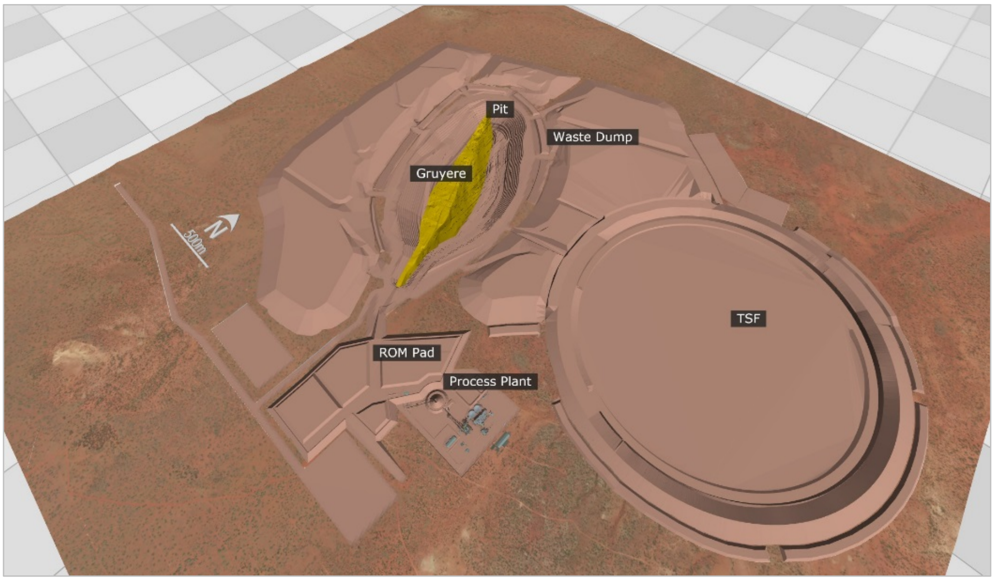
GOLD ROAD RESOURCES
GRUYERE GOLD PROJECT

Located in Western Australia
Feasibility Study Technical Report



ASX Code GOR

ABN 13 109 289 527

COMPANY DIRECTORS

Tim Netscher
Chairman

Ian Murray
Managing Director & CEO

Justin Osborne
Executive Director, Exploration & Growth

Martin Pyle
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Gold Road's Managing Director and CEO, Ian Murray said: *"Following on from the successful announcement of the Gruyere Feasibility Study, Gold Road is pleased to provide a greater degree of detail and information in this Technical Report to investors and shareholders."*

Report Date: 15 November 2016

Effective Date: 19 October 2016

Competent Persons

Mr Justin Osborne, FAusIMM

Mr John Donaldson, MAIG RPGeo (Mining)

Mr David Varcoe, MAusIMM

Mr Bill Gosling, FAusIMM

1000-GE-REP-0003_0

Competent Persons

Mineral Resources

The information in this report that relates to the Mineral Resource Estimation for Gruyere is based on information compiled by Mr Justin Osborne, Executive Director – Exploration and Growth for Gold Road Resources Limited (**Gold Road** or **the Company**) and Mr John Donaldson, Geology Manager for Gold Road.

- **Mr Justin Osborne** is an employee of Gold Road, as well as a shareholder and share option holder, and is a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM 209333)
- **Mr John Donaldson** is an employee of Gold Road as well as a shareholder, and is a Member of the Australian Institute of Geoscientists and a Registered Professional Geoscientist (MAIG RPGeo Mining 10147).

Messrs Osborne and Donaldson have 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 Competent Persons as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Messrs Osborne and Donaldson consent to the inclusion in the report of the matters based on this information in the form and context in which it appears.

Ore Reserves

The information in this report that relates to Ore Reserves is based on information compiled by Mr David Varcoe.

- **Mr David Varcoe** is an employee of AMC Consultants and is a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM).

Mr Varcoe has sufficient experience that is relevant to the style of mineralisation and type of deposits under consideration and to the activity currently being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Varcoe consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

Process Engineering, Design Work and Costing

The information in this announcement that relates to process engineering design work and costing was prepared by GR Engineering Services Limited and was compiled under the guidance of Mr Bill Gosling.

- **Mr Bill Gosling** is an employee of GR Engineering Services Limited and a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM).

Mr Gosling has sufficient experience that is relevant to the style of mineralisation and proposed processing and to the activity currently being undertaken to qualify as a Competent Persons as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Gosling consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

New Information or Data

Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the 22 April 2016 Mineral Resource update, 19 October 2016 Feasibility Study and 7 November 2016 Joint Venture market announcements and that all material assumptions and technical parameters underpinning the market announcements continue to apply and have not materially changed.

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1 SUMMARY

1.1 Purpose of this Report

Gold Road is an Australian incorporated company listed on the Australian Stock Exchange (ASX:GOR). On 19 October 2016, Gold Road publicly disclosed the results of the Gruyere Gold Project Feasibility Study (**FS**)[1] in accordance with the ASX Listing Rules, which wholly incorporates the JORC Code 2012, and in compliance of Australia's statutory continuous disclosure laws as articulated in the Australian Corporations Act (Cth) 2001.

Funding for the project was secured with a 50:50 joint venture agreement between Gold Road and a wholly owned Australian subsidiary of Gold Fields Limited. Details of the agreement were publicly disclosed on 7 November 2016[2] and are not elaborated upon in this report.

The purpose of this report is to provide investors and shareholders with the FS information and results in a NI43-101 compatible format. Since Gold Road is not currently listed on any Canadian exchange, this report is not an official NI43-101[3] report, but is structured in accordance with Form NI43-101F1, taking due consideration for expectations that would be placed on Gold Road under a Canadian listing without compromising Gold Road's current listing obligations. Gold Road notes that the information contained in this report, whilst more expansive, is not materially different to the information already publicly disclosed on the ASX by Gold Road on 19 October 2016. Some statements in this report regarding estimates or future events are forward-looking statements. Gold Road's cautionary statements are presented in Appendix 1.

This report uses the JORC Code 2012 terminology in keeping with Gold Road's listing obligations. These terms are to all intents and purposes compatible with and comparable to the terminology defined by the Canadian Institute of Mining, Metallurgy and Petroleum (**CIM**) Definitions and Standards as required within NI43-101 (see APPENDIX 1 for a direct comparison of key terms). It is worth noting that the NI43-101 accepts the use of "foreign codes", including the JORC Code[4].

A note of departure between the Australian and Canadian requirements is the NI43-101 requirement for an Independent Technical Report, whilst the ASX Listing Rules require disclosure of the full nature of the relationship between Competent Person and the reporting entity, including any issue that could be perceived by investors as a conflict of interest. In addition, Australia's corporations law places the responsibility for disclosure of material information with the board of directors of the listed entity in accordance with the exchange's listing rules[5]. The Board can, however, rely on information and advice provided by others[6].

In presenting this report, Gold Road has relied on contributions of various experts as documented in the FS, the results of which are signed off by Competent Persons, and supported by the outcomes of contributions and technical reviews from a range of independent expert organisations. The Competent Persons, their affiliations and relationships to Gold Road, and the independent expert organisations are outlined and listed in Section 3.

[1] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved"
[2] ASX:GOR Gold Road Resources Public Disclosure, 7 November 2016, "Gruyere Gold Project to be Developed in Joint Venture with Gold Fields LTD"
[3] NI 43-101 Standards of Disclosure for Mineral Reports, Form 43-101F1 Technical Report and Related Consequential Amendments
[4] Ibid, s7.1
[5] Corporations Act (Cth) 2001 s674
[6] Corporations Act (Cth) 2001 s189

1.2 Gruyere Feasibility Study Highlights

Gold Road announced the completion of the FS for the development of its 6.16 million ounce[7] (**Moz**) Gruyere Gold Project (the **Project**), located 200 kilometres east of Laverton in Western Australia on 19 October 2016. The FS was limited to investigating the technical and economic viability of an open pit operation. There is, however, potential for life of mine extensions through transitioning the open pit operation into an underground mine at depth. The FS was also limited to processing ore from the Gruyere deposit only and excluded the potential of processing ores from other satellite deposits (apart from design of the layout of the run-of-mine (**ROM**) pad where consideration has been made for future haulage access).

Key highlights of the FS are:

- The FS confirms Gruyere Gold Project as one of the longest life, lowest cost[8], undeveloped gold deposits in the world
- Updated Ore Reserve of 3.52 million ounces[9], supporting average annual gold production of 270,000 ounces over life-of-mine[10] (**LOM**) of 13 years, elevating Gold Road into the ranks of Australia's mid-tier gold producers
- Gruyere Open Pit averages more than 9,250 reserve ounces per vertical metre to a final depth of 380 metres
- Development to be based on a single large open-pit mine and conventional SAG/Ball Mill Circuit, gravity/carbon-in-leach plant with throughput of 7.5 million tonnes per annum (**Mtpa**) of fresh ore and up to 8.8 Mtpa of oxide ore
- Study findings indicate a technically sound and financially viable Project generating in excess of A$1.2 billion in undiscounted free cash flow (pre-tax, at A$1,500 (US$1,095 at US$0.73:A$1.00) per ounce gold price) over an initial 15-year Project life[11]
- Total forecast capital cost of A$507 million[12,13] (US$370 million[14]) with an additional A$77 million (US$56 million[13]) of sustaining capital over LOM
- Estimated average all-in sustaining cost (**AISC**) of A$945 (US$690[13]) per ounce over LOM with a payback of less than one-third of LOM
- Net Present Value (pre-tax) (**NPV** 8%[15]) of A$486 million (US$355 million[13]) and 24% Internal Rate of Return (pre-tax) (**IRR**) (at A$1,500 (US$1,095 at US$0.73:A$1.00) per ounce gold price)
- NPV 8%[14] increases to A$910 million (US$664 million[13]) with 35% IRR at A$1,750 (US$1,275 at US$0.73:A$1.00) per ounce gold price.

[7] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, "Gruyere Resource Increases to 6.2 Million Ounces"
[8] Australian Gold Miners – Australian equities in a global context – 10 October 2016, Macquarie Equities Research
[9] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved"
[10] Project Life is duration from Construction to end of Processing. LOM is Mine Life duration of Mining and Processing for gold production
[11] Project Life is duration from Construction to end of Processing. LOM is Mine Life duration of Mining and Processing for gold production
[12] Capital cost estimate is as at Q2 2016, and accuracy level is -10% to +15%
[13] Capital cost estimate includes A$43 million (US$31 million) of contingency, and excludes A$7 million escalation to Q4 2018
[14] US$:A$ exchange rate US$0.73:A$1.00
[15] 8% discount rate applied

The FS indicates a technically sound and financially viable Project generating over A$1.2 billion in free pre-tax cash flow (A$0.85 billon in post-tax free cash flow) over the Project life (Table 1-1). The total forecast capital cost is estimated to be A$507 million[16] including a Project contingency of A$43 million. The FS is based on a pit design optimised at A$1,500 per ounce. All base-case financial analyses were completed assuming a A$1,500 per ounce gold price, representing the five-year historic average. Analysis at the more recent spot gold price (A$1,750 per ounce) demonstrates considerable Project upside (Table 1-2 and Figure 1-1).

Table 1-1: Summary of FS Financial Outcomes (all run at A$1,500 per ounce or US$1,095 per ounce at US$0.73:A$1.00)

Measure	Units	FS Outcome A$M	FS Outcome[8] US$M
Gold Produced	koz	3,212	
Gross Revenue	$M	4,817	3,516
Free Cash flow – Pre-Tax	$M	1,222	892
Free Cash flow – Post-Tax	$M	845	617
IRR (Pre-Tax)	%	24.0	
IRR (Post-Tax)	%	19.5	
NPV 8% (Pre-Tax)[1]	$M	486	355
NPV 8% (Post-Tax)[1]	$M	305	223
C1 Cash Costs[2]	$/oz	858	626
C2 Cash Costs[3]	$/oz	1,040	759
C3 Cash Costs[4]	$/oz	1,093	798
AISC[5]	$/oz	945	690
All in Cost (**AIC**)[6]	$/oz	1,103	805
Development Capital Cost[7]	$M	507	370
Development Capital Cost per ounce (Dev. Capex / Gold Produced)	$/oz	158	115
Capital Efficiency (Pre-Tax NPV/Development Capex)		1.0	
Total Project Payback	Months	48	
Payback: LOM	%	33	
Project LOM Costs[9]	$M	3,542	2,586

Notes:
1. *8% discount rate applied*
2. *C1 = Mining + Processing Operating Expenditure + Site General and Administration Expenditure + Transport and Refining Costs.*
3. *C2 = C1 + Depreciation + Amortisation*
4. *C3= C2+ Royalties + Levies + Net Interest Costs*
5. *AISC = C1 + Royalties + Levies + Sustaining Capital + Project related offsite Corporate expenditure*
6. *AIC = AISC + Development Capital Expenditure*
7. *The Development Capital Cost is in Q2 2016 (FS) Real terms. The forecast capital cost including potential escalation of A$7 million to Project completion (Q4 2018) is estimated to be A$514 million*
8. *US$:AS$ exchange rate US$0.73:A$1.00*
9. *Excludes mine site closure costs of A$54 million*

[16] Capital cost estimate is as at Q2 2016, and accuracy level is -10% to +15%. Capital cost estimate includes A$43 million (US$31 million) of contingency, and excludes A$7 million escalation to Q4 2018.

The FS has been evaluated at a A$1,500 per ounce gold price, representing the average price over the last five years. During the period of the FS the Australian dollar gold price traded between a low of A$1,592 to a high of A$1,839[17] per ounce, at an average price of A$1,717 per ounce, with the price above A$1,700 for 65% of the FS period. The Gruyere Project is highly leveraged to the gold price, as identified in Table 1-2 below which displays the potential financial performance at a gold price of A$1,750 per ounce. At this price, the Project generates an additional A$777 million (+63.6%) in pre-tax cash flows while the NPV almost doubles (+87.2%). Figure 1-1 also illustrates the potential uplift in EBITDA generated by a A$1,750 per ounce gold price compared to A$1,500 per ounce over the life of the Project. This price compares favourably with the Company's existing modest hedging position of 50,000 ounces with a forward price of A$1,792 per ounce already secured for the Project.[18]

Completion of the positive FS allows the Company to declare an updated Ore Reserve for Gruyere of 3.52 Moz[19], which supports an average annualised gold production of 270,000 ounces over the LOM. Production at this rate would elevate Gold Road into the ranks of Australia's mid-tier gold producers.

Table 1-2: Summary of FS Key Financial Outcomes and Sensitivities – October 2016

Measure	Units	FS Investment Case (A$1,500/oz)		FS Upside (A$1,750/oz)	
		A$	US$	A$	US$
Free Cash flow – Pre-Tax	$M	1,222	892	1,999	1,459
Free Cash flow – Post-Tax	$M	845	617	1,389	1,014
IRR (Pre-Tax)	%	24		35	
IRR (Post-Tax)	%	19.5		28.5	
NPV 8% (Pre-Tax)	$M	486	355	910	665
NPV 8% (Post-Tax)	$M	305	223	602	440
NPV 5% (Pre-Tax)	$M	692	505	1,217	889
NPV 5% (Post-Tax)	$M	457	334	825	602

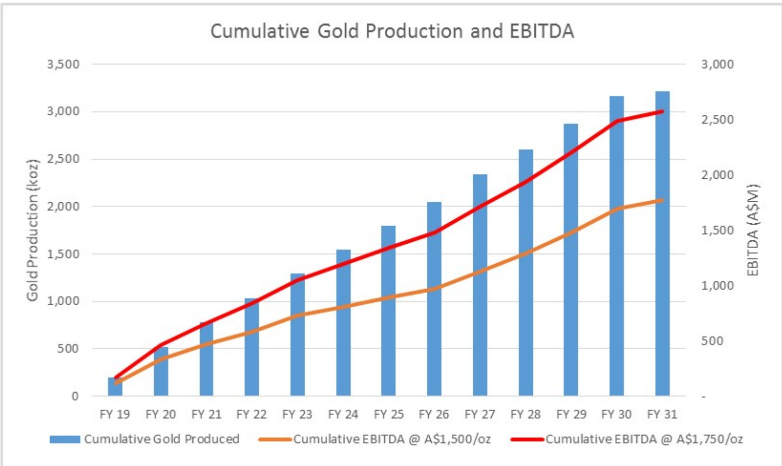

***Figure 1-1:** Annual EBITDA (A$1,500 per ounce and showing increment to A$1,750 per ounce) versus AISC (A$ per ounce)*

Based on the positive FS outcome, the Gold Road Board has approved the FS and recommends progressing the Gruyere Gold Project to the construction phase pending successful completion of financing activities. Gold Road is in the final stages of assessing whether to opt for a combination of debt and equity arrangements or a Joint Venture with a third-party corporation. Project Finance discussions with a number of Australian and International Banking groups commenced in March 2016. The process is now well advanced and the Company is confident of receiving Credit Approved terms supporting a significant debt facility before the end of the year.

[17] A$ gold price as intraday bid asking price from Perth Mint records for the period 8 February to 30 September 2016
[18] ASX:GOR Gold Road Resources Public Disclosure, 1 September 2016, "Gold Road Secures Gold Forward Sales Facility"
[19] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved "

Parallel Joint Venture discussions have also been had with a select number of Australian and International gold mining companies since 2015. These talks are similarly well advanced and provide the Company with a number of potentially viable and attractive funding options.

Given the Company's strong financial position[20], the final financing decisions will be made at a time deemed most appropriate and beneficial to the Gold Road shareholder base.

1.3 Statement of Gruyere Ore Reserves

On the basis of the completed FS Gold Road has updated the Ore Reserve for the Project from the previous Ore Reserve announced on completion of the PFS[21].

An updated Ore Reserves estimate was announced on 19 October 2016[22] for the Project was reported according to the JORC Code 2012 including Table 1 'if not, why not' commentary (Appendix 3).

The Ore Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors. The Proved Ore Reserve estimate is based on Mineral Resource classified as Measured. The Probable Ore Reserve estimate is based on Mineral Resource classified as Indicated. No Inferred Mineral Resources have been included in the Ore Reserve.

Table 1-3 presents a summary of the Ore Reserves on a 100% Project basis at a A$1,500 per ounce gold price (US$1,095 per ounce at US$0.73:A$1.00). Material assumptions underpinning the Gruyere Ore Reserves are presented in Table 1-4.

Table 1-3: Gruyere April 2016 Ore Reserves Statement

Ore Reserve Category	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)
Proved	14.9	1.09	0.52
Probable	76.7	1.22	3.00
Total Ore Reserve	91.6	1.20	3.52

Notes:

- *The Ore Reserve conforms with and uses the JORC Code 2012 definitions, which are directly comparable to the CIM definitions for Proven and Probable Mineral Reserves (see Appendix 2)*
- *The Ore Reserve is evaluated using a gold price of A$1,500 per ounce*
- *The Ore Reserve is evaluated using variable cut off grades: Oxide 0.35 g/t Au, Transitional 0.39 g/t Au and Fresh 0.43 g/t Au*
- *Ore block tonnage dilution averages 3.2%; Ore block gold loss is estimated at 1.4%*
- *All figures are rounded to reflect appropriate levels of confidence*
- *Apparent differences may occur due to rounding*
- *A total of 407 kt at 0.87 g/t Au for 11.4 koz at 0.5 g/t Au cut-off of Inferred Mineral Resource associated with the dispersion blanket Domain is contained within the FS pit design (with the majority located within Stage 2). This oxide material has not been included in the optimisation, the Ore Reserve estimate nor the FS processing schedule and presents potential upside subject to further definition with RC drilling*

The FS on which these Ore Reserves are based was compiled with the assistance of a number of independent, reputable and predominantly Western Australian-based engineering companies as well as other industry experts and qualified Gold Road personnel.

[20] Cash on hand at 30 June 2016 of A$90 million
[21] ASX:GOR Gold Road Resources Public Disclosure, 7 February 2016, "Gruyere Pre-Feasibility confirms long life Gold Mine"
[22] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved"

Table 1-4: Ore Reserves Material Assumptions

Material Assumption	Outcome
Mineral Resources	The Mineral Resource estimate (refer ASX announcement 22 April 2016) for the Gruyere deposit which formed the basis of this Ore Reserve estimate was compiled by the Gold Road Competent Person(s). The estimate is based on 357 Reverse Circulation (RC) holes and 113 diamond holes and associated assay data. The data set, geological interpretation and model was validated using Gold Road's internal processes. An external review was completed by Ian Glacken (Director - Geology at Optiro consultants) who is satisfied that the Mineral Resource estimate has been reported and classified according to the guidelines set out in the JORC Code 2012 and in line with good to best industry practice. The Mineral Resources are reported inclusive of the Ore Reserve.
Mining Method and Assumptions	Gruyere will be mined by open pit mining methods utilising conventional mining equipment. The final pit design is the basis of the Ore Reserve estimate. The selected mining method, design and extraction sequence are tailored to suit orebody characteristics, minimise dilution and ore loss, defer waste movement and capital expenditure, utilise proposed process plant capacity and expedite free cash generation in a safe and environmentally sustainable manner. Mining operating and capital costs were estimated as part of the FS and referenced against contractor budget quotes. The open pit design(s) are based on the recommended geotechnical design parameters and assume dry slopes on the basis of adequate dewatering ahead of mining.
Processing Method and Assumptions	A single stage primary crush, Semi Autogenous Grinding and Ball Milling with Pebble Crushing (SABC) comminution circuit followed by a conventional gravity and carbon in leach (CIL) process is proposed. This process is considered appropriate for the Gruyere ore (which is classified as free-milling) and is commonly used in the Australian and international gold mining industry. Estimated plant gold recovery ranges from 87% to 95% depending on head grade, plant throughput, grind size and ore type. The values are based on significant comminution, extraction, and materials handling test work. No deleterious elements of significance have been determined from metallurgical test work and mineralogy investigations.
Cut-off Grades	Variable economic cut-off grades have been applied in estimating the Ore Reserve. Cut-off grade is calculated in consideration of the following parameters; gold price, operating costs, process recovery, transport and refining costs, general and administrative cost and royalty costs. The Ore Reserve is evaluated using variable cut off grades: Oxide 0.35 g/t Au, Transitional 0.39 g/t Au and Fresh 0.43 g/t Au.
Estimation Methodology	Ordinary Kriging was utilised to estimate the Measured component of the Mineral Resource and Localised Uniform Conditioning was utilised to estimate the Indicated and Inferred components of the Mineral Resource.
Material Modifying factors	Mining dilution and recovery modifying factors were simulated by modelling to a Selective Mining Unit (SMU) which represents the capability of the selected mining fleet. The modelling yielded the following results; mining tonnage dilution factor of 3.2%, mining grade dilution of 4.6% and mining recovery factor of 98.6%. These values reflect the continuity of the orebody with individual ore shape designs hundreds of metres along strike by 20 metres to +50 metres wide.

1.4 Property Description and Ownership

Location and Ownership

The Project is located within the Yamarna Greenstone Belt, approximately 200 kilometres east of Laverton in Western Australia, Australia, at latitude 27° 59′ south and longitude 123° 50′ east on the western fringe of the Great Victorian Desert (**GVD**) (Figure 1-2 and Figure 1-3). The Project can be accessed by road, via the Great Central Road, and by air.

The region is historically under-explored and highly prospective for gold mineralisation. The tenements lie approximately 150 kilometres north of the Tropicana Gold Deposit[23]. The Project is located on Mining Lease M38/1267 which was granted to Gold Road on 5 May 2016 for a period of 21 years (from the date of grant) and is renewable for a further period of 21 years. The mining lease covers an area of 6,845.5 hectare (**ha**) and is wholly within the Yamarna Pastoral Lease. The Yamarna Pastoral Lease covers an area of 149,000 ha and is 100% owned and managed by Gold Road.

Planned infrastructure for the Project will be sited on an additional 14 Miscellaneous Licences which have been granted or are under application. Gold Road also owns the Yamarna Pastoral Lease within which the Gruyere Mining Lease and majority of Project infrastructure will be located. The Pastoral Lease is surrounded by the Cosmo Newberry Aboriginal Reserves (numbers 25051, 22032, 25050 and 20396).

Gold Road also holds an exploration tenement package within the prospective Yamarna Greenstone Belt (Figure 6-1). These exploration tenements totalling approximately 5,000 km^2 are divided into two blocks:

- The northern block of tenements (including ML38/1267) referred to as the North Yamarna Exploration Project, surrounds the Gruyere Gold Project area; and
- The southern block of tenements, totalling approximately 2,900 km^2 constitutes tenements held under a joint venture with Sumitomo Metal Mining Oceania Pty Ltd (**Sumitomo**) (a subsidiary of Sumitomo Metal Mining Co. Limited), is referred to as the South Yamarna Exploration Joint Venture (**SYJV**). Gold Road and Sumitomo each hold 50% ownership of this joint venture.

[23] Tropicana is 70% owned by the global mining company AngloGold Ashanti Ltd

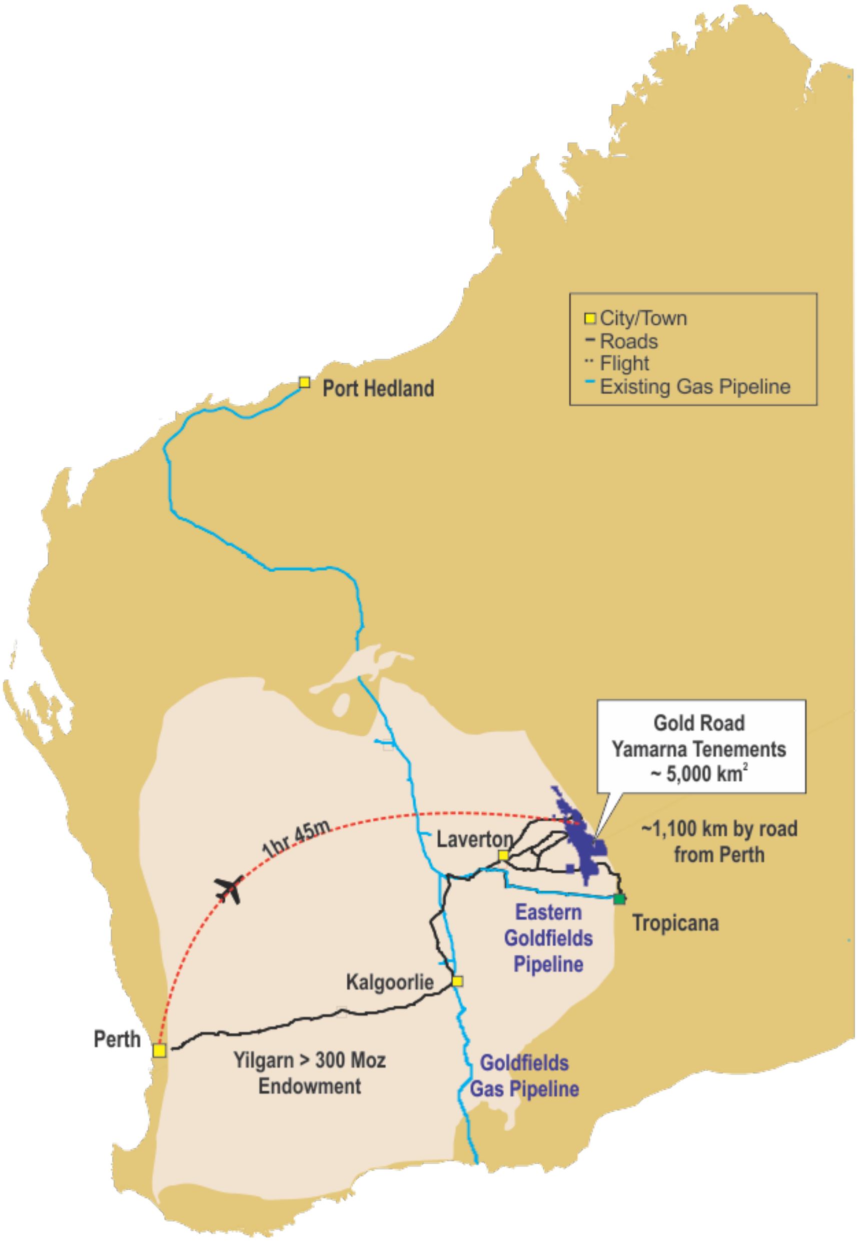

Figure 1-2: *Project Location of Gold Road tenements relative to major cities, towns and relevant infrastructure within the Yilgarn Craton.*

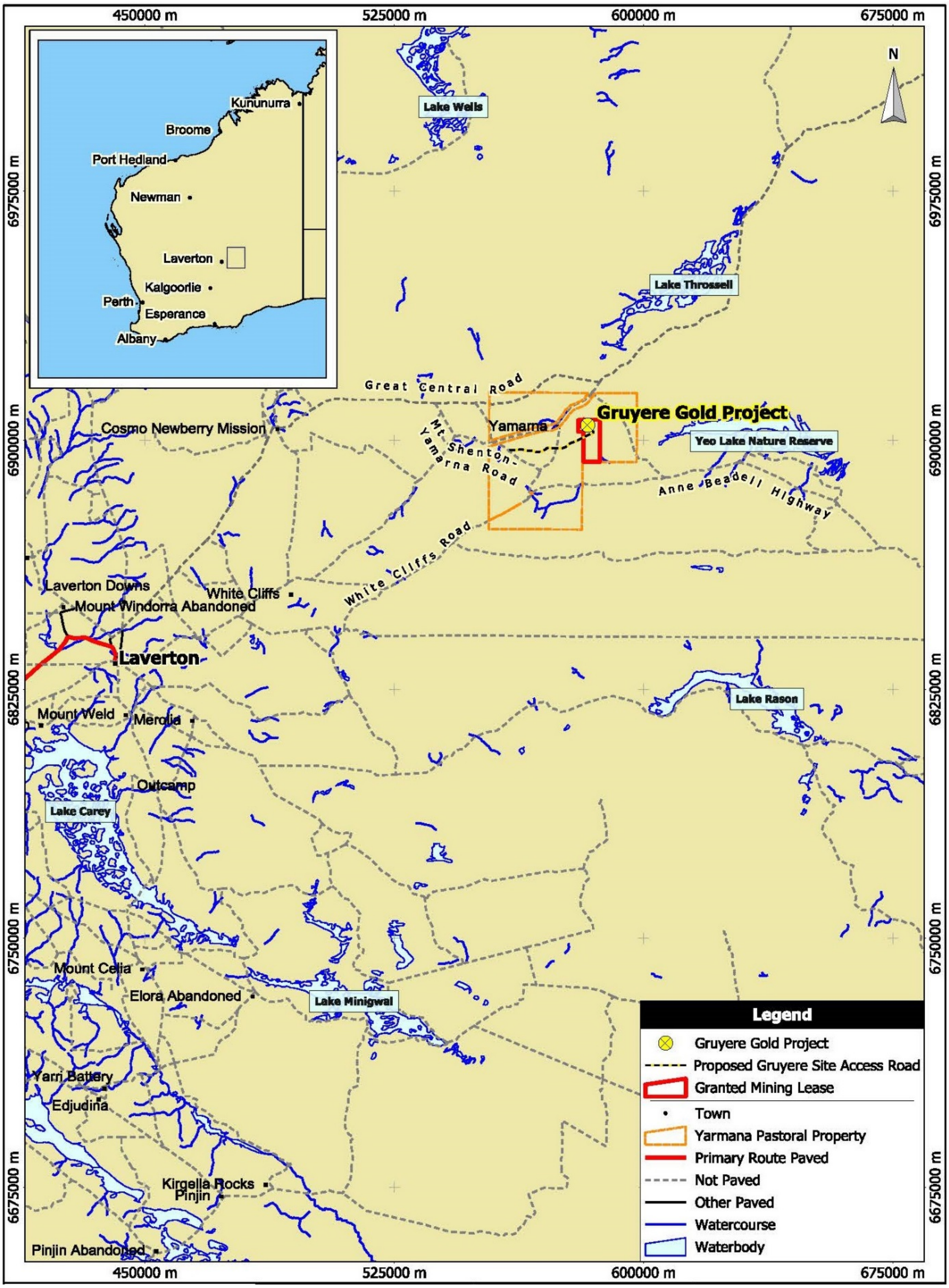

Figure 1-3: *The Project Location*

Mineral Tenure

All tenure required for the Project is subject to the Commonwealth Native Title Act 1993. Gold Road entered into negotiations with the traditional owners of the Project area in 2015. On 3 May 2016 Gold Road reached agreement with the Yilka People and the Cosmo Newberry Aboriginal Corporation (**CNAC**) on the Gruyere-Central Bore Native Title Agreement (**GCBNTA**), which allowed the Department of Mines and Petroleum (**DMP**) to grant the Mining Lease (M38/1267). The GCBNTA covers mining and infrastructure tenements associated with the Gruyere Gold Project, as well as granted mining leases over additional Mineral Resources at Central Bore and Attila-Alaric which do not form a part of the Gruyere Project FS.

The mineral tenements for the Project consist of a granted mining lease and a number miscellaneous licences covering a gas pipeline, water supply pipelines and other Project infrastructure. The main footprint of the planned mining infrastructure within the mining lease covers an area of approximately 2,084 ha. This footprint includes final locations of the open pit, waste rock dumps, the tailings storage facility (**TSF**), mine access roads, processing plant and associated infrastructure.

Outside the mine lease but within the Yamarna Pastoral Lease there are additional footprints for the accommodation village and airstrip locations. In addition, there are mining-related linear infrastructure footprints for the water supply pipelines and the gas pipeline route. These pipeline routes extend outside the Yamarna Pastoral Lease boundary.

Physiography

The Project area lies at the western margin of the Great Victoria Desert of Western Australia. This area consists of a predominantly flat landscape truncated by the Yeo Palaeovalley drainage system that flows towards the Yeo Lake about 60 kilometres east of the Project.

The Project area has varying topography ranging from sand plains and dunes with some regional breakaway areas of indurated, weathered Permian sandstone, rising to small hills up to an elevation of 500 metres AHD or around a maximum of 60 metres above the surrounding landscape.

Average annual rainfall in the Yamarna region is 200 to 230 millimetres and results from both locally generated thunderstorms (October to December) and dissipating tropical cyclones tracking south-east from the coast (January to May). Rain events are infrequent with approximately 30 rain days on average per year. Most of the annual rainfall is received in one or two significant events with some years having close to zero rainfall.

Minor watercourses and drainages at the Project site are ephemeral and dry for the majority of the time. Flows occur periodically following significant rainfall events, particularly during the cyclone season. Flood berms and diversion channels will be constructed at the mine site to control flood events.

The area immediately surrounding the Project has a low population and little established infrastructure.

Access

Road access to the Project site from Laverton is over a distance of approximately 200 kilometres. Access is along the Great Central Road, turning off 153 kilometres from Laverton. The site access road will be 47.7 kilometres in length, comprising 19.2 kilometres on the Mt Shenton-Yamarna Road and 28.5 kilometres on a newly constructed main site access road.

An alternative access route exists directly east from Laverton via White Cliffs Road and the Mt Shenton-Yamarna Road onto the proposed new main site access road however this road is currently in poor condition compared to the Great Central Road route.

Laverton has a commercial airport and is connected to Perth with commercial flights available three days per week. An existing serviceable airstrip is located at Yamarna close to the current Gold Road exploration camp. This airstrip will be used to transport construction personnel until a new airstrip is built at Gruyere in mid-2017. The new sealed airstrip will be built approximately 6 kilometres south-west of the process plant and adjacent to the Project accommodation village; it will be capable of handling 100 seat aircraft.

1.5 Geology and Mineralisation

Geology

The Project and its exploration tenements encompass the Yamarna and Dorothy Hills Greenstone Belts, the eastern most known greenstone belts of the Archaean Yilgarn Craton. The greenstone belts of the Yilgarn Craton are the dominant host for gold mineralisation and mined production in Australia and the Yilgarn Craton is recognised world-wide as a pre-eminent gold district.

The Gruyere deposit is an Archaean orogenic gold deposit. This deposit type is widespread in the greenstone belts of the Yilgarn Craton and in other greenstone belts around the world including in Canada, Africa and India.

The Gruyere deposit is located on a flexure point of the regional scale Dorothy Hills Shear Zone within the Dorothy Hills Greenstone Belt where the shear zone changes from a northerly direction to a north-north-westerly direction. Gold mineralisation is hosted within the steep easterly dipping Gruyere Porphyry, a medium-grained quartz monzonite porphyry that has intruded the country rocks, elongated in the direction of the shear zone.

The cover rocks overlying the Archaean host rocks at Gruyere include Quaternary aeolian sands generally one to three metres thick, with localised sand dunes up to 10 metres in height, and semi-consolidated Permian sandstone which increases in thickness from south to north, attaining a maximum thickness of 30 metres.

The host Gruyere Porphyry averages around 90 metres in horizontal width through the deposit with a maximum width of 190 metres in the centre of the deposit and tapering to around 5 to 10 metres width at the northern and southern extremities.

Mineralisation

The entire Gruyere Porphyry is variably altered and gold grade is related to variations in style and intensity of alteration, structure, veining and sulphide species. Zones containing higher grade gold mineralisation above 1.2 grams per tonne (**g/t**) Au generally have strong albite ± sericite ± chlorite ± biotite alteration and are associated with a sulphide assemblage of pyrrhotite + pyrite ± arsenopyrite, weak to moderate foliation, common micro-fracturing and steeply dipping quartz veining. The total percentage of sulphide minerals is generally in the range 0.5 to 2%.

Below the Permian sandstone cover there is a weathered profile in the Archaean rocks which varies in thickness from 50-90 metres and is divided into an Oxide zone and a Saprock-Transition zone. The Oxide zone contains clay-rich Saprolite rock with complete oxidation of sulphides and leaching and re-mobilisation of gold. A thin gold Dispersion Blanket is interpreted at the base of the Oxide zone; this blanket extends beyond the porphyry contact. The Oxide zone is generally low grade and represents approximately 1% of the total gold mineralisation at Gruyere. The Saprock-Transition zone displays decreasing clay content and decreasing proportion of oxidised sulphide minerals with depth and is gradational into the Fresh (primary) zone.

The boundary between the Oxide and Saprock-Transition zone marks a distinct change in the characteristics of the distribution of gold mineralisation. Above this boundary, gold mineralisation in the Oxide zone exhibits lower grade, higher variance and low continuity whereas below the boundary mineralisation increases in grade and continuity.

The Fresh (primary) zone is hosted entirely in the Gruyere Porphyry and exhibits steep easterly dipping mineralisation. The main northerly strike trend of the mineralisation is interpreted to be parallel to foliation while the steep easterly dip follows the crenulation of the foliation. Mineralisation shows very high continuity in both these orientations.

1.6 Exploration

Exploration by Gold Road commenced in the Yamarna Greenstone Belt in 2006 and initially focussed on the Yamarna Shear Zone on the western side of the greenstone belt. Shear-hosted gold mineralisation was located in an area referred to as the Attila Trend. In 2009 Gold Road located gold mineralisation in an area 3.7 kilometres east of the Attila Trend and subsequently defined a small gold deposit known as Central Bore.

In 2012 Gold Road conducted a detailed 50 metre line-spaced airborne magnetic and radiometric survey over its entire 5,000 km^2 tenement holding. This formed the foundation for a regional targeting program aimed at locating 'world-class gold deposits' in the Yamarna area. The program subsequently identified 10 Camp-scale targets across the Yamarna tenements.

The first Camp-scale target to be tested was the South Dorothy Hills target located approximately 25 kilometres north-east of the Central Bore deposit consisting of priority structural and geochemical targets including Gruyere; initial drilling intersected gold mineralisation over the Gruyere and YAM14 targets. No previous exploration had been conducted on or around the Gruyere deposit prior to Gold Road's discovery.

Gold Road reported a maiden Mineral Resource for the Gruyere gold deposit in August 2014 based on approximately 38,000 metres of resource drilling. By that stage the deposit had been delineated over a strike length of 1,800 metres and to a maximum depth of 500 metres below surface. The deposit remained open at depth. Gold Road completed a Scoping Study in January 2014 which indicated potential for development of a gold mine and justified further evaluation of the deposit.

During 2014 and 2015 total drill meterage increased to 67,665 metres and Gold Road updated its Mineral Resource estimate in September 2015. This new Mineral Resource was used as the basis of a PFS that Gold Road completed in February 2016. The positive outcomes of the PFS led Gold Road to undertake a FS during the remainder of 2016.

1.7 Drilling, Sampling and Assaying

Drilling

The April 2016 Mineral Resource estimate is based on a total of 87,066 metres from 470 drill holes (357 reverse circulation (**RC**) holes for 41,264 metres, 73 holes with RC pre-collars for 14,694 metres RC and 16,506 metres diamond core tail, and 40 full diamond drill holes (**DDH**) for 14,603 metres). The drilling includes 150 close-spaced, grade control equivalent RC holes (14,837 metres) and two DDH (673 metres) completed since the previous Mineral Resource estimate in September 2015.

The deposit extends over a strike length of 2,800 metres of which 1,800 metres is drilled on a 100 metre section spacing to a depth of 600 metres below surface. Drill holes on the 100 metre sections are generally 40 metres apart in the upper 400 metres and approximately 100 metres apart below that. Additional intermediate 50 metre sections have been drilled with at least one to two holes per section over the upper 300 metres. Approximately 75% of the strike length and 100 metres of depth has been drilled to 25 by 25 metres and includes a 100 metre zone drilled to 12.5 by 25 metres spacing in the centre of the deposit. RC drilling dominates in the upper 100 metres with diamond drilling the dominant method below this depth.

The general drill direction of -60° to 270° is approximately perpendicular to the orientation of the main alteration and mineralisation controls and is regarded as a suitable drilling direction to avoid directional bias in the drilling data.

All RC holes were drilled with a 5.25 inch face-sampling bit, with 1 metre samples collected through a cyclone and cone splitter, to form a sample mass of 2 to 4 kilograms. Sample recoveries are recorded as a percentage and no significant sample loss was noted in any part of the drill program. Recovery of the samples was good, generally estimated to be close to 100%, except for some sample loss at the top of the hole.

DDH were drilled at predominantly NQ core size with 40 holes drilled from surface utilising HQ diameter core to the top of fresh rock and 73 holes utilising a component of RC drilling to complete pre-collars through hanging wall waste zones before commencing with NQ core drilling. Sampling of diamond core was based on regular 1 metre intervals or occasional smaller intervals cut to discrete geological contacts.

Logging of RC chips records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples. Logging of diamond drill recorded the same data with the addition of structural information from oriented drill core. All samples are labelled and stored for future reference.

Survey

The majority (97%) of drill hole collar locations have been surveyed using a Differential Geographical Positioning System (**DGPS**) with final collars located to one centimetre accuracy in elevation. Down hole surveys during drilling used an electronic single-shot camera to take dip and azimuth readings at 50 metre intervals, prior to August 2014, and 30 metres interval, post August 2014. Post drilling, holes were surveyed using a north seeking gyroscopic tool.

Gold Road utilises the standard map projection used in Australia which is the Map Grid of Australia (**MGA94**). The Gruyere Project is located in Zone 51 of the Universal Transverse Mercator (**UTM**) grid system. The MGA94 grid is used in conjunction with a local grid (**Gruyere Grid**) which was established with its north-south grid orientation in the same direction as the strike of the Gruyere deposit to assist with geological evaluation.

The Gruyere Grid Northing baseline is set at 340° 00′ 00″ to MGA94 and therefore approximates the strike direction of the deposit. For Australian Height Datum (**AHD**) elevations, 9,000 metres was added to the AHD elevations in the Gruyere Grid to avoid the possibility of negative values in potential underground operations.

An Aerial Lidar and Imagery Survey covering a 2,558 km^2 area including the Gruyere deposit and the Project's main mining infrastructure was completed in January 2016. One metre contours from this survey were used to construct a new topography surface to constrain the resource model.

Sampling and Assaying

Sample preparation for Gold Road's Gruyere drill samples is carried out at the Intertek Genalysis Sample Preparation Facility in Kalgoorlie, Western Australia. Drill samples were oven dried and the whole sample (2 to 4 kilograms) pulverised to 80% passing 75 µm. A sub-sample of approximately 200 grams was retained and a nominal 50 grams was used for gold analysis.

Prepared sample pulps were analysed for gold at the Intertek Genalysis Laboratory (Intertek) in Perth, Western Australia. Samples are analysed for total gold using a 50 gram Fire Assay with ICP-OES (Inductively Coupled Plasma Optical Emission Spectrometry) finish which has a detection limit of 0.005 parts per million (**ppm**) gold. Prior to May 2014 a Fire Assay method with an Atomic Absorption Spectroscopy (**AAS**) finish was used.

All assay information available at 10 February 2016 was used in the grade estimate for the April 2016 Mineral Resource. The resource estimation incorporated 32,293 RC and DDH assays within the resource model.

1.8 Quality Control and Data Verification

Quality Control/Quality Assurance

Gold Road observes standard Quality Control/Quality Assurance (**QA/QC**) protocols for all drilling programs including routine submission of Field Standards (Certified Reference Materials), Blanks, and Field Duplicates. These QA/QC samples are inserted as blind samples within each dispatched drill sample batch. The Gold Road QA/QC protocols have been in place since the initial RC drilling program undertaken in September 2013.

In addition, the contracted laboratory Intertek has its own internal QA/QC protocols. Intertek QA/QC protocols include analysis of Repeats, Laboratory Standards, Checks and Blanks. Intertek also participates in a monthly round-robin inter-laboratory check analysis.

Gold Road arranged for an independent review of QA/QC data for each major drill program and associated resource update completed. Mr David Tullberg (**Tullberg**) of Grassroots Data Services Pty Ltd (**GDS**) reviewed the QA/QC data from the drill hole and assay database used for the maiden Mineral Resource estimate in August 2014. Dr Paul Sauter (**Sauter**), an in-house consultant from Sauter Geological Services Pty Ltd reviewed the QA/QC data from the drill hole and assay database used for the May 2015, September 2015 and April 2016 Mineral Resource estimates.

Overall the reviews indicated that QA/QC results gave acceptable levels of sample accuracy with respect to Field Standard results and precision with respect to Field Duplicate results, with the latter affected somewhat by the presence of coarse gold at Gruyere. There was no significant bias detected in any of the Intertek results or in inter-laboratory Umpire results.

Data Verification

Gold Road has a comprehensive data verification system which validates all stages of the database compilation, including field logs, drill hole survey data and laboratory assay reports.

A formal database audit was carried out by Optiro Pty Ltd (**Optiro**) in July 2014 prior to the reporting of Gold Road's maiden Mineral Resource estimate in August 2014 and prior to Gold Road reporting updates to the Mineral Resource. Optiro is a resource and mining engineering consulting company based in Perth, Western Australia. Optiro was also involved with auditing the Mineral Resource estimation process including all aspects of the data preparation, estimation and modelling process.

Optiro considered the Gruyere database to be of a high standard with respect to data collection, assay quality assurance, geological interpretation, modelling, validation and reporting.

1.9 Mineral Processing and Metallurgical Test Work

Gold Road completed comprehensive ore characterisation and metallurgical test work programs sufficient to establish the optimal processing routes for the ore at Gruyere and estimation of recovery factors. This work was performed on representative samples from all ore type domains within the deposit with the focus on the Fresh (primary) ore type. A total of 50 representative composite samples were generated with an approximate mass of 2,400 kilograms.

Test work samples were classified into ore types based on oxidation zones (Oxide-Saprolite, Saprock, Transition and Fresh), three grade ranges (low <1.0 g/t Au, median 1.0 – 1.4 g/t Au, high >1.4 g/t Au) and from four pit locations (south, central, north and high grade north).

Test work was performed primarily by ALS Metallurgy Limited (**ALS**) in Perth, Western Australia. ALS was responsible for sample preparation, mineralogy, comminution test work, gravity test work, cyanide leaching, including grind size and reagent optimisation, oxygen uptake and viscosity testing, carbon loading kinetics and variability test work. Other specialist companies completed test work on aspects including gravity recoverable gold (**GRG**) test work, materials handling, slurry rheology and tailings thickening testing.

Standard gravity-leach test work demonstrated that 20 to 80% of the gold could be recovered by gravity in the laboratory. A gravity recovery of 35% has been nominated for the process flowsheet. The leach extraction test work showed rapid leaching kinetics and high ultimate leaching extractions with low cyanide and lime consumption for all ore domains.

The ore did not show any preg-robbing characteristics and a Carbon-in-Pulp (**CIP**) circuit would be suitable. A Carbon-in-Leach (**CIL**) circuit was selected because it would be less susceptible to any preg-robbing species that may be treated in the future. However, a pure CIL circuit would limit the gold loading level on the carbon and this would increase the size of acid washing, elution and regeneration circuit significantly. As a compromise a hybrid CIL circuit was selected.

A split Anglo American Research laboratory (**AARL**) elution circuit with separate acid washing and elution columns was selected for carbon elution. The AARL elution circuit with dual columns was chosen for its flexibility. A split circuit was selected to minimise fresh water requirements.

Recovery factors adopted from the test work results for the life of mine (**LOM**) processing model were: Oxide 93.8%, Transition 91.7% and Fresh 90.9%.

Comminution test work indicated that the fresh rock had an average unconfined compressive strength (**UCS**) of 155 Megapascals (**MPa**) (classified as Strong) with an average crusher work index (**CWi**) of 8.3 kWh/t, classified as medium hard. The average abrasion index (**Ai**) was 0.5399 which is classified as highly abrasive. The average rod mill index (**RWi**) of the seven samples was 20.8 kWh/t. This is classified as very hard (> 20 kWh/t) while the average bond ball work index (**BBWi**) was 17.3 kWh/t. This is classified as hard.

1.10 Mineral Resource Estimate

Standards

Mineral Resources for the Project are reported according to the Australasian Code for Reporting Exploration Results, Mineral Resources and Ore Reserves prepared by the Joint Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia, December 2012 (**JORC Code 2012**). The JORC Code 2012 is defined as an 'acceptable foreign code' under NI 43-101.

Resource Estimation Methodology

Gold Road has carried out all resource estimation for the Gruyere deposit in-house, with technical assistance/review by Optiro.

Gold Road produced a combined Ordinary Kriged (**OK**) and recoverable resource estimate for the Gruyere deposit using a 3D block model and a selective mining unit (**SMU**) with dimensions of 5 metres east (across strike) x 12.5 metres north (along strike) x 5 metres RL (vertical). The estimate was achieved using two different estimation techniques dependent on the density of drill data. In areas of close spaced drilling of 12.5 to 25 metres x 25 metres (ultimately classified as a Measured resource), OK was used with a parent block size of 5 metres east x 12.5 metres north x 5 metres RL (the same as the selected SMU). Grade estimation in the areas with drill spacing of 25 to 50 metres x 100 metres or 100 metres x 100 metres (ultimately classified as Indicated and Inferred resources respectively) was carried out using an OK estimate as input to Localised Uniform Conditioning (**LUC**). The initial OK estimation used a parent block size of 25 metres east x 50 metres north x 10 metres RL.

The LUC methodology allows for estimation of SMU-sized blocks from a primary OK grade estimate of larger parent blocks, in this case an SMU of 5 metres east x 12.5 metres north x 5 metres RL (the same size as the OK for the well drilled area). The method provides grade estimates of SMUs from widely spaced data; the estimate is still globally accurate but avoids the inherent smoothing effect on the grade-tonnage curve of a conventional OK model. The LUC method provides an estimate of the grade-tonnage curve expected from a selective mining process at a given SMU size, i.e. normally less tonnes at higher-grade above cut-off than what would be expected from a conventional OK estimate.

A number of factors have been used in combination to derive the Mineral Resource classification of Measured, Indicated and Inferred, with the primary factor being the drill hole spacing. Other factors include the geological continuity, grade continuity, and estimation quality parameters derived from the estimation process.

Validation of the Mineral Resource estimate involved a number of specific checks including detailed comparison of the input data to the output model, to ensure no bias. All validation checks provided acceptable results adding confidence to the quality and validity of the estimation. Optiro completed an independent review of the Mineral Resource estimate.

Mineral Resource Reporting

The maiden estimate for the Project was reported in August 2014[24] and subsequent updates were reported in May 2015[25] and September 2015[26]. The latest Mineral Resource estimate was presented in April 2016[27]. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the estimates in the 22 April 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Project Mineral Resources and the information of the 22 April 2016 announcement.

The April 2016 Mineral Resource estimate was constrained by an optimised pit shell to determine the portion of the total mineralised inventory within the resource model that has a reasonable prospect of eventual economic extraction. The optimisation utilised mining, geotechnical and processing parameters derived from the PFS and a A$1,700 per ounce gold price.

Table 1-5: Gruyere April 2016 Mineral Resource - Tabulation by Resource Category at 0.5 g/t Au cut-off

Resource Category	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)
Measured	13.9	1.18	0.53
Indicated	91.1	1.29	3.79
Measured & Indicated	105.0	1.28	4.31
Inferred	42.7	1.35	1.85

Notes:

- *The Mineral Resource conforms with and uses the JORC Code 2012 definitions*
- *The Mineral Resource is reported using a 0.5 g/t cut-off*
- *The Mineral Resource is constrained within a A$1,700 per ounce optimised pit shell*
- *All figures are rounded to reflect appropriate levels of confidence.*
- *Apparent differences may occur due to rounding*

[24] ASX:GOR Gold Road Resources Public Disclosure, 4 August 2014, "3.84 Million Ounce Gruyere Maiden Gold Mineral Resource"
[25] ASX:GOR Gold Road Resources Public Disclosure, 28 May 2015, "Gruyere Resource Grows to 5.51 Million Ounces Gold"
[26] ASX:GOR Gold Road Resources Public Disclosure, 16 September 2015, "Gruyere Resource Grows to 5.62 Million Ounces Gold"
[27] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, "Gruyere Resource Increases to 6.2 Million Ounces"

1.11 Ore Reserve Estimate

The Ore Reserve for the Project was reported according to the JORC Code 2012. The term 'Ore Reserves' is synonymous[28] with the term 'Mineral Reserves' as used by the Canadian National Instrument NI 43-101 Standards of Disclosure for Mineral Projects (**NI 43-101**). The JORC Code 2012 is defined as an 'acceptable foreign code' under NI 43-101.

Table 1-6 shows a summary of the Ore Reserve, which was presented in a market announcement on 19 October 2016[29]. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Ore Reserves estimates in the 19 October 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Project Mineral Resources and the information of the 19 October 2016 announcement.

The Ore Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors. The Proved Ore Reserve estimate is based on Mineral Resource classified as Measured. The Probable Ore Reserve estimate is based on Mineral Resource classified as Indicated. No Inferred Mineral Resource has been included in the Ore Reserve.

The Ore Reserve was estimated by AMC Consultants Pty Ltd (**AMC**) on behalf of Gold Road in August 2016. A mining model was developed by applying mining ore loss and dilution factors to the resource model. The mining model was subjected to pit optimisation using FS mining and processing costs, processing recoveries, recommended geotechnical slope angles and a gold price of A$1,500 per ounce, in order to determine an optimum pit shell. The selected pit shell was used as a basis for the final open pit design. A mining, schedule, processing schedule, operating cost model and overall Project financial model were developed on the basis of the mineral inventory contained within the open pit design. The mining model within the final pit is reported as the Ore Reserve.

Table 1-6: Gruyere August 2016 Ore Reserves - Tabulation by Reserve Category

Location	Category	Tonnage (Mt)	Grade Au (g/t)	Contained Au (Moz)
Gruyere	Proved	14.9	1.09	0.52
Gruyere	Probable	76.7	1.22	3.00
Total		91.6	1.20	3.52

Notes:

- *The Ore Reserve conforms with and uses the JORC 2012 Code definitions*
- *The Ore Reserve is evaluated using a gold price of A$1,500 per ounce*
- *The Ore Reserve is evaluated using variable cut off grades: Oxide 0.35 g/t Au, Transitional 0.39 g/t Au and Fresh 0.43 g/t Au*
- *Ore block tonnage dilution averages 3.2%, Ore block gold loss is estimated at 1.4%*
- *All figures are rounded to reflect appropriate levels of confidence*
- *Apparent differences may occur due to rounding*

[28] See comparison of CIM's 'Mineral Reserves' and JORC Code's 'Ore Reserves' definitions in APPENDIX 1.
[29] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved "

1.12 Mining

Gold Road engaged AMC to conduct the mining engineering study for the FS. The FS work confirmed the PFS outcomes that the mining will be carried out by open pit contract mining utilising conventional drill and blast, load and haul techniques and ancillary mining equipment provided by the mining contractor. Mining technical services and support will be provided by Gold Road. Consultant Dempers and Seymour was commissioned to undertake the pit slope design for the Project and this work was used by AMC in the preparation of the open pit design.

From the geotechnical assessment the ultimate FS pit and interim cutbacks were designed with an overall average pit slope angle of 50° (varying from 45° to 54°) for the east wall and 48° (varying from 45° to 51°) for the west wall. The pit slopes were designed within the guidelines published by the Department of Mines and Petroleum (**DMP**) with Factor of Safety (**FoS**) greater than 1.2 for the overall pit slopes.

The open pit design process included the design of pit stages and ramp access to the bottom of the pit subject to geotechnical recommendations and mining fleet requirements. The selection of interim pit shells was guided by the objective of maximising cash flows in the initial years of operation with due consideration for practical mining parameters. The pit has been designed to be mined in four stages. Stages 1 and 2 comprise two independent pits, one in the northern end of the deposit and the other in the southern end. Stage 3 will combine the two starter pits and Stage 4 will cut back to the Final Pit Design. Table 1-3 shows the mining inventory within the final pit design.

Table 1-7: Life of Mine Mining Inventory by Mining Stages

Item	Unit	Stage 1	Stage 2	Stage 3	Stage 4	Total
Ore inventory	Mt	18.1	2.7	34.5	36.2	91.6
Contained gold	Moz	0.65	0.16	1.24	1.48	3.52
Grade	g/t	1.11	1.83	1.11	1.27	1.20
Waste inventory	Mt	15.1	13.1	65.3	160.3	253.7
Total inventory	Mt	33.2	15.8	99.8	196.5	345.3
Stripping ratio	W:O	0.8	4.8	1.9	4.4	2.8

Note: *Apparent differences may occur due to rounding*

The mining schedule is structured to optimise cash flows during the initial years of operation (years one to five) in order to minimise the Project payback period and to maximise the Project's debt carrying capacity. The total material movement (**TMM**) per quarter was smoothed to ensure consistent TMM over each quarter (annually). A peak TMM of 7.25 Mt per quarter was set during the first five years of the schedule by testing the lowest TMM that ensured continuous ore supply. When the cutback for Stage 4 commences in year six, it will be necessary to increase the TMM to 11 Mt per quarter to ensure ore supply in later years.

Mining operations will comprise the clearing and stripping of suitable material from all disturbed areas into discrete stockpiles and drilling and blasting of ore and associated internal waste on 5 metre benches, while bulk waste which is outside the ore envelope is blasted on 10 metre benches. Load and Haul will utilise 360 tonne excavators and 180 tonne capacity haul trucks mining on 3 metre high flitches in ore zones and 3 metre to 4 metre high flitches in bulk waste zones. Ore will be direct fed to the crusher or placed on stockpiles for future rehandle as required. Waste dumps will be developed in 10 metre lifts and progressively rehabilitated; raising of the Tailings Storage Facility embankment will be constructed with waste material from the mine as required.

Pit dewatering is expected to be minimal and will be managed by a collection of external dewatering and depressurisation bores and in-pit sumps for use within the mining operation. RC grade control will be provided by a sub-contractor on a predominantly 25 metre x 25 metre pattern and is campaigned during the mine life.

Mining activities will be conducted by a mining contractor with technical and managerial direction provided by Gold Road. The proposed mine operations model will minimise upfront capital expenditure (**Capex**) requirements by Gold Road and access the contractors' specialised open pit mining knowledge, systems and experience lowering operational risk.

1.13 Recovery Methods

The process plant will be a conventional gravity and CIL plant with a throughput capacity of 7.5 Mtpa for fresh ore and up to 8.8 Mtpa of oxide ore (saprolite and saprock) and various blends of ore types producing an average of 265,000 ounces of gold per year based on a nominal head grade of 1.20 g/t. The plant will be designed to operate seven days per week at a nominal treatment rate of 1,100 dry tonnes per hour (**dtph**) on oxide ore, 1,000 dtph on transitional ore and 937 dtph on fresh ore at a grinding circuit utilisation rate of 91.3%. The comminution circuit comprises primary crushing, semi-autogenous grinding (**SAG**) and ball milling with pebble crushing (**SABC**) with a target grind size of 125 µm for estimated gold recovery of 91% to 94%, depending on ore type treated. The plant will have installed capacity and flexibility to grind in the range of 106 µm to 150 µm to provide operational flexibility.

The process plant unit processes are based on proven technology for gold recovery following a processing route of:

- Primary crushing by a gyratory crusher to product size P_{80} of 135 mm
- Grinding in a SABC circuit to a product size P_{80} of 125 µm
- Treatment of a portion of the grinding circuit cyclone underflow by centrifugal gravity concentration, followed by batch intensive leaching of the gravity concentrate and electrowinning of the resulting pregnant solution
- Thickening in a Hi-rate thickener of the grinding circuit cyclone overflow to 50% solids (w/w) prior to treatment in a hybrid CIL circuit
- Acid washing and split AARL elution of the resulting loaded carbon and thermal regeneration of the barren carbon prior to its return to the CIL circuit
- Smelting of cathode sludge from electrowinning to produce a final product of gold doré
- Tailings thickening in a Hi-rate thickener to 60% solids (w/w) prior to disposal of the tailings into the TSF located within an integrated waste landform (**IWL**).

The process plant layout will reflect the sequential nature of the processing operations from ROM ore feed to the facility and tailings disposal of the waste product. Raw and process water will be sourced from two remote borefields within Gold Road's pending tenements and transferred via a system of pipelines and transfer pumps.

1.14 Infrastructure

The three main infrastructure projects are power and water supply and TSF. The power supply is planned to be provided under a Build Own Operate (**BOO**) arrangement for a 40 megawatt (**MW**) gas-fired power station with the fuel supplied by gas pipeline from an existing pipeline near Laverton and a new gas pipeline to Gruyere. The water supply is planned from two water borefields (Yeo and Anne Beadell) which will provide sufficient process and potable water beyond the life of the Project with a contingency borefield to draw on if required.

The TSF will be developed as part an IWL, with a perimeter waste dump surrounding a centrally placed TSF. The circular wall will be constructed from mine waste with internal zone being compacted with approved clayey mine waste and outer zone being bulk waste. The TSF is designed to store 92.4 Mt of tailings, or 61.6 Mm^3.

Other associated infrastructure required for the Project includes:

- Accommodation village
- Sealed airstrip
- Powerlines
- Access and intra-plant roads.

1.15 Environmental Studies, Permitting and Social or Community Impact

Environmental Studies

The Gruyere mining lease granted in May 2016 covers an area of 6,845.5 ha. Over the last three years, Gold Road has commissioned various environmental and cultural heritage surveys within this area and more recently focussed surveys within the main footprint in the northern portion of the tenement relating to the Project. The footprint of approximately 2,084 ha takes into consideration final locations of the open pit, waste rock dumps, the TSF, access roads, process plant and associated infrastructure. Additional surveys were completed during 2016 covering the final mining-related linear infrastructure footprints of the Yeo and Anne Beadell Borefields, water supply and gas pipeline routes, accommodation village and airstrip locations.

Surveys for vertebrate fauna, flora and vegetation, short-range endemic (**SRE**) invertebrate fauna, subterranean fauna and cultural heritage (anthropology) have been completed for the entire Project area. Archaeological surveys will be completed toward the end of 2016.

A work programme for the remainder of 2016 has been developed to complete all remaining environmental baseline studies and archaeological surveys and compile approval documents for submission by Q4 2016 so that assessment is completed by regulators by Q1 2017.

“Adaptive Aquifer Management” of the Yeo Borefield was introduced to mitigate the impact on the stygofauna habitat by extending the potential design length of the Yeo Palaeochannel bores from 65 kilometres to 80 kilometres. The FS design increased access tracks, water pipelines and powerlines by approximately 7 kilometres to 65 kilometres.

Permitting and Approvals

The Project was referred to the Office of the Environmental Protection Authority (**OEPA**) during the FS and received a determination of Assessment on Proponent Information (Category A) (**API-A**) level of assessment under Part IV of the Environmental Protection Act (**EP Act**) (20 June 2016). This level of assessment from the OEPA means that the Project can proceed with the formal environmental applications and assessment without requiring a public environmental review. The approval applications have been submitted in Q4 2016.

The gas pipeline project was referred to the OEPA during the FS and the OEPA determined the level of assessment being “not to be assessed under Part IV of the EP Act (No Appeals)” (18 July 2016). This level of assessment means the OEPA has recommended that DMP manage the environmental approvals of the gas pipeline project.

Gold Road has, to date, obtained the following agreements and approvals:

- The GCBNTA with the Yilka People and the CNAC (3 May 2016).
- Mining Lease M38/1267 granted by the DMP (5 May 2016).
- OEPA determination on the Gruyere Gold Project Referral under Part IV of the Environmental Protection Act 1986 (**EP Act**) (EAG 17), the level of assessment being API-A (20 June 2016).
- OEPA determination on the Gas Pipeline Referral, the level of assessment being "Referral examined, preliminary *investigations and inquiries conducted. Proposal not to be assessed under Part IV of the EP Act (No Appeals)"* (18 July 2016).

Gold Road continues to work closely with all stakeholders to complete all formal environmental assessments and development approvals in accordance with Part IV of the EP Act and the *Mining Act 1978*. Progress on the environmental studies and the required approvals as part of the Native Title and Aboriginal heritage interests in the Project continue, together with conceptual closure planning.

Community

Native Title and Aboriginal heritage aspects within the Project area were addressed by working with the Yilka People resulting in the GCBNTA being signed on 3 May 2016 and the subsequent Mining Lease, M38/1267, being granted on 5 May 2016.

As at 31 August 2016, the final form of the native title determination between the Yilka (the registered native title claim group) and Sullivan/Edwards (an unregistered native title claim group) had not been settled by the Federal Court. Until the final form determination is made by the Federal Court, Gold Road is unable to ascertain the effect of the judgment, if any, on the Company or its Native Title Agreement with the Yilka and any potential impact on the Project.

1.16 Capital and Operating Costs

Project Execution

The Project development and execution will be managed by the Owner's team appropriately resourced to oversee the execution of the design, construction, commissioning and handover to operations. An Operational Readiness (**OR**) Plan, as part of the Whole of Business Framework (**WBF**), has been developed to ensure that Gold Road will have all the systems, standards and procedures in place and an operations team recruited, trained and ready to accept care, custody and control of the Project assets when handed over by the development team.

The Project Execution Schedule is based on a five-month early works programme followed immediately by a 24 month construction and commissioning timeframe with the objective of achieving first gold production by Q4 2018. The Project Execution Strategy is based on Project Finance in place and Project Approval by Q1 2017.

Key milestones for the development and execution of the Project are:

- Q3 2016 - Early commitment on the design of the access roads, borefield tracks, village layout, airstrip, potable water supply and TSF
- Q4 2016 - Conditional award to procure long lead items, namely the primary crusher, SAG and ball mills
- Q4 2016 - Completion of the FS
- Q4 2016 - Early commitment on the design and equipment required on existing (Eastern Goldfields Pipeline) gas pipeline infrastructure
- Q4 2016 - Conditional award on the Engineering, Procurement and Construction (**EPC**), Bulk Earthworks and accommodation village contracts subject to finalisation of financing

- Q4 2016 - Progress the BOO power supply/ gas pipeline contract ready for award on finalisation of financing
- Q1 2017 - Receive Ministerial approval under Part IV of the Environmental Protection Act
- Q1 2017 - Project Finance in place and Project Approval
- Q3 2018 - Complete commissioning of the process plant
- Q4 2018 - First gold production.

Management of Project implementation by the Owner's team on a small number of large contracts.

The Contracting Strategy was developed to support the Project Execution Strategy which is based around an EPC contract model that delivers the design, engineering, construction and commissioning of the process plant and associated infrastructure. The Contracting Strategy also aims to minimise the number of interfaces between contractors on the Project site. The contract model requires an Owner's team to manage the execution of the Project. The total contract packages identified as part of the Project execution strategy are seven major contract packages, listed in Table 1-8, and general packages - approximately 30 consultancies and 20 site services contracts consisting of services required during construction and transitioning to operations.

Table 1-8: Major Contracts and Type

Contract Number	Contract Description	Contract Type
1000-EP-GOR1101	Mine Development and Production	Schedule of Rates
1000-BO-GOR1700	Energy Supply (Power Station and gas pipeline)	Build Own Operate
1000-EP-GOR1100	EPC Process Plant and associated Infrastructure	Fixed Lump Sum
1000-CC-GOR1301	Bulk Earthworks, TSF, access roads and airstrip	Schedule of Rates
1000-DS-GOR1600	Accommodation village supply and construct	Fixed Lump Sum
1000-DS-GOR1601	Communications backbone to site	Fixed Lump Sum
1000-CC-GOR1300	Water bore drilling.	Schedule of Rates

Capital Costs

The total estimated capital cost to design, procure, construct and commission the Project scope consisting of an open pit mine, process plant, non-process infrastructure, Owner's costs, OR and pre-production costs is approximately A$507M with a -10% to +15% level of estimate accuracy. The forecast capital cost expenditure, including potential escalation to Project completion Q4 2018, is estimated to be A$514M. Approximately A$38M is estimated to be directly exposed to foreign exchange variation with exposure predominately to the Euro.

The total estimated life of mine sustaining capital cost for continual mine development, mine rehabilitation, maintenance of process plant, non-process infrastructure and TSF is approximately A$76.7M. A summary of the Project Capital by major area and Sustaining Capital costs are shown in Table 1-9 and Table 1-10 respectively.

Table 1-9: Summary of Total Capital Costs by Major Area

Area	A$M
Direct	
Process Plant & Infrastructure & TSF	178
Infrastructure and Utilities - Site General	79
Mine Development	36
Power Supply and Distribution	20
Site Development and Site Drainage	8
Subtotal Direct	321
Indirect	
Engineering and Contractors	86
Project Owner's team & Pre-production Operations	50
Capital, Operating and Commissioning Spares	7
Subtotal Indirect	143
Contingency	43
Total (Real) Capital Cost	507

Notes:

- *All figures are rounded to reflect appropriate levels of confidence and include Growth Allowances of A$16 m*
- *Apparent differences may occur due to rounding*

Table 1-10: Sustaining Capital - Life of Mine

Calendar Year	LOM Total (A$M)	FY 2019 (A$M)	FY 2020 (A$M)	FY 2021 (A$M)	FY 2022 (A$M)	FY 2023 (A$M)	FY 2024 (A$M)	FY 2025 (A$M)	FY 2026 (A$M)	FY 2027 (A$M)	FY 2028 (A$M)	FY 2029 (A$M)	FY 2030 (A$M)	FY 2031 (A$M)	FY 2032 (A$M)
Mine Development	31.2	2.8	2.2	2.1	1.8	4.2	0.4	3.4	0.4	6.0	1.9	1.2	1.0	1.4	2.4
Plant and Infrastructure	15.9	0.6	0.8	1.6	1.4	2.1	2.7	1.5	1.0	2.7	1.2	0.3	-	-	-
TSF	22.7	-	4.1	4.1	-	4.1	-	4.1	-	4.1	-	-	2.2	-	-
Contingency	7.0	0.3	0.7	0.8	0.3	1.0	0.3	0.9	0.1	1.3	0.3	0.2	0.3	0.1	0.2
Total Sustaining Capital	76.7	3.7	7.8	8.6	3.5	11.5	3.4	9.9	1.5	14.0	3.4	1.7	3.5	1.6	2.7
Total (Cumulative)		3.7	11.5	20.1	23.6	35.1	38.5	48.4	49.9	63.9	67.2	68.9	72.5	74.0	76.7

Notes:

- *All figures are rounded to reflect appropriate levels of confidence*
- *Apparent differences may occur due to rounding*

Operating Costs

The total estimated LOM operating cost for mining, processing, transport and refining and other costs including general and administration, royalties and rehabilitation levy is A$2,958M. Summary of the operating costs is shown in Table 1-11.

Table 1-11: Operating Costs Summary

Item	PFS LOM Cost (A$M)	PFS LOM Cost (A$/oz)	FS LOM Cost (A$M)	FS LOM Cost (A$/oz)
Mining	1,120	384	1,229	383
Processing	1,298	445	1,433	446
Transport and Refining	5	2	5	2
Other Costs[1]	238	82	291	90
Total Opex	2,661	912	2,958	921

Notes:
1. *Other Costs include G&A, royalties and rehabilitation fund levy.*
- *All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.*

All operating costs for the Project have been estimated based on costs prevailing in the Australian minerals industry for Q2 2016. No escalation has been applied as the LOM operating costs are estimated in Real terms consistent with the Financial Model. All costs were estimated to a level of accuracy of -10% to +15%. Rounding errors may occur in the numbers tabulated in this section.

1.17 Economic Analysis

PCF Capital Group (**PCF**) was commissioned to undertake the Project financial modelling for the PFS and FS. The financial model incorporates a start date of August 2016 when commitments to long lead items commence. All Owner's team expenditures relating to studies prior to January 2017 are treated as sunk costs and these include all Project study costs (PFS and FS). Table 1-12 below highlights the key financial inputs and assumptions that were applied in estimating of Project capital costs and financial analysis

Table 1-12: Key Financial Assumptions

Parameter	Units	Assumptions
Gold Price	A$/oz	1,500
Exchange Rate	A$1:US$	0.73
Accumulated Tax Losses	A$	90M*
Corporate Income Tax	%	30
Power Cost (based on gas source)	A$/KWh	0.21
Diesel Price (after rebate)	A$/litre	0.65

Note: ** Estimated Tax Losses as at end of 2016 financial year*

The financial analysis was undertaken using A$1,500 per ounce (five year average historic gold price) and assumes a constant gold price throughout the LOM. Table 1-13 shows the Project financial outcomes in both Australian and US currency.

Table 1-13: Summary of FS Financial Outcomes (all run at A$1,500 or US$1,095 per ounce at US$0.73:A$1.00)

Measure	Units	FS Outcome A$M	FS Outcome[8] US$M
Gold Produced	koz	3,212	-
Gross Revenue	$M	4,817	3,516
Free Cash flow – Pre-Tax	$M	1,222	892
Free Cash flow – Post-Tax	$M	84`5	617
IRR (Pre-Tax)	%	24.0	-
IRR (Post-Tax)	%	19.5	-
NPV (Pre-Tax) [1]	$M	486	355
NPV (Post-Tax) [1]	$M	305	223
C1 Cash Costs [2]	$/oz	858	626
C2 Cash Costs [3]	$/oz	1,040	759
C3 Cash Costs [4]	$/oz	1,093	798
AISC[5]	$/oz	945	690
AIC[6]	$/oz	1,103	805
Development Capital Cost[7]	$M	507	370
Development Capital Cost per ounce (Development Capex/ Gold Produced)	$/oz	158	115
Capital Efficiency (Pre-Tax NPV/ Development Capex)		1.0	-
Payback	Months	48	-
Payback: LOM	%	33	-
Project LOM Costs[9]	$M	3,542	2,586

Notes:

1. *8% Discount rate applied*
2. *C1 = Mining + Processing Operating Expenditure + Site General and Administration Expenditure + Transport and Refining Costs*
3. *C2 = C1 + Depreciation + Amortisation*
4. *C3= C2+ Royalties + Levies + Net Interest Costs*
5. *AISC = C1 + Royalties + Levies + Sustaining Capital + Project related offsite Corporate expenditure*
6. *AIC = AISC + Development Capital Expenditure*
7. *The Development Capital Cost is in Q3 2015 (PFS) and Q2 2016 (FS) Real terms. The forecast capital cost including potential escalation to Project completion (Q4 2018) is estimated to be A$514 million*
8. *US$:A$ exchange rate US$0.73: A$1.00*
9. *Excludes mine site closure costs of $54 million*
10. *Internal Rate of Return (IRR)*
11. *Net Present Value (NPV)*
12. *All in Sustaining Costs (AISC)*
13. *All in Cost (AIC)*

1.18 Conclusions, Risks and Opportunities

FS Conclusions

The FS outcomes indicate a technically sound and financially viable Project that supports the case for Project Financing and development.

The optimum case for the Project is the development of an open pit mine in four stages, with a conventional SABC, CIL process plant and associated infrastructure for throughputs of 7.5 Mtpa for fresh ore and up to 8.8 Mtpa for oxide and transition ores and blends, powered by a gas-fired power station.

Risk

A structured and comprehensive risk assessment and management process was implemented during the FS, in order to characterise and manage the uncertainties of the Project. The purpose of the risk assessment was to identify the critical and significant risks, at the current stage of the Project, to enable a comprehensive mitigation strategy to be developed to reduce or where possible eliminate the impacts of the risks on the Project.

From the risk assessments carried out, no fatal flaws were identified.

Key Project risks during the Feasibility/Commitment phase include potential impact of delays to Project commitment related to tenure, approvals and funding. The key approval is from the Environmental Protection Authority (**EPA**) which could delay site activities. Delays to grant of miscellaneous licences relating to the Project linear infrastructure (i.e. borefields, access roads, gas pipeline etc.) could impact Project approvals and funding which require granted tenure as a pre-condition to submissions. Until Project funding is finalised and the Final Investment Decision is made, funding constraints could impact execution progress.

As at 31 August 2016, the final form of the native title determination between the Yilka People (the registered native title claim group) and Sullivan/Edwards (an unregistered native title claim group) had not been settled by the Federal Court. Until the final form determination is made by the Federal Court, Gold Road is unable to ascertain the effect of the judgment, if any, on the Company or its Native Title Agreement with the Yilka People and any potential impact on the Project.

The key risks during the Construction and Ramp-up phase are potential for increase in capital cost, changing of scope across mine, process plant and associated infrastructure, and potential for construction delays resulting in late commissioning and ramp-up, with direct impact to the Project economics.

In the operational phase, the key risk will be Market-related gold price fluctuations affecting revenue as nearly 100% of Project revenue will be derived from the sale of gold. The gold price will be the single largest variable in assessing Gold Road's ability to service any debt it may have put in place.

Mitigation of Risk

Technical, engineering and infrastructure risks related to the process plant, TSF, water and gas supply are mitigated by the adoption of industry standard design, equipment selection, installation, operation and maintenance strategies.

Permitting risks have been mitigated through the proactive interactions with third parties.

Execution risk has been mitigated through the selection of industry proven suppliers and advanced operational readiness planning ensuring that the key aspects of the execution and operation of the Project have been identified and actions developed to ensure a successful construction period and transition into production. A suitably resourced Owner's team has been recruited and will be supplemented with additional personnel in the execution phase.

Risks to the cost and schedule will be mitigated by implementing a Contracting Strategy that minimises interfaces and contract types that appropriately allocate risk management either to the contractor or Owner. BOO, fixed price, provisional sums and schedule of rates types of contracts will be executed that reflect the level of design and scope definition at the time of award. Overall budget and schedule will be controlled and managed by an experienced Owner's team.

Financial risk exists in the exposure to A$ gold price fluctuations. A program of forward selling gold will be undertaken to mitigate the risk of falling gold prices and to lock in a proportion of sales revenue to fund any Project finance loan repayments.

Opportunities

Opportunities for adding future value will be derived from exploration, resource and reserve upgrades as well as further value engineering on the mining and process plant during the design and engineering phase.

Significant upside to the Gruyere development business case is possible with the discovery of other economic resources as a result of the ongoing regional exploration work on Gold Road's Yamarna tenements. There is potential for further capital cost reduction following gap analysis, engineering design optimisation and through the negotiation of fewer and larger contract packages. The current Project schedule is based on advanced procurement, allowing early procurement of long lead items and thus taking these items off the critical path. Early commitment for engineering will create an opportunity to improve the schedule.

Future volatility in the price of gold provides an opportunity to achieve superior financial returns from the Project during periods of higher gold price above the FS gold price of A$1,500 per ounce, and could impact positively on the Project mine life either as an open cut or an underground operation.

2 INTRODUCTION

2.1 Terms of Reference

The purpose of this report is to provide investors and shareholders with the FS information and results in a NI43-101 compatible format. Since Gold Road is not currently listed on any Canadian exchanges, this report is not an official NI43-101 report, but is structured in accordance with Form NI43-101F1, taking due consideration for expectations that would be placed on Gold Road under a Canadian listing without compromising Gold Road's current listing obligations. Gold Road notes that the information contained in this report, whilst more expansive, is not materially different to the information already publicly disclosed on the ASX by Gold Road on 19 October 2016.

This report uses the JORC Code 2012 terminology in keeping with Gold Road's listing obligations. These terms are to all intents and purposes compatible with and comparable to the terminology defined by the CIM Definitions and Standards as required within NI43-101 (see Appendix 4 for a direct comparison of key terms). It is worth noting that the NI43-101 is accepts the use of "foreign codes", including the JORC Code.

2.2 Details of Inspection

Justin Osborne is one of the Competent Persons and is Gold Road's Executive Director. – Exploration and Growth. He conducts regular site visits and is responsible for all aspects of the Project.

John Donaldson is the second Competent Person and is Gold Road's Geology Manager. He conducts regular specific site visits to focus on understanding the geology as it is revealed in the drilling data. Communication with the site geologists is key to ensuring the latest geological interpretations are incorporated into the resource models.

Both Competent Persons contribute to the continuous improvement of sampling and logging practices and procedures.

2.3 Sources of Information

Sources of information for this document are the Gold Road Feasibility Study and Appendices, and Gold Road's public announcements as presented to the ASX. References are listed in Section 27.

2.4 Effective Date

The effective date of this report is 19 October 2016

2.5 Units and Currency

All measurement units in this report are metric units except for contained gold metal which is expressed as Troy ounces (**oz**). All monetary amounts expressed in this report are in Australian dollars (**A$**) unless otherwise stated.

3 RELIANCE ON OTHER EXPERTS

The FS was compiled under direction of with the assistance of a number of independent, reputable and predominantly Western Australian-based engineering companies as well as other industry experts and qualified Gold Road personnel.

3.1 Competent Persons – Mineral Resources

The information in this report that relates to the Mineral Resource Estimation for Gruyere is based on information compiled by Mr Justin Osborne, Executive Director – Exploration and Growth for Gold Road and Mr John Donaldson, Geology Manager for Gold Road.

- **Mr Justin Osborne** is an employee of Gold Road, as well as a shareholder and share option holder, and is a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM 209333).
- **Mr John Donaldson** is an employee of Gold Road as well as a shareholder, and is a Member of the Australian Institute of Geoscientists and a Registered Professional Geoscientist (MAIG RPGeo Mining 10147).

Messrs Osborne and Donaldson have 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 Competent Persons as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Messrs Osborne and Donaldson consent to the inclusion in the report of the matters based on this information in the form and context in which it appears.

3.2 Competent Persons – Ore Reserves

The information in this report that relates to Ore Reserves is based on information compiled by Mr David Varcoe.

- **Mr David Varcoe** is an employee of AMC Consultants and is a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM).

Mr Varcoe has sufficient experience that is relevant to the style of mineralisation and type of deposits under consideration and to the activity currently being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Varcoe consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

3.3 Competent Persons – Process Engineering Design Work and Costing

The information in this announcement that relates to process engineering design work and costing was prepared by GR Engineering Services Limited and was compiled under the guidance of Mr Bill Gosling.

- **Mr Bill Gosling** is an employee of GR Engineering Services Limited and a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM)

Mr Gosling has sufficient experience that is relevant to the style of mineralisation and proposed processing and to the activity currently being undertaken to qualify as a Competent Persons as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Gosling consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

3.4 Reliance on Independent Experts

In addition to the team of Competent Persons outlined above, the Gruyere FS relied on the review, participation and technical input from a range of experts who are independent of Gold Road (Table 3-1).

Table 3-1: Gruyere FS Independent Experts Contributions

Data and Mineral Resource	
Review of QAQC and Database Integrity	Optiro Grassroots Data Services Pty Ltd Sauter Geological Services Pty Ltd
Review of Geological Interpretation	Optiro
Review of Mineral Resource	Optiro
Ore Reserves	
Mine planning and optimisation, Ore Reserve Statement and peer review of mine geotechnical engineering	AMC
Geotechnical engineering	Dempers and Seymour Pty Ltd
Review of Mining Study	Orelogy Group Pty Ltd
Review of capital cost estimates	Axiom Project Services (**Axiom**)
Environmental surveys and preparation of the environmental approval documents	MBS Environmental Pty Ltd (**MBS**)
Process plant, associated infrastructure	GR Engineering Services Limited (**GRES**)
Metallurgical test work	ALS Laboratories
Hydrogeology	Pennington Scott
Tailings Storage Facility	Coffey Mining
Gravity test work	Gekko Systems
Infrastructure	
Materials handling test work	Jenike & Johanson
Water bore drilling	Aquatech
Gruyere airstrip design	Aerodrome Management Services (**AMS**) Pty Ltd
Access road design	Shawmac Pty Ltd
Power supply	Wayne Trumble
Finance and Economics	
Cost and Schedule Risk Analysis	Broadleaf Capital International Pty Ltd
Assistance with Operational Readiness	KPMG
Financial modelling	PCF Capital

4 PROPERTY DESCRIPTION AND LOCATIONS

4.1 Location

The Gruyere Gold Project is situated approximately 160 km north-east of Laverton in Western Australia, Australia, at latitude 27° 59’ S and longitude 123° 50’ E. The Project area is located on the western fringe of the Great Victorian Desert (**GVD**) within the Yamarna Pastoral Lease which covers an area of 149,000 ha and is 100% owned and managed by Gold Road (Figure 1-2 and Figure 1-3).

4.2 Mineral Tenure

Project

The mineral tenements for the Project consist of a granted mining lease and a number miscellaneous licences covering the gas pipeline, the water supply pipelines and other Project infrastructure. The mineral tenements are listed in Table 4-1.

The granted Mining Lease for the Project which covers an area of 6,845.5 ha is wholly within the Yamarna Pastoral Lease. The Yamarna Pastoral Lease is surrounded on the northern, western and southern boundaries by the Cosmo Newberry Aboriginal Reserves (numbers 25051, 22032, 25050 and 20396 respectively) (Figure 4-1). The Yeo Lake Nature Reserve is to the east of the Yamarna Pastoral Lease and covers an area of approximately 320,000 ha.

Table 4-1: Tenement Details - Gruyere Project as at 31 October 2016

Tenement	Expiry Date	Grant Date	Area
ML38/1267	04/09/2037	5/05/2016	6,845.5 ha
L38/180	20/09/2032	21/09/2011	4,422.0 ha
L38/210	15/05/2034	16/05/2013	59,562.3 ha
L38/211	15/05/2034	16/05/2013	24,801.8 ha
L38/233	16/06/2036	17/06/2015	28,041.2 ha
L38/235	14/07/2037	15/07/2016	42,821.4 ha
L38/237	05/10/2036	06/10/2015	4,299.3 ha
L38/250		Application*	13,090.0 ha
L38/251	02/10/2037	03/10/2016	789.9 ha
L38/252		Application*	3,695.0 ha
L38/253		Application*	788.5 ha
L38/254	26/09/2037	27/09/2016	568.8 ha
L38/255	26/09/2037	27/09/2016	482.8 ha
L38/256	02/10/2037	03/10/2016	190.2 ha
L38/259		Application*	296.5 ha
L38/260		Application*	664.2 ha

Note: ** Tenement applications submitted awaiting grant*

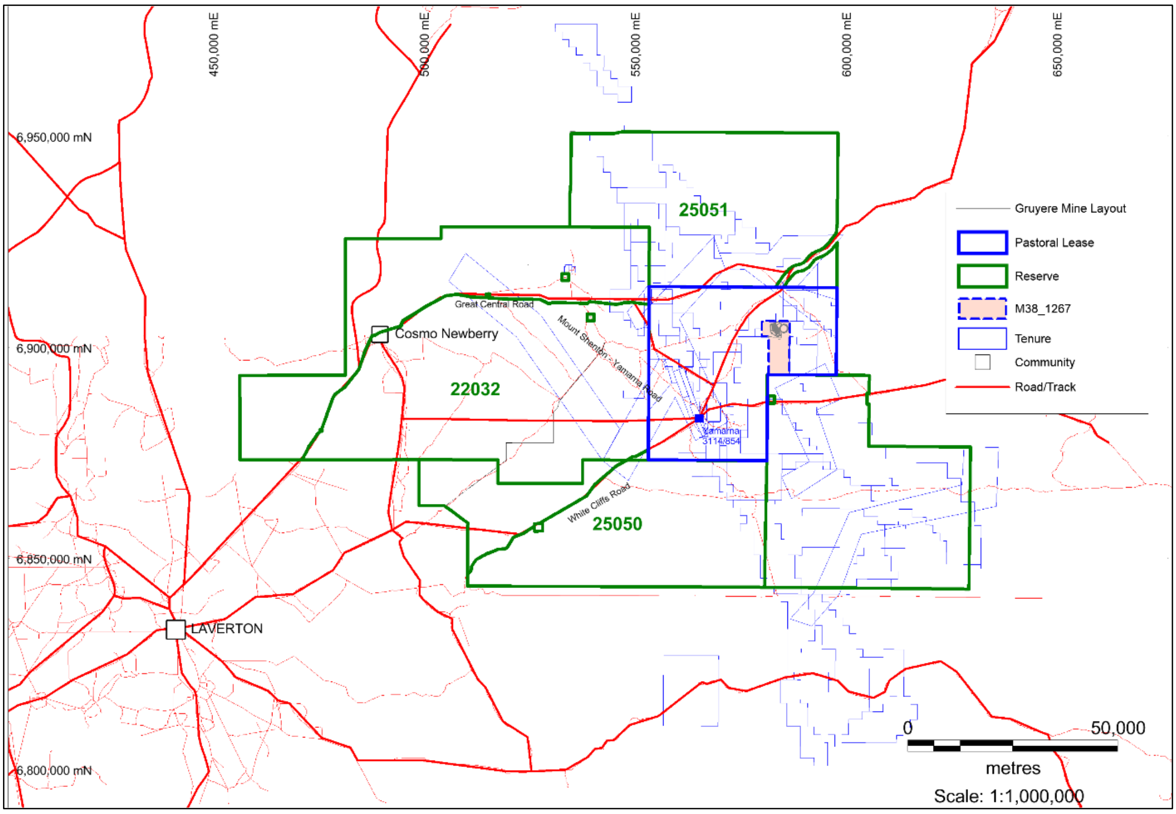

Figure 4-1: *Aboriginal Reserves Surrounding the Yamarna Pastoral Lease*

The Yamarna Pastoral Lease (LA3114/854) is wholly owned by Gold Road. The Yamarna Pastoral Lease renewal application was granted on 1 July 2015 by the Director General of the Department of Land (**DoL**) with the future Lease expiry date being 14 July 2062.

On 5 May 2016, the DMP granted the Mining Lease (M38/1267). The Mining Lease secures tenure over the Project for a period of 21 years (from the date of grant), renewable for a further period of 21 years.

All tenure required for the Project is subject to the Native Title Act 1993 (Commonwealth). The common law of Australia recognises a form of Native Title which, in circumstances where it has not been extinguished, reflects the entitlement of the indigenous inhabitants, in accordance with their laws or customs, to enjoy their traditional lands.

Gold Road entered into negotiations with the traditional owners of the Project area in 2015. The GCBNTA with the Yilka People and the CNAC was signed on 3 May 2016.

Miscellaneous licences L/38/180, 210, 211, 233, 235, 237, 251 and 254-256 have been granted and a further five licences have been lodged. These miscellaneous licences cover the process water borefield, Mt Shenton-Yamarna Road and gas pipeline corridors.

Exploration

Gold Road is authorised to explore for gold on its numerous Exploration Licences within the prospective Yamarna Greenstone Belt. These exploration tenements totalling approximately 5,000 km^2 are divided into two blocks; the northern block of tenements totalling (including ML38/1267) referred to as the North Yamarna Exploration Project, surrounds the Gruyere Gold Project area; the southern block of tenements totalling approximately 2,900 km^2 constitutes tenements held under a joint venture with Sumitomo which is referred to as the South Yamarna Exploration Joint Venture (Figure 7-1). Gold Road and Sumitomo each hold 50% ownership of the joint venture.

Ownership

The Project area (ML38/1267), associated Miscellaneous Licences, the underlying Yamarna Pastoral Lease, and all surrounding North Yamarna Exploration Licences are owned 100% by Gold Road.

The current corporate entity structure for Gold Road is presented in Figure 4-2.

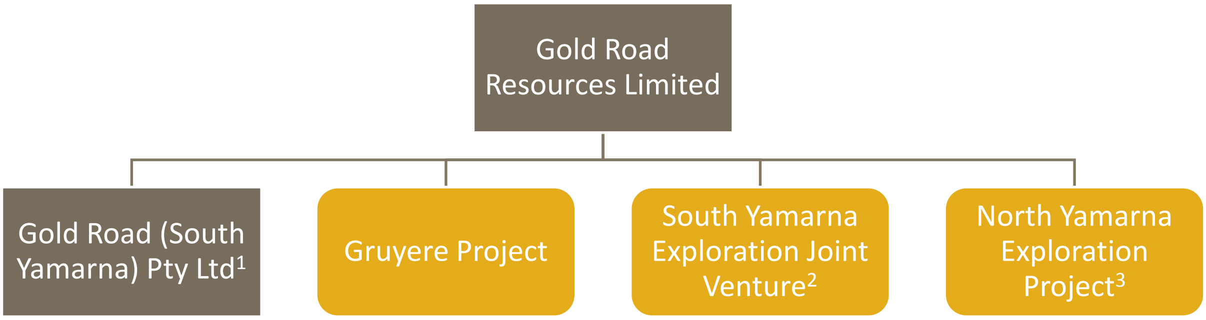

Figure 4-2: *Gold Road Corporate Structure*

Notes:

1. *Gold Road (South Yamarna) Pty Ltd (formerly known as Thatchers Soak Uranium Pty Ltd) currently has no material assets or liabilities and is considered a dormant entity. It was proposed to transfer the Thatcher Soak Uranium project into this vehicle and spin this entity off. The Thatcher Soak Uranium project remains within Gold Road Resources Limited, and forms part of the North Yamarna Exploration Tenements.*
2. *The South Yamarna Exploration Joint Venture is an unincorporated Exploration Joint Venture with Sumitomo, with both parties holding a 50% interest.*
   - *The North Yamarna Exploration project includes the Central Bore, Alaric and Attila deposits as well as other tenements to the north of the South Yamarna Exploration Joint Venture. Brown shading refers to Legal entity*
   - *Gold shading refers to Key asset or Project*

Funding for the project was secured with a 50:50 joint venture agreement between Gold Road and a wholly owned Australian subsidiary of Gold Fields Limited. Details of the agreement were publicly disclosed on 7 November 2016[30].

4.3 Royalties and Agreements

Production royalties will be payable to the State Government and the Yilka People. Gold Road will also have ongoing cost commitments during operations towards environmental and approvals compliance activities as well as progressive rehabilitation and final mine closure activities. Third party agreements will be required for linear infrastructure including the main access road, gas pipeline and water supply pipelines.

Royalties

In Western Australia, mineral royalties are payable either under the Mining Regulations 1981 or various State Agreement Acts. Under Regulation 86AA(4) of the Mining Regulations 1981 the rate of royalty payable for gold metal produced after 30 June 2000 is 2.5% of the value of the gold metal produced.

Agreements

On 3 May 2016 Gold Road reached agreement with the Yilka People and the CNAC on the GCBNTA. Under the GCBNTA, Gold Road has agreed to pay an Aboriginal Heritage Royalty.

[30] ASX:GOR Gold Road Resources Public Disclosure, 7 November 2016, "Gruyere Gold Project to be Developed in Joint Venture with Gold Fields LTD"

Aside from the royalty payment under the GCBNTA, there are no other existing third party royalties or milestone payments.

4.4 Environmental Liabilities

Environmental Assessment

In Western Australia, EP Act provides that, where a development proposal is likely to have a significant effect on the environment, the proposal may be referred to the OEPA for a decision on whether or not it requires formal assessment under Part IV of the EP Act, and if it is to be assessed, the level of assessment. On 20 June 2016, the OEPA advised a level of API-A was required for the Project.

The preliminary environmental factors identified by the OEPA as needing to be addressed in the API-A submission are subterranean fauna, flora and vegetation, heritage, rehabilitation and decommissioning. Gold Road is developing management and mitigation plans to avoid or reduce any potential impacts on these factors to acceptable levels.

A work program for the remainder of 2016 has been developed to complete all remaining archaeological surveys and development of mitigation plans. Final Project EPA Part IV approval is anticipated to be received by January 2017.

Aboriginal Heritage

Compliance with the Western Australia Aboriginal Heritage Act 1972 is a standard condition imposed on mining tenements in Western Australia. The Act applies to all mining tenements in Western Australia and provides protection for significant sites of Aboriginal heritage.

Gold Road entered into the GCBNTA with the Yilka People and CNAC over their respective claim areas following community consultation and negotiation meetings. The GCBNTA includes obligations on Gold Road regarding heritage and the conduct of heritage survey, pursuant to a Cultural Heritage Management Plan.

4.5 Permitting

The approvals strategy for the Project was based on separating the Project into two distinct components: the Mining, Process Plant and associated Infrastructure (inclusive of borefields and access roads); and the Gas Supply.

Mine Permitting

Most of the infrastructure required to support the mining operation is on the granted Mining Lease, however, the access roads and water supply pipelines are within Miscellaneous Licence infrastructure corridors. The airstrip and accommodation village locations are within a Miscellaneous Licence off the Mining Lease.

Gold Road has commenced the formal environmental assessment of the Project and identified that the development approvals pathway will be in accordance with Part IV of the EP Act, in addition to a Mining Proposal under the Mining Act 1978.

Gas Supply Permitting

The proposed power source for the Project is an on-site, gas fired power station with emergency dual fuel (diesel/gas) capability under a BOO contract. Gold Road is planning to deliver gas to the site via a gas pipeline from the Eastern Goldfields Pipeline (**EGP**). The corridor route is from a point on the EGP south-west of Laverton, along the White Cliffs Road reserve through to Gruyere. A Miscellaneous Licence for this gas pipeline alignment was pegged and is currently being negotiated with underlying tenement holders by the BOO contractor. It is anticipated that the grant of tenure will be in early Q1 2017.

Closure and Rehabilitation

Mine Closure Plans are required by DMP for all new Mining Proposal applications and must be prepared in accordance with the Guidelines for Preparing Mine Closure Plans (DMP and EPA, 2015). This requirement is stipulated as a tenement condition under the relevant provisions of the Mining Act 1978 (including Section 84).

4.6 Access

Access to the Project from Laverton is either via White Cliffs Road (road reserve number PIN 1356588) or Mt Shenton-Yamarna Road via Great Central Road (Figure 4-1). Both routes are dual-lane unsealed public roads providing 4WD access. White Cliffs Road is owned by the DoL and maintained by the Shire of Laverton.

The preferred access for the Project is the route via the Mt Shenton-Yamarna Road. A new main site access road will be developed from this road, eastwards for a distance of 28.5 km via a private access road crossing tenements owned by Gold Road. Access across Aboriginal Reserves has been included in the consents obtained under the GCBNTA.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHSIOGRAPHY

5.1 Physiography

The Project area lies at the western margin of the Great Victoria Desert of Western Australia. This area consists of a predominantly flat landscape truncated by the Yeo Palaeovalley drainage system that flows towards the Yeo Lake about 60 km east of the Project (Figure 1-1).

The Project area has varying topography ranging from sand plains and dunes with some regional breakaway areas of indurated, weathered Permian sandstone, rising to small hills up to an elevation of 500 metres AHD or around a maximum of 60 metres above the surrounding landscape.

The area is located in the Great Victoria Desert Shield subregion (GVD1) and is described as an arid active sand-ridge desert of deep Quaternary aeolian sands overlying Permian rocks of the Canning Basin and Archaean rocks of the Yilgarn Craton. The area is characterised by dune sands, red in colour and incoherent with sandplains formed of the same material. Typically, on flatter ground there is a red loam on which Mulga (Acacia aneura) trees are the dominant species. Breakaway areas are typically vegetated with Spinifex hummock grass (Triodia basedowii).

5.2 Climate

The Great Victoria Desert is characterised by an arid climate, with hot summers and cool winters. Summer maximum temperatures average approximately 35°C, while winter average minimum temperatures are approximately 5°C.

Average annual rainfall in the Yamarna region is 200 to 230 mm and results from both locally generated thunderstorms (October to December) and dissipating tropical cyclones tracking south-east from the coast (January to May). Rainfall is sporadic, but slightly higher in the cyclone season (Figure 5-1). Rain events are infrequent with approximately 30 rain days on average per year. Most of the annual rainfall is received in one or two significant events with some years having close to zero rainfall.

Minor watercourses and drainages at the Project site are ephemeral and dry for the majority of the time. Flows occur periodically following significant rainfall events, particularly during the cyclone season. Flood berms and diversion channels will be constructed at the mine site to control flood events.

Figure 5-1 shows the data from the Bureau of Meteorology for Yamarna which operated as a weather station from 1967 to 1998. The nearest presently operating (official) weather station is now at Laverton 160 kilometres to the south-west.

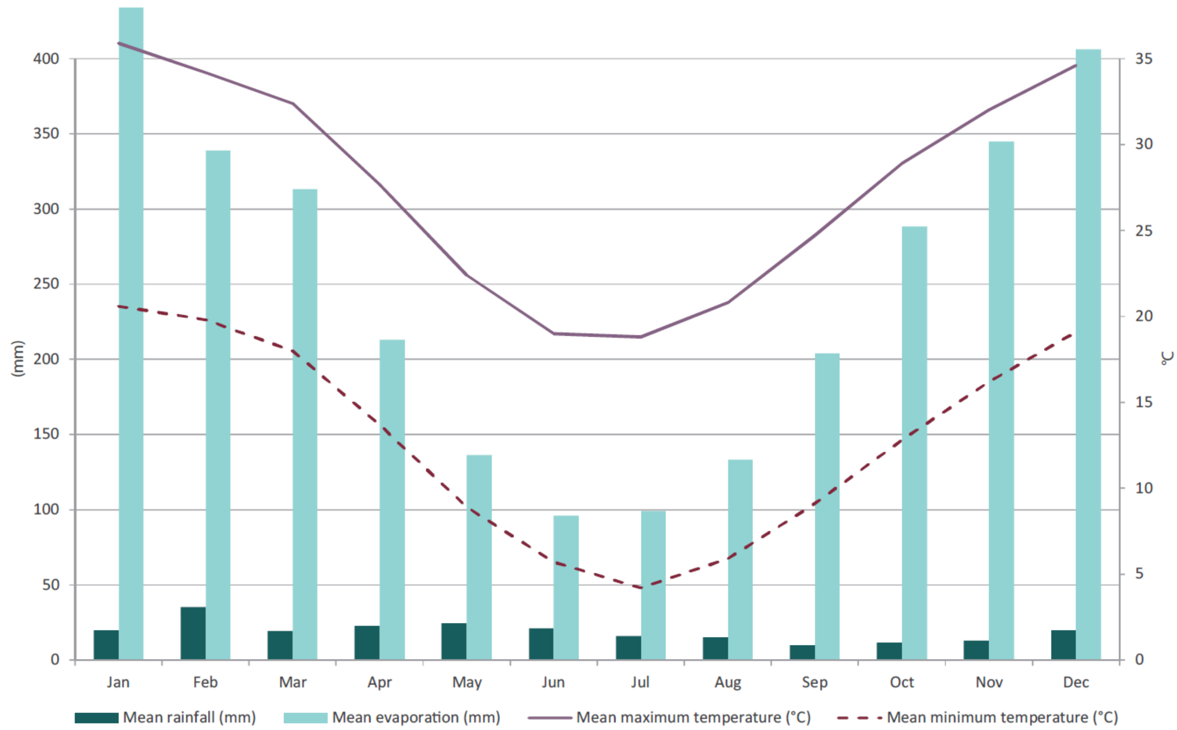

Figure 5-1: *Monthly Mean Rainfall and Evaporation at Yamarna (1967-1998)*

5.3 Access and Transport

Road access to the Project site is from Laverton is over a distance of approximately 200 kilometres. Access from Laverton is along the Great Central Road, turning off 153 kilometres from Laverton; the site access road will be 47.7 kilometres in length, comprising 19.2 kilometres on the Mt Shenton-Yamarna Road and 28.5 kilometres on a newly constructed main site access road (Figure 5.2).

An alternative access route exists directly east from Laverton via White Cliffs Road and the Mt Shenton-Yamarna Road onto the proposed new main site access road.

The accommodation village/airstrip access road will be located approximately 6 kilometres from the end of the main site access road. The main site access road will terminate adjacent to the mine contractors' service area and to the southern entrance to the process plant site. A further 1.2 kilometres of plant access roads will connect the main site access road to the mine contractors' service area and the power station.

The current condition of the Great Central Road is suitable for the construction traffic requirements of the Project. Both the Great Central Road and the White Cliffs Road are unsealed and maintained by the Shire of Laverton. It is anticipated that the maintenance of the road will be continued by the Shire. As part of the Shire's road safety procedure, the Shire closes the roads during excessive wet weather periods or restricts traffic movements.

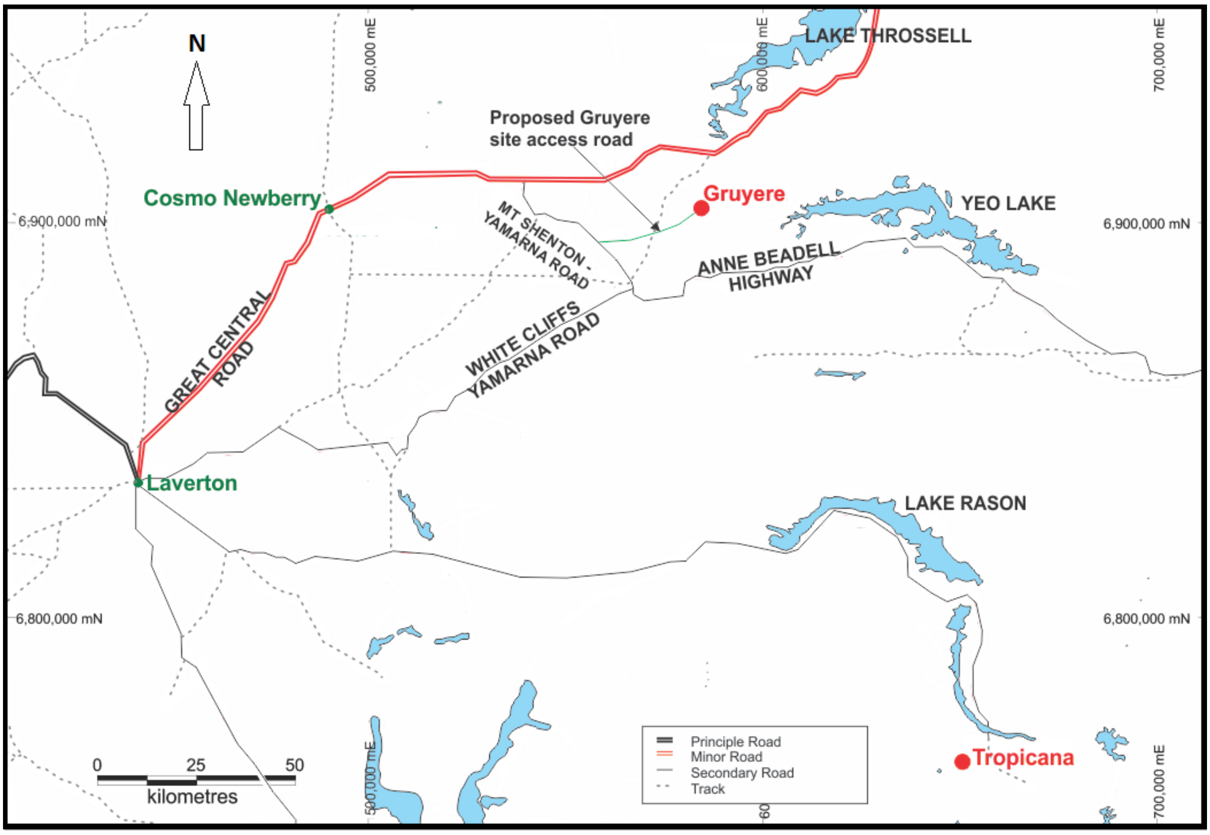

***Figure 5-2**: Gruyere Road Access*

Air Transport

An existing serviceable airstrip is located at Yamarna close to the current Gold Road exploration camp. This airstrip will be used to transport construction personnel until the new airstrip is built at Gruyere in mid-2017.

Laverton has a commercial airport which is located 3 kilometres from the centre of town. Laverton is connected to Perth with commercial flights available currently three days per week.

A 24 hour Civil Aviation Safety Authority (**CASA**) compliant airstrip, including a 2.1 kilometres long runway with bitumen seal, terminal and fuel facility to suit a 100 seat aircraft, will be built approximately 6 kilometres south-west of the process plant and adjacent to the Project accommodation village. The airstrip will be constructed early in the construction phase to minimise the reliance on road transport and the smaller capacity airstrip at Yamarna for personnel access to and from the Project.

5.4 Local Resources

The area immediately surrounding the Project has a low population and little established infrastructure.

Laverton has a population of approximately 1,227 residents of which 417 people permanently reside in the township (2011 census). Laverton was established from the success of the Craiggiemore gold mine in 1897. The town site was surveyed in July 1899 with residential and business areas developed and the town of Laverton was finally gazetted in July 1900.

The town has a community bus service, a gymnasium and a Community Resource Centre which provides communication technology services, a library, the Department of Transport Licensing Agency, secretarial services and training courses. The town also caters for travellers with a fuel station, a shop and several motel options.

Cosmo Newberry, locally referred to as Cosmo, is a small Australian Indigenous community with a population of 71 (2011 census), located approximately 80 kilometres north-west of the Project. The community is managed through its incorporated body, CNAC, incorporated under the Aboriginal Councils and Associations Act 1976 in 1991. In 1994 the community made the decision to become affiliated with Ngaanyatjarra Council.

5.5 Proposed Project Infrastructure

The Gruyere mining lease granted in May 2016 covers an area of 6,845.5 ha. The main footprint of the planned mining infrastructure within the mining lease covers an area of approximately 2,084 ha. This footprint includes final locations of the open pit, waste rock dumps, the TSF, mine access roads, processing plant and associated infrastructure.

Outside the mine lease but within Gold Road's wholly owned Yamarna Pastoral Lease there are additional footprints for the accommodation village and airstrip locations. In addition, there are mining-related linear infrastructure footprints for the water supply pipelines for the Yeo and Anne Beadell Borefields and the gas pipeline route. These pipeline routes extend outside the Yamarna Pastoral Lease boundary.

6 HISTORY

6.1 Exploration and Ownership History

Modern exploration of the Yamarna Greenstone Belt located on the eastern margin of the Yilgarn Craton commenced in 1971 with exploration for uranium by Mining Corp Exploration NL in the North Yamarna area.

Between 1971 and 2006, exploration in Gold Road's Yamarna North Project area included nickel, chromite and gold and was carried out by a number of companies including CRAE, Texas Gulf Australia Ltd, Metal Mining Australia, Zanex NL and Asarco Exploration Company Inc (Asarco). Zanex NL was the first company to delineate a gold deposit, named Attila South, and this marked the start of 20 years of systematic gold exploration in the Yamarna area. Exploration in Gold Road's South Yamarna Project area was commenced by BHP in the late 1980s. Other companies that explored the area included Kilkenny Gold NL, Western Mining Corporation and AngloGold Ashanti Australia in joint venture with Terra Gold Mining (**Terra Gold**).

Exploration in the Dorothy Hills Greenstone Belt on the eastern side of the Yamarna Greenstone Belt commenced in 1993 with Zapopan NL, Pegasus Gold Australia and later Asarco. Asarco exited from Australia in 2005 and Eleckra Mines Ltd (**Eleckra**), which listed in July 2006, purchased Asarco's Yamarna North tenements and also obtained tenements in Yamarna South from Terra Gold. Eleckra achieved a consolidation of tenements covering an area of 3,000 km^2 in the Yamarna Belt. In 2010 Eleckra changed its name to Gold Road Resources Limited to better reflect the gold focus of the junior exploration company.

6.2 Gold Road Exploration

Gold Road secured additional exploration tenements in 2009 and 2010 which brought the total exploration tenement holding to around 5,000 km^2. The additional tenements included E38/2362 over Dorothy Hills that now contains the Gruyere gold deposit.

Exploration by Gold Road initially focussed on the Yamarna Shear Zone on the western side of the greenstone belt. Shear-hosted gold mineralisation was located in an area referred to as the **Attila Trend**.

In 2009 Eleckra located gold mineralisation in an area 3.7 kilometres east of the Attila deposit and subsequently defined a gold deposit known as Central Bore. The Central Bore deposit has a strike length of 800 metres and has been drilled to a depth of 440 metres below surface and remains open at depth and down plunge.

In 2012 Gold Road conducted a detailed 50 metre line-spaced airborne magnetic and radiometric survey over its entire 5,000 km^2 tenement holding. This formed the foundation for a regional targeting program aimed at locating 'world-class gold deposits' in the Yamarna area. The program subsequently identified 10 Camp-scale targets across the Yamarna tenements (Figure 6-1).

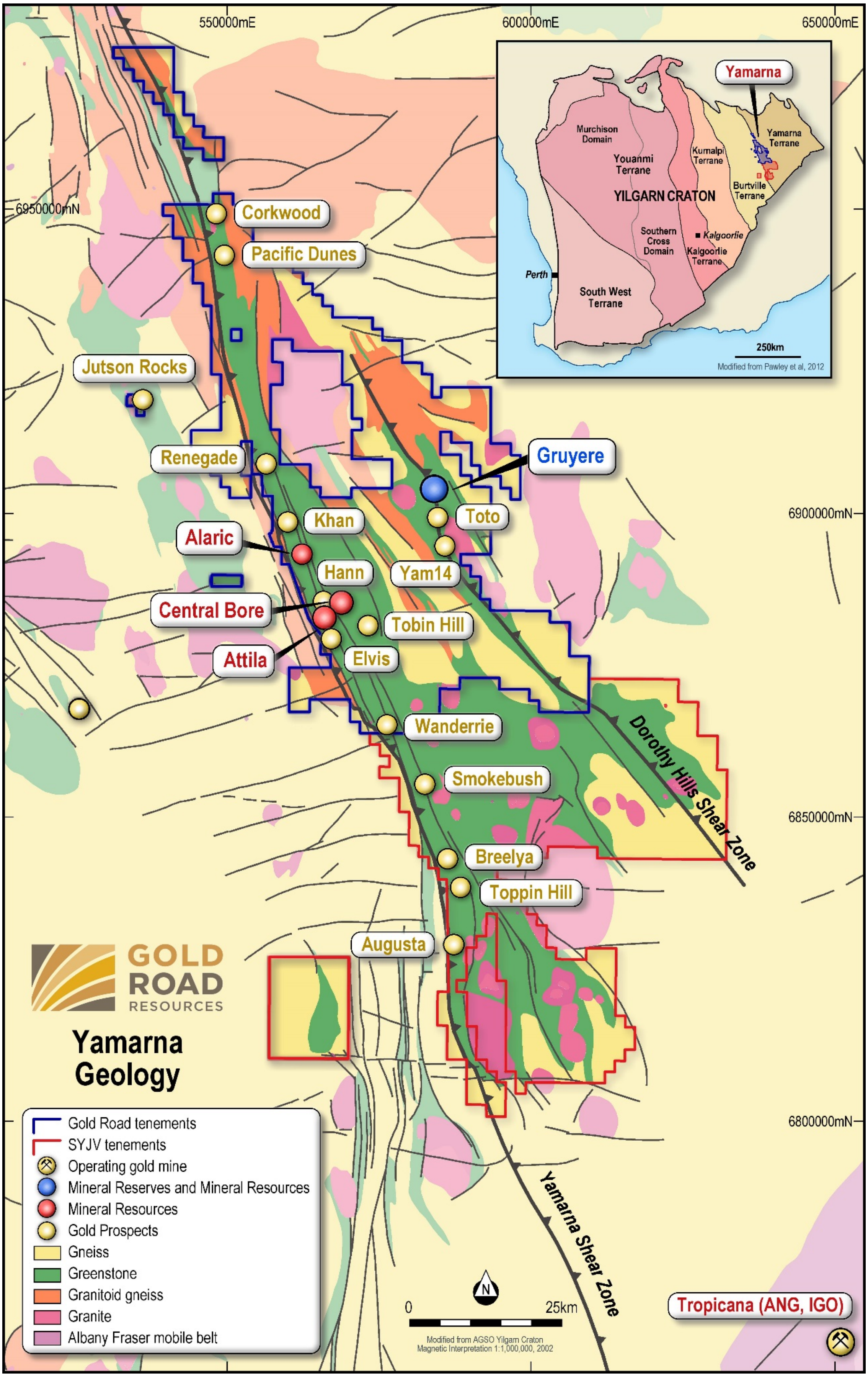

Figure 6-1: *Geology and Location of Prospects and Deposits of the Yamarna Greenstone Belt (MGA94 51)*

The first Camp-scale target to be tested was the **South Dorothy Hills** target located approximately 25 kilometres north-east of the Central Bore deposit and consisting of priority structural and geochemical targets at **Gruyere** and YAM14. Rotary air blast (**RAB**) drilling and follow-up reverse circulation (**RC**) drilling intersected gold mineralisation over the Gruyere target. No previous exploration had been conducted on or around the Gruyere deposit prior to Gold Road's discovery.

By December 2013 Gold Road had delineated gold mineralisation over a strike length of 1,600 metres (Figure 6-1). An early RC drill section through the Gruyere deposit is shown in Figure 6-2.

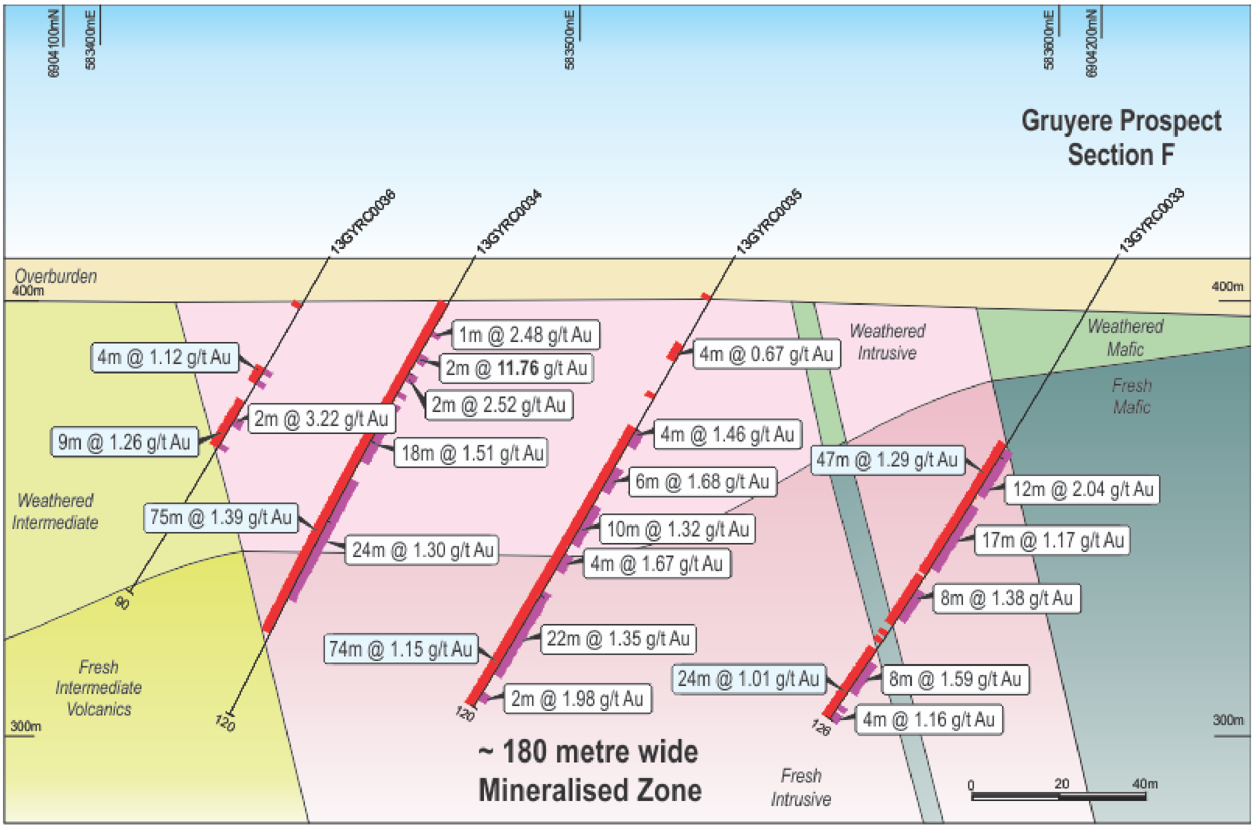

Figure 6-2: *First RC Drilling on Section F (50000N), November 2013*

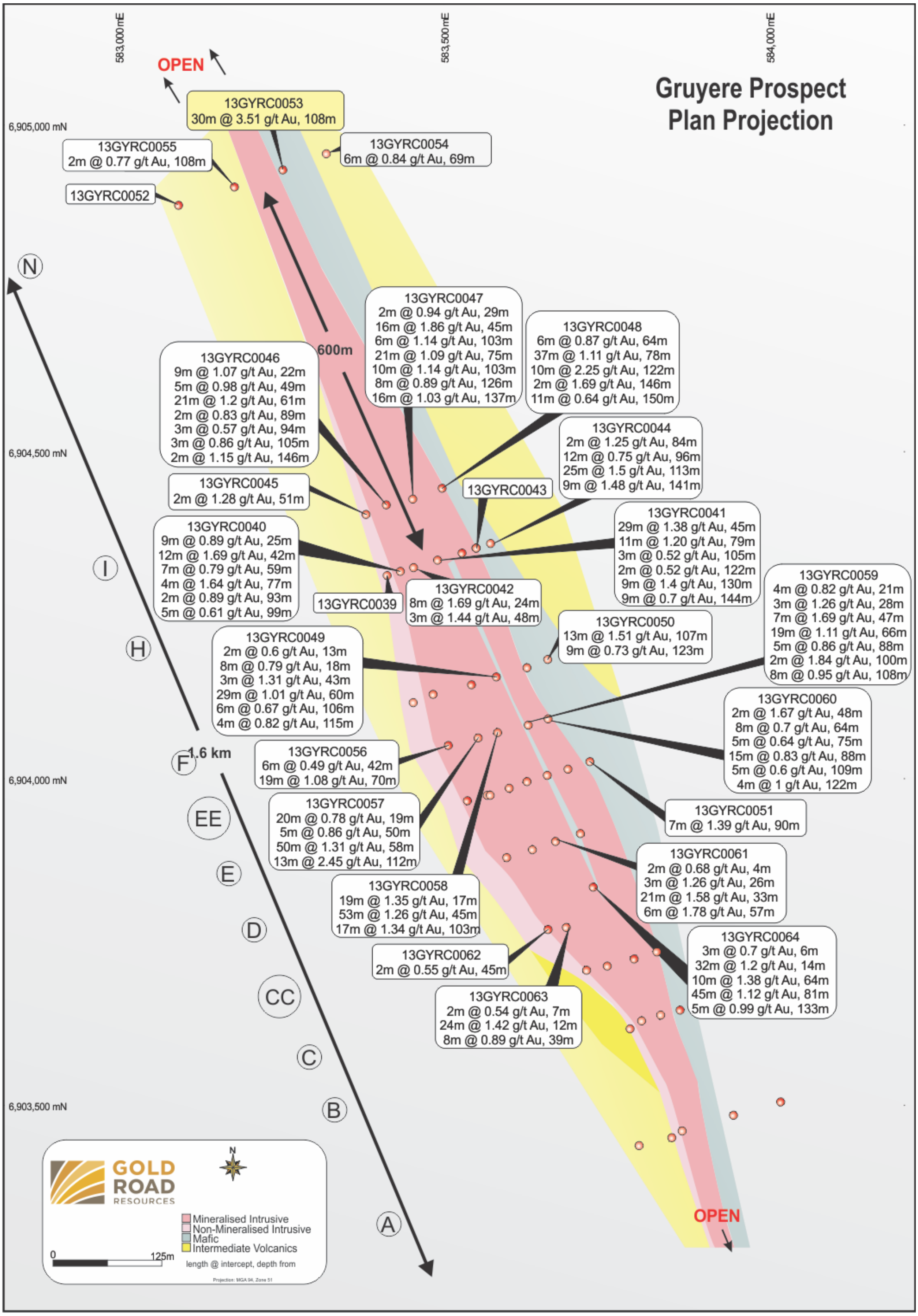

Figure 6-3: *Drill Plan - December 2013 with 1,600 m Mineralised Strike Defined (MGA94 51)*

6.3 Gold Road Mineral Resource Delineation

Gold Road reported a maiden Mineral Resource for the Gruyere gold deposit in August 2014[31] based on approximately 38,000 metres of resource drilling which included both RC holes and diamond drill holes (**DDH**). By that stage the deposit had been delineated over a strike length of 1,800 metres and to a maximum depth of 500 metres below surface. The deposit remained open at depth.

The August 2014 Mineral Resource estimate was reported at a cut-off of 0.7 g/t Au and within a mineralisation envelope constrained by an optimised pit shell at a gold price of A$1,550 per oz. The Measured and Indicated Resource was 40.2 Mt at 1.22 g/t Au with contained gold of 1.58 Moz (Table 6-1). The Inferred Resource was 56.7 Mt at 1.24 g/t Au with contained gold of 2.26 Moz. The Mineral Resource estimate was reported in accordance with the JORC Code 2012.

Additional drilling was undertaken by Gold Road during 2014 and 2015. In late 2014 a deep diamond drill hole extended the depth continuity of the deposit to almost 750m below surface.

Gold Road completed a Scoping Study for the Gruyere Project in January 2015[32]. The study indicated potential for development of a gold mine and justified further evaluation of the deposit.

By May 2015 total drilling had reached 66,000 metres (41,000 metres of RC drilling and 25,000 meters of DDH drilling). Gold Road reported[33] an updated Mineral Resource estimate in May 2015 based on these drilling results. The Gruyere May 2015 Mineral Resource was estimated using a 0.7 g/t Au cut-off within a mineralisation envelope constrained by an optimised pit shell at a gold price of A$1,600 per oz (A$50 per oz higher than the price used for the August 2014 estimate). The Measured and Indicated Resource estimate was 87.5 Mt at 1.21 g/t with 3.4 Moz of contained gold (Table 6-1). This estimate represented an increase on the August 2014 estimate of 118% in tonnes and 116% in contained gold.

Between May and September 2015, Gold Road increased total drill meterage to 67,665 metres from 207 RC drill holes and 108 DDH holes. In September 2015, the deposit was extended to 1,150 metres below surface by further deep diamond drilling. A further update of the Mineral Resource was reported in September 2015[34].

The September 2015 Mineral Resource estimate was used as the basis of a Preliminary Feasibility Study that Gold Road completed in February 2016[35]. The resource was estimated using a 0.7 g/t Au cut-off within a mineralisation envelope constrained by an optimised pit shell at a gold price of A$1,600 per oz (the same parameters as in May 2015). The Measured and Indicated Resource estimate was 95.1 Mt at 1.35 g/t with 4.1 Moz of contained gold. The Inferred Resource estimate was 33.3 Mt at 1.40 g/t Au with contained gold of 1.5 Moz (Table 6-1).

Drilling post September 2015 included 150 grade control equivalent RC holes (14,837 metres) and two DDH holes. This brought the total drill metres to 87,066 metres for the Gruyere gold deposit (55,958 metres of RC and 31,109 metres of DDH).

Gold Road published a paper detailing the exploration history with respect to the discovery of Gruyere in the conference proceedings of the NewGenGold conference in November 2015 (Reference 1).

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[31] ASX:GOR Gold Road Resources Public Disclosure, 4 August 2014, “3.84 Million Ounce Gruyere Maiden Gold Mineral Resource”

[32] ASX:GOR Gold Road Resources Public Disclosure, 27 January 2015, “*Gruyere Scoping Study a Robust Long Life Gold Project’*

[33] ASX:GOR Gold Road Resources Public Disclosure, 28 May 2015, “Gruyere Resource Grows to 5.51 Million Ounces Gold”

[34] ASX:GOR Gold Road Resources Public Disclosure, 16 September 2015, “Gruyere Resource Grows to 5.62 Million Ounces Gold”

[35] ASX:GOR Gold Road Resources Public Disclosure, 7 February 2016, “*Gruyere Pre-feasibility Study Confirms Long Life Gold Mine”*

The Mineral Resource for Gruyere was subsequently updated in April 2016[36] for input to a Feasibility Study. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the 22 April 2016 market announcement, and that all material assumptions and technical parameters underpinning the estimates in the April 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Gruyere Gold Project Mineral Resources and the information of the 22 April 2016 announcement. The April 2016 Mineral Resource estimate is presented and discussed in more detail in Section 14.

Table 6-1: *Gruyere Mineral Resource Reporting (August 2014, May 2015, September 2015) by Resource Category*

August 2014[1]			
Resource Category	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz)**
Measured	1.43	1.36	62
Indicated	38.76	1.22	1,515
Measured & Indicated	40.19	1.22	1,578
Inferred	56.74	1.24	2,260
Total	96.93	1.23	3,838
May 2015[2]			
Resource Category	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz)**
Measured	1.45	1.43	67
Indicated	86.09	1.21	3,337
Measured & Indicated	87.54	1.21	3,403
Inferred	50.27	1.30	2,108
Total	137.81	1.24	5,512
September 2015[3]			
Resource Category	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz)**
Measured	1.58	1.41	0.07
Indicated	93.48	1.35	4.05
Measured & Indicated	95.07	1.35	4.12
Inferred	33.31	1.40	1.49
Total	128.38	1.36	5.62
April 2016[4]			
Resource Category	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz)**
Measured	13.9	1.18	0.53
Indicated	91.1	1.29	3.79
Measured & Indicated	105.0	1.28	4.31
Inferred	42.7	1.35	1.85
Total	147.7	1.30	6.16

Notes:

1. *The August 2014 Mineral Resource is reported at a lower cut-off grade of 0.70 g/t Au. The Resource is constrained with an A$1,550 per ounce optimised pit shell based on parameters derived from an ongoing Scoping Study.*
2. *The May 2015 Mineral Resource is reported at a lower cut-off grade of 0.70 g/t Au. The Resource is constrained with an A$1,600 per ounce optimised pit shell on parameters derived from an ongoing Pre-Feasibility Study.*
3. *The September 2015 Mineral Resource is reported at a lower cut-off grade of 0.70 g/t Au. The Resource is constrained with an A$1,600 per ounce optimised pit shell on parameters derived from an ongoing Pre-Feasibility Study.*
4. *Gruyere Mineral Resource reported at 0.5 g/t Au cut-off, constrained within a A$1,700 per ounce Au optimised pit shell based on mining and processing parameters from the PFS (ASX announcement dated 8 February 2016), and geotechnical parameters consistent with the previous Mineral Resource estimate (ASX announcement dated 16 September 2015).*
5. *All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.*

[36] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, “Gruyere Resource Increases to 6.2 Million Ounces”

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project and its exploration tenements encompass the Yamarna and Dorothy Hills Greenstone Belts, the eastern most known greenstone belts of the Archaean Yilgarn Craton. The greenstone belts of the Yilgarn Craton are the dominant host for gold mineralisation and mined production in Australia and the Yilgarn Craton is recognised world-wide as a pre-eminent gold district (inset Figure 6-1).

The Yamarna and Dorothy Hills Greenstone Belts form a part of the Yamarna Terrane. The western margin of the terrane is marked by the 350 km long Yamarna Shear Zone which is a broad, crustal scale, east-dipping listric shear zone separating the Yamarna Terrane from the older Burtville Terrane to the west (Reference 2). The eastern margin of the terrane is typically sheared against interpreted metagranitic rocks which are entirely under cover. Trending north-west to south-east, the Yamarna Greenstone Belt extends over 250 kilometres in strike length, varies in width from three to 30 kilometres and is located on the western margin of the Yamarna Terrane.

Approximately 25 kilometres to the east is the north-west to south-east trending Dorothy Hills Greenstone Belt which extends for over 90 kilometres in strike, varies in width from 3 kilometres to 10 kilometres and is poorly exposed. The Dorothy Hills Greenstone Belt is host to the Gruyere Deposit.

Mafic rocks and intermediate to dacitic volcanics and volcaniclastics dominate the Yamarna Greenstone Belt, with subordinate ultramafic, felsic volcanic, feldspar porphyry, clastic sediment and chert units identified. The mafic rocks are primarily basaltic, variably deformed to schists, with locally preserved pillows and flow top breccias. Dolerite and gabbro sills are noted throughout the succession. Thin units of ultramafic rock are interlayered with mafics on the western margin of the Belt, extending for approximately 50 kilometres along the central part of the Belt. Felsic volcanic and volcaniclastic sequences are found throughout the Belt.

The mafic rocks of the Dorothy Hills Greenstone Belt are predominantly foliated and metamorphosed basalts. The basalts include concordant sheets of dolerite which may represent thicker volcanic flows. Sedimentary rocks are interbedded with mafic rocks in the western part of the Dorothy Hills Belt, while felsic schists and intrusions interlayered with strongly sheared mafic rocks dominate the central portion. In the centre of the northern end of the Dorothy Hills Greenstone Belt a granite intrudes a regular body of foliated monzogranite (the Ziggy Monzonite), which has sheared contacts with the greenstone. The Gruyere Deposit is hosted entirely within the Gruyere Porphyry, a quartz monzonite intrusive emplaced into the regional scale Dorothy Hills Shear Zone.

The geology of the Yamarna Terrane, including the Yamarna and Dorothy Hills Greenstone Belts, remains poorly understood in comparison to other greenstone belts in the Yilgarn. Ongoing doctoral studies which commenced in 2014 are focussed on developing the holistic stratigraphic and geotectonic understanding of the Yamarna Terrane geology, in comparison with the well-known and gold-prolific Kalgoorlie-Norseman and Laverton Greenstone Belts.

7.2 Deposit Geology

The Gruyere Deposit is located on a flexure point of the regional scale Dorothy Hills Shear Zone within the Dorothy Hills Greenstone Belt where the shear zone changes from a northerly direction to a north-north-westerly direction (Figure 7-1). Gold mineralisation is hosted within the steep easterly dipping Gruyere Porphyry, a medium-grained quartz monzonite porphyry (plagioclase, quartz and ferromagnesian minerals) that has intruded the country rocks, elongated in the direction of the shear zone.

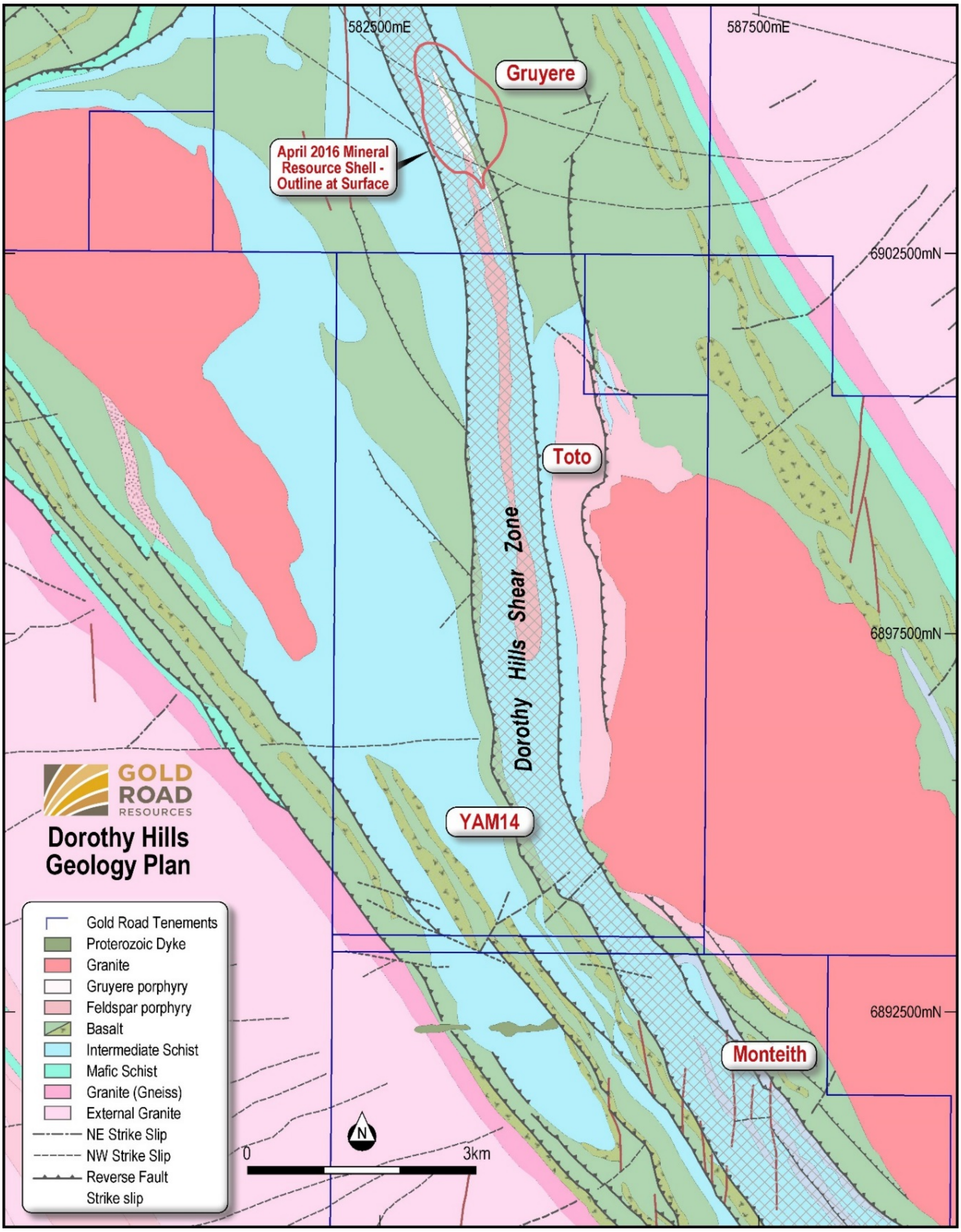

Figure 7-1: *Geology of the Dorothy Hills Greenstone Belt (MGA94 51)*

The cover unconformably overlying the Archaean rocks at Gruyere includes Quaternary aeolian sands generally 1 to 3 metres thick, with localised sand dunes up to 10 metres in height. A semi-consolidated Permian sandstone (Paterson Formation) underlying the sand is absent over the southern end of the Gruyere Porphyry and gradually increases in thickness to 30 metres at the northern end. Weathering of the Archaean rocks increases in depth from 45 metres (to base of weathering) in the south, to 85 metres in the north. The weathering profile is truncated and comprises a minimal clay zone progressing to a Saprock-transition zone into Fresh rock. Mineralisation occurs within all weathered zones of the Gruyere Porphyry, with approximately 93% of the Mineral Resource in fresh rock.

The host Gruyere Porphyry averages around 90 metres in horizontal width through the deposit with a maximum width of 190 metres in the centre of the deposit and tapering to around 5-10 metre width at the northern and southern extremities. A persistent 1 to 5 metre wide steeply dipping mafic dyke (Main Dyke) is located proximal to the hanging wall. Other localised thin sub-parallel, intensely sheared, mafic to intermediate dykes or rafts are noted throughout the porphyry

The stratigraphic sequence comprises a tholeiitic (low thorium) basalt with preserved pillow lava textures overlain by a sequence of intermediate to mafic volcaniclastics, often described as fine grained laminated clastic meta-sediments. The tholeiitic basalt is observed on the hanging wall of the Gruyere Porphyry south of the cross cutting Alpenhorn Fault, with volcaniclastics in the hanging wall position north of the fault. Footwall stratigraphy comprises volcaniclastics and to the south of the Alpenhorn Fault a second, poorly mineralised, felsic to intermediate porphyry is observed which appears texturally similar to the Gruyere Porphyry.

Shearing is variably developed in the country rock and the Gruyere Porphyry. Shear intensity is very high at the contact of the porphyry, with the contact being sharp on both hanging wall and footwall margins. A strong foliation fabric in the Gruyere Porphyry is invoked by the Dorothy Hills Shear Zone and has the same orientation as the porphyry, steeply dipping (70 to 80°) to the east and striking to the north. Foliation intensity within the porphyry varies from very weak to very strong, with measured kinematic indicators showing both sinistral and dextral and reverse and normal movement indicating a complex structural history. Increased localised deformation forms a crenulation of the foliation with a steep down-dip lineation consistent with the orientation of observed gold grade continuity. The gross movement sense on the Dorothy Hills Shear Zone is interpreted as dextral, with strong sinistral overprint evident in the Gruyere Deposit area.

A plan view and a cross section through the deposit showing the main interpreted geological features of the Gruyere deposit are shown in Figure 7-2 and Figure 7-3 respectively. Both figures are using a local survey grid, Gruyere Grid, on which grid north is orientated 340° to true north.

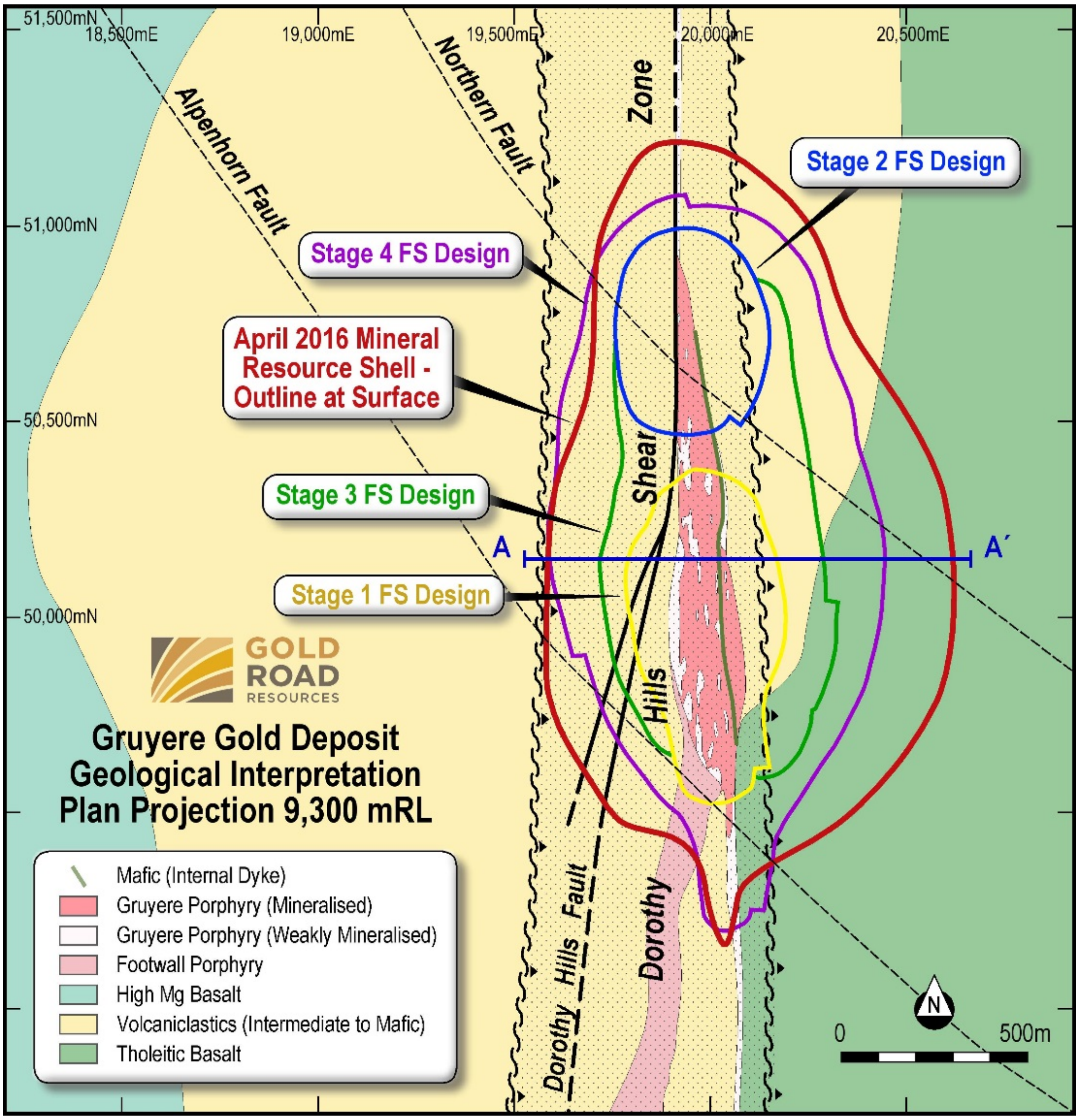

Figure 7-2: *Gruyere Deposit Geological Interpretation Plan View (Gruyere Grid)*

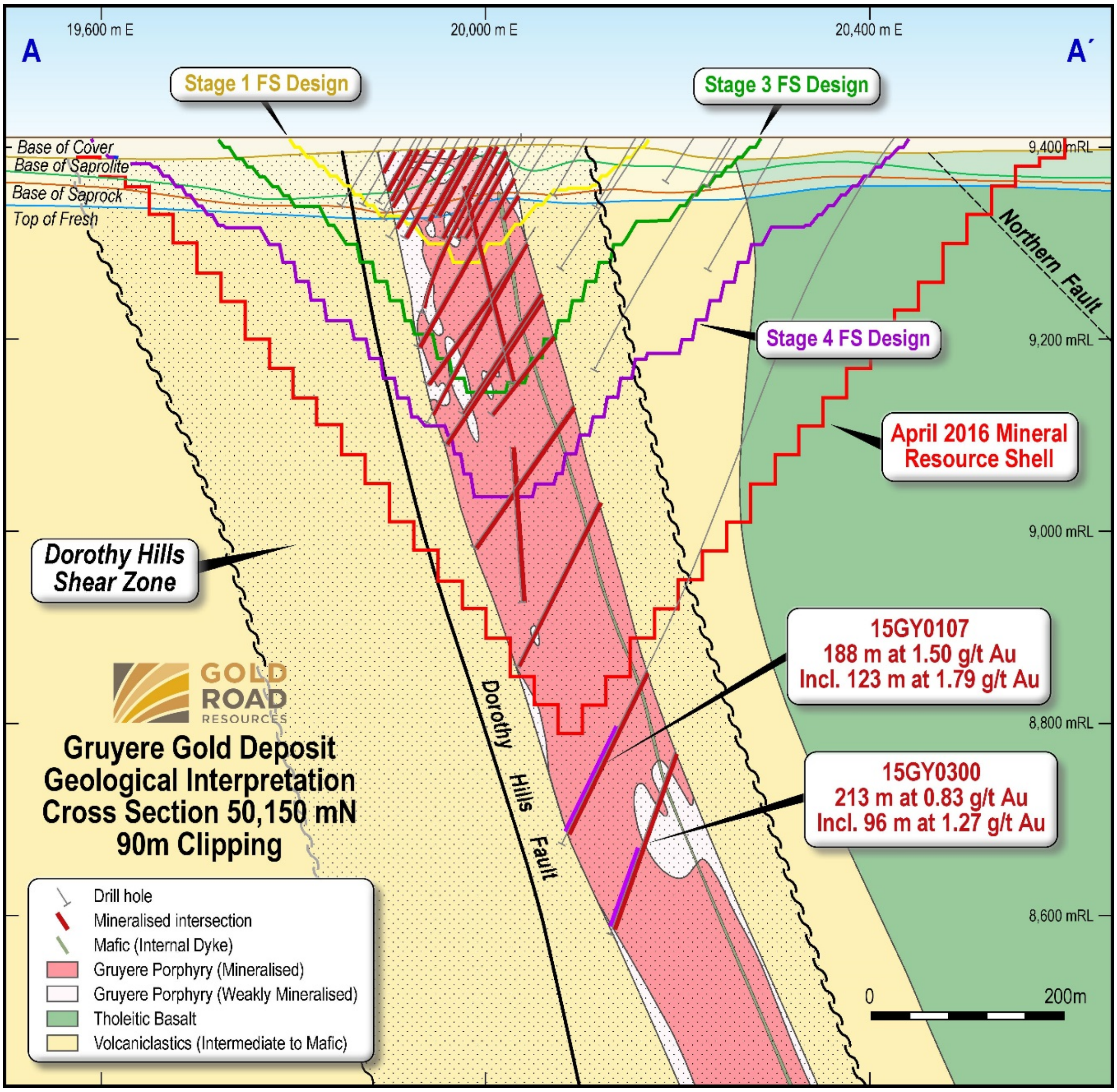

Figure 7-3: *Gruyere Deposit Geological Interpretation Cross Section (Gruyere Grid)*

7.3 Mineralisation

Gold mineralisation at Gruyere has developed in response to a complex reverse shearing structural event. The porphyry, a more competent and brittle body compared to the relatively ductile host rocks, responded to deformation with significant cracking and fracturing resulting in increased permeability. Gold-bearing mineralising fluids were able to flow freely through the rock mass, resulting in uniform and disseminated gold mineralisation ubiquitous to the porphyry.

North-west striking, cross-cutting arcuate thrust faults, initially interpreted from magnetic data and mapped changes in stratigraphy are believed to be an important gross control to mineralisation. These faults are interpreted as early features, growth faults or thrusts that offset the regional stratigraphy, but not the Dorothy Hills Shear Zone or Gruyere Porphyry. The faults are coincident with zones of thickening of the Gruyere Porphyry (Alpenhorn Fault), areas of higher-grade development in the north (Northern Fault) and are interpreted to have acted as additional conduits to fluid flow during the gold mineralising event.

The entire Gruyere Porphyry is variably altered and gold grade can be related to variations in style and intensity, of alteration, structure, veining and sulphide species (Table 7-1). Zones containing higher grade gold mineralisation above 1.2 g/t Au generally have strong albite ± sericite ± chlorite ± biotite alteration and are associated with a sulphide assemblage of pyrrhotite + pyrite ± arsenopyrite, weak to moderate foliation, common micro-fracturing and steeply dipping quartz veining.

Sulphides are common throughout the zone of gold mineralisation, with pyrite dominant in the upper areas and pyrrhotite-arsenopyrite increasing with depth. The total percentage of sulphide minerals is generally in the range 0.5-2%. Quartz ± carbonate vein sets observed in diamond core and optical televiewer surveys show multiple character: early shear veins parallel to the shear foliation; late tabular veins (0.01 to 1 metre thick) at high angle to foliation with variable albite alteration halos; veins with strong chlorite margins; chlorite fractures ± albite halos; and fine stock work veins in areas of intense alteration.

Table 7-1: Summary Gruyere Alteration, Structure, Veining, Sulphide and Gold Grade Range

Approximate Gold Grade Range (g/t)	0.01 to 0.30	0.30 to 0.80	0.60 to 1.50	1.20 to 1.80	1.20 to 2.50+
Implicit Model	Weakly Mineralised	Mineralised			
Alteration Assemblage	Hematite ± magnetite	Sericite ± albite ± chlorite ± biotite	Albite ± sericite ± chlorite ± biotite	Albite ± sericite ± chlorite ± biotite	Albite ± sericite ± chlorite ± biotite
Alteration Intensity	Weak to Strong	Weak - moderate	Weak	Moderate	Strong (no primary textures preserved)
Sulphide Assemblage		pyrite ± pyrrhotite	pyrite + pyrrhotite ± arsenopyrite	pyrite + pyrrhotite ± arsenopyrite	pyrrhotite + pyrite ± arsenopyrite
Core Photos (4 by 3 cm)					
Structure	Weak foliation	Weak foliation	Weak foliation	Weak foliation (locally moderate), Crenulation, occasional microfracturing.	Weak to moderate foliation (locally strong), Crenulation, microfracturing common.
Veining	Shallow dipping quartz	Steep and shallow dipping quartz, some internal fractures, occasional infilled by amphibole±py	Steep and shallow dipping quartz, some internal fractures, occasional quartz-carbonate	Predominantly steep dipping quartz, some internal fractures, occasional quartz-carbonate	Predominantly steep dipping quartz, some internal fractures, occasional quartz-carbonate

Weathering

Below the Permian sandstone cover there is a weathered profile in the Archaean rocks which varies in thickness from 50 to 90 metres and is divided into an Oxide zone and a Saprock-Transition zone (Figure 7-4). The Oxide zone contains clay-rich Saprolite rock with complete oxidation of sulphides and leaching and re-mobilisation of gold. A thin gold dispersion blanket is interpreted at the base of the Oxide zone; this blanket extends beyond the porphyry contact. The Oxide zone is generally low grade and represents approximately 1% of the total gold mineralisation at Gruyere. The Saprock-Transition zone displays decreasing clay content and decreasing proportion of oxidised sulphide minerals with depth and is gradational into the Fresh (primary) zone.

The boundary between the Oxide and Saprock-Transition zone (solid line in Figure 7-4) marks a distinct change in the characteristics of the distribution of gold mineralisation. Above this boundary gold mineralisation in the Oxide zone exhibits lower grade, higher variance and low continuity whereas below the boundary mineralisation increases in grade and continuity.

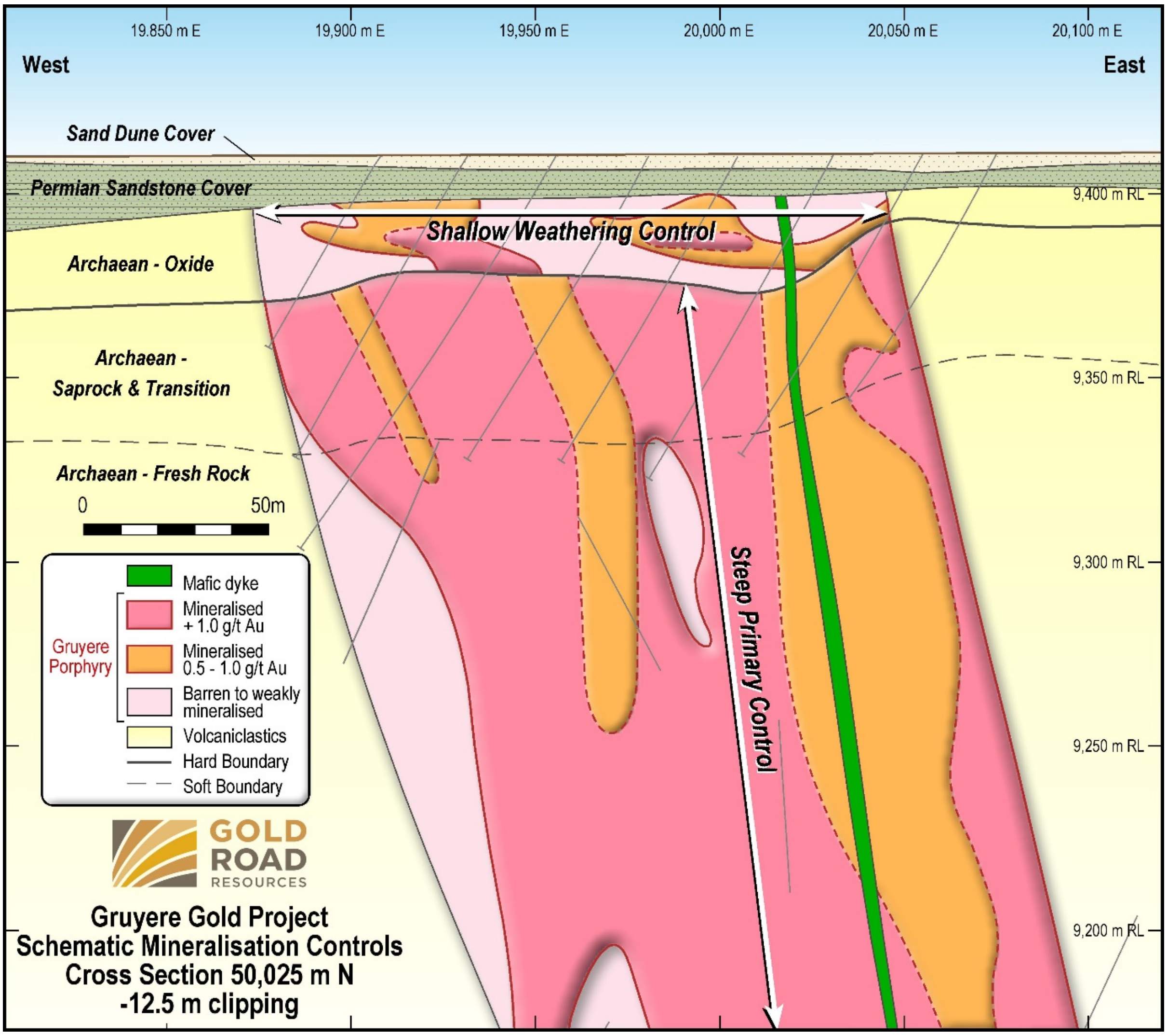

Figure 7-4: *Schematic Cross Section - Mineralisation Oxidation Zones (Gruyere Grid)*

The Fresh (primary) zone is hosted in the Gruyere Porphyry and exhibits steep easterly dipping mineralisation. The strike extent of the Gruyere mineralised system appears to be co-incident with the interpreted cross-cutting faults. The intersection of the Alpenhorn Fault with the Gruyere Porphyry defines a steep plunge, considered the gross plunge of the system. The higher-grade zone at the northern end of the deposit is associated with the Northern Fault and is characterised by: stronger and more ductile deformation mineralisation across the full width of the porphyry, lack of internal mafic units; and higher density of quartz veining. The main northerly strike trend is interpreted to be parallel to foliation while the steep easterly dip follows the crenulation of the foliation. Mineralisation shows very high continuity in these orientations.

A shallow south plunging shoot control is observed on a local and gross scale and relates to a number of geological features:

- Relationship to the intersection of the tabular quartz vein set and foliation orientations from diamond structural data
- Trends defined by alteration and other geological features, such as interpretation of higher grades corresponding to higher intensity alteration, sulphide zonation patterns, mineral mapping by CSIRO showing detailed distribution patterns of sulphides, white micas and iron-oxide, and distribution of high-density of quartz veining and increased deformation.

8 DEPOSIT TYPES

The Gruyere gold deposit is an Archaean orogenic gold deposit. This deposit type is widespread in the greenstone belts of the Yilgarn Craton and in other greenstone belts around the world including in Canada, Africa and India.

Archaean orogenic gold deposits share a number of similar geological characteristics including location in greenstone belts, strong structural control of orebodies, relative timing of gold mineralisation with respect to peak metamorphism, consistent metal association and broadly uniform hydrothermal fluid chemistry. However individual deposits display a diverse range of depositional site characteristics including host lithologies, structural setting, alteration and mineralisation styles, and various aspects of fluid and ore chemistry (oxidation state, gold fineness) (Reference 3).

Most orogenic gold deposits are hosted in mafic-ultramafic extrusive and intrusive rocks whereas the Gruyere deposit is hosted in a granitoid. As the Gruyere deposit is a recent discovery and the first major deposit in the Yamarna area, insufficient studies on the mineralised system have been completed to indicate how closely Gruyere compares with major orogenic gold deposits in other locations such as the better known Kalgoorlie Terrane in the central Yilgarn Craton.

The scale and continuity of mineralisation in the Gruyere deposit makes it unusual to other similar style deposits in the Yilgarn; the large mineralised volume is due to the brittle-ductile failure of the porphyry body, preconditioned by first phases of albite alteration. The ubiquitously altered quartz monzonite porphyry is fractionated; making it unusual compared to typical porphyry hosted gold deposits, with a distinctive negative Eu-anomaly.

Gold mineralisation at Gruyere is characterised by varying intensity albite-sericite-chlorite-biotite-calcite alteration, with associated pyrite-pyrrhotite disseminations, and coarse arsenopyrite proximal to high grade zones. Grade commonly increases with the intensity of albite-sericite-chlorite-biotite-calcite alteration; high-grade zones are commonly overprinted with limited porphyry textural retention. Intense alteration and pyrrhotite>pyrite+arsenopyrite mineralisation is often observed proximal to sheared, recrystallised south-east plunging quartz veins.

Lower grade gold mineralisation (commonly < 0.3 g/t Au) within the porphyry is characterised by reddened (hematite dusted) albite-quartz-muscovite-biotite-oligioclase-magnetite alteration with only minor pyrite disseminations. Visible gold is commonly observed within brittle-ductile chlorite±pyrite bearing fractures, which are common throughout the porphyry.

While fractionated gold-only porphyry analogues are uncommon, the Canadian Malarctic deposit, hosted along the Cadillac Fault Zone – Abitibi Greenstone Belt, is similar in host lithologies, intrusive lithology, mineralised volume and primary structural features. Both deposits are hosted on major shear or fault zones along secondary dextral events, and include similar intrusive hosts - quartz monzonite porphyry (Gruyere) and quartz monzodiorite porphyry (Canadian Malarctic), intruding into volcanic/sedimentary country rock sequences.

Mineralisation within the Canadian Malarctic is primarily hosted in the host Pontiac group clastic sediments, and with the remaining mineralisation within the quartz monzodiorite porphyry, whereas mineralisation at Gruyere is entirely hosted within the quartz monzonite porphyry. Porphyry geometries vary, with the Malartic porphyry showing multiple dykes extending from a deeper pluton-stoping up into the Pontiac sediments, while the Gruyere porphyry has intruded the host volcanics as a singular intrusive, possibly as a dyke in a higher relative position to a deeper pluton.

9 EXPLORATION

9.1 Regional Deposit Targeting

In 2012 Gold Road completed a detailed 50 metre line-spaced airborne magnetic and radiometric survey totalling 70,000 line-kilometres and covering its entire 5,000 km^2 tenement holding. This dataset was used in combination with other datasets in order to carry out a regional deposit targeting program.

Additional datasets used included regional geology, gravity, aeromagnetics and Aster satellite imagery over the Yamarna greenstone belt and other greenstone belts in the Yilgarn Craton. The regional data was required to compare the signatures of the large gold deposits in the well-established gold belts of the Yilgarn to the signatures identified on the Yamarna Belt.

Gold Road took a mineral systems approach to the regional targeting, combining multiple datasets and multi-scale concepts to identify discrete Camp-scale targets capable of hosting multi-million ounce gold systems. This approach integrated geological concepts at the regional, district and local prospect scales.

The targeting scales used relate to different geodynamic zones or depths, respectively from lower crust/upper mantle, to upper crustal, and finally to the near-surface environment (<2 kilometres). The primary fluid sources and/or driving forces for large deposits were interpreted to be related and controlled by lower crustal to upper mantle characteristics that generally manifest only as subtle near-surface features or lineaments in most conventional datasets.

The data and concepts utilised by Gold Road included the following:

- Open file data on the Yilgarn Craton
- New detailed aeromagnetic data over the Gold Road tenements
- Regional-scale concepts including identification of major regional high-strain shear zones, inverted rift axial structures, inverted syn-rift transfer fault intersections, and antiformal culminations or domes
- District-scale concepts based on a redox targeting method focussing on contacts of oxidised iron-rich magnetic units with reduced non-magnetic units
- Prospect-scale (5 to 50 km^2 areas) structural targets were generated from the combined datasets.

A total of 40 discrete prospect-scale targets were identified and ranked based on identification of dilational structural sites, competency contrast in stratigraphy, and magnetic destruction or alteration features coincident with cross-cutting faults.

Combining the compiled data and results from all three targeting scales identified 10 Camp-scale targets within Gold Road's tenements. The Camp targets were further ranked into six requiring immediate testing (within two years) and four warranting longer term testing. The first Camp that was drill tested was the South Dorothy Hills Camp which yielded the Gruyere discovery within three months of starting the new regional exploration program.

10 DRILLING

10.1 Drilling Programs

Gold Road completed a total of 87,066 metres of drilling at Gruyere during the period September 2013 to December 2015. Drilling was conducted over seven separate drilling programs, consisting of 357 RC holes, 73 pre-collared RC holes with diamond core tails and 40 fully cored diamond holes. A summary of drilling is shown in Table 10-1.

Table 10-1: Summary of Gruyere Resource Drilling Physicals (RC and DDH)

Gruyere Resource Drilling Physicals				
Hole Type	No of Holes	RC metres	DDH metres	Total metres
Reverse Circulation (RC)	357	41,264		41,264
DDH with RC Pre-collar	73	14,694	16,506	31,199
DDH only	40		14,603	14,603
Total	470	55,958	31,109	87,066

The first program of seven RC holes in September 2013 confirmed the initial intersection of gold mineralisation in previous RAB drilling. The maiden resource estimate in August 2014 was undertaken after another three drilling programs brought the total drilling to 38,000 metres. This was increased to 67,665 metres with three additional programs by September 2015 when a resource update for the PFS was completed.

The final drill program consisting of close spaced (25 by 25 metres) grade control type RC holes was completed in the December 2015 quarter for use in a further update of the resource model which formed the basis for the FS.

Drill sections are oriented west to east (270° to 90° Gruyere Grid) with the majority of holes oriented -60° to 270°. Thirteen holes in this orientation are shallow to dip and four are steep to dip as shown in Table 10-2. A small component of holes has been drilled to the north of which five are deep DDH holes drilled along the strike of the deposit (-60° towards 10°) to specifically test along strike continuity. Other orientations tested are to the north-east, east and to the south.

The general drill direction of -60° to 270° is approximately perpendicular to the main alteration packages and important quartz vein orientation and is a suitable drilling direction to avoid directional bias.

Table 10-2: Summary of Gruyere Resource Drilling Orientation Data - (holes used for resource estimation only)

Azimuth (Gruyere Local Grid)	Dip	DDH	RC	Total	Comment
250 to 290	-40 to -50	7	7	14	Perpendicular to strike and shallow to dip
250 to 290	-51 to -75	69	291	360	Perpendicular to strike and dip
250 to 290	-76 to -85	2	2	4	Perpendicular to strike and steep to dip
291 to 020	-55 to -70	11		11	Along strike/down dip - includes 1 wedge
021 to 100	-60 to -80	12	14	26	To north-east and east
101 to 249	-60 to -70	2	4	6	To south
na	-90		2	2	Water bores
	Total	103	320	423	

Drilling at Gruyere extends for approximately 2,800 metres north-south with the main 1,800 metres long zone of mineralisation drilled on a 100 metre section spacing to a depth of 600 metres below surface. Drill holes on the 100 metre sections are generally 40 metres apart in the upper 400 metres and approximately 100 metres apart below 400 metres. Additional intermediate 50 metre sections have been drilled with at least one to two holes per section over the upper 300 metres, proving good continuity of both geology and gold mineralisation between the 100 metre sections. Approximately 75% of the strike length and 100 metres of depth has been drilled to 25 by 25 metres and includes a 100 metre zone drilled to 12.5 by 25 metres spacing in the centre of the deposit. RC drilling dominates in the upper 100 metres with diamond drilling the dominant method below this depth.

The drill locations in plan and on a longitudinal section are shown in Figure 10-1 and Figure 10-2 respectively. Gaps in the drilling noted in these Figures reflect the areas that could not be accessed due to the presence of sand dunes which restrict or prevent drilling access (two gaps of 50 metres and two of 100 metres). These areas will be drilled to grade control drill density when the sand dunes are removed as part of the Project development pre-strip.

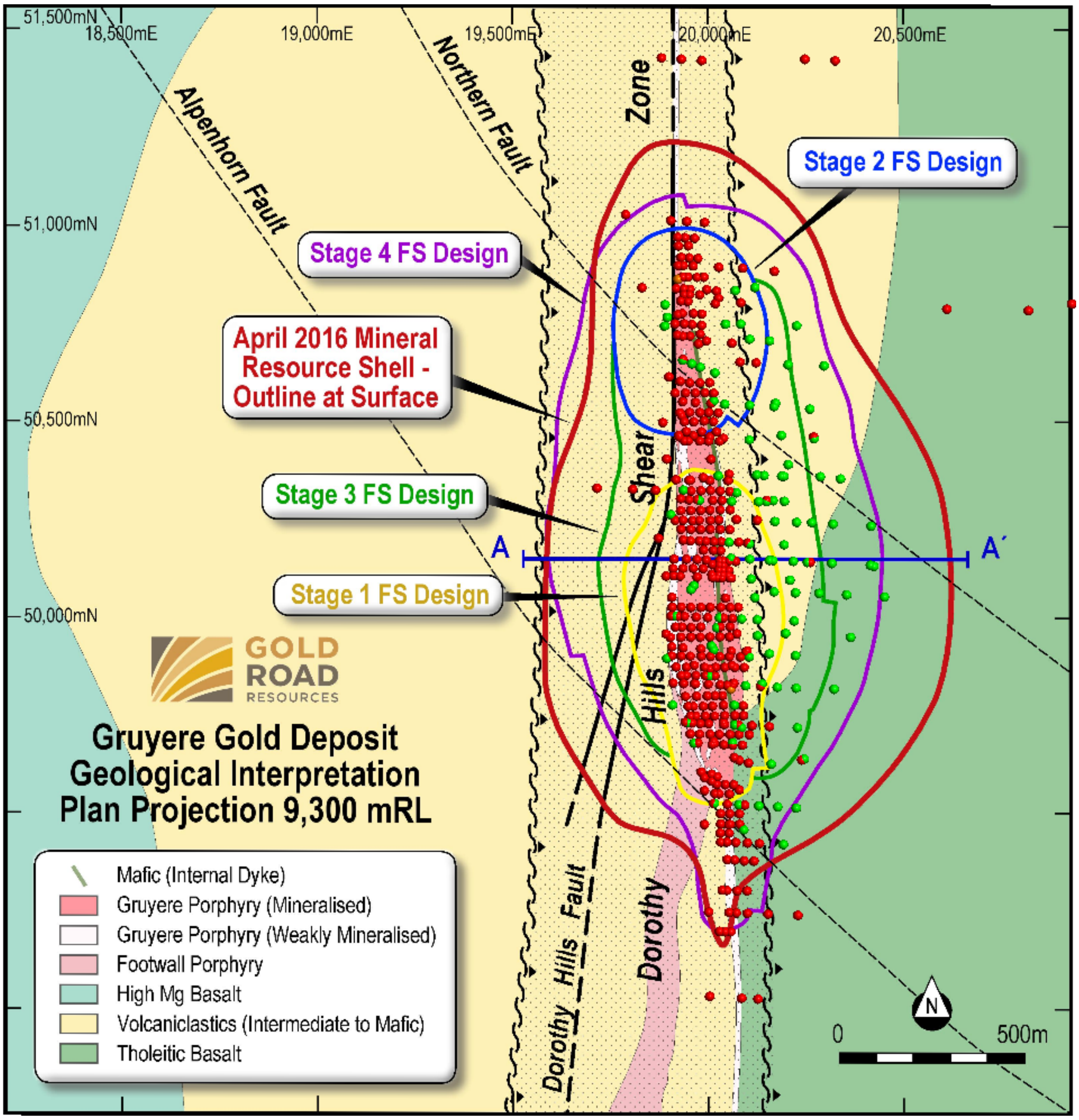

Figure 10-1: *Drill Hole Location Plan - Geological Interpretation at 9300 mRL (Gruyere Grid)*

Note: *Red dots are RC drill holes, green dots are diamond drill holes, orange dots are waterbore holes (RC)*

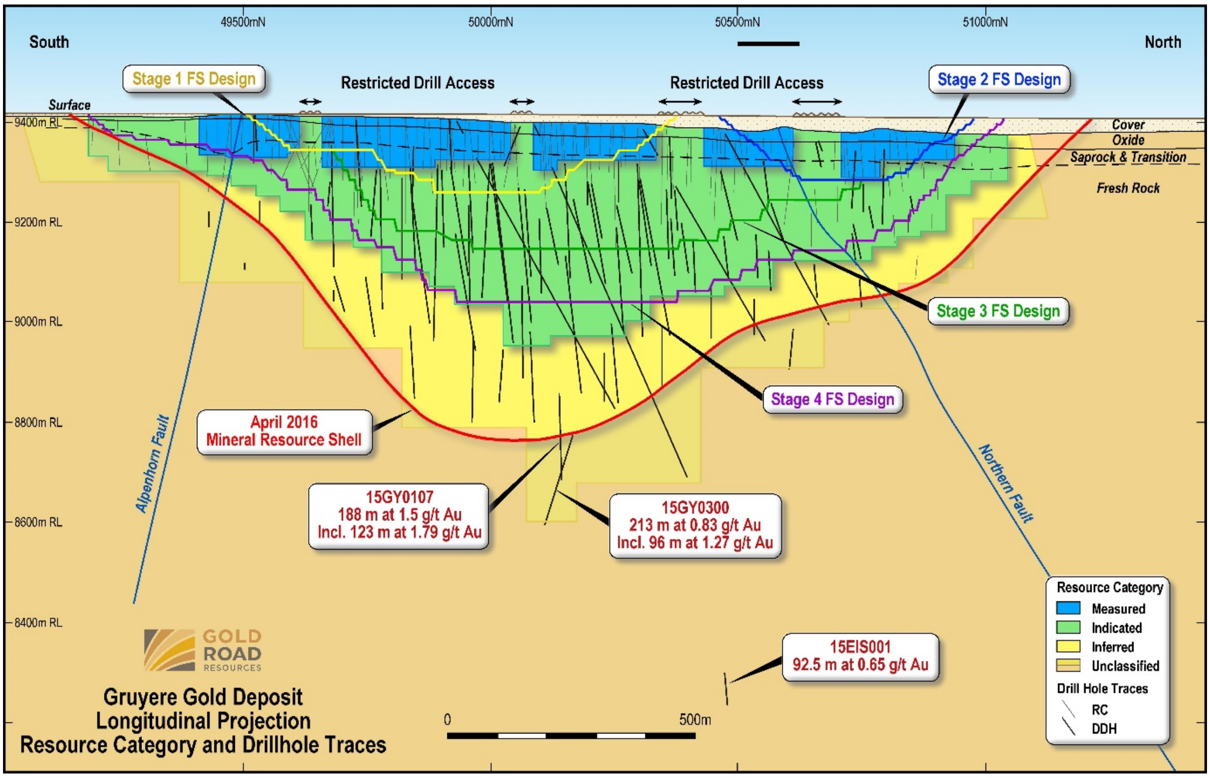

***Figure 10-2**: Drill Location Long Section looking West (Gruyere Grid)*

10.2 Sampling Procedures

All RC holes were drilled with a 5.25 inch face-sampling bit, with 1 metre samples collected through a cyclone and cone splitter, to form a sample mass of 2-4 kg. Sample recoveries are recorded as a percentage and no significant sample loss was noted in any part of the drill program. Recovery of the samples was good, generally estimated to be close to 100%, except for some sample loss at the top of the hole. All assays derived from RC drilling used in the Mineral Resource are based on the original 1 metre sample intervals collected during operations.

A total of 27 RC holes from the early drilling in 2013/2014 featured compositing over waste intervals. None of these composited samples have been used in the Mineral Resource estimate. No compositing has been employed in the diamond drilling and no sample compositing has been used during reporting; all reported intersections represent full length weighted average grades across the intersection length.

The majority of RC samples were dry. Ground water egress occurred in some holes at variable depths from 100 to 400 metres. Drill operators ensured that water was lifted from the face of the hole at each rod change to ensure that water did not interfere with drilling and that all samples were collected dry. When water was not able to be isolated from the sample stream the drill hole was stopped and drilling was completed with a diamond tail.

DDH holes were drilled at predominantly NQ core size with 40 holes drilled from surface utilising HQ diameter core to the top of fresh rock and 73 holes utilising a component of RC drilling to complete pre-collars through hanging wall waste zones before commencing with NQ core drilling. Drill operators measure core recoveries for every drill run completed using a 3 metre barrel. The core recovered is physically measured by tape measure and the length recovered is recorded for every 3 metre drill run. Core recovery is calculated as a percentage recovery. Close to 100% recovery was achieved for the majority of diamond drilling completed at Gruyere.

Core is oriented using downhole Reflex surveying tools, with orientation marks provided after each drill run.

Sampling of diamond core was based on regular 1 metre intervals or occasional smaller intervals cut to discrete geological contacts. The core was cut in half for both NQ and HQ core diameters to produce a mass of 3-4 kg per sample, with the remaining half core retained on site for reference.

10.3 Survey

Drill Hole Surveys

The majority (97%) of drill hole collar locations have been surveyed using a Differential Geographical Positioning System (**DGPS**) with final collars located to one centimetre accuracy in elevation.

Drillers use an electronic single-shot camera to take dip and azimuth readings inside the stainless steel rods, at 50 metre intervals, prior to August 2014, and 30 metre intervals, post August 2014. Downhole directional surveying using a north-seeking gyroscopic tool was completed on site and live (down drill rod string) or after the rod string had been removed from the hole. Most DDH holes were progressively surveyed live whereas most RC holes were surveyed upon exiting the hole.

Additional down-hole surveys were completed to collect physical rock property data, including density and magnetic susceptibility and optical and acoustic televiewer surveys. The additional geotechnical and structural geological data was used in the construction of the geological models.

An Aerial Lidar and Imagery Survey covering a 2,558 km^2 area including the Gruyere deposit and the Project's main mining infrastructure was completed in January 2016 by Trans Wonderland Holdings as part of the ongoing FS. One metre contours from this survey were used to construct a new topography surface to constrain the resource model. The survey showed good agreement with the existing DGPS drill hole collar data.

Australian Map Grid

Gold Road utilises the standard map projection used in Australia which is the Map Grid of Australia (**MGA94**). The Gruyere Project is located in Zone 51 of the Universal Transverse Mercator (**UTM**) grid system. The MGA94 grid is used in conjunction with a local grid which was established with its north-south grid orientation in the same direction as the strike of the Gruyere deposit to assist with geological evaluation.

Local Survey Grid

A local grid (Gruyere Grid) was established to create an accurate and practical co-ordinate system aligned along the strike of the deposit. The local grid was established by survey contractor Land Surveys Pty Ltd (Land Surveys) using geodetic DGPS units in rapid static mode and post processed with Trimble Business Centre (**TBC**) software.

The origin of the Gruyere Grid is Northing 50,000.000 and Easting 20,000.000. The MGA94 Zone 51 equivalent is Northing 6,904,145.935 and Easting 583,541.290. A permanent survey mark (PGU003) consisting of a steel picket set in concrete was installed at this point. Two additional permanent survey marks were installed along the north-south baseline, either side of the origin. An accurate transformation between Gruyere Grid and MGA94-51 was established by Land Surveys.

The Gruyere Grid Northing baseline is set at 340° 00′ 00″ to MGA94 and therefore approximates the strike direction of the deposit. For Australian Height Datum (**AHD**) elevations, 9,000 metres was added to the AHD elevations in the Gruyere Grid to avoid the possibility of negative values in potential underground operations. The AHD RL for survey mark PGU003 is 409.809 metres and the Gruyere Grid RL is 9,409.809 metres.

10.4 Logging

Logging of RC chips records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples. All samples are wet-sieved and stored in numbered and labelled chip trays for future reference. Logging of diamond drill core records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples, along with structural information from oriented drill core. All samples are stored in numbered and labelled core trays for future reference.

Geological logging and sampling data is collected in the field using LogChief software and uploaded digitally. The software utilises lookup tables, fixed formatting and validation routines to ensure data integrity prior to upload to the central database.

All core is photographed in the trays, prior to cutting, with individual photographs taken of each tray both dry and wet. Photos are uploaded to and stored in the Gold Road database. Hand held or Field Portable X-ray fluorescence (**FPXRF**) devices are used for indicative identification of litho-geochemistry and alteration to aid logging and subsequent interpretation. Calibration of the FPXRF tools is completed at start-up before the commencement of any data capture.

A weathering profile guide is used as a reference for logging and mapping activities and to provide clear definition of the different material types; this system is aimed at achieving consistency of logging from the geological team.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

All Gold Road's sampling and analytical techniques are industry-standard and have been implemented since the initial RC drilling program in September 2013. Only minor changes and improvements have been made since that time.

11.1 Sample Preparation

Sample preparation for Gold Road's Gruyere drill samples is carried out at the Intertek Genalysis Sample Preparation Facility in Kalgoorlie, Western Australia.

Drill samples were oven dried and the whole sample (2 to 4 kg) pulverised to 80% passing 75 µm. A sub-sample of approximately 200 g was retained and a nominal 50 g was used for gold analysis.

11.2 Sample Analysis

Prepared sample pulps were analysed for gold at the Intertek Genalysis Laboratory (Intertek) in Perth, Western Australia.

Samples are analysed for total gold using a 50 g Fire Assay with Inductively Coupled Plasma Optical Emission Spectrometry (**ICP-OES**) finish which has a detection limit of 0.005 ppm gold. Prior to May 2014 a Fire Assay method with an AAS (Atomic Absorption Spectroscopy) finish was used. Fire Assay with either AAS or ICPES finish for gold is considered to be appropriate for the Gruyere material and mineralisation. The method gives a near total digestion of the gold in the sample.

During 2013/2014, Gold Road submitted 675 samples for LeachWELL™ analysis which provides a total gold value and an approximation of gold recovery. The method uses a larger sample mass (400 to 1,000g) which is effective in capturing potential coarse gold in the sample. Samples are leached for 24 hours with the resulting leach solution then assayed for its dissolved gold content by AAS or ICP-OES techniques. The remaining pulp material is washed and reground, and an additional fire assay is completed on a representative 50g sample (with AAS or ICP-OES finish) to determine the unleached gold content, which is approximately representative of the unrecoverable gold, or tail, in the sample. A combination of the two assay results (leach plus tail) represents the total gold grade, and an approximation of gold recovery is represented by the proportion of leachable gold compared to the total gold grade.

The LeachWell samples were re-assayed by Fire Assay (ICPES finish). Dr Paul Sauter (in-house consultant Sauter Geological Services Pty Ltd) concluded that there is no significant bias between the assay techniques, and, that Fire Assay is the most appropriate sample for resource estimation purposes.

Fire Assay results are used exclusively in the April 2016 Mineral Resource.

11.3 Sample Security

Logging, sample labelling and sample storage of RC and DDH samples take place in secure facilities on site. Samples are placed in pre-numbered calico sample bags, collected in plastic bags (five calico bags per single plastic bag), sealed and transported by Gold Road to the Intertek Laboratory in Kalgoorlie where sample preparation is undertaken. Intertek arrange sample transport for the prepared sample pulps that are sent to the Intertek Laboratory in Perth for analysis. All pulps are returned to site for storage. RC and DDH pulps and residues from analyses are returned from the laboratory after 60 days and retained on site.

11.4 Bulk Density

Bulk density is determined using two main methods: for RC drilling downhole rock property surveys are completed by ABIMS Pty Ltd which provide a density measurement every 0.1 metres downhole; for DDH drilling, a core sample is taken every metre in weathered and every 10 metres in fresh material and subjected to the water immersion method (weight in air/water) to determine bulk density.

The physical measurements derived from the air/water immersion method were compared to the down-hole tool measurements. Good correlation between RC data and the DDH core data was observed for Saprolite, Saprock and Transition material. The down-hole tool values for Fresh rock did not match the other two methods and so was set aside pending review by the provider. Bulk density values determined from DDH core samples were used for Fresh rock.

Average bulk density values determined by lithology and oxidation type. Values were modified by a moisture percentage so that dry tonnage is reported. The moisture percentages used were overburden and Saprolite 5%, Saprock 3%, Transition 2% and Fresh 1%. Average dry bulk density values are summarised in Table 11-1.

Table 11-1: Gruyere Bulk Density - Average Values from RC Logging and DDH Samples

Lithology	Oxidation Zone	Bulk Density (t/m^3)
Quaternary Cover	Oxide	1.45
Permian Sandstone	Oxide	1.70
Gruyere Porphyry	Saprolite	1.85
Gruyere Porphyry	Saprock	2.45
Hanging Wall Volcaniclastics	Saprock	2.35
Gruyere Porphyry	Transition	2.50
Hanging Wall Volcaniclastics	Transition	2.60
Gruyere Porphyry	Fresh	2.65
Hanging Wall Volcaniclastics	Fresh	2.90

11.5 Quality Control/Quality Assurance Procedures

QA/QC Protocols

Gold Road observes standard QA/QC protocols for all drilling programs including routine submission of Field Standards (Certified Reference Materials), Blanks, and Field Duplicates. These QA/QC samples are inserted as blind samples within each dispatched drill sample batch.

The Gold Road QA/QC protocols have been in place since the initial RC drilling program undertaken in September 2013.

Protocols for RC and DDH drilling consist of Field Standards and Blanks inserted at a rate of 3 Standards and 3 Blanks per 100 samples. RC Field Duplicates are generally inserted at a rate of approximately 1 in 40. The RC duplicate field sample is taken from the cone splitter at the same time as the primary sample. DDH Field Duplicates in the form of half core samples are also inserted at a rate of approximately 1 in 40.

In addition, the contracted laboratory Intertek has its own internal QA/QC protocols. Intertek QA/QC protocols include analysis of Repeats, Laboratory Standards, Checks and Blanks.

QA/QC Data Review

An independent review of QA/QC data for each major drill program and associated resource update have been completed.

Mr David Tullberg (Tullberg) of Grassroots Data Services Pty Ltd (GDS) reviewed the QA/QC data from the drill hole and assay database used for the maiden Mineral Resource estimate in August 2014.

Dr Paul Sauter (Sauter), an in-house consultant from Sauter Geological Services Pty Ltd reviewed the QA/QC data from the drill hole and assay database used for the May 2015, September 2015 and April 2016 Mineral Resource estimates.

QA/QC Review for Maiden Mineral Resource Estimate - August 2014

Total sample submission for this Mineral Resource estimate was 30,810 samples including 30,135 samples for 50 g fire assays and 675 samples for LeachWELL™ analysis. The QA/QC samples consisted of 2,683 or approximately 9% of total samples, including 1,011 Field Blanks, 983 Field Standards and 689 Field Duplicates.

In addition, 841 Laboratory Blanks (including 77 Acid Blanks), 1,664 Laboratory Checks, and 1,420 Laboratory Standards were inserted and analysed by Intertek.

A total of 236 Umpire Laboratory check assays were also submitted with five Laboratory Blanks and 10 Laboratory Checks inserted and analysed by Minanalytical Laboratories in Perth, Western Australia.

Results of the Field and Laboratory QA/QC assays were checked on assay receipt using QAQCR software.

Tullberg reported that all three Standards used by Gold Road (G301-3, 1.96 ppm Au, G311-1, 0.52 ppm Au, and G998-3, 0.81 ppm Au) returned means close to the expected means. A few result outliers were recorded and in each instance Intertek was contacted to verify results. A plot of the results for Field Standard G301-3 is shown in Figure 11-1. This plot shows that one result fell outside expected limits. In this case the sample was re-checked and found to be in error; the laboratory repeated the analysis for the whole batch of samples. A plot of the results for all three standards is shown in Figure 11-2.

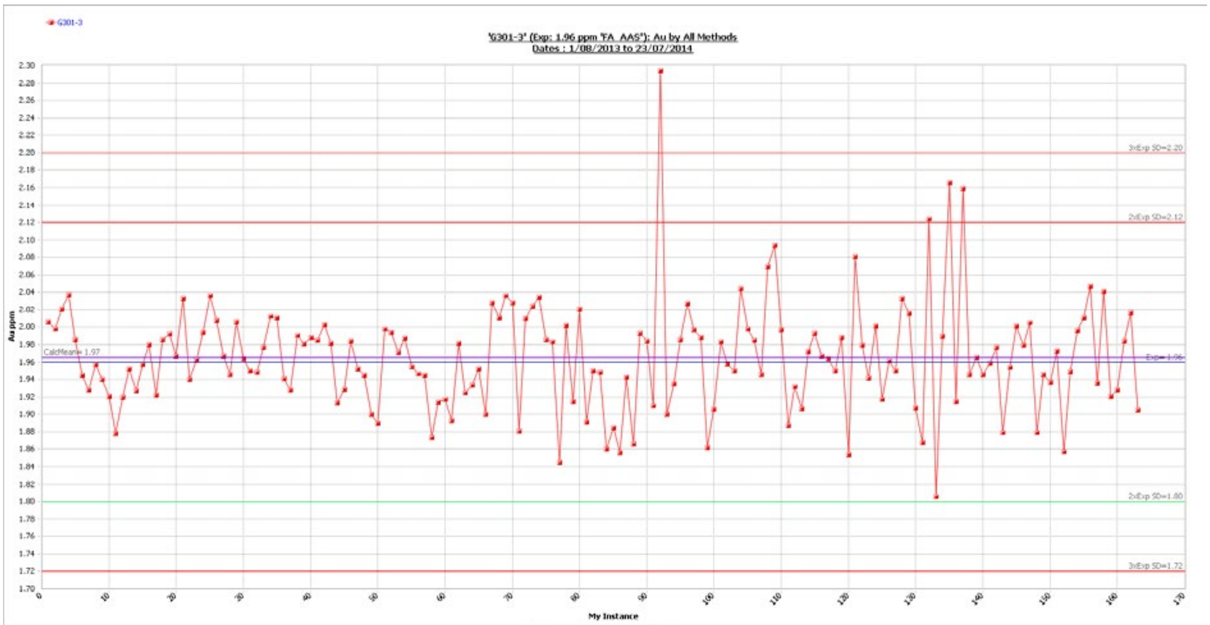

Figure 11-1: *QA/QC Data review for 2013-2014 Drilling - Plot of Field Standard G301-3*

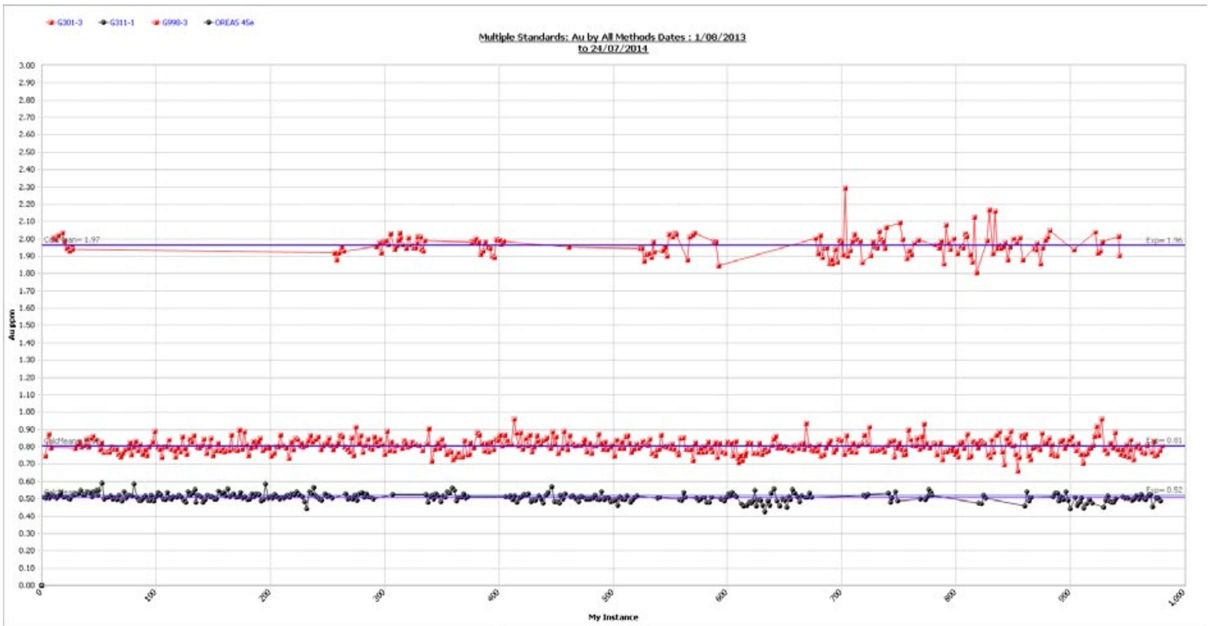

Figure 11-2: *QA/QC Data Review for 2013-2014 Drilling - Plot of Field Standards G301-3, G311-1, G998-3*

Three Blanks (two Laboratory Blanks and one Field Blank) were used during the drilling. Both the Laboratory Blanks performed as expected. The Field Blank used by Gold Road for the most part returned results at or near the expected value however a number of results showed unexpected grade. Follow up test work by Intertek indicated that the Field Blank may carry some gold and needed to be reviewed.

Both RC and DDH Field Duplicates were taken in the course of the drilling programs with RC Duplicates taken via a cone splitter and DDH Duplicates as half core. Tullberg considered the RC Duplicates to be true duplicates but the DDH Duplicates strictly as Field Replicates. Correlation studies on the RC original and duplicate samples indicated a correlation coefficient (R^2) value differing considerably by fire assay analytical finish method, between 0.51 for ICP-OES and 0.84 for AAS. AAS was replaced by ICP-OES as the preferred method by the laboratory in April 2014. Tullberg considered the low correlation R^2 value could be due to poor sampling procedures but more likely to be related to the sample homogeneity and location/nature of the gold particles particular to the Gruyere deposit. Tullberg recommended Gold Road undertake umpire test work on duplicate sampling.

QA/QC Review for Mineral Resource Estimate - May 2015

Drilling completed in the June 2014 to April 2015 period was added to the Gruyere database for the May 2015 resource update. Sauter carried out two reviews of the QA/QC data generated from this drilling; the first review in April 2015 was of the results of 300 Umpire sample pulps (RC 75, DDH 225) analysed by ALS Laboratories (ALS), and the second review completed in May 2015 was of the QA/QC data for Field Standards (222), Field Blanks (222) Field Duplicates (307) and Laboratory Standard and Check samples (597).

Sauter plotted the 300 Umpire pulps analysed at ALS as X-Y plots and used measures of the repeatability expressed as the percentage of the mean absolute paired difference (%MAPD) including the mean, 2 standard deviations, and percentage passing 30%MAPD (Figure 11-3). Low %MAPD is considered to reflect better accuracy and low 2SD MAPD to reflect better precision. Sauter reported that there does not appear to be a bias between the two laboratories (apart from a few outliers in the DDH results), however there is significant scatter in the results. This applies especially to the DDH samples, resulting in only 64% of DDH pairs passing 30%MAPD and a low R^2 value of 0.59. Sauter suggested that the DDH results could be as a result of differences in material handling, resulting in an uneven gold distribution even in the -75 µm pulps. Sauter recommended further investigation of the issue including particle size passing -75 µm.

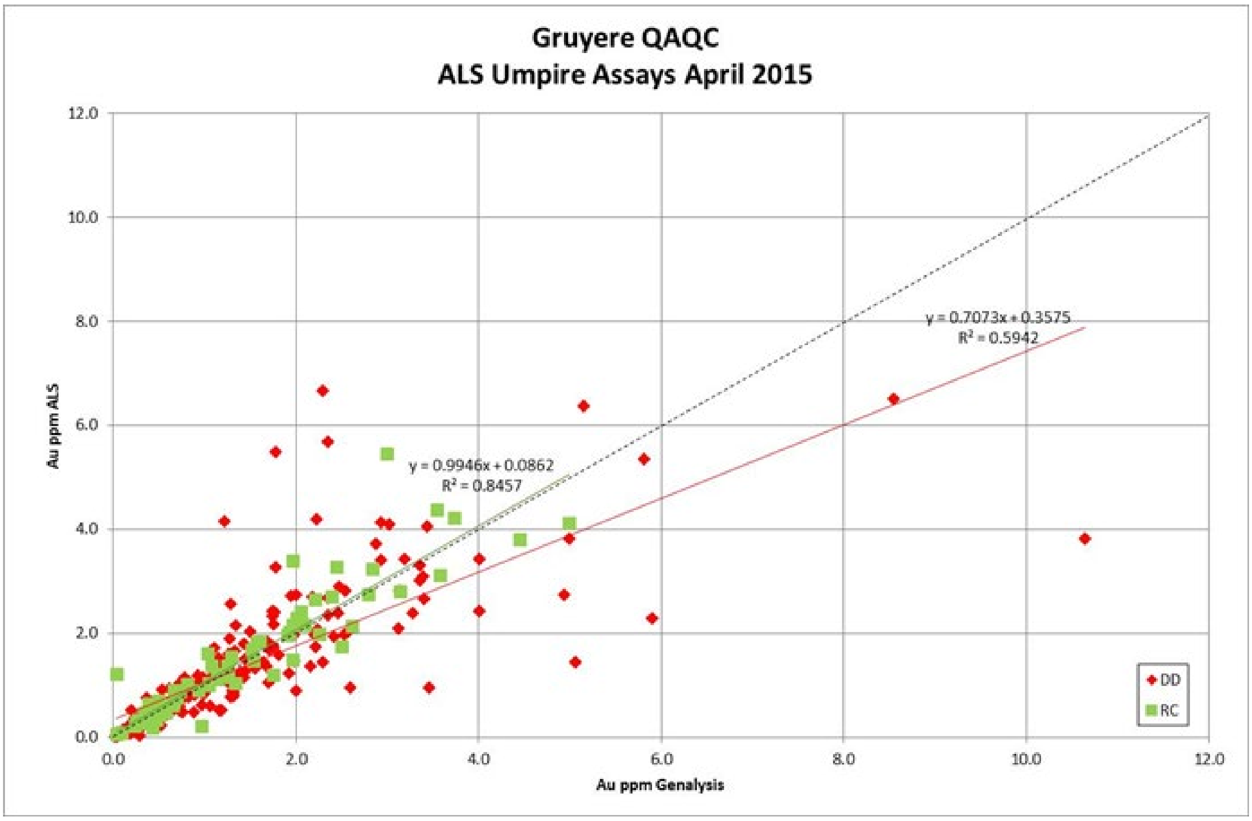

Figure 11-3: *QA/QC Data Review for 2014-2015 Drilling - Plot of Intertek Genalysis Original Pulps and ALS Umpire Pulps*

Sauter reported that all Field Standards behaved well with most results within the 2SD limits. Of the 222 Field Blanks, four samples returned results higher than 0.05 ppm Au. These results were not considered by Intertek to be blank/standard mix-ups and therefore warranted further investigation. Results for Field Duplicates were poor, especially for DDH samples with only 40% of DDH pairs passing 30%MAPD and a low R^2 value of 0.39. RC repeatability was poorer at lower grades, whereas with DDH samples the poor repeatability was fairly constant through the entire grade range.

Results of Laboratory Checks (pulp checks in the same batch) were better, but showed a significant difference between lower and higher grade samples, as well as between RC and DDH samples. The latter suggested that there was still inhomogeneity in the DDH pulps, which was also the case in the ALS Umpire assays. RC passing 30% MAPD was 74% and DDH was 58% and R^2 values 0.97 and 0.90 respectively.

Results of Laboratory Repeats (pulp checks in different batches), behaved better than the Laboratory Checks with RC 85% and DDH 63% passing 30%MAPD. However, there was still a significant difference between RC and DDH.

Sauter concluded that the difference between RC and DDH samples seen in the Field Duplicates, ALS Umpire analysis and the Intertek Laboratory Check and Repeats was not well understood but suggested these results reflected a significant amount of gold inhomogeneity at the primary sample level, possibly as a result of the presence of coarse gold at the Gruyere deposit. Sauter recommended further investigation including looking at the particle size distribution in the pulp samples.

QA/QC Review for Pre-Feasibility Study Mineral Resource Estimate - September 2015

The resource update in September 2015 included drilling results for the period April to June 2015. Sauter reviewed the QA/QC data from this drilling in August 2015, including results for 121 Field Standards, 121 Field Blanks, 167 Field Duplicates and Laboratory Standards, Check and Repeat samples totalling 481 samples.

Sauter reported that results for the Field Standards and Field Blanks were acceptable except for a downward trend in Field Standard G998-3 towards the end of the period which should be monitored. All Field Blanks were below 0.05 ppm Au, with 19 results (16% of 121 Field Blanks) above the detection limit of 0.005 ppm Au. Sauter recommended that the current dune sand blank material being used by Gold Road be replaced with a different material for testing of contamination during pulverization.

The RC and DDH Field Duplicates were again poor and comparable to previously reported results; RC and DDH passing 30%MAPD were both 37% with R^2 values of 0.57 and 0.47 respectively. Laboratory Checks were acceptable with little difference between RC and DDH samples. RC and DDH passing 30%MAPD were 79% and 73% respectively with R^2 values of 0.96 and 0.88 respectively. An X-Y plot of the Laboratory Checks is shown in Figure 11-4.

Sauter reported a significant spread, and a negative bias, in the Laboratory Repeats. Rp1 Repeats were carried out by the laboratory on higher grade samples, so were not random, and results were often lower-grade than the high-grade original assay. For most of the larger variations, a second Repeat (Rp2) was assayed (19 out of 110 Rp1 assays), which generally confirmed the lower-grade Rp1. This variability again suggests the presence of coarse gold, although its effects are more pronounced in this dataset than in previous data.

Sauter also reported on the checks carried out on the percentage passing -75 µm particle sizing as an explanation as to why results for RC and DDH Field Duplicates previously differed. Percentage passing -75 µm was shown to be similar for RC and DDH pulps at Intertek, 89% and 90% passing respectively, and well in excess of the minimum protocol of P80 -75 µm. Sauter concluded that the differences in Field Duplicate results was likely due to the presence of coarse gold at Gruyere.

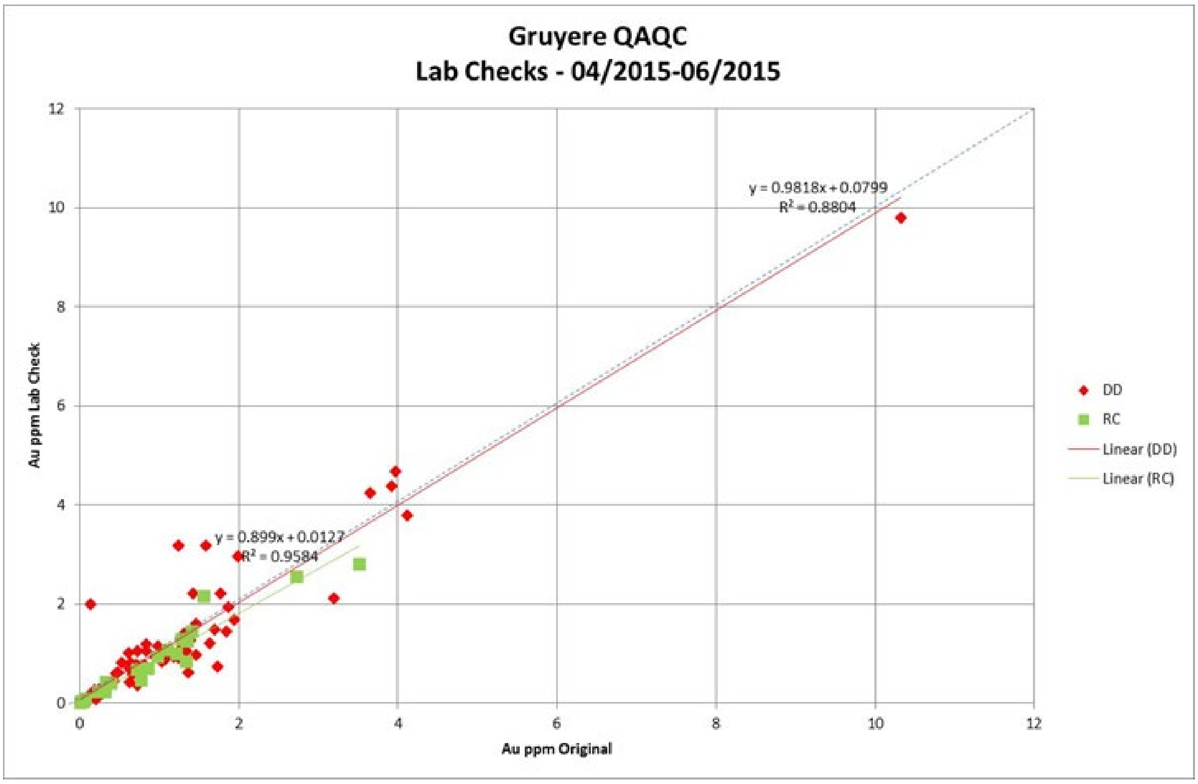

Figure 11-4: *QA/QC Data Review for 2015 Drilling - Plot of Intertek Genalysis Laboratory Check Pulps*

QA/QC Review for Feasibility Study Mineral Resource Estimate - April 2016

The Mineral Resource estimate for the Feasibility Study in April 2016 included the close spaced grade control drilling completed in September and October 2015. Sauter reviewed the QA/QC data from this drilling in December 2015, including results for 404 Field Standards, 403 Field Blanks, 335 RC Field Duplicates and Laboratory Standards, Check and Repeat samples totalling 862 samples.

Sauter reported that results were generally acceptable and an improvement on previous results. Recommendations included further Umpire laboratory testing and changing the Field Blanks to a more appropriate material.

Results for the two Field Standards used in this drilling program showed slight negative bias however overall Sauter considered results comparable to previously reported data. Sauter recommended the use of a third Field Standard, as previously, but with a grade of around 1.2 to 1.3 ppm Au. Field Blanks showed a higher percentage of results above detection limit, 33% compared with the previous 16%. Sauter again recommended that a replacement material for the dune sand Blanks be found such as unmineralised quartz.

Results for RC Field Duplicates improved compared with previous results, with 60% passing 30%MAPD and a R^2 value of 0.79. Laboratory Standards, Check and Repeats were acceptable and comparable to previously reported results. An X-Y plot of the RC Field Duplicates is shown in Figure 11-5.

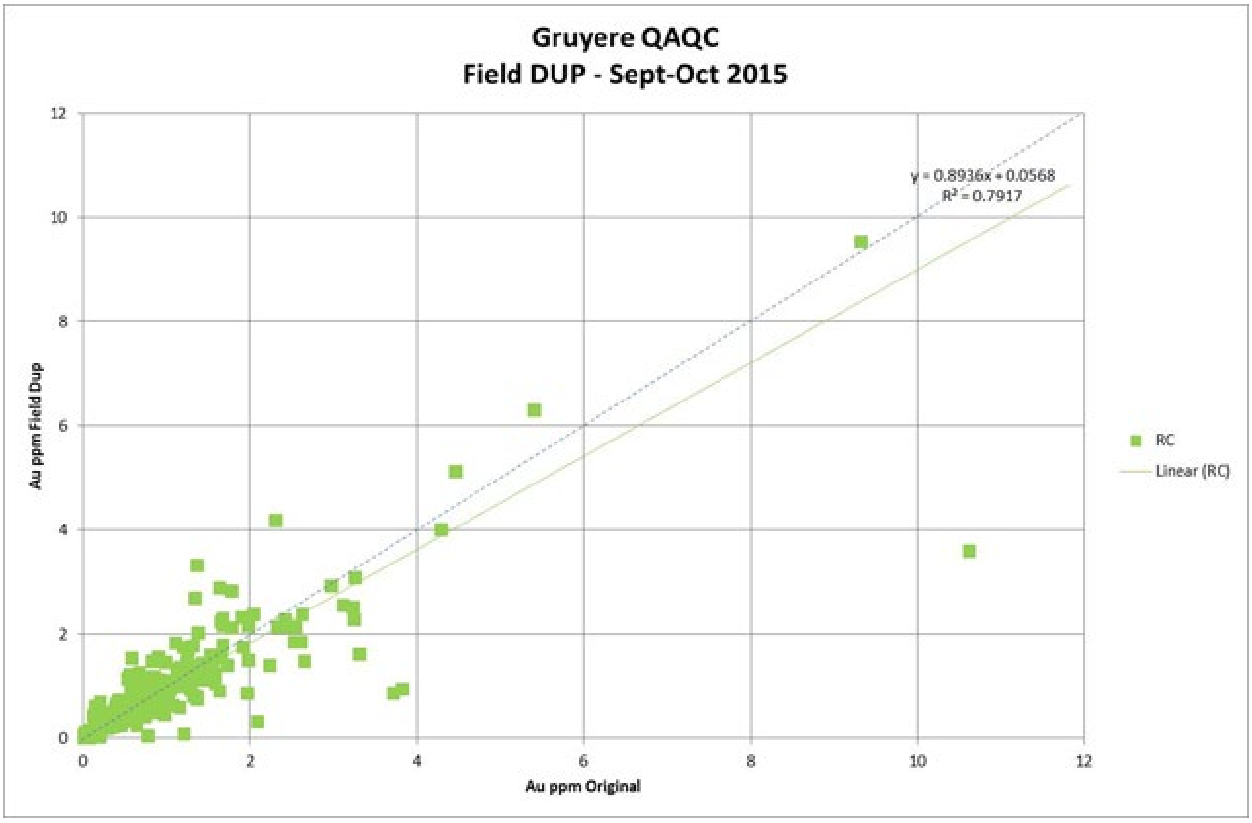

Figure 11-5: *QA/QC Data Review for Sep-Oct 2015 RC Drilling - Plot of Field Duplicate Pulps*

12 DATA VERIFICATION

12.1 Database Procedures

All field logging at Gruyere is carried out on Toughbooks using LogChief data capture software. Logging data is submitted electronically to the Database Geologist in the Perth office. Assay files are received electronically from the Intertek Laboratory. All data is stored in a Datashed/SQL database system, and maintained by the Gold Road Database Manager. The Database Manager is responsible for the integrity and efficient use of the system. Only the Database Manager or their Data Entry Clerk has permission to modify the data.

DataShed software has validation procedures that include constraints, library tables, triggers and stored procedures. Data that does not pass validation tests must be corrected before upload.

Assay data must pass laboratory QA/QC before database upload. Gold Road utilises QAQR software to further analyse QA/QC data, and batches which do not meet QA/QC criteria are requested to be re-assayed. Sample grades are checked visually in three dimensions against the logged geology and geological interpretation. No assay data was adjusted. The laboratory's primary Au field is the one used for plotting and resource purposes. No averaging is employed.

Drill hole collar pickups are checked against planned and/or actual collar locations. A hierarchical system is used to identify the most reliable down hole survey data. Drill hole traces are checked visually in three dimensions. The Project geologist and resource geologist are responsible for interpreting the down hole surveys to produce accurate drill hole traces.

Significant results were compiled by the Database Manager and reported for release by the Exploration Manager/Executive Director. Data was routinely checked by the Senior Exploration and Project Geologist, Principal Resource Geologist or Consulting Geologists during drilling programs.

12.2 Independent Database Verification

A formal database audit was carried out by Optiro in July 2014 prior to the reporting of Gold Road's maiden Mineral Resource estimate in August 2014. Optiro is a resource and mining engineering consulting company based in Perth, Western Australia.

Optiro's audit involved the checking of original assay, collar and downhole survey data records against Gold Road's resource database, covering approximately 10% of the Gruyere holes available at the time of the audit. Similar audits of the database were completed prior to updates of the resource model in May 2015, September 2015 and April 2016.

Optiro was also involved with auditing the Mineral Resource process, initially through a series of reviews leading up to the publication of the maiden resource figures in August 2014. This process was undertaken by Ian Glacken, Director and Principal Consultant of Optiro, who is a geologist and geostatistician with over 30 years worldwide mining industry experience. During these reviews, all aspects of the data preparation, estimation and modelling process were audited.

Optiro considered the Gruyere database to be of a high standard with respect to data collection, assay quality assurance, geological interpretation, modelling, validation and reporting.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Gold Road completed comprehensive test work programs sufficient to establish the optimal processing routes for the ore at Gruyere, and were performed on samples of mineralisation that were typical of the deposit, supporting estimation of recovery factors.

Nine separate test work programs were completed for the PFS and FS; these programs were reported on during the period November 2015 to May 2016. The objective of the additional test work carried out for the FS was to complete recommended work identified at the completion of the PFS and to expand on the PFS test work to provide an adequate level of metallurgical information for the process flowsheet and plant design.

Test work was performed by the following companies:

- ALS Metallurgy Limited (**ALS**), Perth, Western Australia
- ALS was responsible for sample preparation, mineralogy, comminution test work, gravity test work, cyanide leaching, including grind size and reagent optimisation, oxygen uptake and viscosity testing, carbon loading kinetics and variability test work. This test work was covered under seven separate ALS technical reports – A16624, A16652, A16682, A16698, A16857, A16983 and A17012.
- Gekko Systems Pty Limited (**Gekko**), Ballarat, Victoria, Australia
- Gekko carried out GRG test work, modelling and intensive leach test work, report T1408.
- Jenike & Johanson Pty Limited (J**enike & Johanson**), Perth, Western Australia
- Jenike & Johanson carried out materials handling testing, report 70356-1
- Outotec Pty Limited (**Outotec**), Perth, Western Australia
- Outotec undertook thickening test work
- Newpark Drilling Fluids (Australia) Limited (**Newpark**), Perth, Western Australia
- Newpark carried out Rheogram testing on flocculated tailings slurry from Outotec.

13.2 Sample Selection

A total of 50 representative composite samples were generated with an approximate mass of 2,446 kg. These were delivered to ALS's laboratory in Perth. Samples were collected over a large range of down hole depths as shown in Figure 13-1. The depth range was from 6 metres (Comp #88 South Oxide Median, 6 metres to 12 metres) to 411 metres (Comp #59 Central Fresh High 392 metres to 411 metres). All selected samples were from within the PFS optimised pit shell which was based on a A$1,400 per ounce gold price.

Test work samples were classified into ore types based on exploration oxidation zones (Oxide-Saprolite, Saprock, Transition and Fresh), three grade ranges (low <1.0 g/t Au, median 1.0 to 1.4 g/t Au, high >1.4 g/t Au) and from four pit locations – south, central, north and high grade north.

Composite samples consisted of the following material:

- A total of 477 kg of RC chip samples from the Saprolite and Saprock ore zones were sampled to generate four individual Saprolite and Saprock composites for testing under test work program A16857
- Fresh, half NQ diamond drill core with a total mass of 1,158 kg were sampled to generate 28 individual composites for testing under the comminution and extractive leaching test work program A16624
- Fresh, half NQ diamond drill core with a total mass of 811 kg were sampled to generate 18 individual fresh composites for testing under the comminution and extractive leaching test work program A16652

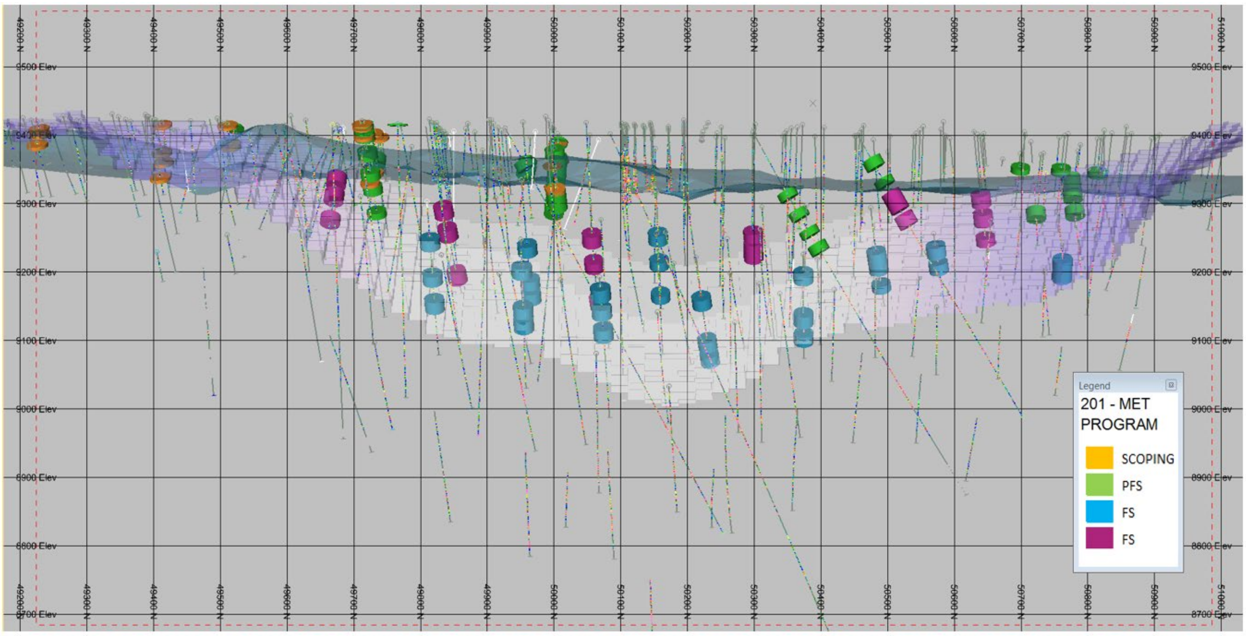

Figure 13-1: *Metallurgical Sample Selection for Gruyere - Longitudinal Section Looking West (Gruyere Grid)*

A total of 17 Master Composite samples were produced by ALS using test samples from earlier test work programs. Details of the Master Composites are shown in Table 13-1. These Master Composites or sub-samples from them were used for mineralogical analysis, oxygen uptake rate testing, and test work for bulk gravity/direct cyanidation leach, slurry rheology, carbon adsorption, gravity recoverable gold and for thickening and Rheogram testing.

Table 13-1: Gruyere Metallurgical Test Work - Master Composite Samples

Master Composite	Ore Domain	Sample Weight (kg)
Composite #1	South Fresh	80.0
Composite #2	Central Fresh	80.0
Composite #3	North Fresh	80.0
Composite #4	South Fresh	30.0
Composite #5	Central Fresh	30.0
Composite #6	North Fresh	30.0
Composite #7	HG North Fresh	30.0
Composite #8	South Fresh	23.5
Composite #9	Central Fresh	23.5
Composite #10	North Fresh	23.5
Composite #11	HG North Fresh	24.0
Composite #12	South Fresh	24.0
Composite #13	Central Fresh	24.0
Composite #14	North Fresh	24.0
Composite #15	HG North Fresh	20.5
Composite Oxide	North+South	40.0
Composite Saprock	North+South	40.0

Note: *High Grade (**HG**)*

13.3 Mineralogy

A total of 1 to 2 kg of each of six composite samples were milled to a target grind size of P_{80} 125 µm. The resulting milled samples were each separated into a gravity concentrate and a gravity tail using a 3 inch Knelson concentrator followed by hand-panning before being submitted for mineralogical investigation.

The six gravity concentrates and gravity tails were analysed to provide information on the mode of occurrence of gold as well as sulphide minerals as these are potentially important host minerals for the gold. These samples are listed in Table 13-2. The mineralogical analysis was carried out by ALS Metallurgy. The gravity concentrates were characterised by Qualitative Evaluation of Mineral by Scanning Electron Microscopy (**QEMSCAN**) while the gravity tail was analysed by X-ray Diffraction (**XRD**).

Table 13-2: Gravity Concentrate Samples Submitted for Mineralogical Analysis

Sample ID	Sample Type	Sample Description	Head Assay (Au g/t)
Sample A	North Fresh	Master Composite #10/14 - North Fresh	1.45
Sample B	HG North Fresh	Master Composite #11/15 - HG North Fresh	2.07
Sample C	South Fresh	Master Composite #8/12 - South Fresh	1.23
Sample D	Central Fresh	Master Composite #9/13 - Central Fresh	1.24
Sample E	Transition	Comp#13 Main South Transition Median	1.72
Sample F	Saprock	Saprock Master Composite (Composites #89 and #91 equal portions combined) Ex-A16857	1.91

The main sulphide minerals in the gravity concentrates derived from the four Fresh composites are pyrite, pyrrhotite and arsenopyrite. Trace amounts of galena, sphalerite and chalcopyrite were also present. The sulphide minerals were generally coarse grained (> 120 µm) and well liberated (> 70%). The major minerals present in the gravity tail for all samples are quartz and albite. Minor but variable proportions of clinochlore, mica and calcite are also present.

QEMSCAN analysis indicates that gold is present in the deposit as three grain types with varying gold fineness; the first two types, free gold and electrum, represent gold alloyed with varying amounts of silver (Ag). These two types probably form a continuous series between gold having a silver content of 0-20% and electrum containing more than 20%; silver content in electrum grains average around 25% Ag. The third type of gold present is the Ag-Au telluride (Te) group. Of the 65 grains of gold detected, the coarsest grain measured 45 µm by 20 µm. Most grains were much smaller (typically < 10 µm).

Gold present is generally fine grained with grains less than 10 µm; the coarsest grain detected measured 45 by 20 µm. The data indicated that although some of the gold was liberated at the grind size tested (P80 125 µm), most of the gold remained locked in a variety of host minerals. Pyrite was the most common host mineral with other hosts including arsenopyrite, pyrrhotite, Fe-oxides/hydroxides (mainly in Saprock and Transition samples), various silicate minerals and calcite.

The mineralogy in the Saprock and Transition composites were generally the same as for the Fresh composites except for elevated Fe-oxides/hydroxides and Mn-oxides/hydroxides, and the absence of pyrrhotite and arsenopyrite. Pyrite was the only significant sulphide present in the Saprock and Transition samples.

13.4 Communition Test Work

A total of 35 Fresh ore samples had comminution testing conducted across two campaigns at ALS. The samples selected came from a range of depths from 85 to 372 metre downhole, and from all four ore domains (South, Central, North and HG North).

Comminution tests carried out included:

- Unconfined Compressive Strength (**UCS**)
- Bond Impact Testing (Crusher Work Index) (**CWi**)
- SAG Mill Comminution (**SMC**)
- Bond Abrasion Index (**Ai**)
- Rod Mill Work Index (**RWi**)
- Bond Ball Work Index (**BBWi**)
- Ore Specific Gravity (**SG**).

The comminution test work results are summarised in Table 13-3. The comminution characteristics can be summarised as follows:

- 14 Fresh samples were tested for UCS determination and the average UCS for these samples was 155 MPa (classified as Strong). The highest sample tested recorded 258 MPa
- 11 Fresh samples were tested for CWi determination. The average CWi was 8.3 kWh/t and is classified as medium hard
- 14 Fresh samples were tested for SMC test work. The average Drop Weight Index (**DWi**) was 7.69 kWh/m^3, the average resistance to impact breakage (**Axb**) was 34.9 (85th percentile was 32.5) classified as high resistance to impact breakage and the average abrasion resistance parameter was 0.34
- 11 Fresh samples were tested for Ai determination. The average Ai of 0.5399 is classified as highly abrasive
- Seven Fresh samples were tested for RWi determination. The average RWi of the seven samples was 20.8 kWh/t. This is classified as very hard (> 20 kWh/t)
- 14 Fresh samples were tested for BBWi determination. The average BBWi was 17.3 kWh/t. This is classified as hard. The closing screen size selected for the BBWi test was 150 µm to achieve a product grind size of 125 µm
- The average SG for the fresh ore tested was 2.70.

Table 13-3: Gruyere Fresh Ore Comminution Test Work Summary

Sample ID	BHID	From m	To m	UCS MPa	CWi kWh/t	RWi kWh/t	BBWi Screen □m	BBWi kWh/t	Ai g	A	b	Axb	SG
Comp#42 South Fresh Low	15GY0091	196	213	-	-	-	-	-	0.552	-	-	-	-
Comp#43 South Fresh Median	15GY0091	254	271	216	6.3	18.9	-	-	-	-	-	-	2.73
Comp#44 South Fresh High	15GY0091	297	317	-	-	-	150	16.7	-	94.5	0.34	32.1	2.68
Comp#45 South Fresh Low	15GY0092	238	256	-	5.8	22.1	-	-	-	-	-	-	2.73
Comp#46 South Fresh Median	15GY0092	302	320	63	-	-	150	18.2	-	85.7	0.38	32.6	2.72
Comp#47 South Fresh High	15GY0092	324	344	-	-	-	-	-	0.508	-	-	-	-
Comp#49 South Fresh Median	14GYDD0065	276	295	167	-	-	150	17.0	-	94.4	0.36	34.0	2.68
Comp#51 Central Fresh Low	15GY0094	303	320	-	6.8	20.8	-	-	-	-	-	-	2.73
Comp#52 Central Fresh Median	15GY0094	336	356	102	-	-	150	17.1	-	92.0	0.38	35.0	2.73
Comp#53 Central Fresh High	15GY0094	271	291	-	5.2	20.1	-	-	-	-	-	-	2.72
Comp#54 Central Fresh Low	14GYDD0050	284	298	-	-	-	-	-	0.586	-	-	-	-
Comp#55 Central Fresh Median	14GYDD0050	179	197	258	-	-	150	17.1	-	86.8	0.42	36.5	2.68
Comp#56 Central Fresh High	14GYDD0050	225	240	-	-	-	-	-	0.582	-	-	-	-
Comp#58 Central Fresh Median	14GYDD0025	292	310	182	-	-	150	17.7	-	91.1	0.41	37.4	2.69
Comp#60 North Fresh Low	15GY0100	250	266	-	5.9	22.1	-	-	-	-	-	-	2.72
Comp#61 North Fresh Median	15GY0100	354	372	208	-	-	150	16.9	-	81.8	0.44	36.0	2.68
Comp#62 North Fresh High	15GY0100	319	339	-	5.6	21.7	-	-	-	-	-	-	2.71
Comp#63 North Fresh Low	14GYDD0039	265	278	-	-	-	-	-	0.475	-	-	-	-
Comp#64 North Fresh Median	14GYDD0039	211	231	50	-	-	150	17.3	-	86.0	0.40	34.4	2.67
Comp#65 North Fresh High	14GYDD0039	231	247	-	-	-	-	-	0.617	-	-	-	-
Comp#66 North Fresh Median	14GYDD0040	226	243	189	-	-	150	17.3	-	84.2	0.45	37.9	2.67
Comp#68 HG North Fresh Median	14GYDD0053	234	253	138	-	-	150	18.5	0.560	100.0	0.34	34.0	2.69
Comp#69 HG North Fresh High	14GYDD0053	215	234	-	5.7	19.8	-	-	-	-	-	-	2.71
Comp#70 South Fresh Low	13GYRC0028	175	195	-	9.3	-	-	-	-	-	-	-	2.72
Comp#71 South Fresh Median	13GYRC0028	138	157	-	-	-	150	16.6	-	100.0	0.35	35.0	2.67
Comp#74 South Fresh Median	14GYDD0019	85	105	246	-	-	-	-	0.616	-	-	-	-

Sample ID	BHID	From m	To m	UCS MPa	CWi kWh/t	RWi kWh/t	BBWi Screen □m	BBWi kWh/t	Ai g	A	b	Axb	SG
Comp#75 South Fresh High	14GYDD0019	157	171	-	14.2	-	-	-	-	-	-	-	2.72
Comp#77 Central Fresh Median	14GYDD0048	231	249	-	-	-	150	16.8	-	90.0	0.39	35.1	2.69
Comp#78 Central Fresh High	14GYDD0048	299	319	-	12.0	-	-	-	-	-	-	-	2.73
Comp#79 Central Fresh Low	13GYRC0048	172	187	80	-	-	-	-	0.593	-	-	-	-
Comp#80 Central Fresh Median	13GYRC0048	191	203	-	-	-	150	16.7	-	100.0	0.33	33.0	2.68
Comp#82 North Fresh Low	14GYDD0012A	120	135	-	13.9	-	-	-	-	-	-	-	2.71
Comp#83 North Fresh Median	14GYDD0012A	103	121	-	-	-	150	17.8	-	90.4	0.40	36.2	2.66
Comp#85 North Fresh Low	14GYDD0047	179	193	-	-	-	-	-	0.358	-	-	-	-
Comp#86 North Fresh Median	14GYDD0047	109	129	112	-	-	-	-	0.492	-	-	-	-
Max				258	14.2	22.1	-	18.5	0.617	100.0	0.5	37.9	2.73
Min				50	5.2	18.9	-	16.6	0.358	81.8	0.3	32.1	2.66
Std Dev				66	3.3	1.2	-	0.6	0.074	5.8	0.0	1.7	0.02
Average				155	8.3	20.8	-	17.3	0.540	91.2	0.4	34.9	2.70
85th Percentile				222	12.9	22.1	-	17.8	0.604	-	-	-	2.73
15th Percentile				-	-	-	-	-	-	-	-	32.5	-

Notes:

- *A and b are ore hardness parameters used by the SAG mill model in **JKSimMet.***
- *The product of A and b, referred to as Axb is the universally accepted parameter which represents an ore's resistance to impact breakage.*

13.5 Gravity Gold Test Work

GRG test work was completed on seven master composite Fresh ore samples representing the South, Central, North and HG North Fresh ore domains. Master composites #1, #2 and #3 were each subjected to three stage GRG testing whilst master composites #4, #5, #6 and #7 were each subjected to single stage GRG testing. The gravity recoverable gold content was determined using a laboratory sized Knelson batch centrifugal concentrator.

The three-stage GRG test established gold liberation and recovery data for the samples tested. The final gravity gold recoveries are summarised in Table 13-4.

Table 13-4: Gravity Gold Test Work - Three-Stage GRG Summary

Composite ID	Head Assay (g/t)	Mass Yield (%)	GRG (%)	Concentrate Grade (g/t)
Master Composite #1 - South Fresh	1.07	0.99	66.1	71
Master Composite #2 - Central Fresh	1.19	0.90	67.6	89
Master Composite #3 - North Fresh	1.47	0.83	66.4	117

The three-stage GRG results were comparable between the three master composites. On average, the first stage, at a grind size of P_{100} 850 µm, recovered 29% of the gold, of which approximately half was contained in the coarse +106 µm size fraction. This indicates that approximately 15% of the gold in feed is coarse gravity recoverable gold. As the grind size was further reduced to P_{80} 75 µm for the third pass, gravity gold recovery on average, increased to approximately 67%. The majority (about two-thirds) of the additional gold recovered was contained in the -106 µm size fraction, indicating the gravity recoverable gold is predominantly fine grained.

The single stage GRG test establishes recovery data only. Overall the average GRG for the four master composites is slightly lower than the three-stage GRG recovery value. In summary, these GRG results indicate the gold in the ore is highly amenable to gravity concentration by batch centrifugal concentration.

The single stage GRG recoveries for the four master composites are summarised in Table 13-5.

Table 13-5: Gravity Gold Test Work - Single Stage GRG Summary

Composite ID	Head Assay (g/t)	Mass Yield (%)	GRG (%)	Concentrate Grade (g/t)
Master Composite #4 - South Fresh	1.23	0.38	59.7	195
Master Composite #5 - Central Fresh	1.27	0.38	64.6	214
Master Composite #6 - North Fresh	1.45	0.38	63.0	238
Master Composite #7 - HG North Fresh	2.22	0.38	68.3	397

Intensive cyanidation leach tests were carried out on master composites #4, #5 and #6, single stage GRG concentrates. The results indicated comparable leaching kinetics between each sample. Given a grind size of P_{80} 75 µm, more than 99% of the gold was leached into solution within four hours. The final leach solution, analysed by ICP, indicates the concentrations of various deleterious elements were low and are unlikely to affect leaching or downstream electrowinning.

13.6 Leach Extraction Test Work

Extensive leach extraction testing was carried out on Oxide, Saprock and Fresh ore samples. Project site water from the Yeo borefield was used for all test work.

The main objective of the gold extractive test work was to determine the relationship between total gold extraction (gravity plus cyanide leach) and grind size for the Fresh ore.

Gravity-Direct Cyanidation (Bottle Roll) Test Work

The standard gravity-leach test work included:

- Sample preparation of composite sub-samples
- Head assay analysis including comprehensive head assay for gold by Fire Assay in duplicate with two replicate sub-samples (four assays in total) plus 34 other elements, including SG determination and gold analysis by Screen Fire Assay (SFA) on 1 kg charges
- Determinations of grind establishment times on three selected grind size P80 106 µm, 125 µm and 150 µm
- For each composite tested, samples underwent gravity separation and intensive (LeachWELL™) 24 hour cyanidation leach of gravity concentrates. The tailings from the intensive cyanide leach of the gravity concentrate were combined with the gravity tailings and subjected to a 24 hour direct bottle roll cyanidation leach with oxygen sparging using site water.

Calculated head grades tested ranged from 0.54 g/t Au (Comp #42 South Low) to 3.52 g/t Au (Comp #69 HG North High). Significant variability was observed between the multiple gold fire head assays suggesting the presence of coarse gold in the test composites. The screen fire assays confirmed the presence of coarse gold, with the oversize (+75 µm) material gold grade on average more than 14 times higher than the undersize material.

Organic/graphitic carbon levels were at or below the detection limit (0.03%) except in the case of the Saprock master composite which reported 0.06% organic carbon; this reduces the probability of preg-robbing of gold from the leach liquor during cyanidation. Later Preg-Robbing Index (**PRI**) Determination testing indicated that the Oxide and Saprock master composites did not exhibit preg-robbing behaviour.

Each of the composites contained low levels of base metals, including copper, zinc and nickel, decreasing the possibility of excess reagent consumption and reduced gold dissolution kinetics caused by cyanide soluble minerals. Arsenic was also relatively low (in the range < 10 ppm to 1,160 ppm) reducing the probability of gold locked in refractory mode in solid solution with minerals such as arsenopyrite.

A total of 138 batch scale gravity-leach extraction tests were carried out on Fresh ore from 46 composites. A summary of the standard gravity-leach extraction test work results for the three grind sizes tested is summarised in Table 13-6.

Table 13-6: Leach Extraction Test Work - Summary of Gravity-Direct Cyanidation for Fresh Ore

Grind Size P806 (µm)	% Gold Extraction				Au Grade (g/t)		Consumption (kg/t)	
	Gravity	Leach	Leach Overall	Total	Calc'd Head	Leach Residue	NaCN	Lime
106	62.1	31.6	83.7	93.7	1.43	0.08	0.27	2.23
125	61.2	30.8	79.2	92.0	1.40	0.11	0.26	2.24
150	57.6	33.1	78.4	90.8	1.44	0.12	0.26	2.21

The level of gold extraction varied according to grind size. Highest extractions were generally observed at the finest grind sizes. Figure 13-2 displays the gold extraction against the calculated head assay for all gravity-leach tests carried out at grind sizes P_{80} 106 µm, 125 µm and 150 µm on Fresh ore composites. At a LOM head grade of 1.20 g/t the total gold extraction was 93.3%, 91.8% and 90.4% respectively. The data has a standard error of approximately ± 2 %. While tested samples representing the Fresh ore is classified as free milling the gold extraction is generally sensitive to grind size, particularly above P_{80} 125 µm.

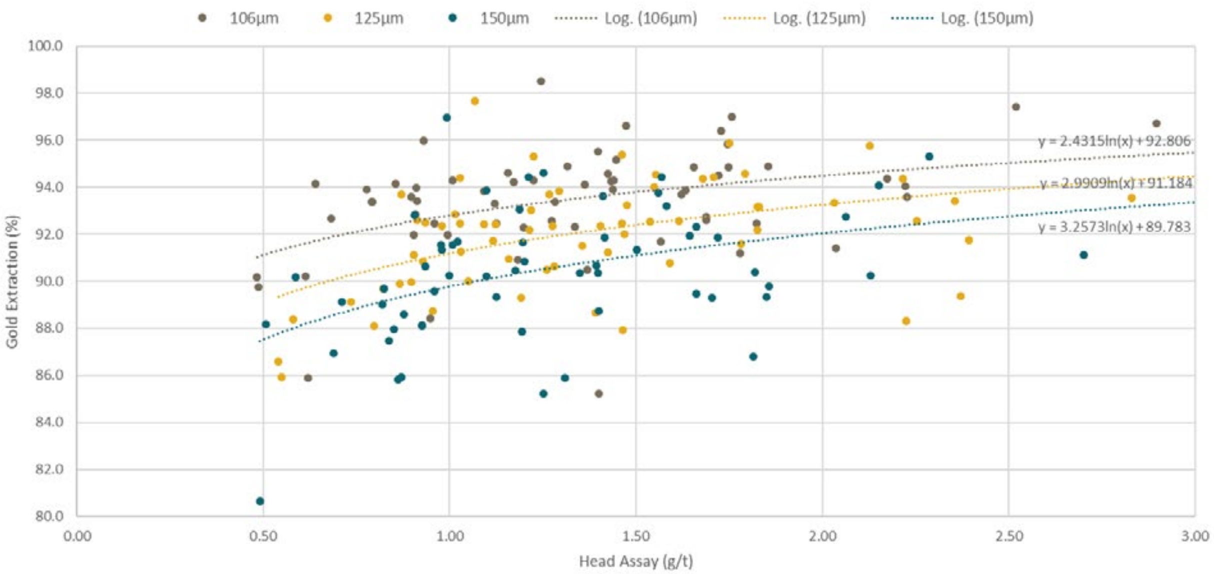

Figure 13-2: *Gold Gravity-Leach Extraction for Fresh Ore - Regression Analysis*

A summary of the standard gravity-leach extraction test work results for grind size at P_{80} 125 µm is summarised in Table 13-7.

Table 13-7: Leach Extraction Test Work - Summary of Gravity-Direct Cyanidation on Fresh Ore at P_{80} 125 µm

Composite ID	% Gold Extraction				Au Grade (g/t)		Consumption (kg/t)	
	Gravity	Leach	Leach Overall	Total	Calc'd Head	Leach Residue	NaCN	Lime
Comp#42 South Fresh Low	51.8	34.8	72.2	86.6	0.54	0.07	0.25	2.66
Comp#43 South Fresh Median	68.6	23.8	75.9	92.4	1.12	0.09	0.32	1.97
Comp#44 South Fresh High	52.9	35.4	75.2	88.3	2.22	0.26	0.27	2.21
Comp#45 South Fresh Low	61.7	31.3	81.8	93.0	1.22	0.09	0.27	2.54
Comp#46 South Fresh Median	62.5	29.8	79.6	92.3	0.98	0.08	0.20	2.28
Comp#47 South Fresh High	61.6	31.0	80.7	92.6	2.25	0.17	0.30	2.52
Comp#48 South Fresh Low	57.7	31.9	75.5	89.7	0.82	0.09	0.25	2.21
Comp#49 South Fresh Median	65.4	27.8	80.4	93.2	1.48	0.10	0.22	2.10
Comp#50 South Fresh High	72.9	21.1	77.9	94.0	1.55	0.09	0.25	2.22
Comp#51 Central Fresh Low	64.8	27.7	78.6	92.4	1.09	0.08	0.27	2.40
Comp#52 Central Fresh Median	73.8	20.7	79.1	94.5	1.55	0.09	0.25	2.38
Comp#53 Central Fresh High	68.2	25.0	78.5	93.2	1.83	0.13	0.27	2.13
Comp#54 Central Fresh Low	70.4	23.3	78.7	93.7	0.87	0.06	0.27	2.27
Comp#55 Central Fresh Median	60.5	31.0	78.5	91.5	1.36	0.12	0.22	2.41
Comp#56 Central Fresh High	67.8	26.5	82.4	94.3	1.68	0.10	0.25	2.52
Comp#57 Central Fresh Low	37.3	52.6	83.9	89.9	0.87	0.09	0.25	2.43
Comp#58 Central Fresh Median	71.7	20.8	73.7	92.6	1.62	0.12	0.22	2.45
Comp#59 Central Fresh High	80.3	15.6	79.0	95.9	1.75	0.07	0.22	2.21
Comp#60 North Fresh Low	49.3	39.8	78.5	89.1	0.73	0.08	0.25	2.23
Comp#61 North Fresh Median	70.6	23.1	78.5	93.7	1.27	0.08	0.22	2.71
Comp#62 North Fresh High	68.0	26.5	83.0	94.6	1.79	0.10	0.22	2.41
Comp#63 North Fresh Low	63.6	28.9	79.3	92.5	1.03	0.08	0.22	2.52
Comp#64 North Fresh Median	65.7	23.6	68.9	89.4	2.37	0.25	0.25	2.69
Comp#65 North Fresh High	62.3	29.8	79.3	92.2	1.83	0.14	0.27	2.25
Comp#66 North Fresh Median	62.9	29.6	79.9	92.5	1.54	0.12	0.25	2.77
Comp#67 North Fresh High	74.7	20.7	81.7	95.4	1.46	0.07	0.22	2.61
Comp#68 HG North Fresh Median	67.2	26.0	79.2	93.2	1.83	0.13	0.25	2.78
Comp#69 HG North Fresh High	73.6	21.7	82.0	95.2	3.52	0.17	0.25	2.93
Comp#70 South Fresh Low	48.3	42.8	82.8	91.1	0.90	0.08	0.25	1.71
Comp#71 South Fresh Median	55.1	36.6	81.5	91.7	1.12	0.09	0.25	1.72
Comp#72 South Fresh High	54.9	37.3	82.7	92.2	1.21	0.10	0.25	1.76
Comp#73 South Fresh Low	55.1	33.3	74.1	88.4	0.58	0.07	0.27	1.79
Comp#74 South Fresh Median	58.8	32.0	77.8	90.8	0.93	0.09	0.22	1.50
Comp#75 South Fresh High	48.1	40.6	78.2	88.7	1.39	0.16	0.20	1.49
Comp#76 Central Fresh Low	66.0	24.0	70.5	90.0	0.90	0.09	0.20	1.69
Comp#77 Central Fresh Median	53.0	38.2	81.4	91.3	1.03	0.09	0.22	1.62
Comp#78 Central Fresh High	77.1	18.2	79.5	95.3	1.23	0.06	0.27	1.96
Comp#79 Central Fresh Low	57.9	34.7	82.4	92.6	0.91	0.07	0.27	1.92
Comp#80 Central Fresh Median	56.8	34.4	79.7	91.2	1.42	0.13	0.32	1.94

Composite ID	% Gold Extraction				Au Grade (g/t)		Consumption (kg/t)	
	Gravity	Leach	Leach Overall	Total	Calc'd Head	Leach Residue	NaCN	Lime
Comp#81 Central Fresh High	56.7	34.2	79.1	90.9	1.16	0.11	0.32	2.10
Comp#82 North Fresh Low	51.9	41.0	85.2	92.9	1.01	0.07	0.25	2.34
Comp#83 North Fresh Median	54.5	37.9	83.5	92.5	1.46	0.11	0.32	2.17
Comp#84 North Fresh High	60.3	33.3	83.8	93.6	2.83	0.18	0.35	2.52
Comp#85 North Fresh Low	46.3	39.6	73.8	85.9	0.55	0.08	0.35	2.71
Comp#86 North Fresh Median	47.1	42.9	81.1	90.0	1.05	0.11	0.32	2.29
Comp#87 North Fresh High	59.1	34.3	83.9	93.4	2.35	0.16	0.35	2.15
Average	61.2	30.8	79.2	92.0	1.40	0.11	0.26	2.24

Figure 13-3 displays the calculated head assay versus solid residue for all Fresh ore samples tested at a grind size of P_{80} 125 µm.

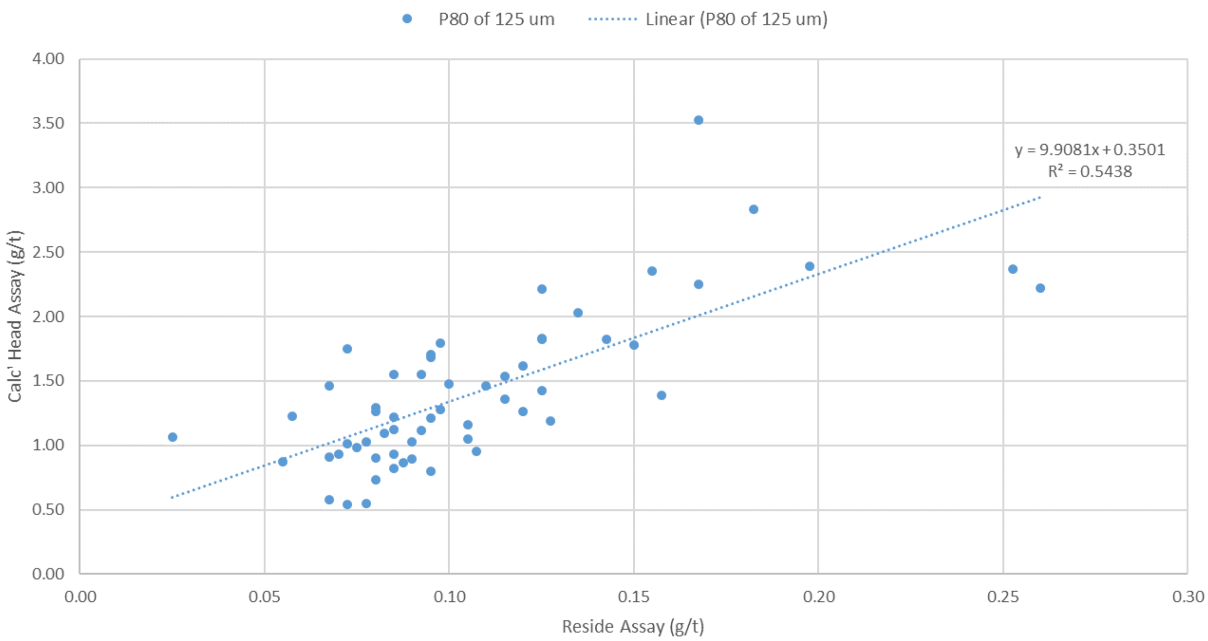

Figure 13-3: *Calculated Head Assay versus Residue Assay for Grind Size P_{80} 125 µm - Fresh ore*

In summary, the test work results for the gravity-direct cyanidation at a grind size of P_{80} 125 µm indicated that gravity recovery was wide ranging from 37.3% to 80.3%, averaging 61.2% recovery and that total gold extraction from 46 Fresh composite samples averaged 92.0% extraction. Leach kinetics were generally rapid, with the majority of gold dissolution completed within a residence time of four hours. Average cyanide consumption was 0.26 kg/t and average lime consumption 2.24 kg/t. The level of reagent consumption variability was typical of bench-scale laboratory testing.

There is a moderate correlation between head assay and residue assay indicating higher residue for higher head assays. The average solid leach residue was 0.11 g/t. The deepest sample tested was composite #59 (Central Fresh High) at a depth of 411 metres with a head grade of 1.75 g/t Au; this composite gave a total gold extraction of 95.9%.

Whole of Ore Leach Test Work

Direct cyanidation (bottle roll) 24 hour testing was conducted on 12 Fresh ore sub-samples as part of a program (A17012) of whole of ore leach test work. The 12 samples tested were targeted for selection having a head assay of around 1.21 g/t. Test work was conducted at a grind size of P_{80} 125 µm only. A summary of the results is shown in Table 13-8.

Table 13-8: Leach Extraction Test Work - Summary of Whole of Ore Leach Direct Cyanidation on Fresh Ore at P_{80} 125 µm

Sample ID	Au Head Grade (g/t)			Leach Au Extraction (%)					Au Tail Grade (g/t) [Avg of 4 FA]	Reagent Consumption (kg/t)	
	Assay [Avg]	SFA	Calc.	2-hr	4-hr	8-hr	16-hr	24-hr		NaCN	Lime
Comp#14 Main South Fresh Low	0.82	0.88	0.88	56.0	75.2	83.2	84.3	84.9	0.13	0.15	2.74
Comp#15 Main South Fresh Median	1.26	1.13	1.54	59.3	80.8	91.1	91.7	91.7	0.13	0.27	2.77
Comp#25 Main Central Fresh Low	1.10	1.10	1.02	62.1	83.9	87.8	91.2	91.2	0.09	0.20	2.90
Comp#31 Main North Fresh Low	1.01	0.85	0.77	67.2	82.0	89.0	90.9	89.7	0.08	0.17	2.90
Comp#46 South Fresh Median	1.14	0.81	0.90	62.8	80.4	88.6	90.8	92.5	0.07	0.25	2.69
Comp#51 Central Fresh Low	0.93	1.19	1.40	63.6	82.4	93.7	94.5	95.2	0.07	0.27	2.73
Comp#57 Central Fresh Low	0.81	0.92	0.92	76.6	85.2	87.9	89.5	92.7	0.07	0.27	2.59
Comp#61 North Fresh Median	1.39	0.96	1.98	55.6	78.2	91.7	94.2	94.2	0.12	0.25	2.82
Comp#63 North Fresh Low	1.26	1.05	1.10	72.4	86.4	93.2	93.2	93.2	0.08	0.20	3.35
Comp#70 South Fresh Low	1.13	0.97	0.99	68.3	81.2	89.7	90.2	92.7	0.07	0.15	2.96
Comp#71 South Fresh Median	1.40	1.73	1.55	74.0	86.7	93.1	93.1	93.1	0.11	0.27	3.10
Comp#75 South Fresh High	1.48	1.52	1.38	83.9	86.1	86.1	88.2	88.2	0.16	0.25	2.34

Statistical comparison (paired t-test) between the total extraction of the whole of ore leach test data and the gravity-leach test data for the same samples is shown in Table 13-9.

Table 13-9: Leach Extraction Test Work - Paired t-test for Gravity-Leach and Whole of Ore Leach on Fresh Ore

Sample ID	Gold Extraction (%)		
	Gravity and Leach	Leach Only	Difference
Comp#14 Main South Fresh Low	88.1	84.9	3.20
Comp#15 Main South Fresh Median	89.3	91.7	-2.40
Comp#25 Main Central Fresh Low	90.5	91.2	-0.70
Comp#31 Main North Fresh Low	93.8	89.7	4.20
Comp#46 South Fresh Median	92.3	92.5	-0.10
Comp#51 Central Fresh Low	92.4	95.2	-2.70
Comp#57 Central Fresh Low	89.9	92.7	-2.80
Comp#61 North Fresh Median	93.7	94.2	-0.50
Comp#63 North Fresh Low	92.5	93.2	-0.70
Comp#70 South Fresh Low	91.1	92.7	-1.60
Comp#71 South Fresh Median	91.7	93.1	-1.40
Comp#75 South Fresh High	88.7	88.2	0.40
Average			-0.43
Standard Deviation			2.18
Confidence Limit			30%
Critical t Value (30% level)			0.40
t Value			-0.68
±			0.25

There were two samples (Comp #14 and Comp #31) which showed a significantly higher extraction for the gravity-leach test compared to the whole of ore leach test. Most of the composites tested showed that the whole of ore leach extraction was slightly higher than the gravity-leach test. The paired t-test data shows that there is an average difference of -0.43% extraction ± 0.25% (i.e. a range of -0.68% to -0.18% extraction) at a very low confidence limit of 30%. Results indicate that there is no statistically significant difference between the whole of ore leach test data and the gravity-leach test data for the samples tested.

Bulk Leach Test Work

Bulk leach tests were carried out on sub-samples of four Fresh ore master composites (composites #8, #9, #10 and #11) representing the major ore domains within the Gruyere deposit, and the two Oxide/Saprock master composites. A third composite (#20 Main Central Saprock Median) was also tested in order to produce sufficient cyanidation leach tailings for downstream testing. All tests were carried out at a grind size of P80 125 µm. Calculated head assays ranged from 1.03 g/t Au (Comp #8) to 2.13 g/t Au (Comp #11).

A summary of the bulk leach extraction results is shown in Table 13-10. Rate of extraction curves for Fresh ore and Oxide/Saprock ore are presented in Figures 13-4 and 13.5 respectively.

Table 13-10: Bulk Leach Extraction Summary

Composite ID	Gold Extraction (%)			Au Grade (g/t)		Consumption (kg/t)	
	Gravity	Leach	Total	Calc'd Head	Leach Residue	Cyanide	Lime
Master Composite #8 - South Fresh	47.7	46.7	94.4	1.03	0.06	1.10	1.60
Master Composite #9 - Central Fresh	49.6	41.0	90.6	1.28	0.12	1.05	1.38
Master Composite #10 - North Fresh	49.0	43.3	92.4	1.41	0.11	0.89	1.51
Master Composite #11 - HG North Fresh	61.5	34.2	95.8	2.13	0.09	0.72	1.65
Master Composite Oxide	34.8	61.0	95.8	1.84	0.08	0.26	4.38
Master Composite Saprock	37.1	59.7	96.8	2.05	0.07	0.18	3.81
Composite #20 Main Central Saprock	40.0	55.7	95.7	1.10	0.05	0.76	2.51

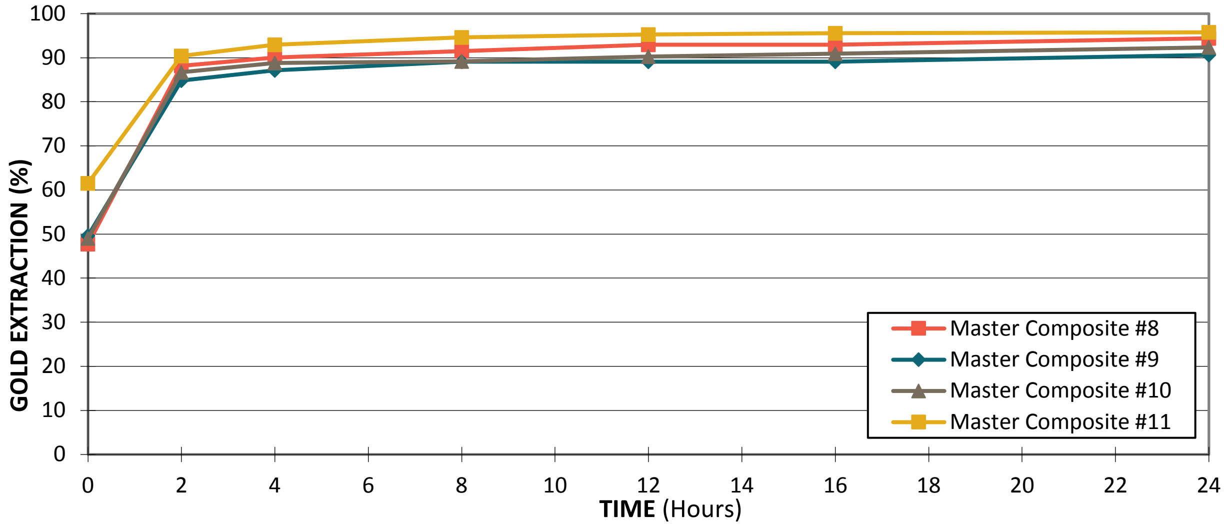

Figure 13-4: *Bulk Leach Extraction Fresh Ore - Rate of Gold Extraction*

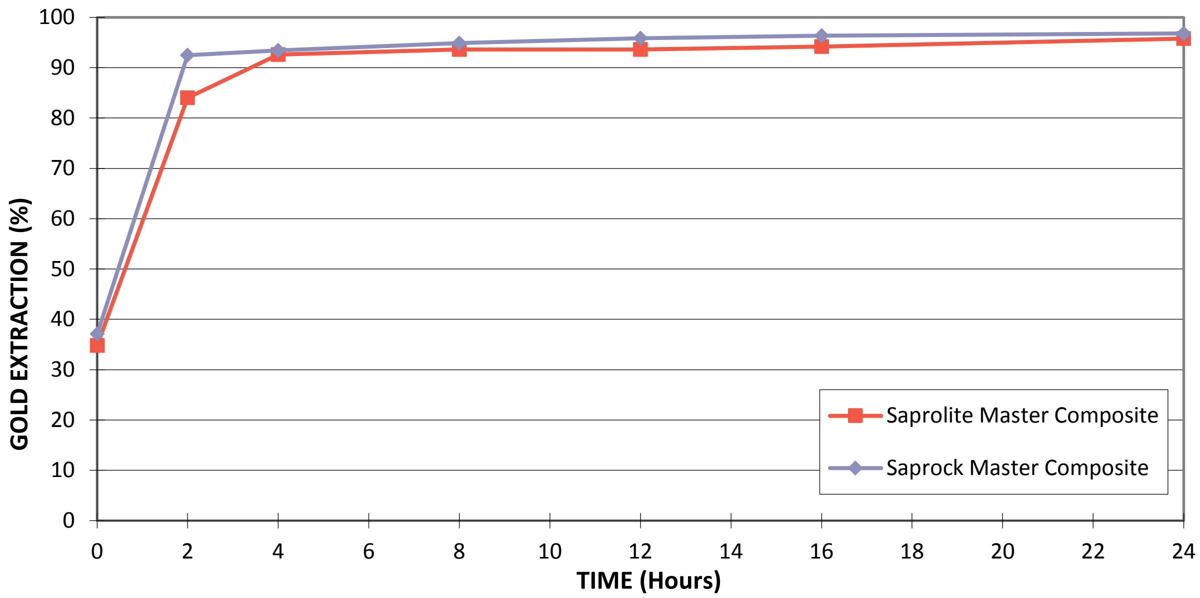

Figure 13-5: *Bulk Leach Extraction Oxide/Saprock - Rate of Gold Extraction*

Bulk leach test results indicate a gold extraction for Fresh ore of over 90% with the highest extraction at 95.8% from HG North composite #11 and the lowest at 90.6% from the Central composite #9. Gravity recovery of gold was moderate, ranging between 47.7% and 61.5%. Leach kinetics were rapid, with the majority of gold dissolution completed after a residence time of four hours. Lime consumption was moderate and cyanide consumption was higher than what would be expected in a full-scale plant due to the significantly higher agitation rate during the bulk leach test.

Bulk leach results for the Oxide/Saprock indicate low to moderate gravity recovery in the range 34.8% for the Oxide Saprolite to 40.0% for the Saprock. Total leach extraction was very high in the range 95-97% for both Oxide Saprolite and Saprock. Leach kinetics were similar to the Fresh ore with the majority of gold dissolution completed after a residence time of four hours. Lime consumption was moderately high and cyanide consumption relatively low.

13.7 Carbon Adsorption Test Work

Sequential triple contact Carbon-in-Pulp (**CIP**) adsorption and equilibrium carbon loading test work were carried out on the following samples: Fresh ore master composites #8 to #11; the Oxide and Saprock master composites; and composite #20 Main Central Saprock Median.

Sequential Triple Contact Carbon Adsorption CIP Test Work

Carbon adsorption test work was undertaken to determine the Fleming kinetic adsorption constants, k and n for gold extraction. The Fleming empirical rate constant, k, is dependent upon slurry mixing efficiency, pulp viscosity and carbon particle size. A summary of these test results is shown in Table 13-11.

Table 13-11: Summary of Sequential Triple Contact Carbon Adsorption CIP Test Work

Composite ID	Fleming Gold Adsorption Constants		Loaded Carbon Au Metal Content (g/t)
	k	n	
Master Composite #8 - South Fresh	148	1.09	2,221
Master Composite #9 - Central Fresh	146	0.85	2,047
Master Composite #10 - North Fresh	180	0.81	2,867
Master Composite #11 - HG North Fresh	346	0.87	5,013
Saprolite Master Composite	99	0.64	784
Saprock Master Composite	82	0.68	1,014
Saprock Master Composite Duplicate	58	0.57	731
Comp#20 Central Saprock Median	252	0.74	1,738

Based on the derived k values, the results indicate that the Fresh ore generally has medium to good adsorption kinetics. The adsorption kinetics for composite #11 - HG North in particular was excellent. Both the Oxide Saprolite and Saprock composite samples had low k values indicating poor adsorption kinetics however it is suspected that these two samples which were derived from RC chip samples, were contaminated by drilling fluid. Subsequently, testing was carried out on an alternative Saprock composite sample derived from drill core, Comp #20. The k value for Comp #20 was very high (252) indicating very good adsorption kinetics for the Saprock ore.

Equilibrium Carbon Loading Test Work

Equilibrium carbon loading test work was carried out to determine the gold loading capacity of activated carbon. This is determined from an adsorption isotherm and is defined as the mass of gold (in mg) adsorbed on to 1 g of carbon at 1 mg/L of gold in solution i.e. the equilibrium loading on carbon in contact with 1 ppm Au solution. The summary of these test results are shown in Table 13-12.

Table 13-12: Summary of Equilibrium Carbon Loading Test Work

Composite ID	Equilibrium Carbon Loading Au (g/t) @ Sol'n Concentration		
	1.0 ppm	0.5 ppm	0.1 ppm
Master Composite #8 - South Fresh	3785	2983	2177
Master Composite #9 - Central Fresh	4179	3265	2356
Master Composite #10 - North Fresh	4987	3812	2672
Master Composite #11 - HG North Fresh	6174	4714	3299
Saprolite Master Composite	5205	3886	2641
Saprock Master Composite	3763	2925	2096

Results indicate moderate to excellent gold adsorption characteristics, with relatively high gold loading at 1 ppm gold in solution, at equilibrium, ranging from 3.785 to 6.174 kg of gold per tonne of carbon for the Fresh ore. The Oxide Saprolite and Saprock master composites achieved 5.205 kg and 3.763 kg of gold per tonne of carbon respectively.

Gravity/CIL Cyanidation Test Work

Gravity-CIL cyanidation testing was carried out on Saprock ore samples to determine if there would be any adverse plant recovery impacts from processing this type of ore as preliminary carbon adsorption test work indicated poor adsorption kinetic rates which may have been as a result of RC drilling fluid contamination. Two samples were tested for CIL test work: the Saprock master composite; and the composite #20 Main Central Saprock Median.

The Saprock master composite was tested twice. The first test was as per the standard CIL test procedure. The second test included a scavenger carbon contact stage prior to the commencement of the standard CIL test (i.e. prior to the addition of cyanide) using 20 g of carbon for 1 hour to remove any possible contaminants (e.g. hydrocarbons, organic matter) that were suspected to have been present in the ore. A summary of the CIL test work results for each of three samples tested are summarised in Table 13-13.

Table 13-13: Summary of Saprock CIL Test Work

Composite ID	% Gold Extraction			Au Grade (g/t)		CIL Solution Loss (ppm)	Consumption (kg/t)	
	Gravity	CIL	Total	Calc'd Head	CIL Solid Residue		NaCN	Lime
A16857 Saprock Master Composite	57.9	39.5	97.4	2.45	0.06	0.003	0.69	3.67
A16857 Saprock Master Composite with scavenger carbon contact stage	46.3	50.0	96.2	1.52	0.06	0.003	0.67	3.50
A16207 Comp#20 Central Saprock Median	50.6	46.3	96.9	1.20	0.04	0.003	0.63	2.72

The total CIL extraction result for all three Saprock samples were excellent at over 96% with moderately high gravity gold recovery ranging from 46.3% to 57.9%. The rate of gold extraction was very high. Very low leach residue grades of 0.04 g/t to 0.06 g/t Au were achieved. Lime consumption was moderately high at 2.7-3.7 kg/t and cyanide consumption low.

13.8 Other Test Work

Slurry Rheology

Slurry rheology testing was carried out on a number of composite samples representing the major ore domains within the Gruyere deposit. The samples tested were: Fresh ore master composites #8 to #11; the Oxide and Saprock master composites; composite #13 Main South Transitional Median; and composite #18 Main Central Oxide Median.

Viscosity testing using a Bohlin Visco 88 meter was carried out over a range of slurry densities. All test work was carried out using Project site water, at a grind size of P_{80} 125 µm and at ambient temperature.

The viscosity test work results indicated that for all samples tested, the viscosity was generally low and there would be no issues pumping and mixing the slurry from the various ore types.

Dynamic Thickening Test Work

Dynamic thickening and Rheogram testing on thickened underflow samples was carried out to determine thickening and pumping requirements for process plant design. Outotec was sent Fresh ore samples from master composites #12-#15, and from Transition ore composite #13 and Oxide ore composite #18. All samples were at a grind size of P_{80} 125 µm.

The results for all samples tested showed that the material can be thickened by high rate thickening at a flux rate range of 0.50 to 1.50 t/m^2/h. Over this flux rate range the sample reached minimum thickener underflow densities of 57.3% for the Oxide Saprolite, 60.6% for the Transition and 64.2% solids (w/w) for the Fresh tailings. The flocculant dosage rate was 30 g/t for the Oxide Saprolite, 20 g/t for the Transition and 10 g/t for the Fresh. Yield stress were generally very low and overflow clarity were good for all Fresh ore samples tested.

Rheogram testing was carried out on all thickened underflow samples from the dynamic thickening test work program. The slurry samples were prepared to a range of slurry densities (50, 55, 60 and 70% solids). Rheograms of each samples were produced using a VT 550 rheometer with SV 2 sensor system at ambient temperature. The shear rate ranged from 600 s^{-1} to 0.05 s^{-1} and the shear stress was recorded. The Rheograms for all samples tested showed that the various ore types exhibit shear thinning behaviour. Solids settled very quickly for all samples over the range of slurry densities tested.

Materials Handling Test Work

A range of materials handling testing was carried out by Jenike & Johanson, a specialist bulk materials engineering firm. The test program consisted of the following tests:

- Particle size analysis
- Worst-case moisture determination with regards to cohesive strength and wall friction angle at continuous flow conditions
- Cohesive strength for calculating the critical outlet dimensions, to prevent bridging and rat-holing
- Compressibility to determine the bulk density versus consolidating pressure relationship
- Particle density - liquid displacement method to determine the true density of the particles
- Wall friction for calculating mass flow hopper angles
- Permeability for predicting critical steady state flow rates of de-aerated material
- Chute test for calculating critical chute clean-off angle
- Bench scale angle of repose and drawdown angle.

Two 50 kg samples were tested – an Oxide Saprolite composite sample and a Fresh ore composite sample. The Oxide Saprolite sample originated from RC drill samples whilst the Fresh ore samples were comminution testing reserve core samples. The Oxide sample was screened to -6.35 mm prior to testing and the Fresh ore sample was screened to -12.5 millimetres prior to testing due to the lack of natural fine material present in the sample.

The Oxide material at 10.1% moisture exhibited the worst flowability. The Fresh ore was classified as easy flowing for all moisture contents tested. Both the Oxide and Fresh samples were shown to have very low permeability. Critical steady solids state flow rate was determined to be high for the Fresh ore and low for Oxide ore.

The chute test results indicate that both ore types are impact pressure sensitive. Higher impact pressures resulted in higher recommended minimum chute angles. Angle of repose and drawdown tests results are summarised in Table 13-14.

The materials handling test results were used as inputs into the process plant design, particularly with regards to chute angles and stockpile capacity.

Table 13-14: Materials Handling Test Work - Summary of Angle of Repose and Drawdown Angles

Material	Particle Size Tested	Moisture Content	Angle of Repose		Drawdown Angle	
	(mm)	(%)	Average	Range	Average	Range
Oxide Ore	-6.35	10.1	41	40-42	70	70
Fresh Ore	-12.5	6.2	38	35-42	45	44-45

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Gold Road has carried out all resource estimation for the Gruyere deposit in-house, with technical assistance and review by Optiro. All Mineral Resources are reported in accordance with the JORC Code 2012, which is regarded as an acceptable foreign code under NI 43-101 definitions.

The maiden estimate for the Gruyere Gold Project was reported in August 2014[37] and subsequent updates were reported in May 2015[38] and September 2015[39]. The latest Mineral Resource estimate was presented in April 2016[40]. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the estimates in the 22 April 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Gruyere Gold Project Mineral Resources and the information of the 22 April 2016 announcement.

The April 2016 Mineral Resource estimate was independently reviewed by Optiro and has been used as input to the FS for the Project.

The resource estimation software used by Gold Road includes:

- Leapfrog Geo – Drill hole validation, material type, lithology, alteration and faulting wireframes, domaining and mineralisation wireframes, geophysics and regional geology
- Snowden Supervisor - geostatistics, variography, declustering, kriging neighbourhood analysis (KNA), validation
- Datamine Studio RM – Drill hole validation, cross-section, plan and long-section plotting, block modelling, geostatistics, quantitative kriging neighbourhood analysis (QKNA), OK estimation for validation and input to LUC, block model validation, classification, and reporting
- Datamine Studio RM Uniform Conditioning Module – LUC grade estimation; the module is an interface to the code in Isatis software for change of support, information effect calculation, uniform conditioning and grade localisation; Isatis is a highly regarded geostatistical software in the industry and is used by many of the top gold mining companies worldwide.

[37] ASX:GOR Gold Road Resources Public Disclosure, 4 August 2014, "3.84 Million Ounce Gruyere Maiden Gold Mineral Resource"
[38] ASX:GOR Gold Road Resources Public Disclosure, 28 May 2015, "Gruyere Resource Grows to 5.51 Million Ounces Gold"
[39] ASX:GOR Gold Road Resources Public Disclosure, 16 September 2015, "Gruyere Resource Grows to 5.62 Million Ounces Gold"
[40] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, "Gruyere Resource Increases to 6.2 Million Ounces"

14.2 Drilling Data

The April 2016 Mineral Resource estimate is based on a total of 87,066 metres from 470 drill holes (357 RC holes for 41,264 metres, 73 holes with RC pre-collars for 14,694 metres RC and 16,506 metres diamond tail, and 40 full DDH holes for 14,603 metres). The drilling includes 150 grade control equivalent RC holes (14,837 metres) and two diamond holes (673 metres) completed since the previous Mineral Resource estimate in September 2015.

The deposit extends over a strike length of 2,800 metres of which 1,800 metres is drilled on a 100 metre section spacing to a depth of 600 metres below surface. Drill holes on the 100 metre sections are generally 40 metres apart in the upper 400 metres and approximately 100 metres apart below that. Additional intermediate 50 metre sections have been drilled with at least one to two holes per section over the upper 300 metres. Approximately 75% of the strike length and 100 metres of depth has been drilled to 25 by 25 metres and includes a 100 metre zone drilled to 12.5 by 25 metre spacing in the centre of the deposit. RC drilling dominates in the upper 100 metres with diamond drilling the dominant method below this depth.

14.3 Resource Model

The modelled resource has dimensions of 1,800 metres along strike and a variable width of 7 to 190 metres, averaging 90 metres. The vertical depth of the resource model extends from the surface (mineralisation commences 2 metres below surface) to a lower limit of 600 metres below surface.

A full set of 25 metre spaced cross-sections was generated and manually interpreted, with focus on the oxidation profile and material type boundaries. These sections were spatially referenced using 3D imaging software and used to guide digital construction of the resource model wireframes, resulting in a smoother and more realistic interpretation of these boundaries.

Deposit lithologies and oxidation zones are coded into the block model and are used in the allocation of rock properties such as bulk density and in constraining mineralisation.

Three mineralisation domains used to constrain the estimation of gold grades in the model. These domains are named Primary, Weathered and Dispersion Blanket. The three domains are shown in a 3D isometric projection in Figure 14-1.

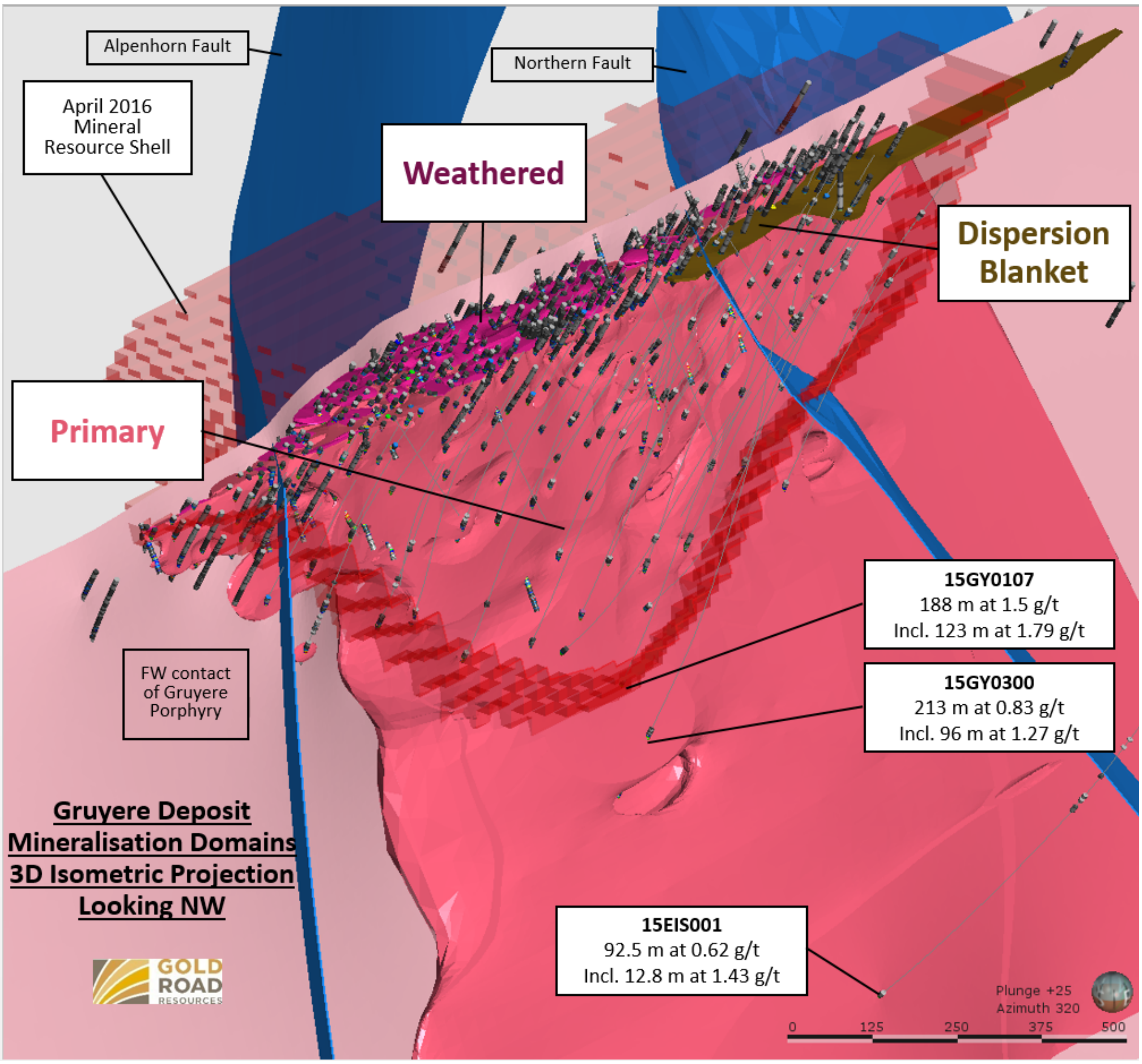

***Figure 14-1**: Mineralisation Domains - 3D Isometric Projection*

The Primary domain includes Saprock, Transition and Fresh mineralisation hosted within the Gruyere Porphyry and represents 99% of the total Mineral Resource. In the September 2015 model the Primary domain boundary was interpreted between the Saprock and Transition boundary; in the April 2016 model it has been interpreted at the Oxide (Saprolite) and Saprock boundary based on geological observations and spatial analysis of the mineralisation (including variography) and therefore is approximately 20 to 45 metres higher in elevation.

A cut-off grade of 0.3 g/t Au is used for implicit modelling of the Primary Domain. The 0.3 g/t Au cut off corresponds approximately to the boundary between barren to very weakly mineralised hematite-magnetite alteration and weak to strongly mineralised albite-sericite-carbonate ± pyrite, pyrrohotite, arsenopyrite alteration, and is also recognised as an inflection point subdividing the non-mineralised and mineralised populations on log probability plots.

The Weathered domain consists of Oxide (Saprolite) and minor Saprock mineralisation hosted within the Gruyere Porphyry and contains 0.5% of the Mineral Resource. The domain has dominant flat lying controls that are consistent with re-mobilisation and/or leaching of gold above the leaching boundary as a result of oxidation/weathering processes. The flat lying controls have been verified by spatial analysis. To achieve the continuity required for adequate grade modelling and estimation in the Weathered Domain the cut-off grade for implicit modelling is decreased to 0.15 g/t.

For both the Primary and Weathered domains, a maximum of 2 metres of internal waste and a minimum intersection of 2 metres of mineralisation are applied to the modelling.

The minor Dispersion Blanket domain is a thin flat lying zone of mineralisation hosted at the Oxide (Saprolite)-Saprock boundary within hanging wall and footwall lithologies. The Dispersion Blanket domain is constrained by an automated traditional wire framing method.

14.4 Gold Assay Statistics

All assay information available at 10 February 2016 was used in the grade estimate for the April 2016 Mineral Resource. The resource estimation incorporated 32,293 RC and DDH assays within the mineralisation wireframe. The raw assays were composited to 2 metre lengths to remove sample length biases and improve estimation quality. Univariate statistics are summarised in Table 14-1.

The highly consistent nature of the Primary domain of the Gruyere gold mineralisation is demonstrated by the low coefficient of variation (CV) of 0.89 in the uncut 2 metre composited data, the minimal reduction 0.07% on the mean with the application of a top-cut of 30 g/t Au, and long ranges in the variograms (discussed further in section 14.5). The low CV and global mean grade in the Primary domain remains virtually unchanged from the previous resource estimation completed in September 2015.

The higher CV in the Weathered domain is consistent with the higher variability implied by the gold re-mobilisation and leaching interpretation.

Top-cuts (all samples included method) were applied to 2 metre composites selected within mineralisation wireframes. The top-cut level was determined through the analysis of histograms, log histograms, log probability plots and spatial analysis. The Primary domain samples were cut to 30 g/t Au which affected only one sample resulting in a 0.1% reduction in mean grade. Weathered domain samples were cut to 10 g/t Au which affected three samples resulting in a 1.0% reduction in mean grade. There were no samples cut in the Dispersion Blanket domain.

Table 14-1: Basic Statistics by Domain

Domain	Composite Length	Number of Samples	Min g/t	Max g/t	Mean g/t	Standard Deviation g/t	Variance g/t²	Co-efficient of Variation	Number of Samples Cut	% Samples Cut	% Reduction in Mean
Weathered	Raw	1,526	0.01	21.88	0.72	1.54	2.38	2.14			
	2.0 m	820	0.02	12.22	0.70	1.14	1.29	1.61			
	2.0 m top-cut	820	0.02	10.00	0.70	1.07	1.15	1.54	3	0.37%	1.00%
Primary	Raw	30,573	0.01	84.88	1.28	1.50	2.24	1.17			
	2.0 m	15,285	0.01	43.17	1.28	1.14	1.29	0.89			
	2.0 m top-cut	15,285	0.01	30.00	1.28	1.11	1.23	0.87	1	0.01%	0.07%
Dispersion Blanket	Raw	194	0.09	5.87	0.83	0.86	0.74	1.03			
	2.0 m	106	0.11	3.45	0.85	0.71	0.50	0.83			
	2.0 m top-cut	106	0.11	3.45	0.85	0.71	0.50	0.83	-	0.00%	0.00%

Table 14-2: Variogram Parameters by Domain

Domain	Host Rock and Predominant Material Type	Strike/Dip	Variogram Values (variance)		Variogram Ranges (m)		
Weathered	Gruyere Porphyry Saprolite	000 / 00	Nugget	0.35	Dip	Strike	Perpendicular
			C1	0.36	10	35	3
			C2	0.24	22	60	6
			C3	0.05	50	80	15
Primary Measured	Gruyere Porphyry Saprock and Transition	000 / 75 E	Nugget	0.35	Dip	Strike	Perpendicular
			C1	0.25	25	25	4
			C2	0.25	65	110	7
			C3	0.15	275	525	75
Primary Indicated and Inferred	Gruyere Porphyry Fresh	000 / 75 E	Nugget	0.35	Dip	Strike	Perpendicular
			C1	0.25	25	25	4
			C2	0.33	115	145	8
			C3	0.07	275	350	60
Dispersion Blanket	Mafic and Intermediate Sequence Saprolite	000 / 00	Same as Weathered				

14.5 Resource Estimation Methodology

Gold Road produced a combined OK and recoverable resource estimate for the Gruyere deposit using a 3D block model and an SMU with dimensions of 5 metres east (across strike) x 12.5 metres north (along strike) x 5 metres RL (vertical).

An estimate was achieved using two different estimation techniques dependent on the density of drill data. In areas of close spaced drilling of 12.5 to 25 metres x 25 metres (ultimately classified as a Measured resource), OK was used with a parent block size of 5 metres east x 12.5 metres north x 5 metres RL (the same as the selected SMU). In this case the parent block size is regarded as acceptable with respect to drill hole spacing, i.e. approximately equivalent to 50% of the maximum drill spacing in the area of drilling.

Grade estimation in the areas with drill spacing of 25 to 50 metres x 100 metres or 100 metres x 100 metres (ultimately classified as Indicated and Inferred resources respectively) was carried out using an OK estimate as input to LUC (**LUC**). The OK estimation used a parent block size of 25 metres east x 50 metres north x 10 metres RL which is approximately equivalent to 25 to 50% of the maximum drill spacing in these areas, and again considered acceptable.

The LUC methodology allows for estimation of SMU-sized blocks from a primary OK grade estimate of larger parent blocks, in this case an SMU of 5 metres east x 12.5 metres north x 5 metres RL (the same size as the OK for the well drilled area). The method provides grade estimates of SMUs from widely spaced data; the estimate is still globally accurate but avoids the inherent smoothing effect on the grade-tonnage curve of a conventional OK model. The LUC method provides an estimate of the grade-tonnage curve expected from a selective mining process at a given SMU size, i.e. normally less tonnes at higher-grade above cut-off than what would be expected from a conventional OK estimate.

Variography

Spatial continuity of mineralisation at Gruyere was defined using directional variograms.

A number of twin holes were purposely drilled or coincidently provided data to assess the short scale variability of mineralisation; all tests showed good comparison of thickness and grade between holes. Three twin RC holes were completed with their collars being less than 5 metres distant from the parent collar and two twin RC versus diamond sub-parallel holes were completed with their collars being less than 10 metres distant from the parent collar. In addition one diamond pair provided a twin data set over a length of 120 metres at a spacing of less than 4 metres apart. This twinned data also provided accurate data for testing the nugget effect at Gruyere.

A detailed drill programme completed in 2015 included a number of holes on an approximate 12.5 metres x 12.5 metres to 25 metres x 25 metres drill spacing. The data derived from this drilling was used to confirm short scale mineralisation continuity and refine statistical and geostatistical relationships in the data.

Details of variogram models are shown in Table 14-2. Models have low to moderate nuggets and three range structures.

A new variogram with shorter ranges than the September 2015 model was interpreted for the Weathered domain using the new drilling data. This model is consistent with the lower continuity implied by the gold leaching interpretation of this domain. There was insufficient data to obtain a meaningful variogram for the Dispersion Blanket domain and therefore the Weathered domain variogram was used.

A new variogram was also interpreted for the densely drilled upper section of the Primary domain which is predominantly hosted in Saprock and Transition mineralisation. The ranges of the new variogram are similar to the previous September 2015 model. The variography remained unchanged for the remainder of the Primary domain, predominantly hosted in Fresh mineralisation.

Grade Estimation

Kriging neighbourhood analysis (KNA) was undertaken to optimise the search neighbourhood used for the estimation and to test the parent block size. The search ellipse and selected samples by block were viewed in three dimensions to verify the parameters.

Hard boundaries are used for all domain boundaries to control grade estimation. Dry bulk density values were applied to the block model using average values for different lithologies and oxidation zones.

Grade estimation parameters are listed separately for the OK estimates for the 5 metres east x 12.5 metres north x 5 metres RL parent blocks, the primary OK estimates for the 25 metres east x 50 metres north x 10 metres RL parent blocks, and the LUC estimates for 5 metres east x 12.5 metres north x 5 metres RL SMUs.

OK grade estimation parameters for 5 metres east x 12.5 metres north x 5 metres RL parent blocks:

- Smallest sub-cell used for volume calculations of material types – 1 metre east x 12.5 metres north x 1 metre RL
- Block discretisation - 3 metres east x 5 metres north x 2 metres RL
- Search ellipse aligned to mineralisation trend, dimensions: Dispersion Blanket - 50 metres east x 80 metres north x 15 metres RL; Weathered – 50 metres east x 80 metres north x 15 metres RL; Primary - 35 metres east x 60 metres north x 15 metres RL
- Sample search – Dispersion Blanket: maximum samples per drill hole 5; first pass minimum 20 samples, maximum 60; second pass minimum 10, maximum 60, search size expanded by x2; third pass minimum 2, maximum 60, search size expanded by x3
- Sample search – Weathered: maximum samples per drill hole 5; first pass minimum samples 30, maximum 60; second pass minimum 30, maximum 60, search size expanded by x2; third pass minimum 10, maximum 60, search size expanded by x2
- Sample search – Primary: maximum samples per drill hole 4; first pass minimum samples 16, maximum 36; second pass minimum 16, maximum 36, search size expanded by x2; third pass minimum 8, maximum 36, search size expanded by x2.

OK grade estimation parameters for 25 metres east x 50 metres north x 10 metres RL parent blocks:

- Smallest sub-cell used for volume calculations of material types – 1 metre east x 12.5 metres north x 1 metre RL
- Block discretisation - 3 metres east x 5 metres north x 2 metres RL
- Search ellipse aligned to mineralisation trend, dimensions: Dispersion Blanket - 50 metres east x 80 metres north x 15 metres RL; Weathered – 50 metres east x 80 metres north x 15 metres RL; Primary - 200 metres east x 350 metres north x 60 metres RL
- Sample search – Dispersion Blanket: maximum samples per drill hole 5; first pass minimum 20 samples, maximum 60; second pass minimum 10, maximum 60, search size expanded by x 2; third pass minimum 2, maximum 60, search size expanded by x 3
- Sample search – Weathered: maximum samples per drill hole 5; first pass minimum samples 30, maximum 60; second pass minimum 30, maximum 60, search size expanded by x2; third pass minimum 1, maximum 60, search size expanded by x3

- Sample search – Primary: maximum samples per drill hole 7; first pass minimum samples 30, maximum 60; second pass minimum 15, maximum 60, search size expanded by x1; third pass minimum 5, maximum 60, search size expanded by x 3
- Maximum distance of extrapolation from data points – 50 metres from sample data to Inferred boundary.

LUC grade estimation parameters for 5 metres east x 12.5 metres north x 5 metres RL SMUs:

- 12.5 metres east x 25 metres north x 5 metres RL de-clustering of input data in Supervisor (the de-clustering weight is inversely proportional to the number of data points in each cell); note that change in grade through de-clustering with respect to the use of the cell size optimiser is minimal
- Discretisation 3 metres east x 5 metres north x 2 metres RL
- Information Effect planned sample spacing 25 metres east x 25 metres north x 1 metres RL, and 9 metres east x 9 metres north x 5 metres RL planned number of samples
- 40 SMUs (5 metres east x 12.5 metres north x 5 metres RL) per parent block (25 metres east x 50 metres north x 10 metres RL)
- 70 cut-offs at 0.1 g/t intervals
- 7 iso-frequencies.

Resource Model Validation

Validation of the Mineral Resource estimate involved a number of specific checks including detailed comparison of the input data to the output model, to ensure no bias. All validation checks provided acceptable results adding confidence to the quality and validity of the estimation.

The following validation checks were performed:

- QQ plots of RC vs DDH input grades
- Statistical comparison of different drilling orientations including local spot checks
- Comparison of twinned RC, twinned DDH and twinned RC v DDH holes
- Comparison of the volume of wireframe vs the volume of block model
- Checks on the sum of gram metres prior to compositing vs the sum of gram metres post compositing
- A negative gold grade check
- Comparison of the model average grade and the de-clustered sample grade by domain
- Generation of swath plots by domain, northing and elevation
- Comparison of LUC estimate to OK estimate
- Visual check of drill data vs model data in plan, section and three dimensions
- Comparison to previous models.

The resource model itself provides a very good predictor of the Gruyere mineralisation. As an example, a significant quantity of Measured Resource in the April 2016 estimate (approximately 13 Mt) which has been upgraded from previously classified Indicated Resource in September 2015 by additional drilling shows no material change in the contained tonnes, grade or ounces. Tonnes increased by 3%, grade changed from 1.21 g/t Au to 1.20 g/t Au (-1%) and contained gold changed from 493,000 ounces to 502,800 ounces (2%).

Further validation was undertaken by testing an alternative geological interpretation. This involved running an unconstrained model (constrained only by the Gruyere Porphyry) to compare against the actual model which is constrained by a 0.3 g/t Au mineralisation envelope within the Gruyere Porphyry. Analysis showed that at 0.0 g/t Au cut-off, the unconstrained estimate showed 20% higher tonnage at 19% lower grade and 2% less contained gold. This is a result of the smearing of gold grade into weakly mineralised areas less than 0.3g/t Au within the Porphyry. The closeness of the contained gold in the two models provides confidence in the actual estimate.

Plan and cross section views of the resource model are shown in Figure 14-2 to Figure 14-5.

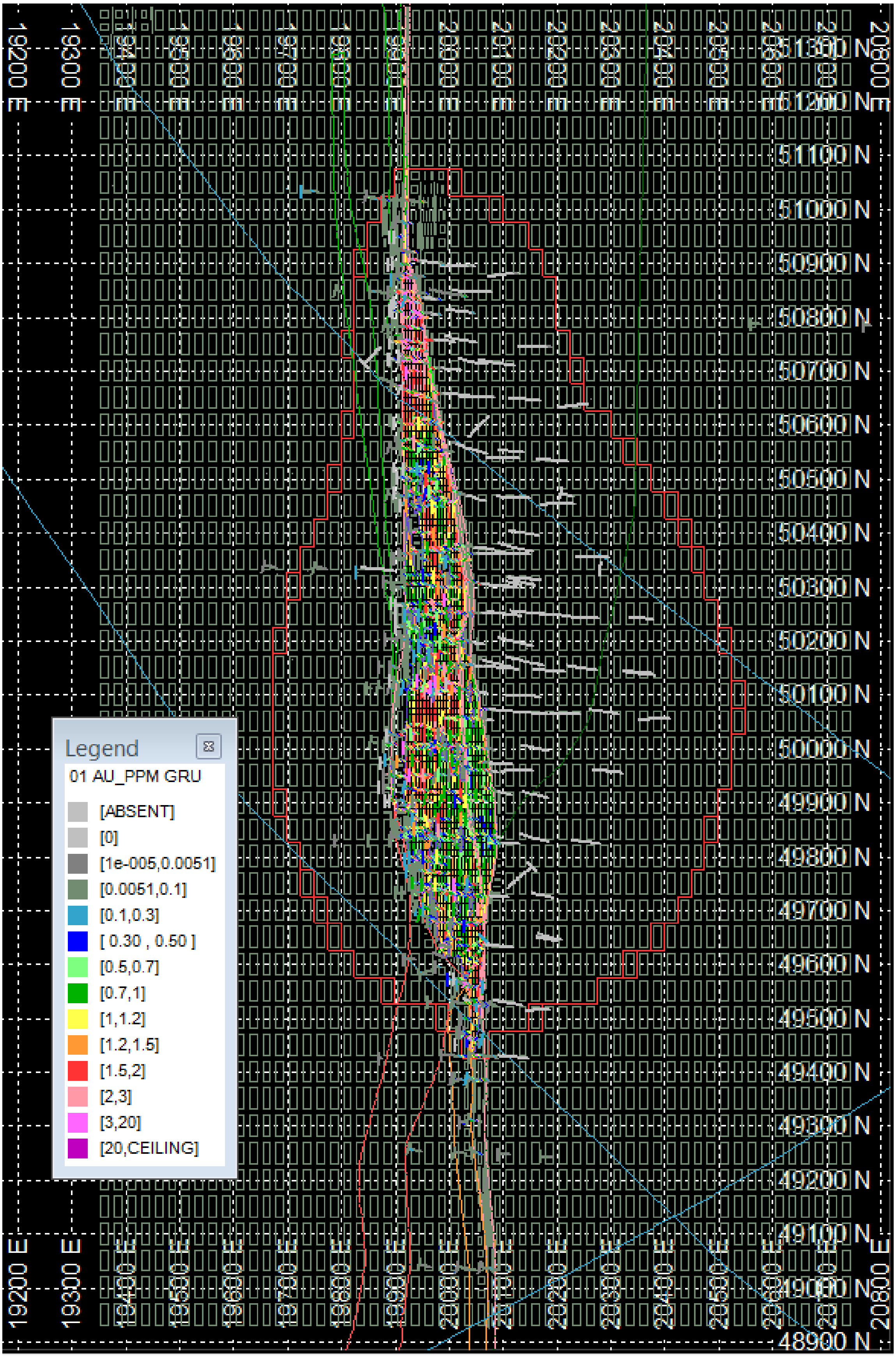

Figure 14-2: *Resource Block Model - Plan View at 9,300 mRL Showing Gold Grade (clipped to 50 m)*

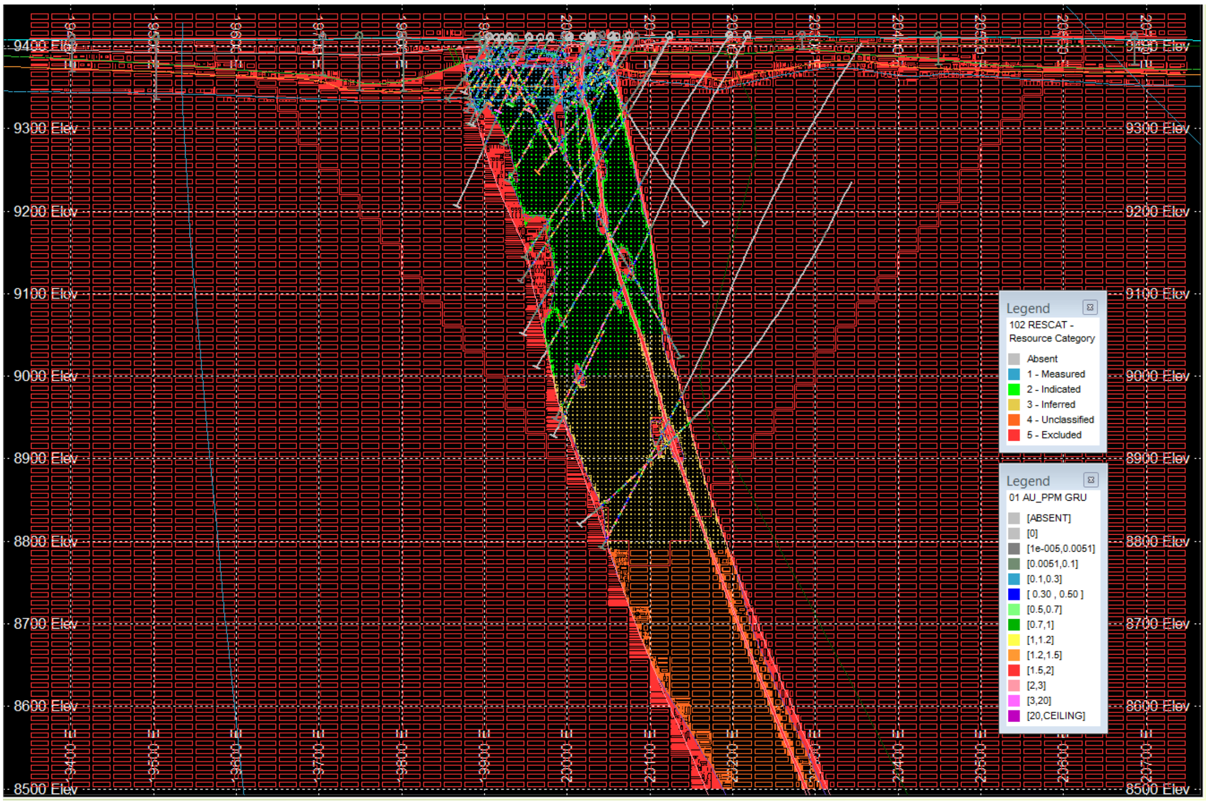

Figure 14-3: *Resource Block Model - Cross Section at 50,000N Showing Resource Category and Gold Grade (clipped to 50 m)*

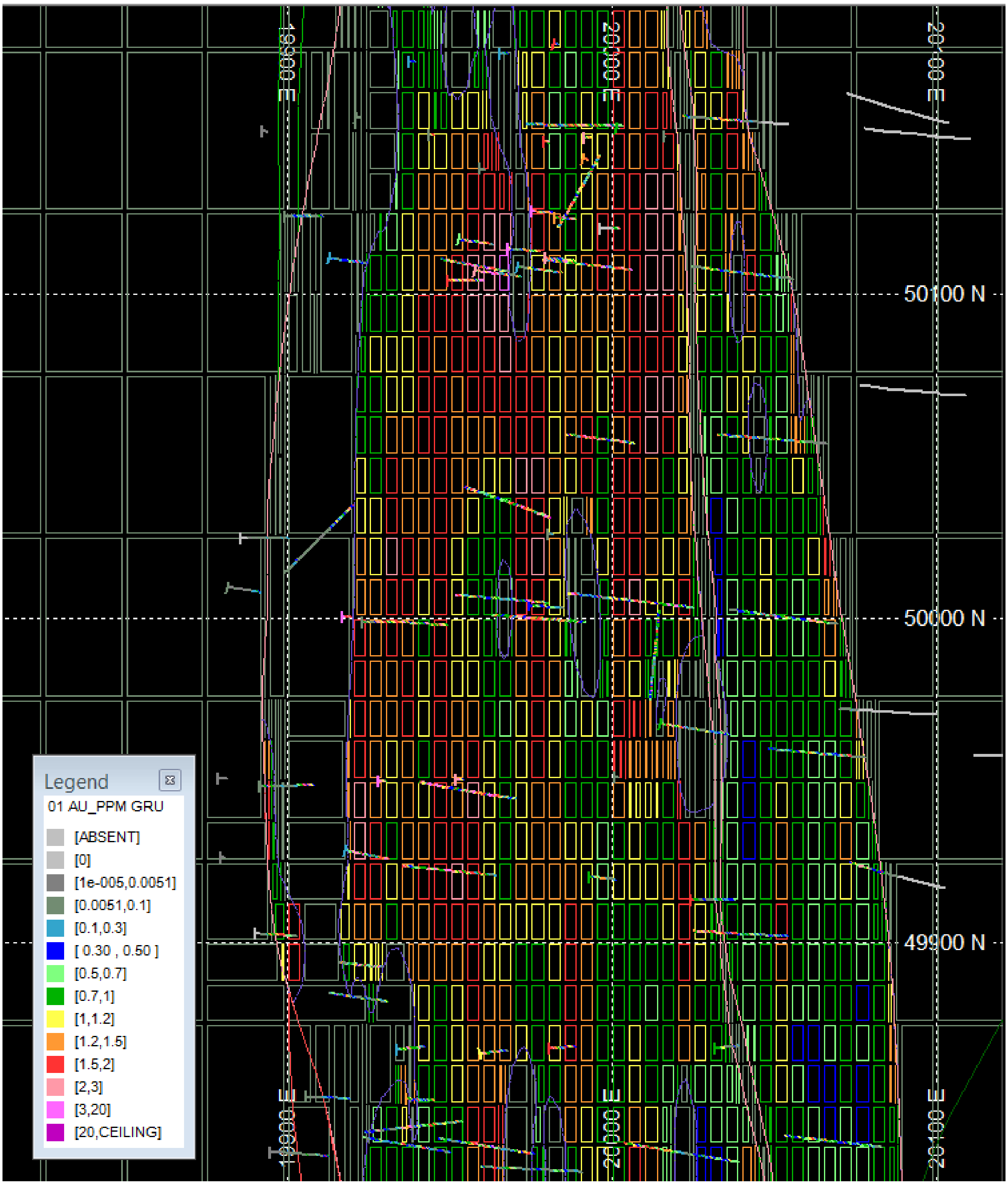

Figure 14-4: *Resource Block Model - Plan View at 9,300 mRL South Central Section of Deposit with Gold Grade (clipped to 25 m)*

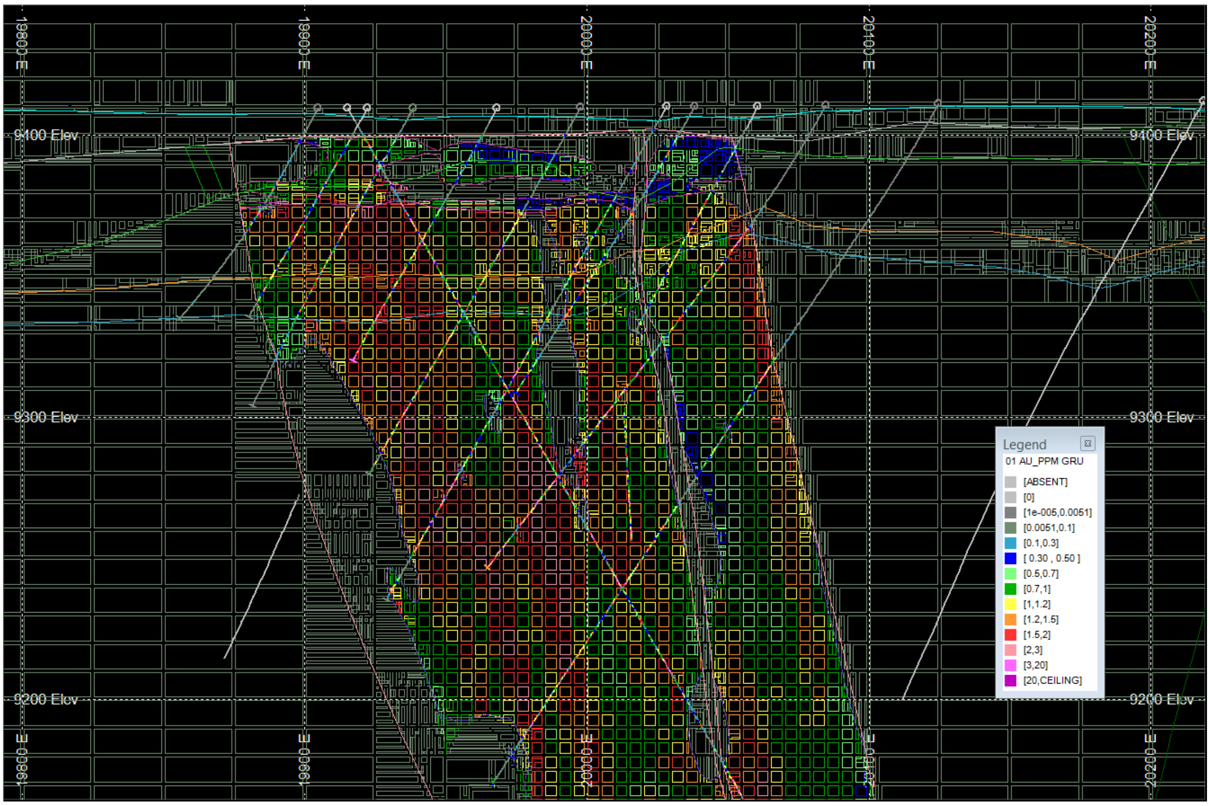

Figure 14-5: *Resource Block Model - Cross Section at 50,000N with Gold Grade in Upper Section of Deposit (clipped to 25 m)*

Resource Classification

A number of factors have been used in combination to derive the Mineral Resource classification, with the primary factor being the drill hole spacing. Other factors include the geological (lithology) continuity, in particular defining the full width (hanging wall and footwall contacts) of the Gruyere Porphyry, grade continuity, and estimation quality parameters derived from the estimation process.

With respect to drill hole spacing, the following classification was used:

- Area drilled with holes spaced at least 12.5 to 25 metres across strike and down dip (E) by 25 metres along strike (N) was classified as Measured resource
- Area with at least 25 to 50 metres east x 100 metres north was classified as Indicated resource; this area is also supported by 20 scissor holes on and between 100 metre sections, five strike-parallel holes demonstrating along strike continuity and nine off-angle holes testing alternate structural orientations
- Area with at least 100 metres east x 100 metres north was classified as Inferred resource; this area was limited to the maximum extent of the resource model at depth (600 m) and a maximum distance of 50 metres along strike beyond the extent of drilling.

Low confidence mineralisation within the resource model that does not satisfy the above criteria for Mineral Resource was flagged as unclassified material.

14.6 Mineral Resource Reporting

The April 2016 Mineral Resource estimate has been reported by Gold Road at 0.5 g/t Au cut-off in accordance with the JORC Code 2012. The Measured, Indicated and Inferred resources are as follows: Measured Resource is 13.9 Mt at 1.18 g/t Au with contained gold of 0.53 Moz; Indicated Resource is 91.1 Mt at 1.29 g/t Au with contained gold of 3.79 Moz; and Inferred Resource is 42.7 Mt at 1.35 g/t Au with contained gold of 1.85 Moz (Table 14-3).

No modifying factors, including dilution and mining recovery, have been applied to the Mineral Resource estimate.

Table 14-3: Gruyere April 2016 Mineral Resource - Tabulation by resource Category at 0.5 g/t Au cut-off

Resource Category	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)
Measured	13.9	1.18	0.53
Indicated	91.1	1.29	3.79
Measured & Indicated	105.0	1.28	4.31
Inferred	42.7	1.35	1.85

Note: *Apparent differences may occur due to rounding; Mineral Resources are reported inclusive of Ore Reserves.*

The Mineral Resource at a range of cut-off grades from 0 to 1.5 g/t Au is shown in Table 14-4.

The Competent Persons under the JORC Code 2012 for reporting this Mineral Resource is Mr Justin Osborne who is the Executive Director - Exploration and Growth and Mr John Donaldson who is Geology Manager, both employees of Gold Road.

Resource Constraints

The Mineral Resource is constrained by an optimised pit shell to determine the portion of the total mineralised inventory within the resource model that has a reasonable prospect of eventual economic extraction. The optimisation utilised mining, geotechnical and processing parameters selected for the FS and a gold price of A$1,700 per ounce. Only Measured, Indicated and Inferred categories of mineralisation at a 0.5 g/t Au cut-off within this optimised pit shell are reported as Mineral Resource. There is additional gold mineralisation outside the Mineral Resource, some of which may convert to Mineral Resource with further drilling and/or underground mining evaluation. The limits of the April 2016 Mineral Resource shell are shown in plan and longitudinal section in Figure 10-1 and Figure 10-2 respectively. These Figures also provide for a comparison between the Mineral Resource shell and the Ore Reserve optimised pit shells.

Independent Review

Optiro was engaged to externally review the technical aspects of the April 2016 Mineral Resource, the previous Mineral Resource updates in 2015 and the maiden Mineral Resource estimate in August 2014. A formal review was undertaken and suggestions for improvement were sought and applied where appropriate. A database audit was also undertaken by Optiro for the 2015 Mineral Resource updates and maiden Mineral Resource estimate.

Optiro is of the opinion that Gold Road's Gruyere Mineral Resource of April 2016 has been generated, estimated and classified in accordance both with the JORC Code 2012 and with commonly-accepted best practice for gold resource evaluation.

Table 14-4: Gruyere April 2016 Mineral Resource - Grade Tonnage Tabulation

	Measured			Indicated			Inferred			Total Measured & Indicated		
Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz)
0.0	14.2	1.16	0.53	92.0	1.28	3.80	43.0	1.34	1.85	106.1	1.27	4.33
0.5	13.9	1.18	0.53	91.1	1.29	3.79	42.7	1.35	1.85	105.0	1.28	4.31
0.7	12.7	1.23	0.50	84.9	1.34	3.66	40.4	1.39	1.80	97.6	1.32	4.16
0.8	11.6	1.28	0.47	78.7	1.39	3.51	38.0	1.43	1.74	90.3	1.37	3.98
0.9	10.1	1.34	0.43	71.1	1.44	3.29	34.8	1.48	1.65	81.1	1.42	3.73
1.0	8.4	1.42	0.38	62.5	1.51	3.03	31.1	1.54	1.54	70.8	1.50	3.41
1.1	6.7	1.51	0.33	53.7	1.58	2.73	27.3	1.60	1.41	60.4	1.57	3.06
1.2	5.3	1.60	0.28	45.2	1.66	2.41	23.5	1.68	1.27	50.5	1.65	2.69
1.5	2.5	1.92	0.15	24.8	1.92	1.53	13.6	1.92	0.84	27.2	1.92	1.68

Note: *Apparent differences may occur due to rounding; Mineral Resources are reported inclusive of Ore Reserves.*

15 ORE RESERVE ESTIMATES

15.1 Introduction

Gold Road engaged AMC to estimate the Ore Reserve in August 2016. AMC estimated the Ore Reserve in accordance with the JORC Code 2012. All Ore Reserves are reported in accordance with the JORC Code 2012, which is regarded as an acceptable foreign code under NI 43-101 definitions.

An Ore Reserves estimate was presented on 19 October 2016[41]. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Ore Reserves estimates in the 19 October 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Gruyere Gold Project Mineral Resources and the information of the 19 October 2016 announcement.

The Ore Reserve represents the economically mineable part of the Measured and Indicated Mineral Resources. The Ore Reserve estimate is the result of a detailed FS completed by a team consisting of Gold Road personnel and independent external consultants.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology which is widely utilised in the goldfields of Western Australia. Financial modelling completed as part of the FS shows that the Project is economically viable under current assumptions. Material Modifying Factors (mining, processing, infrastructure, environmental, legal, social and commercial) have been considered during the Ore Reserve estimation process.

15.2 Mining Model

The Mineral Resource model for the Project was prepared by Gold Road using a block size for gold grade estimation of 5 metres east x 12.5 metres north x 5 metres RL. The model contains a number of fields including geological rock code, in situ gold grade estimate, resource category and rock density. The resource model indicates that oxidation/weathering of the mineralisation is deeper at the northern end relative to the southern end. The deeper lying mineralisation in the northern end has relatively higher-grade compared to the southern end. Figure 15-1 shows a longitudinal view looking west of the resource model with gold grade distribution in g/t Au.

A mining model was developed that allowed for modifying factors relating to mining ore loss and dilution. AMC used Datamine software to add an estimate for mining dilution and ore loss to the resource model. The ore dilution modelling has been simulated through a process of block expansion which adds a 0.5 metres skin of waste to ore zones and also bulks up narrow ore and internal waste zones to a minimum mining width of 5 metres.

[41] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, “Gruyere Feasibility Study Approved”

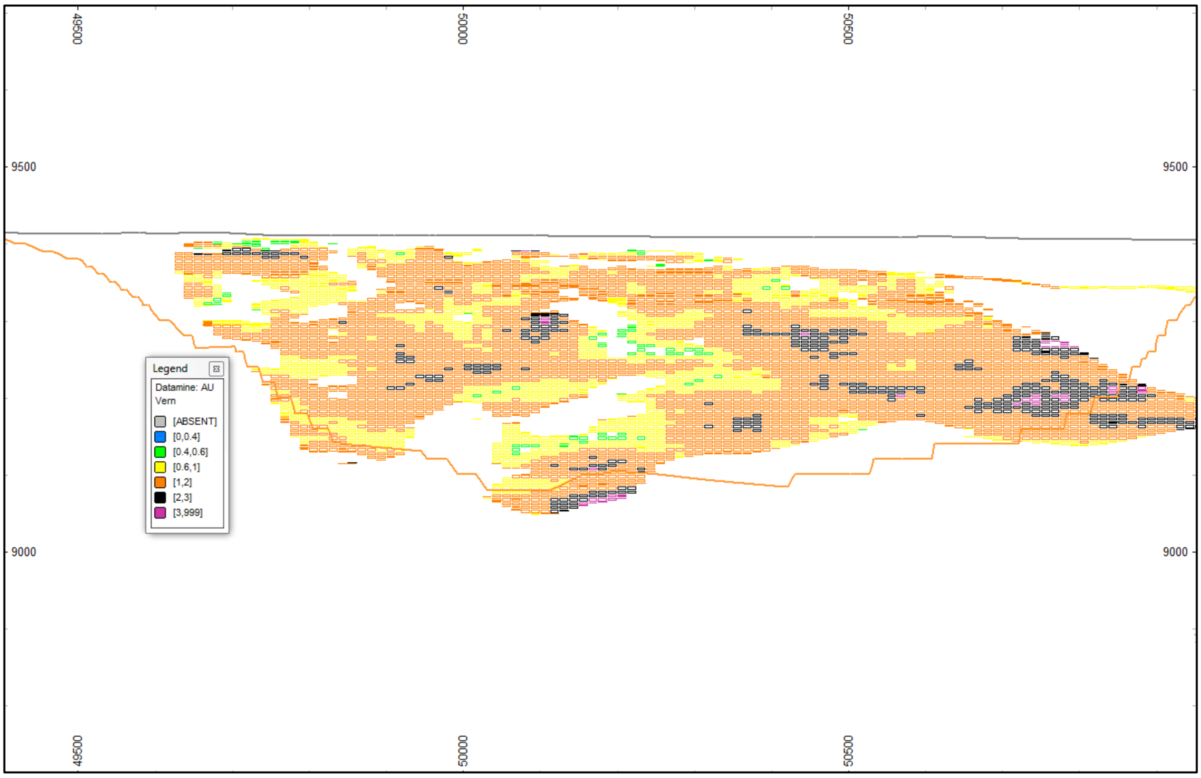

Figure 15-1: *Longitudinal Section of Geological Model looking west (Gruyere Grid)*

The dilution process considers a selective mining unit which represents the minimum practical block size that would be delineated during ore and waste selection. The chosen selective mining unit measures 5 metres east x 12.5 metres north x 5 metres RL (i.e. the same as the resource block size). Selective mining unit selection takes into account the available loading equipment sizes that can meet total material movement requirements and likely operating conditions such as grade control practices, mining methods, the direction of mining and the style and behaviour of the mineralisation.
The modifying factors were also calibrated against an actual ore block design created in the predominantly Measured area below the oxide zone.

The resource model and mining model are compared at a 0.5 g/t cut-off grade within the PFS pit design in Table 15-1. Note that the PFS pit design was the current pit design at the time of re-evaluating modifying factors.

Table 15-1: Mining Model - Dilution and Ore Loss in the PFS Pit Design

Cut-off value 0.5 g/t	In situ Resource			Diluted Mineable Inventory		
Resource Category	Tonnes (Mt)	Au (g/t)	Au (Moz)	Tonnes (Mt)	Au (g/t)	Au (Moz)
Measured	13.9	1.18	0.53	14.5	1.10	0.51
Indicated	66.1	1.27	2.70	68.1	1.22	2.67
Total (MI)	80.0	1.25	3.22	82.6	1.20	3.18
Dilution impact				103.2%	96.0%	98.6%

Note: *The FS Pit Design is shown in Table 15-3 for comparison; apparent differences may occur due to rounding*

15.3 Open Pit Optimisation

Optimisation Parameters

The open pit optimisation has been undertaken utilising the Dassault Systèmes Geovia Whittle implementation of the Lerchs-Grossman algorithm, to determine optimal mining limits.

Table 15-2 presents the pit optimisation parameters that were applied for the FS. Process engineering studies that were completed after the pit optimisation process resulted in minor changes (within 1% of pit optimisation assumptions) to process recovery assumptions for all material types. The final overall recoveries by ore type are reported in Table 16-12.

Inferred Mineral Resources were considered as waste during the pit optimization.

A gold price of A$1,500 per ounce (US$1,095 per ounce, at exchange rate of A$/US$ 0.73) was applied in the financial modelling for the Ore Reserve calculation process. This price forecast was established by Gold Road on the basis of historical A$ gold price trends over the last 5 years. Over the review period the price of gold has ranged between A$1,300 per ounce and A$1,800 per ounce and averaged approximately A$1,500 per ounce.

Table 15-2: Open Pit Optimisation Parameters

Parameter	Units	Value	Source
Reference Gold Price	A$/oz	1,500	Gold Road
Gold Price	US$/oz	1,095	Gold Road
Exchange Rate	A$/US$	0.73	Gold Road
Transport and Refining Costs	A$/oz	1.60	Gold refinery/Gold Road
Process Gold Recovery – Oxide Ore	%	93	Metallurgical test work via Gold Road
Process Gold Recovery – Transitional Ore	%	92	
Process Gold Recovery – Fresh Ore	%	3.1818 ln(x) + 90.362 where ln(x) is the natural logarithm of x and x = feed grade in g/t Resulting Recovery range 89% (0.6 g/t) to 92% (1.7 g/t)	
Processing Cost – Oxide Ore	A$/t	13.72	Process engineering consultant via Gold Road
Processing Cost – Transitional Ore	A$/t	15.06	
Processing Cost – Fresh Ore	A$/t	16.07	
ROM Ore Rehandle Cost	A$/t	0.26	PFS
Grade Control	A$/t	0.05	Gold Road
General and Administration Costs	A$/t	1.08	Gold Road
Rehabilitation	A$/t	0.04	Benchmark
Mining Tonnage Dilution	%	3.2%	AMC
Mining Ore Loss	%	1.4%	AMC
Reference Mining Cost – Fresh Rock	A$/t	3.12	Mining contractor quote
Mining Cost Adjustment per 10m Bench - Fresh Rock	A$/t	0.06	
Overall Slope Angle		Varies by rock type by depth Oxide 38°-42° Fresh 38°-57°	Geotechnical consultant

Note: *Optimisation parameters may differ from final FS Study outcomes due to timing.*

Optimisation Results

The output from the pit optimisation process produced a series of nested pit shells corresponding to various gold prices (as defined by Revenue Factors with revenue factor 1 corresponding to a gold price of A$1,500 per ounce). The pit optimisation results formed the basis of determining the economic mining limits for the open pit. The pit optimisation was prepared with the mining model and only valued ore blocks with Mineral Resource codes of Measured and Indicated.

Figure 15-2 compares optimisation results by nested pit shell and shows the following information in relation to each pit shell:

- Total material mined (ore and waste)
- Undiscounted net cash flow
- Best case (mining each incremental pit shell as a pushback) net cash flow
- Worst case (mining each bench completely before the next bench) net cash flow
- Expected case net cash flow, which is based on the assumption that the final pit design and schedule will show a net cash flow value between the best and worst cases. The expected case was simulated by a weighting of 70% to the Worst case and 30% to the Best case. The 30% weighting towards Best case was selected based on analysis of similar deposits and assessment of the PFS mining strategy of two interim pits and two pushbacks to the ultimate pit limit.

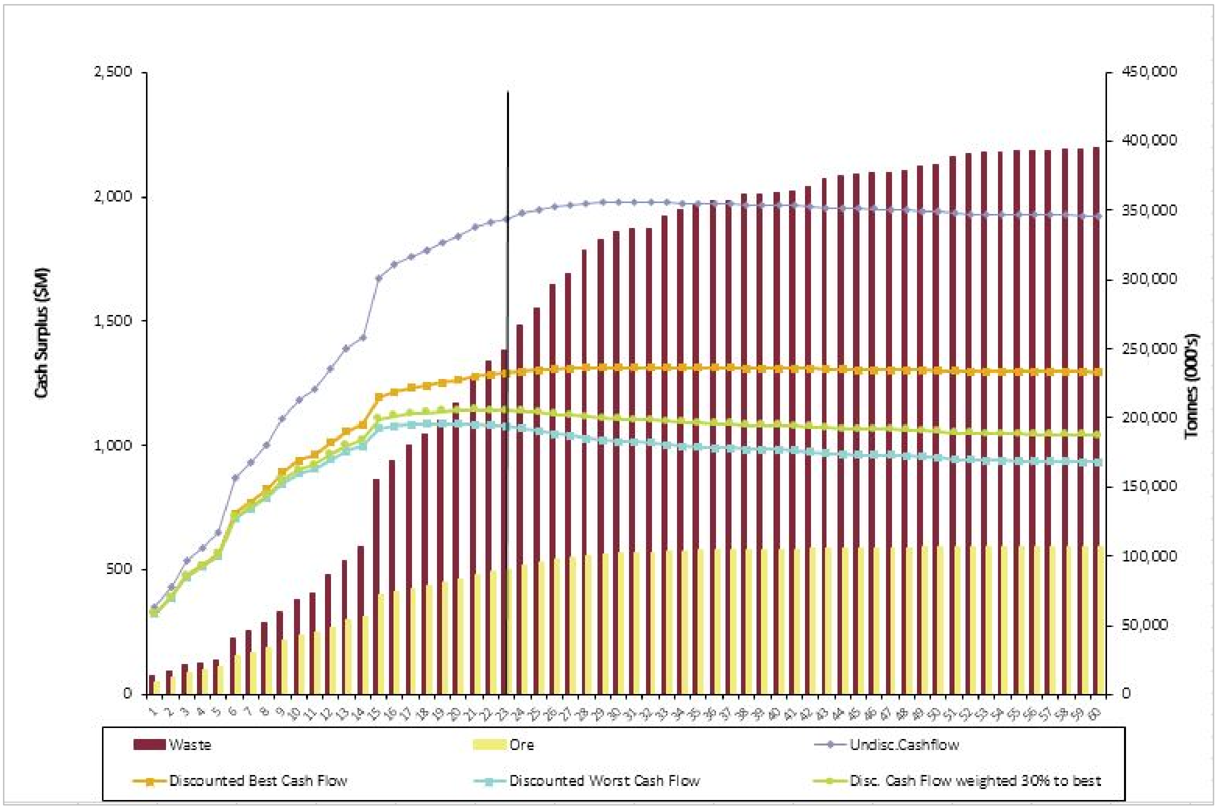

Figure 15-2: *Pit Optimisation Results Showing Cash Flow by Pit Shell*

Figure 15-2 indicates that the optimum pit shell (depending on the selection criteria) could range from Shell 20 to Shell 31; the values of other pit shells were normalised with Shell 31 (undiscounted cash flow optimum pit shell) to provide an alternative method of determining the final pit shell.

Figure 15-3 shows the optimisation results normalised against Shell 31, and indicates that Shell 23 presents 99.8% of the value of the best result on the discounted curve (weighted 30% towards the Best case). Shells larger than Shell 23 show minimal increase in value despite an increase in ore tonnes when analysed on the basis of the discounted curve (weighted 30% towards Best).

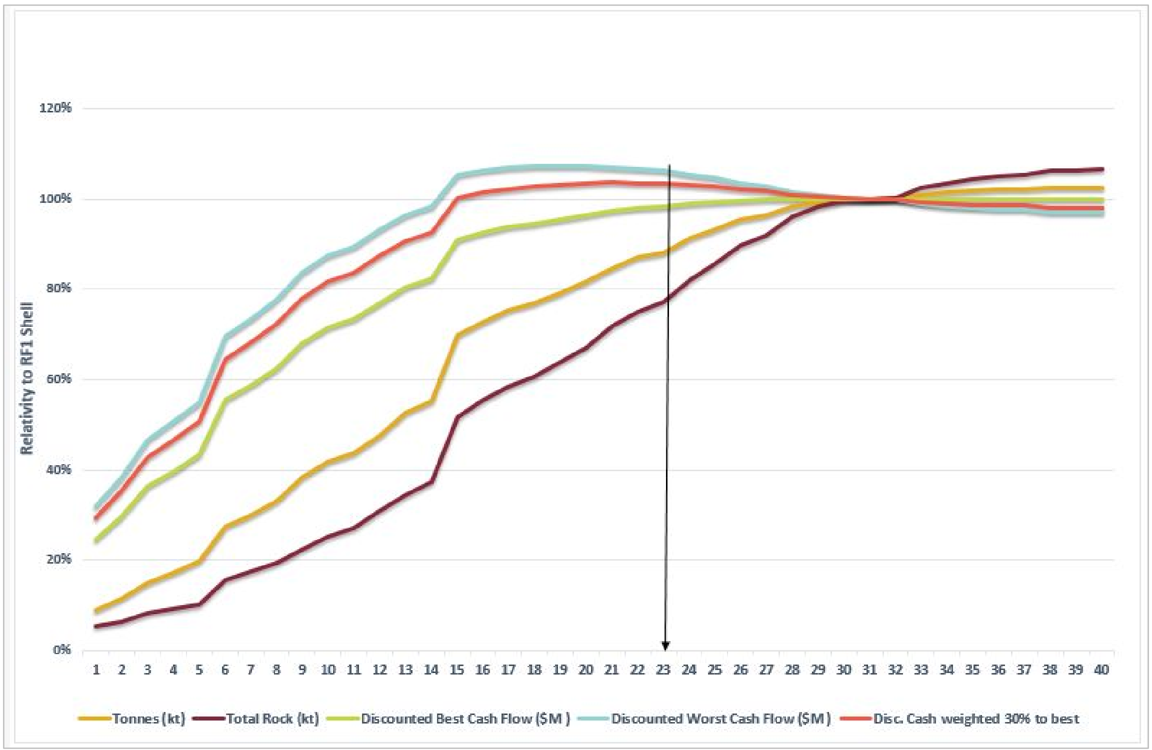

Figure 15-3: *Normalised Pit Optimisation Results*

Based on the information presented in Figure 15-2 and Figure 15-3, Shell 23 was selected as the guideline for designing the final pit.

Optimisation Sensitivity

Additional optimisation runs were undertaken to establish the sensitivity of the selected shell to key variables. The results of the sensitivity analysis provide guidance to risk and opportunities around the selected pit shell. Sensitivities were run on:

- Gold price referencing the base price of A$1,500 per ounce to -10% to +10% range with 5% increments
- Mining operating cost to -10% to +10% range with 5% increments
- Processing operating cost -10% to +10% range with 5% increments
- Processing recovery -10% to +10% range with 5% increments
- Slope angles to variations of -5 degrees to +5 degrees range.

Sensitivity analysis indicates that the optimisation output is most sensitive to changes in gold price, processing recovery and overall slope angles. The selected shell and resulting pit design is shown to be robust to significant (up to 10%) changes in key input parameters; changes beyond 10% may necessitate revising the ultimate pit design. Figure 15-4 shows the results of the sensitivity analysis.

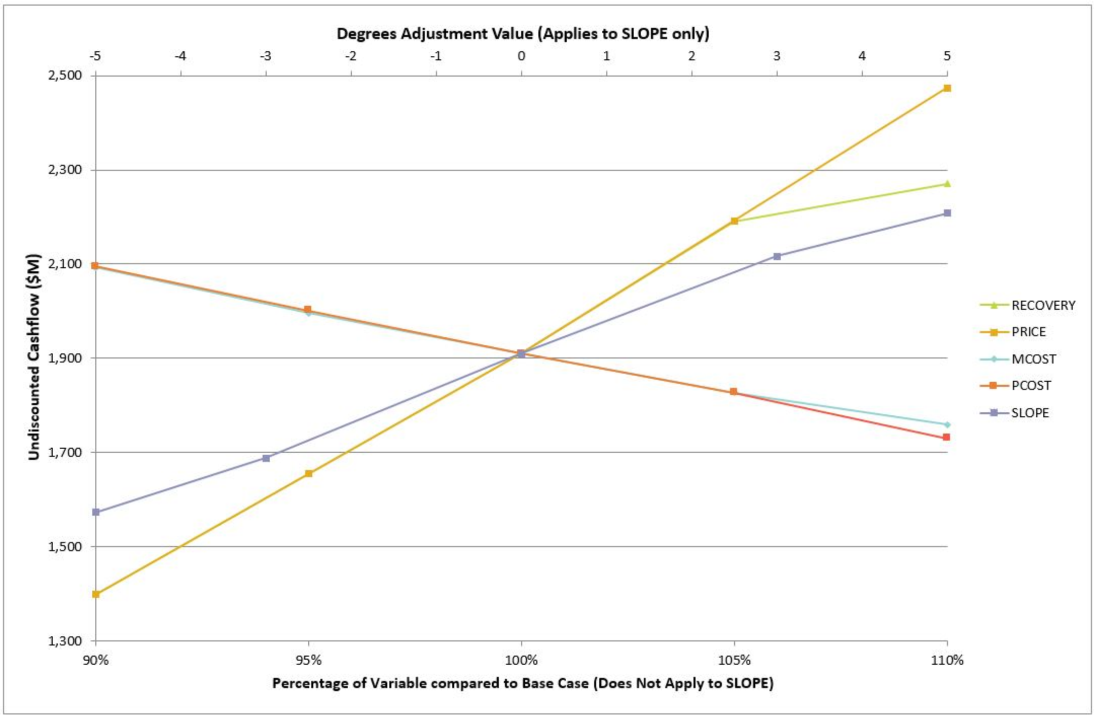

***Figure 15-4**: Pit Optimisation - Summary of Sensitivity Analysis*

15.4 Final Open Pit Design

The final pit design is the basis of the Ore Reserve estimate. The pit has been designed to be mined in four stages. Stages 1 and 2 comprise two independent pits, one in the northern end of the deposit which has a higher strip ratio but accesses higher average grades and the other in the southern end with a lower strip ratio and lower average grades. Stage 3 will combine the two starter pits and Stage 4 will cut back to the final pit design. Figure 15-5 shows a 3 dimensional view of the pit stages. Stage 1 is shown in green, Stage 2 in orange, Stage 3 in blue and Stage 4 in red.

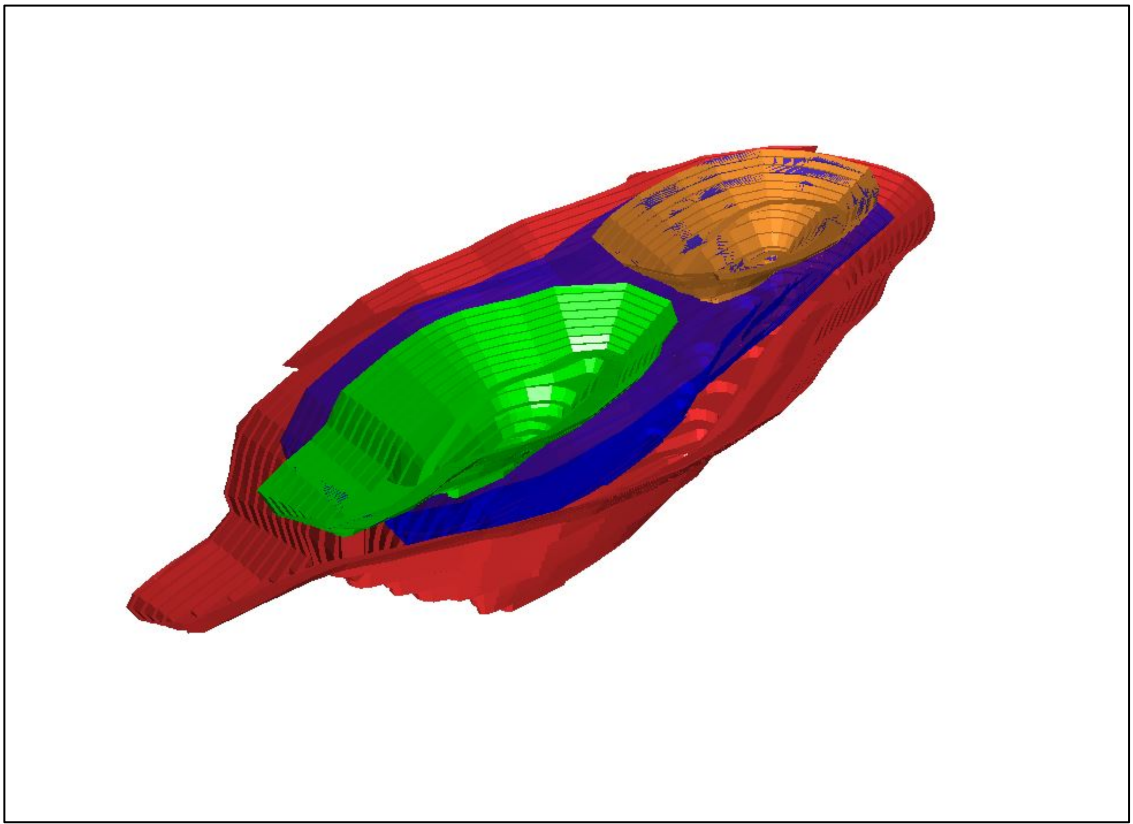

Figure 15-5: *Final Pit Design - 3D View of Pit Stages*

The final pit design was compared to the optimised pit shell 23 that formed the basis of the design. Table 15-3 shows a summary of the comparison.

Table 15-3: Final Pit Design Inventory vs Optimised Shell Inventory (Measured and Indicated Resource Categories)

Object	Ore Tonnes (Mt)	Grade (g/t)	Waste Tonnes (Mt)	Total Rock (Mt)	Strip Ratio w:o	Recovered Ounces (Moz)
Shell 23	90.2	1.20	249.0	339.3	2.76	3.48
Final Design	91.6	1.20	253.7	345.3	2.77	3.52
Comparison	102%	100%	102%	102%	100%	101%

Note: *Apparent differences may occur due to rounding.*

The cut-off grades used to determine the Ore Reserves within the final pit design were based on the recovery and cost parameters used for the Whittle pit optimisation. The variable cut-off grades are: for Oxide ore 0.35 g/t Au; Transition ore 0.39 g/t Au; and for Fresh ore 0.43 g/t Au.

15.5 Ore Reserve Reporting

The Ore Reserve was estimated by AMC in July 2016 in accordance with the JORC Code 2012. The Ore Reserve consists of 16% Proved and 84% Probable; Proved Ore Reserve is based on the Mineral Resource classified as Measured; and Probable Ore Reserve is based on the Mineral Resource classified as Indicated. No Inferred Mineral Resource has been included in the Ore Reserve. The Ore Reserve was estimated using a gold price of A$1,500 per ounce.

A summary of the Ore Reserve is shown in Table 15.4. An Ore Reserves estimate was presented on 19 October 2016[42]. Gold Road confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Ore Reserves estimates in the 19 October 2016 market announcement continue to apply and have not materially changed. There is no material difference in the information presented below concerning the Gruyere Gold Project Mineral Resources and the information of the 19 October 2016 announcement.

Table 15-4: Gruyere August 2016 Ore Reserve by Reserve Category

Ore Reserve Category	Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)
Proved	14.9	1.09	0.52
Probable	76.7	1.22	3.00
Total Ore Reserve	91.6	1.20	3.52

The Competent Person under the JORC Code 2012 for reporting this Ore Reserve is Mr David Varcoe who is an employee of AMC.

15.6 Factors Impacting on Ore Reserves

Gold Road is not aware of any mining, metallurgical, environmental, infrastructure, permitting, or other factors that may materially affect the current Ore Reserves at Gruyere.

[42] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, “Gruyere Feasibility Study Approved”

16 MINING METHODS

Gold Road engaged AMC to conduct the mining engineering study for the FS. The FS work confirmed the PFS outcomes that the mining will be carried out by open pit contract mining utilising conventional drill and blast, load and haul techniques and ancillary mining equipment provided by the mining contractor. Mining technical services and support will be provided by Gold Road.

The scope of the FS was limited to investigating the technical and economic viability of an open pit operation. There is, however, potential for LOM extensions through transitioning the open pit operation into an underground mine at depth.

Consultant Dempers and Seymour was commissioned to undertake the pit slope design for the Project and this work was used by AMC in the preparation of the open pit design.

16.1 Pit Slope Geotechnical Evaluation

The scope of work completed for the pit slope geotechnical evaluation for FS included additional diamond drilling, geotechnical core logging and associated laboratory test work on selected core samples. Structural and rock mass modelling, kinematic structural analysis, and probabilistic and deterministic structural and limit equilibrium analyses were carried out in determining slope parameters.

Geotechnical Data

Data applied to this geotechnical study consisted of geotechnical logging of 13 geotechnical drill holes drilled specifically for the FS, 21 drill holes from the Scoping Study and PFS, optical and acoustic televiewer survey data from 139 drill holes and core photos from 111 drill holes.

Figure 16-1 shows the spatial location of the logged drill holes relative to the proposed (April 2016) pit. The red holes show the drilling that was the basis of the Scoping Study. The green holes show the additional drilling that was completed between July 2014 and September 2015 for the PFS and the blue holes show the drilling that was completed for the FS between November 2015 and February 2016.

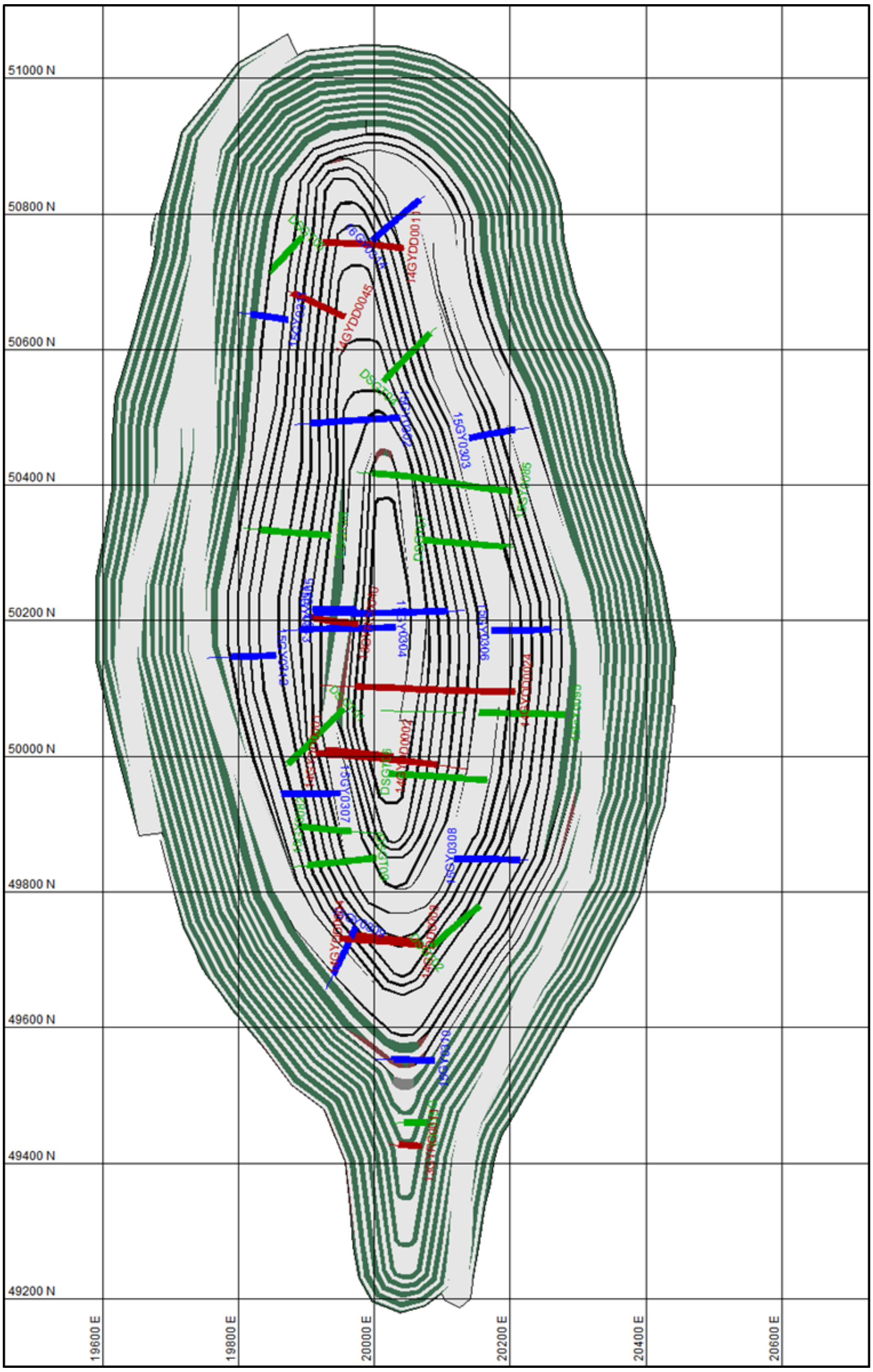

Figure 16-1: *Plan View Showing Logged Drill Holes (Geotechnical) and the Proposed Final Pit*

Significant Geotechnical Features

Significant geotechnical features were modelled from the geotechnical data set. The geotechnical database was interrogated to identify significant intersections that comprise broken zones with characteristics. Televiewer survey data was reviewed to determine structural orientations for broken zones and core photos were used to confirm identified significant features.

Five geotechnical feature types were identified with primary characteristics as follows:

- Type 1 - fault breccia related zone with significant thickness
- Type 2 - broken zone with brittle joints
- Type 3 - micro fractured zone, possibly related to Type 1 but at a smaller scale
- Type 4 - dyke related, broken along foliation
- Type 5 - unclassified, comprising a combination of mixed feature types with low geotechnical significance.
- Type 1 structures are considered to be the most significant geotechnical features in the context of pit slope designs. These features have been modelled as a single continuous structure averaging 7 metres in thickness and occasionally up to 15 metres and less than 25 metres of true thickness. The structure is continuous for 950 metres on strike and 700 metres down dip and is limited by available drilling data. This structure follows the foliation/ bedding orientation within the dominant joint set orientation in the area, with an average dip of 75° and dip direction between 70° to 110°. The structure is shown in Figure 16-2 and has been termed the Dorothy Hills Fault by the geology team.

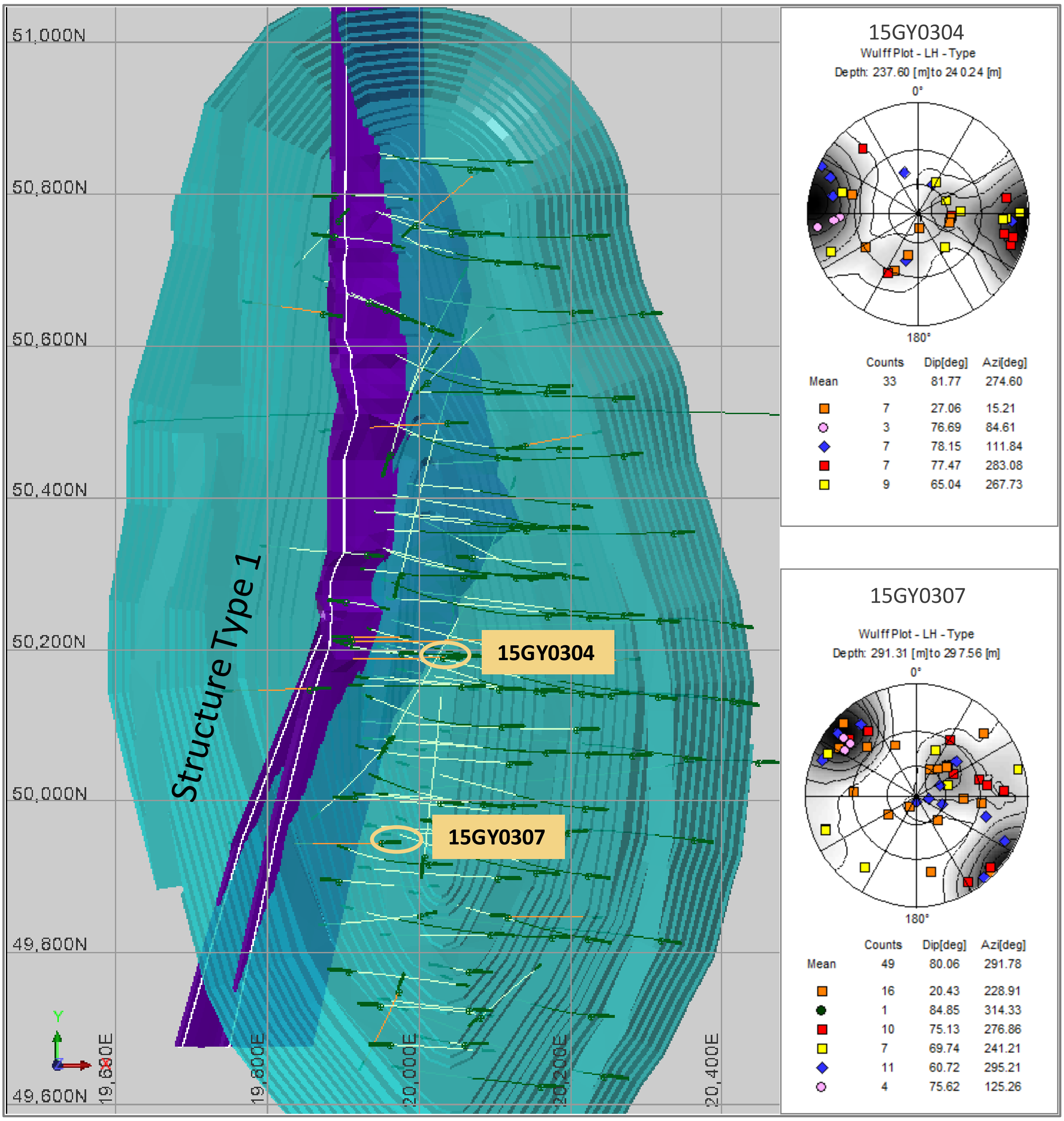

Figure 16-2: *Plan View Showing Structure Type 1*

Mining Rock Mass Model

The Rock Mass Rating (**RMR**) classification system is used in measuring the geotechnical and geological parameters and the in situ RMR is adjusted in consideration of the expected mining conditions to give the Mass Rock Mass Rating (**MRMR**). A 3D Mining Rock Mass Model (**MRMM**) was constructed based on geotechnical and structural logging of drill core, laboratory testing of selected drill core and defined rock types. The MRMM allows all the logged values and calculated geotechnical parameters to be represented in three dimensional block models. These models are constrained by the available geological and structural data and are analogous to a resource block model. For the Project, the following block models were created:

- RMR and MRMR.
- Inter ramp slope angle
- Hardness - rock strength index from geotechnical logging
- Rock Block size - RQD/Jn gives a measure of block size (where Jn is the joint set number)
- Discontinuity Shear Strength - Jr/Ja represents the roughness and frictional characteristics of the joint wall and infill material (where Jr is the joint roughness number and Ja is the joint alteration number)
- Geological Strength Index, Cohesion and angle of internal friction (Phi) to input directly for pit slope modelling
- Fracture Frequency to determine rock bridge for rigorous pit slope design.

The block models are interrogated to allow classification of the rock mass and to provide input parameters for rigorous pit slope analyses. The rock can be classified as follows:

- The weathered material is classified as Very Poor to Poor
- The main pit wall forming rock units are classified as Fair to Good
- The Fault structure is classified as Very Poor to Poor.

Summary rock mass characteristics are presented in Table 16-1.

Table 16-1: Summary of Rock Mass Characteristics

Rock Unit	Rock Strength (MPa)	Joint Condition	Fracture Frequency	RMR	MRMR
Oxide	1 - 10	Smooth and undulating with soft sheared fine infill	9 fractures/m Spacing 0.11 m	6 - 30 Average 21	5 - 24 Average 17
Transitional	4 - 25	Smooth and undulating with soft sheared coarse infill	7 fractures/m Spacing 0.14 m	20 - 40 Average 31	16 - 32 Average 25
Intermediate Volcanic	50 - 100	Smooth and undulating with non-softening fine infill	9 fractures/m Spacing 0.11 m	40 - 45 Average 42	32 - 36 Average 34
Tonalite	100 - 150	Smooth and undulating with non-softening medium infill	2 fractures/m Spacing 0.50 m	55 - 67 Average 61	44 - 54 Average 49
Basalt	100 - 135	Slickensided and undulating with non-softening medium infill	1.6 fractures/m Spacing 0.62 m	54 - 68 Average 59	43 - 55 Average 47
Fault	4 - 25	Rough and planar with soft sheared fine infill	40 fractures/m Spacing 0.02m	22 - 44 Average 34	17 - 35 Average 28
Intermediate Volcaniclastic	130 - 150	Slickensided and undulating with non-softening medium infill	1.8 fractures/m Spacing 0.55 m	55 - 67 Average 60	44 - 54 Average 49

Eleven geotechnical domains were defined on the basis of detailed analysis of the MRMM. Preliminary overall slope angles were then determined for each geotechnical domain for further rigorous analyses. Figure 16-3 shows the different geotechnical domains.

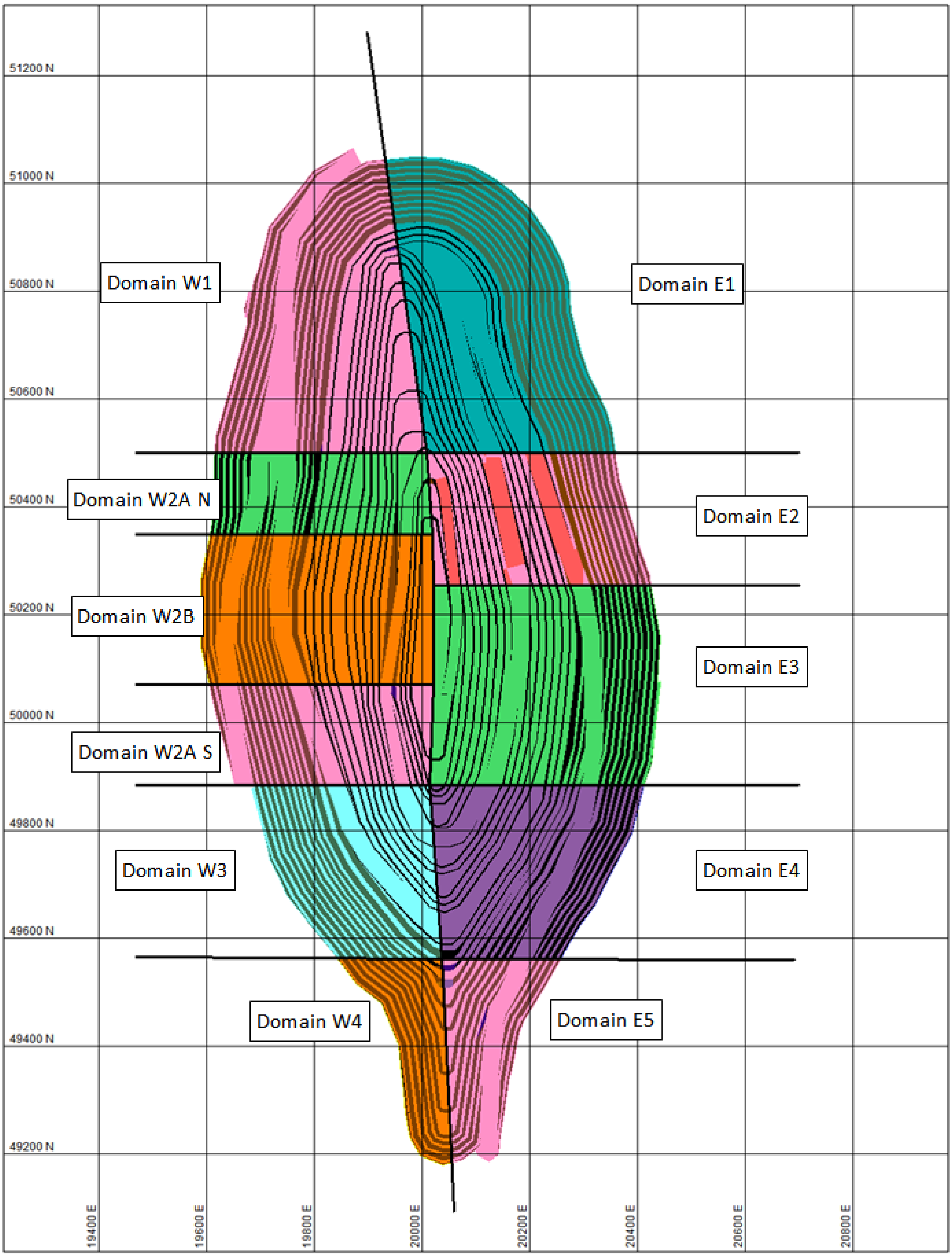

Figure 16-3: *Plan View Showing Geotechnical Domains on the Final Pit Design*

Rock Bridge

Overall pit slope failure is a combination of failure through intact rock (termed rock bridge) and structure. Hoek-Brown failure criteria is used to determine the likelihood of failure occurring through intact rock and the Barton shear strength envelope is used to assess the likelihood of structurally controlled failure. The strength data determined from either, or both, of these criteria are then independently applied to limit equilibrium models to assess slope stability. However, the respective strength parameters are usually adjusted to allow for a rock bridge or structure.

In order for both methods to be applied simultaneously to slope stability design, fracture frequency determined in the MRMM is used to determine the ratio of rock bridge to structure expressed as a percentage of rock bridge. This ratio per rock type within each geotechnical domain is input into limit equilibrium models. Appropriate strength parameters are then assigned from Hoek-Brown and Barton which have been statistically analysed from parameters modelled in the MRMM.

Laboratory Testing

Representative samples of predominantly fresh rock were recovered from diamond drilling during the PFS and FS. A summary of average test results is presented in Table 16-2 and Table 16-3. Detailed laboratory test results and analyses are presented in the Dempers and Seymour report (Reference 4).

Table 16-2: Summary of Laboratory Rock Strength Test Results

Rock Unit	UCS (MPa)	Dry Density (t/m3)	Youngs Modulus (GPa)	Poissons Ratio	Brazilian Tensile Strength (MPa)	S-wave Velocity (m/s)	P-wave Velocity (m/s)
Oxide	10	2.06	-	-	-	-	-
Transitional	27	2.54	51	0.15	5.0	2,763	5,141
Tonalite	163	2.69	85	0.23	18.8	3,231	5,969
Basalt	79	2.87	78	0.30	18.5	3,217	6,306
Fault	3	2.56	1	0.37	0.5	-	-
Intermediate Volcaniclastic	75	2.75	62	0.23	13.7	3,143	6,075

Table 16-3: Summary of Laboratory Shear Strength Test Results

Rock Unit	Triaxial Compression - Cohesion (MPa) - Single Shear Failure	Triaxial Compression - Friction Angle (°) - Single Shear Failure	Triaxial Compression - Cohesion (MPa) - Multiple Shear Failure	Triaxial Compression - Friction Angle (°) - Multiple Shear Failure	Direct Shear - Cohesion (kPa)	Direct Shear - Friction Angle (°)
Transitional	-	-	-	-	25	35.5
Basalt	7.01	28.1	21.8	43.2	46	39.0
Intermediate Volcaniclastic	21.42	32.2	15.0	46.2	37	33.0

Test work data inputs and outcomes are considered to be appropriate for the FS. The results show that geotechnical conditions at Gruyere are likely to be consistent with general conditions in the Eastern Goldfields of Western Australia.

Structural Assessment

The structural dataset consists of 81,914 structures measured in diamond core and from televiewer survey data. In addition to the orientation of the defect, entries contained information on the micro-scale roughness and joint infill.

Principal Rock Types

Using the available geological model, the structural data was coded on the principal rock types as follows:

- ABU - basalt 5% of data
- AGT - tonalite 15% of data
- AIU - intermediate volcanic < 1% of data
- FAULT - interpreted fault 2% of data
- GENROCK - intermediate volcaniclastics 22% of data
- OXIDE - oxidised material 11% of data, and
- TRANS - transitional weathered rock 44% of data.

The dataset was subsequently interrogated based on rock type with exception for AIU due to lack of structural data (five measurements).

Structural Stability Analysis

Eight main wall orientations were assessed for structural stability; three each on the east and west walls and one each on the north and south walls. Using the lithology model as a guide, structural stability analyses were carried out for each rock type and wall orientation. As most of the rock types are present in both sides of the proposed pit, the structural data for each rock type was separated into east and west to better represent the structural orientations found on each side of the pit. The data for the FAULT was not split as it only intersects the west wall of the pit. Parameters were assigned for shear strength and density based on the averages for each rock type from the laboratory test results. Stability analyses were carried out on the following batter and berm geometries for the weathered and fresh rock types:

- Weathered rock - (OXIDE and TRANS) 10 metre high batters, at 45°, 50° and 55° with 5 metre wide berms
- Fresh rock - (ABU, AGT and GENROCK) 20 metre high batters, at 65°, 70°, 75°, 80° and 85° with 8 metre wide berms.

The failure modes for each wall were assessed using kinematic analysis to ascertain the sets that have the potential to cause failure. Further deterministic analyses were then undertaken on the identified potential failure sets for toppling, wedge and planar failure.

The results of the probabilistic methods for the failure modes are reported as a percentage of failed poles or probability of failure (**PoF**), classified using a risk ranking shown in Table 16-4.

Table 16-4: Ranking of Probabilistic Results

Rank	% Failed
1	>50
2	25 - 50
3	5 - 25
4	0 - 5

The analyses indicate that greater than 50% of failures (Risk Rank 1) have a frequency of greater than 15% when batters were steeper than 75°. A summary of Risk Rank 1 for each wall is shown in Table 16-5.

Table 16-5: Risk Rank 1 Frequency Summary

Batter Angle (°)	East Wall % of Risk Rank 1	West Wall % of Risk Rank 1
75	0	15
80	11	35
85	75	50

Pit Slope Modelling

Deterministic analysis using two dimensional limit equilibrium analyses and finite element slope stability analyses applying the Shear Strength Reduction method was carried out. This method involves a systematic search for a stress reduction factor or FOS that brings a slope to the limits of failure. Material strength properties were determined from the MRMM for each rock unit in each geotechnical domain. The analyses were carried out for each geotechnical domain shown in Figure 9-8.

The assessment of each geotechnical domain included the calculation of FoS and Shear Strength Reduction Factor. Deterministic analyses were carried out for the design sections.

The weathered rock is variable and comprises Saprock, high plasticity silts, relict structure and clayey silts. Material strength properties for the weathered profile and transitional profile were based on the geotechnical logging and laboratory test data. Inter ramp slope angles of 38° to 43° with batter angles of 45° to 55° were analysed and returned FoS of >1.2 which are considered appropriate for this study and are within the guidelines published by the DMP in Western Australia.

Overall slope angles from 50° to 65° in the fresh rock were analysed. The analyses were carried out under dewatered slope conditions.

Seismic loading was not applied during the analyses since seismicity was not considered to be a risk to overall wall performance.

The results from these analyses are presented in Figure 16-4 and Figure 16-5.

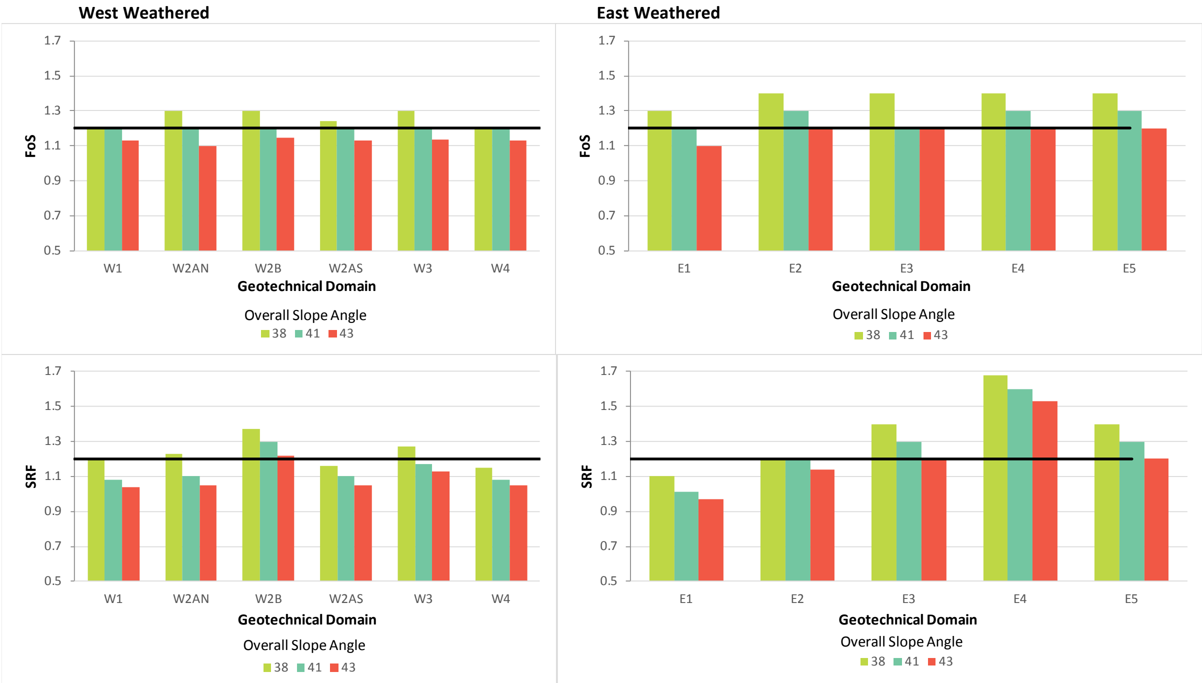

Figure 16-4: *Limit Equilibrium and Stress Reduction Analyses - Weathered Profile*

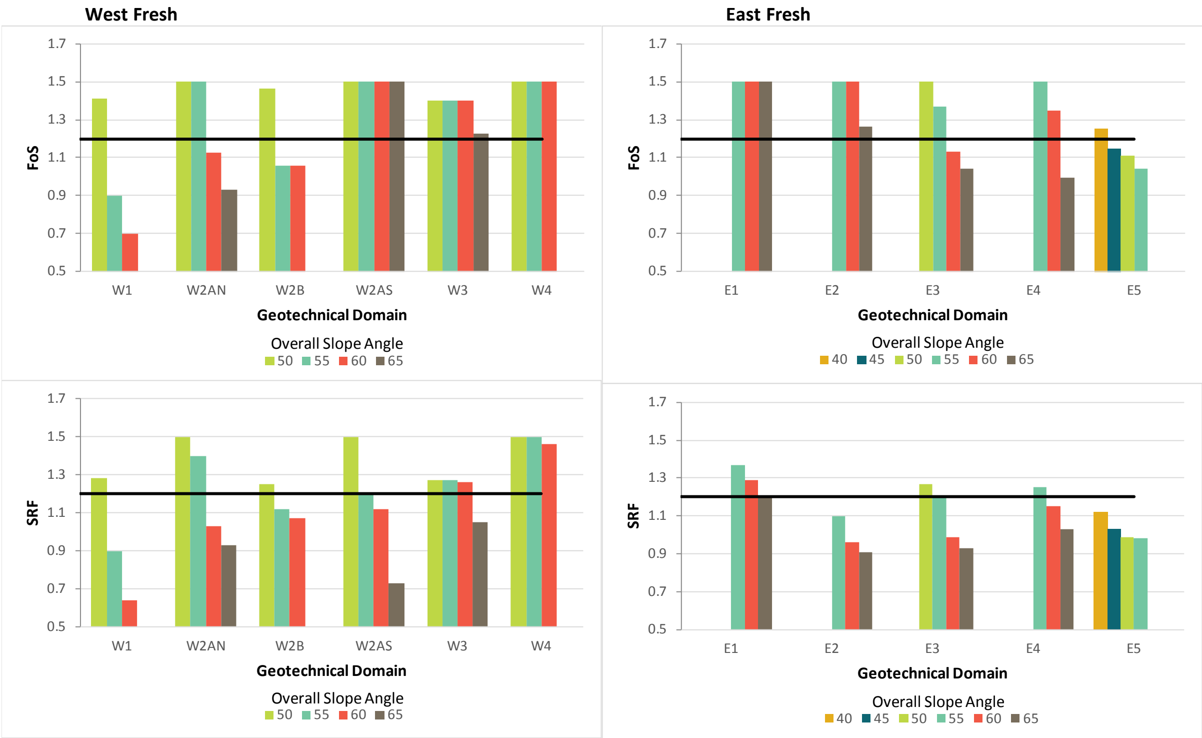

Figure 16-5: *Limit Equilibrium and Stress Reduction Analyses - Fresh Profile*

Pit Slope Configuration

Two pit slope configurations were developed based on the analyses completed for the FS as follows:

- **Base Case**. Inter ramp slope angles return minimum FoS of 1.2 for all geotechnical domains and Shear Strength Reduction Factor of 1.2 for all domains except E2 and E5. The probability of batter scale failure is less than 10% for most failure modes analysed.
- **Feasibility Study Case**. A steeper design than the Base Case with minimum FoS for inter ramp angles of 1.2 but in some geotechnical domains on the east and west walls the Shear Strength Reduction Factor is less than 1.2. The probability of batter scale failure increases, exceeding 25% for a number of failure modes analysed.
- Table 16-6 summarises the slope angles used for pit design.

Table 16-6: Pit Design Slope Angles

Geotechnical Domain	Material	Overall Slope Angle (º)
West wall	Weathered	38 to 41
	Fresh	45 to 57
East wall	Weathered	38 to 42
	Fresh	38 to 48

Monitoring and Slope Management

A slope monitoring programme will be implemented at the start of mining to predict and minimise the adverse effects of instability on slopes. The programme will consist of tension crack mapping, survey monitoring and ground water monitoring.

Geotechnical Assessment Conclusion and Recommendations

The following conclusions and recommendations were made by Dempers and Seymour.

The rock mass at Gruyere is classified as Fair to Good. The rock mass comprises sub-vertical foliation and low angle joints. The rock mass is influenced by low angle healed joints and foliation planes which reduce the rock mass strength.

The oxide profile averages 40 metres in thickness but is variable from 5 metres to 77 metres. The material in the zone comprises clayey silts, high plasticity silts, relict structure, Saprock, laterite, calcrete and gravels.

The transitional profile averages 35 metres in thickness varying from 9 metres to 85 metres. The profile is highly fractured and has zones of highly weathered rock within good zones of transitional rock.

The west wall of the pit is dominated by sub-vertical foliation dipping steeply towards the east and a major fault striking north-south. Foliation planes are generally smooth to rough and planar and are considered to be continuous. The dip of the foliation planes ranges from 65° to 85° with an average dip of 75° degrees. Batters for the Base Case have been designed at 75° taking the average dip of the foliation into consideration. For the FS Case, batters are mined steeper than the average dip of the foliation planes and increased occurrences of slabbing of the batters along the foliation can be expected. The pit design has taken this into account and it is anticipated that slabbing will be adequately managed by the wide berms incorporated in the design.

The fault zone is contiguous with the western wall and is approximately 5 metres to 10 metres thick with a dip and orientation similar to the foliation. However, the dip can vary from 60° to 85°. The fault will impact on wall stability and failures may occur when batters are steeper than the fault potentially causing large failure volumes which may need to be caught by the berms. To mitigate this risk, wide berms have been designed where the fault is most likely to be exposed and where the rock mass is less competent and these wider berms have been incorporated in the FS pit designs.

The east wall has sub-vertical foliation planes dipping into the wall towards the east and shallower defect sets dipping towards the west. The influence of the healed low angle defects is evidenced in UCS and triaxial test data where failures have occurred mainly on healed defects dipping at shallow angles towards the west and healed foliation planes.

In general terms, inter ramp failure on the west and east walls will be structurally controlled. Foliation planes together with low angle structures can result in step path/toppling type failure if this wall is mined too steeply.

Two pit slope configurations were developed:

- Base Case with an overall pit slope angle of 46° (varying from 41° to 50°) for the east wall and 44° (varying from 40° to 48°) for the west wall
- FS Case with an overall pit slope angle of 50° (varying from 45° to 54°) for the east wall and 48° (varying from 45° to 51°) for the west wall.

The design configurations are based on dry dewatered slopes. Both options were designed within the guidelines published by the DMP with FoS greater than 1.2 for the overall pit slopes. Whilst the FS Case presents a steeper slope design with a higher risk profile than the Base Case, it is considered to be a practical option. Issues associated with batter scale failures will be managed with wider berms to catch failed material and regular pit monitoring.

The ultimate FS pit and interim cutbacks were designed using the recommended FS Case pit slope configuration. These pits have been assessed for overall stability using numerical modelling techniques. In all cases, the FoS are equal to or greater than 1.2.

The following activities are recommended to be carried out during Operations:

- Pit mapping will be carried out as mining proceeds to determine the intensity and orientation of the structural sets identified during this study.
- A monitoring and slope management programme will be implemented to establish benchmark criteria for the pit and then provide ongoing data to determine overall slope performance.
- A ground control management plan will be implemented when mining commences.

16.2 Open Pit Design

The open pit design process included the design of pit stages and ramp access to the bottom of the pit subject to geotechnical recommendations and mining fleet requirements. The selection of interim pit shells was guided by the objective of maximising cash flows in the initial years of operation with due consideration for practical mining parameters. Initial mine schedules were developed to test the pushback sizes. A number of iterations were developed for Stages 1, 2 and 3 to balance the supply of sufficient ore for the first five years of the schedule while minimising waste stripping and deferring Stage 4 waste. The schedule was shown to be most sensitive to the size of the Stage 3 pit. Deferring too much waste to Stage 4 results in either a very high stripping requirement or lack of ore supply in years 6 and 7.

The Stage designs were prepared utilising optimum ramp exit points for waste material. Waste from Stages 1 and 2 is used for construction of the ROM, initial construction of the TSF with the balance reporting to the eastern dumps. Stage 3 waste is placed on the TSF and eastern dumps. Stage 4 is the only stage that has waste placed on western and northern dumps. A minimum mining width of 50 metres is established between Stages with minor areas reducing to 40 metres.

The pit has been designed to be mined in four stages. Stages 1 and 2 comprise two independent pits, one in the northern end of the deposit which has a higher strip ratio but accesses higher average grades and the other in the southern end with a lower strip ratio and lower average grades. Stage 3 will combine the two starter pits and Stage 4 will cut back to the final pit design.

Figure 16-6 to Figure 16-9 show plan views of the final pit profile for each of the four pit stages.

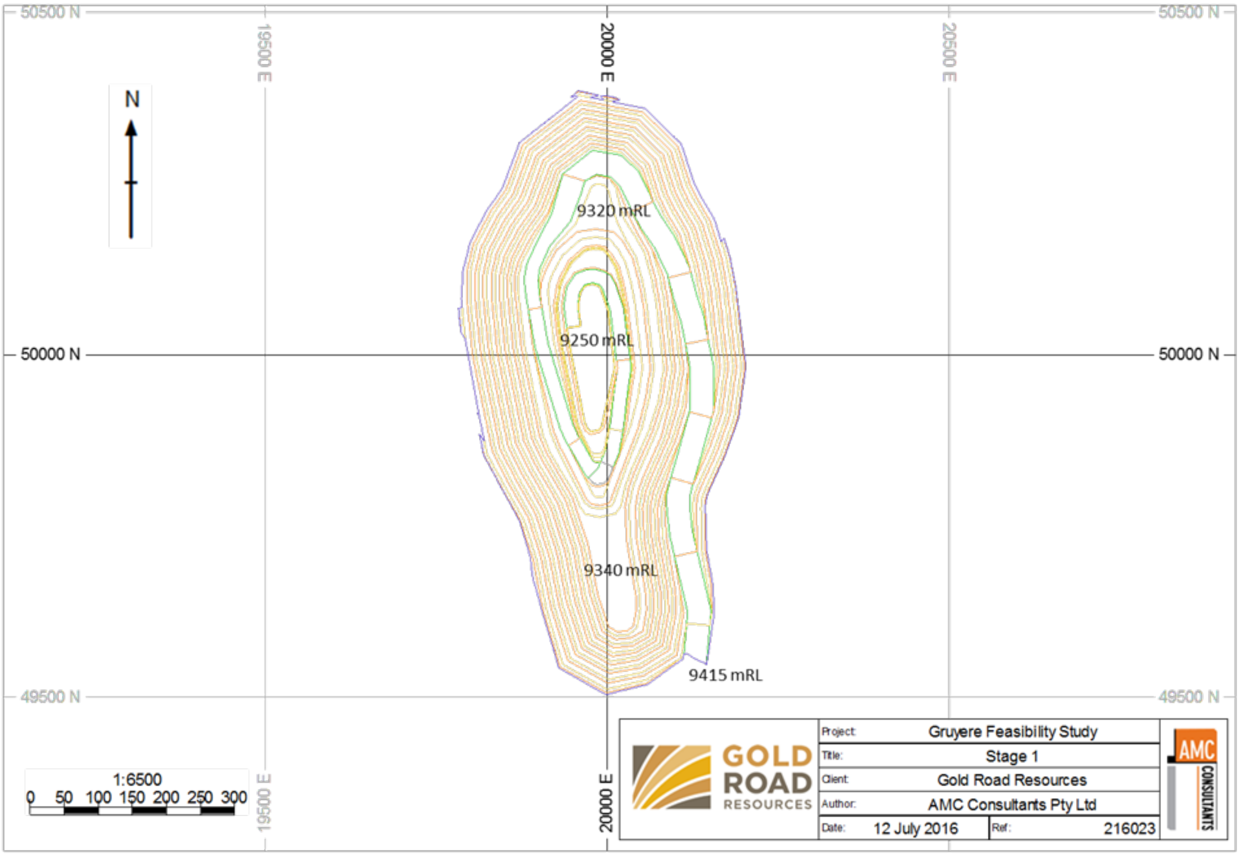

Figure 16-6: *Plan View Showing Stage 1 of the Pit Design (Gruyere Grid)*

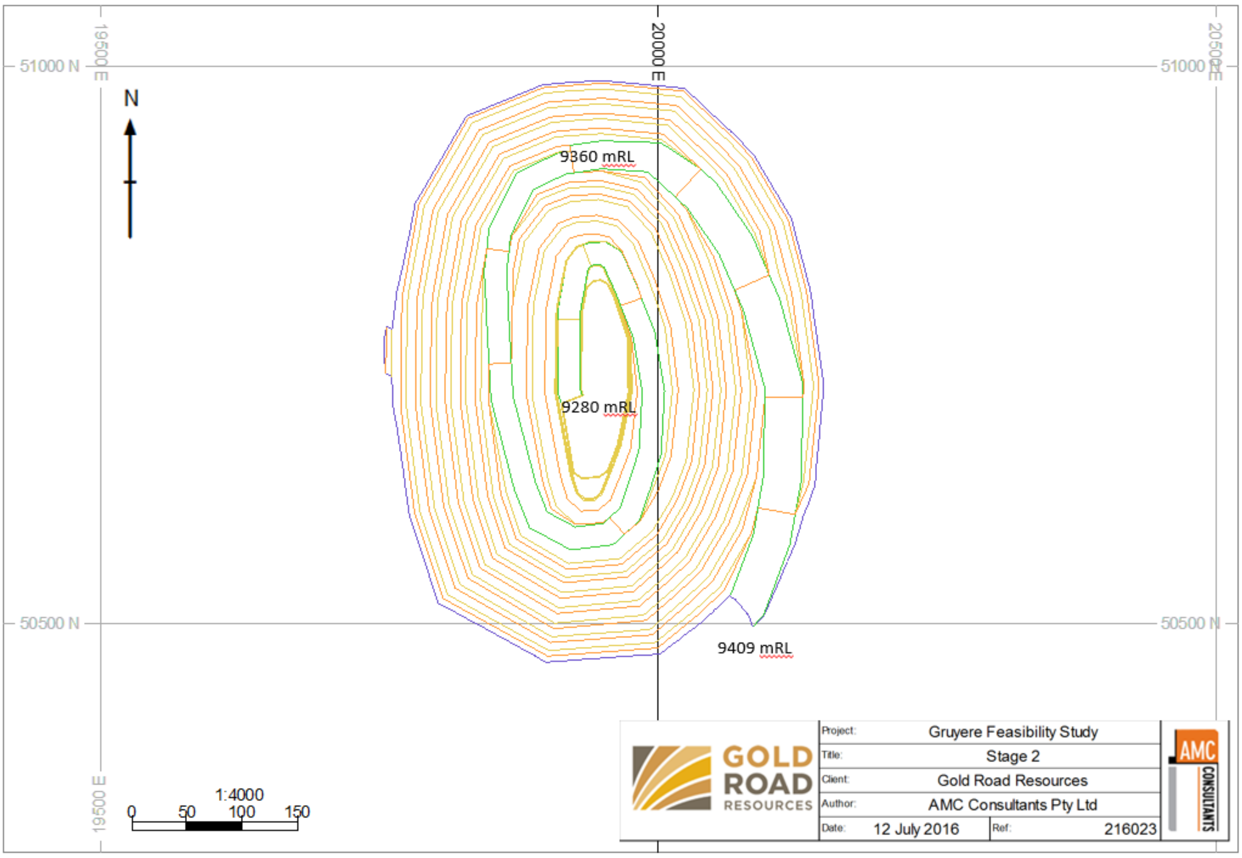

Figure 16-7: *Plan View Showing Stage 2 of the Pit Design (Gruyere Grid)*

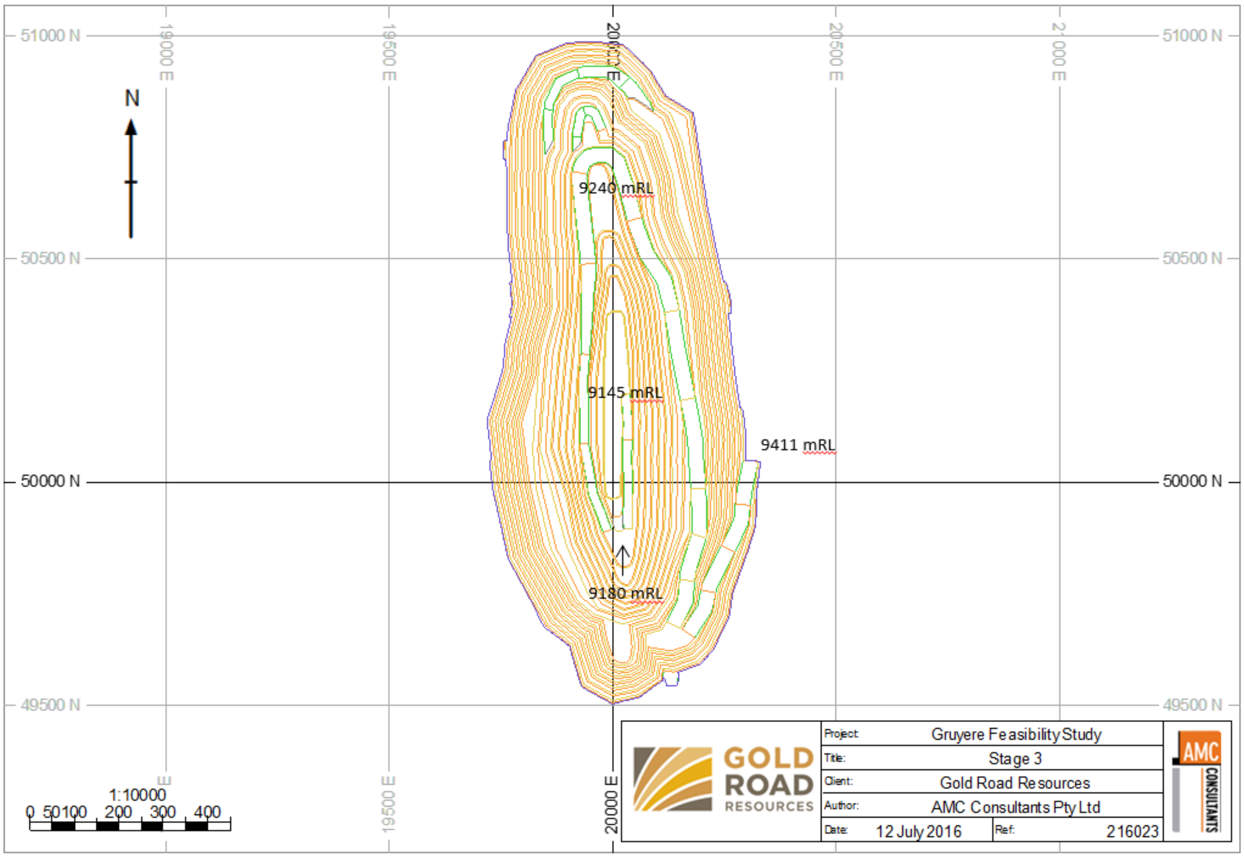

Figure 16-8: *Plan View Showing Stage 3 of the Pit Design (Gruyere Grid)*

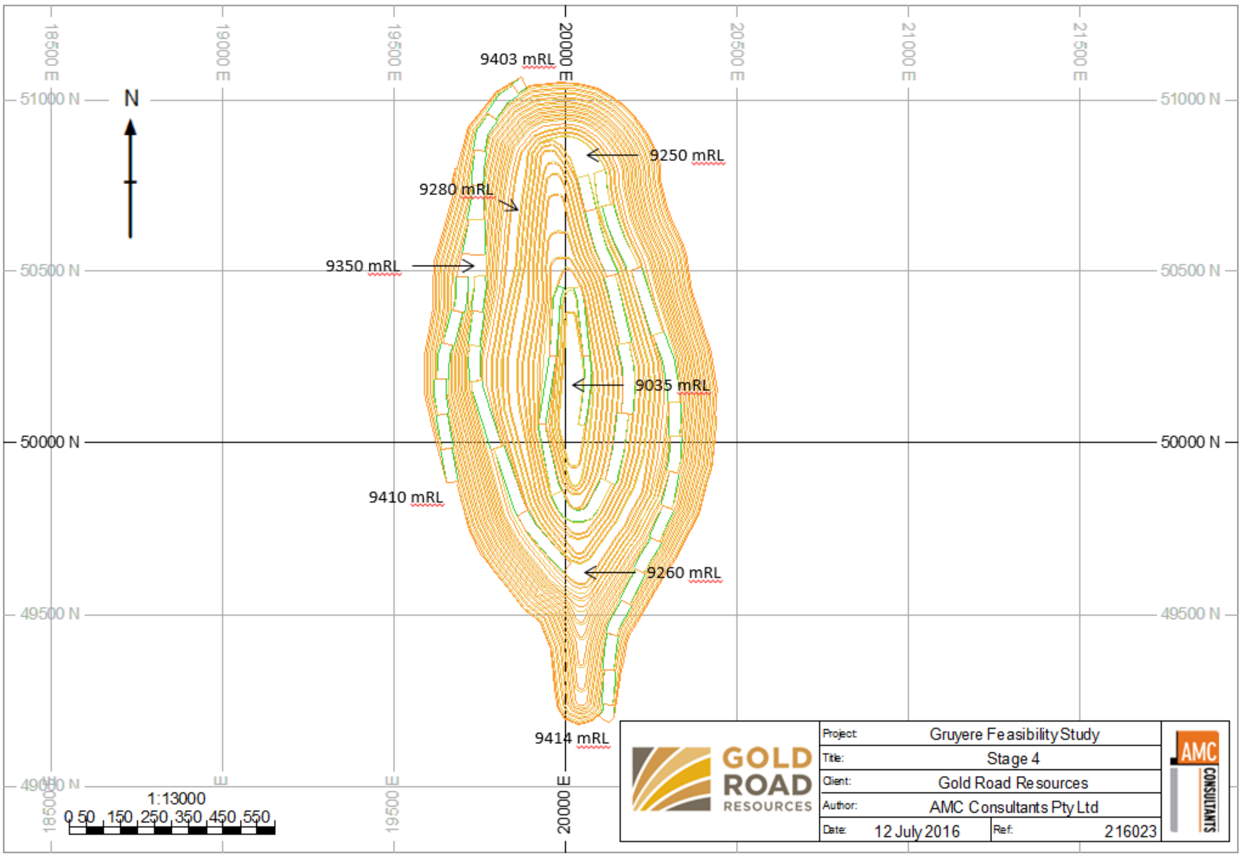

Figure 16-9: *Plan View Showing Stage 4 of the Pit Design (Gruyere Grid)*

Figure 16-10 shows a typical cross sectional view in the southern end of the pit showing Stage 1, Stage 3 and Stage 4, and the mining model with colour-coded gold block grades above 0.5 g/t. Stage 2 is off section as this stage is in the northern end of the pit.

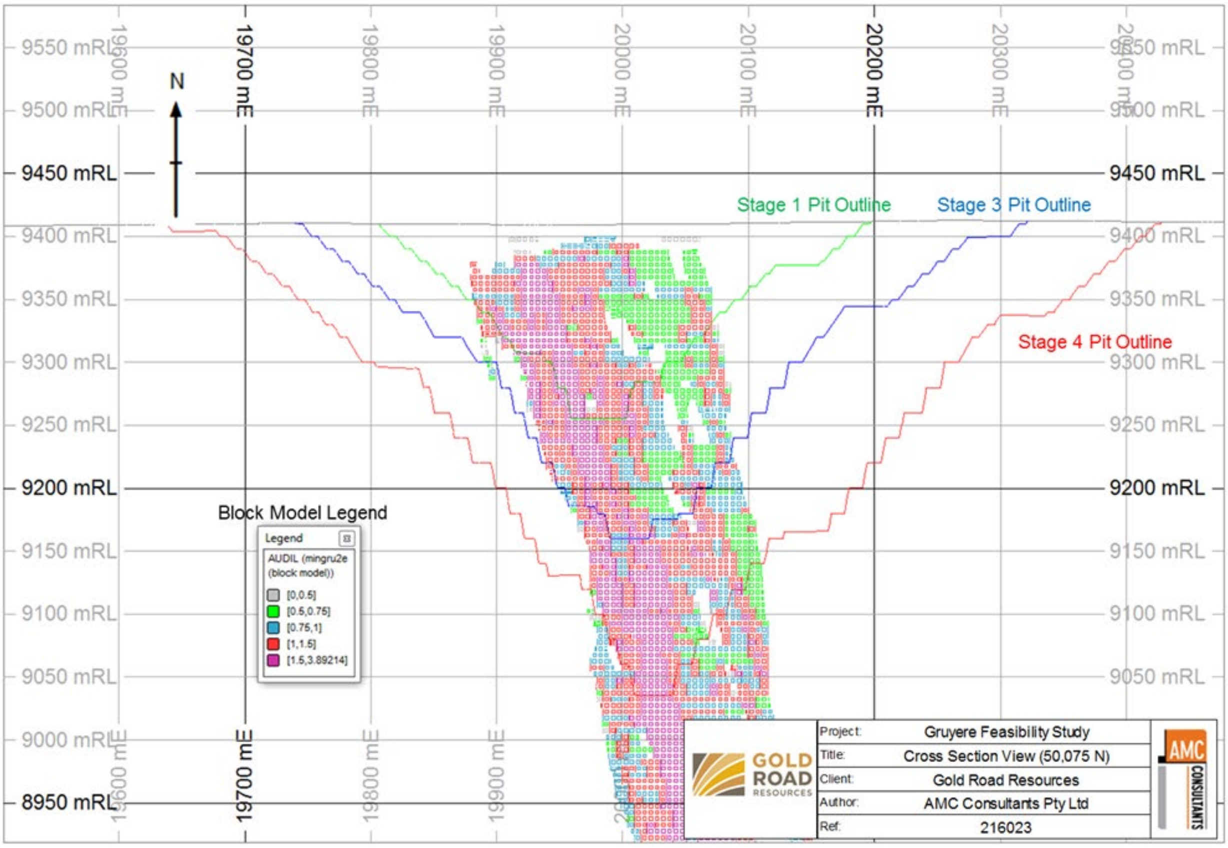

Figure 16-10: *Cross Sectional View (50,075N) Showing Pit Stages and Diluted Mining Model*

Ramp and Haul Road Design

The geometry and structure of the haul roads have been designed with consideration of the safe and economic operation of the pit. For the purposes of the FS, there has been no separation of light and heavy vehicles via separate haul roads. The roads will be constructed in a conventional manner utilising mine sourced materials. Ongoing road maintenance will be in the form of grading, watering, rolling and sheeting as required.

Ramps and haul roads have been designed to accommodate a Cat 789 haul truck or equivalent, being the largest mobile equipment in the proposed mining fleet. Double lane ramps and haul roads are designed to a total width of 35 metres. Single lane ramps are designed in the lower benches of the pit to a width of 20 metres. All ramps are designed with a maximum gradient of 10% with provision for safety berms and water drainage facilities.

Mining Inventory

The mining inventory within the final pit design was evaluated in Datamine against the mining model with breakdown by rock type and by mining stage, both as quantities and proportions, as shown in Table 16-7 to Table 16-10.

Table 16-7: Life of Mine Mining Inventory by Rock Type

Material Type	Ore Tonnes (Mt)	Grade (g/t)	Contained Gold (Moz)	Waste Tonnes (Mt)	Total Tonnes (Mt)
Transported	0	0	0	4.1	4.1
Permian	0	0	0	23.4	23.4
Saprolite	1.0	0.86	0.03	32.9	33.9
Saprock	11.3	1.12	0.41	33.7	45.0
Transitional	3.5	1.13	0.13	16.2	19.7
Fresh	75.8	1.21	2.96	143.4	219.1
Total	91.6	1.20	3.52	253.7	345.3

Note: *Apparent differences may occur due to rounding.*

Table 16-8: Proportional Life of Mine Mining Inventory by Rock Type

Material Type	Ore Tonnes (%)	Contained Gold (%)	Waste Tonnes (%)	Total Tonnes (%)
Transported	0.0	0.0	1.6	1.2
Permian	0.0	0.0	9.2	6.8
Saprolite	1.1	0.8	13.0	9.8
Saprock	12.3	11.5	13.3	13.0
Transitional	3.8	3.6	6.4	5.7
Fresh	82.7	84.0	56.5	63.5
Total	100	100	100	100

Note: *Apparent differences may occur due to rounding.*

Table 16-9: Life of Mine Mining Inventory by Mining Stage

Item	Unit	Stage 1	Stage 2	Stage 3	Stage 4	Total
Ore inventory	Mt	18.1	2.7	34.5	36.2	91.6
Contained gold	Moz	0.65	0.16	1.24	1.48	3.52
Grade	g/t	1.11	1.83	1.11	1.27	1.20
Waste inventory	Mt	15.1	13.1	65.3	160.3	253.7
Total inventory	Mt	33.2	15.8	99.8	196.5	345.3
Stripping ratio	W:O	0.8	4.8	1.9	4.4	2.8

Note: *Apparent differences may occur due to rounding.*

Table 16-10: Proportional Life of Mine Mining Inventory by Mining Stage

Item	Unit	Stage 1	Stage 2	Stage 3	Stage 4	Total
Ore inventory	%	20	3	38	40	100
Contained gold	%	18	5	35	42	100
Waste inventory	%	6	5	26	63	100
Total inventory	%	10	5	29	57	100

Note: *Apparent differences may occur due to rounding.*

16.3 Mining Schedule

Mine scheduling was completed utilising Minemax scheduling software based on a quarterly scheduling period. The Minemax software generates an NPV optimised schedule based on criteria and constraints set by the user.

The mining schedule is structured to optimise cash flows during the initial years of operation (years one to five) in order to minimise the Project payback period and to maximise the Project's debt carrying capacity.

An initial schedule was developed based on annual periods to assist with the development of a dump strategy. This schedule scenario was developed by allowing the scheduler to choose from different dump locations at the same time as optimising ore and waste mining costs and the Project revenue stream. The LOM dump strategy was determined and set for all subsequent scheduling iterations. The dump schedule is constrained to meet fill volume requirements for TSF and ROM pad construction activities.

Quarterly based schedules were developed to focus on maximising value (measured as discounted cash flow) by maximising revenue and minimising waste movement in the initial five year period and to also ensure schedule practicality.

The TMM per quarter was smoothed to ensure consistent TMM over each quarter (annually). A peak TMM of 7.25 Mt per quarter was set during the first five years of the schedule by testing the lowest TMM that ensured continuous ore supply. When the cutback for Stage 4 commences in year six, it will be necessary to increase the TMM to 11 Mt per quarter to ensure ore supply in later years.

The mining schedule has been constrained by setting a maximum vertical advance rate of 60 metres per annum in Stages 1 and 2 and 80 metres in Stages 3 and 4 (due to more bulk waste mining activity in Stages 3 and 4) to allow sufficient time for drill and blast, load and haul, dewatering and grade control. Stages 3 and 4 were split into north and south ends to allow a lag in bench advance between each end of the pit. The maximum vertical lag between benches was set at 20 metres.

Initial mine development and pre-stripping activities are scheduled to defer capital expenditure and land disturbance, and to provide sufficient material required to construct the initial TSF embankment, site roads and ROM pad.

The Minemax schedule was imported into the AMC OPMincost cost estimation system (**Cost Model**) to develop mining costs. The schedule that was developed in quarters was then modified in the Cost Model to ensure the schedule for the first year met monthly requirements for provision of specific construction material including TSF embankment material. As a result, some waste production scheduled by Minemax from Stages 3 and 4 was deferred to later periods but without adverse impact on ore presentation.

In the following Figures 16-11 to 16-14, Y1-Q1 is the starting point corresponding to the quarter ending in June 2018.

Figure 16-11 shows TMM by period split into ore and waste.

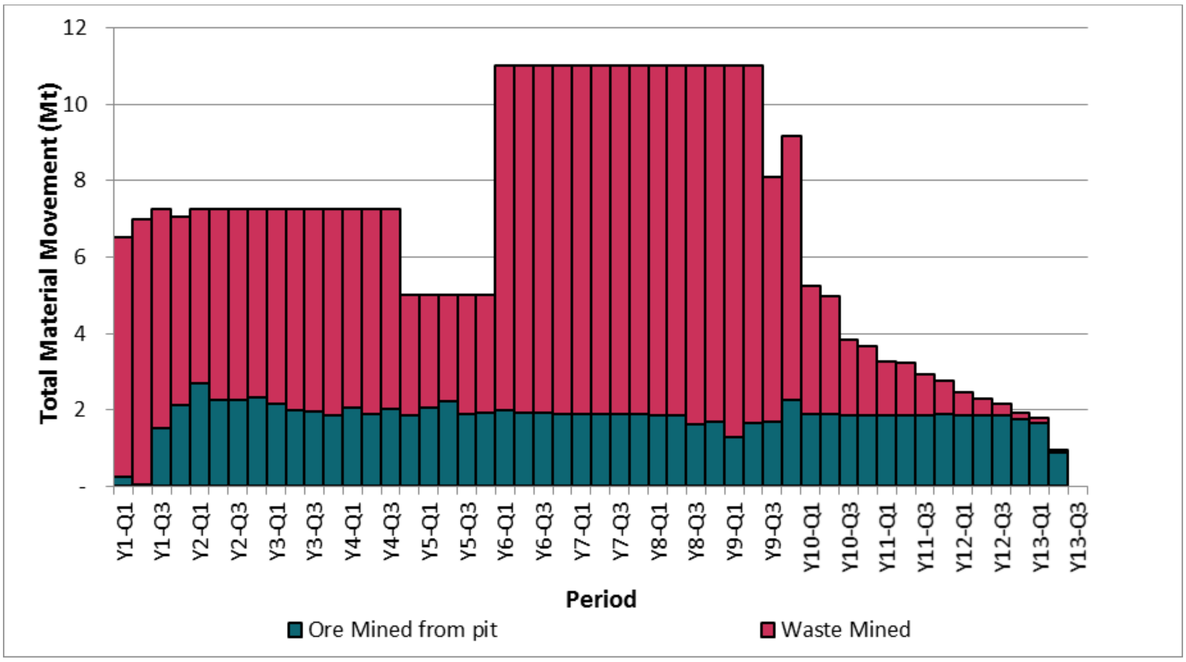

***Figure 16-11**: Total Material Movement by Period by Ore and Waste*

Figure 16-12 shows material movement by period by pit stage. The Minemax schedule output shows some Stage 3 and Stage 4 material mined in Y1-Q1 due to vertical advance limits on Stages 1 and 2 balanced with achieving a smooth mining schedule. This material has been rescheduled to a later period in the Project life within the Mining Cost Model without impacting ore presentation. Quaternary material which is located generally within 10 metres of the natural surface has been assumed to be free dig, with production rates the same as blasted oxide material.

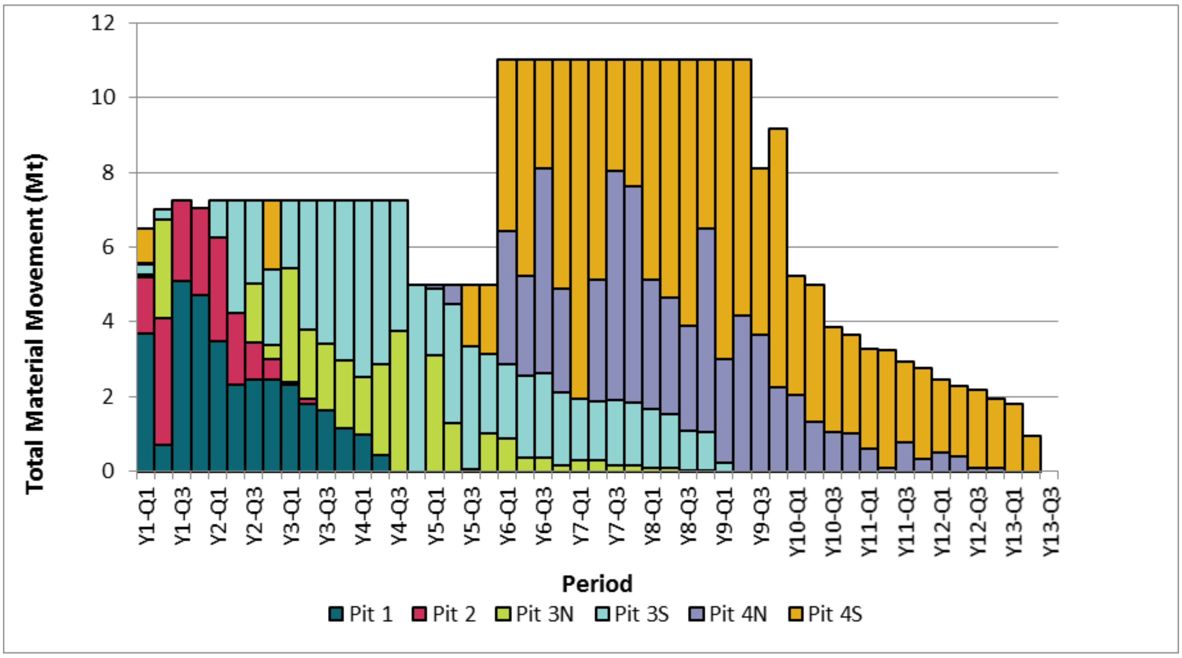

***Figure 16-12**: Total Material Movement by Period by Pit Stage*

Figure 16-13 shows the ore processing schedule by source and plant feed grade, with material designated "Direct" including material rehandled to/ from stockpiles within a quarter, and material "Reclaimed" includes material rehandled from long term stockpiles. The proportion of ore directly tipped into the crusher from the pit is expected to be 50% over the LOM.

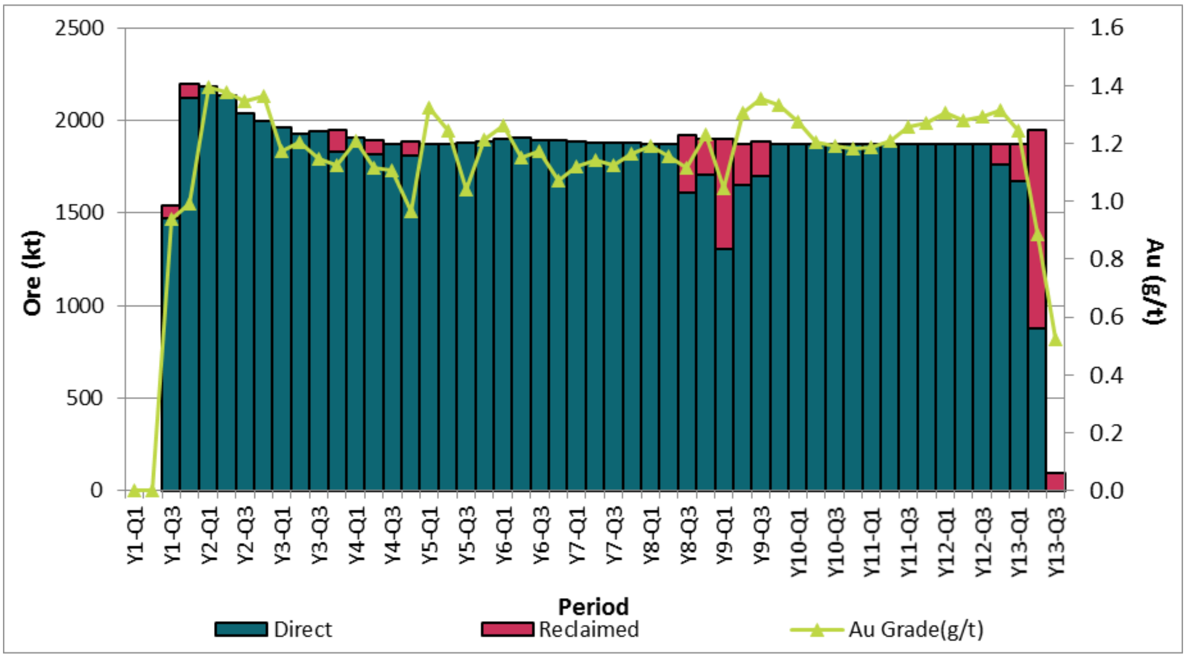

***Figure 16-13:** Plant Feed Tonnes and Grade by Source ("Direct") includes ROM rehandle*

The ore processing schedule allows variable process throughput rates and comminution grind sizes for ore based on material type and gold grade. The maximum process plant throughput rate is set at 2.2 Mt per quarter for 100% Oxide ore feed (equivalent to 8.8 Mtpa) and 1.875 Mt per quarter for 100% fresh ore (equivalent to 7.5 Mtpa). The optimum grind size is determined by the Minemax scheduling software in consideration of net block values and material availability in each quarter. Grind size options of 106, 125 and 150 µm are selectable by the Minemax scheduler with finer grinds having a higher cost and higher recovery (assessed using a gold price of A$1,500 per ounce). In nearly all periods the optimum grind size selected by Minemax was 125 µm. The scheduler did not select the 106 µm grind option which is only optimal when ore grades exceed 2.8 g/t (Figure 16-14). It has been assumed that a grind size change may occur once per month with no impact of this changeover on operating costs or plant production.

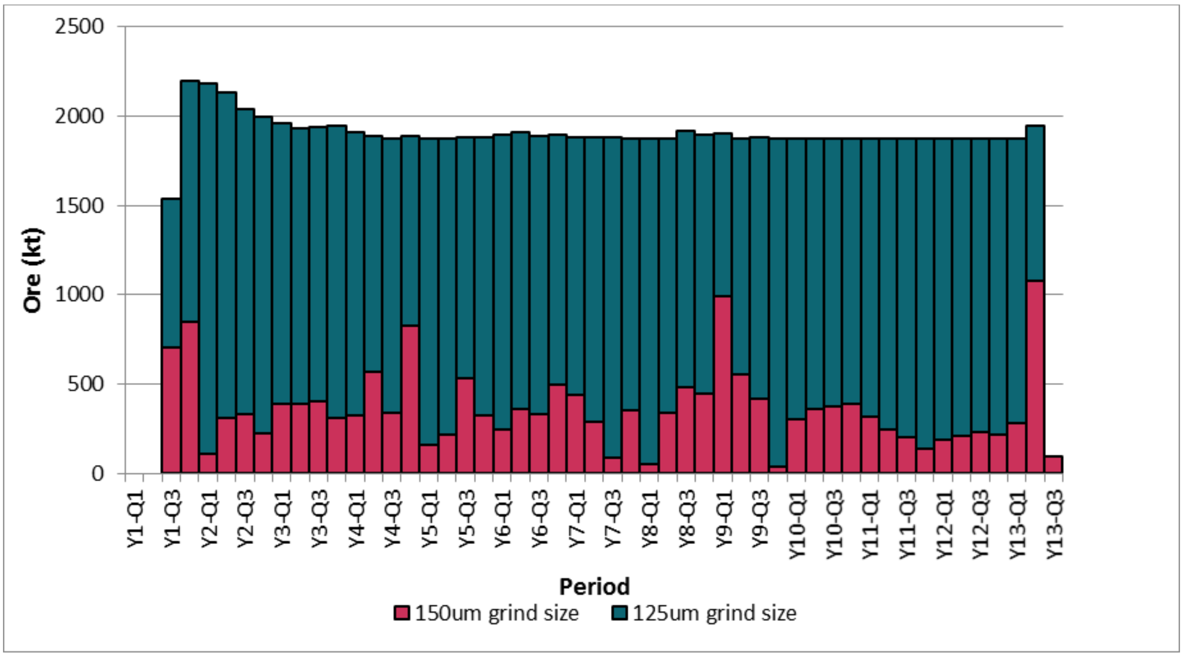

Figure 16-14*: Plant Feed Tonnes by Grind Size*

The mining schedule and processing schedule are shown on a Financial Year basis in Table 16-11 and Table 16-12 respectively.

Table 16-11: Annual Mining Schedule

		Financial Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Ore Tonnes Mined	Unit	LOM Total															
Oxide	kt	12,287	258	6,145	3,807	1,304	-	302	426	45	-	-	-	-	-	-	-
Transition	kt	3,516	-	176	2,162	627	233	23	261	34	-	-	-	-	-	-	-
Fresh	kt	75,767	-	40	3,039	5,934	7,592	7,695	6,968	7,434	6,492	7,494	7,513	7,509	7,184	874	-
Total Ore Mined	kt	91,570	258	6,361	9,008	7,865	7,825	8,020	7,655	7,513	6,492	7,494	7,513	7,509	7,184	874	-
Waste Mined	kt	253,747	5,846	19,849	22,718	21,135	16,675	17,980	36,345	36,487	37,508	25,986	8,233	3,883	1,019	84	-
Total Material	kt	345,317	6,103	26,210	31,725	29,000	24,500	26,000	44,000	44,000	44,000	33,480	15,746	11,392	8,203	958	-
Total Material	kbcm	139,824	3,698	13,202	15,134	11,570	8,812	11,715	19,081	16,034	15,413	11,818	5,694	4,205	3,084	361	-
Material blasted	kt	341,199	3,088	25,838	31,555	29,000	24,466	25,473	44,000	44,000	44,000	33,480	15,746	11,392	8,203	958	-
Material free dig	kt	4,119	3,016	371	171	-	34	527	-	-	-	-	-	-	-	-	-
Topsoil removed	kbcm	2,810	1,202	334	565	-	573	136	-	-	-	-	-	-	-	-	-
Topsoil replaced	kbcm	2,734	-	-	-	-	-	-	-	-	-	327	648	-	-	609	1,150
Production drilling	km	10,319	113	769	1,034	905	765	789	1,191	1,184	1,093	919	578	504	427	50	-
Pre-split drilling	km	860.40	-	-	2.93	35.70	99.40	57.40	68.80	143.40	176.40	122.30	63.80	45.10	38.20	7.00	-
Strip Ratio	W:O	2.80	22.70	3.10	2.50	2.70	2.10	2.20	4.70	4.90	5.80	3.50	1.10	0.50	0.10	-	-
Gold Grade Mined																	
Oxide	g/t	1.10	0.88	1.08	1.19	0.98	0.48	1.04	1.02	1.18	-	-	-	-	-	-	-
Transition	g/t	1.13	-	1.13	1.19	1.03	1.05	0.84	0.96	1.25	-	-	-	-	-	-	-
Fresh	g/t	1.21	-	1.13	1.38	1.21	1.11	1.16	1.14	1.15	1.21	1.33	1.19	1.26	1.30	1.33	-
Average Grade Mined	g/t	1.20	-	1.08	1.25	1.16	1.10	1.16	1.12	1.16	1.21	1.33	1.19	1.26	1.30	1.33	-
Contained Gold																	
Oxide	koz	434	7	214	146	41	-	10	14	2	-	-	-	-	-	-	-
Transition	koz	127	-	6	82	21	8	1	8	1	-	-	-	-	-	-	-
Fresh	koz	2,957	-	1	135	231	270	288	255	276	252	319	288	304	301	37	-
Total Contained Gold	koz	3,519	7	222	363	292	278	299	277	279	252	319	288	304	301	37	-

Note: *Apparent differences may occur due to rounding*

Table 16-12: Annual Processing Schedule

	Unit	Financial Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Ore Tonnes Processed		LOM Total														
Oxide	kt	12,287	-	5,735	3,592	1,250	66	255	406	45	437	65	-	-	-	435
Transition	kt	3,516	-	153	1,552	612	272	10	237	29	508	-	-	-	-	143
Fresh	kt	75,767	-	31	2,984	5,860	7,189	7,273	6,932	7,434	6,651	7,445	7,500	7,500	7,500	1,467
Total Tonnes Processed	kt	91,570	-	5,919	8,128	7,723	7,527	7,538	7,575	7,509	7,596	7,510	7,500	7,500	7,500	2,046
Total Ore Mined	kt	91,570	258	6,361	9,008	7,865	7,825	8,020	7,655	7,513	6,492	7,494	7,513	7,509	7,184	874
Stockpile closing balance	kt		258	699	1,579	1,721	2,020	2,501	2,581	2,586	1,481	1,465	1,479	1,488	1,172	97
Gold Grade Processed																
Oxide	g/t	1.10	-	1.13	1.23	1.01	0.73	1.12	1.03	1.18	0.71	0.49	-	-	-	0.49
Transition	g/t	1.13	-	1.20	1.36	1.08	1.09	1.14	1.00	1.32	0.72	-	-	-	-	0.51
Fresh	g/t	1.21	-	1.22	1.40	1.22	1.13	1.19	1.14	1.15	1.20	1.32	1.19	1.26	1.28	1.02
Average	g/t	1.20	-	1.13	1.32	1.17	1.13	1.19	1.13	1.16	1.14	1.32	1.19	1.26	1.28	0.87
Gold Processed in feed																
Oxide	koz	434	-	207	142	40	2	9	14	2	10	1	-	-	-	7
Transition	koz	127	-	6	68	21	9	-	8	1	12	-	-	-	-	2
Fresh	koz	2,957	-	1	134	229	262	279	254	276	256	317	287	304	309	48
Total	koz	3,519	-	215	344	291	273	289	275	279	278	318	287	304	309	57
Process Recovery																
Oxide	%	93.80	-	93.80	93.90	93.80	93.00	94.00	93.90	93.90	93.00	93.00	-	-	-	93.00
Transition	%	91.70	-	91.70	91.90	91.80	91.50	91.90	91.50	91.80	91.00	-	-	-	-	91.00
Fresh	%	90.90	-	90.90	91.30	90.90	90.60	90.80	90.60	90.70	90.80	91.20	90.80	91.00	91.10	90.30
Average Recovery	%	91.30	-	93.80	92.50	91.30	90.60	90.90	90.80	90.80	90.90	91.20	90.80	91.00	91.10	90.60
Gold Recovered																
Oxide	koz	407	-	195	134	38	1	9	13	2	9	1	-	-	-	6
Transition	koz	117	-	5	62	19	9	-	7	1	11	-	-	-	-	2
Fresh	koz	2,687	-	1	122	208	237	253	230	250	233	289	261	277	282	43
Total Gold Recovered	koz	3,212	-	201	318	266	247	262	250	253	253	290	261	277	282	52

Note: *Apparent differences may occur due to rounding*

Haulage Profiles

Haulage profiles were developed utilising Alastri Software's Haul Infinity program which analyses site topographic data and proposed locations for ore and waste material destination. Travel times and fuel burn rates were estimated from each mining bench's haul profile and applied as an input into the Mining Cost Model.

Stockpiling Strategy

A variable plant feed cut-off grade was applied to maximise the feed grade in the initial years of operation within the constraints of maximum plant throughput. Table 16-13 presents the grade ranges that were applied in the stockpiling strategy.

Marginal grade material (below economic cut-off but greater than 0.3 g/t) will be stockpiled, as "mineralised waste", adjacent to the ROM pad and may be processed at a later date depending on future gold price and processing economics. No additional cost is attributed to stockpiling this material which may or may not be processed. Peak stockpile size of marginal grade material will be approximately 0.9 Mt. Figure 16-15 shows stockpile movements over the LOM.

Table 16-13: Stockpile Grade Bins

Stockpile	Symbol	Minimum Grade	Maximum Grade	Material Type
Low-grade	LGO	Fresh (0.43g/t) Transition (0.38g/t) Oxide (0.35g/t)	0.6	Fresh, Transition, Oxide
Medium-grade	MGO	0.6	0.8	Fresh, Transition, Oxide
High-grade 1	HGO1	0.8	1.1	Fresh, Transition, Oxide
High-grade 2	HGO2	1.1	1.3	Fresh, Transition, Oxide
High-grade 3	HGO3	1.3	999.0	Fresh, Transition, Oxide

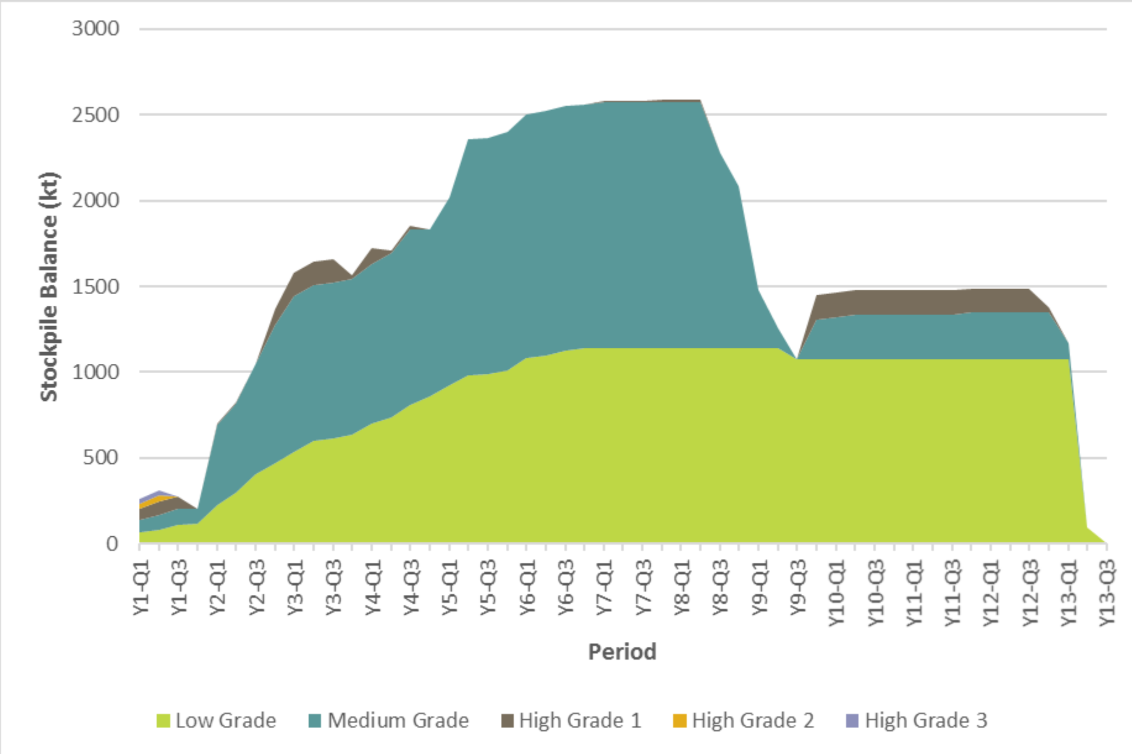

Figure 16-15: *Stockpile Balances Over LOM*

16.4 Mining Infrastructure

The design and operating strategy for mining infrastructure focussed on optimising development capital and operating costs whilst minimising environmental impacts. This is achieved by minimising haul lengths to the ROM and dumps where possible. Infrastructure is limited to areas outside a stand-off distance of 140 metres from the pit crest. The area outside the stand-off distance is used for the location of pit bunds, surface haul roads and topsoil stockpiles. Sterilisation drilling, waste rock characterisation and waste material movement optimisation studies have been conducted as part of the FS. A waste dump optimisation study has been carried out to provide guidance to the overall materials handling strategy. Figure 16-16 shows the proposed site layout incorporating the following key features:

- The TSF location was established during the PFS and optimised the location relevant to the plant and mine while avoiding the larger sand dunes to the west of the open pit.
- Waste Dumps are located adjacent to the open pit to minimise haul lengths
- The ROM pad is designed to maximise its ore storage capacity fitting in to an allocated area between the pit and plant
- The ROM pad access roads and layout have been designed to accommodate potential transport of ore from other regional satellite ore bodies for treatment in the process plant
- West waste dumps are limited by the location of the main diversion channel running to the west of the Project
- Dump development to the north of the pit is constrained by a culturally sensitive area
- Mining workshops are located to ensure easy access to the main site access road and the open pit
- Explosives facilities are located at a required separation distance from other occupied buildings while maintaining good access to the main site access road and the open pit.

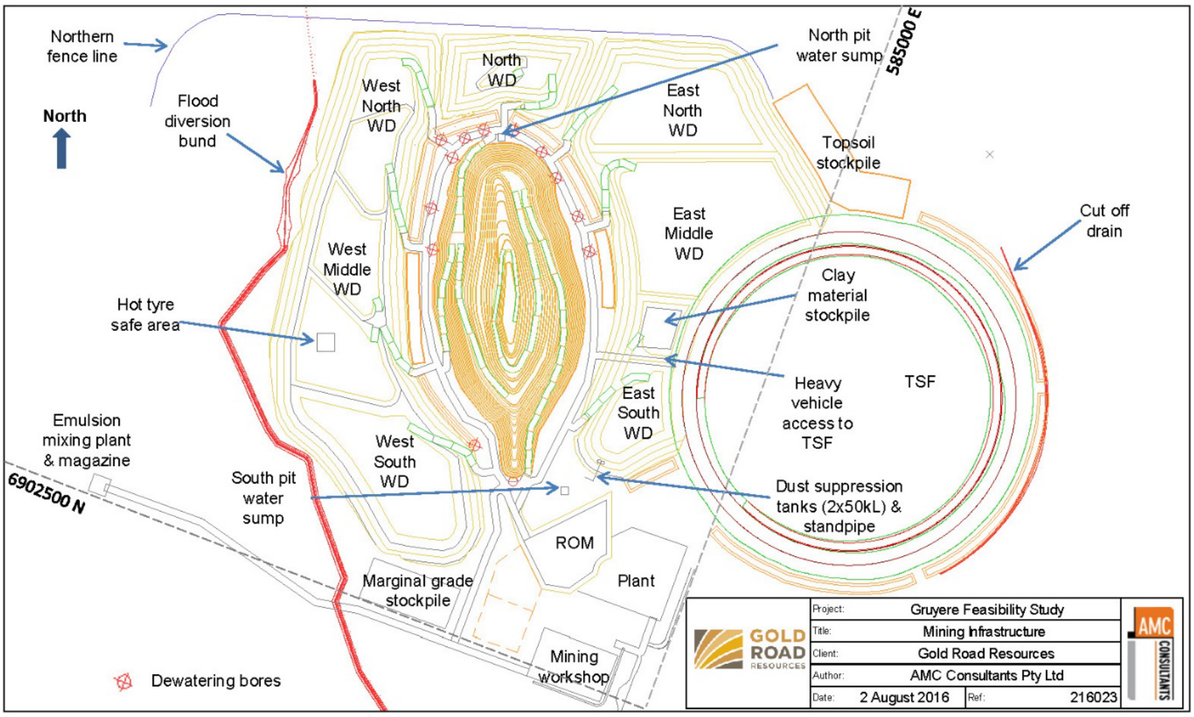

Figure 16-16: *Mine Site Layout*

Topsoil Management

Topsoil is classed into two categories, a higher value gravelly loam (nominal thickness to be recovered of 0.25 metres) and a lower value but more common swale sand (recovery depth varies from 0.2 metres from the waste dump footprints to 1 metre from the mine footprint). Topsoil will be removed from areas requiring disturbance and, when possible, immediately used for rehabilitation of final landforms. Where immediate use is not possible the soils will be stored separately in dedicated stockpiles of maximum 3 metre height for gravelly loams and 10 metre height for swale sands for future rehabilitation requirements. Final stockpile locations will be determined during operations based on quantities recovered and the ultimate site layout.

Waste Rock Dump

Waste rock from the pit will be disposed of at designated waste rock dumps located adjacent to the pit, and will also be utilised to construct the ROM pad and external walls of the TSF. Waste dumps will be sited as close to the pit as safely possible taking into account necessary width for infrastructure around the open pit and DMP guidelines on the construction of safety bund walls around open pit mines. Approximately 103 Mbcm of waste rock will be disposed of at waste rock dumps. The dumps will be monitored on an ongoing basis in accordance with relevant environmental requirements.

Tailings Storage Facility

Waste material from the pit will also be utilised to develop walls for a TSF which is integrated with the adjoining waste rock dump east of the open pit. Mine waste rock will be initially utilised to construct starter embankment walls and the base of the TSF. Subsequent TSF lifts will utilise waste rock from the mining process for perimeter embankment walls. Approximately 1.3 Mlcm of selected clay/ Saprolite waste rock material will be utilised for the perimeter embankment with an additional 14.6 Mlcm to construct the final waste dump embankment around the TSF. Additional costs for hauling and placing (including additional dozer hours) of bulk material to the TSF instead of the nearest waste dump are estimated and reported separately in the Mining Cost Model.

Run of Mine Pad

The ROM pad will be constructed using waste rock sourced from the open pit. The pad will be built up in stages over the Project development period to support the crusher chamber construction. The start-up phase will fully enclose the primary crushing unit and allow enough working area. The ore handling capacity of the pad will support the maximum stockpile size estimated during mine scheduling of 1.5 Mt of high, medium and low-grade ore. Selected low-grade and marginal material may be tipped off the south-west side of the ROM for additional storage. The ROM also has facility for acceptance of ore from other mines via road train haulage.

The final ROM pad will have a footprint of 44 ha and will be constructed to a height of approximately 21.4 metres with a skyway arrangement to allow for building higher stockpiles as well as minimising vehicle interaction. Approximately 3 Mlcm of material will be required to construct the pad.

16.5 Mine Operations and Management

Mining activities will be conducted by a mining contractor with technical and managerial direction provided by Gold Road. The proposed mine operations model has the following advantages:

- Minimisation of upfront Capex requirements by Gold Road
- Application of contractors' specialised open pit mining knowledge, systems and experience to lower operational risk.

The general mining method is summarised as follows:

- Clearing and stripping of suitable material from all disturbed areas into discrete stockpiles
- Drilling and blasting of ore and associated internal waste on 5 metre benches, while bulk waste which is outside the ore envelope is blasted on 10 metre benches. Trim blasts and pre-splits will be used to provide wall control in fresh rock as required. The majority (70%) of the explosives usage is bulk emulsion and the remainder is ANFO, with all explosives supply provided by a subcontractor
- Loading and hauling utilising 360 t excavators and 180 tonne capacity haul trucks mining on 3 metre high flitches in ore zones and three to 4 metre high flitches in bulk waste zones. Ore material is planned to be marked out by paint or tapes on the ground, supported by dedicated ore spotters as required. Ore will be direct fed to the crusher or placed on stockpiles for future rehandle as required
- Waste dumps will be developed in 10 metre lifts and progressively rehabilitated. Raising of the TSF embankment will be constructed with waste material from the mine as required
- Ancillary plant support for floor control, haul road construction and maintenance, rehabilitation, drill support, waste dump battering and the like provided by a fleet of dozers, graders and water carts
- Pit dewatering is expected to be minimal and will be managed by collection of water by in-pit sumps for use within the mining operation
- Crusher feed is provided by a combination of direct tip from the mine (50% of crusher feed) and rehandle from ROM or long term stockpiles using either a front end loader (FEL) only or a FEL and 135 t capacity haul trucks (dependent on haul distance)
- Grade control will be provided by a subcontractor on a 25 metre x 25 metre pattern of RC drill holes, and is campaigned during the mine life.

The following sections provide more details on the various mining activities.

Drilling and Blasting

Drill and blast technical parameters were developed following a review by an independent specialist drill and blast consultant, and based on analysis of material properties and drill hole logging developed during the FS. It was recommended that ore should be blasted on five metre benches using 102 mm and 127 mm diameter holes to optimise fragmentation. Waste is generally drilled and blasted on a 10 metre bench height. All material with an in situ dry density greater than 2.0 t/m^3 was classed as hard, otherwise it was categorised as soft. The transported cover material (27 Mt) was assumed to not require any drill and blast. AMC developed drill and blasting costs and equipment fleet requirements based on the parameters provided by the specialist consultant.

Pre-split blasting was assigned to fresh rock walls in Stage 3 and 4. The estimation of pre-split to Stage 3 is a conservative decision based on limited geotechnical input and may overstate the pre-split drill metres required. Information gathered during the mining of Stages 1, 2 and 3 will be used to refine the final design for Stage 4. Pre-split drilling is designed on a spacing of 1.3 metres, drilled on 10 metre benches with 0.7 metres sub-drill using 102 mm diameter holes.

Table 16-14 presents drill and blast design parameters that were applied for hard and soft material classes. All fresh rock walls will be trim blasted adjacent to the wall but are not specifically allocated a separate drill and blast pattern as there is no difference in costs between trim and production blasts. Allowance for trim blasts that may be required on transitional walls (where there was no pre-split blasting) and a 5% redrill allowance for all drilling types were made.

Table 16-14: Drill and Blast Design Parameters

Parameter	Units	Hard Ore	Soft Ore	Hard Waste	Soft Waste	Trim Blast
Dry Density	t/bcm	2.60	1.80	2.60	1.80	2.60
Moisture Content	%	2.50	10.00	2.50	10.00	2.50
Swell Factor	%	30.00	30.00	30.00	30.00	30.00
Wet Bank Density	t/m3	2.67	1.98	2.67	1.98	2.67
Hole Diameter	mm	102.00	127.00	165.00	200.00	102.00
Bench Height	m	5.00	5.00	10.00	10.00	5.00
Instantaneous Penetration Rate	m/hr	27.00	43.00	28.00	43.00	27.00
Burden	m	2.70	3.40	4.20	5.40	4.00
Spacing	m	3.10	3.90	4.80	6.20	3.10
Sub-drill	m	0.80	1.00	1.30	1.60	0.80
Hole Length	m	5.80	6.00	11.30	11.60	5.80
Drill rig		T45	T45	DML	DML	T45
Explosive Type		Emulsion	ANFO	Emulsion	ANFO	Emulsion
Powder Factor	Kg/bcm	0.91	0.52	1.03	0.57	0.62
Volume blasted	kbcm	38,749.00	734.00	63,838.00	31,367.00	2,326.00

Loading and Hauling

The primary loading fleet will consist of a maximum of three hydraulic excavators in the 360 tonne class. This class of machine is considered the largest option that could also practically excavate bulk waste and more selective ore zones on 3.5 metres to 4 metres flitches. Smaller machines would result in a higher operating cost and introduce scheduling issues by the need to create additional dig locations. The excavator model selected for the purpose of the FS was the Hitachi EX3600 unit.

The ore and waste haulage fleet will consist of 180 tonne mechanical drive haul trucks. The truck model selected for the purpose of the FS was the Cat 789 haul truck which is the largest unit that could direct tip to the primary crusher and is well matched to the proposed excavator. Cat 785 trucks are selected to operate with the 992 FEL loading topsoil and on ROM rehandle. This fleet could supplement mining activities although not scheduled to do so.

Table 16-15 shows operating assumptions for the load and haul mining fleet.

Table 16-15: Loading and hauling Equipment Operating Parameters

Parameter	Units	Soft Material	Hard Material
Material Details			
Dry Density	t/bcm	1.80	2.60
Moisture Content	%	10.00	2.50
Swell Factor	%	30.00	30.00
Wet Loose Density	t/m³	1.52	2.05
Wet Bank Density	t/m³	1.98	2.67
Excavator Details (Hitachi EX3600)			
Bucket Heaped Capacity	m³	22.00	22.00
Fill Factor	%	98.00	85.00
Bucket Capacity volume	bcm	16.60	14.40
Truck Details (Cat 789)			
Tray Capacity	m³	108.00	108.00
Fill Factor	%	95.00	95.00
Volume Limit	bcm	78.90	78.90
Rated Payload	t	180.00	180.00
Assumed Overload/Underload	%	0.00	0.00
Adjusted Payload	t	180.00	180.00
Weight Limit	bcm	90.90	67.50
Adopted Capacity	bcm	78.90	67.50
Minimum Bucket Fill	%	25.00	25.00
Calculated Passes Per Load	No	4.76	4.70
Passes Per Load (Rounded)	No	5.00	5.00
Actual Truck Load	bcm	78.90	67.50
Actual Truck Load	t	156.00	180.00
Dump Time	min	1.20	1.20
Excavator Productivity (Maximum scheduled)			
Cycle Time	sec	28.00	30.00
Loading Cycle Time Penalty	sec	0.00	0.00
Adjusted Cycle Time	sec	28.00	30.00
Efficiency Factor	%	83.00	83.00
First Pass	sec	10.00	10.00
Truck Exchange	sec	35.00	35.00
Loading Time	min	2.62	2.75
Maximum Productivity	bcm/op.hr	1,502.00	1,223.00
Utilisation	%	92.80	92.80
Productivity	bcm/op.hr	1,394.00	1,135.00
Productivity	t/op.hr	2,509.00	2,951.00
Annual Operating Time	hours	6,342.00	6,342.00
Productivity	Mbcmpa	8.84	7.20
Productivity	Mtpa	15.91	18.72

The following loaded travel speed limits were used to derive travel times and fuel use calculation for haul trucks:

- Pit bench 20 kph maximum speed, 3% rolling resistance
- Ramps 40 kph maximum speed, 2% rolling resistance, at a 10% gradient
- Out of pit flats 60 kph maximum speed, 2% rolling resistance
- Waste dump bench 40 kph maximum speed, 3% rolling resistance
- TSF bench 20 kph maximum speed, 3% rolling resistance
- ROM ramp 20 kph maximum speed, 2% rolling resistance, at a 10% gradient.

Downhill speeds were derived from equipment retard curves.

Auxiliary Mining Equipment

Haul road dust suppression will be carried out using water recovered from pit dewatering and will be supplemented by raw water from the borefields. Water requirements for dust suppression for the mining operation have been estimated to be approximately 0.6 gigalitre (**GL**) per annum, based on pit and dump road length and estimated water demand to maintain road moisture. A fleet of two Cat 777 water trucks or equivalent operating 3,000 hours per year will be utilised for dust suppression.

Track dozers are assigned at a rate of one base unit plus 0.4 units per excavator or active mining area and an additional wheel dozer. Total dozer fleet supports multiple mining areas, development of the TSF and a relatively large drill fleet.

Graders are assigned to the model at one base unit plus one for every 20 trucks operating.

Table 16-16 shows the proposed mining equipment that will complement the primary mining fleet.

Table 16-16: Mining Equipment

Equipment	Manufacturer	Model	Annual Hours	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Excavator	Hitachi	EX3600	6,342	1	2	2	2	1	2	3	3	3	2	1	1	1	-
Truck	Cat	789	6,079	3	11	11	11	10	13	17	17	17	14	8	6	5	-
Truck	Cat	785	5,992	2	1	1	1	1	1	1	1	1	1	1	1	1	1
Track Dozer	Cat	D10T	4,000	1	3	3	3	2	3	3	3	3	3	2	2	2	1
Water Truck	Cat	777 WT	3,000	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Grader	Cat	16M	4,000	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Front end loader	Cat	992K REH	6,371	1	2	1	1	2	1	1	1	1	2	1	1	1	1
Support excavator	Hitachi	EX1200	1,500	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Wheel Dozer	Cat	834H	4,000	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Tyre Handler	Cat	980 Tyre Handler	750	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Rock-Breaker	Cat	336DL RB	750	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Dewatering Pump	Sykes	HH160i	1,200	-	1	1	2	2	2	3	3	3	3	4	4	4	-
Pump	Sykes	Sump Pump	1,000	-	1	2	3	3	3	3	3	3	3	3	3	3	-
Lighting Plant	Allight	Light Plant	3,000	8	14	13	13	12	14	15	14	14	13	11	11	7	-
Fuel Truck	Highway	Hway FT	3,000	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Small Service Truck	Highway	Hway ST	2,000	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Troop Carrier	Toyota	LV-TC	1,200	2	3	3	3	3	3	3	3	3	3	3	3	3	-
Pit Bus	Toyota	Coaster	1,000	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Crane	S. Cranes	Crane	500	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Forklift	Generic	Large	750	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Forklift	Generic	Small	750	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Light Vehicle Contractor	Toyota	Land Cruiser	1,500	12	15	15	15	15	15	15	15	15	15	12	12	8	-
Light Vehicle Owner	Toyota	Land Cruiser	1,250	6	11	11	11	11	11	11	11	11	11	7	7	5	-
Drills	Atlas Copco	DML	5,689	1	3	3	4	3	4	7	7	7	4	2	1	-	-
Drills	Atlas Copco	ROC T45	5,689	1	5	7	6	6	6	6	7	7	7	5	5	4	-

Mine Equipment Support Infrastructure

The following support infrastructure will be constructed by the mining contractor:

- Three-bay heavy vehicle maintenance workshop, with separate tyre change and boilermaker bays, with overhead gantry crane and store
- Light vehicle maintenance facility
- Fuel storage and dispensing facility (approximately 600 kL capacity)
- Oil and lubricant storage and dispensing facility
- Heavy and light vehicle washdown facility
- Associated offices, crib rooms, training facilities, communications etc.
- Pit dewatering and associated distribution infrastructure (pipelines, turkey nests etc.). Dewatering bores will be developed by others during the construction period.

Technical Services

A site based Gold Road technical services team will provide short-term and long-term technical direction and support to the operations in the following disciplines:

- Mine management
- Survey
- Geological control and grade control
- Mine planning
- Geotechnical monitoring
- Information management.

Manpower requirements to sustain the mining operation are estimated by AMC in the Mining Cost Model. The estimated manning level for the Gold Road mining department were 31 during the main mining period and for the contractor ramping up from an initial 108 to around 200 with peak contractor numbers of 223.

Pit Dewatering

Pit dewatering requirements were analysed by external consultant Pennington Scott as part of the FS. Areas of potential pit influx include:

External Flood Water

Two significant catchment areas have been modelled on the south-western end of the pit and process plant. In the design of the pit water management system, provision has been made to divert flood water generated in the upstream catchment away from the Project in general, including the open pit.

Internal Storm Water Run-off

The designed final pit outline has a catchment area of approximately 118 ha. Storm water generated within this catchment will drain down the haul ramp and pit slopes and accumulate on the lowest bench of the pit. An in-pit sump will be installed on the lowest bench to capture all internal run-off. Water in the sump will be pumped out to the mine raw water storage dam for use in dust suppression. Any excess water will be pumped to the process plant. An allowance for pump capacity to manage a one in two year storm event, (estimated at 36 ML) is included in the Mining Cost Model.

Groundwater

Ground water will initially be extracted by four out-of-pit bores targeting high permeability zones in the weathered zone. An additional nine out-of-pit depressurisation holes will be drilled approximately 12 months later targeting areas where the Saprolite is thickest to lower pore pressures. Out-of-pit dewatering bores will be drilled and equipped at designated locations around the perimeter of the pit outline. Dewatering will commence during the Project construction phase.

Horizontal seep wells will be drilled into the pit walls through the transition zone to depressurize and assist with drainage of the overlying lower Saprolite zone at approximately 9,340 mRL. The costs for these drainholes are included in the Miscellaneous Overheads in the Mining Cost Model.

Discharge from the out-of-pit boreholes and horizontal seep wells will be pumped out to the main raw water storage dam and will be utilised for dust suppression. The rate of water made from mine dewatering will be governed by the rate of mining and is expected to average in the order of 18 to 35 L/s (1,555 to 3,024 kL per day) over the first several years of mining, declining to less than 12 L/s (1,037 kL per day) by the end of mining.

Grade Control Drilling

Grade control drilling requirements for ore definition is planned to be performed by RC drilling and sampling on a predominantly 25 metre x 25 metre pattern, 60° inclination from the horizontal toward 270° Gruyere Grid and 1 metre downhole sample. Sample assay will be by Fire Assay and Atomic Absorption Spectroscopy or Inductively Coupled Plasma Mass Spectrometry finish using an offsite laboratory. Grade estimation is planned to incorporate geological interpretation and geostatistical analysis, consistent with current mineral resource estimation methodology.

The drilling will be campaigned throughout the mine life, with a nominal 50 metre vertical depth per grade control hole assumed in the Mining Cost Model. A cost of A$70/m drilled (including assay) has been assumed in the Mining Cost Model.

Third Party Review of AMC Study

Orelogy was commissioned to undertake an independent third party review of the AMC mining study focussing on the ability to implement designs, achieve production schedules and costs. Excluded from their assessment was the independent generation of an ore reserve model, performing optimisation runs, review of geotechnical parameters and drill and blast studies.

The review raised one serious, five moderate and 14 minor flags with the serious and moderate flags noted below. Orelogy believe:

- the methodology used to calculate truck productivities underestimates the truck hours required to meet the schedule
- execution risk may exist in the interface between 5 metre bench height blasts in ore and 10 metres in waste
- wait times for trucks in the truck cycle may be under-called when an excavator is operating in an under-trucked mode
- The assumptions for proportion of ore rehandled from ROM stockpiles to the crusher are too low
- Haulage analysis (truck haul routes and cycle times) was done on an annual basis and the validity of this approach was not verified when applied to a quarterly schedule
- Fuel burn for trucks may be under-estimated due to cycle time, truck productivity and engine fuel burn rate assumptions.

AMC addressed the items raised by Orelogy and disagreed with the assessment risks associated with truck productivities, bench heights and truck haulage analysis. They modelled the changes to ROM feed, fuel burn and wait times for trucks when the excavators are under-trucked and concluded the impact was well within the accuracy of the FS. No changes were made to the FS financial model based on Orelogy's review.

17 RECOVERY METHODS

17.1 Design Concepts

From the mineral processing and metallurgical testing set out in Section 13 various design concepts were developed.

Comminution

The ore can be characterised as varying from soft (Saprolite ore) to hard (Fresh ore). As the majority of the Ore Reserve is fresh ore and direct tipping into the primary crusher is proposed, the opportunity to blend ores will be limited. The process plant design has therefore been based on the 85th percentile for hardness in the fresh ore samples.

Given the high UCS values measured for the fresh ore (above 200 MPa) and the porphyry lithology, a crushing work index of 21 kWh/t has been used in the design as this is typical of an ore of this nature.

Based on the options reviewed in the PFS, a single stage crushing circuit followed by a SABC circuit operating at a rate of 7.5 Mtpa with a target grind size of P_{80} of 125 μm on fresh ore was selected for the FS. This circuit would be able to process 8.8 Mtpa on Oxide ore types (included both Saprolite and Saprock) and 8.0 Mtpa on transitional ore types to a product size P_{80} of 125 μm.

To mitigate risk related to variability in ore size and hardness in a SABC circuit when employing large diameter grinding mills, the selection of both mills was conservative. As the circuit design is based on hard fresh ore the installed motor power is significantly greater than that required in soft and medium hardness ores. Given the variability in hardness inherent in the orebody it was determined that both the SAG and ball mills will have variable speed drives to compensate for change in hardness. This will permit power consumption to be optimised to ore type, assist in protecting the SAG mill liners from damage as a result of grinding media impact and minimise the mill motor current requirements on mill start up.

Gravity Gold

Throughout the metallurgical test work program, standard duplicate, triplicate and quadruplicate head assays for gold showed poor reproducibility. Screen fire assays proved more consistent with the back calculated assays from the leaching test work. This is indicative of coarse free gold (i.e. greater than 75 µm). Standard gravity-leach test work demonstrated that between 20% and 80% of the gold could be recovered by gravity in the laboratory. The laboratory gravity recovery is likely to overstate the result that will be achieved in the plant practice due to the much higher mass recoveries in the laboratory (typically 6% to 7% mass recovery in the laboratory against a mass recovery of 0.03% in the plant design).

To better quantify the plant gravity recovery, Master Composites representing the three major fresh ore domains (South, Central, and North) were subjected to 3-stage GRG test work. A gravity recovery of 35% has been nominated based on vendor modelling of this data but the electrowinning circuit has been designed to cope with a gravity recovery of up to 50%. The plant design incorporates a gravity circuit to process approximately 40% of the cyclone underflow at a 300% circulating load. At lower circulating loads a greater proportion of the cyclone underflow can be treated. The split to the gravity circuit will be controlled by adjusting the number of cyclones directed to the gravity circuit and the number of cyclones directed back to the ball mill.

Leaching

Overall the test work showed rapid leaching kinetics and high ultimate leaching extractions with low cyanide consumptions for all ore domains. Leaching was substantially complete within four hours and complete for all practical purposes within 24 hours. Oxygen markedly increased the leaching kinetics in the first four hours but did not appear to increase the ultimate extraction at 24 hours. There was no evidence of preg-robbing (ore that has a natural affinity to adsorb gold from solution) species in the ores tested.

Leaching test work was conducted using Project site (Yeo Borefield) water. The water is saline, with a TDS content of approximately 20,000 ppm, and therefore it was expected that reagent consumptions would be slightly elevated when compared to using fresh water. However, the measured cyanide and lime consumptions were modest and water quality is not considered to be of any significant concern. The proposed bore water was buffered at about pH 10.2 on lime. This would indicate the target pH for the leaching circuit will be 9.5 to 10 to minimise both lime consumption and HCN gas evolution.

The ore did not show any preg-robbing characteristics and a CIP circuit will be suitable. A CIL circuit was selected because it would be less susceptible to any preg-robbing species that may be treated in the future. However, a pure CIL circuit will limit the gold loading level on the carbon and this would increase the size of acid washing, elution and regeneration circuit significantly. As a compromise a hybrid CIL circuit was selected.

The design will include a single leaching stage with a nominal pulp residence time of four hours and six stages of adsorption with a nominal pulp residence time of 20 hours on fresh ore. Therefore, the plant design provides an overall nominal pulp residence time of 24 hours for fresh ore. The four hours of leaching time was selected because the leaching was substantially completed (typically greater than 80%) at this point and the total nominal pulp residence time of 24 hours was selected based on the leaching being practically complete at this point.

The inclusion of six stages of adsorption was based on modelling of the circuit using carbon activities typical of an operation with a clean ore and good plant carbon acid washing and regeneration practices. Such a circuit should achieve a solution loss of less than 0.015 ppm and a loaded carbon gold assay of 1,060 g/t on a head grade of 1.2 g/t and 35% gravity recovery on fresh ore. If preg-robbing ores are treated in future, carbon can be circulated to the leaching stage and allowed to return to the first adsorption tank to offset the impact of such ores on gold recovery. Carbon adsorption performance confirmed by test work showed no indication of anything in the ore that would cause complications in this regard.

To minimise the volume of the leaching and adsorption stage, a pulp density of 50% solids (w/w) was selected. This will require pre-leach thickening. Pre-leach thickening will have the additional benefit of improving classification efficiency in the grinding circuit as lower pulp densities in the cyclone feed are more feasible than would otherwise be the case. In addition, thickening of the CIL tailings to 60% solids (w/w) has been incorporated into the design to maximise the recovery of reagents and soluble gold in the tailings. This will have the additional benefit of maximising water recovery (thereby minimising the raw water demand of the plant) and maximising the ultimate settled density in the tailings. This has the compound effect of minimising the TSF embankment raising requirements and minimising seepage from the TSF. The specific area capacity for thickening used in the design of 1.0 t/^{m2}/h was selected on the basis of dynamic thickening test work conducted on representative samples of Oxide, Transitional and Fresh ore. Leaching, thickening, and rheological test work on representative samples of Oxide, Transitional and Fresh ore has confirmed that the CIL and tailings disposal circuits will be able to operate at these elevated densities.

Elution and Gold Recovery

A split AARL elution circuit with separate acid washing and elution columns was selected for carbon elution. The AARL elution circuit with dual columns was chosen for its flexibility. A split circuit was selected to minimise the fresh water requirements. Full chemical analysis on the loaded carbon from test work indicated that there are no major impurities or deleterious elements that could complicate this process. Likewise, given the clean nature of the ore it is not expected that thermal regeneration of the carbon will present any difficulties.

17.2 Process Plant Description

Introduction

The Project processing facility will be designed to process 7.5 Mtpa of Fresh ore but the design will enable the latent capacity of the major comminution equipment (based on the installed power) to be utilised when treating ore types with lower work index than fresh ore. The design will enable the processing of:

- Oxide ore (Saprock and Saprolite) at a rate of 8.8 Mtpa
- Transition ore at a rate of 8.0 Mtpa
- Fresh ore at a rate of 7.5 Mtpa.
- Various tonnages of ore blends.

The process plant will be designed to operate seven days per week at a nominal treatment rate of 1,100 dtph on Oxide ore, 1,000 dtph on Transitional ore and 937 dtph on Fresh ore at a grinding circuit utilisation rate of 91.3%. Availability is defined as the percentage of total time that the process plant is mechanically and electrically ready to operate while utilisation is defined as the percentage of the total time that the process plant is actually operated.

The process plant unit processes are based on proven technology for gold recovery following a processing route of:

- Primary crushing by a gyratory crusher to product size P_{80} of 135 mm
- Grinding in a SABC circuit to a product size P_{80} of 125 µm
- Treatment of a portion of the grinding circuit cyclone underflow by centrifugal gravity concentration, followed by batch intensive leaching of the gravity concentrate and electrowinning of the resulting pregnant solution
- Thickening in a Hi-rate thickener of the grinding circuit cyclone overflow to 50% solids (w/w) prior to treatment in a hybrid CIL circuit
- Acid washing and split AARL elution of the resulting loaded carbon and thermal regeneration of the barren carbon prior to its return to the CIL circuit
- Smelting of cathode sludge from electrowinning to produce a final product of gold doré
- Tailings thickening in a Hi-rate thickener to 60% solids (w/w) prior to disposal of the tailings into the TSF located within an integrated waste landform (**IWL**).

The process plant layout will reflect the sequential nature of the processing operations from ROM ore feed to the facility and tailings disposal of the waste product. A general schematic layout of the process plant is shown in Figure 17-1.

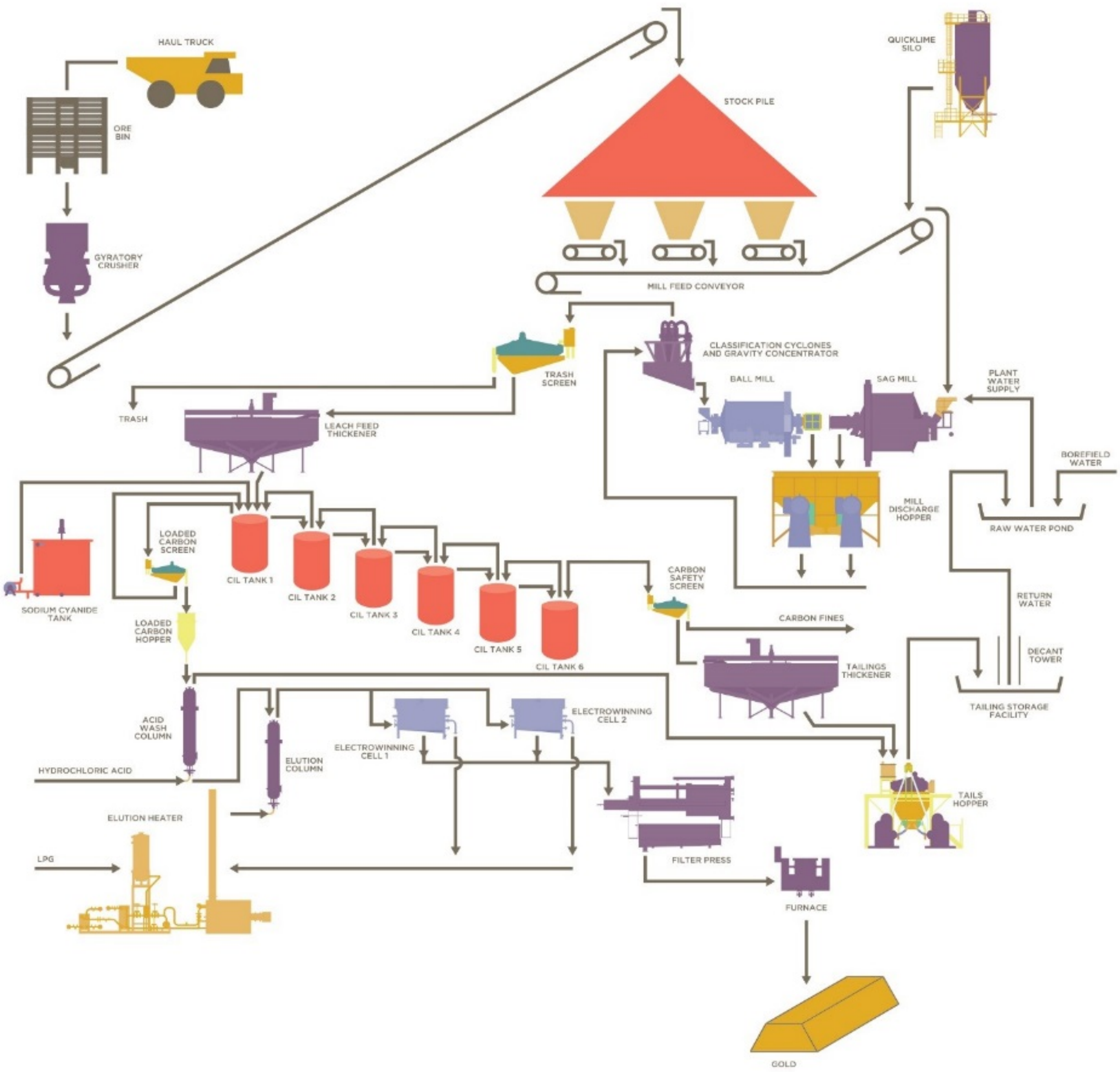

Figure 17-1: *Process Plant Flowsheet*
(Note: for simplicity of the diagram, the gravity circuit and pebble crusher circuits are not included)

Primary Crushing

The crushing circuit will be a single stage, open circuit gyratory crusher. Product from the crushing circuit will be conveyed to the coarse ore stockpile. The circuit will crush 1,300 dtph to a product size P_{80} of 135 mm. The crushing plant will operate with utilisation range of 66% to 78%, depending on ore type being crushed.

ROM ore will be trucked from the mine to a ROM pad and will either be tipped directly into the primary crusher dump pocket or stockpiled on the ROM pad for reclaim at a later stage by FEL. Any oversize material fed into the dump pocket bridging the opening to the gyratory crusher will be fragmented by a fixed rock breaker to permit it to pass into the primary crusher. The FEL will be supplied and operated by the mining contractor.

The primary crusher will be a 1.370 metres by 1.905 metres (54 to 75 inch) gyratory crusher with a 650 kW motor. It will be operated with an open side setting of 160 mm and a 37 mm throw. The primary crusher will discharge onto a 1.2 metre wide by 6.5 metres long apron feeder (55 kW drive) that will, in turn, discharge onto the crusher discharge conveyor. The crusher discharge conveyor (71 metres in length, 1,500 mm wide, 200 kW drive) will feed onto the stockpile feed conveyor (156 metres in length, 1,500 mm wide, 300 kW drive). A self-cleaning magnet located at the crusher discharge conveyor head chute will remove magnetic tramp metal from the ore stream and discharge it into a tramp metal bin. The stockpile feed conveyor will discharge onto the coarse ore stockpile.

To minimise dust emissions, the primary crusher discharge chamber will be serviced by a dust extraction system comprising a filter bag unit with reverse air pulse cleaning. The dust collected by the system will be discharged onto the crusher product conveyor.

Coarse Ore Storage and Handling

Crushed ore will be reclaimed from the crushed ore stockpile via three apron feeders under the stockpile, discharging ore onto the mill feed conveyor which runs within the tunnel beneath the stockpile. The total capacity of the coarse ore stockpile is approximately 70,000 t and a live storage capacity of approximately 16,000 t, equivalent to 14.5 to 17.1 hours of milling time, depending on ore type. The mill feed conveyor will feed the grinding circuit.

The reclaim apron feeders (55 kW drive) will each be 1.2 metres wide by 6.5 metres long, variable speed and any single feeder will be able to supply the entire mill feed rate. These Apron Feeders are identical to the Primary Crusher Discharge Apron Feeder. The feed control philosophy envisions that the feeders will be operated together to minimise variation in the mill feed particle size distribution by controlling the drawdown of ore from the stockpile.

A 600 t quicklime silo, fitted with a variable speed rotary valve and a fixed speed drive weighing screw feeder (200 mm diameter by four metre long with maximum feed rate of 2,800 kg/hr), will dose lime onto the mill feed conveyor to provide protective alkalinity in the leaching and adsorption circuit. Lime will be delivered to site using triple or quad road trains and the lime will be pneumatically transferred into the silo from the tanker trailers.

The reclaim area and the quicklime silo area will be serviced by dedicated vertical spindle centrifugal sump pumps for clean-up, with floor slopes appropriately graded to the relevant sumps to facilitate ease of cleaning.

Grinding and Classification

The mill feed conveyor (178 metres in length, 1,500 mm wide, 220 kW drive) will feed the two stage grinding circuit. The first stage will be a grate discharge SAG mill in open circuit with pebble crushing and the second stage will be an overflow discharge ball mill in closed circuit. The circuit will grind 1,100 dtph of Oxide ore, 1,000 dtph of transitional ore and 937 dtph of Fresh ore to a product size P_{80} of 125 µm. The comminution circuit will operate with utilisation of 91.3%.

The SAG mill will have an inside shell diameter of 10.97 metres and effective grinding length (**EGL**) of 5.79 metres. The mill will have a grate discharge configuration and dual pinion variable speed drive with 7,700 kW low speed synchronous motors (15.4 MW installed motor power combined). The SAG mill will be charged with 125 mm grinding media and will be designed to operate with a 15% ball charge. The ball charge and mill speed will be adjusted to suit the ore type. The mill discharge grate will have 15 mm apertures and 65 mm pebble ports. SAG mill discharge will be screened on a 3.6 metres wide by 5.8 metres long, double deck, horizontal, wet vibrating screen with top and bottom deck apertures of 50 mm by 50 mm and 8 mm by 16 mm respectively. The screen oversize from both screen decks will be conveyed to the pebble crushing circuit. A twin deck screen is required for screening capacity and to ensure clean product to the pebble crushing circuit. The screen undersize will produce a slurry transfer size of approximately 1 mm which will be transferred to the ball mill discharge hopper.

The pebble crushing circuit will consist of a feed bin fitted with dual vibrating plate feeders, feeding two 1.12 metre diameter short head cone crushers fitted with 220 kW motors (HP300 or equivalent). Both feeders and crushers will be duty units. The pebble crushers will be operated with a 15 mm closed side setting. Pebble crusher discharge will be returned to the SAG mill via the mill feed conveyor. To protect the pebble crushers from damage by grinding media, a self-cleaning magnet will be fitted on the head chute of the pebble transfer conveyor. Grinding media removed by the magnet will discharge into the pebble crusher magnet bunker for re-use in the ball mill or reject (broken balls or miss-shaped steel scats). In addition, a metal detector will be fitted to the pebble crusher feed conveyor. In the event of a metal detection signal, a flop gate at the head chute of the conveyor will be activated to temporarily divert feed directly back onto the mill feed conveyor for return to the SAG mill. The pebbles will be directed to ground by a flop gate to bypass the crushers should this be required. Dual pebble crushers have been selected to provide a degree of redundancy which may be required for certain ore types.

The SAG mill discharge screen undersize will flow by gravity into the mill discharge hopper. The SAG mill discharge screen undersize will combine with the ball mill discharge pulp in the mill discharge hopper. One of two centrifugal slurry pumps (20 by 18 inch) with 1,500 kW drives, arranged in a duty/ standby configuration, will pump the combined mill discharge pulp to a cyclone cluster for classification.

The cyclone cluster will consist of 12 mm by 650 mm diameter cyclones, eight to ten duty cyclones and two to four standby cyclones, depending on ore type. Cyclone overflow will gravitate to the trash screens. Cyclone underflow will be split between the ball mill and the gravity circuit and this will be accomplished by partitioning the cyclone underflow launder into three compartments. One compartment will be fed by the underflow from six cyclones and will be directed to the ball mill. The other two compartments will be fed by the underflow from three cyclones each and both will be directed to the gravity circuit. The arrangement will provide the ability to adjust the proportion of the underflow treated by the gravity circuit and will suit varying circulating loads that may result from treating the different ore types or blended feed.

The ball mill will be a 7.93 metre (Inside Shell) diameter by 11.58 metres EGL overflow discharge with a dual pinion variable speed drive with 7,700 kW low speed synchronous motors (15.4 MW installed motor power combined). It will be charged with 65 mm grinding media. The ball mill speed and ball charge will be adjusted to suit the ore type. The ball mill will discharge onto a 5.4 metre diameter by 4m long trommel screen with 8 mm by 16 mm apertures. Trommel screen oversize will discharge into the ball mill scats bunker whilst trommel screen undersize will discharge into the mill discharge hopper.

The grinding area will be serviced by three vertical spindle centrifugal sump pumps (150 mm pump size) for clean-up, with floor slopes appropriately graded to the relevant sumps to facilitate ease of cleaning.

Grinding media will be delivered in bulk and stored in ball bunkers (one bunker for each media size). The SAG mill will be charged via a ball loader and the mill feed conveyor and the ball mill will be charged via an electro-magnet hoist.

Gravity Recovery

The gravity circuit will consist of four centrifugal concentrators treating a portion of the cyclone underflow. Gravity concentrate will be leached using a vendor supplied intensive leach reactor to yield a pregnant solution from which precious metals will be recovered by electrowinning.

The cyclone underflow launder will have three separate compartments. Two of these compartments will feed the gravity circuit. The number of cyclones servicing each gravity compartment can be adjusted as required. Each of the two gravity feed compartments will feed a dedicated 2.4 metres wide by 6 metres long, horizontal, wet vibrating screen. The screen deck panels will have alternating rows of 2.4 mm by 6 mm and 2.4 mm by 18 mm slots. Screen oversize will return to the ball mill feed. Screen undersize will feed the centrifugal gravity concentrators. Each screen will supply two 1.219 metre (48 inch) diameter centrifugal concentrators. The concentrators will operate in a staggered discharge cycle so that while one unit is flushing the other units are collecting concentrate. The gravity circuit has been designed for a 40-minute collection cycle followed by a standard flushing cycle.

The tailings from the gravity concentrators will return to the ball mill feed.

Concentrate from the gravity concentrators will discharge to the intensive leach reactor. The batch leach process will be initiated on a daily basis. The leaching sequence will be controlled by a programmable logic controller (**PLC**). After leaching, the residue will be returned to the mill discharge hopper by a centrifugal slurry pump and the pregnant solution will be forwarded to electrowinning located in the gold room.

Electrowinning will be carried out in a dedicated 800 mm by 800 mm electrowinning cell fitted with 12 cathodes and 13 anodes. Electrical current will be supplied from a 1,200 A rectifier. The cathodes will be stainless steel and the precious metal precipitate will be removed by washing loaded cathodes in a cathode washing station and filtering the resulting sludge. The filter cake will be dried in an oven and then combined with fluxes and smelted to produce gold doré.

Leaching and Adsorption

After screening to remove trash, the cyclone overflow from the grinding circuit will be thickened using a 38 metre diameter Hi-rate thickener and then leached with cyanide in a hybrid CIL circuit that consists of a single stage of leaching and six stages of leaching and adsorption. The total nominal pulp residence time in the hybrid CIL circuit will be 24 hours.

The cyclone overflow from the grinding circuit will gravitate to one of two duty trash screens. The trash screens will be 3.66 metres wide by 6.1 metres long, horizontal, wet vibrating screens. The screen aperture will be 0.8 mm by 18 mm. Trash screen oversize will discharge into the trash bunker and will be periodically removed for disposal by an integrated tool handler. Trash screen undersize will gravitate to the pre-leach thickener. The two trash screens provide a degree of redundancy.

Pre-leach thickener feed will be dosed with flocculant and thickened in the 38 metre diameter Hi-rate thickener to 50% solids (w/w). The thickener underflow (leach feed) will be pumped by one of two centrifugal slurry pumps (14 by 12 inch) with 315 kW drives, arranged in a duty/ standby configuration, to the CIL tanks. Dual transfer lines have been incorporated into the design to permit both pumps to be run for short periods of time in the event of high loads in the pre-leach thickener. Cyanide will be dosed into the suction of the duty thickener underflow pump, and oxygen will be injected into the leach feed line. The thickener overflow will gravitate to the process water pond via a sedimentation pond.

The leaching and adsorption circuit will consist of a 5,000 m³ leaching tank with a nominal pulp residence time for Fresh ore of four hours and six 4,200 m³ CIL tanks with a nominal 20 hour pulp residence time (leaching and CIL). For Oxide ore the residence time will be a total of 20.4 hours, for the Transition it will be 22.7 hours and for Fresh ore it will be 24 hours. The design will include the ability to bypass any tank in the train should this be required.

Cyanide will be stage dosed into the discharge of the leach tank and the first CIL tank as required. Oxygen will also be injected down the agitator shaft of the leach tank and the first two CIL tanks as required. The oxygen manifold could be extended down the leach train to CIL TK-06 but this is not necessary for Gruyere ore.

Each CIL tank will have two 20 m² mechanically wiped, inter-tank screens with 1 mm aperture stainless steel wedge wire to retain carbon. The design carbon concentration will be 9 g/L. Carbon will be advanced through the CIL circuit counter current to the pulp, on a batch basis, by recessed impeller pumps. Loaded carbon from the first stage of the CIL will be pumped to the loaded carbon screen. The loaded carbon screen will be a 1.5 metres wide by 3.6 metres long, horizontal, wet vibrating screen. Loaded carbon from the loaded carbon screen will gravitate into the acid wash column. The design advance rate for the circuit is 15 t/d. Barren carbon from the kiln (or directly from the elution column) will be returned to the circuit via the barren carbon screen. The barren carbon screen will be a 1.5 metres wide by 3.6 metres long, horizontal, wet vibrating screen.

The pre-leach thickener area will be serviced by one vertical spindle centrifugal sump pump and the leaching and adsorption area will be serviced by two vertical spindle, centrifugal sump pumps for clean-up, with floor slopes appropriately graded to the relevant sumps to facilitate ease of cleaning.

Elution and Gold Recovery

The carbon handling and gold recovery system will comprise the following:

- 18 t mild steel, rubber lined, acid wash column
- 18 t stainless steel elution column
- 6,500 kW elution heater
- A split AARL elution system with two 249 m³ pregnant solution tanks and a 249 m³ barren solution tank
- 1.5 tph carbon regeneration kiln and its associated quench tank
- An eduction water system for carbon transfer including a recycle system with a settling cone to remove carbon fines from the circuit for bagging and subsequent treatment (by others)
- An electrowinning circuit with four 800 mm by 800 mm electrowinning cells with each cell fitted with 12 cathodes and 13 anodes and supplied by a 1,200 A rectifier
- A cathode washing station and filter to recover precious metal precipitate

- An A300 smelting furnace and crucible to produce gold doré
- A secure gold room with a vault and safe for the storage of bullion.

The carbon handling system has been designed for (nominally) six elution cycles per week. However, the design of the acid washing and elution circuit will provide the flexibility to increase the elution frequency should this prove necessary. The flexibility of the design has been enhanced by the addition of a second pregnant solution tank into the elution system which will permit a new second elution cycle to be commenced whilst electrowinning of the pregnant solution from the previous elution cycle is still in progress. A barren solution tank has been incorporated into the design to permit the barren solution from electrowinning to be returned to the leaching and adsorption circuit over an extended period to minimise the disruption to the circuit. A split AARL circuit was selected in preference to a conventional AARL circuit to reduce the requirement for fresh water.

The elution area and the gold room will each be serviced by a dedicated vertical spindle centrifugal slurry pump for clean-up, with floor slopes appropriately graded to the relevant sumps to facilitate ease of cleaning.

Gold security from production to transportation logistics has been considered in the design of the process plant. The gold room will be a secured building with remote security monitoring, motion detection, seismic monitors and electronic surveillance including closed circuit television (**CCTV)** recording. Industry standard procedures involving key holder systems, swipe cards and double lock systems for vault and safe access will be employed. The two-person policy and gold room entry procedures including gold shipment procedures will be implemented. Strategies will be employed to minimise the storage of large quantities of gold on site and it is envisaged that weekly gold shipments occur. Regular external auditing will be carried out to ensure the risk of gold theft is low or eliminated. These gold security standards and procedures will be developed as part of the OR scope.

Gold shipments will be carried out by a third party specialist security firm. The security firm will be responsible for the liability of each shipment from time of collection from the gold room until delivered to the refinery.

17.3 Tailings Disposal

Final tailings from the leaching and adsorption circuit will be screened to recover carbon fines and then thickened prior to being pumped to the TSF by a two-stage pumping system. At the commencement of the Project only the first stage of pumping will be required so only this stage will be installed. The second stage of pumping will be installed once the embankment height of the TSF requires it.

The tailings from the leaching and adsorption circuit will gravitate to one of two duty tailings screens. The tailings screens will be 3.66 metres wide by 6.1 metres long, horizontal, wet vibrating screens. The screen aperture will be 0.8 mm by 18 mm. Tailings screen oversize (predominantly carbon fines) will be collected into carbon bags for subsequent treatment. Tailings screen undersize will gravitate to the tailings thickener. The two tailings screens provide a degree of redundancy.

Tailings thickener feed will be dosed with flocculant and thickened in the 38 metre diameter Hi-rate thickener to 60% solids (w/w) for all ore types. The thickener underflow will be pumped by one of two centrifugal slurry pumps (14 by 12 inch) with 75 kW drives, arranged in a duty/ standby configuration, to the tailings hopper. Dual transfer (500 mm rubber-lined steel) lines have been incorporated into the design to permit both pumps to be run for short periods of time in the event of high loads in the tailings thickener. The thickener overflow will gravitate to the process water pond via a sedimentation pond.

The contents of the tailings hopper will be pumped to the TSF by one of two pump trains, arranged in a duty/ standby configuration. Each pump train will consist of two centrifugal slurry pumps (12 by 10 inch) with 350 kW motors in series. Decant return from the TSF will be returned to the process water pond using a submersible, decant return water pump.

The tailings thickener area and the tailings area will each be serviced by one vertical spindle, centrifugal sump pump for clean-up, with floor slopes appropriately graded to the relevant sumps to facilitate ease of cleaning.

17.4 Grinding Media and Reagent Management

The following process additives will be necessary to operate the processing facilities:

- Grinding media (steel balls)
- Flocculant
- Quicklime (dry powder)
- Sodium cyanide
- Oxygen
- Carbon
- Sodium hydroxide
- Hydrochloric acid
- Liquefied Petroleum Gas (**LPG**)
- Smelting fluxes.

Grinding media and reagents will be received to site, mixed and dosed as shown in Table 17- 1. The nominal storage capacity for key consumables is at least nine days' consumption to allow for possible road closures due to inclement weather.

Table 17-1: Details of Grinding Media and reagent Systems

Consumable	Packaging	Mixing	Storage	Storage capacity (days)	Dosing	Annual Average Consumption
Grinding Media	Bulk	-	Bunkers	9	Ball loader (SAG mill); Electro-magnet hoist (ball mill)	11,700 t
Quicklime	Bulk	-	600 t	9	Variable speed drive (**VSD**) rotary valves and fixed speed drive weigh screw feeders	22,500 t
Flocculant	750 kg bags	Automated batch system with 2,100 kg bin, feeder, wetting head and 20 m^3 agitated mixing tank	50 m^3	9	VSD dosing pumps	150 t
Sodium Cyanide	26 t liquid solution in isotainers	-	440 m^3	10	Circulating pumps and dosing valves	4,500 t

Consumable	Packaging	Mixing	Storage	Storage capacity (days)	Dosing	Annual Average Consumption
Oxygen	-	-	-	-	Generated on-site by a pressure swing absorption plant	-
Sodium Hydroxide	Bulk	-	30 m³	21	Circulating pumps and dosing valves	697 m³
Hydrochloric Acid	Bulk	-	70 m³	20	Dosing pumps	1,355 t
LPG	Bulk	-	66 m³	8		2,721 m³
Smelting Fluxes	25 kg bags on a pallet	-	4 t	195	Manual mixing	7.5 t

17.5 Process Plant Services

Water Services

Raw and process water will be sourced from the Yeo Borefield (Figure 18-1) from bores fitted with multi-stage centrifugal submersible pumps and transferred to the process plant via a system of buried pipelines with a single transfer pumping station. All pipelines will be buried as far as practicable. At strategic points the pipelines will be provided with scour pits to permit the draining and air purging of the pipelines for maintenance purposes.

The Yeo Borefield will provide the raw water supply for the process plant at the required rate of 20,500 kL/d. Within the process facility, raw water will be delivered into the raw water pond which will supply the requirements for raw, gland and dedicated fire water supplies. The raw water pond will overflow into the process water pond which supplies the process water requirements for the plant. The raw water pond will be a 12,000 m³ capacity pond lined with 1 mm HDPE.

The process plant requirements for potable, safety shower and fresh water will be provided by a reverse osmosis (**RO**) plant with water sourced from the Anne Beadell Borefield. The raw water distribution system will comprise of the following:

- Two raw water pumps (222 L/s horizontal centrifugal)
- Two gland water pumps (11 L/s multi-stage centrifugal)
- Two high pressure gland water pumps (2.8 L/s multi-stage centrifugal)
- An electric fire water pump (with a diesel powered back-up pump) and electric jockey fire water pump (both 40 L/s horizontal centrifugal).

The raw water pumps arranged in a duty/ standby configuration will draw water from the raw water pond to feed non-contaminated raw water to the following areas:

- Gravity concentrators
- Intensive leach reactor
- Elution circuit
- Flocculant mixing.

One of two gland water pumps arranged in a duty/ standby configuration will draw water from the raw water pond to feed gland seals on all the horizontal slurry pumps in the process plant, except the second stage of the tailings pumps.

One of two high pressure gland water pumps, arranged in a duty/ standby configuration, will be used to boost the gland water pressure for gland seals on the second stage of the tailings pumps.

An electric fire water pump (with a diesel powered, back-up pump) and electric jockey fire water pump will draw water from the base of the raw water pond to supply a dedicated fire water supply to hydrants and hose reels located throughout the process plant. The raw water pond will have a dedicated fire water reserve of at least 600 m^3 capable of sustaining four fire hydrants for four hours.

One of two potable water electric pumps arranged in a duty/ standby configuration (a diesel potable water pump will provide back-up to the electric potable water pumps) will feed water services around the processing facility and will keep a 23 m^3 capacity, safety shower water tank topped up via a flow valve. An electric safety shower circulating water pump will also draw from the safety shower water tank to feed the safety showers distributed throughout the process plant. A diesel safety shower circulating water pump will provide back-up to the electric safety shower circulating pump in the event of power failure or shutdown of the safety shower circulating pump. The safety shower water tank will be fitted with a chiller system to maintain an acceptable water temperature during the summer months. The safety shower pipework will be lagged to provide thermal insulation.

The raw water pond and all thickeners will overflow to a 48,000 m^3 capacity process water pond lined with 1 mm High Density Polyethylene (**HDPE**). The process water pond will overflow to the site drainage pond. Two of three process water pumps (400 L/s horizontal centrifugal), arranged in a duty/ standby configuration, will draw from the base of the process water pond to feed process water to the following:

- General service points throughout the plant
- Grinding area dilution water
- Flocculant dilution
- Flushing water for the tailings lines.

Compressed Air Services

A set of two wet screw air compressors with 75 kW motors will generate plant air which will be stored in a plant air receiver with 5.0 m^3 capacity for distribution around the process facility.

Plant air will be filtered and dried in a refrigerated drier before being directed to a 1.0 m^3 capacity instrument air receiver. Instrument air will be reticulated to instruments throughout the process plant from this air receiver.

Electrical Services

Power will be supplied from a gas powered power station (by BOO provider) located in close proximity to the process plant. Power will be generated at 11 kV and will be distributed throughout the plant areas and the around the tailings storage areas via cable. Power to peripheral areas, such as the Yeo Borefield and the village will be reticulated at 22 kV.

An 11 kV substation will be established adjacent to the grinding area to feed large drives in the wet plant area as well as substations feeding low voltage drives in the grinding, leaching, water services, gold room and reagents areas. The crushing area substation will be fed directly from the power station.

An 11 kV powerline will be constructed from the power station to the tailings decant area.

Supplies to the village and airstrip, warehouse and workshops, offices and mining workshop will be fed via a step-up transformer located adjacent to the power station. Power to the Yeo Borefield, village and airstrip will be reticulated via a 22 kV overhead powerline.

Kiosk type substations will be installed to service the warehouse and workshop area, the mining workshops and office areas.

The borefield transfer pumps will be supplied via a pad-mounted 1,500 kVA transformer with switchgear and variable speed drives installed in a prefabricated switch-room. Other borefield loads will generally be supplied by pole-mounted transformers and outdoor motor starters.

Substations in the plant area will comprise of one or more transformers located in a bunded and fenced compound located adjacent to a transportable prefabricated switch-room. Motor Control Centres (**MCC**), variable speed drives and Plant Controls System (**PCS**) equipment will be located within the switch-rooms.

The substations in the plant area will be as follows:

- Crushing 2 Megavolt-ampere (**MVA**), 415 V
- Crushing 2.5 MVA, 690 V
- Grinding 2.5 MVA, 415 V
- Grinding 2.5 MVA, 690 V
- Leaching 2.5 MVA, 690 V
- Wet Plant 2 MVA, 415 V
- Water Services 2.5 MVA, 690 V.

The switch-rooms will be prefabricated, insulated, air-conditioned buildings constructed of non-combustible materials, on a substantial steel base and fitted with Very Early Smoke Detection Apparatus (**VESDA**) smoke detection and hand held fire extinguishers. The switch-rooms will be elevated above the ground on either concrete or steel plinths to allow for the installation of cables from below.

The switch-rooms will house the medium voltage switchboards, MCCs, variable speed drives, lighting and small power distribution boards and PLC cubicles.

The 415 V and 690 V MCCs will be of Form 4 design either single sided or back to back construction and arranged for connection from below. The PLC equipment associated with the motor control modules will be built into one or more tiers of the MCC and the PLC inputs and outputs (**I/O**) will be hard wired between drive modules and the PLC racks.

Communications between the MCC and control system human machine interface (**HMI**) will be via ethernet and by fibre or copper as appropriate. Communications with the borefield will be via telemetry.

Low voltage variable speed drives will be the variable voltage, variable frequency (**VVVF**) six pulse type and will be either wall or floor mounted depending upon their size and weight.

All drives will have local control stations with start and stop buttons adjacent to the drive to provide local control for maintenance. Selected drives will also be remotely operable from the central control room via the operator interface terminal. The operating status of all drives will be displayed on the operator interface mimic pages. Any drive fault will be reported by the control system and an alarm will be initiated and logged.

Control voltage for all drives and local control stations will be at 24 V DC.

Cable ladders to be installed throughout the plant will be National Electrical Manufacturers Association 20B type, hot dipped galvanised. Where necessary cable ladder bends, risers, tees and reducers of the same specification will be installed. Peaked cable ladder covers will be installed where cables in the ladder are subjected to direct sunlight or the potential for mechanical damage.

Screened cable will be used for all variable speed drive applications.

All high voltage cables will be cross-linked polyethylene PVC with copper screened conductors, wire armouring and HDPE outer when installed below ground.

High pressure sodium and metal halide light fittings will be used for the general plant lighting. Battery back-up lighting will be installed in all switch-rooms and access ways to ensure safe evacuation in the event of a blackout. All low voltage power circuits and sub circuits will be protected by instantaneous residual current devices (**RCD**).

17.6 Process Control

The process plant controls system will be a PLC based system. PLCs included in vendor supplied equipment that interface with the main control system will be specified by Gold Road to achieve commonality and standardisation. The HMI will utilise standard personal computers running Citect software to provide control. The main control room will be located in the grinding area with a subsidiary control room in the crushing area.

Process Control System Description

The I/O associated with the PCS will comprise three different control areas:

- Field instrumentation including control valves and actuators
- Rotary equipment driven by motors
- Electrical power equipment.

Analogue I/O signals are predominantly associated with the process measurements of flow, pressure, density, pH and temperature and the control of modulating valves and actuators, and 6 HV and 23 LV Variable Speed Drives (**VSD**s).

Digital I/O is typically associated with the status and control of drives, valves and actuators and mechanical plant. In addition, the digital I/O is used to detect alarm conditions and annunciate warnings.

Both analogue and digital I/O are associated with the status of the electrical power equipment and power monitoring. It is estimated that there will be approximately 2,500 I/Os in total.

The process plant comprises the following areas:

- Primary Crushing
- Grinding and Classification
- Gravity Recovery
- Pre-Leach Thickening
- Leaching and Adsorption
- Carbon Handling and Gold Recovery
- Tailings Thickening and Disposal
- Reagents
- Air and Water Supply.

The process facility will be controlled from one main control room.

The I/O in each area associated with the MCC will be installed in one or more tiers of the MCC and will be hard wired to the starter modules within the MCC. The digital and analogue I/O associated with the process instrumentation will be wired to the Process Control Cubicle (**PCC**).

In the crushing and water services areas the PCC will be combined with the MCC I/O tiers. In other areas within the plant, the PCC will be stand alone.

Provision will be made in each PCC for the power distribution to the field instrumentation associated with that PCC.

Four visual display units will be installed within the control room to provide operator interface. These units will present the operator with graphical process information in the form of trends, mimic pages, alarm summaries, logs and reports. This interface will also enable the operator to start and stop equipment, control VSDs and alter process set-points. A single visual display unit will be installed in the crusher control room with similar control and display capabilities. Major items of plant will be covered by CCTV with viewing available in the main and local control rooms.

The adjustment of controller parameters will be made from the controller face plate and it will be possible to password protect this to prevent unauthorised adjustments. Display screens will be configured for the trending of individual or related parameters and a number of alarm pages will be developed to allow the setting of alarm points attached to various parameters. All analogue input signals including outputs from flow, pressure, temperature and weighing instruments will be displayed appropriately on mimic pages. A short-term trend plot for each I/O from the system can be provided where required on the mimic pages. A data historian will be used for long-term storage of all process plant data, data analysis and reporting.

The analogue and digital I/O associated with the plant instrumentation will be cabled to one or more PCCs within the plant areas. These units will be located within the area switch-rooms and will house the PLC racks, instrumentation power supplies and communication hardware. Communications between these units and the control system HMI will be via ethernet over fibre optic or copper cable as appropriate.

An uninterrupted power supply (**UPS**) will be installed to provide a 30 minute back-up power to the process control system in the event of a loss of power.

General Process Control Philosophy

Control of equipment will be designed with three modes of operation, as follows:

- **Remote (Group start/ stop - Normal operating mode)** - in this mode, a group of drives can be started sequentially and stopped automatically (sequentially) by an operator from the PCS. All drive interlocks and process interlocks will be operational.
- **Remote (Individual start/ stop)** - in this mode, an individual drive can be started and stopped by an operator from the PCS. All drive interlocks and process interlocks will be bypassed (except for critical interlocks associated with that particular drive).
- **Local** - in this mode each drive can only be started and stopped from a local control station adjacent to the drive. Drive and process interlocks are bypassed in local mode. Local mode is intended for use for testing or maintenance purposes.

In any of the above modes, the drives can be stopped from the Local Control Station (**LCS**). Each drive will have a LCS located adjacent to the drive.

All rotating equipment will be equipped with local physical lock-out capability to facilitate safe isolation by means of locks and tags for maintenance and similar purposes.

18 PROJECT INFRASTRUCTURE

The infrastructure for the Project includes all non-mining facilities outside the process plant that are required to support the plant or processing functions and the Project. Infrastructure includes the:

- Upgrading of the Mt Shenton-Yamarna road from the intersection of Great Central Road
- Site access road from Mt Shenton-Yamarna Road to the process plant site
- Borefield access tracks
- Access roads to the accommodation village, airstrip and general site
- Magazine and emulsion compound access track
- Airstrip including the runway strip and terminal facilities
- Surface water diversion channels
- Construction of temporary and permanent accommodation villages
- Process and site drainage water ponds
- Emergency fuel storage at the power station
- Borefield raw water supply for processing
- Potable water supply from a Reverse Osmosis (RO) plant
- Sewage treatment
- Tailings storage facility
- Turkey nests and scour pits for road and pipeline maintenance
- Power station.

The proposed location of the infrastructure is shown in Figures 18-1 and 18-2.

Infrastructure design takes into consideration the environmental factors for the area including:

- Climate temperature extremes
- The flows captured by the surface water diversion channels to the west of the Project open pit, east of the TSF and along the airstrip. Rainfall run-off ponds and channels/ culverts surrounding both sides of the plant area are designed for a 1 in 100 year ARI storm event and time of concentration varying due to the catchment area
- The FS water balance for the TSF uses an average evaporation rate of 3,000 mm per year
- All buildings and structures will be compliant with relevant AS/NZS 1170.2, local regulations and building codes including provision for cyclone rating.

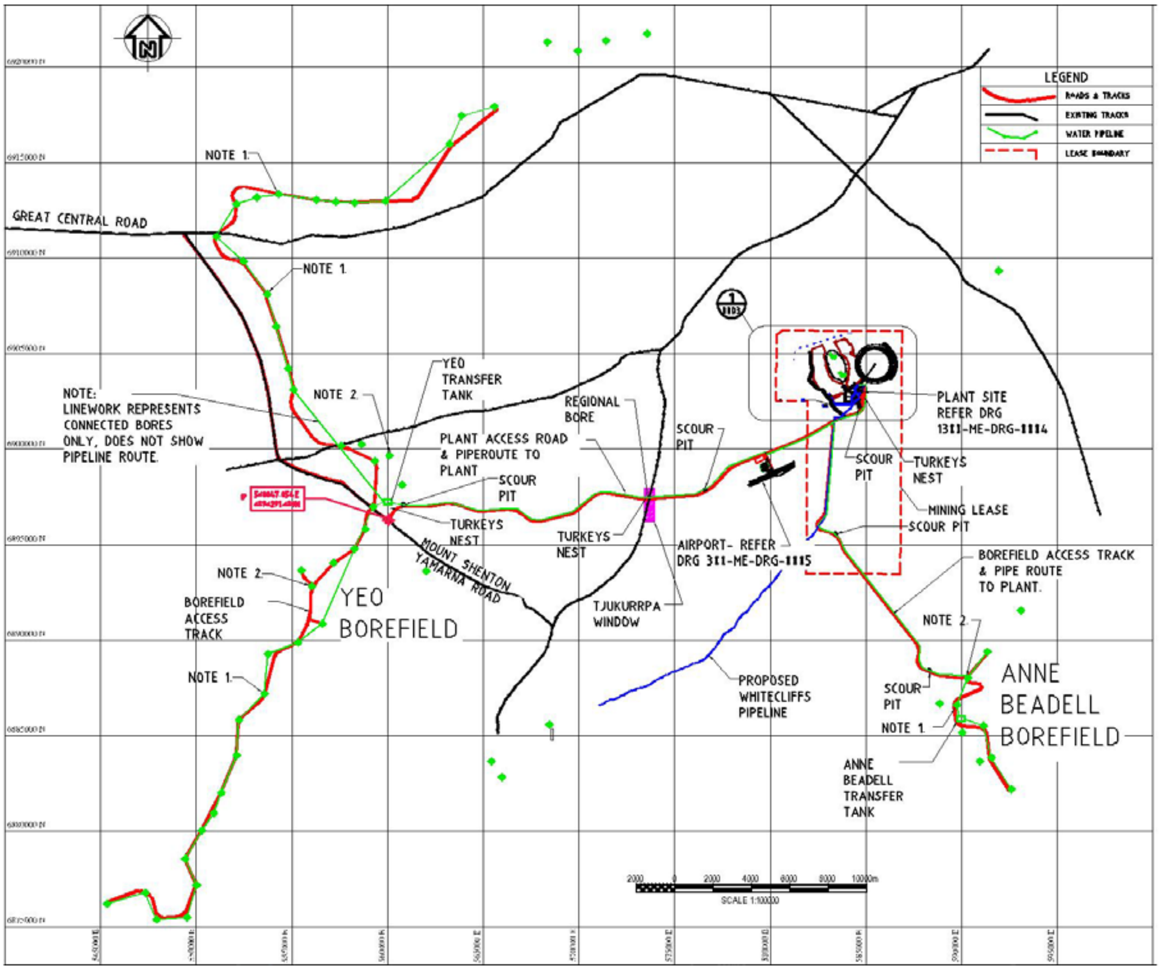

Figure 18-1: *Overall Site Plan of the Project*

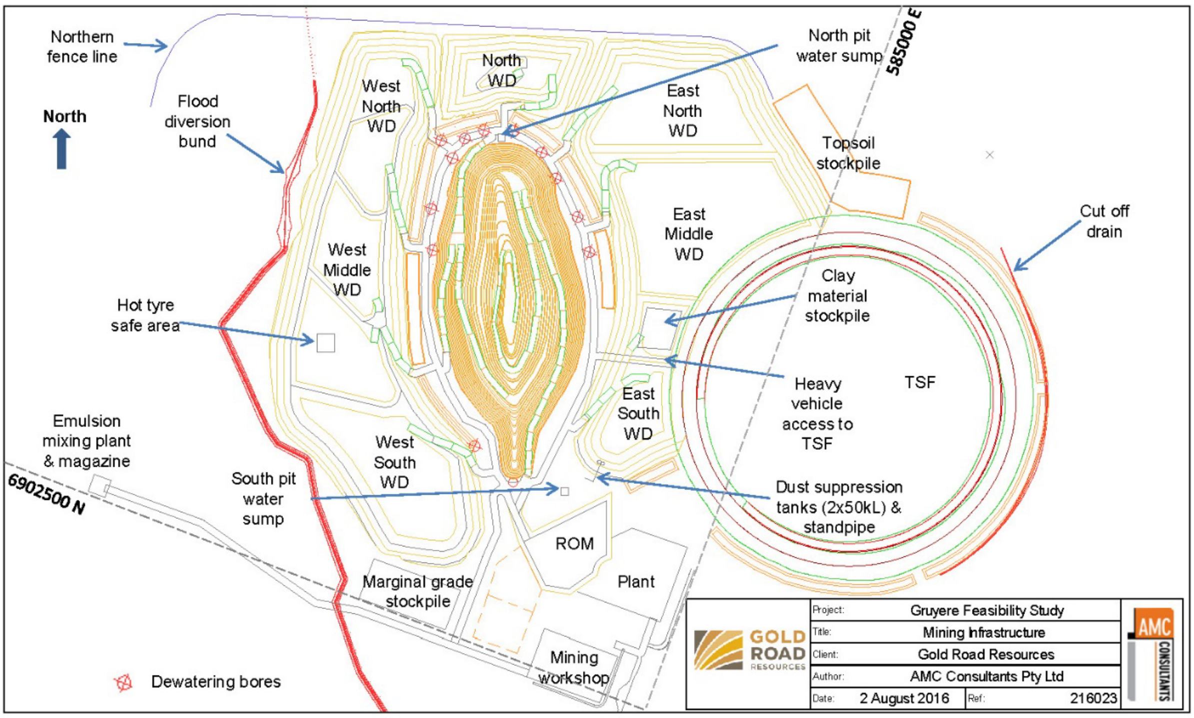

Figure 18-2: *Layout Plan of the Gruyere Pit and Surrounding Infrastructure*

18.1 Access Roads and Tracks

Road access to the Project site will be from Laverton along the Great Central Road, turning off 153 kilometres from Laverton; the site access road will be 47.7 kilometres in length, comprising 19.2 kilometres on the Mt Shenton-Yamarna Road and 28.5 kilometres of the main site access road. The accommodation village/airstrip access road will be located approximately six kilometres from the end of the main site access road. The main site access road will terminate adjacent to the mine contractors' service area and to the southern entrance to the process plant site. A further 1.2 kilometres of plant access roads will connect the main site access road to the mine contractors' service area and the power station.

The current condition of the Great Central Road is suitable for the construction traffic requirements of the Project and it is noted that regular maintenance of the road is undertaken by the Shire of Laverton. In order to maintain a trafficable surface during the construction period it is anticipated that the maintenance of the road will be continued by the Shire. As part of the Shire's road safety procedure, the Shire closes the roads during excessive wet weather periods or restricts traffic movements.

The site access road will include the upgrade of 19.2 kilometres of the Mt Shenton-Yamarna Road and 28.5 kilometres of new construction (main site access road) to the process plant site. The site access road will require regular maintenance to retain a trafficable surface during the construction period. The capital estimate includes the cost of the road contractor maintaining the road surface for two days per week following the road construction period. Heavy vehicle traffic movement will be restricted during wet weather periods to minimise damage to the trafficable surface. The road and track sections are detailed in Table 18-1.

Table 18-1: Major Road and track Design Sections

Section	Road	Length (Km)
1	Mt Shenton-Yamarna Road (Shire road)	19.2
2	Main site access road (private road)	28.5
3	Accommodation Village and Airstrip access road (private road)	1.1
4	Yeo Borefield access track (private track)	67.0
5	Anne Beadell Borefield access track (private track)	29.2
6	Magazine access track (private track)	3.4
7	Plant Road 1 (private road) Plant Road 2 (private road)	0.6 0.6

The pavement for access roads will be gravel (or calcrete), locally sourced from borrow pits. Approximately 110,000 m^3 of gravel material will be required for the Mt Shenton-Yamarna Road and the main site access road.

The pavement for tracks shall be formed from in situ material cut from the adjacent drainage and compacted and shaped to final formation profile.

18.2 Airstrip

A sealed airstrip, with associated facilities, to service the operation will be constructed adjacent to the accommodation village six kilometres south-west of the process plant. The airstrip's sealed runway will be 2,100 metres long and be suitable for use by a Fokker F-100 (or similar) aircraft and capable of carrying up to 100 passengers.

Geometric design of the airstrip will be performed to the CASA Manual of Standards 139, Version 1.13 – March 2016 (MOS139) for compliance as a Code 3C airstrip; the runway orientation will be 70°/ 250°.

Access to the airstrip will be via the accommodation village/ airstrip access road.

18.3 Accommodation Village

The accommodation village, containing both the temporary and permanent facilities, will be constructed approximately six kilometres south-west of the process plant and sited adjacent to the airstrip within a well-drained, elevated area 400 metres long and 400 metres wide. The earthworks will be a balanced cut to fill with disturbance of vegetation kept to a minimum.

The works will comprise the supply and site installation of 600 accommodations rooms and support services in total consisting of:

- 300 construction (temporary) rooms at the accommodation village
- Support services building, including administration, wet mess and dry mess
- 300 permanent rooms at the accommodation village.

The accommodation village will consist of a 300 person construction camp and a 300 person permanent village, and will be made available in two initial stages to facilitate the changes through the phases of the Project. A final third stage will focus on clean-up and the establishment of landscaping and sport facilities.

As described, the accommodation and services will be developed in the following defined stages:

- Stage 1 300 rooms at the construction camp by February 2017
- Stage 2 300 rooms at the Permanent accommodation village by June 2017
- Stage 3 final establishment of facilities, connections and clean-up – post June 2017.

It is anticipated that all 600 rooms will be utilised during the construction/ commissioning stages of the Project. Upon completion the initial 300 person construction camp will be utilised for contract staff as required during shut-downs.

18.4 Process Plant Infrastructure

The process plant and administration facilities will be contained in an area approximately 400 metres long and 400 metres wide and located to avoid the major local water courses. The infrastructure will contain the following items:

- Administration office complex
- Emergency response buildings
- Process plant buildings
- Process plant workshop, warehouse and store
- Reagent store
- Assay laboratory
- Diesel storage and other miscellaneous facilities

18.5 Tailings Storage Facility

The TSF will be developed as part an IWL, with a perimeter waste dump surrounding a centrally placed TSF. The scope of work for Coffey comprised:

- Geotechnical investigation of the selected site
- TSF design
- Development of a closure/ rehabilitation concept for the TSF
- Determination of quantities for TSF construction to allow cost estimation
- Development of a construction technical specification for earthworks and associated works
- Compilation of cost estimate
- Preparation of preliminary design drawings for Stage 1 and final embankments including underdrainage and decant details.
- Preparation of an Engineering report.

The proposed TSF has been designed in accordance with the following guidelines:

- DMP (2015), 'Guide to the preparation of a design report for tailings storage facilities (TSFs)'
- DMP (2013), 'Code of Practice: Tailings Storage Facilities in Western Australia'.

The following design parameters and assumptions have been adopted for the FS:

- Total ore production 92.4 Mt which relates to an approximate production rate of 8.0 to 8.2 Mtpa for the first 3 years, reducing to 7.5 Mtpa for the remaining 9.2 years
- Process CIL with tailings to TSF at 60% solids (w/w)
- Tailings at P80 of 125 µm
- Tailings dry density of 1.5 t/m³
- Tailings beach slope 1 vertical:100 horizontal (V:H)

The tailings parameters are based on laboratory testing carried out as part of the PFS and also considers tailings performance on similar gold tailings projects in the Western Australian Goldfields. The adopted beach slope was assumed based on Coffey's experience and the expected tailings properties.

The proposed process plant site is approximately one kilometres from the pit and the proposed TSF immediately north of the plant site and east of the pit. The proposed plant and TSF are located out of the pit rim failure zone.

The Project area terrain is flat to gently undulating. Drainage in the pit and plant areas is to the north-east and ultimately towards Yeo Lake.

Consultant MBS's Waste Rock Characterisation Report completed in July 2015 indicated that the mine waste to be used in TSF starter embankment and IWL construction is likely to be benign, with no significant metal enrichment and is assessed as Non-Acid Forming. However, there are materials tested that are potentially dispersive and would need to be managed (i.e. dispersive material will not be placed on outer wall slopes).

Results of geochemical testing of tailings from metallurgical sampling indicates the tailings solids are likely to have low levels of total sulphur at around 2 mg/L and are Non-Acid Forming. The tailings supernatant will likely be alkaline with no significant enrichment of metals. The results of geochemical characterisation test work indicate that the tailings will be relatively benign and lining of the TSF should not be required, provided seepage is adequately managed and controlled. MBS indicated that the tailings are saline and that consideration should be given in the facility closure design for capillary rise in covers. Options will likely require a capillary break layer of rock waste or sand.

Based on the FS mining schedule it is anticipated there will be sufficient material from mining for use in tailings storage construction. Pre-stripping of waste and ore will produce approximately 13 Mt of predominantly cover and saprolite material over a nine-month period for the completion of the Stage 1 of the TSF. As an indication of adequate material requirements, approximately 0.2 Mm^3 of saprolite and 2.3 Mm^3 of waste will be required for Stage 1 embankment construction and a further 0.3 Mm^3 of saprolite required for basin lining.

The following design objectives were adopted:

- Reduce upfront capital costs and minimise the overall construction cost
- Minimise daily inputs required for tailings storage operation and management
- Maximise the tailings density and storage capacity by rotating the deposition points
- Provide adequate tailings and stormwater storage capacity
- Maximise return water to the plant
- Minimise environmental impacts (i.e. reduce seepage water losses).

The proposed TSF perimeter embankment will be raised in six stages comprising four lifts of five metres and one lift of 5.5 metres from the Stage 1 (starter) crest RL 412 metres to Stage 6 crest RL 437.5 metres and then one small lift of 1.6 metres to the final Stage 7 crest RL 439.1 metres. The maximum embankment height of Stages 1 and 7 will be approximately 14 metres and 41 metres respectively. Further refinement of the stage heights will be undertaken in the next phase of engineering design. Details of the embankment geometry of Stage 1 and future stages are shown in Figure 18-3.

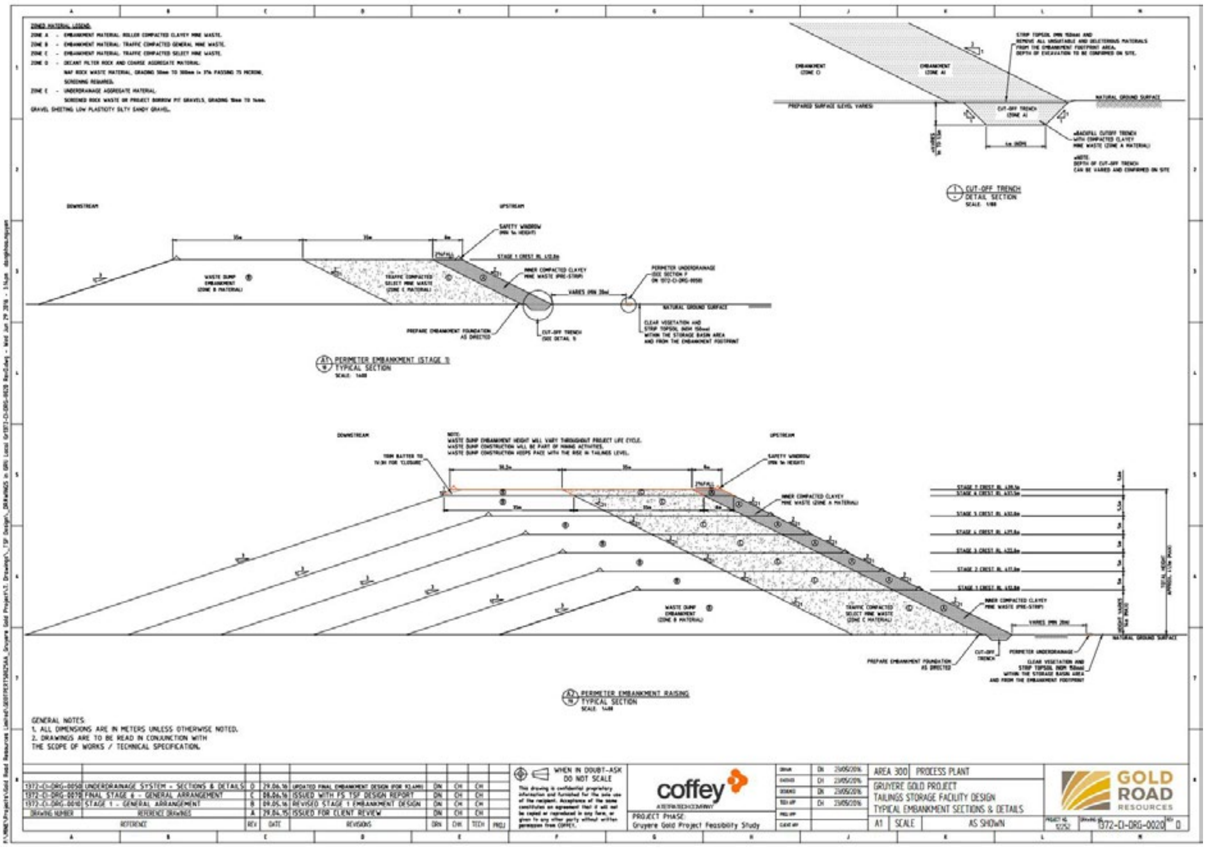

***Figure 18-3**: Tailings Storage Facility Embankment Geometry*

Storage capacity, earthworks volumes for development of the proposed TSF, including the seven stages of construction are outlined in Table 18-2. Waste dump volumes have been calculated assuming horizontal-layered, full dump width construction. The waste dump design and volume estimation will be conducted and confirmed by Gold Road's mining consultant.

The total design storage capacity is estimated at 62 Mm^3 (or 92 Mt based on a tailings dry density of 1.5 t/m^3). The total storage life will be 12.2 years (based on a tailings production of 7.5 Mtpa).

Table 18-2: Tailings Storage Facility - Capacity and Earthworks Volumes

Stage	Embankment Crest RL (m)	Cumulative Storage Volume (Mm^3)	Cumulative Storage Capacity (Mt)	Cumulative Storage Life (Years)	Perimeter Embankment Volume - Compacted Material (per stage) (m^3)	Mine Waste Dump Embankment Volume (per Stage) (m^3)
1 (Starter)	412	5.98	8.98	1.1	211,000	2,338,000
2	417	14.16	21.25	2.6	197,000	2,772,000
3	422	24.45	36.67	4.7	207,000	3,637,000
4	427	34.99	52.48	6.8	209,000	4,425,000
5	432	45.78	68.67	9.0	212,000	5,244,000
6	437.5	57.96	86.94	11.5	235,000	7,211,000
7 (Final)	439.1	61.62	92.43	12.2	69,000	589,000

The TSF is designed to store 92.4 Mt of tailings, or 61.6 Mm^3, based on a deposited tailings dry density of 1.5 t/m^3. Should additional nearby orebodies or an underground resource be developed during operations, the TSF will need to expand to accommodate the increased production. The current design is adequate for anticipated Project production and a decision on expansion can be delayed until later in the life of the facility.

18.6 Water Supply

Groundwater Resources

Groundwater occurs in the region surrounding the Gruyere deposit within relatively shallow Quaternary alluvial and calcrete unconfined aquifers, and in a deeper, confined aquifer in Eocene sediments of the Werillup Formation. These sediments occupy the Yeo Palaeochannel. The Perkolilli Shale between the Quaternary and Werillup Formation forms an aquitard between the two aquifers. A summary of the aquifers present in the Project area are presented in Table 18-3.

Gold Road has targeted the Yeo water supply area (**WSA**) and the Anne Beadell WSA to source the water supply for the Project. Groundwater salinity in the Yeo Borefield is interpreted to be stratified, with better quality brackish water overlying saline to hypersaline water at depth. Groundwater in the shallow calcrete in the Anne Beadell Borefield is often brackish and classified as 'hard' due to the high carbonate content. A series of 62 shallow bores drilled into the Quaternary deposits over the Yeo Palaeochannel and Anne Beadell tributary for the Project encountered water with salinity values between about 580 mg/L to 23,200 mg/L, with an average of around 7,800 mg/L Total Dissolved Solids (**TDS**).

Figure 18-4 is a geological long-section through the Yeo Borefield within the Yeo WSA which shows the Yeo Palaeochannel sedimentary succession. Figure 18-5 shows the locations of the Yeo and Anne Beadell WSAs.

Table 18-3: Summary of Aquifer Types and Yields in the Gruyere Region

Aquifer	Geological unit	Max Saturated thickness (m)	Bore yield (kL per d)	Aquifer potential	Water quality
Palaeochannel					
Alluvial and calcrete	Quaternary deposits	14	0-500	Low - moderate	Brackish – saline
Perkolilli Shale	Perkolilli Shale	29	-	Aquitard	
Yeo Palaeochannel aquifer	Werillup Fm.	+81	200-2000	High	Saline– hypersaline
Permian	Paterson Fm.	+100	-	Low – moderate	Brackish – hypersaline
Archean Basement	Upper Saprolite	~50	-	Low	Brackish – saline
	Lower Saprolite – Saprock (transition zone)	~100	0-1000	Low - moderate	Brackish – saline

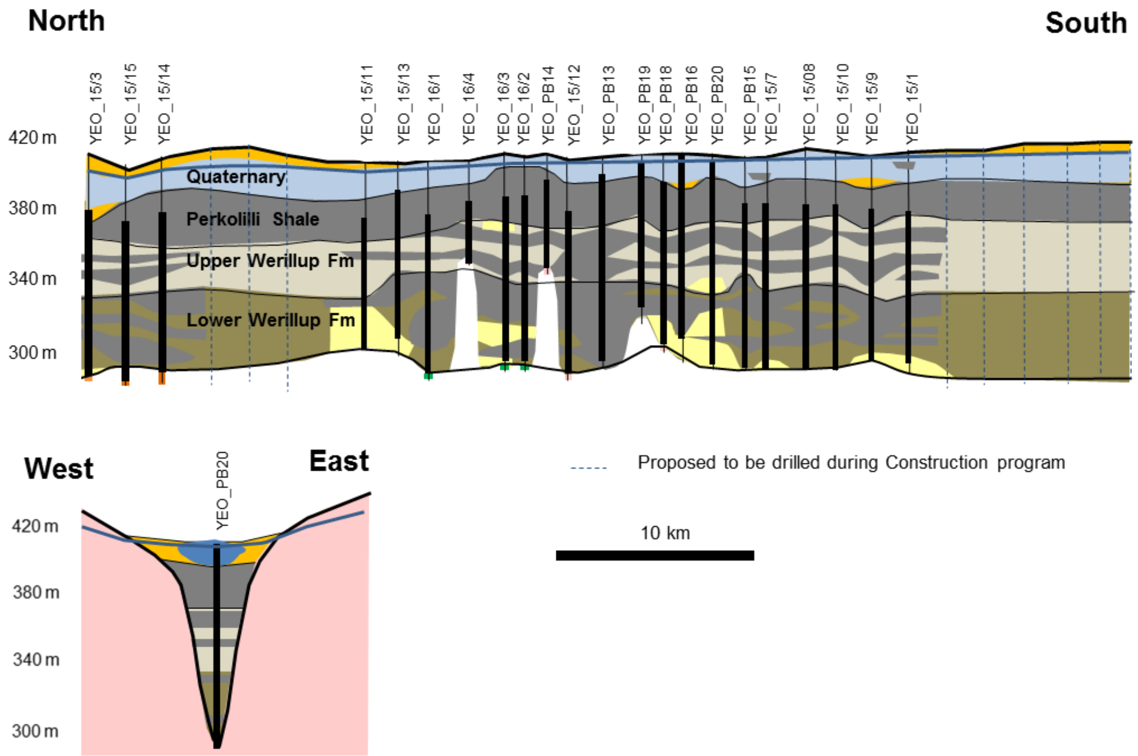

Figure 18-4: *Geological Long Section North-South through the Yeo Water Supply Area with Representative Bores*

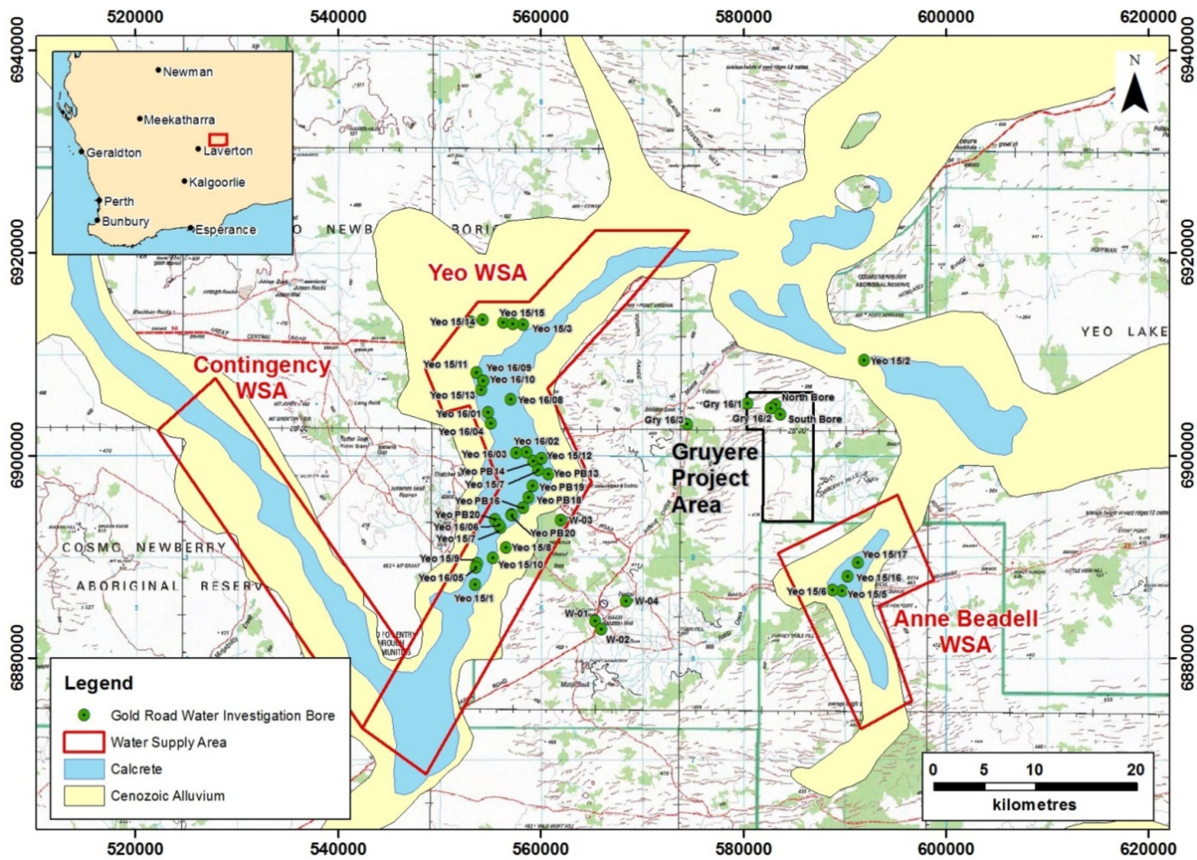

Figure 18-5: *Location of Yeo and Anne Beadell Water Supply Areas in the Yeo Palaeochannel*

Site Water Balance

The Project will require water during the development and operational phases. A full Project water balance study has been prepared and Tables 18-4 and 18-5 summarise the operational water losses and water sources.

Table 18-4: Summary of the Project Water Losses

Water Balance Losses	Water Balance (GL per year)			
	Construction Years -2 to 0	Operations – startup Years 0 to 0.5	Operations – oxide Years 0.5 to 2.5	Operations – fresh Years 2.5 to 13
Dust suppression – regionally	0.3	0.1	0.1	0.1
Dust suppression – Project area	0.8	0.4	0.4	0.4
Village and admin potable water	0.1	0.1	0.1	0.1
Concrete batching	0.2			
Elution circuit		0.1	0.1	0.1
Nett seepage/evaporation losses in IWL		2.8 to 3.6	2.4 to 2.5	2.4 to 2.5
Moisture retained in IWL		2.3 to 2.6	2.3	2.3
Total	1.4	5.7 to 6.8	5.3 to 5.4	5.3 to 5.4

Table 18-5: Summary of Project Water Sources

Water Balance Sources	Water Balance (GL per year)			
	Construction Years -2 to 0	Operations – startup Years 0 to 0.5	Operations – oxide Years 0.5 to 2.5	Operations – fresh Years 2.5 to 13
Anne Beadell Borefield to RO	0.4 to 0.1	0.3	0.3	0.3
Anne Beadell Borefield – construction makeup	0.1			
Regional bores (Central Bore)	0.1			
Moisture contained in ROM		1.0	0.8	0.4
Makeup water from Yeo Borefield		3.4 to 4.9	3.2 to 3.7	3.9 to 4.2
Pit dewater/depressurisation bores	0.2 to 0.8	0.8 to 0.2	0.8 to 0.2	0.5 to 0.1
Supply bores in plant area	0.3 to 0.0			
Regional bores	0.3	0.1	0.1	0.1
Rainfall over IWL cell		0.1 to 0.3	0.1 to 0.3	0.1 to 0.3
Total	1.4	5.7 to 6.8	5.3 to 5.4	5.3 to 5.4

During the operational phase, the Project will require an estimated peak total of 5.5 GL per year of groundwater from the Yeo and Anne Beadell Borefields drawing from palaeochannel and fractured rock aquifers (i.e., excluding moisture input in the ROM and in rainfall over the IWL). Water demand is expected to be highest at the beginning of the operational phase due to the following:

- Process plant throughput rates are expected to be highest during the processing of the oxide and transitional ore types, anticipated to be 8.8 Mtpa for the first 2.5 years, then declining to 7.5 Mtpa for fresh ore
- No or very low decant return water from the TSF in the first six months of the operational phase.

Consequently, annualised water demand is expected to peak in the first year of the operational phase at between 5.7 and 6.8 GL per year and then decrease to an ongoing requirement of around 5.4 GL per year in subsequent years. Water balancing has confirmed that the Yeo Borefield can supply greater than 100% of the instantaneous peak process water demand (if required) for the maximum process plant ore throughput rate of 8.8Mtpa even during unlikely scenario of having no tailings thickener on-line, no decant return from the TSF, no other additional water sources and allowing for the site requirement for dust suppression.

Site Water Supply

The bulk of the raw water supply for the Project will be sourced from the Yeo Borefield, located 25 kilometres west of the Project. The Yeo Borefield will not be ready until the completion of the construction phase of the Project. In the interim, water for the construction phase would be sourced from advanced development of out-of-pit dewatering and mine site water bores in the Project area, supplemented by the brine reject from the RO plant and an additional 0.2 GL per year (400 kL per day) of untreated makeup water supplied from the Anne Beadell Borefield as required.

Yeo Borefield Water Supply

The Yeo Borefield will be capable of providing 7.5 GL per year of moderate to high salinity raw water to the process plant for use as process water. Features of the borefield include the following:

- Thirty-two water bores (23 duty and nine standby) will be installed along the 65 kilometres length of the Yeo Borefield for the supply of raw water to the process plant. The water quality is estimated to be in the range of 25,000 mg/L to 100,000 mg/L TDS
- The borefield will consist of two branches, each approximately 33 kilometres long stretching roughly north and south from the intersection point with the Mt Shenton-Yamarna Road
- Bores on each branch will deliver into a header pipe which will terminate at the raw water break tank at the intersection point of the road. Each branch pipeline will incorporate a pair of in-line booster pumps (duty and standby)
- The two raw water break tanks will have a total capacity of 3,000 m^3 and will be steel with a plastic lining, equipped with a pair of transfer pumps (duty and standby). The pumps will deliver into the transfer pipeline
- The 560 mm outside diameter, HDPE transfer pipeline will be installed parallel to the site access road and will deliver the water to the raw water pond at the process plant
- All borefield pipelines will be buried wherever possible. If the pipeline route encounters hard digging conditions that prevent cost effective excavation, alternative installation measures including installing the pipeline at ground level and covering with suitable spoil for fire protection will be considered

- All pumps in the Yeo Borefield will be powered by a 22 kV overhead powerline from the power station at the process plant
- All pumps will be controlled by telemetry from the process plant
- Water flow monitoring of each input to the bore water pipeline will be incorporated to enable real-time monitoring of the water flows.

All bores will be fitted with stainless steel bore pumps, impellers and motors and stainless well head arrangements and HDPE discharge pipelines. All bore compounds will be fenced.

Anne Beadell Borefield Raw Water Supply

The Anne Beadell Borefield will provide raw water supply to the process plant. The borefield will consist of six bores (four duty and two standby), located an average distance, by track, of 29 kilometres from the process plant and will provide brackish raw water to the raw water tank.

The bore pumps will deliver into a break tank with a total storage capacity of 266 m^3. The pumps will be powered by diesel generators due to the remote location and small number of bores. A pair of raw water transfer pumps (duty and standby) will deliver the water through a single 180 mm co-extruded HDPE pipeline to the RO feed water tank at the process plant.

The brackish to saline water from the Anne Beadell Borefield will be pumped to the RO plant feed tank in the process plant area. This water will be processed at 1,100 m^3 per day through the RO plant, producing 700 m^3 per day of permeate. The permeate will supply the operation and accommodation village with fresh and potable water.

All bores will be fitted with stainless steel bore pumps, impellers and motors, stainless well head arrangements and HDPE discharge pipelines. The borefield pipeline will be buried wherever possible for fire protection purposes. All bore compounds will be fenced.

The potable water bores will be powered by self-bunded power generating sets and will be controlled through a telemetry system at the process plant.

Sewage and Waste Water Supply

Two organic waste treatment systems will be installed to process waste water streams from ablutions and other facilities at the village (construction and permanent), and the process plant site.

A 250 m^3 per day capacity containerised waste water treatment plant will be installed at the village. The plant will be rated for peak manning at approximately 350 L per person and will be sufficient to treat the combined volume of effluent from both the temporary and permanent villages. The waste water will be aerobically treated to Class A standard which will allow recirculation of water to the village for garden reticulation.

Sewage sumps, pumping equipment and piping will be installed in the administration, processing and mine service areas, pumping effluent to a 35 m^3 per day capacity sewage plant. Treated waste water will be discharged to a dripper field to the east of the process plant.

Treated, benign sludge from the waste water treatment plants will be transported from site to an approved waste storage facility.

Diversion Channels

The minor watercourses and drainages in the vicinity of the Project site and included within the various catchment areas are ephemeral and will be dry for the majority of the time. Flows will occur periodically following significant rainfall events, particularly during the summer months from January to March, when the potential exposure to remnant cyclone and depression related high intensity rainfall is greatest. Consequently, run-off will report to the watercourses in the vicinity of the Project and, on occasion, flows may be high and may cause flooding leading to asset damage or loss and operational delays if appropriate measures are not in place. The design of the surface water management measures was based on the 100 year ARI peak flow estimates.

18.7 Power Generation

Introduction

Development of the power generation solution for the Project has followed a formal process in determining the most reliable and cost effective option.

During PFS the Power Options Study (Q3 2015) recommended a gas fired power station fuelled by gas pipeline. This is the final outcome for the FS. The power generation facility will be provided on a Build-Own-Operate basis under a Development Agreement followed by an Energy Supply Agreement.

Energy Supply Agreement Request

The Scope of Work for the power supply included the requirements for the acquisition and delivery of fuel and design, construction, operation and maintenance of the power station. The Scope of Work provided a baseline for tender.

During the tender process tenderers were requested to provide proposals for two options:

- **Option A**: BOO and maintain a gas fired power station and provide all turnkey services for the acquisition and transport of gas and construction and operation of gas pipeline lateral to the site
- **Option B**: BOO and maintain a power station to start production based on a diesel fired facility (including diesel supply and storage) with an ability to convert to gas delivered by pipeline at a later date.

Technical and financial analysis and evaluation of the proposals has shown that the gas fired facility from day one provides the least cost outcome and is therefore the preferred option.

A Miscellaneous License for the gas pipeline route which follows White Cliffs Road has been applied for. Baseline environmental surveys of the pipeline alignment have been completed and the pipeline project has been referred to the EPA under Part IV of the EP Act (outcome was “Not assessed – no advice given” on 18 July 2016). At the point of award, the Company will enter into an access arrangement to allow the Independent Power Provider (**IPP**) access to the Miscellaneous License and use of associated baseline study reports for the purposes of the IPP applying to DMP for a Native Vegetation Clearing Permit and necessary Petroleum Pipeline Act approvals for construction. Should the IPP choose, it will become responsible for any changes to tenure of the pipeline route (e.g. easement under the Land Administration Act 1997) as they will be responsible for ownership and operations.

Gold Road will be responsible for environmental approvals for the mine process plant and mining operations including the power station built, owned and operated by the IPP.

Scope of Work

The design of the power station is based on a 35 MW peak load requirement, 32 MW average load producing 255 GWh per year and capable of producing the peak load requirement at an N-2 level of reliability in all ambient conditions. This will provide generation capacity capable of meeting both the average and peak power demands of the process facilities in all circumstances with up to two of the units out of service.

The IPP will design and construct the power station based on their standard configurations suitable for the following general parameters. The general parameters for design and construction of power station is shown in Tables 18-6.

Table 18-6: General Parameters for Design and Construction of Power Station

Item	(MW)
Installed Load	42
Max Demand	35
Average Demand	32
Annual Consumption (MWh)	255,000
Largest Single Loads	14.2

The annual consumption forecast for the site is shown in Table 18-7.

Table 18-7: Power Draw and Annual Consumption for Process Plant and Infrastructure

Load Area	Installed (kW)	Consumed (kW)	Annual Usage (MWh)
Crushing	1,403	927	6,342
Grinding	32,560	25,390	202,806
Leaching/ Absorption	1,641	1,149	8,687
Elution and Gold Recovery	398	263	1,111
Tailings Disposal	952	666	4,924
Reagents Storage and Distribution	57	39	225
Water Services	3,125	2,187	17,997
Air Services	650	498	4,073
Workshop	36	25	128
Laboratory	60	42	185
Infrastructure	1,353	947	7,645
Total Power	**42,236**	**32,134**	**254,127**

Note: *Apparent differences may occur due to rounding.*

Design criteria for the power facility considered the following:

- Due to the potential for a number of electrical loads to be added to site over the Project lifetime, the design to be implemented, caters for a variable power station capacity. The plant should then be able to have additional generation installed incrementally to meet the potential future demand increase of up to 5 MW or a reduction in requirements of up to 10 MW. The IPP will be required to ensure that the power station is capable of providing 3 MW of emergency supply at all times.
- The power generation unit(s) will enable direct and controllable connection to the proposed 11 kV main distribution board. Power is used on site at either 11 kV or stepped up to 22 kV for distribution to the borefields.
- The solution must allow for future possible use of alternative or renewable energy sources such as solar or wind generation to provide a reduction in fuel fired energy.
- Economic assessment of the IPP's offer based on a 15 year economic lifetime, even though the useful equipment lifetime may be extended. This assessment is based on the current LOM of 13 years.
- Adequate metering and measurements will be included in the power station for monitoring of power generation performance. Interfacing with the process plant process control system will also be critical.
- Appropriate operations and maintenance resources are in place to guarantee the minimum operating requirements and required availability are met by the IPP.

22 kV Power Distribution to Yeo Borefield

Power for the Yeo Borefield will be reticulated below ground to a point outside of the process plant footprint and then via overhead powerlines at 22 kV.

The powerlines will extend approximately 27 kilometres from the power station along the main site access road to the intersection of the Mt Shenton-Yamarna Road and main site access road. The powerlines will then extend approximately 65 kilometres in a north-south direction to cover the Yeo Borefield.

Pole top mounted transformers will be required at each of the Yeo bores to step down to 415 V. Ground mounted transformers will be required at the borefield transfer pumps. Bore areas will be fenced.

22 kV Power Distribution to Village and Airstrip

An overhead spur line from the Yeo Borefield powerlines, approximately six kilometres west of the process plant, will extend one kilometres south to supply power to the accommodation village and airstrip.

Pad-mounted, kiosk type 22 kV/ 415 V transformers will be required at the accommodation village and a pole mounted transformer at the airstrip.

11 kV Power Distribution to TSF

An 11 kV overhead powerline from the power station will supply power to 11 kV/ 415 V transformers at the TSF for the decant pump power supply and the underdrain pump power supply.

18.8 Communications

Robust and reliable Information Technology and Communications (**IT&C**) Infrastructure is planned for the Project construction and operations phases. The plan is for these services to be installed before any significant site construction activities commence. These requirements are:

- User end device(s) connection to company data
- Voice and video telephony
- Site two-way radio communication
- Camp entertainment systems
- Stable and secure access to the Internet.

The infrastructure is also required to allow data flow from company assets and activities at the Project site back to central data repositories, such as fixed and mobile assets, mining and exploration activities, as well as staff welfare and safety in the field.

Where possible single points of failure have been engineered out of the design to provide operational redundancy. In the event of an outage, there should be no material impact on operations while repairs are made in the background. Should a point fail that creates less than an operational connection, a pre-determined priority of service for critical applications will be instigated until repairs have been made. Critical spares will be held in the site inventory so that any degraded services are kept to a minimum.

For maximum operational flexibility, Gold Road will own all of the IT&C assets (apart from the existing towers at Laverton, Nambrook Range, Bailey Range and Trusscott Range). This provides Gold Road the flexibility to avoid being locked in with a management vendor and the ability to upgrade the assets in the future should better technology become available.

The design references the IT Strategy plans to support an invisible difference in performance, from the user's perspective, between desktop and the Cloud. The short to mid-term intention to move to Windows 10 and Office 2016 is heavily reliant on Microsoft's Cloud offering. It will be imperative to have robust connections to the Internet to support that type of end-user computing model. No decision has been made on how to refresh the desktop fleet, whether it is new hardware with high-powered local compute capabilities or a Virtual Desktop Infrastructure which uses a lighter weight desktop device to connect to centralised compute power for the better protection of user data. The former will still need robust communication while the latter approach requires very high resilience to failure in the communication link. The proposed design will support both options.

Requirements for the construction and operations stages of the Project include:

- Communication access to Site
- Overall Wide Area Network (**WAN**)
- Administration Local Area Network (**LAN**) routers and switches
- Ratification of the IT Strategy
- Internet packet telephony and unified communications system
- Village entertainment system
- Intra-site microwave communications
- Wi-Fi network and Wi-Fi connection for mobile assets
- Two-way radio, both Very High Frequency and Digital Mobile Radio (**VHF/DMR**) and VHF Airport Base Station for aircraft communication.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Gold projects are in the unique position of not having to market product, other than to establish an agreement with a refiner to take product on normal commercial terms for precious metals doré production.

- **Refiner Selection** - For the purposes of the Gold Road FS it has been assumed that gold will be refined at the Perth Mint and costs reflect this option. Prior to gold production this service will be tendered for contract award.
- **Pricing Strategy** - Gold Road will negotiate the general terms of product sales with the intended refiner. For modelling purposes a gold price of A$1,500 per ounce has been used in calculating the revenue from sales for the economic analysis in Section 22.

19.2 Contracts

As part of the development and construction of the Gruyere Project, Gold Road's Owner's team will develop and manage a number of major contracts for the Project. These contracts include:

- EPC for processing facilities and associated infrastructure
- Bulk earthworks for the process plant area, access roads, airstrip and TSF
- Supply and installation of the accommodation village
- BOO for power station and gas pipeline
- Communications backbone to site
- Water bores drilling and development
- Catering and accommodation village maintenance
- Air charter services
- Mining development and pre-strip.
- All contractors (and their sub-contractors) required to conduct work for or on behalf of Gold Road must satisfy the requirements of the Company's Contractor Evaluation Process before being awarded a contract.

These contracts will be awarded as part of the Project Execution Plan prepared by Gold Road.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

The Gruyere mining lease granted in May 2016 covers an area of 6,845.5 ha. Over the last three years, Gold Road has commissioned various environmental and cultural heritage surveys within this area and more recently focussed surveys within the main footprint in the northern portion of the tenement relating to the Project. The footprint of approximately 2,084 ha takes into consideration final locations of the open pit, waste rock dumps, the TSF, access roads, processing plant and associated infrastructure. Additional surveys were completed during 2016 covering the final mining-related linear infrastructure footprints of the Yeo and Anne Beadell Borefields, water supply and gas pipeline routes, accommodation village and airstrip locations.

Surveys for vertebrate fauna, flora and vegetation, SRE invertebrate fauna, subterranean fauna and cultural heritage (anthropology) have been completed for the entire Project area and summaries of the results are presented below. Archaeological surveys will be completed toward the end of 2016.

Baseline Surveys of the Mining Lease Area

Flora and Vegetation

Levels 1 and 2 flora and vegetation desktop and field surveys were conducted over the spring and autumn seasons of 2014 and 2015, (References 5 and 6). Thirty-two broad vegetation communities were identified within the survey area. These communities were represented by a total of 44 Families, 104 Genera and 240 Taxa (including sub-species and variants).

No Threatened Flora Taxa, pursuant to Sub-section (2) of Section 23F of the *Wildlife Conservation Act 1950* (**WC Act**), the *Commonwealth Environment Protection and Biodiversity Conservation Act 1999* and as listed by the Department of Parks and Wildlife (**DPaW**), were identified within the survey area. No Priority Flora Taxa, as listed by DPaW, were identified within the survey area. None of the vegetation communities were found to have National Environmental Significance as defined by the Environment Protection and Biodiversity Conservation Act (**EPBC Act**). No Threatened Ecological Communities pursuant to Commonwealth and State legislation or Priority Ecological Communities as listed by DPaW were recorded within the survey area. The survey area is not located within an Environmentally Sensitive Area or Schedule 1 Area, as described in Regulation 6 and Schedule 1, clause 4 of the *Environmental Protection (Clearing of Native Vegetation) Regulations 2004*. The survey area is not located within any DPaW managed land. The Yeo Lake Nature Reserve, which is listed as a Class "A" Nature Reserve managed by DPaW, is located approximately 15 kilometres to the east of the Project (outside the Mining Lease).

Vertebrate Fauna

As with the flora and vegetation surveys, Gold Road also commissioned Levels 1 and 2 fauna desktop and field surveys over the spring and autumn seasons of 2014 and 2015 (Reference 7). A total of 116 vertebrate fauna species including 45 reptile, 54 bird and 17 mammal species (including three bats) were recorded. The reptile, bird and mammal assemblages recorded from the Project area were representative of the Great Sandy and Gibson Deserts, with no unexpected species or range extensions.

One species, the Rainbow Bee-eater (*Merops ornatus*), of conservation significance was considered highly likely to occur in the Project area while five species of conservation significance were considered possible to occur in the Project area (medium likelihood). A further six species of conservation significance were considered unlikely (low likelihood) to occur in the Project area.

SRE Invertebrates

A Level 1 SRE invertebrate survey was undertaken during spring of 2014 (References 7 and 8). A total of 37 potential SRE specimens were collected during the surveys. These comprised eight spiders, 25 scorpions and two pseudoscorpions. No snails were found during the survey. Further taxonomic identifications revealed that of the specimens collected, only six were potential SRE species (**sp**.), comprising three species of Mygalomorph spider and three species of scorpion. A Level 2 SRE survey was completed in October 2015 (Reference 6).

Subterranean Fauna

A Level 1 stygofauna and troglofauna (subterranean aquatic fauna that live in groundwater systems or aquifers) survey was completed in May 2015 with sampling of seven sites located within the anticipated zone of pit dewatering impact and two sites located outside of the potential impact zone for regional context (Reference 9). Three stygofauna species were recorded:

One species of *Tubificida* (Class Oligochaete), (within the impact area)

One species of Syncarida (*nr Atrpobathynella* sp. B19), (within and outside the impact area)

One species of *Copepoda (Parastenocaris* sp. B30), (within the impact area).

The *Syncarida* and *Copepoda* are new undescribed crustacean species based on morphological differences. The new species of *Syncarida (nr Atopobathynella* sp. B19) has been inferred to have wider distribution, through hydraulic connection within the aquifer system. The new *Copepoda* species (*Parastenocaris* sp. B30) is a widespread species known from a number of locations across Western Australia including West Kimberley, East Kimberley, Pilbara, Murchison and north-eastern Goldfields as well as 20 kilometres north-west of Laverton at the Windarra Nickel Project. The *Tubificida* worm (*Enchytraeidae* sp.) is regarded as being widespread having been recorded in the Pilbara and South Coast bioregions as well as 20 kilometres north-west of Laverton at the Windarra Nickel Project and within the northern Goldfields at Gidgee.

No troglofauna were recorded due to the absence of suitable habitat around the Project and this is also likely within the Yeo Palaeochannel.

Stygofauna which were recorded within the Yeo Palaeochannel 25 kilometres west of the Project comprised of a species rich copepod-dominated community, which is generally typical of surveys in the Yilgarn calcretes. None of these stygofauna species were recorded at the mining Project area. This is not unexpected as the hydrogeology of the Gruyere orebody is likely to be disconnected and too distant from the Yeo Palaeochannel and associated calcrete aquifer. A summary of the stygofauna survey work undertaken within the Yeo Palaeochannel is presented below in the section titled *Subterranean Fauna*.

Cultural Heritage

A search of the Heritage Council's State Heritage register and the Department of Aboriginal Affairs' (**DAA**) register was undertaken to identify any heritage sites within a radius of 20 kilometres of the Project. No registered aboriginal heritage sites are located within the mining lease M38/1267. There are six Registered Heritage sites occurring within the wider tenement. A search of the EPBC Act – Protected Matters Database was also undertaken to determine the presence of any Registers of the National Estate (**RNE**) listed under the *Australian Heritage Council Act 2003* within the area. The search identified one RNE, named Pildpirl Protected Area, located within an eight kilometres radius of the central point which was listed for its mythological and ceremonial site values in November 1979.

Ethnographic cultural mapping surveys were completed in September and November 2015 by expert anthropologists across the greater Project region, taking in the likely footprint including infrastructure corridors.

Traditional Owners participated in these surveys as well as 'Senior Men' who are the responsible cultural custodians of the region.

Archaeological surveys are planned to commence from August to November, 2016 for the mining area, related infrastructure corridors and the access roads. The Yeo Borefield area will be surveyed in February, 2017.

Baseline Surveys of the Gas Pipeline Route

Flora and Vegetation

A Level 1 flora and vegetation desktop and field survey was conducted over the spring 2015 season of the gas pipeline route from a point west of the EGP Main Line Valve 1 near Granny Smith Gold Mine to the north-east towards Laverton and follows the White Cliff public road reserve eastward to the Project (Reference 10).

Fifty vegetation communities were identified within the White Cliffs Road survey area. These communities comprised eight different landform types and seven major vegetation groups. These communities were represented by a total of 54 Families, 132 Genera and 310 Taxa (including sub-species and variants).

Since this survey, the gas pipeline alignment has been modified south of Laverton to avoid impacted underlying tenement holders. The alignment now trends in a south-westerly direction from Laverton to join the EGP some 15 kilometres west of Main Line Valve 1. This new alignment will be surveyed in Q3 2016 once the alignment has been agreed by all underlying tenement holders. Given the consistency and uniformity of vegetation assemblages south of Laverton, no additional species are expected to be found as a result of this re-alignment.

No Threatened Taxa, pursuant to Sub-section (2) of Section 23F of the WC Act and the EPBC Act were identified within the survey area. Three Priority Flora Taxa as listed by DPaW were identified within the survey area.

None of the vegetation communities within the survey area were found to have National Environmental Significance as defined by the EPBC Act. No Threatened Ecological Communities (**TEC**) pursuant to Commonwealth or State legislation were recorded within the survey area. The re-aligned section south-west of Laverton intersects a Priority Ecological Community (**PEC**); "*Mount Jumbo Range vegetation complex (banded ironstone formation)*" (Priority level 3). Development within the proposed pipeline corridor is not expected to have a significant impact on this PEC given the strategy that pipeline installation will occur off the banded ironstone formation ridge and keep within the PEC buffer zone.

The survey area is not located within an Environmentally Sensitive Area (**ESA**) listed under the *Environmental Protection (Clearing of Native Vegetation) Regulations 2004*. The White Cliffs Road survey area intersects two Schedule 1 Areas; one centred on the abandoned Mount Morgans Gold Mine and a section of the Old Laverton Road extending south-west of Mount Morgans; the second is centred on Laverton town site. The survey area is not located within a listed or proposed conservation area managed by DPaW.

Ten introduced Taxa were identified within the gas pipeline survey area. According to the Department of Agriculture and Food of Western Australia one of these taxon is listed as a Declared Plant under Section 22 of the *Biosecurity and Agriculture Management Act 2007*.

Vertebrate Fauna

As with the flora and vegetation surveys, Gold Road also commissioned a Level 1 fauna desktop and field survey over the spring 2015 season of the gas pipeline route (Reference 11). A total of 48 native fauna species were recorded within the survey area. Observations of three introduced species using the survey area were also gathered.

As with Flora and Vegetation in above, the new pipeline alignment south-west of Laverton will need to be surveyed.

A review of the EPBC Act threatened fauna list, DPaW's Threatened Fauna Database and Priority List, unpublished reports and scientific publications identified 27 specially protected, migratory or priority fauna species as having been previously recorded or as being potentially present in the general vicinity of the survey area. No vertebrate fauna species of conservation significance (listed under State or Federal threatened/migratory species lists or as a DPaW priority species) were positively identified as utilising the study area during the survey period. The current status on site and/or in the general area of those species of conservation significance considered likely to occur is difficult to determine. However, based on the habitats present and, in some cases, recent nearby records, eight species can be regarded as possibly utilising the survey area for some purpose at times.

Given the relatively narrow and linear nature of the gas pipeline route, the fact that the pipeline will be buried and the presence of large areas of similar habitat in adjoining areas, impacts on fauna and fauna habitat at any one point are anticipated to be small or negligible and therefore manageable.

Cultural Heritage

Once the Miscellaneous Licence is granted for the gas pipeline alignment, a search of the Heritage Council's State Heritage register and the DAA register will be undertaken to identify any heritage sites within five kilometres of the gas pipeline corridor. A search of the EPBC Act – Protected Matters Database will also be undertaken to determine the presence of any RNE listed under the *Australian Heritage Council Act 2003* within the same area. Archaeological surveys will also be undertaken by expert archaeologists along the alignment of the gas pipeline route toward the end of 2016.

Baseline Surveys of the Water Supply Routes and Borefields

Flora and Vegetation

A Level 1 flora and vegetation desktop and field survey was conducted over the spring and autumn 2015 and 2016 seasons of the Yeo Palaeochannel and Anne Beadell Borefields or Gruyere Borefields and associated water supply routes, (Reference 12).

The Gruyere Borefields survey areas comprise of forty-three broad vegetation communities. No Threatened Flora Taxa, pursuant to Sub-section (2) of Section 23F of the WC Act and the Commonwealth EPBC Act were identified within the survey areas. No Priority Flora Taxa as listed by DPaW were identified within the survey areas.

None of the vegetation communities within the survey areas was found to have National Environmental Significance as defined by the Commonwealth EPBC Act. No TEC pursuant to Commonwealth or State legislation were recorded within the survey areas. The survey areas are not located within an ESA as listed under the EP Act, or Schedule 1 Areas. The survey areas are not located within a listed or proposed conservation area managed by DPaW. However, the Yeo Lake Nature Reserve, which is listed as a “Class A” Nature Reserve managed by DPaW, is located approximately 700 metres east of the north-eastern most extent of the Anne Beadell Borefield survey area.

One introduced taxon was identified within the Gruyere Borefields survey areas. According to the Department of Food and Agriculture, WA, it is not listed as a Declared Plant under Section 22 of the *Biosecurity and Agriculture Management Act*.

Vertebrate Fauna

As with the flora and vegetation surveys, Gold Road also commissioned a Level 1 fauna desktop and field survey over the spring and autumn 2015 and 2016 seasons of the water supply routes and Gruyere Borefields (Reference 6). Records indicate that 28 mammals (including eight bat species), 103 bird, 105 reptile and nine frog species have previously been recorded in the general area. Opportunistic fauna observations during the surveys identified a total of 56 native fauna species and four introduced species.

A review of the EPBC Act threatened fauna list, DPaW’s Threatened Fauna Database and Priority List, unpublished reports and scientific publications identified a number of specially protected, migratory or priority fauna species as having been previously recorded or as being potentially present in the general vicinity of the survey area. Two vertebrate fauna species of conservation significance (listed under State or Federal threatened/migratory species lists or as a DPaW priority species) were positively identified as utilising the study area for some purpose during the survey period. An additional six species of conservation significance can be regarded as possibly utilising the survey area for some purpose at times. The habitat within the survey areas while considered possibly suitable, may be marginal in extent/quality and the species listed may therefore only visit the area for short periods or as rare/uncommon vagrants. It was concluded that no terrestrial invertebrate or vertebrate fauna species of significance would be significantly impacted on by installation and operation of the Gruyere Borefields. Given the relatively small size of the impact footprint at any one point, the linear nature of pipeline routes and the extensive habitat connectivity with adjoining areas, impacts on fauna and fauna habitat at any one point are anticipated to be small/negligible and therefore manageable.

Subterranean Fauna

Level 1 and 2 subterranean fauna desktop and field surveys were commenced in 2012 and more recently in 2015 and 2016 of the Gruyere Borefields (Reference 13).

Any risk to troglofauna in the Gruyere Borefields was considered to be low to negligible given that no excavation of habitat would occur in either of the two borefields. As such, no further sampling was conducted. It was considered unlikely that the Project would reduce the amount of habitat suitable for troglofauna. Drawdown of the water table is unlikely to lead to significant reduction in above water table humidity and in fact, drawdown may increase the amount of troglofauna habitat available.

Stygofauna however may be impacted from drawdown and as such, were subject to extensive survey effort both within and outside the drawdown area over several sampling periods to gain an understanding of the distribution of species.

The updated report from additional sampling undertaken from May-July 2016 will be finalised in Q3 2016. Indications at the time of writing are that several species are likely to be restricted to the Yeo Palaeochannel, although species' ranges are expected to extend beyond the area sampled in the surveys. Few species were collected from the Anne Beadell Borefield area.

On the basis of the EPA Environmental Assessment Guideline No. 8: Environmental Principles, Factors and Objectives (EPA 2015c), the EPA identified that stygofauna in the Yeo Palaeochannel is the key environmental factor for the Project. This determination is significant because groundwater abstraction may alter stygofauna habitat and as such, provided guidance to Gold Road in the level of detail required to be addressed in the Part IV approval process.

Cultural Heritage

Once the Miscellaneous Licences are granted for the Gruyere Borefields and water supply routes, a search of the Heritage Council's State Heritage register and the DAA register will be undertaken to identify any heritage sites within five kilometres of these corridors. A search of the EPBC Act – Protected Matters Database will also be undertaken to determine the presence of any RNE listed under the *Australian Heritage Council Act 2003* within the same area. Archaeological surveys will also be undertaken by expert archaeologists along the alignment of the pipeline routes toward the end of 2016.

Waste Management

The Project is expected to generate the typical gold mining process wastes such as waste rock and tailings as well as construction and maintenance wastes, general refuse, liquid effluent, chemical and hydrocarbon wastes.

Waste rock will be placed in the waste rock dumps detailed in Section 16. Process plant tailings will be placed in the TSF detailed in Section 17. All non-mining and non-processing waste streams associated with the Project that need to be managed have been identified. Where possible, a calculation of the anticipated waste production rate has been completed. Waste management and treatment (or handling) objectives and the approach to be adopted have been developed including the associated decommissioning activities required at mine closure.

TSF Groundwater Monitoring

As part of the TSF operation there is planned to be a monitoring system which will comprise of a network of approximately 10 groundwater monitoring bores located around the TSF. The locations are to be determined based on the advice of the Project Hydrogeologists. It is planned that the monitoring bores will each have trigger levels assigned (for groundwater quality and levels) to initiate the future installation of recovery bores.

20.2 Permitting

Statutory Approvals

Statutory approvals for the construction and operational phases of the Project are required under State and Commonwealth legislation. Table 20-1 provides a register of the State and Commonwealth legislation which applies to the Project.

Table 20-1: Applicable Environmental and Heritage Permitting Legislation

Legislation	Agency	Regulates	Project Component
Commonwealth Legislations			
Environment Protection and Biodiversity Conservation Act 1999	Department of the Environment	Matters of national environmental significance – relevant category rare flora and fauna	All
Native Title Act 1993	National Native Title Tribunal	Native Title/Tenure	All
Civil Aviation Act 1988, Civil Aviation Safety Regulations 1998	Civil Aviation Safety Authority	Safety of civil aviation, with particular emphasis on preventing aviation accidents and incidents	Airstrip
Australian Jobs Act 2013	Australian Industry Participation Authority	Major projects with capital expenditure of $500 million or more to increase Australian industry participation	All - excluding BOO power contract
State Legislation (Primary Statutory Approvals)			
Mining Act 1978	Department of Mines and Petroleum	Land access and tenure	All
		Environmental assessment and management (Mining Proposal)	All
		Petroleum pipeline licence, consent to construct and operate	Gas pipeline
Environmental Protection Act 1986	Office of the Environmental Protection Authority	Environmental impact assessment and management	All
Aboriginal Heritage Act 1972	Department of Aboriginal Affairs	Aboriginal archaeological and ethnographic heritage	All
State Legislation (Secondary Statutory Approvals)			
Agriculture and Related Resources Protection Act 1976	Department of Agriculture and Food	Weeds and feral pest animals	All
Building Regulations 1989	Shire of Laverton	Building Licences	Accommodation village, administration buildings
Bushfires Act 1954	Bush Fire Service	Wild fire control	All
Conservation and Land Management Act 1984	Department of Parks and Wildlife	Flora and fauna, habitat, weeds, pests, diseases	All
Dangerous Goods Safety Act 2004	Department of Mines and Petroleum	Transport and management of explosives and dangerous goods including gases.	Explosives storage facility

Legislation	Agency	Regulates	Project Component
Electricity Act 1945	Department of Finance (Public Utilities Office)	Control of electricity generating stations and the transmission, distribution and use of electricity	Power station and transmission lines
Environmental Protection Act 1986 Environmental Protection Regulations 1987	Department of Environmental Regulation	Part V Works Approvals and Licences	Process plant, power station, TSF, waste water treatment plants, landfill
Environmental Protection (Clearing of Native Vegetation) Regulations 2004	Department of Mines and Petroleum	Clearing of native vegetation on Mining Act tenure (that is not covered by Part IV approvals)	Gas pipeline
Health Act 1911	Department of Health	Human health impacts and management	Accommodation village, waste water treatment plants
Local Government Act 1955 and Local Government (Miscellaneous Provisions) Act 1960	Shire of Laverton	Building licenses	Accommodation village, waste water treatment plant, use of public roads
Main Roads Act 1930	Main Roads Western Australia	Safety on public roads, including access and egress from private to public roads	Use of main roads and intersections with main roads
Mines Inspection and Safety Act 1994	Department of Mines and Petroleum	Worker safety on mine sites (Project Management Plan approval required)	Operations phase of all project components
Occupational Safety and Health Act 1984	Department of Commerce	Worker safety on construction sites (Construction Safety Management Plan required, but not subject to WorkSafe approval)	Construction of the process plant, accommodation village, mine buildings and power lines
Rights in Water and Irrigation Act 1914	Department of Water	Access to and use of water resources	All
Soil and Land Conservation Act 1945	Department of Agriculture and Food	Protection of soil resources	All
Waterways Conservation Act 1976	Department of Water	Protection of surface and groundwater	All
Wildlife Conservation Act 1950	Department of Parks and Wildlife	Protection of native wildlife	All
Australian Pipeline Industry Association Code of Practice Onshore Pipelines 2009	Australian Pipeline Industry Association Ltd	This Code is intended to encapsulate the best techniques and methods presently available to mitigate or to eliminate the environmental impacts of activities	Gas pipeline

Legislation	Agency	Regulates	Project Component
AS2885.1-2007 Pipelines – Gas and Liquid Petroleum – Design and Construction	Standards Australia	Pipeline design and construction methods and management	Gas pipeline
AS2885.2-2007 Pipelines – Gas and Liquid Petroleum – Welding	Standards Australia	Pipeline welding methods and management	Gas pipeline
AS2885.3-2001 Pipelines – Gas and Liquid Petroleum – Operation and Maintenance	Standards Australia	Operations and maintenance of gas pipelines	Gas pipeline
AS2885.3-2001 Pipelines – Gas and Liquid Petroleum – Operation and Maintenance – Field Pressure Testing	Standards Australia	Field pressure testing of pipelines	Gas pipeline

The approvals strategy for the Project was based upon selecting the applicable Mining Act tenure for the various mining and mining-related activities. Most of the infrastructure required to support the mining operation is on the granted Mining Lease, however, the access roads, gas pipeline and water supply pipelines are within Miscellaneous Licence infrastructure corridors. The airstrip and accommodation village locations are within a Miscellaneous Licence off the Mining Lease.

The Project is made up of two discrete aspects:

- Mining, Process Plant and associated Infrastructure (inclusive of borefields and access roads)
- Gas supply pipeline.

By separating these two aspects, a delay in the approval of a component of one aspect will not cause a delay in the approval of the other aspect.

State Environmental Approvals

Gold Road has commenced the formal environmental assessment of the Project and identified that the development approvals pathway will be in accordance with Part IV of the EP Act, in addition to a Mining Proposal under the *Mining Act 1978*. The mining and water supply aspects of the Project were referred to the OEPA on 3 March 2016 and it was deemed that formal environmental assessment was required at an *"Assessment on Proponent Information, Category A"* level of assessment.

The management and protection of stygofauna was the key environmental factor identified by OEPA that required formal impact assessment. Gold Road is progressing stygofauna studies and approval processes to manage and mitigate the risks to stygofauna that have been identified in the Yeo Borefield area of the Project. All other baseline environmental surveys of flora, vegetation, vertebrate fauna and SRE invertebrates have been completed, as discussed earlier, with no significant species identified that would be impacted by the Project in a manner that could not be managed.

The management and protection of stygofauna that have been identified in the Yeo Borefield area of the Project is the key environmental factor identified by OEPA that requires formal impact assessment. Gold Road completed the stygofauna studies and API-A approval document was submitted to the EPA for assessment on 4 October 2016.

A work program for the remainder of 2016 has been developed to complete all remaining archaeological surveys and development of mitigation plans. Final Project EPA Part IV approval is anticipated to be received by January 2017.

The gas pipeline aspect of the Project was referred separately to the OEPA which resulted in a decision of *"Not Assessed - No Advice Given"* being given on 18 July 2016. The OEPA recommended that environmental approvals of the gas pipeline project via a Native Vegetation Clearing Permit and Mining Proposal be managed through the DMP approvals process. As the gas pipeline aspect is being contracted out to a third party on a BOO model, this third party will be responsible for obtaining the DMP approvals. The BOO contractor will be responsible for obtaining environmental approval under a Part V Vegetation Clearing Permit and will also submit applications to the DMP for the Consent to Construct Approval (Petroleum) and Consent to Operate Approval (Petroleum).

Commonwealth Environmental Approvals

The EPBC Act is the Australian Government's central piece of environmental legislation that provides a legal framework to protect and manage nationally and internationally important flora, fauna, ecological communities and heritage places defined in the EPBC Act as matters of national environmental significance. The Act is administered by the Department of the Environment. Approvals are required from the Commonwealth if projects are likely to impact on matters of national environmental significance.

With regard to nationally threatened species, the Level 1 and 2 fauna surveys undertaken across all Project footprints specifically targeted nationally listed species over multiple seasons. No sightings were made or other evidence identified of these potentially occurring species despite the survey effort. Gold Road has self-assessed the potential risks to matters of national environmental significance and believe there are no potential impacts or triggers to warrant Referral and the Project is not considered to be a 'Controlled Action' under the EPBC Act.

State Mining and Secondary Approvals

The DMP is the lead agency for mining approvals and is the agency responsible for administering the *Mining Act 1978* (Western Australia). With EPA assessing the significant environmental impact components of the Project, DMP will still need to assess the Project under Section 70 O(1) of the *Mining Act 1978* (Western Australia) which requires a Mining Proposal approval. A Mining Proposal will be prepared and will contain detailed information on identification, evaluation and management of environmental impacts relevant to the Project and the surrounding environment.

Secondary State Government environmental approvals are also required for the construction and operation of the Project and includes Works Approvals and Environmental Licences (under Part V of the EP Act).

The Project accommodation facility will be constructed on Crown Land covered by the Yamarna Pastoral Lease. Typically, a Development Application/ Planning Consent and Building Permit Application would still need to be made to the local Shire however, within the Shire of Laverton, the Shire's jurisdiction in this instance does not extend beyond the town limits. Therefore, none of these types of applications are required for the Project.

Commonwealth Native Title Approvals

All tenure required for the Project is subject to the *Native Title Act 1993*. The tenure required is subject to either the Section 29 *Native Title Act 1993* 'right to negotiate' process or the Section 24MD *Native Title Act 1993* 'infrastructure process' which gives a right to be consulted.

Native Title and Aboriginal heritage aspects of the Project area have been addressed by the Company by working with the Yilka People resulting in the GCBNTA being signed on 3 May 2016 and the subsequent Mining Lease, M38/1267 being granted on 9 May 2016.

However, on 29 June 2016, native title was jointly determined with respect to the Yilka applicant (a registered native title claimant group) and the Sullivan/Edwards applicant (an unregistered native title claimant group) over the Cosmo Newberry Aboriginal Reserves and Yamarna Pastoral Lease on which the Project lies (*Murray on behalf of the Yilka Native Title Claimants v State of Western Australia (No. 5)* [2016] FCA 752).

The Court's judgment will require the Yilka and the Sullivan/Edwards groups to reach an intra-indigenous agreement regarding how they work together in the future and how they share the rights and benefits of the GCBNTA, noting that the GCBNTA requires the Yilka People to procure the execution by the registered native title holder(s) of a deed of assumption (in a form and substance acceptable to Gold Road) in which the registered native title holder(s) agrees to be bound by the GCBNTA. If the parties cannot agree, or there are delays in the parties reaching a resolution, it may impact on the grant of miscellaneous licences applications for the Project.

As at 31 August 2016, the final form of the native title determination between the Yilka (the registered native title claim group) and Sullivan/Edwards (an unregistered native title claim group) had not been settled by the Federal Court. Until the final form determination is made by the Federal Court, Gold Road is unable to ascertain the effect of the judgment, if any, on the Company or its Native Title Agreement with the Yilka and any potential impact on the Project.

The Mining Lease has been granted and all necessary native title approvals have been obtained.

Gas Pipeline Corridor Approvals

The proposed power source for the Project is an on-site, gas fired power station with emergency dual fuel (diesel/gas) capability. Gold Road is planning to deliver gas to the site via a gas pipeline from the EGP. The corridor route is from a point on the EGP south-west of Laverton, along the White Cliffs Road reserve through to Gruyere. A Miscellaneous Licence for this gas pipeline alignment was pegged and is currently being negotiated with underlying tenement holders. It is anticipated that the grant of tenure will be in early Q1 2017.

Once the Miscellaneous Licence is granted, Gold Road will provide access to the BOO contractor for the gas pipeline and power station who will obtain environmental approval under a Part V Vegetation Clearing Permit. The BOO contractor will also submit applications to the DMP for the Consent to Construct Approval (Petroleum) and Consent to Operate Approval (Petroleum).

Project Water Licences

Gold Road currently has two 5C groundwater licences issued by the Department of Water (**DoW**) in the Great Victoria Desert Sub-area of the Goldfields groundwater management area. Ground Water Licence (**GWL**) 176189 allows Gold Road to abstract 600,000 kL per year from the palaeochannel aquifer, while GWL 177087 allows abstraction of 600,000 kL per year from the fractured rock aquifer.

Gold Road has applied to the DoW to increase the allocation limit on the palaeochannel licence GWL 176189 to abstract up to 7.8 GL per year from the Yeo and Anne Beadell Borefields (which allows for some contingency); and to increase the allocation on the fractured rock licence, GWL 177087, to 800,000 kL per year to cover the Project's mine dewatering and construction water supplies. The DoW has advised that the hydrogeological studies submitted in support of the Project "*indicate that the required amount of water can be abstracted with acceptable impacts on the groundwater resource and other users*" (DoW letter dated 14 March 2016). The issuing of the Project water licence is currently pending the outcome of the environmental assessment by the EPA.

Closure and Rehabilitation

Mine Closure Plans are required by DMP for all new Mining Proposal applications and must be prepared in accordance with the Guidelines for Preparing Mine Closure Plans (DMP and EPA, 2015). This requirement is stipulated as a tenement condition under the relevant provisions of the *Mining Act 1978* (including Section 84).

A Conceptual Mine Closure Plan has been submitted as part of the impact assessment documentation and Mining Proposal and is expected to be approved by late 2016. The Regulators (DMP and EPA) accept that not all the necessary detail for final closure will be available in this early stage of the Project, however they must be able to understand the issues that require management at closure and have confidence that all relevant issues have been identified and appropriately planned for and managed.

DMP and EPA define the Project as a longer-term project (i.e. more than ten years). Consequently, less detailed information on the final closure may be required at the Project approval stage due to the longer time before planned closure.

Central to this understanding is a progressive rehabilitation strategy which ensures that the post-mining landscape is safe and stable, that the quality of surrounding water resources is protected, that the agreed post-mining land use is established and that agreed success criteria are monitored and reported to stakeholders.

Gold Road has estimated a preliminary mine site closure costs at A$54 million.

20.3 Social and Community Impact

Community Consultation

A stakeholder consultation programme has been in place since 2009, ensuring that the relevant regulatory authorities and Traditional Owners continue to be consulted in relation to the Project.

The Traditional Owners have been actively involved in baseline flora and fauna surveys for the Project. Representatives of the Yilka People participated in the Level 2 Spring flora survey, the Level 2 Autumn fauna survey and more recently, the Level 2 stygofauna survey. During the surveys, their interests have related to exclusion zones, rocky breakaway vegetation communities and the conservation of species of cultural importance. In addition, specific consultation with Yilka has occurred throughout 2015 during the Native Title negotiations in which Gold Road met with Yilka representatives each month and concluded with the GCBNTA being signed on 3 May 2016. This Agreement takes into consideration Yilka's concerns and requirements around potential impacts to their traditional native title rights and access to lands.

Gold Road has provided Project briefing presentations and held discussions with the relevant Government departments including DER, DPaW, DMP and OEPA. Pre-referral meetings were held with the EPA in August 2015 (Mine and Borefield Project) and again in February 2016 (Gas Pipeline Project). Feedback was used and factored into the Project approvals strategy and licencing requirements.

Gold Road will continue to engage with relevant stakeholders on matters associated with the Project to ensure stakeholder concerns are addressed. Any potential issues or impacts will be updated in the Project Risk Register and managed through implementation of the selected management and mitigation measures. Some of the main topics of discussion and outcomes of consultations with stakeholders are summarised below:

- Meetings with DMP to discuss the Project, approvals issues and timing
- Meetings with the EPA to obtain an appropriate Level of Assessment decision on the referrals
- Meetings with Yilka People and CNAC to provide Project updates, obtain consents, discuss employment and contracting opportunities, communication and consultation, management measures and compensation and financial benefits
- Meetings with the Shire of Laverton to provide updates on the Project and seek advice on local government approvals required and timing.

Local Communities

Laverton has a population of approximately 1,227 residents of which 417 people permanently reside in the township (2011 consensus). Laverton was established from the success of the Craiggiemore gold mine in 1897. The town site was surveyed in July 1899 with residential and business areas developed and the town of Laverton was finally gazetted in July 1900.

Cosmo Newberry, locally referred to as Cosmo, is a small Australian Indigenous community with a population of 71 (2011 census), located approximately 80 kilometres north-west of the Project. The community is managed through its incorporated body, CNAC, incorporated under the *Aboriginal Councils and Associations Act 1976* in 1991. In 1994 the community made the decision to become affiliated with Ngaanyatjarra Council.

Community and Indigenous Engagement

Key social issues identified during Project design included:

- A reduced contribution to local infrastructure and services by the Project due to the adoption of a Fly In Fly Out (**FIFO**) workforce. Local government preference is for a residential workforce based in Laverton but the commute distance is too great for daily drive-in-drive-out to operate
- A restriction to land access due to Project operations and the resultant impact on indigenous heritage values
- An increase to traffic in the region due to transportation of materials, consumables and equipment to the Project.

Gold Road has addressed indigenous impact issues in the GCBNTA and has developed a Community and Indigenous Engagement Strategy. This strategy is driven by the Company's Diversity in Engagement and Employment Policy. These address the Company's requirements for appropriate equal employment opportunities and anti-discrimination, as well as employment and contracting opportunities for local businesses and individuals.

In addition to mitigating the identified impacts, opportunities were identified during the FS which were included in the Community and Indigenous Engagement Strategy. These opportunities are:

- Local employment and procurement of services
- Contribution to the local economy through rates, taxes, charges and community investment by the Company. Gold Road is already providing vital community support services such as emergency response and air access for the Royal Flying Doctor Service (**RFDS**). These capabilities will increase as infrastructure such as the sealed all weather airstrip is built at Gruyere and specialist staff such as paramedics are recruited into operational roles.

Gold Road plans to minimise any potential social issues or impacts resulting from the Project's development and promote the identified opportunities during the construction and operations phases. Gold Road believes that implementing these opportunities will result in an overall positive change to the demographics and population statistics of the region.

21 CAPITAL AND OPERATING COSTS

21.1 Project Execution

The Project development and execution will be managed by the Owner's team appropriately resourced to oversee the execution of the design, construction, commissioning and handover to operations. An OR Plan, as part of the WBF, has been developed to ensure that Gold Road will have all the systems, standards and procedures in place and an operations team recruited, trained and ready to accept care, custody and control of the Project assets when handed over by the development team.

The Project Execution Schedule (Figure 21-1) is based on a five-month early works programme followed immediately by a 24 month construction and commissioning timeframe with the objective of achieving first gold production by Q4 2018. The Project Execution Strategy is based on Project Finance in place and Project Approval by Q1 2017.

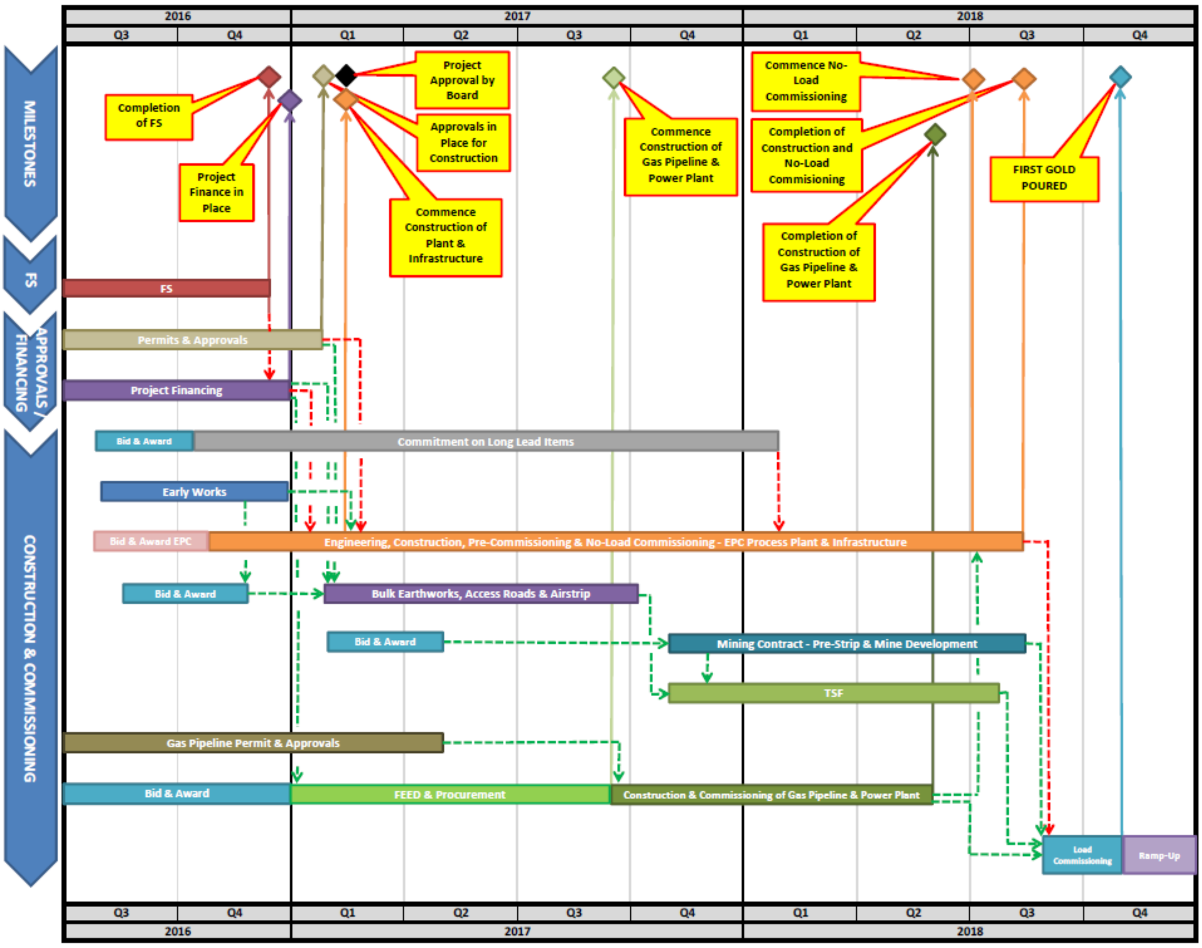

Figure 21-1: *High-level Project Execution Schedule*

The Contracting Strategy was developed to support the Project Execution Strategy which is based around an EPC contract model that delivers the design, engineering, construction and commissioning of the process plant and associated infrastructure. The Contracting Strategy also aims to minimise the number of interfaces between contractors on the Project site. The contract model requires an Owner's team to manage the execution of the Project. A Contracts Responsibility Matrix has been developed showing the internal ownership of each contract development and award process, the management and administration and the transitioning of the contracts into operations.

The Contracting Strategy and Bidders' List were developed through a rigorous process commencing with workshops involving the Project team followed by expressions of interests, pre-qualifications, clarifications and shortlisting of potential bidders to the Project.

The total contract packages identified as part of the Project execution strategy are as follow:

- Major contracts – seven packages
- General packages – approximately 30 consultancies and 20 site services contracts consisting of services required during construction and transitioning to operations. This list also includes services contracts specific for operations requirement only but will be developed and awarded as part of the OR stream of work.

Table 21-1 lists the seven major contracts identified and the type of contract. Figure 21- shows the Major Contracts Matrix.

Table 21-1: Major Contracts and Types

Contract Number	Contract Description	Contract Type
1000-EP-GOR1101	Mine Development and Production	Schedule of Rates
1000-BO-GOR1700	Energy Supply (Power Station and gas pipeline)	Build Own Operate
1000-EP-GOR1100	EPC Process Plant and associated Infrastructure	Fixed Lump Sum
1000-CC-GOR1301	Bulk Earthworks, TSF, access roads and airstrip	Schedule of Rates
1000-DS-GOR1600	Accommodation village supply and construct	Fixed Lump Sum
1000-DS-GOR1601	Communications backbone to site	Fixed Lump Sum
1000-CC-GOR1300	Water bore drilling.	Schedule of Rates

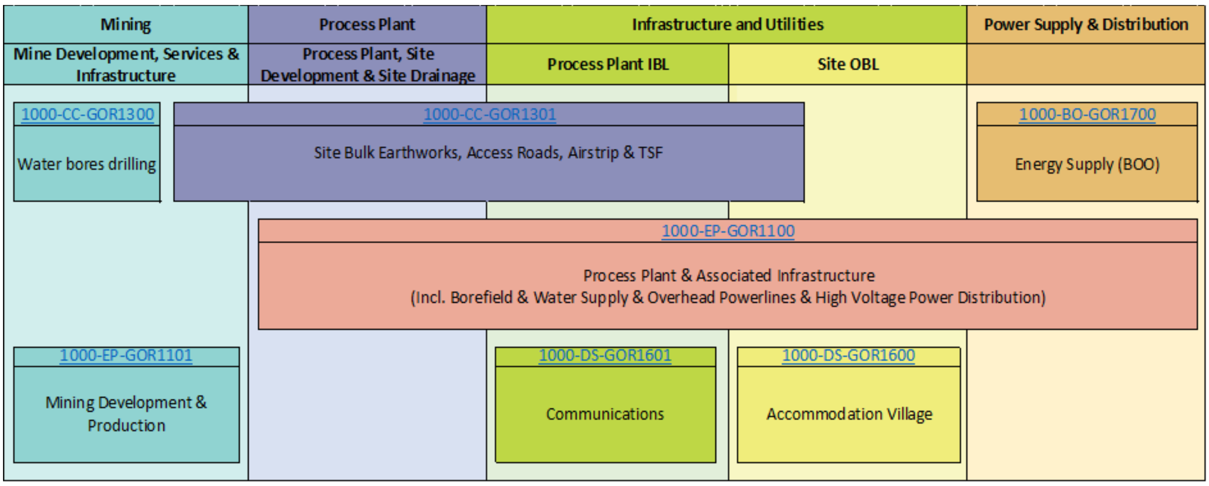

***Figure 21-2:** Major Contracts Matrix*

21.2 Capital Costs

The capital cost estimate represents costs for the overall Project development. The estimate includes direct costs for the open pit mine pre-strip and mine development, the process plant, the non-process infrastructure (NPI) and, indirect costs associated with the contractors, Owner's team and pre-production operations. The capital costs include allowances for contingency and estimated growth. The capital costs associated with the gas-fired power station and gas delivery pipeline are not included in the estimate as these are provided under a BOO contract and are captured in the power unit cost used in the operating cost estimates. Similarly, the capital cost estimate does not include the cost of the mining mobile equipment fleet as this will be incorporated in the mining contract rates.

The cost estimate has been developed with input primarily from GRES, AMC and the Owner's team. Axiom completed a peer review of the non-mining capital cost estimate, (Reference 14). Broadleaf completed a capital cost and schedule risk analysis to determine the capital contingency over a range of probabilistic outcomes, (Reference 15). The capital cost basis of estimate has been developed from the preliminary Project Execution Strategy included in the Project Execution Plan (PEP). The Project Execution Strategy is based on a 24 month construction and commissioning timeframe, beginning in Q1 2017, with completion of commissioning and ramp-up by Q4 2018. An early works program including commitment of long lead time equipment and early engineering will be required during Q3 and Q4 2016.

The estimate is based upon preliminary engineering, quantity take-offs, tendered price quotations for mills, crushers and accommodation village and budget price tendered quotations for major equipment and bulk commodities. Unit rates for installation were based on market enquiries specific to the Project and benchmarked to those achieved recently on similar projects undertaken in the Australian minerals processing industry. The estimate includes an allowance for Project contingency based on a P80 outcome (80% certainty of achieving the estimated capital cost).

The Sustaining Capital Expenditure (**Susex**) estimate represents cost expended to sustain and/ or maintain the capital assets to perform to the Project design criteria during the LOM. The estimate includes all costs for the pit expansion, ongoing mine rehabilitation, the process plant and infrastructure capital maintenance, and the TSF wall lifts.

The accuracy of the estimate is -10% to +15% as per recommended practice No. 47R-11 for process industries set out by the Association for the Advancement of Cost Engineering (**AACE**) - Cost Estimate Classification guidelines for Class 3 estimates.

Capital Cost Estimate Summary

The total estimated cost to design, procure, construct and commission the Project scope inclusive of an open pit mine development, process plant and supporting infrastructure, Owner's team, OR and pre-production costs is A$507M in Q2 2016 (estimate Base Date) terms. The forecast capital cost including potential escalation to Project completion (Q4 2018) is estimated to be A$514M.

The capital cost estimate includes:

- Direct costs of the Project development
- Indirect costs associated with the design, construction and commissioning of the new facilities
- Owner's cost associated with the management of the Project from design, engineering, construction up to the handover to operations and Project close-out
- Insurance, operating spares and first fills
- Costs associated with OR and pre-production operations
- Growth allowance on quantity, pricing and unit rate variance
- Contingency on Project scope definition and risks.

Total Project capital expenditure by major area, and quarterly and annual expenditure are summarised in Tables 21-2 and 21-3 respectively. No allowances have been made for interest payments, financing costs and foreign exchange rate variations during construction. Figure 21-3 summarises the real monthly expenditures histogram and cumulative capital expenditures.

Table 21-2: Direct and Indirect Cost Summary

Area	A$M
Direct	
Process Plant & Infrastructure & TSF	178
Infrastructure and Utilities - Site General	79
Mine Development	36
Power Supply and Distribution	20
Site Development and Site Drainage	8
Subtotal Direct	321
Indirect	
Engineering and Contractors	86
Project Owner's team & Pre-production Operations	50
Capital, Operating and Commissioning Spares	7
Subtotal Indirect	143
Contingency	43
Total (Real) Capital Cost	507

Table 21-3: Expenditure Schedule for Total Project Development (Q2 2016 Estimate Base Date Cost)

	2016		2017				2018			
A$M	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Direct Cost	1	12	42	61	82	65	36	13	8	1
Indirect Cost	-	2	11	18	32	33	20	12	13	3
Contingency	-	1	5	7	10	9	5	2	2	-
Total (Quarterly)	2	15	59	85	125	107	61	27	22	4
Total (Annual)	17		376				114			
Total (Cumulative)	2	17	76	162	286	393	454	481	503	507

Note: *Apparent differences may occur due to rounding*

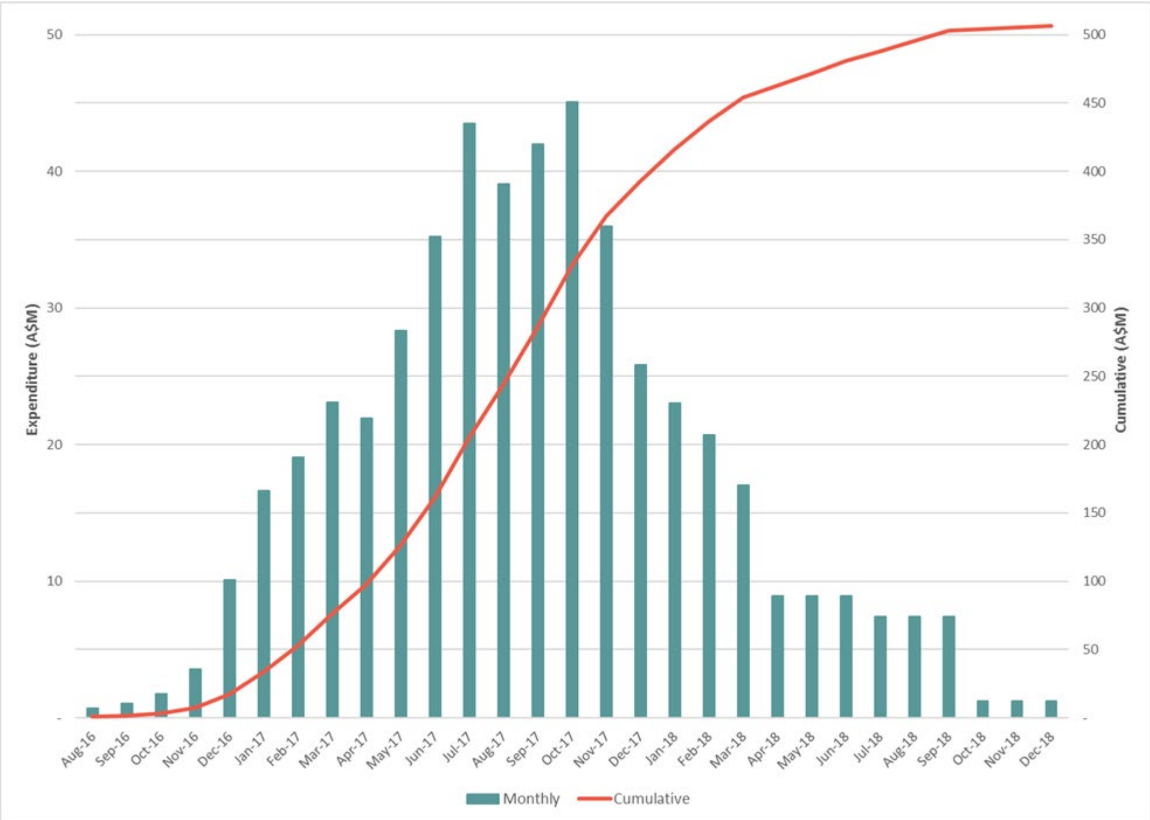

Figure 21-3: *Monthly and Cumulative Expenditure Profile (Q2 2016 Estimate Base Date Cost)*

Sustaining Capital Cost Estimate Summary

The FS Susex estimate is currently A$76.7M, a decrease of A$64.4M from the PFS. Table 21-4 is a summary of the variance between the PFS and FS which largely relate to reclassification of Susex to either Capex or Opex. Table 21-5 details the reasons for changes from PFS to FS. Table 21-6 summarises the annual expenditure schedule. Figure 21-4 shows the sustaining cost variance from PFS to FS as a waterfall graph.

Table 21-4: Summary of Total Sustaining Capital Cost by Major Area PFS vs FS

Area	PFS Total LOM Cost (A$M)	FS Total LOM Cost (A$M)	Variance (A$M)
Mine Development	80	31	-49
Processing and Infrastructure	30	16	-14
TSF	18	23	5
Contingency	13	7	-6
Total	**141**	**77**	**-64**

Note: *Apparent differences may occur due to rounding.*

Table 21-5: Reasons for Variance from PFS to FS

Area	Variance	Reason/s
Pit Expansion	Decrease of A$62M from A$80M to A$18M. Note TSF overhaul costs are included in Mine Development in Table 21-4	PFS assumed that in the operations phase all mining costs incurred in moving cover material were sustaining capital costs. In the FS, cover material is treated as general waste and included as capital or operating costs.
Mechanical Equipment	Decrease of A$22M from A$22M to A$0M	PFS had an allowance of 2.5% of mechanical equipment cost for replacement cost. FS has no replacement of equipment necessary for LOM assuming an establishment of overall proactive sustainable asset management strategy is in place.
TSF Lifts	Increase of A$5M from A$18M to A$23M	PFS had five lifts in sustaining capital whereas FS has six lifts based on a smaller initial wall height.
TSF Overhaul	Increase of A$13M from A$0M to A$13M	This cost was allocated to operating costs during the PFS. In the FS overhaul distance beyond the waste dump has been reclassified as Susex so as to avoid overstating the mining cost.
Contingency	Decrease of A$6M from A$13M to $7M	Contingency is a 10% allowance and was reduced relative to the reduced estimated capital cost.

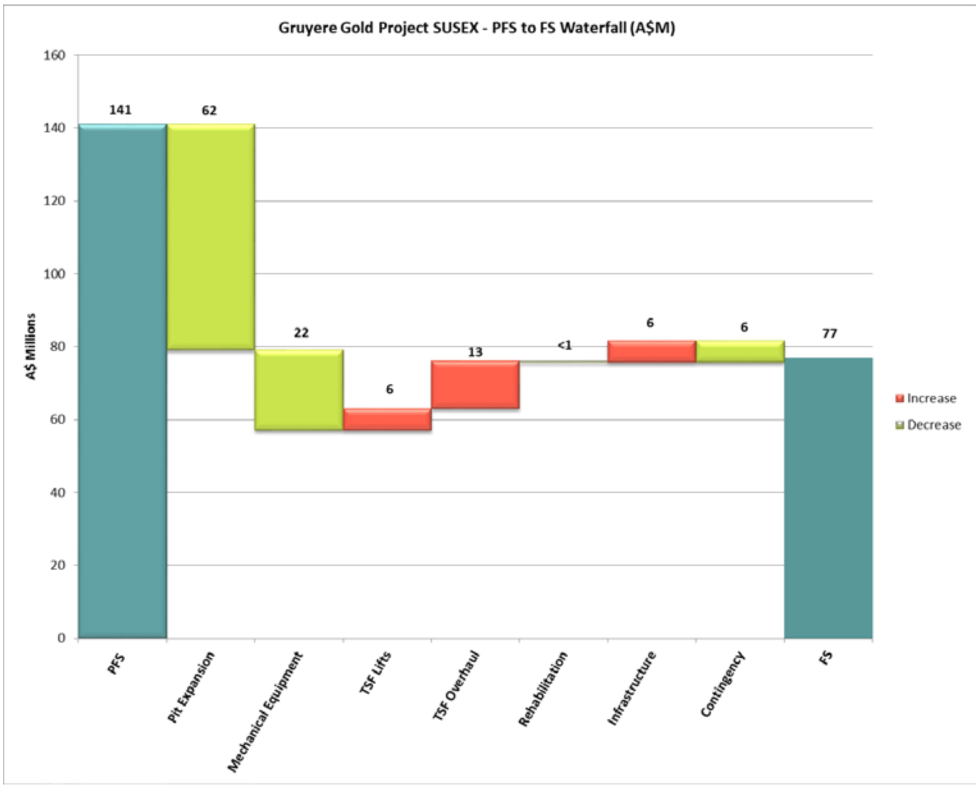

Figure 21-4: *Sustaining Capital PFS to FS Waterfall*

Table 21-6: Susex Expenditure Schedule over Life of Mine

Calendar Year	LOM Total (A$M)	FY 2019 (A$M)	FY 2020 (A$M)	FY 2021 (A$M)	FY 2022 (A$M)	FY 2023 (A$M)	FY 2024 (A$M)	FY 2025 (A$M)	FY 2026 (A$M)	FY 2027 (A$M)	FY 2028 (A$M)	FY 2029 (A$M)	FY 2030 (A$M)	FY 2031 (A$M)	FY 2032 (A$M)
Pit Expansion	31.2	2.8	2.2	2.1	1.8	4.2	0.4	3.4	0.4	6.0	1.9	1.2	1.0	1.4	2.4
Plant and Infrastructure	15.9	0.6	0.8	1.6	1.4	2.1	2.7	1.5	1.0	2.7	1.2	0.3	-	-	-
TSF	22.7	-	4.1	4.1	-	4.1	-	4.1	-	4.1	-	-	2.2	-	-
Contingency	7.0	0.3	0.7	0.8	0.3	1.0	0.3	0.9	0.1	1.3	0.3	0.2	0.3	0.1	0.2
Total Sustaining Capital	76.7	3.7	7.8	8.6	3.5	11.5	3.4	9.9	1.5	14.0	3.4	1.7	3.5	1.6	2.7
Total (Cumulative)		3.7	11.5	20.1	23.6	35.1	38.5	48.4	49.9	63.9	67.2	68.9	72.5	74.0	76.7

Note: *Apparent differences may occur due to rounding*

Accuracy of Estimate

The development, Susex and capital cost estimates have been developed to a FS level definition based on estimation plan (the Estimating Plan) and guidelines developed by Gold Road. Based on the current state of design and pricing, the accuracy of the estimate will be within -10% to +15% of the most likely value of the estimated final Project costs including contingency. The accuracy range is consistent with a Class 3 Estimate as noted in the AACE guidelines. In the development of the capital cost estimate, some of the Project definition deliverables have been completed to a higher maturity level than a Class 3 Estimate requirement.

The cost estimates were developed from engineering at the level of approximately 10%, including quantity take-offs, tendered price quotations for mills, crushers and accommodation village and budget tendered price quotations for major equipment and bulk commodities. Unit rates for installation were based on market enquiries specific to the Project and benchmarked to those achieved recently on similar projects undertaken in the Australian minerals processing industry.

Work Breakdown Structure

The overall cost estimate comprises five discrete sections:

- Open pit mining
- Mine infrastructure
- Process plant, infrastructure including EPC services and other contracts
- OR and pre-production operations
- Project Owner's costs.

All estimates were developed in accordance with the current Work Breakdown Structure (**WBS**). The WBS identifies the area and facility breakdown of the Project scope of work.

Basis of Estimate

Gold Road developed an Estimate Plan which set out the guidelines for the estimating basis and methodology used by consultants in preparing the inputs to the cost estimate. Gold Road has collated input from all consultants and will be responsible for the completeness of the Project capital cost estimate.

A capital cost estimate responsibility matrix was developed by Gold Road to ensure the estimate included the entire Project scope. The cost estimates for execution and implementation are based on the work being done under EPC and other contracts.

The capital cost estimate base currency is in Australian dollars and the cost estimate Base Date is Q2 2016.

An estimate was undertaken of the possible escalation impact on the Project from Q1 2017 through to Q4 2018. It was assessed that the potential exposure to the Project is A$7M.

The estimate for escalation was calculated using a combination of forecast rates from various sources. The rates have been applied to all Direct and Indirect costs in the capital expenditure profile.

Open Pit Mine Development and Infrastructure

The estimate includes contractor mobilisation, clearing and grubbing of the north and south pit, removal of topsoil, mine pre-strip, haul roads, waste dumps and ROM pad. Clearing, grubbing, topsoil removal and preparation of the hardstand for the mining contractor's facilities is part of the Bulk Earthworks cost estimate. All mining costs incurred prior to gold production are capitalised. The ongoing mining costs for clearing and grubbing, waste dump rehabilitation and overhaul of material to the TSF are classified as Susex.

The Capex estimate excludes the mining infrastructure facilities consisting of the heavy and light vehicle workshops, fuel and oil storage facilities, wash-down pad, mine administration office, crib rooms and mine change rooms. These costs are included in the mine operating costs.

The level of maturity in scope definition for mine development is considered well defined and the accuracy is consistent with a FS at -10% to +15%. The mine planning and scheduling is based on optimised mine pit design selection, waste dump optimisation and IWL with the TSF. A number of mine scheduling scenarios were investigated, varying in criteria for maximising early cash flow and Net Present Value (**NPV**).

Process Plant and Infrastructure

The estimate includes all site preparations, process plant, first fills and spares, buildings, site access roads, power and water supply, accommodation facilities, administration offices and Owner's mobile equipment. The total Direct and Indirect costs were estimated by GRES based on an EPC approach for execution which included all contractors' overheads and profit margins. A growth allowance has been included in the estimates to cover for any potential deviation in:

- Material take-offs due to level of engineering and design maturity
- Construction unit rates and productivity
- Material and equipment pricing costs
- Finalisation of site plans and infrastructure locations
- Ground conditions and terrain assumptions.

The Capex estimates are based on the purchase of new equipment.

The Base Date of the capital cost estimate is Q2 2016. The Owner's team made an assessment of the potential exposure to current material and equipment pricing and rates due to escalation. The completion of the facilities is planned to be in Q4 2018 (~2.5 years from the estimate Base Date).

The capital cost was derived by estimating the quantities of major commodities and the associated labour hours and costs.

In general, all estimate components have been built up on a first principles basis with quantified detailed activities defining the scope requirements and applicable pricing rates applied. The estimate is based upon preliminary engineering, quantity material take-offs, market quotations for major equipment and current cost data for similar activities and equipment. Approximately 95% of the supply cost is based on current market quotes and the installation costs are based on GRES's internal Enterprise Bargaining Agreement (**EBA**) and established structural, mechanical, piping, electrical and instrumentation productivities and construction practices.

Basis of Estimate – Direct Costs

The capital cost estimate is based on the design, construction and commissioning of a new process plant and associated infrastructure and facilities.

Based on the design criteria and flowsheets developed for the Project, preliminary plant equipment selections were made and plant layouts were developed for each process area of the plant.

Sufficient engineering design was undertaken to ensure the feasibility of the layouts, the accuracy of the equipment specifications and to enable material quantities to be estimated to the nominated level of accuracy for the FS.

Owner's Costs, Operational Readiness and Pre-Production Operating Costs

The Owner's estimate has been based on the staffing plan and organisation charts prepared by the Owner's team, set out in the FS Project Execution and the Integrated Project Master Schedule.

The Project will be executed based on the contracting strategy with the following seven major contract packages:

- EPC contract for the process plant and associated infrastructure
- Bulk Earthworks, TSF, access roads and airstrip
- Mine development and production
- Energy supply (power station and gas pipeline)
- Accommodation village supply and construct
- Water bore drilling
- Communications backbone to site.

This Project Execution Strategy is based on:

- Owner's team appropriately resourced to manage the Project
- OR plan to develop all the systems, standards and procedures
- Recruitment and training of an operations team ready to accept care, custody and control of the assets when handed over by the Owner's team
- Owner's team management and supervision of Bulk Earthworks, communications, accommodation village and bore drilling contracts and other contracts
- An EPC contracting strategy to execute the process plant and associated infrastructure
- Development of the contracting and procurement strategy for operations phase
- Minor contracts to support construction.

Contingency

Contingency has been calculated by probabilistic analysis of the perceived schedule and cost risks. This process was facilitated and modelled by Broadleaf International who conducted a series of risk workshops with Gold Road personnel. Results have been documented in Broadleaf's report Cost and Schedule Risk Analysis, (Reference 15).

The analysis indicated that the schedule, labour rates and productivity are large drivers of capital cost uncertainty. These should be reduced once the EPC and Bulk Earthworks contractors are appointed and the other early works initiated.

Capital Cost Estimate Peer Review

An independent third party, Axiom, was engaged to undertake a review of the Project capital cost estimate of the non-mining capital. The estimate was considered to have met or exceeded the criteria for FS level of quality.

21.3 Sustaining Capital Cost

The Susex estimate includes all mine development during operations, ongoing mine rehabilitation, process plant and infrastructure and TSF expansion.

Basis of Estimate – Mine Development and Rehabilitation

The mine development Susex costs include:

- Pre-Stripping and Sustainable Ore Production during Construction
- The pre-strip work from the north pit after the south pit achieves sustainable ore production during construction.

Pre-stripping During Operations

The pre-strip works for the rest of the pit outline (i.e. excluding the north and south pit), includes clearing and grubbing. Table 21-7 shows the pit expansion stage numbers and the timing of the pit stage expansion.

Table 21-7: Year of Pit Expansion for Each Stage

Stage	Year of Pit Expansion
1 and 2	2018/2019
3	2021/2022
4	2023

Mine Rehabilitation

The rehabilitation of the mine includes the following activities:

- Loading and hauling of the topsoil from the stockpiles to the waste dump
- Re-vegetating the waste dump area
- Monitoring the rehabilitated area.

Mine rehabilitation costs are estimated by applying a rate of A$0.03/t to the waste material mined.

Basis of Estimate – Process Plant and Infrastructure

The Susex estimate for the process plant and infrastructure includes:

- Replacement of light vehicles and mobile equipment (non-mining)
- Maintenance of the airstrip, camp, roads and administration buildings.

Basis of Estimate – Tailings Storage Facility

The TSF Susex cost includes all staged expansion of the TSF. There are six stages in total that will be undertaken during the LOM where Stage 1 of the TSF has been captured as part of the development capital cost estimate. Stages 2 to 7 have been included as part of Susex.

Coffey developed the design and quantities for all expansion stages of the TSF. Table 21-8 shows the total cost for the six expansion stages (Stages 2 to 7) and the year of construction, assuming construction occurs in the six months prior to the previous stage reaching capacity.

Table 21-8: Capital Cost and Year of Construction for each Stage of Tailings Storage Facility Expansion

Stage	Capital Cost (A$M)	Year of Construction
2	4.1	2019
3	4.1	2021
4	4.1	2023
5	4.1	2025
6	4.4	2027
7	2.2	2029

Contingency

The total contingency has been estimated by applying 10% to the total Susex.

21.4 Operating Costs

This section summarises operating cost estimates for the Project. Operating costs are sub-divided into mining, processing, transport and refining, site and corporate General and Administration (**G&A**) costs.

All operating costs for the Project have been estimated based on costs prevailing in the Australian minerals industry for Q2 2016. No escalation has been applied as the LOM operating costs are estimated in Real terms consistent with the Financial Model. All costs were estimated to a level of accuracy of -10% to +15%. Rounding errors may occur in the numbers tabulated in this section.

Operating Cost Summary

Operating costs are shown in Table 21-9.

Table 21-9: Summary of Operating Costs

Item	LOM Cost (A$M)	Unit Cost/t Milled (A$/t)	Unit Cost/oz Produced (A$/oz)	Proportional Cost
Mining	1,228.6	13.42	383	44.1%
Processing	1,433.4	15.65	446	51.4%
Transport and Refining	5.1	0.06	2	0.2%
G&A (Site & Corporate)*	120.8	1.32	38	4.3%
Total	2,788.0	30.45	869	100.0%

Notes:

- **General and Administration (**G&A**) costs in the table above include site and corporate*
- *Apparent differences may occur due to rounding.*

Mining Operating Costs

Mining costs for the Project were estimated by AMC based on the quarterly mining schedule. The mining operating cost estimate has been prepared using the Cost Model. The Cost Model has been refined by comparison against a range of projects including both owner mining and contract mining estimates. The operating costs for drilling, blasting, loading and hauling, topsoil removal and replacement, and crusher feed activities were developed from first principles. This includes operating hours, haul cycles, labour rates, fuel consumption, maintenance requirements and consumables. This section covers mining costs incurred from Q4 2018, which is the quarter during which gold production commences. All mining costs incurred prior to this date are classed as capital costs.

The Cost Model has several key assumptions. There will be two 12 hour shifts per day for a continuous operation with twenty production shifts per annum lost to weather interruptions. Mining operations and maintenance personnel will work a 2 weeks on, 1 week off (2:1) roster while management and technical staff will work a 9 days on, 5 days off (9:5) roster. Ownership costs of all equipment (which will be owned by the mining contractor) is based on a financing model. The diesel fuel cost assumption for the mine operating costs is A$0.65/L (net of Diesel Fuel Rebate). In estimating the costs a mining contractor margin of 11.5% was assumed.

The cost model includes an allowance for the cost of the Gold Road mining team including mining department management and technical functions such as engineers, geologists and surveyors.

The average LOM mine operating cost is A$3.56/t mined or A$13.42/t milled. The main cost centres by activity are drill and blast, and haulage.

Table 21-10 and Figure 21-5 provide a summary of the LOM mine operating cost by activity.

Table 21-10: Mining Operating Costs by Activity

Activity	LOM Cost (A$M)	LOM Cost (A$/t mined)	LOM Cost (A$/bcm mined)	LOM Cost (A$/t processed)	LOM Cost (A$/oz)	Proportional Cost
Load and Haul	468.4	1.35	3.35	5.12	146	38%
Drill and Blast	389.1	1.13	2.79	4.25	121	32%
Other Mining	169.6	0.50	1.20	1.85	53	15%
Management and Overheads	201.4	0.58	1.44	2.19	62	15%
Total	1,228.6	3.56[1]	8.79[1]	13.42	383	100%

Notes: *The A$/t mined and A$/bcm mined are calculated to include material mined during the construction period for which the estimated cost is capitalised; apparent differences may occur due to rounding.*

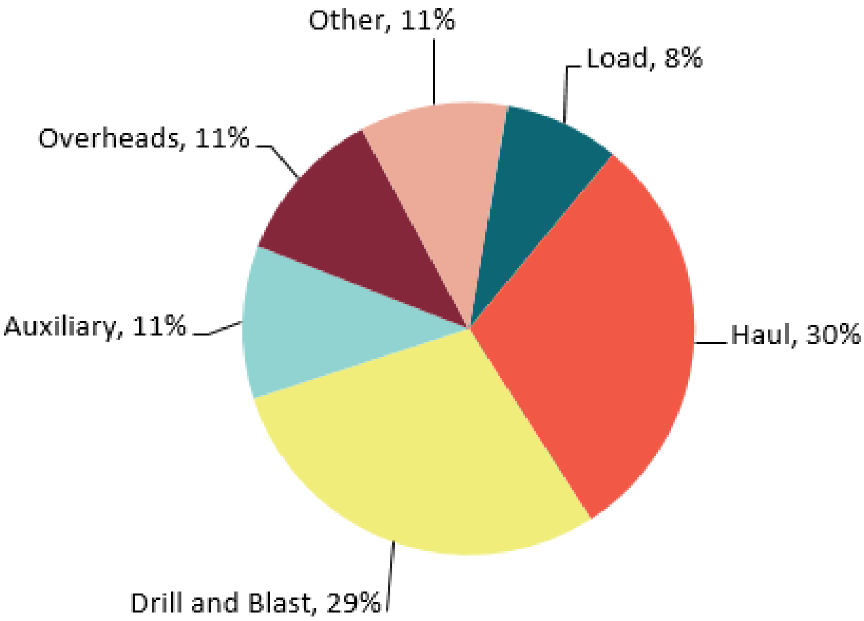

Figure 21-5: *Life of Mine Operating Costs by Activity*

Figure 21-6 illustrates mine operating cost on an annual basis. The step change increase in FY 2024 is due to the commencement of the Stage 4 cutback. Table 21-11 shows the mining costs by activity on an annual basis.

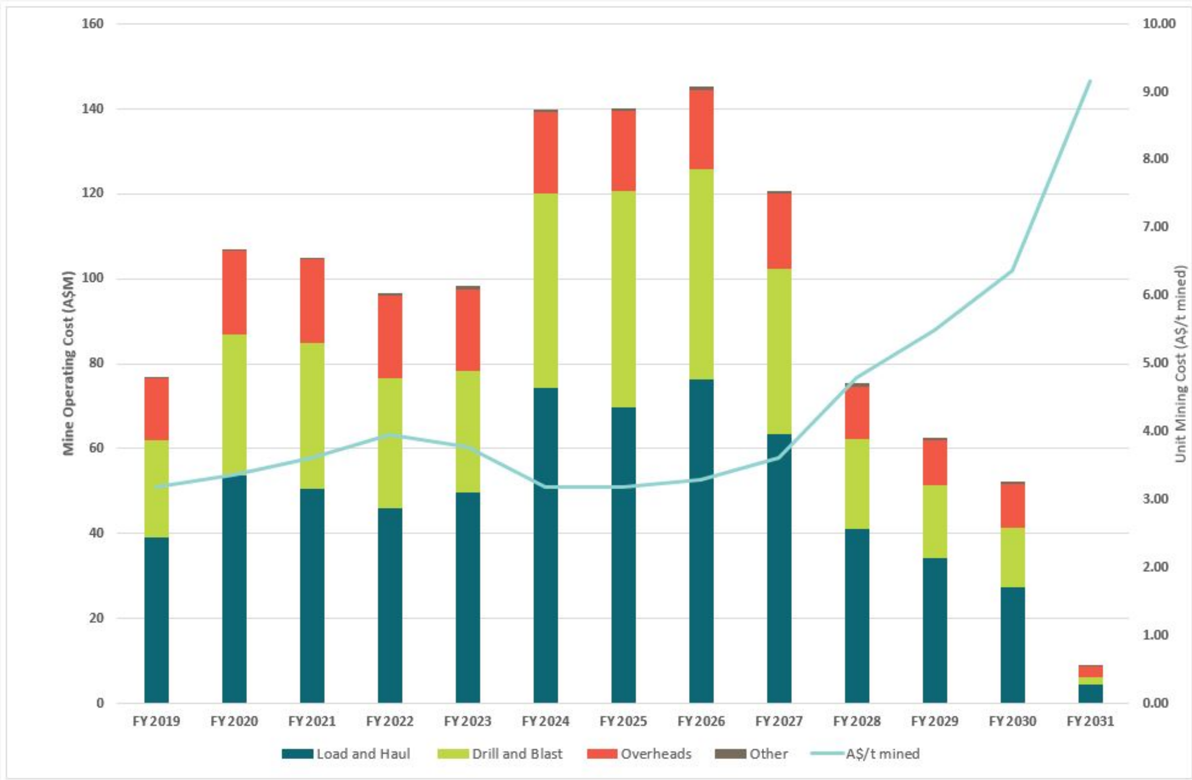

Figure 21-6: *Total Mine Operating Costs*

Table 21-11: Annual Mine Operating Costs by Activity

Activity	TOTAL A$M	FY 2019 A$M	FY 2020 A$M	FY 2021 A$M	FY 2022 A$M	FY 2023 A$M	FY 2024 A$M	FY 2025 A$M	FY 2026 A$M	FY 2027 A$M	FY 2028 A$M	FY 2029 A$M	FY 2030 A$M	FY 2031 A$M
Load and Haul	468.4	27.9	39.3	36.6	33.3	36.6	59.8	55.5	60.9	49.5	28.3	21.5	16.9	2.2
Drill and Blast	389.1	23	33	34.1	30.6	28.6	45.9	50.8	49.8	39	21.3	17.1	13.8	1.9
Other Mining	169.6	11.4	14.7	14.6	13.3	13.9	15.3	15.1	16.1	14.9	13.4	13.6	11.1	2.3
Management and Overheads	201.4	14.5	19.8	19.7	19.5	19	18.8	18.7	18.4	17.4	12.2	10.6	10.3	2.4
TOTAL	1228.6	76.8	106.8	105	96.7	98.1	139.8	140.1	145.2	120.8	75.2	62.8	52.1	8.8

Note: *Apparent differences may occur due to rounding.*

Process Operating Costs

Life of Mine Summary

The LOM operating cost estimate for the process plant was completed for a blend of different ore types (Fresh, Transition and Oxide) and grind sizes (P_{80} of 125 µm and 150 µm) at various throughput rates. This was based on the annual operating cost estimates for the different ore types and grind sizes completed by GRES.

Figure 21-7 shows the breakdown of process plant feed material by ore type on an annual basis. Fresh ore constitutes 83% of total LOM feed, Transition ore accounts for 4% and Oxide ore makes up the remaining 13%.

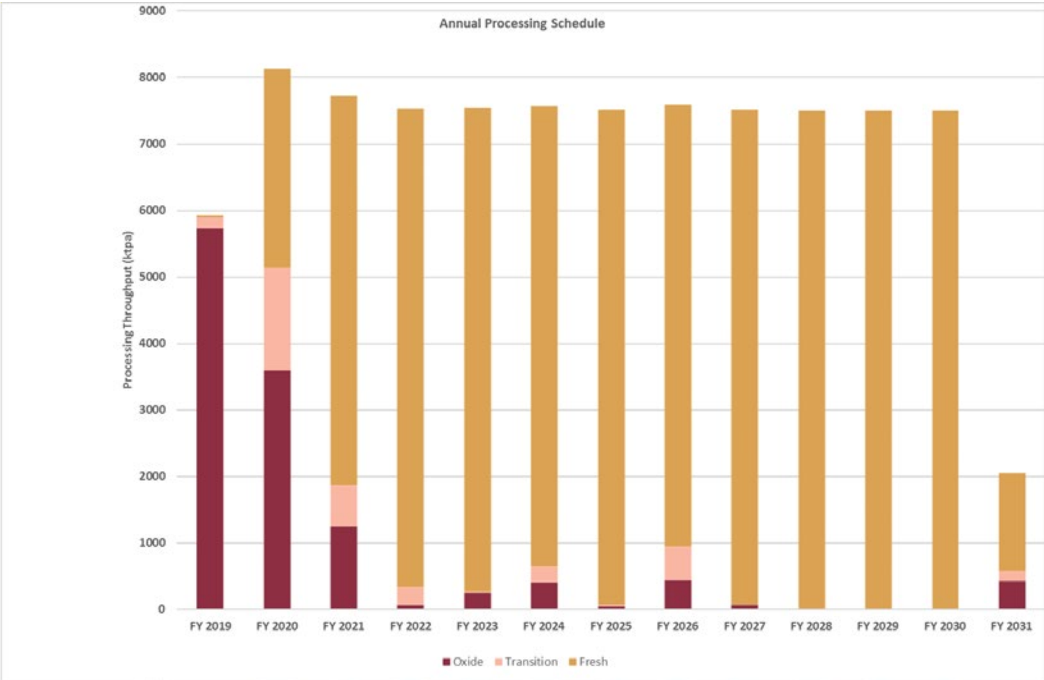

***Figure 21-7**: Annual Process Plant Feed by Ore Type*

Table 21-12 and Figure 21-8 provides a summary of the LOM operating cost by cost centre based on the process plant feed schedule of the LOM.

Table 21-12: Life of Mine Average Process Operating Cost Estimate - Summary by Cost Centre (for Fresh, Transition and Oxide Material)

Cost Centre	LOM Cost (A$M)	Unit Cost (A$/t processed)	Unit Cost (A$/oz)	Proportional Cost
Power	639.2	6.98	199	45%
Reagents and Grinding Media	434.8	4.75	135	30%
Labour	124.4	1.36	39	9%
Wear Materials	111.3	1.22	35	8%
Maintenance Spares, Consumables and Contractors	64.7	0.71	20	4%
Other	59.1	0.65	18	4%
Total	1,433.5	15.65	446	100%

Note: *Apparent differences may occur due to rounding.*

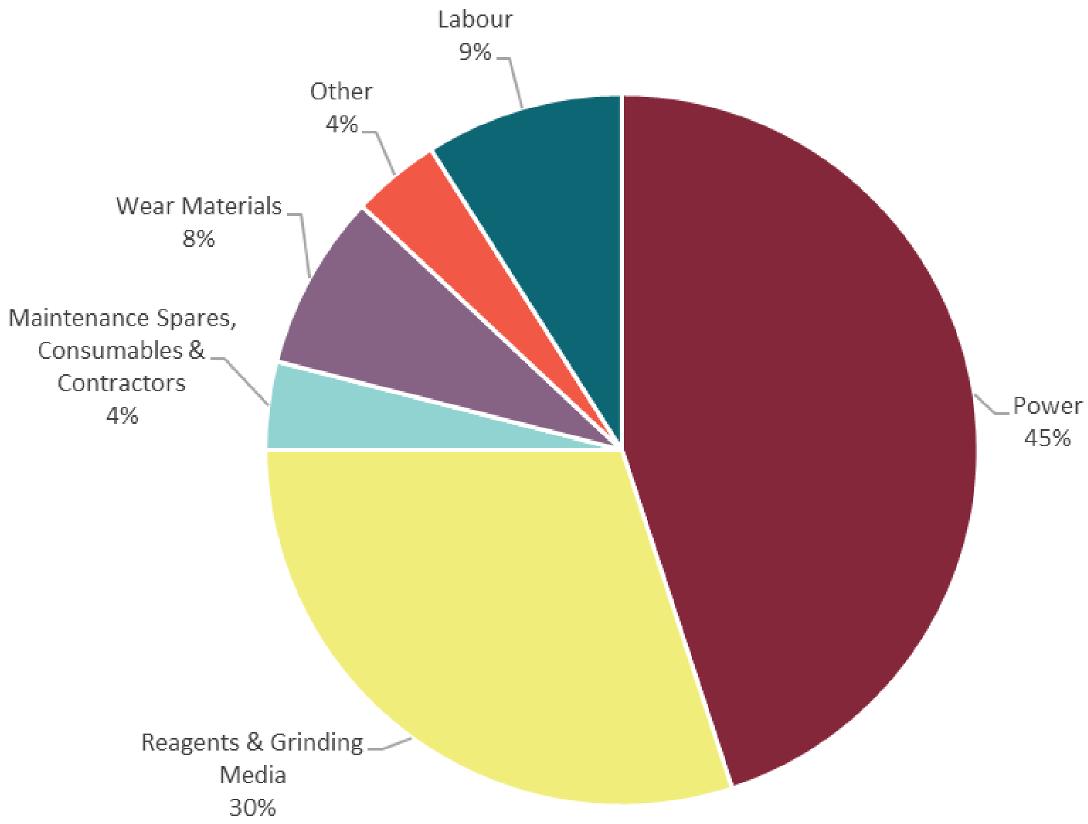

Figure 21-8: *Chart Illustrating Process Operating Cost Breakdown*

Figure 21-9 shows the process operating costs on an annual basis. Unit cost are relatively low in the first three years of production due to a significantly high proportion of Oxide and Transition ore in comparison to the LOM average.

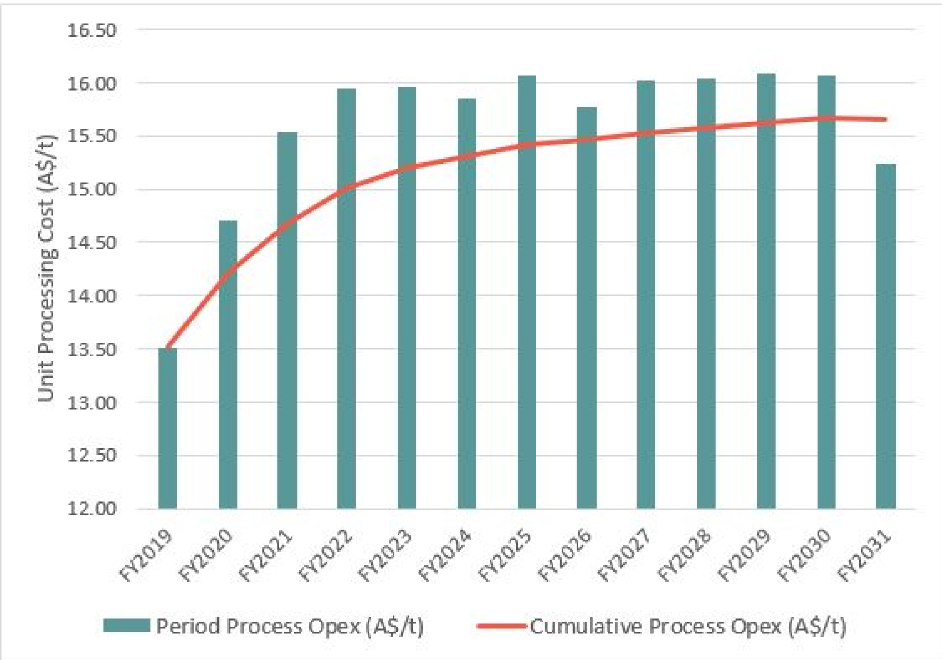

Figure 21-9: *Process Operating Costs*

Other Operating Costs

Transport and Refining Costs

Gold bullion will be transported from site by chartered plane under armed security escort by an established provider to the Perth Airport. Secured transport will transfer the doré to the Perth Mint refinery located within the Perth Airport complex for refining. The gold refining fee is estimated on the basis of information provided by a leading gold refinery in Perth. Refining fees are likely to be negotiable and are dependent on annual bullion sales and the final contract terms negotiated with the refinery.

General and Administration Costs

The General and Administration cost include the Site G&A and the Corporate costs. Site G&A costs include all personnel relating to site management, administration, health, safety, environment and community relations. This cost also includes the supplies needed to sustain the activities related to the various non-operating support departments such as procurement, environment, Health, Safety and Environment (**HSE**) and Human Resources (**HR**). Corporate costs were allocated based on an assessment of individual role's level of support to the Project and includes investor relations, marketing and legal fees.

Table 21-13 shows the total of the other operating costs over the LOM.

Table 21-13: Other Operating Costs Over Life of Mine

Cost Centre	Cost over LOM (A$M)	Unit Cost/oz Produced (A$/oz)
Transport and Refining	5	2
General and Administration	121	38
Total	125	40

Note: *Apparent differences may occur due to rounding.*

22 ECONOMIC ANALYSIS

PCF was commissioned to undertake the PFS and FS Project financial and debt modelling. All Owner's team expenditures prior to January 2017 are treated as sunk costs, including all Project study costs (PFS and FS). The financial analysis is based on a quarterly production plan. The period commencing from August 2016 to the end of commissioning and handover (October 2018) is analysed on a monthly basis.

This section presents financial results on a financial year (July to June) basis. A summary of that information and how it compares to the PFS is outlined below in Table 22-1.

Table 22-1: Summary of FS Financial Outcomes (all run at A$1,500 per ounce or US$1,095 per ounce at US$0.73:A$1.00)

Measure	Units	PFS Outcome[8]	FS Outcome A$M	FS Outcome US$M[8]
Gold Produced	koz	2,917	3,212	-
Gross Revenue	A$M	4,375	4,817	3,516
Free Cash flow - Pre-Tax	A$M	1,087	1,222	892
Free Cash flow - Post-Tax	A$M	772	845	617
IRR (Pre-Tax)	%	27.5	24.0	-
IRR (Post-Tax)	%	21.2	19.5	-
NPV (Pre-Tax)[1]	A$M	464	486	355
NPV (Post-Tax)[1]	A$M	293	305	223
C1 Cash Costs[2]	A$/oz	853	858	626
C2 Cash Costs[3]	A$/oz	1,058	1,040	759
C3 Cash Costs[4]	A$/oz	1,109	1,093	798
AISC[5]	A$/oz	961	945	690
AIC[6]	A$/oz	1,117	1,103	805
Development Capital Cost[7]	A$M	456	507	370
Development Capital Cost per ounce (Development Capex/Gold Produced)	A$/oz	157	158	115
Capital Efficiency (NPV (Pre Tax)/ Development Capex)		1.0	1.0	-
Payback	Months	42	48	-
Payback: LOM	%	32	33	-
Project LOM Costs[9]	A$M	3,258	3,542	2,586

Notes:

1. *8% Discount rate applied*
2. *C1 = Mining + Processing Operating Expenditure + Site General and Administration Expenditure + Transport and Refining Costs*
3. *C2 = C1 + Depreciation + Amortisation*
4. *C3= C2+ Royalties + Levies + Net Interest Costs*
5. *AISC = C1 + Royalties + Levies + Sustaining Capital + Project related offsite Corporate expenditure*
6. *AIC = AISC + Development Capital Expenditure*
7. *The Development Capital Cost is in Q3 2015 (PFS) and Q2 2016 (FS) Real terms. The forecast capital cost including potential escalation to Project completion (Q4 2018) is estimated to be A$514 million.*
8. *US$:A$ exchange rate as per Table 22-2*
9. *Excludes mine site closure costs of $54 million*

22.1 Financial Data and Assumptions

Table 22-2 shows the key financial inputs and assumptions that were applied in the estimation of the Project costs and financial analysis.

Table 22-2: Key Financial Assumptions

Parameter	Units	PFS Assumptions	FS Assumptions
Gold Price	A$/oz	1,500	1,500
Exchange Rates	A$1:US$	0.73	0.73
Accumulated Tax Losses	A$	90M*	90M*
Corporate Income Tax	%	30	30
Power Cost	A$/KWh	0.21	0.21
Diesel Price (after rebate)	A$/litre	0.75	0.65

Note: **Estimated Tax Losses as at end of 2016 financial year.*

Gold Price

The financial model assumes a constant gold price of A$1,500 or US$1,095 per ounce at US$0.73:A$1.00 throughout the LOM. This assumption is based on the historical five-year average gold price.

Exchange Rate

The financial model assumes a constant A$:US$ exchange rate of US$0.73:A$1.00 throughout the LOM. Approximately 8% of the Project development capital cost estimate is denominated in foreign currency. This is considered to constitute low foreign currency risk exposure (on the cost input side).

Power Cost

Power generation facilities are planned to be constructed on a BOO arrangement utilising gas fuel piped to site. The cost of power generation is calculated at an average of A$0.21/KWh based on detailed power studies completed in-house and validated by the Energy Supply Agreement tender process conducted during the FS. Major risks to the power cost forecast include higher than projected gas commodity pricing, higher than anticipated compression requirements and a potential introduction of a carbon tax by the Federal Government.

Diesel Price

The diesel price during construction and operations has been set at a flat rate of A$0.65/litre (net of government rebate) based on benchmarking of bulk diesel price supply costs to similar remote Western Australian mines as at May 2016. Major risks for the use of diesel include higher than forecast prices, changes in excise/ duties/ rebates, logistics and storage constraints.

22.2 Marketing

Gold projects are in the unique position of not having to market product, other than to establish an agreement with a refiner to take product on normal commercial terms for precious metals doré production.

Refiner Selection

For the purposes of the FS it has been assumed that gold will be refined by a leading gold refinery in Perth and costs reflect this option. Prior to gold production this service will be tendered for contract award.

Pricing Strategy

Gold Road will negotiate the general terms of product sales with the intended refiner. For modelling purposes a gold price of A$1,500 per ounce has been used in calculating the revenue from sales.

22.3 Gold Production and Revenues

Total gold production over the LOM is 3.2 Moz with average annual production of 265,126 oz. Gold production per year is shown in Figure 22-1. Total gross revenue from the sale of gold over the LOM, using the assumed gold price of A$1,500 per ounce is estimated to be A$4,817M. Table 22-3 shows estimated gold production and gross revenues on an annual basis.

Gold will be delivered first into any derivative (hedging) contracts that may be in place with the remaining gold sold into the spot market via commercial arrangements with the chosen refiner. The FS assumes that 100% of gold is sold each month/quarter with no gold kept on hand (other than standard processing amounts remaining "in-circuit").

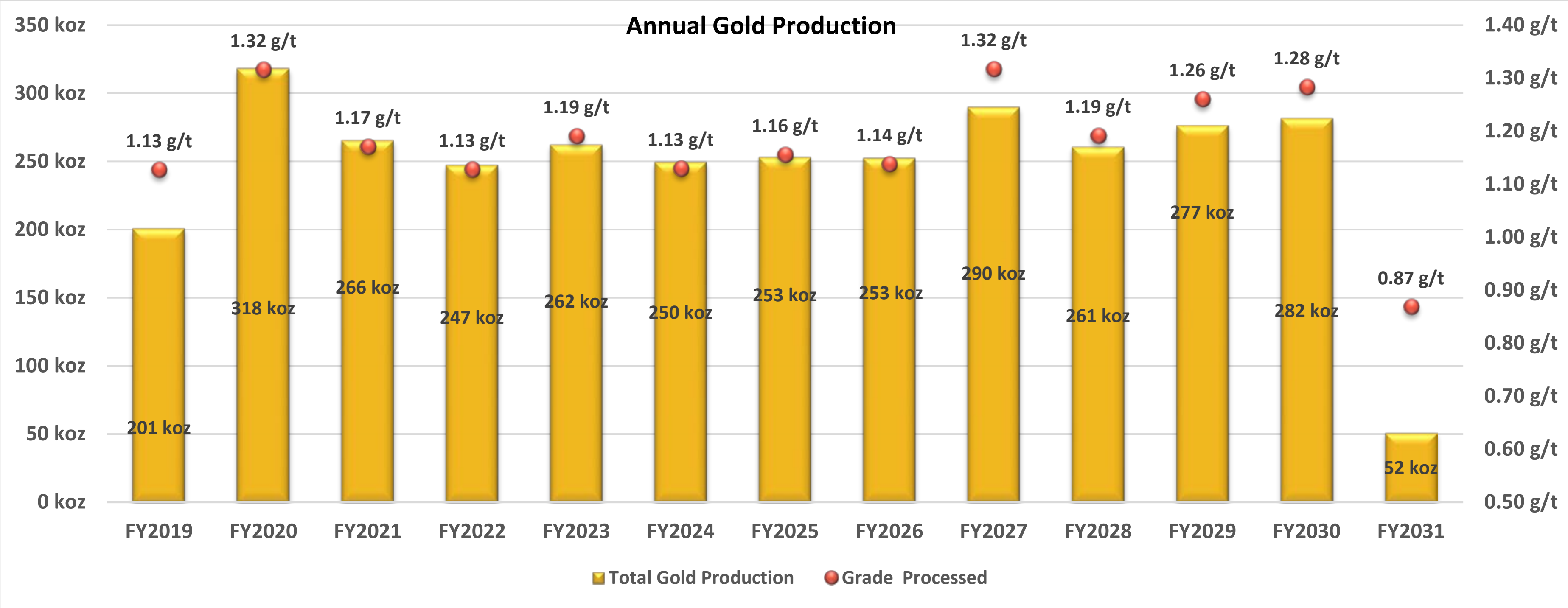

Figure 22-1: *Annual Gold Production (LHS) and Grade of Ore Processed (RHS)*

Table 22-3: Gold Production and Revenues

	Units	Total	FY2019	FY2020	FY2021	FY2022	FY2023	FY2024	FY2025	FY2026	FY2027	FY2028	FY2029	FY2030	FY2031
Gold Production	koz	3,212	201	318	266	247	262	250	253	253	290	261	277	282	52
Gold Revenue	A$M	4,817	302	477	399	371	394,	375	380	379	435	391	415	423	78

Note: *Apparent differences may occur due to rounding.*

22.4 Project Capital and Operating Costs

Operating Costs

The total estimated LOM operating cost for mining, processing, transport and refining and other costs including general and administration, royalties and rehabilitation levy is A$2,958M. Summary of the operating costs is shown in Table 22-4.

Table 22-4: Operating Costs Summary

Item	PFS LOM Cost (A$M)	PFS LOM Cost (A$/oz)	FS LOM Cost (A$M)	FS LOM Cost (A$/oz)
Mining	1,120	384	1,229	383
Processing	1,298	445	1,433	446
Transport and Refining	5	2	5	2
Other Costs[1]	238	82	291	90
Total Opex	2,661	912	2,958	921

Notes:

1. *Other Costs include G&A, royalties and rehabilitation fund levy.*
2. *All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.*

Capital Costs

The Project capital costs are outlined in Section 21: Capital and Operating Costs and summarised in Table 22-5. They have been categorised as either Development Capital or Sustaining Capital.

Table 22-5: Capital Costs Summary

Item	PFS LOM Cost ($AM)	FS LOM Cost (A$M)
Development Capital		
Direct		
Process Plant & Infrastructure & TSF	186	178
Infrastructure and Utilities – Site General	59	79
Mine Development	33	36
Power Supply and Distribution	19	20
Site Development and Site Drainage	6	8
Indirect		
Engineering and Contractors	81	86
Project Owner's Team and Pre-Production Operations	35	50
Capital, Operating and Commissioning Spares	4	7
Contingency	35	43
Sub Total – Development Capital	456	507
Sustaining Capital		
Mine Development	80	31
Plant & Infrastructure	30	16
TSF	18	23
Contingency	13	8
Sub Total – Sustaining Capital	141	77
Total Capital Cost	597	584

Note: *All figures are rounded to reflect appropriate levels of confidence and apparent differences may occur.*

Salvage Costs

The financial model assumes that all Project assets have no salvage value.

Working Capital

Project working capital is required to purchase consumables prior to commercial gold production. An inventory of supplies to cater for nine days of reagent and grinding media consumption has been estimated for the Project. This level of inventory is considered adequate for the scale and geographic location of the Project relative to nearby supply centres. A total of A$3.65M is estimated (as a percentage of operating expenditure) for working capital costs and this amount is in addition to the capital cost estimate for first fills and spares.

Rehabilitation and Mine Closure Costs

Rehabilitation and mine closure costs have been estimated based on guidelines provided by DMP. The total Rehabilitation Liability Estimate for the Project is A$61.7M, of which A$7.9M is sustaining capital over LOM.

Tax

It is estimated that the Project will incur a total tax liability of approximately A$376M over the LOM. Table 22-6 shows the estimated tax liability on an annual basis. Income tax is not payable until FY2020 due to carry forward tax losses providing a shelter for the first two production years.

Table 22-6: Project Tax Schedule

	Total A$M	FY 2018 A$M	FY 2019 A$M	FY 2020 A$M	FY 2021 A$M	FY 2022 A$M	FY 2023 A$M	FY 2024 A$M	FY 2025 A$M	FY 2026 A$M	FY 2027 A$M	FY 2028 A$M	FY 2029 A$M	FY 2030 A$M	FY 2031 A$M
Revenue	4,817	-	302	477	399	371	394	375	380	379	435	391	415	423	78
Opex[1]	-3,012	-4[2]	-176	-253	-249	-240	-243	-283	-284	-288	-266	-219	-208	-197	-101[3]
EBITDA	1,805	-4	126	224	149	131	151	92	96	91	169	172	208	225	-23
D&A	-584	-23	-89	-75	-62	-52	-43	-37	-31	-27	-22	-21	-17	-15	-70[3]
EBIT	1,222	-27	38	149	87	79	108	55	65	64	146	151	190	211	-94
Net Interest[4]	-	-	-	-	-										
EBT	1,222	-27	38	149	87	79	108	55	65	64	146	151	190	211	-94
Tax Payable	-376	0	0	-21	-26	-24	-32	-16	-19	-19	-44	-45	-57	-63	-9
NPAT	845	-27	38	128	61	56	75	38	45	45	102	106	133	147	-103

Notes:

1. *Includes Total Opex Cost A$2,958M (Table 22-4) plus mine closure costs*
2. *Includes values from FY2017*
3. *Includes values from FY2032*
4. *Interest Income not included as Gold Road does not consider it to be specifically related to the Project*
5. *Apparent differences may occur due to rounding*
6. *EBITDA Earnings Before Interest, Tax, Depreciation and Amortisation*
7. *D&A Depreciation and Amortisation*
8. *EBIT Earnings Before Interest and Tax*
9. *EBT Earnings Before Tax*
10. *NPAT Net Profit After Tax*

22.5 Project Financial Outcomes

Table 22-7 shows key Project financial outcomes.

Table 22-7: Project Financial Performance Outcomes (A$1,500 per ounce or US$1,095 per ounce at US$0.73:A$1.00)

Measure	PFS Outcome	FS Outcome
Pre-Tax Net Present Value/Development Capex	1.0	1.0
IRR (Pre-Tax)	27.5%	24.0%
IRR (Post-Tax)	21.2%	19.5%
All in Sustaining Costs per ounce	A$961/US$701	A$945/US$690
Payback Period	42 months	48 months
Payback Period: LOM	32%	33%

Internal Rate of Return (IRR)

The Project IRR on a Capex to completion basis is 24.0% before tax and 19.5% after tax.

Payback Period

The payback period on a Capex to completion basis is 48 months. This represents approximately 33% of the Project life.

Project Cash Flow

The estimated total undiscounted cash flow pre-tax is A$1,222M and after tax A$845M. This estimate is calculated on a Capex to completion basis. Capex commitments start in August 2016. Table 22-8 shows the annual Project cash flow.

Table 22-8: Project Annual Cash Flow Summary

Summary Cash Flow	Total A$M	FY 2018 A$M	FY 2019 A$M	FY 2020 A$M	FY 2021 A$M	FY 2022 A$M	FY 2023 A$M	FY 2024 A$M	FY 2025 A$M	FY 2026 A$M	FY 2027 A$M	FY 2028 A$M	FY 2029 A$M	FY 2030 A$M	FY 2031 A$M
Revenue	4,817	0	302	477	399	371	394	375	380	379	435	391	415	423	78
Transport, Refining Charges, Royalties, Rehab	-175	-0.5[1]	-11	-17	-14	-14	-14	-14	-14	-14	-16	-14	-15	-15	-3
Net Revenue	4,642	-0.5	291	460	384	358	379	361	366	365	419	377	400	407	75
Operating Costs[2]	-2,837	-4	-165	-236	-235	-226	-228	-269	-270	-274	-251	-205	-193	-182	-98[3]
Change in Working Capital	-	-	14	1	0.2	-2	5	0.2	0.1	0.7	-5	-1	-1	-0.7	-10
Net Operating Cash flows	1,805	-4	140	225	150	129	156	92	96	91	163	171	206	225	-34
Capital Costs	-584	-481	-30	-8	-9	-4	-11	-3	-10	-2	-14	-3	-2	-4	-4
Net Cash flow Before Tax	1,222	-485	110	217	141	126	144	88	86	90	149	167	204	221	-38
Tax	-376	-	-	-21	-26	-24	-32	-16	-19	-20	-44	-45	-57	-63	-9
Net Cash Flows	845	-485	110	197	115	102	112	72	67	71	105	122	147	158	-47

Notes:

1. *Includes values from FY2017*
2. *Includes Mine Closure and off site Corporate General and Administration costs for Project*
3. *Includes values from FY2032*
4. *Apparent differences may occur due to rounding*

EBITDA vs AISC

Figure 22-3 shows the estimated annual EBITDA against the annual AISC at a gold price of A$1,500 per ounce.

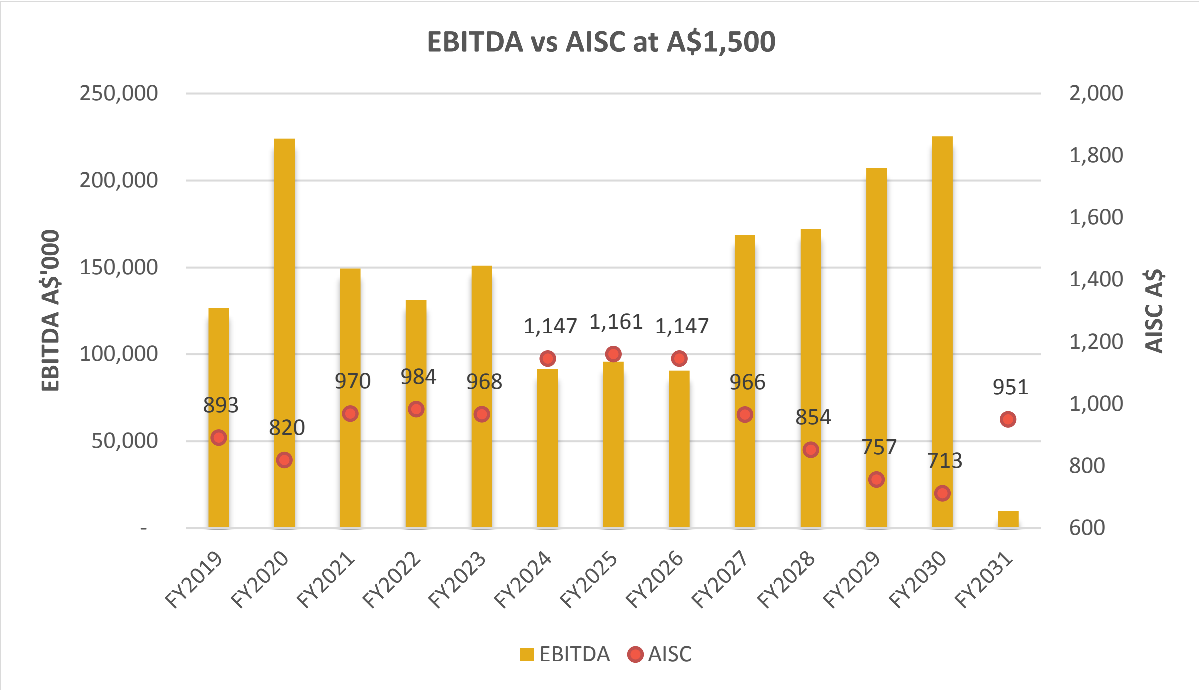

Figure 22-2: *EBITDA vs AISC at A$1,500 per ounce*

22.6 Sensitivity Analyses

Key Project and market driven variables were subjected to sensitivity analyses to assess their impact on Project economic viability (measured in NPV and IRR terms). Tables 22-9 to 22-14 and Figures 22-4 to 22-7 below presents the outcome of the sensitivity analyses.

The Project operating cost estimate has minor exposure to the US$ diesel price.

Diesel Costs

Table 22-9: Diesel Cost Sensitivity

Diesel Price (A$/l)	Opex (A$M)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
0.65	2,788	24.0	486	19.5	305
0.75	2,811	23.7	474	19.2	296
0.85	2,834	23.3	461	18.9	287
0.95	2,858	22.9	448	18.6	278
1.05	2,881	22.6	435	18.3	269

Operating Costs

Table 22-10: Operating Cost Sensitivity

% Var	Opex (A$M)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
-15	2,375	30.2	713	24.4	464
-10	2,513	28.2	637	22.8	411
-5	2,650	26.2	562	21.2	358
0	2,788	24.0	486	19.5	305
5	2,925	21.9	411	17.7	252
10	3,063	19.6	335	15.8	199
15	3,201	17.2	260	13.9	146
20	3,338	14.7	184	11.8	93

Feed Grade

Table 22-11: Feed Grade Sensitivity

% Var	Feed Grade (g/t)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
-20	0.96	7.1	-22	5.4	-58
-15	1.02	12.0	105	9.6	36
-10	1.08	16.3	232	13.1	126
-5	1.14	20.3	359	16.4	216
0	1.20	24.0	486	19.5	305
5	1.26	27.6	613	22.3	394
10	1.31	31.0	741	25.1	483

Process Recovery

Table 22-12: Process Recovery Sensitivity

% Var	Recovery (%)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
-3	88.6	21.8	410	17.7	251
-2	89.5	22.6	436	18.3	269
-1	90.4	23.3	461	18.9	287
0	91.3	24.0	486	19.5	305
1	92.2	24.8	512	20.1	323
2	93.1	25.5	537	20.6	341

Development Capital Expenditure

Table 22-13: Development Capex Sensitivity

% Var	Capex (A$M)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
-15	431	28.8	555	23.4	360
-10	456	27.0	532	22.0	342
-5	481	25.5	509	20.7	323
0	507	24.0	486	19.5	305
5	532	22.7	464	18.4	287
10	557	21.5	441	17.4	268
15	583	20.4	418	16.4	250
20	608	19.3	395	15.5	231
25	633	18.3	372	14.7	213

Gold Price

Table 22-14: Gold Price Sensitivity

% Var	Gold Price ($/oz)	Pre-Tax IRR (%)	Pre-Tax NPV$_{8\%}$ (A$M)	Post-Tax IRR (%)	Post-Tax NPV$_{8\%}$ (A$M)
-17	1,250	10.4	62	8.2	5
-13	1,300	13.4	147	10.8	66
-6	1,400	19.0	317	15.4	186
0	1,500	24.0	486	19.5	305
6	1,600	28.7	656	23.3	424
12	1,700	33.2	826	26.8	543
17	1,800	37.4	995	30.2	662
21	1,900	41.5	1,165	33.5	781

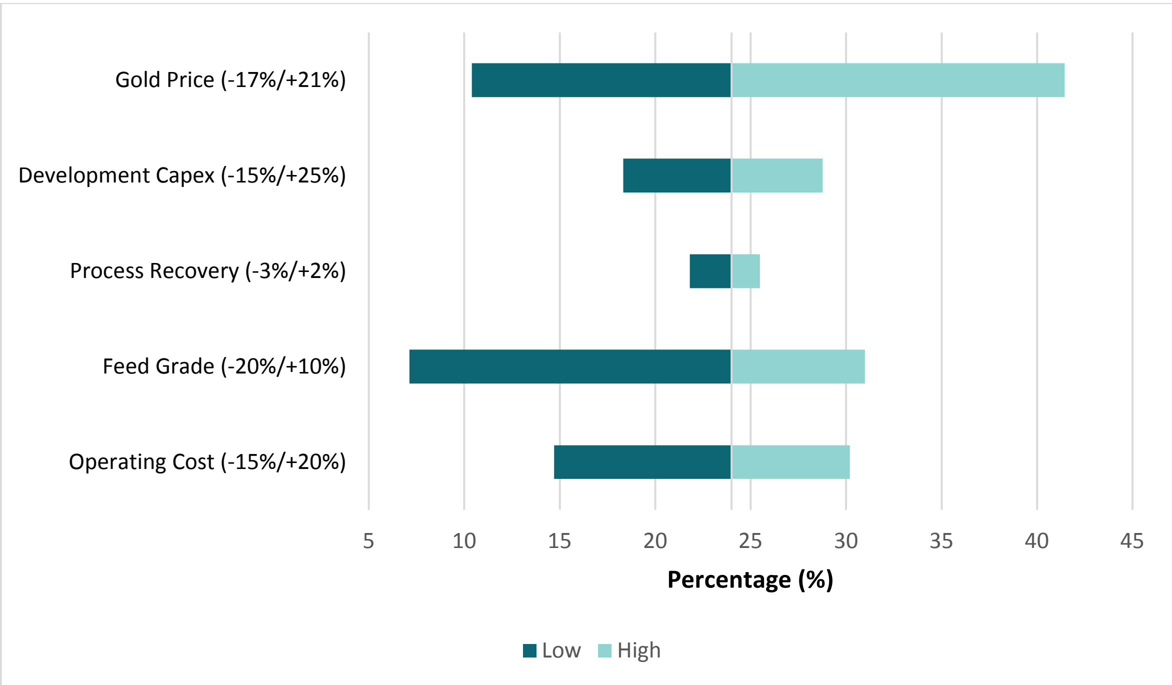

***Figure 22-3:** Pre-Tax Internal Rate of Return Sensitivity Chart*

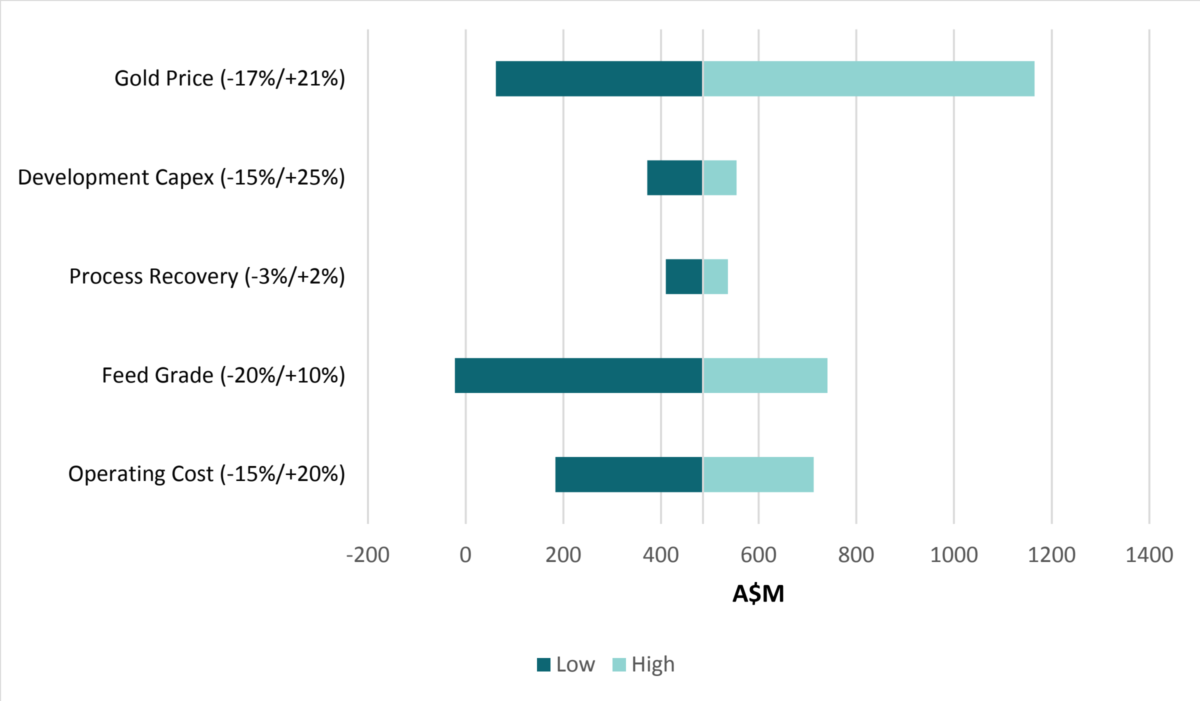

__Figure 22-4:__ Pre-Tax Net Present Value Sensitivity Chart

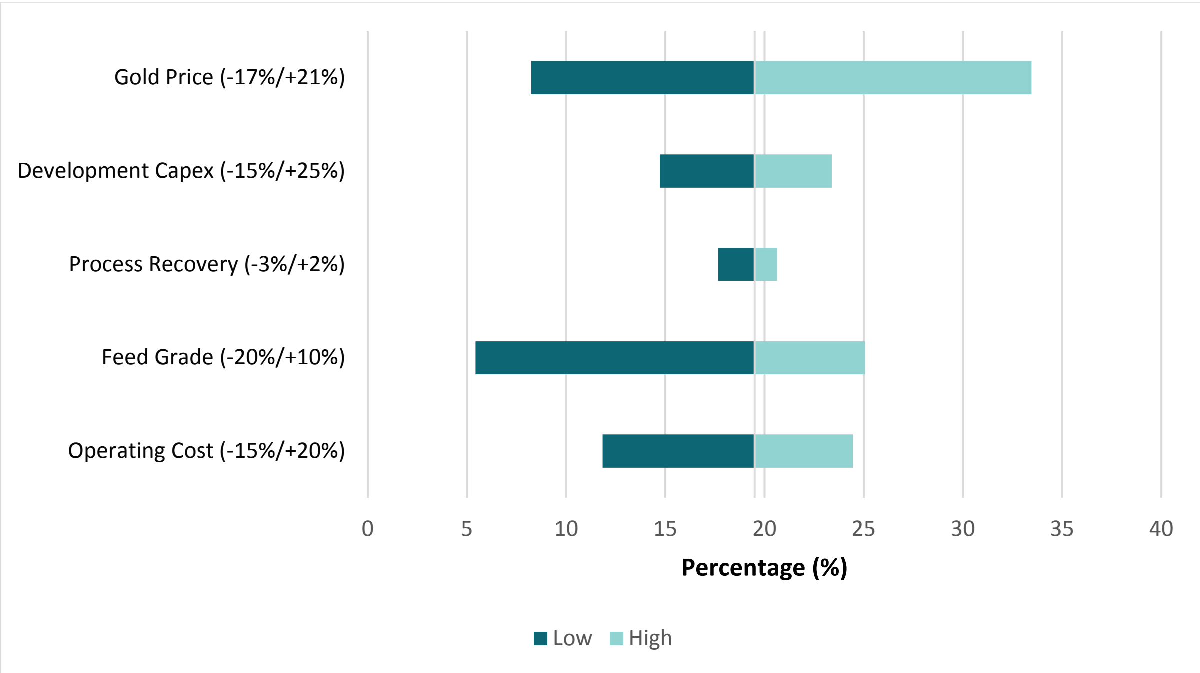

__Figure 22-5:__ Post-Tax Internal Rate of Return Sensitivity Chart

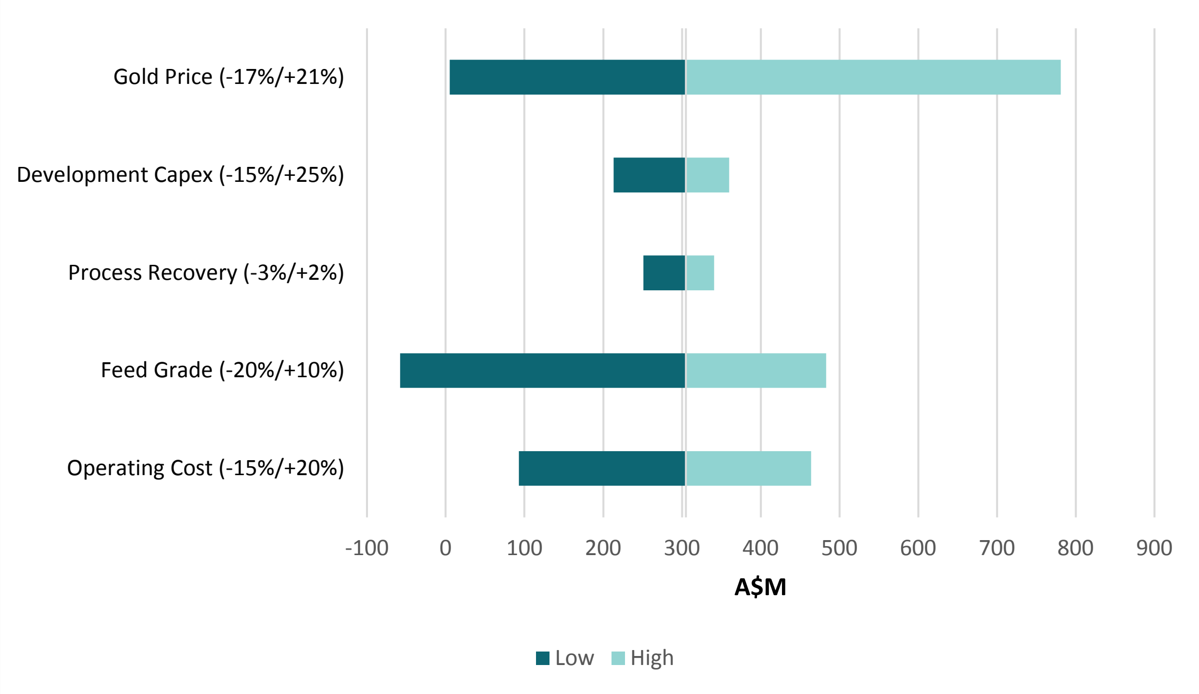

Figure 22-6: *Post-Tax Net Present Value Sensitivity Chart*

The sensitivity analysis identifies that the following parameters are the major Project value drivers:

- Gold price
- Feed grade
- Operating costs.

22.7 Project Funding

Gold Road's proposed financing strategy for the development of the Project will include, but not be limited to, the following factors:

- Securing a fully funded solution for the development of the Project
- Minimising potential dilution to existing Gold Road shareholders
- Providing flexible funding solutions to:
    - Ensure the continuation of exploration activities
    - Facilitate additional development opportunities.
    - Capitalise on favourable external factors such as gold price. (e.g. hedging when the spot price is substantially above the FS gold price assumptions).

The Company will be reviewing and assessing the available funding options in order to maximise the benefits to shareholders.

Potential funding options being considered include:

- Traditional debt and equity structures - preliminary discussions have been held with a number of local and international banking groups with a view to developing a short list of preferred banks as Gold Road moves towards securing Project funding.
- Sale of potential Gruyere joint venture Project interest: To date, the Company has received a number of indicative, incomplete and non-binding proposals from selected international and domestic mining companies. The Company has not made any decision in relation to these proposals and will consider them, at the appropriate time, in the context of the Company's various funding options.

23 ADJACENT PROPERTIES

There are no other exploration or mining properties adjacent to Gold Road's Gruyere Gold property.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data and information or explanation necessary to make the Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

The FS outcomes indicate a technically sound and financially viable Project that supports the case for Project Financing and development.

The optimum case for the Project is the development of an open pit mine in four stages, with a conventional SABC, CIL process plant and associated infrastructure for throughputs of 7.5 Mtpa for fresh ore and up to 8.8 Mtpa for oxide and transition ores and blends, powered by a gas-fired power station.

25.2 Risk and Opportunity

Assessment

A detailed risk and opportunity identification and evaluation process was undertaken during the FS. The focus was on the Project development phase, with consideration given to Operational Readiness in order to manage the subsequent commissioning and handover to operations.

Gold Road is developing a whole-of-business framework (**WBF**) as part of OR that provides a clear understanding of the value drivers and operating risks inclusive of residual risks and opportunities identified within the Project and Operations (Figure 25-1).

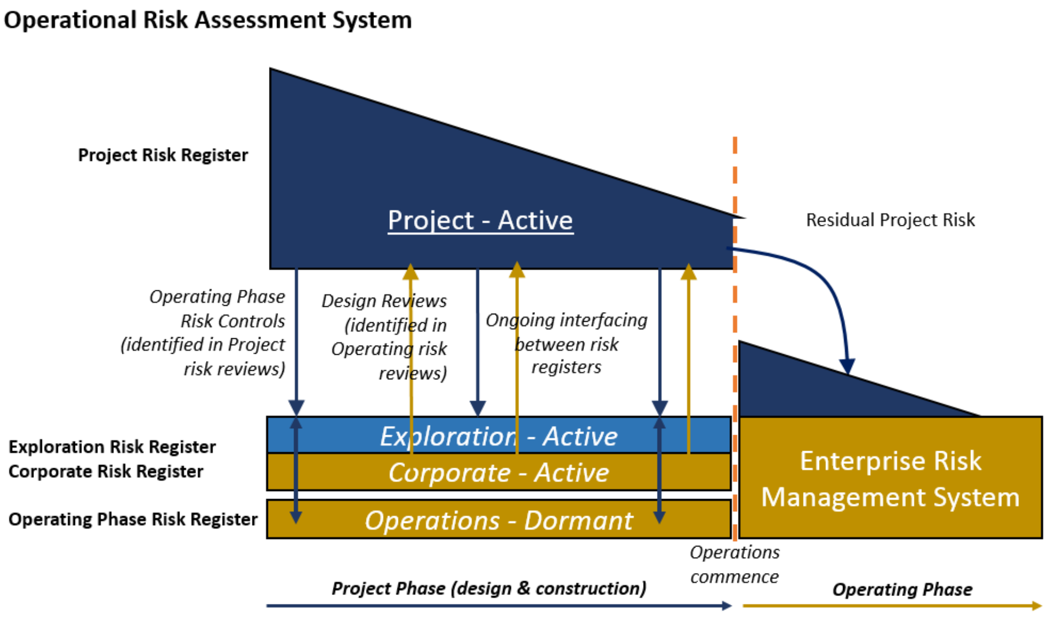

Figure 25-1: *Gruyere Project and Operating Phase Risk*

Risk assessment and management of risk on the Project will be an ongoing process.

Risks associated with corporate governance, financial management, Company strategy, Board performance, legal issues, statutory compliance, investor relations, human resources, and the like are referred to as corporate risk. Risk management across the Company has oversight at Board level through the Audit and Risk Committee, a sub-committee of the Gold Road Board. These risks, while persistent over time, will change as the Company transitions from an explorer to a developer and finally to a mine operator. The role of the Committee is to monitor, advise and, if required, to intervene in the risk management process across the Company.

Risks

From the risk assessments carried out, no fatal flaws were identified however key Project risks during the Feasibility/Commitment phase include:

- The impact of delays to Project commitment related to tenure, approvals and funding. The key approval is from the Environmental Protection Authority (**EPA**) which could delay site activities
- Delays to grant of miscellaneous licences relating to the Project linear infrastructure (i.e. borefields, access roads, gas pipeline etc.) could impact Project approvals and funding which require granted tenure as a pre-condition to submissions
- Until Project funding is finalised and the Final Investment Decision is made, funding constraints could impact execution progress
- As at 31 August 2016, the final form of the native title determination between the Yilka (the registered native title claim group) and Sullivan/Edwards (an unregistered native title claim group) had not been settled by the Federal Court. Until the final form determination is made by the Federal Court, Gold Road is unable to ascertain the effect of the judgment, if any, on the Company or its Native Title Agreement with the Yilka and any potential impact on the Project.

Potential risks during the Construction and Ramp-up phase were identified as:

- Increase in capital cost, changing of scope across mine, process plant and associated infrastructure as well as considerable non-process infrastructure establishment costs during execution
- Potential for construction delays resulting in late commissioning and ramp-up, with direct impact to the Project economics.

During the Operational phase key risks that could impact on operating margins (and return on capital), are:

- Market-related gold price risks affecting revenue; nearly 100% of Project revenue will be derived from the sale of gold with minor silver revenue generated as a by-product hence the gold price will be the single largest variable in assessing Gold Road's ability to service any debt it may have put in place
- Production level risks flowing through to unit costs; key risks in relation to production revolve around the efficiency of the operation to maximise production and minimise costs.
- Mitigation steps have been identified to reduce the potential effect on the Project outcomes with a risk management plan implemented which supports the Project schedule.
- Rigorous EPC contractor selection and expanding the capability of the Owner's team for Project delivery is an area that can reduce significant schedule and cost risk; despite commercial arrangements being proposed to address some of these execution risks, there will be a number of areas between contractors and key stakeholders which will require close supervision and management of change throughout the Project execution phase.

The gold price assumed for the FS is a flat A$1,500 per ounce per annum which is regarded as appropriately conservative in relation to the current positive sentiment towards gold.

An important philosophy adopted by Gold Road during Project evaluation was to ensure that appropriate parameters and assumptions were made in the design phase such that the Project will survive in both strong and weak gold price market scenarios. The robust economics of the Gruyere Project as derived from the FS indicates that this has been achieved. With an AISC for the life of the Project of A$945 per ounce of gold, the Project is expected to be able to generate acceptable returns throughout the range of gold prices experienced over the past five years.

Opportunities

Opportunities for adding future value will be derived from exploration, resource and reserve upgrades as well as further value engineering on the mining and process plant during the design and engineering phase.

The FS risk assessment process also identified key areas of opportunity around:

- Significant upside to the Gruyere development business case is possible with the discovery of other economic resources as a result of the ongoing regional exploration work on Gold Road's Yamarna tenements. Exploration efforts are focused on the discovery of another world class orebody which may lend itself to processing at the Project process plant. Improvements in the Gruyere orebody grade could result in significant upside to the development business case
- Further capital cost reduction following gap analysis, engineering design optimisation and through the negotiation of contract packages
- Contracting strategy developed for fewer and larger bid packages, attracting tier-1 and tier 2 contractors and negotiating risk-reward incentives for cost reduction and schedule improvements
- The current Project schedule is based on advanced procurement, allowing early procurement of long lead items and thus taking these items off the critical path. Progress is closely monitored with additional opportunities to improve on schedule, e.g. suppliers/ vendors are invited to provide suggestions and recommendations to accelerate deliveries where possible. Early commitment for engineering will create an opportunity to improve schedule
- The volatility in the price of gold, in addition to the risks already discussed, provides an opportunity to achieve superior financial returns from the Project during periods of higher gold price, and will enhance the likelihood of an increase in Project mine life either as an open cut or an underground operation.

26 RECOMMENDATIONS

Gold Road has recommended that that the Project progresses to development with commencement of the early works programme.

Limited early works have commenced following the granting of several key Miscellaneous Licenses by the DMP.

The early works include construction of the Gruyere accommodation village and an access road from the village to the main site in the December 2016 quarter. The estimated cost of these works is approximately A$18 million.

The planned key schedule milestone dates for the Project are presented in Table 26-1. Early works have commenced.

Table 26-1: Project Key Milestone

Activity Item	Start
Commence Early Works Infrastructure Engineering	Jul 2016
Accommodation Village Stage 1 Operational	Jan 2017
Project Finance in Place	Dec 2016
Site Access for Construction (All Approvals in place)	Feb 2017
EPC Contract Award	Oct 2016
Award Supply and Delivery of Long lead Equipment	Oct 2016
Bulk Earthworks Contract Award	Nov 2016
Bulk Earthworks commences on site	Feb 2017
Commence TSF Construction	Nov 2017
Commence No-Load Commissioning	Jun 2018
Commence Load Commissioning	Sep 2018
First Gold	Oct 2018
Complete Production Ramp up	Q4 2018

27 REFERENCES

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2. MJ Pawley, MTD Wingate, CL Kirkland, S Wyche, CE Hall, SS Romano & MP Doublier 2012, Adding pieces to the puzzle: episodic crustal growth and a new terrane in the north-east Yilgarn Craton, Western Australia, Australian Journal of Earth Sciences: An International Geoscience Journal of the Geological Society of Australia, vol. 59:5, pp. 603-623

3. KF Cassidy and SG Hagemann, 'World-class' Archaean orogenic gold deposits, eastern Yilgarn Craton: Diversity in timing, structural controls and mineralisation styles, Geoscience Australia, 2001

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6. Botanica Consulting Pty Ltd (2), Level 2 Flora and Vegetation Survey of the Gruyere Project, July 2015

7. Rapallo Limited, Level 2 Vertebrate Fauna Survey of the Gruyere Project Area, July 2015

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9. Martinick Bosch Sell Pty Ltd, Level 1 Subterranean Fauna Survey, Gruyere Project, August 2016

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11. Botanica Consulting Pty Ltd (1), Level 1 Fauna Assessment, White Cliffs Yamarna Road Gas Pipeline, Gold Road Resources Ltd, January 2016

12. Botanica Consulting Pty Ltd (2), Level 1 Flora and Fauna Survey of the Gruyere Borefields, June 2016

13. Bennelongia Environmental Consultants, Gruyere Gold Project: Borefields Stygofauna Assessment, March 2016

14. Axiom Project Services Pty Ltd, Estimate Review, Gruyere Feasibility Study, August 2016

15. Broadleaf Capital International Pty Ltd, Cost and Schedule Risk Analysis, August 20

APPENDIX 1 - FORWARD-LOOKING AND CAUTIONARY STATEMENTS

Some statements in this report regarding estimates or future events are forward-looking statements. They include indications of, and guidance on, future earnings, cash flow, costs and financial performance. Forward-looking statements include, but are not limited to, statements preceded by words such as “planned”, “expected”, “projected”, “estimated”, “may”, “scheduled”, “intends”, “anticipates”, “believes”, “potential”, “could”, “nominal”, “conceptual” and similar expressions. Forward-looking statements, opinions and estimates included in this announcement are based on assumptions and contingencies which are subject to change without notice, as are statements about market and industry trends, which are based on interpretations of current market conditions. Forward-looking statements are provided as a general guide only and should not be relied on as a guarantee of future performance. Forward-looking statements may be affected by a range of variables that could cause actual results to differ from estimated results, and may cause the Company’s actual performance and financial results in future periods to materially differ from any projections of future performance or results expressed or implied by such forward-looking statements. These risks and uncertainties include but are not limited to liabilities inherent in mine development and production, geological, mining and processing technical problems, the inability to obtain mine licenses, permits and other regulatory approvals required in connection with mining and processing operations, competition for among other things, capital, acquisitions of reserves, undeveloped lands and skilled personnel, incorrect assessments of the value of acquisitions, changes in commodity prices and exchange rate, currency and interest rate fluctuations, various events which could disrupt operations and/or the transportation of mineral products, including labour stoppages and severe weather conditions, the demand for and availability of transportation services, the ability to secure adequate financing and management's ability to anticipate and manage the foregoing factors and risks. There can be no assurance that forward-looking statements will prove to be correct.

Statements regarding plans with respect to the Company’s mineral properties may contain forward-looking statements in relation to future matters that can only be made where the Company has a reasonable basis for making those statements.

This announcement has been prepared in compliance with the JORC Code 2012 and the current ASX Listing Rules.

The Company believes that it has a reasonable basis for making the forward-looking statements in this announcement, including with respect to any production targets and financial estimates, based on the information contained in this announcement and in particular:

The FS which was completed by independent engineering firm, GRES and AMC, who are considered to be Western Australian experts, together with Gold Road’s Project Development Team under the direction of Sim Lau, Gold Road Project Director (BEng.(Civil) Monash University 1981). As is normal for this type of study, the FS has been prepared to an overall level of accuracy of approximately -10% to +15% .

The Company has a Mineral Resource Estimate for the Gruyere[43] Resource of 147.71 Mt at 1.30 g/t Au for 6.16 Moz (at a 0.5 g/t Au cut-off grade) of which 70%, being 104.98 Mt at 1.28g/t Au for 4.31 Moz, is classified in the Measured and Indicated Mineral Resource category under the JORC Code 2012.

The Gruyere Mineral Resource was estimated by Mr Justin Osborne and Mr John Donaldson of Perth, Western Australia in April 2016[44].

Metallurgical testwork, consistent with that required for this level of study, which forms the basis for estimates of metallurgical recoveries was managed by Gold Road’s Principal Metallurgist, Mr Max Briggs who is a competent person and a Member of The Australasian Institute of Mining and Metallurgy, and performed by a number of specialist laboratories in Australia. Based on a nominal head grade of 1.20 g/t, estimated gold recoveries for the oxide, transitional and fresh ores are 94%, 92% and 91% respectively at the target grind size of 125 µm.

The mine planning and scheduling for the 7.5 Mtpa to 8.8 Mtpa production range was supervised by Mr David Varcoe of AMC Consultants, Mr Wayne Foote, General Manager – Operations, Mr Andrew Hollis, Project Mining Manager and

[43] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, “Gruyere Resource Increases to 6.16 Million Ounces”
[44] ASX:GOR Gold Road Resources Public Disclosure, 22 April 2016, “Gruyere Resource Increases to 6.16 Million Ounces”

Mr Asam Shaibu, Principal Mining Engineer of Gold Road (mining engineers with considerable mine planning and operations experience and Members of the Australasian Institute of Mining and Metallurgy) utilising the Whittle Optimisation software (for open pit mine optimisation) and Studio 3 (for open pit mine planning). The entire mining inventory[45] is in Proved and Probable Ore Reserve categories, accounting for the entire 13 years of mine life.

GRES prepared the detailed process flowsheet based on metallurgical test work.

Geotechnical Engineering has been completed by Clive Seymour of Dempers and Seymour using modern geotechnical techniques and methods, and based on testwork consistent with this level of study. Dempers and Seymour are industry recognised experts in the field of mining geotechnical engineering.

The Project has been granted Lead Agency Status Level 2 by the Government of Western Australia. This means, by way of recognition of the size and significance of the Project to the State of Western Australia, all necessary State approval processes will be coordinated by specific individuals within the Department of Mines and Petroleum.

The Company believes that the investigations and studies carried out on the process flowsheet and the mine planning for this Study meet or exceed what would normally be expected for a FS.

Gold Road has had a very successful track record of adding Mineral Resources through greenfields and brownfields exploration across its tenements within the Yamarna Greenstone Belt. Gold Road is confident that there is a reasonable probability that it will continue to increase the Mineral Resources at the Project through exploration to extend the mine life past what is currently assumed in the FS. Attila Trend and Central Bore resources have not been contemplated in the FS. The Gruyere deposit is located in the Yamarna Greenstone Belt which is highly prospective.

The Project's positive technical and economic fundamentals provide a platform for Gold Road to advance discussions with potential strategic partners and traditional financiers. Continued support from key institutional shareholders and strategic partners, current market conditions and an encouraging outlook for the global gold market enhance the Company's view of the fundability of the Project. The Board is confident the Company will be able to finance the Project through a combination of debt and equity or strategic partnerships.

Gold Road's Board and Management team includes Managing Director and CEO, Mr Ian Murray a qualified Chartered Accountant and mining industry professional with 20 years international corporate and mining experience, Executive Director Exploration and Growth, Mr Justin Osborne a geologist with more than 26 years exploration, mining, development and corporate experience, Non-Executive Chairman, Mr Tim Netscher who has extensive mining operational, project development and business development experience primarily with the larger international mining companies, General Manager Operations, Mr Wayne Foote, a mining engineer, who has more than 29 years' experience in the mining industry, the last 16 years at senior and executive management level. Gold Road Non-Executive Director, Sharon Warburton is a highly regarded company director, who has predominantly worked in the construction, mining and infrastructure sectors throughout a career that has spanned more than 25 years.

Additional experience is added by Gruyere Steering Committee Chairman, and Consultant to the Board, Mr Robin Marshall, who has more than 40 years' experience in the Mining and Mineral Processing Industry in Project Development, Execution and Operations/Engineering.

The Board and Management are well qualified and experienced to deal with any funding and project development challenges as they occur. In addition, the current state of the mining professional labour market is such that expert specialist input, when required, is available in Western Australia and can be sourced by Gold Road on a part-time or full-time basis.

[45] ASX:GOR Gold Road Resources Public Disclosure, 19 October 2016, "Gruyere Feasibility Study Approved"

The Study is based on the assumption that all gold produced will be refined at and sold to the Perth Mint, a statutory authority of the Government of Western Australia. The Perth Mint refines almost all gold doré bars produced in Western Australia. The gold market is a highly liquid international market with no need for offtake agreements.

PREVIOUSLY REPORTED INFORMATION

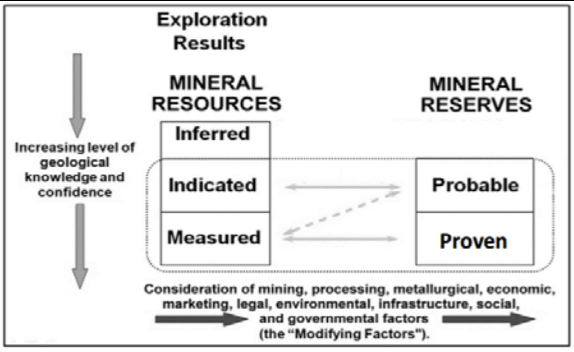
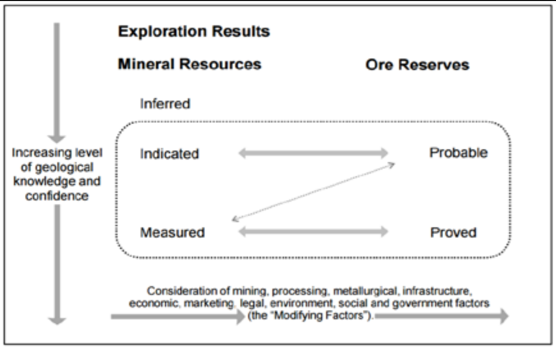
This annoucement includes information that relates to Mineral Resources and exploration results which were prepared and first disclosed under the JORC Code 2012. This information was included in the Company's previous annoucements as follows:

- ASX announcement dated 4 August 2014, Maiden Gruyere Resource
- ASX announcement dated 15 October 2014, Annual Report To Shareholders
- ASX announcement dated 20 January 2015, Mineralisation At Gruyere Extended To 750 Metres Depth
- ASX announcement dated 27 January 2015, Gruyere Scoping Study confirms long life Gold Project
- ASX announcement dated 28 May 2015, Gruyere Resource Grows To 5.51 Million Ounces Gold
- ASX announcement dated 3 August 2015, Gruyere Pre-Feasibility Study Stage 1 completed
- ASX announcement dated 16 September 2015, Gruyere Resource Increases To 5.62 Million Ounces
- ASX announcement dated 7 February 2016, Gruyere Pre-Feasibility Study confirms long life Gold Mine
- ASX announcement dated 22 April 2016, Gruyere Resource Increases To 6.16 Million Ounces (JORC Code 2012 Table 1 Sections 1 to 3 republised in Appendix 4)
- ASX announcement dated 19 October 2016, Gruyere Feasibility Study Approved (JORC Code 2012 Table 1 Sections 1 to 4 republised in Appendix 3).
- ASX announcement dated 7 November 2016, Gruyere Gold Project to be Developed in Joint Venture with Gold Fields LTD

These announcements are available at the Company's website www.goldroad.com.au.

The Company confirms that it is not aware of any new information or data that materially affects the information included in the original market announcements and, in the case of estimates of Mineral Resources and Ore Reserves, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcements continue to apply and have not materially changed. The Company confirms that the form and context in which the Competent Persons' findings are presented have not materially changed from the original market announcement.

APPENDIX 1 - COMPATIBILITY OF CIM DEFINITIONS AND STANDARDS (2014) AND JORC CODE 2012

	CIM Definitions and Standards (2014)	JORC Code 2012
Figure 1		
Mineral Reserves/Ore Reserves	**Mineral Reserves:** A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.	**Ore Reserves:** An Ore Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.
Proven/	**Proven:** "A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors."	**Proved:** "A 'Proved Ore Reserve' is the economically mineable part of a Measured Mineral Resource. A Proved Ore Reserve implies a high degree of confidence in the Modifying Factors"

	CIM Definitions and Standards (2014)	JORC Code 2012
Expertise	Qualified Person (defined in NI43-101): "qualified person" means an individual who (a) is an engineer or geoscientist with a university degree, or equivalent accreditation, in an area of geoscience, or engineering, relating to mineral exploration or mining; (b) has at least five years of experience in mineral exploration, mine development or operation, or mineral project assessment, or any combination of these, that is relevant to his or her professional degree or area of practice; (c) has experience relevant to the subject matter of the mineral project and the technical report; (d) is in good standing with a professional association; and (e) in the case of a professional association in a foreign jurisdiction, has a membership designation that (i) requires attainment of a position of responsibility in their profession that requires the exercise of independent judgment; and (ii) requires A. a favourable confidential peer evaluation of the individual's character, professional judgement, experience, and ethical fitness; or B. a recommendation for membership by at least two peers, and demonstrated prominence or expertise in the field of mineral exploration or mining;	**Comeptent Person**: A Comeptent Person is a minerals industry professional who is a Member or Fellow of The Australiasian Institute of Mining and Metallurgy, or of the Australian Institute of Geoscientists, or of a 'Recognised Professional Organisation' (RPO), as included in a list available on the JORC and ASX websites. These organisations have enforceable disciplinary processes including the powers to suspend or explel a member. A Competent Person must have a minimum of five years relevant experience in the style of mineralisation or type of deposit under consideration and in the activity which that person is undertaking. If the Competent Person is preparing documentation on Exploration Results, the relevant experience must be in exploration. If the Competent Person is estimating, or supervising the estimation of Mineral Resources, the relevant experience must be in the estimation, assessment and evaluation of Mineral Resources. If the Competent Person is estimating, or supervising the estimation of Ore Reserves, the relevant experience must be in the estimation, assessment, evaluation and economic extraction of Ore Reserves.
Independence	Independent Technical Report required (NI43-101, s5.3) for first time mineral resource or mineral reserves, or more than a 100% change in total mineral resources or mineral reserves, unless a producing issuer or in a joint venture with a producing issuer.	Discloure of the full nature of the relationship between Competent Person and the reporting entity, including any issue that could be perceived by investors as conflict of interest.

APPENDIX 3 - JORC CODE 2012 TABLE 1

Section 4 Estimation and Reporting of Ore Reserves

The information below was previously presented in the ASX announcement dated 19 October 2016.

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code (2012) explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	The Mineral Resource estimate for the Gruyere deposit which formed the basis of this Ore Reserve estimate was compiled by the Gold Road Competent Person(s) utilising relevant data. The estimate is based on 357 Reverse Circulation (RC) holes and 113 diamond holes of exploration drilling and assay data. The data set, geological interpretation and model was validated using Gold Road's internal and Quality Assurance and Quality Control (QAQC) processes and reviewed by an independent external consultant. Ordinary Kriging was utilised to estimate the Measured component of the resource and Localised Uniform Conditioning was utilised to estimate the Indicated and Inferred components of the resource. The individual block size for estimation was 5 mE x 12.5 mN x 5 mRL for both methods. The Mineral Resources are reported inclusive of the Ore Reserve (refer ASX announcement 22 April 2016).
Site visits	Comment on any site visits undertaken by the Competent Person and the outcome of those visits. If no site visits have been undertaken indicate why this is the case.	The Competent Person conducted a site visit in October 2015. The following activities were completed: ▪ Gained general familiarisation with the site including likely mining conditions, proposed pit location, waste dump location, site drainage and site access ▪ Assessed proposed locations of mining related infrastructure relative to the designed open pit ▪ Observed resource drilling activities ▪ Inspected air core drill hole sites to get an understanding of the variations in weathering profiles across the deposit ▪ Viewed diamond drill core from selected samples.
Study status	The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves. The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	The Ore Reserve estimate is the result of a detailed Feasibility Study (FS) completed by a team consisting of Gold Road personnel and independent external consultants. The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology which is widely utilised in the goldfields of Western Australia (WA). Financial modelling completed as part of the FS shows that the project is economically viable under current assumptions. Material Modifying Factors (mining, processing, infrastructure, environmental, legal, social and commercial) have been considered during the Ore Reserve estimation process.

Criteria	JORC Code (2012) explanation	Commentary
Cut-off parameters	The basis of the cut-off grade(s) or quality parameters applied.	Variable economic cut-off grades have been applied in estimating the Ore Reserve. Cut-off grade is calculated in consideration of the following parameters: ▪ Gold price ▪ Operating costs ▪ Process recovery ▪ Transport and refining costs ▪ General and administrative cost ▪ Royalty costs.
Mining factors or assumptions	The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.	Gruyere will be mined by open pit mining methods utilising conventional mining equipment. Final pit and interim stage designs were completed as part of the FS. The final pit design is the basis of the Ore Reserve estimate. The selected mining method, design and extraction sequence are tailored to suit orebody characteristics, minimise dilution and ore loss, defer waste movement and capital expenditure, utilise proposed process plant capacity and expedite free cash generation in a safe and environmentally sustainable manner. Mining operating and capital costs were estimated as part of the FS and referenced against contractor budget quotes.
	The assumptions made regarding geotechnical parameters (e.g. pit slopes, stope sizes, etc), grade control and pre-production drilling.	Geotechnical modelling has been completed by an external consultant on the basis of field logging and laboratory testing of selected dedicated diamond drill core samples. The recommended geotechnical design parameters assume dry slopes on the basis of adequate dewatering ahead of mining. Eleven geotechnical domains were identified: ▪ Domain West 1: - Weathered material: batter heights of 10m, batter angles of 50° - 55° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 75° and berm widths of 9m. ▪ Domain West 2AN: - Weathered material: batter heights of 10m, batter angles of 55° - 60° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 70° - 80° and berm widths of 6m. ▪ Domain West 2B: - Weathered material: batter heights of 10m, batter angles of 55° - 60° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 12m. ▪ Domain West 2AS: - Weathered material: batter heights of 10m, batter angles of 55° - 60° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 6m. ▪ Domain West 3, East 4: - Weathered material: batter heights of 10m, batter angles of 55° - 60° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 9m. ▪ Domain West 4: - Weathered material: batter heights of 10m, batter angles of 50° - 55° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 8m. ▪ Domain East 1: - Weathered material: batter heights of 10m, batter angles of 50° - 55° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 9m. ▪ Domain East 2:

Criteria	JORC Code (2012) explanation	Commentary
		- Weathered material: batter heights of 10m, batter angles of 55° - 60° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 8m. ▪ Domain East 3: - Weathered material: batter heights of 10m, batter angles of 55° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 60° - 80° and berm widths of 11m. ▪ Domain East 5: - Weathered material: batter heights of 10m, batter angles of 55° and berm widths of 5m - Fresh material: batter heights of 20m, batter angles of 55° and berm widths of 6m. A separate hydrogeological report was prepared by independent consultants which considered the infrastructure required to effectively dewater the open pit and pit slopes. This study was supported by the development of test bores and field test pumping analysis.
	The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate). The mining dilution factors used. The mining recovery factors used. Any minimum mining widths used.	Mining dilution and recovery modifying factors were simulated by modelling to a Selective Mining Unit (SMU) then applying a dilution skin at each ore to waste contact across the orebody, and then re-estimating the resultant tonnes and grades of neighbouring blocks due to the impact of including dilution at that contact. A configuration of 5 mE x 12.5 mN x 5 mRL with a 0.5 m dilution skin was applied which represents the capability of the selected mining fleet. The modelling yielded the following results: ▪ Mining tonnage dilution of 3.2% ▪ Mining grade dilution of 4.6% ▪ Mining recovery factor of 98.6% (gold loss of 1.4%) These values reflect the fact that Gruyere is a relatively simple continuous orebody with individual ore shape designs of hundreds of metres along strike and 20 to 50 m wide.
Mining factors or assumptions	The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.	The mining schedule is based on supplying variable throughput rates to a processing plant with a name plate capacity of 7.5 Mtpa for fresh ore material with the capability to treat up to 8.0 Mtpa of transition material and up to 8.8 Mtpa of oxide material. The mining schedule is based on realistic mining productivity and equipment utilisation estimates and also considered the vertical rate of mining development. Inferred Mineral Resources were considered as waste during the pit optimisation and production scheduling process. Waste material from mining activities will be disposed of as follows: ▪ Topsoil will be disposed of at designated stockpiles for application in on-going rehabilitation activities; ▪ Initial saprolite waste will be utilised to construct the base and starter embankment of the Tailings Storage Facility (TSF); ▪ Some waste rock will be utilised to construct the Run Of Mine (ROM) pad; ▪ Some waste rock will be utilised to construct on-going TSF lifts; ▪ Excess waste rock will be disposed of at designated waste rock dumps.
	The infrastructure requirements of the selected mining methods.	The proposed mine plan includes waste rock dumps, a ROM pad, a surface water diversion channel, surface dewatering bores, light and heavy vehicle workshop facilities, explosives storage and supply facilities and technical services and administration facilities.

Criteria	JORC Code (2012) explanation	Commentary
Metallurgical factors or assumptions	The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. Whether the metallurgical process is well-tested technology or novel in nature. The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. Any assumptions or allowances made for deleterious elements. The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	A processing flowsheet, materials balance, water balance, equipment identification, mechanical and electrical layouts were all developed to FS standard. A single stage primary crush, Semi Autogenous Grinding and Ball Milling with Pebble Crushing (SABC) comminution circuit followed by a conventional gravity and carbon in leach (CIL) process is proposed. This process is considered appropriate for the Gruyere ore, which is classified as free-milling. The proposed metallurgical process is commonly used in the Australian and international gold mining industry and is considered to be well-tested and proven technology. Significant comminution, extraction, and materials handling testing has been carried out on approximately 2,000kg of half-NQ (NQ core diameter = 47.6mm) diamond drilling core samples, and 480kg of RC chip samples. This has been carried out on oxide, saprock, transitional, and fresh ore types which were obtained across the Gruyere deposit (South to North) and to a depth of approximately 300m. Estimated plant gold recovery ranges from 87% to 95% depending on head grade, plant throughput, grind size and ore type. Significant comminution, extraction, and materials handling testing has been carried out on material selected from approximately 2,000kg of half-NQ core. No deleterious elements of significance have been determined from metallurgical test work and mineralogy investigations.
Environmental	The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	Baseline environmental studies of flora, vegetation, vertebrate fauna, short-range endemic invertebrates and subterranean fauna are all completed. Environmental approvals for the mining and water supply aspects of the project will be assessed by the EPA and the Department of Mines and Petroleum WA (DMP). The approvals document to EPA has and the approvals document to the DMP will be submitted in Q4 2016. Environmental approvals for the gas pipeline aspect of the project has been assessed by the EPA, and will be assessed by the DMP for a petroleum pipeline licence and clearing permit in 2017. Waste rock and tailings characterisation work has been completed and all waste types and tailings are non-acid forming and have limited metal leachate potential. Waste rock and tailings storage locations have been selected based on suitable geographical characteristics and proximity to the pit and plant.
Infrastructure	The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	The project site is within economic distances of existing infrastructure of the Eastern Goldfields region. Services and consumable supplies will be delivered by existing roads from Laverton some 150km to the west. A gas supply lateral from the Eastern Goldfields Pipeline will be built from Laverton to site to supply gas to a purpose built gas fired power station. The workforce will be Fly In-Fly Out (FIFO) and based at a camp on site during rostered days on. An on-site airstrip is to be built as part of the project. Pump testing and modelling of the potential yield from the Yeo and Anne Beadell borefields indicate that there is sufficient groundwater to service the needs of the Project for the life-of-mine. The primary source of water for the project will be developed over approximately 65 km of tested palaeochannel. In addition to the tested palaeochannel length, approximately 100 km of palaeochannel is available for potential development on tenements with granted miscellaneous water search licences. Miscellaneous licence applications have been lodged to secure the tenure required for the water and gas pipelines and a new section of road for site access. Granting of the remaining miscellaneous licence applications for the Yeo borefield is expected in Q4 2016 and for the gas pipeline infrastructure is expected in Q1 2017.
Costs	The derivation of, or assumptions made, regarding projected capital costs in the study	All capital estimates are based on market rates as at the second quarter of 2016.

Criteria	JORC Code (2012) explanation	Commentary
		It is assumed that all mining equipment required for the project will be supplied by a mining contractor. It is assumed that power infrastructure will be supplied by a third party under a Build-Own-Operate arrangement to supply power at a cost to the project. The capital cost estimate accuracy is -10% /+15%. Mine development costs were developed from a combination of inputs from Gold Road, AMC Consultants, GR Engineering Services and Pennington Scott Hydrogeologists. The basis of estimate is: ▪ Contract mining ▪ Mobilisation of mining equipment and personnel from Perth ▪ Earthworks quantities determined from detailed site inspections by a competent civil engineer and geological modelling ▪ Mine dewatering requirements developed from FS level hydrogeological modelling ▪ A mining schedule developed on a quarterly basis ▪ A contingency allowance on capital cost items calculated to reflect the relevant level of confidence in the estimate Processing and infrastructure development capital costs have been estimated by GR Engineering Services (GRES) on the basis of: ▪ Earthworks quantities determined from detailed site inspections by a competent civil engineer ▪ Concrete and structural quantities developed from site layouts and similar designs from other projects ▪ A mechanical equipment list developed from the recommended process design criteria ▪ Budget pricing from local and international suppliers ▪ Contingency allowances calculated on a line by line basis relevant to the source and confidence in market rates
Costs	The methodology used to estimate operating costs.	The operating cost estimate accuracy is -10% /+15%. Operating costs assume a FIFO scenario with various rosters on site. Mining operating costs have been estimated by AMC on the basis of scheduled material movement and mining rates for a contractor mining scenario with technical services supplied by Gold Road employees. Mine design and scheduling was prepared by competent mining engineers. Process and infrastructure operating costs have been estimated by GRES on the assumption that: ▪ A conventional SABC circuit will be utilised to treat ore at a rate of 7.5 Mtpa for fresh ore with the capability to treat up to 8.8 Mtpa of oxide material ▪ Comminution grind sizes will be in the range of 106µm to 150µm for all material types ▪ Power will be generated on site utilising gas delivered by pipeline ▪ The process plant will be operated by Gold Road employees. The operating cost estimate is considered to be appropriate for the current market in the eastern goldfields of WA.
	Allowances made for the content of deleterious elements.	No allowance is made for deleterious elements since testwork to date on ore from Gruyere has not shown the presence of deleterious elements.
	The source of exchange rates used in the study.	Capital Costs for process plant and infrastructure are estimated in 2016 Australian dollars.

<table>
<tr><th>Criteria</th><th>JORC Code (2012) explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>Foreign currency exchange rates were derived as tabled below.
<table>
<tr><th></th><th>ate (A$1 = X)</th><th>Source</th></tr>
<tr><td>United States Dollar</td><td>0.75</td><td>Gold Road</td></tr>
<tr><td>Euro</td><td>0.66</td><td>online</td></tr>
<tr><td>Chinese Renminbi</td><td>4.87</td><td>online</td></tr>
</table></td></tr>
<tr><td></td><td>The derivation of, or assumptions made, regarding projected capital costs in the study. Derivation of transportation charges.
The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.
The allowances made for royalties payable, both Government and private.</td><td>Transport charges - Gold bullion transportation charges are derived on the basis of a quote provided by a leading industry bullion shipment organisation.
Treatment and refining charges are estimated on the basis of a quote from a leading Perth Gold Refinery.
An allowance has been made for all royalties, including an allowance of 2.5% of revenue for royalties payable to the Western Australian State Government and an allowance for other royalties payable to private parties (these royalties being commercially sensitive and covered by confidentiality).</td></tr>
<tr><td>Revenue factors</td><td>The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.
The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.</td><td>The mined ore head grades are estimated utilising industry accepted geostatistical techniques with the application of relevant mining modifying factors.
Gold price and exchange rates have been determined by an external financial expert group on the basis of current market trends.
A Life-of-mine (LOM) gold price forecast of A$1,500/oz (Real 2016) is applied in the financial modelling for the Ore Reserve calculation process. This price forecast was established by Gold Road on the basis of historical A$ gold price trends over the last 5 years. Over that review period the price of gold has ranged between A$1,300/oz and A$1,800/oz and averaged approximately A$1,500/oz.</td></tr>
<tr><td>Market assessment</td><td>The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.
A customer and competitor analysis along with the identification of likely market windows for the product.
Price and volume forecasts and the basis for these forecasts.
For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.</td><td>There is a transparent market for the sale of gold.</td></tr>
<tr><td>Economic</td><td>The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.
NPV ranges and sensitivity to variations in the significant assumptions and inputs.</td><td>

Discounted cash flow modelling and sensitivity analysis has been completed to evaluate the economic performance of the Ore Reserve. Key value driver inputs into the financial model included:

Gold price at A$1,500/oz based on historical trends over the last 5 years.
Discount rate of 8% as determined by the Board of Directors of Gold Road.

The Ore Reserve returns a positive NPV of A$305M (post-tax) under the assumptions detailed herein.
The table below shows the results of sensitivity analysis on key project variables.

<table>
<tr><th>n Variable</th><th>-10%</th><th>0%</th><th>10%</th></tr>
<tr><td></td><td colspan="3">% Change in Project NPV (Post-Tax)</td></tr>
<tr><td></td><td>-59%</td><td>-</td><td>59%</td></tr>
<tr><td>covery</td><td>-59%</td><td>-</td><td>59%</td></tr>
<tr><td>t</td><td>16%</td><td>-</td><td>-16%</td></tr>
</table></td></tr>
</table>

Criteria	JORC Code (2012) explanation	Commentary
		<table><tr><td>Cost</td><td>18%</td><td>-</td><td>-18%</td></tr><tr><td>nt Capex</td><td>12%</td><td>-</td><td>-12%</td></tr></table> ▪ The project NPV (Post Tax) is most sensitive to variations in the gold price and process recovery. ▪ A 10% reduction in gold price or process recovery reduces NPV by 59%. A 10% increase in gold price or process recovery increases NPV by 59%.
Social	The status of agreements with key stakeholders and matters leading to social licence to operate.	A Native Title Mining Agreement has been signed for the Project (ASX Announcement 4 May 2016: Historic Native Title Agreement In Place for Gruyere Project). Subsequent to the Native Title Agreement, a Mining Lease was granted over the project area (ASX Announcement 9 May 2016: Yamarna Mining Leases Granted). Several key miscellaneous licences have also been granted (ASX Announcement 29 September 2016: Gruyere Gold Project to Commence Limited Early Works).
Other	To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: Any identified material naturally occurring risks. The status of material legal agreements and marketing arrangements. The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	▪ Flooding risk has been analysed by an independent external expert and appropriate mitigation strategies have been included in the FS. ▪ No significant species have been identified that would be significantly impacted by the Project in a manner that could not be adequately managed. ▪ Mining and gas pipeline contract negotiations have commenced. There are reasonable prospects to anticipate that contract terms as assumed in the Ore Reserves estimate will be achieved. Project commissioning is estimated for 2018.
Classification	The basis for the classification of the Ore Reserves into varying confidence categories. Whether the result appropriately reflects the Competent Person's view of the deposit. The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	The main basis of classification of Ore Reserves is the underlying Mineral Resource classification. All Proved Ore Reserves derive from Measured Mineral Resources and all Probable Ore Reserves derive from Indicated Mineral Resources in accordance with JORC Code 2012 guidelines. The results of the Ore Reserve estimate reflect the Competent Person's view of the deposit. No Probable Ore Reserves are derived from Measured Mineral Resources. No inferred Mineral Resource is included in the Ore Reserves. 16% of the Ore Reserve is in the Proved category with the balance (84%) being Probable.
Audits or reviews	The results of any audits or reviews of Ore Reserve estimates.	The FS which forms the basis of the Ore Reserve estimate was subjected to various reviews and audits: ▪ Metallurgical testwork was reviewed by Gold Road metallurgists and process engineers and confirmed to be adequate for a FS. ▪ Geotechnical input was reviewed by external independent consultants and found to be acceptable for a FS. ▪ Open pit designs, production schedules and mining cost models were reviewed through AMC's internal peer review system and externally by an independent technical expert. ▪ The basis of design for the process plant and infrastructure was reviewed by Gold Road metallurgists and process engineers and was deemed appropriate for a FS.

Criteria	JORC Code (2012) explanation	Commentary
		▪ Capital cost estimates were reviewed by an external independent consultant and were considered to be appropriate for a FS. ▪ The financial model applied for project valuation was reviewed by Gold Road financial accountants and was considered to be appropriate for a FS. ▪ The overall FS was reviewed by an independent technical expert and was considered to be appropriate.
Discussion of relative accuracy/ confidence	Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	The Gruyere FS resulted in a technically robust and economically viable business case. This is deemed to be an appropriate basis for a high level of confidence in the Ore Reserves estimate. In the opinion of the Competent Person, cost assumptions and modifying factors applied in the process of estimating Ore Reserves are reasonable. Gold price and exchange rate assumptions were set out by Gold Road and are subject to market forces and present an area of uncertainty. In the opinion of the Competent Person, there are reasonable prospects to anticipate that all relevant legal, environmental and social approvals to operate will be granted within the project timeframe.

Section 1 Sampling Techniques and Data

The information below was previously presented in ASX announcements dated 19 October 2016 and 22 April 2016. The data for the 25 by 25 metre RC program has not been publicly released as it is considered to be operational in nature. These holes were treated with the same geological protocols as described in Table 1 below.

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	*Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.*	The sampling has been carried out using a combination of Reverse Circulation (**RC**) and Diamond Drilling (**DDH**). RC drill samples are collected through a rig-mounted cone splitter designed to capture a one metre sample with optimum 3 to 4kg sample weight. Drill core is logged geologically and marked up for assay at approximate one metre intervals based on geological observation. Drill core is cut in half by a diamond saw and half core samples submitted for assay analysis. Detailed descriptions of drilling orientation relative to deposit geometries, and full sample nature and quality are given below.
	Include reference to measures taken to ensure sample representation and the appropriate calibration of any measurement tools or systems used.	Sampling was carried out under Gold Road's protocols and QAQC procedures as per industry best practice. See further details below.
	Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	RC holes were drilled with a 5.25 inch face-sampling bit, 1 m samples were collected through a cyclone and cone splitter to form a 2-4 kg sample. All holes with reported assays from RC drilling are from the original 1 m samples collected from the splitter except for 1% of RC samples, which were four metre composite samples collected through logged waste zones. The 4 m composite samples were created by spear sampling of the total 1 m samples collected in large plastic bag from the drilling rig and were deposited into separate numbered calico bags for sample despatch. No assays collected by four metre composite sampling were used in the Resource estimation. Diamond drilling was completed using an HQ or NQ drill bit for all holes. Core is cut in half for sampling, with a half core sample sent for assay at measured intervals. Both RC and diamond samples were fully pulverised at the laboratory to -75um, to produce a 50g charge for Fire Assay with an AAS finish up until May 2014 and ICPES finish post this date.
Drilling techniques	*Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	RC drilling rigs, owned and operated by Raglan Drilling, were used to collect the RC samples. The face-sampling RC bit has a diameter of 5.25 inches (13.3 cm). Diamond drilling rigs operated by Terra Drilling Pty Ltd collected the diamond core as NQ or HQ size. The majority of diamond holes used RC pre-collars to drill through barren hanging-wall zones to specified depth, followed by diamond coring at NQ size from the end of the pre-collar to the end of hole. This ensured diamond core recovery through the mineralised zones within the Gruyere Porphyry. Core is oriented using downhole Reflex surveying tools, with orientation marks provided after each drill run.

Criteria	JORC Code explanation	Commentary
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.*	The majority of RC samples were dry. Ground water egress occurred in some holes at variable depths between 100 and 400 m. Drill operators ensured that water was lifted from the face of the hole at each rod change to ensure that water did not interfere with drilling and that all samples were collected dry. When water was not able to be isolated from the sample stream the drill hole was stopped and drilling was completed with a diamond tail. RC recoveries were visually estimated, and recoveries were recorded in the log as a percentage. Recovery of the samples was good, generally estimated to be close to 100%, except for some sample loss at the top of the hole. All diamond core collected is dry. Drill operators measure core recoveries for every drill run completed using a 3 m core barrel. The core recovered is physically measured by tape measure and the length recovered is recorded for every 3 m "run". Core recovery is calculated as a percentage recovery. Close to 100% recoveries were achieved for the majority of diamond drilling completed at Gruyere.
	Measures taken to maximise sample recovery and ensure representative nature of the samples.	RC face-sampling bits and dust suppression were used to minimise sample loss. Drilling air pressure lifted the water column above the bottom of the hole to ensure dry sampling. RC samples were collected through a cyclone and rotary cone splitter. The rejects were deposited in a large plastic bag and retained for potential future use. The sample required for assay is collected directly into a calico sample bag at a designed 3 - 4 kg sample mass which is optimal for whole-of-sample pulverisation at the assay laboratory. Diamond drilling results in uncontaminated fresh core samples which are cleaned at the drill site to remove drilling fluids and cuttings to present clean core for logging and sampling.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	All RC samples were dry with the exception of a few samples (<5%) that were reported as slightly damp to the end of the hole. Apart from the tops of the holes while drilling through the sand dune cover, there is no evidence of excessive loss of material and at this stage no information is available regarding possible bias due to sample loss. There is no significant loss of material reported in any of the Diamond core.
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All chips and drill core were geologically logged by Gold Road geologists, using the Gold Road logging scheme. This provides data to a level of detail adequate to support Mineral Resource Estimation activities. Approximately 30% of holes have been surveyed using downhole optical (OTV) and/or acoustic (ATV) televiewer tools which provide additional information suitable for geotechnical and specific geological studies. A full set (49,425 to 50,950 mN) of 25 m spaced manually interpreted cross-sections were geo-referenced and used to guide digital construction of material type wireframes. A weathering profile guide was developed as part of the process in order to document the features and provide a guide for further logging and open pit mapping. Nine specific geotechnical diamond holes were drilled to support the PFS and a further 12 drilled to support the FS. The holes were designed and logged in geotechnical detail by Dempers and Seymour Pty Ltd Geotechnical Mining Consultants. Collaboration between the geological and geotechnical groups has resulted in refinement of the geological interpretation, particularly the understanding of significant faults and shear zones. Metallurgical composite samples selected over the life of the project have been based on the detailed logging information, gold grades and geological interpretation.

Criteria	JORC Code explanation	Commentary
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	Logging of RC chips records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples. All samples are wet-sieved and stored in a chip tray. Logging of drill core records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples, along with structural information from oriented drill core. All samples are stored in core trays. All core is photographed in the trays, with individual photographs taken of each tray both dry, and wet; all photos are uploaded to and stored in the Gold Road server database.
	The total length and percentage of the relevant intersections logged	All RC and diamond holes were logged in full.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.*	Core samples were cut in half using an automated Corewise diamond saw. Half core samples were collected for assay, and the remaining half core samples are stored in the core trays.
	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	One metre RC drill samples are collected via a rotary cone-splitter, installed directly below a rig mounted cyclone, and an average 2-3 kg sample is collected in an un-numbered calico bag, and positioned on top of the plastic bag. >95% of samples were collected dry (dry to slightly damp). Four-metre composite samples were created by spear sampling of the total one metre samples collected in large plastic bag from the drilling rig and deposited into separate numbered calico bags for sample despatch. A number of RC holes utilised 4 m composite samples for waste intervals. *If composite samples returned anomalous gold values, the intervals were resampled as one metre samples by collecting the sample produced from the rotary cone-splitter.* No 4 m sample assays were used in this Mineral Resource Estimate.
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	Samples were prepared at the Intertek Laboratory in Kalgoorlie. Samples were dried, and the whole sample (both RC and DDH) was pulverised to 80% passing 75um, and a sub-sample of approx. 200g was retained. A nominal 50g was used for the analysis. The procedure is better than industry standard for this type of sample as most labs split the 2-3 kg prior to pulverising.
	Quality control procedures adopted for all sub-sampling stages to maximise representation of samples.	A duplicate RC field sample is taken from the cone splitter at the same time as the primary sample a rate of approximately 1 in 40 samples. A twinned half core sample is taken at a frequency of 1 in 40 samples, with one half representing the primary result and the second half representing a twinned result. At the laboratory, regular laboratory-generated repeats and check samples are assayed, along with laboratory insertion of its own standards and blanks.
	Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	Duplicate samples were collected at a frequency of 1 in 40 for all drill holes. RC duplicate samples are collected directly from the rig-mounted rotary cone splitter. Core duplicate samples utilise the second half of core after cutting.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	Sample sizes are considered appropriate to give an indication of mineralisation given the particle size and the preference to keep the sample weight below a targeted 3kg mass which is the optimal weight to ensure the requisite grind size in the LM5 sample mills used by Intertek in sample preparation.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td rowspan="3">Quality of assay data and laboratory tests</td><td>The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.</td><td>Samples were analysed at the Intertek Laboratory in Perth. The analytical methods used for RC and diamond drilling methods for raw (not composited) samples in a 10km square region surrounding the deposit were as follows:
<table>
<tr><th>Azimuth (Gruyere Grid)</th><th>DDH</th><th>RC</th><th>Total</th></tr>
<tr><td>50 gram Fire Assay with AAS finish</td><td>6,295</td><td>13,888</td><td>20,183</td></tr>
<tr><td>50 gram Fire Assay with ICPES finish</td><td>17,206</td><td>20,337</td><td>37,543</td></tr>
<tr><td>Total</td><td>23,501</td><td>34,225</td><td>57,726</td></tr>
</table>
Fire Assay with either AAS or ICPES finish for gold is considered to be appropriate for the Gruyere material and mineralisation. The method gives a near total digestion of the material intercepted in diamond core drilling. ICPES provides improved quality compared to AAS and all fire assay protocols for Gold Road samples were changed to this finish during May 2014.</td></tr>
<tr><td>For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.</td><td>Calibration of the hand-held XRF tools is applied at start-up. XRF results are only used for indicative assessment of lithogeochemistry and alteration to aid logging and subsequent interpretation.
Downhole survey of rock property information for selected holes reported has been completed. ABIMS is the contractor which compiled this work. This involved downhole surveying using a variety of tools with real time data capture and validation. The tools were calibrated on a regular basis. This data was used in conjunction with other data in the determination of specific gravity (SG) data for the Resource Model.</td></tr>
<tr><td>Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.</td><td>The Gold Road protocol for RC programs is for Field Standards (Certified Reference Materials) and Blanks to be inserted at a rate of 3 Standards and 3 Blanks per 100 samples. RC Field Duplicates and DDH Field Twins are generally inserted at a rate of approximately 1 in 40. Samples are processed at Intertek Laboratories, where regular assay Repeats, Laboratory Standards, Checks and Blanks are inserted and analysed in addition to the blind Gold Road QAQC samples.
For the reported resource the relevant assays and QAQC numbers are as follows:
<table>
<tr><th rowspan="2">Assay and QAQC Numbers</th><th colspan="2">April 2016</th></tr>
<tr><th>Number</th><th>Comment</th></tr>
<tr><td>Total Sample Submission</td><td>58,137</td><td></td></tr>
<tr><td>Field Blanks</td><td>1,536</td><td></td></tr>
<tr><td>Field Standards</td><td>1,526</td><td></td></tr>
<tr><td>Filed Duplicates</td><td>1,148</td><td></td></tr>
<tr><td>Laboratory Blanks</td><td>1,259</td><td>including 98 Acid Blanks</td></tr>
<tr><td>Laboratory Checks</td><td>1,855</td><td></td></tr>
<tr><td>Laboratory Standards</td><td>1,868</td><td></td></tr>
<tr><td>Umpire Checks - Minanalytical</td><td>236</td><td>including 5 Laboratory Blanks and 10 Laboratory Standards</td></tr>
<tr><td>Umpire Checks - ALS Laboratories</td><td>62</td><td>including 4 Laboratory Blanks and 6 Laboratory Standards</td></tr>
</table></td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		Results of the Field and Laboratory QAQC assays were checked on assay receipt using QAQCR software. All assays passed QAQC protocols, showing acceptable levels of contamination or sample bias, including diamond half core v. half core Field Twins. QAQC Audits for each major drill program and associated resource update have been completed and reported by Mr David Tullberg (Grassroots Data Services Pty Ltd) and by Dr Paul Sauter (in-house consultant Sauter Geological Services Pty Ltd).
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.*	Significant results were compiled by the Database Manager and reported for release by the Exploration Manager/Executive Director. Data was routinely checked by the Senior Exploration and Project Geologist, Principal Resource Geologist or Consulting Geologists during drilling programs. All results, except for the 25 by 25 m RC data, which is considered operational, have been reported in ASX announcements listed in Appendix 2.
	The use of twinned holes.	Three twin RC holes were completed and data analysed in the reported resource, with their collars being less than 5 metres distant from the parent collar. ▪ 14GYRC0026A (twin pair with hole 13GYRC0026) ▪ 14GYRC0033A (twin pair with hole 14GYRC0033) ▪ 14GYRC0060A (twin pair with hole 13GYRC0060) Two twin RC vs DDH sub-parallel holes were completed and data analysed in the reported resource, with their collars being less than 10 metres distant from the parent collar. ▪ 13GYDD0003 (twin pair with hole 13GYRC0027) ▪ 13GYDD0002 (twin pair with hole 13GYRC0049) One diamond pair (14GYDD0012A and 14GYDD0012B) provide a twin data set over a length of 120 m at a spacing of less than less than 4 m apart. This twinned data provided accurate data for validating the nugget effect at Gruyere. As part of the Maiden Mineral Resource reported in August 2014 a detailed drill program was completed which included a number of holes on an approximate 12.5 by 12.5 m to 25 by 25 m drill spacing. The data derived from this drilling and the recent 25 by 25 m drilling was used to confirm short scale mineralisation continuity and refine statistical and geostatistical relationships in the data which are useful in resource estimation.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	All field logging is carried out on Toughbooks using LogChief data capture software. Logging data is submitted electronically to the Database Geologist in the Perth office. Assay files are received electronically from the Laboratory. All data is stored in a Datashed/SQL database system, and maintained by the Gold Road Database Manager.
	Discuss any adjustment to assay data.	No assay data was adjusted. The laboratory's primary Au field is the one used for plotting and resource purposes. No averaging is employed.
Location of data points	*Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	The drill hole locations were initially picked up by handheld GPS, with an accuracy of 5m in northing and easting. All holes were later picked using DGPS to a level of accuracy of 1 cm in elevation and position. For angled drill holes, the drill rig mast is set up using a clinometer, and rigs aligned by surveyed positions and/or compass. Drillers use an electronic single-shot camera to take dip and azimuth readings inside the stainless steel rods, at 50 m intervals, prior to August 2014, and 30 m interval, post August 2014. Downhole directional surveying using north-seeking gyroscopic tool was completed on site and live (down drill rod string) or after the rod string had been removed from the hole. Most diamond drill holes were surveyed live whereas most RC holes were surveyed upon exiting the hole.

Criteria	JORC Code explanation	Commentary
	Specification of the grid system used.	A local grid (Gruyere Grid) was established by contract surveying group Land Surveys. The purpose of the local grid is to have an accurate and practical co-ordinate system along strike of the deposit. A high density survey control network and an accurate transformation between Gruyere Grid and MGA94-51 has been established. All ongoing studies, geological and resource activities are now conducted in Gruyere Grid.
	Quality and adequacy of topographic control.	An Aerial Lidar and Imagery Survey was completed January 2016 by Trans Wonderland Holdings as part of the ongoing FS covering 2,558 km^2 over the project area. One metre contours from this survey were used to construct a new topography surface to constrain the resource model. The survey showed good agreement with the existing DGPS drill hole collar data. All drill holes used in the resource grade estimate have a final collars survey by DGPS which are has a 1cm elevation accuracy.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.*	Drill spacing is at an approximate 50 m section spacing and 40 to 80 m on section over the top 200 vertical metres of the deposit; the spacing is at a 100 m sections at 50 to 100 m spacing from 150 to 600 vertical metres. Approximately 75 % of the pit strike length has been drilled to 25 by 25 m spaced holes to a depth of 70 to 100 m below surface. Drill spacing in relation to Resource Classification is discussed further in Section 3 below.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	Spacing of the reported drill holes is sufficient to demonstrate the geological and grade continuity of the deposit, and is appropriate for resource estimation procedures. Detailed description of the relationship between drill spacing and Resource classification is provided in Section 3 below.
	Whether sample compositing has been applied.	A total of 246 RC samples (out of a total 22,072 RC samples) featured compositing over waste intervals. This is the equivalent of <1% of all RC sample collected. None of these composited samples have been used in the Resource Estimate. No compositing has been employed in the diamond drilling. No sample compositing has been used during reporting – all reported intersections represent full length weighted average grades across the intersection length.
Orientation of data in relation to geological structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drill sections are oriented west to east (270° to 090° Gruyere Grid) with the majority of holes oriented approximately perpendicular to dip and strike at -60° to 270°, 14 holes in this orientation are shallow to dip and four are steep to dip. A small component of drilling has been drilled in a northward orientation, five of these are deep diamond drill holes drilled along the strike of the deposit (-60 towards 010°) to specifically test along strike continuity. Twenty-six holes are drilled to the northeast and east, and six are drilled to the south. The table below details the drilling orientation by drill type.

Azimuth (Gruyere Grid)	Dip	DDH	RC	Total	Comment
250 to 290	-40 to -50	7	7	14	Perpendicular to strike and shallow to dip
250 to 290	-51 to -75	69	291	360	Perpendicular to strike and dip
250 to 290	-76 to -85	2	2	4	Perpendicular to strike and steep to dip
291 to 020	-55 to -70	11		11	Along strike / down dip - includes 1 wedge
021 to 100	-60 to -80	12	14	26	To northeast and east
101 to 249	-60 to -70	2	4	6	To south
na	-90		2	2	Water bores
	Total	**103**	**320**	**423**	

Criteria	JORC Code explanation	Commentary
	If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	Detailed structural logging of diamond drill core identified important quartz veins sets with an approximate shallow dip to the east. Drilling angled at either -60 to the east or west does not introduce any directional bias given the current understanding of the structural orientations and the dip and strike of mineralisation.
Sample security	*The measures taken to ensure sample security.*	For all RC drilling and diamond drilling pre-numbered calico sample bags were collected in plastic bags (five calico bags per single plastic bag), sealed, and transported by company transport to the Intertek laboratory in Kalgoorlie. Prepared pulps were then despatched by Intertek to its laboratory in Perth for assaying.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	Sampling and assaying techniques are industry-standard. Internal and Consultant reviews of QAQC have been completed and documented. Company laboratory audits have been complete at the Intertek Laboratory in Perth. No independent laboratory or sample audits have been completed.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	*Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The RC and diamond drilling occurred within tenement E38/2362, which is fully owned by Gold Road. The tenement is located on the Yamarna Pastoral Lease, which is owned and managed by Gold Road. Tenement E38/2362 is located inside the Yilka Native Title Claim, WC2008/005, registered on 6 August 2009. The 2004 "Yamarna Project Agreement" between Gold Road and the Cosmo Newberry Aboriginal Corporation governs the exploration activities respectively inside the Pastoral Lease. As part of the ongoing FS Yilka and Gold Road reached an in-principle native title mining agreement in December 2015 and are working to sign the final agreement within Q2 2016 as a precursor to grant of the lodged mining lease application.
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	The tenement is in good standing with the WA DMP.
Exploration done by other parties	*Acknowledgment and appraisal of exploration by other parties.*	No previous exploration has been completed on this prospect by other parties.
Geology	*Deposit type, geological setting and style of mineralisation.*	The Gruyere Deposit comprises a narrow to wide porphyry intrusive dyke (Gruyere Porphyry – a Quartz Monzonite) which is between 35 and 190 m in width and which strikes over a current known length of 2,200 m. The Gruyere Porphyry dips steeply (65-80 degrees) to the east. A sequence of intermediate to mafic volcaniclastic rocks defines the stratigraphy to the west of the intrusive and intermediate to mafic volcanics and a tholeiitic basalt unit occur to the east. Mineralisation is confined ubiquitously to the Gruyere Porphyry and is associated with pervasive overprinting albite-sericite-chlorite-pyrite (±pyrhhotite±arsenopyrite) alteration which has obliterated the primary texture of the rock. Minor fine quartz-carbonate veining occurs throughout. Pyrite is the primary sulphide mineral and some visible gold has been observed in logged diamond drill core. The Gruyere Deposit is situated at the north end of the regional camp-scale South Dorothy Hills Target identified by Gold Road during its regional targeting campaign completed in early 2013. The Gruyere Deposit comprises coincident structural and geochemical targets within a major regional-scale structural corridor associated with the Dorothy Hills Shear Zone. This zone occurs within the Dorothy Hills Greenstone Belt at Yamarna in the eastern part of the Archaean Yilgarn Craton. The Dorothy Hills Greenstone is the most easterly known occurrence of outcropping to sub-cropping greenstone in the Yilgarn province of Western Australia.
Drill hole Information	*A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ▪ easting and northing of the drill hole collar ▪ elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar ▪ dip and azimuth of the hole ▪ down hole length and interception depth ▪ hole length. *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	Appendix 2 outlines previous general ASX announcements that contain reported drill hole information for all relevant RC and Diamond holes included in the reported resource estimation. The 25 by 25 m RC data has not been reported in detail as it is considered operational.
Data aggregation methods	*In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.*	All drill assay results (except for the previously mentioned 25 by 25 m RC holes) used in this estimation of this resource have been published in previous releases; refer to Appendix 2 for a list of previous releases.

Criteria	JORC Code explanation	Commentary
	Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	All drill assay results (except for the previously mentioned 25 by 25 m RC holes) used in this estimation of this resource have been published in previous releases; refer to Appendix 2 for a list of previous releases.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	No metal equivalent values are used.
Relationship between mineralisation widths and intercept lengths	*These relationships are particularly important in the reporting of Exploration Results.* *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	Mineralisation is hosted within a steep east-dipping, N-S striking porphyry. The porphyry is mineralised almost ubiquitously at greater than 0.3 g/t Au and is characterised by pervasive sub-vertical shear fabrics and sericite-chlorite-biotite-albite alteration with accessory sulphides dominated by pyrite-pyrrhotite-arsenopyrite. Higher grade zones occur in alteration packages characterised by albite-pyrrhotite-arsenopyrite alteration and quartz and quartz-carbonate veining. These vein packages dip at approximately -45^0 to the SSE, with strike extents of over 100 m. The general drill direction of 60^0 to 270^0 is approximately perpendicular to the main alteration packages and is a suitable drilling direction to avoid directional biases.
Diagrams	*Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Refer to Figures and Tables in the body of the release.
Balanced reporting	*Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	All drill assay results (except for the previously mentioned 25 by 25 m RC holes) used in this estimation of this resource have been published in previous releases; refer to Appendix 2 for a list of previous releases.
Other substantive exploration data	*Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Drill hole location data are plotted in Figures in the body text.
Further work	*The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Possible extensions at depth and to the south at depth will be tested in a strategic manner.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	*Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*	Geological metadata is stored centrally in a relational SQL database with a DataShed front end. Gold Road employs a Database Manager who is responsible for the integrity and efficient use of the system. Only the Database Manager or their Data Entry Clerk has permission to modify the data. Sampling and geological logging data is collected in the field using LogChief software and uploaded digitally. The software utilises lookup tables, fixed formatting and validation routines to ensure data integrity prior to upload to the central database. Sampling data is sent to, and received from, the assay laboratory in digital format. Drill hole collars are picked up by differential GPS (DGPS) and delivered to the database in digital format. Down hole surveys are delivered to the database in digital format. The Mineral Resource estimate only uses Gold Road RC and DDH assay data. There is no historical data.
	Data validation procedures used.	DataShed software has validation procedures that include constraints, library tables, triggers and stored procedures. Data that does not pass validation tests must be corrected before upload. The LogChief software utilises lookup tables, fixed formatting and validation routines to ensure data integrity prior to upload to the central database. Geological logging data is checked visually in three dimensions against the existing data and geological interpretation. Assay data must pass laboratory QAQC before database upload. Gold Road utilises QAQR software to further analyse QAQC data, and batches which do not meet pass criteria are requested to be re-assayed. Sample grades are checked visually in three dimensions against the logged geology and geological interpretation. Drill hole collar pickups are checked against planned and/or actual collar locations. A hierarchical system is used to identify the most reliable down hole survey data. Drill hole traces are checked visually in three dimensions. The project geologist and resource geologist are responsible for interpreting the down hole surveys to produce accurate drill hole traces.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits. If no site visits have been undertaken indicate why this is the case.*	Justin Osborne is one of the Competent Persons and is Gold Road's Executive Director. He conducts regular site visits and is responsible for all aspects of the project. John Donaldson is the second Competent Person and is Gold Road's Principal Resource Geologist. He conducts regular specific site visits to focus on understanding the geology as it is revealed in the drilling data. Communication with the site geologists is key to ensuring the latest geological interpretations are incorporated into the resource models. Both Competent Persons contribute to the continuous improvement of sampling and logging practices and procedures.

Criteria	JORC Code explanation	Commentary
Geological interpretation	*Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.*	The predominance of diamond drilling at Gruyere has allowed a robust geological interpretation to be developed, tested and refined over time. Early establishment of lithology and alteration coding and detailed structural logging has given insight into geological and grade trends that have been confirmed with geostatistical analysis, (including variography). Other sources of data (see next commentary) have also added confidence to the geological interpretation. The type and thickness of host lithology and main hangingwall mafic dyke is predictable. Other non-mineralised mafic and intermediate dykes are less predictable. The footwall and hangingwall lithologies are less well known due to the focus of drilling on mineralised units. However, the hangingwall lithologies are understood better as holes are collared on this side of the deposit. Results from the EIS hole (ASX announcement dated 8 September 2015) have improved the understanding of hangingwall lithologies and this will improve with further study. Continued drilling has shown that the approximate tenor and thickness of mineralisation is also predictable, but to a lesser degree than the geology. Results from the 25 by 25 m RC grade control drilling data have confirmed the geological interpretation and mineralisation model. As the deposit has good grade and geological continuity, which has been confirmed by grade control drilling, the Competent Persons regard the confidence in the geological interpretation as high.
	Nature of the data used and of any assumptions made.	All available data has been used to help build the geological interpretation. This includes geological logging data (lithology and structure), gold assay data (RC and DDH), portable XRF multi-element data (Niton and laboratory), geophysics (airborne magnetics and gravity), down hole Televiewer data (optical images and structural measurements, specific gravity, resistivity and natural gamma) and mineral mapping and multi-element data from research conducted in partnership with the CSIRO. An assumption regarding some gold remobilisation has been made at the more deeply weathered northern end of the deposit where a small flat lying gold dispersion blanket has been interpreted near the saprolite / saprock boundary. This is believed to represent dispersion of gold due to weathering processes. Justification for this interpretation lies in the lack of visual control to the mineralisation and its position in the weathering profile.
	The effect, if any, of alternative interpretations on Mineral Resource estimation.	A model constrained only by lithology (Gruyere Porphyry) was run to compare against the implicitly (and lithologically) constrained at 0.3 g/t model (actual model). Results showed that at 0 g/t cut-off the estimate of ounces was within 2%, and, as expected the lithologically constrained model had higher tonnage at lower grade. At 0.5 g/t, grade is 10% less and ounces are 7% less, and at 1.0 g/t grade is 1% less and ounces are 19% less in the lithologically constrained model. Moreover, in previous updates, one other potential mineralised trend, keeping all other constraints constant, was been modelled and showed little effect on the global estimate of volume.
	The use of geology in guiding and controlling Mineral Resource estimation.	Regionally the deposit is hosted in an Archaean basin to the East of the crustal scale Yamarna Shear Zone. The Gruyere deposit is located on an inflection of the NW (MGA) striking Dorothy Hills Shear Zone which transects the basin. The Dorothy Hills Shear Zone is the first order control into which the host Gruyere Porphyry has intruded. The bulk of the mineralisation has been constrained to the host intrusive below the base of Quaternary and Permian cover. Several NNE dipping cross-cutting arcuate and linear faults have been interpreted from airborne magnetics, the distribution of lithology and diamond core intersections of faults. The Alpenhorn Fault and to a lesser degree the Northern Fault have been used to constrain the distribution of mineralisation.

Criteria	JORC Code explanation	Commentary
		Mineralisation within the intrusive host has been implicitly modelled to the mineralisation trends discussed below at a constraining 0.3 g/t cut-off. The cut-off was established using two lines of reasoning: 1. All of the assay data internal to the host rock was plotted on a log probability plot; a value of 0.3 g/t was recognised as an inflection point subdividing the non-mineralised and mineralised populations. This is further supported through a reduction in the CV in the unconstrained case from 1.0 to 0.9 in the constrained case i.e. a reduction in stationarity supporting the domaining. 2. 0.3 g/t corresponds to the approximate grade cut-off between barren to very weakly mineralised hematite-magnetite alteration and weak to strongly mineralised albite-sericite-carbonate ± pyrite, pyrrhotite, arsenopyrite alteration. Three mineralisation Domains have been modelled; Primary, Weathered and the minor Dispersion Blanket. 1. The Primary Domain corresponds to mineralisation hosted in fresh, transitional and saprock Gruyere Porphyry. The mineralisation trend is along strike and steeply down dip. The trend was established using observations of alteration, sulphide and gold grade distribution, together with the following structural observations from diamond core: ▪ The along strike component corresponds to the main foliation within the intrusive host. ▪ The steep down dip component corresponds to a strong down-dip lineation parallel to the axes of tight to isoclinal folds of the pre-existing foliation within the intrusive host. The strike and dip components for the Primary Domain were readily confirmed in the variography. 2. A secondary Domain corresponds to mineralisation hosted in deeply weathered (saprolite) Gruyere Porphyry. The mineralisation trend is flat lying, reflecting the weathering processes. The trend was established using observations of gold grade distribution and the position relative to the weathering profile. The strike and dip components for the Weathered Domain were readily confirmed in the variography. 3. A minor third Domain corresponds to a flat lying, 4 to 5 m thick, gold dispersion blanket interpreted near the saprolite boundary and hosted within hangingwall and footwall lithologies.
	The factors affecting continuity both of grade and geology.	Apart from the controls discussed previously, one narrow (1 to 5 m wide), steeply dipping non-mineralised internal mafic dyke has been modelled as barren within the intrusive host. Other narrow (generally less than 1 m wide) mafic and intermediate intrusives / dykes occur but have very short scale continuity and insignificant to the scale of mineralisation.
Dimensions	*The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	Length along strike: 1,800 m Horizontal Width: 7 to 190 m with an average of 90 m. The vertical depth of Mineral Resource from surface to the upper limit is 2 m and to the lower limit is 600 m. The Mineral Resource has been constrained by an optimised Whittle shell that considers all available mineralisation in the geological model. The optimisation utilises realistic mining, geotechnical and processing parameters from the latest information available from the ongoing FS. The gold price used was A$1,700/oz Au. Only Measured, Indicated and Inferred categories within this shell have been reported as Mineral Resource. Mineralisation in the geology model outside the shell has not been reported. Approximately 39,000 oz of unclassified* mineralisation falls within the shell and is not reported. *Low confidence mineralisation within the geological model that does not satisfy the criteria for Mineral Resource has been flagged as unclassified.

Criteria	JORC Code explanation	Commentary

	The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	**Software used:**
		▪ Datashed – frontend to SQL database
		▪ MapInfo – geophysics and regional geology
		▪ Stereonet – compilation and interpretation of diamond structural data.
		▪ Core Profiler – compilation of downhole photographs in core trays for geo-referencing in 3D software.
		▪ Leapfrog Geo – Drill hole validation, material type, lithology, alteration and faulting wireframes, domaining and mineralisation wireframes, geophysics and regional geology
		▪ Snowden Supervisor - geostatistics, variography, declustering, kriging neighbourhood analysis (KNA), validation
		▪ Datamine Studio RM – Drill hole validation, cross-section, plan and long-section plotting, block modelling, geostatistics, quantitative kriging neighbourhood analysis (QKNA), OK estimation (for validation and input to LUC), block model validation, classification, and reporting.
		▪ Datamine Studio RM Uniform Conditioning Module – LUC grade estimation. The module is an interface to the code in Isatis software for change of support, information effect calculation, uniform conditioning and grade localisation. Isatis is the most highly regarded geostatistical software in the industry and is used by many of the top gold mining companies worldwide.
		Localised Uniform Conditioning:
		▪ LUC was selected as at technique to estimate the Indicated and Inferred areas of this resource update as the method provides estimates of Selective Mining Units (SMU) from widely spaced data. The LUC model is globally accurate but the estimate of the grade tonnage curve is not over smoothed (as in conventional OK) resulting in less tonnes at higher grade above a given cut-off (ie. an estimate of the grade control grade tonnage curve).
		▪ The improved resolution of LUC adds value to economic evaluation at higher cut-offs (e.g. 1.0 g/t): however, at lower cut-offs (e.g. 0.5 g/t) used for reporting there are no significant differences between the direct block (OK) estimate and the LUC estimate.
		▪ In models prior to September 2015 grades were estimated using an OK methodology into large parent blocks resulting in a globally accurate but smoothed grade tonnage curve (more tonnes at lower grade above cut-off).
		Block model and estimation parameters:
		▪ Treatment of extreme grade values – Top-cuts (all samples included method) were applied to 2m composites selected within mineralisation wireframes. The top-cut level was determined through the analysis of histograms, log histograms, log probability plots and spatial analysis.
		    ▪ Primary - one sample was cut using a 30 g/t top-cut resulting in a 0.1% reduction in mean grade.
		    ▪ Weathered - 3 samples were cut using a 10 g/t top-cut resulting in a 1.0% reduction in mean grade.
		    ▪ Dispersion Blanket - no samples were top-cut.
		▪ Estimation technique for Measured – OK – at this data spacing (25 by 25 m grade control) OK is the appropriate technique, where LUC is appropriate for broader spaced drilling. The data is sufficiently dense for a correct direct block estimate.
		▪ Estimation for technique Indicated and Inferred - LUC - with an OK estimate (25 m X by 50 m Y by 10 m Z panels) required as input.

Criteria	JORC Code explanation	Commentary

		▪ KNA was undertaken to optimise the search neighbourhood used for the estimation and to test the parent block size. The search ellipse and selected samples by block were viewed in three dimensions to verify the parameters.
		▪ Model rotation – none required – local Gruyere Grid used.
		▪ Parent block size for Measured estimation of gold grades by OK - 5 m X by 12.5 m Y by 5 m Z (parent cell estimation with full subset of points).
		▪ LUC inputs for Indicated and Inferred estimation of gold grades (note that 6 estimation scenarios were tested and analysed before deciding on the final input parameters);
		  ▪ 12.5 m X by 25 m Y by 5 m Z declustering of input data in Supervisor (the declustering weight is inversely proportional to the number of data points in each cell). Note that change in grade through declustering with respect to the use of the cell size optimiser is minimal.
		  ▪ Discretisation 3 X by 5 Y by 2 Z
		  ▪ Information Effect planned sample spacing 25 m X by 25 m Y by 1 m Z, and 9 X by 9 Y by 5 Z planned number of samples
		  ▪ 40 SMUs (5 m X by 12.5 m Y by 5 m Z) per panel (25 m X by 50 m Y by 10 m Z)
		  ▪ 70 cut-offs at 0.1 g/t intervals
		  ▪ 7 iso-frequencies
		▪ Smallest sub-cell – 1 m X by 12.5 m Y by 1 m Z (a small X dimension was required to fill internal mafic dyke and a small Z dimension was required to fill to material type boundaries).
		▪ Panel discretisation - 3 X by 5 Y by 2 Z (using the number of points method)
		▪ Measured Search ellipse – aligned to mineralisation trend, dimensions;
		  ▪ Fresh - 35 m X by 60 m Y by 15 m Z.
		  ▪ Weathered – 50 m X by 80 m Y by 15 m Z.
		  ▪ Dispersion Blanket - 50 m X by 80 m Y by 15 m Z.
		▪ Indicated and Inferred Search ellipse – aligned to mineralisation trend, dimensions;
		  ▪ Fresh - 200 m X by 350 m Y by 60 m Z (the longest range in variogram is 350 m).
		  ▪ Weathered - 50 m X by 80 m Y by 15 m Z (the longest range in variogram is 80 m).
		  ▪ Dispersion Blanket - 50 m X by 80 m Y by 15 m Z.
		▪ Measured - number of samples:
		  ▪ Fresh – maximum per drill hole = 4, first search 16 min / 36 max, second search 16 min / 36 max and a volume factor of 2, third search 8 min / 36 max with a volume factor of 2
		  ▪ Weathered– maximum per drill hole = 5, first search 30 min / 60 max, second search 30 min / 60 max and a volume factor of 2, third search 10 min / 60 max with a volume factor of 2
		  ▪ Dispersion Blanket – maximum per drill hole = 5, first search 30 min / 60 max, second search 30 min / 60 max and a volume factor of 2, third search 6 min / 60 max with a volume factor of 2
		▪ Indicated and Inferred - number of samples:

Criteria	JORC Code explanation	Commentary
		▪ Fresh – maximum per drill hole = 7, first search 30 min / 60 max, second search 15 min / 60 max and a volume factor of 1, third search 5 min / 60 max with a volume factor of 3 ▪ Weathered – maximum per drill hole = 5, first search 30 min / 60 max, second search 30 min / 60 max and a volume factor of 2, third search 1 min / 60 max with a volume factor of 3 ▪ Dispersion Blanket – maximum per drill hole = 5, first search 20 min / 60 max, second search 10 min / 60 max and a volume factor of 2, third search 2 min / 60 max with a volume factor of 3 ▪ Maximum distance of extrapolation from data points – 50 m from sample data to Inferred boundary Domain boundary conditions – Hard boundaries are applied at all domain boundaries.
	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	Several internal models and three public models were produced prior to the publication of this Mineral Resource. These were used to plan drilling programs, manage performance and expectation and test geological interpretation on an ongoing basis during and after the various drilling campaigns. Analysis shows that this model has performed well globally and locally against the original internal and publically released models. In particular, and locally at a 0.5 g/t cut-off, in the Measured (grade control defined) portion of this model (13.9 Mt at 1.18 g/t for 526 koz) the variance has been minimal +4% for tonnes, -4 % for grade and +1% for ounces in comparison to the same volume in the previous model (Indicated). There is no previous production.
	The assumptions made regarding recovery of by-products.	There are no economic by-products.
	Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	No deleterious elements of significance have been determined from metallurgical test work and mineralogical investigations. Waste rock characterisation work has been completed and all waste types and tailings are non-acid forming and have limited metal leachate potential.
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	For the Measured (OK estimate). The parent block size of 5 m X by 12.5 m Y is approximately: • 50% of the maximum drill spacing of 25 m X by 25 m Y in Measured areas For the Indicated and Inferred (OK estimate as input to LUC) The parent block size of 25 m X by 50 m Y is approximately: • 25% of the minimum drill spacing of 50 m X by 100 m Y in Indicated areas • 12.5% of the maximum drill spacing of 100 m X by 100 m Y in Inferred areas
	Any assumptions behind modelling of selective mining units.	The selective mining unit (SMU) of 5 m X by 12.5 m Y by 5 m Z was chosen as it gives 40 SMU's per 25 m X by 50 m Y by 10 m Z parent cell (a minimum of around 24 SMU's are required for adequate grade / tonnage definition) and corresponds well with mining equipment and mining flitch sizes selected in the PFS. A separate fleet sizing study will be completed during the FS.
	Any assumptions about correlation between variables.	No correlation between variables was analysed or made.
	Description of how the geological interpretation was used to control the resource estimates.	The geological interpretation was used at all stages to control the estimation. If geostatistics, variography and/or visual checks of the model were difficult to interpret then the geological interpretation was questioned and refined.
	Discussion of basis for using or not using grade cutting or capping.	Top-cuts were used in the estimate as this is the most appropriate way to control outliers when estimating block grades from assay data.

Criteria	JORC Code explanation	Commentary
	The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	The following validation checks were performed: ▪ QQ plots of RC vs DDH input grades. ▪ Statistical comparison of different drilling orientations including local spot checks. ▪ Comparison of twinned RC, twinned DDH and twinned RC v DDH holes. ▪ Comparison of the volume of wireframe vs the volume of block model. ▪ Checks on the sum of gram metres prior to compositing vs the sum of gram metres post compositing ▪ A negative gold grade check ▪ Comparison of the model average grade and the declustered sample grade by Domain. ▪ Generation of swath plots by Domain, northing and elevation. ▪ Comparison of LUC estimate to OK estimate. ▪ Visual check of drill data vs model data in plan, section and three dimensions. ▪ Comparison to previous models ▪ Comparison to alternative interpretations (see above) All validation checks gave suitable results. There has been no mining so no reconciliation data available.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Average bulk density values have been modified by a moisture percentage so that dry tonnage is reported. These are: overburden and saprolite 5%, saprock 3%, transition 2% and fresh 1 %.
Cut-off parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied.*	The cut-off grade used for reporting is 0.5 g/t gold. This has been determined from mining and processing parameters and input costs from the latest information available from the ongoing FS.
Mining factors or assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	The mining method assumed is conventional open pit with a contract mining fleet appropriately scaled to the size of the deposit. Whittle optimisation input parameters are outlined in Table 11 of the main text. The de facto minimum mining width is a function of parent cell size (25m X by 50m Y by 10m Z). No allowance for dilution or mining recovery has been made in the Mineral Resource estimate.

Criteria	JORC Code explanation	Commentary
Metallurgical factors or assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	A single stage primary crush, Semi Autogenous Grinding and Ball Milling with Pebble Crushing (SABC) comminution circuit followed by a conventional gravity and carbon in leach (CIL) process is proposed. This process is considered appropriate for the Gruyere ore, which has been classified as free-milling. The proposed metallurgical process is commonly used in the Australian and international gold mining industry and is considered to be well-tested technology. Metallurgical recovery is applied to the resource model by material type and grind size (106µm, 125µm and 150µm) according to test work values for weathered material and grade recovery curves for fresh rock. 106µm was selected for input to optimisation. No recovery factors are applied to the Mineral Resource numbers themselves. Significant comminution, extraction, and materials handling testing has been carried out on over 4,500 kg of half-core diamond drilling core samples (NQ core diameter = 47.6mm). The testing has been carried out on saprolite (oxide), saprock, transitional and fresh ore types which were selected to represent different grade ranges along the strike length of the deposit and to a depth of around 410 m. For the fresh rock samples, 62 composites representing four major mineralised zones (South, Central, North and High Grade North) were subjected to gold extractive test work by gravity separation and direct cyanidation of gravity tails. In total, 183 individual gravity-leach tests were completed at various grind size P80 ranging from 106 µm to 150 µm. Gravity gold recoveries are estimated at 35%. Estimated plant gold recovery ranges from 87% to 96% depending on head grade, plant throughput, grind size and ore type and are summarised in the table below.

Material Type	Metallurgical Recovery at P80			
	106 µm	125 µm	150 µm	Comments
Saprolite (oxide)	94%	93%	92%	
Saprock	94%	93%	92%	
Transition	93%	92%	91%	
Fresh	2.6130 x ln head grade (g/t) + 92.199 %	3.1818 x ln of head grade (g/t) + 90.362 %	3.3997 x ln of head grade (g/t) + 88.929 %	capped at 96%

No deleterious elements of significance have been determined from metallurgical test work and mineralogical investigations.

Criteria	JORC Code explanation	Commentary
Environmental factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	Surface waste dumps and infrastructure (e.g. tailings dam) will be used to store waste material from open pit mining. Conventional storage facilities will be used for the process plant tailings. Test work has been completed for potential acid mine drainage material types. Results show that all material types are non-acid forming and are unlikely to require any special treatment. Baseline environmental studies of flora, vegetation, vertebrate fauna, short-range endemic invertebrates and subterranean fauna have commenced and are due for completion within the timeframe of the FS schedule.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td rowspan="3">Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.</td><td>Bulk density has been determined using 2 main methods and cross checked with data from recent metallurgical test work:
1. RC drilling – downhole rock property surveys completed by ABIMS Pty Ltd which provide a density measurement every 0.1 m downhole.
2. DDH drilling – weight in air / weight in water – measurements every 1 m in weathered every 10 m in fresh rock, using approximate 0.1 m core lengths.
The physical measurements derived from the air/water method were compared to the down hole tool measurements and metallurgical test work. Good correlation was observed between methods for saprolite, saprock and transitional. The down-hole tool values for fresh rock did not match the other two methods and so was set aside pending review by the provider.</td></tr>
<tr><td>The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.</td><td>Vacuum sealed bags were used where required to account for void spaces in the core.
Bulk density has been applied by lithology and weathering type.</td></tr>
<tr><td>Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>Data was coded by method, lithology (including mineralisation and cover) and weathering type. The three methods were compared and found to be in agreement except for the down hole tools values for fresh rock. Averages were derived both by lithology and weathering type. Assumptions for moisture percentages were made and accounted for in the final value used for bulk density.</td></tr>
<tr><td rowspan="3">Classification</td><td>The basis for the classification of the Mineral Resources into varying confidence categories.</td><td>The Mineral Resource has been constrained within an optimised Whittle pit shell. Blocks in the geological model within that shell have been classified as Measured, Indicated or Inferred. Several factors have been used in combination to aid the classification;
▪ Drill hole spacing:
<table>
<tr><th>Domain</th><th>Criteria</th><th>Measured</th><th>Indicated</th><th>Inferred</th><th>Unclassified</th></tr>
<tr><td rowspan="5">Primary</td><td>Target Spacing</td><td>25 m X by 25 m Y</td><td>50 m X by 100 m Y</td><td>100 m X by 100 m Y</td><td></td></tr>
<tr><td rowspan="2">Actual Spacing</td><td rowspan="2">12.5 m X by 12.5 m Y to 25 m X by 25 m Y</td><td rowspan="2">25 m X to 65 m X by 100 m Y with extra holes on 50 m Y</td><td>100 m X by 100 m Y</td><td>"Potential" beyond Inferred to limits of geological model.</td></tr>
<tr><td>Footwall contact of along strike hole 14GYDD0061</td><td></td></tr>
<tr><td rowspan="2">Boundary Extension</td><td>10 to 15 m along strike</td><td>25 m along strike</td><td>50 - 100 m along strike</td><td></td></tr>
<tr><td>Closet 5 m Rl from bottom of hole</td><td>Minimal down dip - except North end 30 m from drilling. Drilling needs to define full width of intrusive host.</td><td>Minimal down dip - except North end 50 m from Indicated boundary</td><td></td></tr>
<tr><td rowspan="2">Weathered</td><td>Target Spacing</td><td>12.5 to 25 m X by 25 m Y</td><td>50 m X by 100 m Y</td><td></td><td></td></tr>
<tr><td>Actual Spacing</td><td>12.5 m X by 12.5 m Y to 25 m X by 25 m Y</td><td>25 m X to 50 m E by 100 m Y with extra holes on 50 m Y</td><td></td><td></td></tr>
<tr><td>Dispersion Blanket</td><td>Actual Spacing</td><td></td><td></td><td>25 to 50 m X by 25 to 100 m Y</td><td>"Potential" beyond Inferred to limits of geological model.</td></tr>
</table>
▪ Level of geological continuity.
▪ Level of grade continuity.
▪ Consideration of estimation quality parameters derived from the OK process.</td></tr>
<tr><td>Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).</td><td>All relevant factors have been taken into account in the classification of the Mineral Resource.</td></tr>
<tr><td>Whether the result appropriately reflects the Competent Person’s view of the deposit.</td><td>The Mineral Resource estimate appropriately reflects the Competent Person’s view of the deposit.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*	Ian Glacken (Director - Geology at Optiro consultants) was engaged to externally review the technical aspects of this update, and the three previous Mineral Resource estimates. A formal review was undertaken and suggestions for improvement were sought and applied where appropriate. An endorsement letter/summary report of the review has been completed for this update and the three previous Mineral Resource estimates. Optiro is satisfied that the Mineral Resource estimate has been reported and classified according to the guidelines set out in the JORC Code 2012 and in line with good to best industry practice. An external database audit was not undertaken for this update due to the operational nature of the drilling. Lisa Bascombe of Optiro conducted audits for the three previous Mineral Resource estimates. Internal geological peer review by the Executive Director, Exploration manager and/or geological team, and handover meetings with the development and operational teams were held and documented at appropriate times. An informal internal peer review, as part of a board briefing, was conducted with the Non-executive Directors on the Gold Road board, who are also geologists, for the previous Mineral Resource estimate. A QAQC report was completed by Dr Paul Sauter (internal consultant – Sauter Geological Services Pty Ltd) for data collected for this update to the resource. Results are acceptable and an improvement on previous results. Recommendations include further umpire lab testing and changing the blanks to a more appropriate material. A QAQC report was completed by Mr Dave Tullberg (Grassroots Data Services Pty Ltd) for data collected for the maiden resource. A QAQC report was completed by Dr Paul Sauter (internal consultant – Sauter Geological Services Pty Ltd) for data collected for the previous two updates to the resource. This included analysis of umpire lab test-work.
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.*	Variances to the tonnage, grade and metal of the Mineral Resource estimate are expected with further definition drilling. It is the opinion of the Competent Persons that these variances will not significantly affect economic extraction of the deposit. The mean grade of raw assay data in the mineralised domains compare extremely well upon the collection of additional data;

Model Release	Number of Mineralised Samples (>0.3 g/t)	Mean g/t
April 2016	32,293	1.245
September 2015	24,156	1.305
May 2015	22,490	1.268
August 2014	15,320	1.266
February 2014*	4,240	1.230

*in house model

Previous tests to determine the performance of the Inferred category as it has been upgraded with drilling to Indicated and Measured have been made. The results showed that a robust estimate of Inferred can be made as acceptable variances of tonnage, grade and/or metal were calculated from the original Inferred model in comparison to the same area in the Indicated or Measured model.

Performance of the Indicated category has been assessed in this update compared to previous estimates. At a 0.5 g/t cut-off, the Measured (grade control defined) portion of this model (13.9 Mt at 1.18 g/t for 526 koz) has

Criteria	JORC Code explanation	Commentary
		performed well against the same volume in the previous model (Indicated). The variance is minimal at +4% for tonnes, -4 % for grade and +1% for ounces. The model performance was also assessed visually. As new drilling data came in it was compared to the existing model; in the majority of cases the existing model matched the tenor and thickness of the new assay data.
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	Confidence in the Mineral Resource estimate is such that the Measured portions of the model will provide adequate accuracy for ore block design, monthly mill reconciliation and short to medium term scheduling. For the Indicated and Inferred portions it will provide adequate accuracy for global resource evaluation and for more detailed evaluation at a large scale. Bench evaluations show that tonnages greater than 5 million may be mined over a 20 m vertical height. This is twice the parent cell vertical height of 10 m, so an unbiased estimate at that scale is expected. For Indicated this equates to annual and quarterly production windows and to an annual production window for Inferred. Relative accuracy is expected to decrease at depth as smaller tonnages are mined as the pit width decreases.
	These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	No previous mining.

APPENDIX 4 - GLOSSARY OF TERMS AND ABBREVIATIONS

Term/ Abbreviation	Description
A$	Australian Dollar
AACE	Association for the Advancement of Cost Engineering
AARL	Anglo American Research Laboratory
AAS	Atomic Absorption Spectroscopy
AC	Aircore drilling
Ai	Abrasion Index
AIC	All In Cost
AMC	AMC Consultants Pty Ltd
AMS	Aerodrome Management Services Pty Ltd
ANFO	Ammonium Nitrate Fuel Oil
API-A	Assessment on Proponent Information (Category A)
ARI	Average Recurrence Interval
AS/NZS	Australian Standard/New Zealand Standards
ASIC	Australian Securities and Investment Commission
ASX	Australian Securities Exchange
Au	Gold
Axb	Resistance to impact breakage
Axiom	Axiom Project Services
BBWi	Bond Ball Work Index
bcm	bank cubic metres
bcm/op.hr	bank cubic metres per operating hour
BOO	Build Own Operate
C1	C1 = Mining + Processing Operating Expenditure + Site General and Administration Expenditure + Transport and Refining Costs
C2	C2 = C1 + Depreciation + Amortisation
C3	C3 = C2+ Royalties + Levies + Net Interest Costs
Capex	Capital expenditure
CASA	Civil Aviation Safety Authority
CCTV	Closed Circuit Television
CIL	Carbon-In-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-In-Pulp
CNAC	Cosmo Newberry Aboriginal Corporation
Coffey	Coffey Mining Ltd
Company	Gold Road Resources Limited
Cost Model	AMC OPMincost cost estimate system
CWi	Crusher Work Index
DAA	Department of Aboriginal Affairs
DDH	Diamond Drill Hole
DER	Department of Environmental Regulation
DMP	Department of Mines and Petroleum
DMR	Digital Mobile Radio - Two-way radio
DoE	Department of the Environment
DoL	Department of Lands
DoW	Department of Water
dtph	dry tonnes per hour
DWi	Drop Weight Index
EBA	Enterprise Bargaining Agreement

Term/ Abbreviation	Description
EBIT	Earnings Before Interest and Tax
EBITDA	Earnings Before Interest, Taxes, Depreciation and Amortisation
EBT	Earnings Before Tax
EGL	Effective Grinding Length
EGP	Eastern Goldfields Pipeline
EP Act	Environmental Protection Act 1986
EPA	Environmental Protection Authority
EPBC Act	Environment Protection and Biodiversity Conservation Act
EPC	Engineering, Procurement and Construction
EPCM	Engineering, Procurement and Construction Management
ESA	Environmentally Sensitive Area
FEL	Front End Loader
FIFO	Fly In Fly Out
FoS	Factor of Safety
FPXRF	Field Portable X-ray Fluorescence
FS	Feasibility Study
ft	foot (measurement)
FY	Financial Year
G&A	General and Administration
g/t	grams per tonne
GCBNTA	Gruyere and Central Bore Native Title Agreement
GL	Gigalitre
Gold Road	Gold Road Resources Limited
GRES	GR Engineering Services Limited
GRG	Gravity Recoverable Gold
GWL	Groundwater Licences
ha	hectare
HDPE	High Density Polyethylene
HG	High-Grade
HMI	Human Machine Interface
HQ	Diamond Core Diameter – 63.5 mm
hr	hour
HR	Human Resources
HSE	Health, Safety and Environment
I/O	Input/Output
ICP	Inductively Coupled Plasma
IPP	Independent Power Provider
IRR	Internal Rate of Return
IT	Information and Technology
IT&C	Information Technology and Communications
IWL	Integrated Waste Landform
JKSimMet	Comminution Simulation Software for Comminution Circuits
JORC Code	Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (2012 Edition)
kg	kilogram
kL	kilolitre
km	kilometre
koz	kilo ounces
kVA	kilovolt-ampere
kWh	kilowatt hour
L/s	Litres per second
LAN	Local Area Network

Term/ Abbreviation	Description
LCS	Local Control Station
lcm	loose cubic metres
LOM	Life of Mine
LPG	Liquefied Petroleum Gas
LUC	Localised Uniform Conditioning
m	metre
M	Million
m^2	square metre
m^3	cubic metre
MBS	MBS Environmental Pty Ltd
MCC	Motor Control Centre
min	minute
ML	Megalitre
mm	millimetre
Moz	Million ounces
MPa	Megapascal
MRMM	Mining Rock Mass Model
MRMR	Mining Rock Mass Rating
Mtpa	Million tonnes per annum
MVA	Megavolt-ampere
MW	Megawatt
MWh	Megawatt-hour
NI 43-101	National Instrument NI 43-101
NPI	Non-Process Infrastructure
NPV	Net Present Value
$NPV_{8\%}$	Net Present Value calculated at a discount rate of 8%
NQ	Diamond Drill Core Diameter - 47.6 mm
NT Act	Native Title Act 1993
OEPA	Office of the Environmental Protection Authority
OK	Ordinary Kriged
Opex	Operating expenditure
Optiro	Optiro Mining Consultants
OR	Operational Readiness
Owner's team	Gold Road Owner's team
oz	Troy Ounces
P_{80}	80% passing
pa	per annum
PCC	Process Control Cubicle
PCF	PCF Capital Group
PCS	Process Controls System
PEC	Priority Ecological Community
PEP	Project Execution Plan
PFS	Pre-Feasibility Study
pH	A measure of acidity or alkalinity
PLC	Programmable Logic Controller
PoF	Probability of Failure
ppm	parts per million
Project	Gruyere Gold Project
Q1, 2, 3, 4	Quarters 1, 2 3, 4 – Calendar Year
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Qualitative Evaluation of Mineral by Scanning Electron Microscopy
RAB	Rotary Air Blast drilling

Term/ Abbreviation	Description
RC	Reverse Circulation
RCD	Residual Current Device
RFDS	Royal Flying Doctor Service
RL	Reduced Level
RMR	Rock Mass Rating
RNE	Registers of the National Estate
RO	Reverse Osmosis
ROM	Run-of-Mine
RWi	Rod Mill Work Index
SABC	Semi Autogenous Ball Milling with Pebble Crushing
SAG	Semi-Autogenous Grinding
SFA	Screen Fire Assay
SG	Specific Gravity
SMC	SAG Mill Comminution
SMU	Selective Mining Unit
sp.	Species
SRE	Short-range Endemic
Sumitomo	Sumitomo Metal Mining Oceania Pty Ltd
Susex	Sustaining Capital Expenditure
t	tonne
TDS	Total Dissolved Solids
TEC	Threatened Ecological Communities
Technical Report	NI 43-101 Technical Report
TMM	Total Material Movement
tph	tonnes per hour
TSF	Tailings Storage Facility
UCS	Unconfined or Uniaxial Compressive Strength
UPS	Uninterrupted Power Supply
US$	United States Dollar
V:H	Vertical to Horizontal ratio
VESDA	Very Early Smoke Detection Apparatus
VHF	Very High Frequency
VSD	Variable Speed Drive
VVVF	Variable Voltage, Variable Frequency
w/w	Percent weight/weight
W:O	Waste to Ore ratio
WA	Western Australia
WAN	Wide Area Network
WBF	Whole-of-Business Framework
WBS	Work Breakdown Structure
WC Act	Wildlife Conservation Act
WSA	Water Supply Area
XRD	X-ray Diffraction