Fe	iron
FTD	flow-through drain
FTE	full-time employee(s)
FUFG	flotation & ultra-fine grind
g	gram(s)
G&A	general and administration
GHD	GHD Limited
GOP	Gladstone Open Pit
g/t	grams per metric tonne
GTA	graphite tube atomisation
GWS	GWS Limited Consulting
H&SC	Hellman and Schofield Pty Ltd
ha	hectare(s)
HDC	Hauraki District Council
HDPE	high density polyethylene
Hg	mercury
HQ	is a reference to the ~ 96mm diameter of drill rods used to recover diamond drill core
HR	hydraulic radii
ID2	Inverse Distance weighting to the second power method
ID3	Inverse Distance weighting to the third power method
Indicated Mineral Resource	as defined under the CIM Standards, is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence, sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information, gathered trough appropriate techniques from locations such as outcrops, tranches, pits, workings and drill holes that are spaced close enough for geotechnical and grade continuity to be reasonably assumed.

Inferred Mineral Resource	as defined under the CIM Standards is that part of a Mineral Resource for which a quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
IRA	inter-ramp angles
JK	JKTech Pty Ltd
JORC Code	the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves which become effective 20th December 2012 and mandatory from 1st December 2013. The JORC Code is the accepted reporting standard for the ASX and the NZX.
kg	kilogram(s)
km	kilometre(s)
km^2	square kilometres(s).
koz	thousand troy ounces.
kt	thousand metric tonnes.
kV	kilovolts.
kWh	kilowatt hour(s)
kWh/t	kilowatt-hours per tonne
lb	pound(s)
LG	Lerch Grossman
LHD	load haul dump machines
LHOS	long hole open stoping
LINZ	Land Information New Zealand
LOM	Life of mine
µm	micron or micrometre
m	metre(s)
M	million(s)
m^3	cubic metre(s)
$m^{3/}h$	cubic metres per hour
m/s	metres per second
Ma	million years
Measured Mineral Resources	as defined under the CIM Standards
MEO	Mt Eden Old Cadastral grid
Metso	Metso Technology PTSI Pty Ltd

Mineral Reserve	as defined under the CIM Standards is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justifie4d. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined. The term “Mineral Reserve”, when used in the Technical Report, is consistent with “Ore Reserve” as defined by the JORC Code.
Mineral Resource	as defined under the CIM Standards is a concentration of occurrence of diamonds, natural solid inorganic material or natural solid fossilised organic material including base and precious metals, coal and industrial minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted form specific geological evidence and knowledge. Mineral Resources are subdivided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.
mineralisation	the concentration of minerals in a body of rock
MLR	Multiple Linear Regression
mm	millimetre(s)
MMT	Multipartite Monitoring Team
MOP	Martha Open Pit
MOP4	Martha Phase 4
MOP5	Martha Phase 5
Moz	million troy ounces
MP	Mining Permit
MRF	Mine Rehabilitation Fund
m RL	Reduced Level from mien datum
MSO	Mineable Stope Optimiser software
Mt	million metric tonnes
Mtpa	million tonnes per annum
MUG	Martha Underground
multiple indicator kriging	is a grade estimation technique
MW	megawatt(s)
N	North
NAF	non-acid forming rock
NAPP	negative acid producing potential

NATA	National Association of Testing Authorities, the body which accredits laboratories and inspection bodies within Australia.
NE	Northeast
NI 43-101	National Instrument 43-101 – Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators.
NNE	North East
NPV	net present value
NQ	is a reference to the ~ 76mm diameter drill rods used to recover diamond drill core.
NRS	Northern Rock Stack
NMV	Net Metal Value
NSR	net smelter return
NW	Northwest
NZMG	New Zealand Map Grid
NZPAM	New Zealand Petroleum and Minerals
NZD	New Zealand Dollar
NZDM	New Zealand Dollar Millions
NZTM	New Zealand Transverse Mercator
OceanaGold	means OceanaGold Corporation and/or any of its subsidiaries.
OCEANAGOLD or OGC	means OceanaGold Corporation
OEM	Original Equipment Manufacturer
OHPL	Overhead Power Line
ordinary kriging or OK	is a grade estimation technique
Outotec	Outotec Pty Ltd
oz	troy ounce (31.103477 grams)
PAF	potentially acid forming rock
Pb	lead
PEA	Preliminary Economic Assessment
PIMA	Portable Infrared Mineral Analyser
PMP	Probable Maximum Precipitation storm event
polygonal method	is a grade estimation technique
ppb	parts per billion
ppm	parts per million
PPS	Polishing Ponds Stockpile
PQ	is a diamond tube size equivalent to 85mm inside diameter

Preliminary Feasibility Study	as defined under the CIM Standards is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve. The CIM Standards require the completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.
Probable Mineral Reserve	as defined under the CIM Standards is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. The term “Proven Mineral Reserve”, when used in this Technical Report, is consistent with “Probable Ore Reserve” as defined by the JORC Code.
Proven Mineral Reserve	as defined under the CIM Standards is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. The term “Proven Mineral Reserve”, when used in this Technical Report, is consistent with “Proved Ore Reserve” as defined by the JORC Code.
PSE	Pollution Source Equipment
PSM	PSM Consultants Pty Ltd
pXRF	portable X-ray fluorescence
Q1	Quarter beginning 1 January and ending 31 March
Q2	Quarter beginning 1 April and ending 30 June
Q3	Quarter beginning 1 July and ending 30 September
Q4	Quarter beginning 1 October and ending 31 December
QA/QC	quality assurance / quality control
Qualified Person or QP	as defined under the CIM Standards means and individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation, or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the Technical Report; and is a member or licensee in good standing of a professional association.
PLI	Point Load Index

RAB	rotary air blast
RAR	return air rise
RC	Reverse Circulation drilling
RCF	Rehabilitation Cash Fund
RMA	Resource Management Act 1991
RMI	Rick Management Intercontinental Pty Ltd
ROM	Run-of-mine
RPS	Waikato Regional Policy Statement
RQD	Rock Quality Designation index of rock quality
S	South
SABC	SAG mill / Ball mill / pebble crusher
SAG	semi-autogenous grinding mill
SCSR	self-contained self-rescuer
SE	Southeast
SEDAR	System for Electronic Document Analysis And Retrieval (www.sedar.com)
SG	specific gravity
SGS	SGS Laboratory Waihi
SIA	Social Impact Assessment
SIMP	Social Impact Management Plan
SMU	selective mining unit
SRK	SRK Consulting Pty Ltd
SSC	Southern Stability Cut
SSM	small scale mining or miners
STDEV	standard deviation
SW	Southwest
t	metric tonne (1,000 kilograms)
TCDC	Thames Coromandel District Council
TEM	technical economic model
the District	Waihi District
t/m^3	tonnes per cubic metre
tpa	tonnes per annum
tpd	tonnes per day
tpm	tonnes per month
TSF	Tailings Storage Facility
TSP	total suspended particulate

TSS	total suspended solids
TSX	Toronto Stock Exchange
TVZ	Taupo Volcanic Zone
UCS	Uniaxial Compressive Strength
USD	United States dollars
UTM	Universal Transverse Mercator
UTS	Uniaxial Tensile Strength
W	West
WKP	Wharekirauponga
WRC	Waikato Regional Council
WRD	waste rock dump
wt	weight
WTP	water treatment plant
XRF	x-ray fluorescence
Zn	zinc
3D	three-dimensional
@	at
%	percent
º	degrees
ºC	degrees Celsius

CERTIFICATE OF QUALIFIED PERSON

I, Trevor Maton, MAusIMM CP(Min) do certify that:

1. I am the Chief Metallurgist for OceanaGold New Zealand of 43 Moresby Avenue, Waihi, New Zealand.
2. This certificate applies to the Technical Report titled "Waihi District Study - Martha Underground Feasibility Study NI 43-101 Technical Report" dated March 31, 2021 (the "Technical Report").
3. I graduated with a Bachelor of Science in Mining Engineering (Hons) from the Royal School of Mines, London in 1982 and a Master of Science in Economics from Curtin University, Perth in 2001.
4. I am member and Chartered Professional of the Australian Institute of Mining and Metallurgy (MAusIMM CP Min).
5. I have worked as a mining engineer for 38 years since graduation. My relevant experience includes underground miner, shift supervisor, foreman, planning and design engineer, technical services manager, mine manager, study manager, consulting geotechnical engineer in surface and underground base metal mining, open pit and underground gold mining, open cut and underground coal mining.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a technical association, (as defined in NI43-101) and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of NI43-101.
7. I am based in the Waihi mine site.
8. I am responsible for the preparation of Sections 1 to 5 and 15 to 26 inclusive of the Technical Report.
9. I am not independent of the issuer applying all the tests in section 1.5 of NI43-101 because I am an employee of OceanaGold Management Pty.
10. Prior to my employment with OceanaGold in 2003 I had involvement with the Waihi Project since 2002.
11. I have read NI43-101 and the Technical Report has been prepared in compliance with NI43-101.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of March 2021.

Signature

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Trevor William Maton, MAusIMM CP(Min)

CERTIFICATE OF QUALIFIED PERSON

I, Peter Church, MAusIMM CP(Geo) do certify that:

1. I am the Principal Geologist for OceanaGold Waihi, 43 Moresby Avenue, Waihi, New Zealand.
2. This certificate applies to the Technical Report titled "Waihi District Study - Martha Underground Feasibility Study NI 43-101 Technical Report" dated March 31, 2021 (the "Technical Report").
3. I graduated with a Bachelor of Science in Geology from the University of Otago in 1992.
4. I am member and Chartered Professional of the Australian Institute of Mining and Metallurgy (MAusIMM CP Geo).
5. I have worked as a geologist for 28 years since graduation. My relevant experience includes epithermal gold solver, Archean Gold, mine geology resource estimation and exploration.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a technical association, (as defined in NI43-101) and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of NI43-101.
7. I have been employed by OceanaGold Corporation since 2015 and I have been employed at the Waihi Project for a period of 9 years.
8. I am responsible for the preparation of Sections 6, 7, 8, 9, 10, 11 and 14 of the Technical Report.
9. I am not independent of the issuer applying all the tests in section 1.5 of NI43-101 because I am an employee of OceanaGold (New Zealand) Limited.
10. I have read NI43-101 and the Technical Report has been prepared in compliance with NI43-101.
11. As of the aforementioned Effective Date, to the best of my knowledge, information and belief the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of March 2021.

Signature

Peter Charles Church, MAusIMM CP(Geo)

CERTIFICATE OF QUALIFIED PERSON

I, David Carr, MAusIMM CP(Met) do certify that:

1. I am the Chief Metallurgist for OceanaGold Corporation 99 Melbourne Street, Brisbane, Australia.
2. This certificate applies to the Technical Report titled "Waihi District Study - Martha Underground Feasibility Study NI 43-101 Technical Report" dated March 31, 2021 (the "Technical Report").
3. I graduated with a Bachelor of Engineering in Metallurgical Engineering (Hons) from the University of South Australia in 1993.
4. I am member and Chartered Professional of the Australian Institute of Mining and Metallurgy (MAusIMM CP Met).
5. I have worked as a metallurgist for 27 years since graduation. My relevant experience includes base metal flotation, flotation and leaching of gold ores, pressure oxidation of refractory sulphide ores, ultrafine grinding, process plant design, project evaluation and plant commissioning.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a technical association, (as defined in NI43-101) and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of NI43-101.
7. I last visited the Waihi Project in March 2021.
8. I am responsible for the preparation of Sections 13,17 of the Technical Report.
9. I am not independent of the issuer applying all the tests in section 1.5 of NI43-101 because I am an employee of OceanaGold Management Pty.
10. Prior to my employment with OceanaGold in 2003 I had no involvement with the Waihi Project.
11. I have read NI43-101 and the Technical Report has been prepared in compliance with NI43-101.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of March 2021.

Signature

David Read Carr, MAusIMM CP(Met)