Annual Qualified Persons Report for the Ballarat Gold Project, Australia for the Year Ended 31 March 2015

LionGold Corp Ltd

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APPENDICES

Appendix A Checklist of assessment and reporting criteria, based on Table 1 of the 2012 JORC Code

1 EXECUTIVE SUMMARY

1.1 Report Scope

LionGold Corporation Limited ("LionGold") subsidiary Castlemaine Goldfields Propriety Limited ("CGT") has delivered an updated resource estimate and maiden reserve for its Ballarat East gold mine (the Ballarat mine), located at Ballarat, Victoria, Australia. The Mineral Resource and Ore Reserve are reported in accordance with The JORC Code 2012 (JORC, 2012) which will be publically reported by LionGold to the Singapore Exchange Securities Trading Limited ("SGX") and used by CGT to plan mining operations over the financial year 2015-2016. The Ballarat mine operation is described as follows.

1.2 Project Description

The Ballarat mine is owned and operated by CGT. CGT holds an Exploration Licence (EL3018), which covers the historic Ballarat East, Ballarat West and Ballarat South goldfields. Importantly this area includes two Mining Licenses; MIN5396 which covers the Ballarat East mine site, process plant and tailings storage facility, and MIN4847, which covers the Ballarat South goldfield. The Ballarat mine is located beneath the city of Ballarat. The field extends over a strike length of three kilometres and with historic records documenting over 1.2 Moz Au production from underground sources since 1858.

The mine has established infrastructure, including surface buildings, a fully operating plant, a fleet of mining vehicles and underground decline access to development. Production areas are accessed via the 1,205 m long Woolshed Gully decline and the 3,715 m long Woah Hawp decline, which has reached a point about 690 m below the portal. Overall, the current mine extends 3,422 m from the portal to the end of the decline. The entire underground network comprises some 19 km of tunnels.

The current mine production plan is based on a combination of ore generated from the development along the strike of the ore zone, mechanised drift and fill and longhole bench stoping. Geotechnical conditions and geometry of the ore bodies are highly variable and the mining method is selected to suit. Long hole stoping is a combination of "up-hole retreat" stopes with no backfill, and stopes where a top and bottom access is present allowing the stope void to be backfilled. The bulk of future production is scheduled from three main areas - Llanberris, Sovereign and Britannia ore lodes.

The CGT 2015-2016 plan is to mine ore from the current resource (Table 1.2). The overall 2015-2016 plan is to extract 246,000 t at 7.8 g/t Au for around 52,000 oz Au recovered. This is scheduled such that 76% (188,000 t at 6.9 g/t Au) of the tonnes are mined in 2015-2016 are from the current resource. Additional mill feed will come from 'not in resource' sources.

The resource is depleted during the 2015-2016 forecast year, such that only 19% (50,000 t at 6.3 g/t Au) of the forecast total of 246,000 t for 2016-2017 will come from the resource. The remaining 81% of the forecast total for 2016-2017 is expected from on-going exploration success which will be achieved from drilling of the exploration targets from within the existing mine footprint and this will identify further ore sources to allow for economic extraction in 2016-2017 at production rates, grades and costs similar to the 2015-2016 budget year.

Over the past year mining has successfully extracted approximately 60% of the Inferred Resource ounces, with the remainder being sterilised or deemed sub-economic under current conditions. This conversion rate is reflected in the 2015-2016 plan.

Three diamond drill rigs operate underground on a 24/7 basis, producing around 5,600 m of drill core per month. CGT has, over the last three years, demonstrated its capacity to replace resources depleted for mining. The existing infrastructure allows quick exploitation of areas identified during drilling and over the next 12 months.

The maiden Ore Reserve defined at Ballarat is based on the Indicated Mineral Resources and represent a relatively small portion of the mine's overall resources. The low rate of conversion from Inferred to Indicated Resource to support the Ore Reserve principally relates to a lack of close-spaced drilling to resolve geological and grade continuity. In particular, a high to extreme nugget effect exists on the gold grades. In addition, localised variations in vein complexity leading to poor geometric prediction are present. Economic decisions are thus based on a combination of Probable Ore Reserves and Inferred Mineral Resources. The project has appropriate infrastructure and plant in place. Mining costs, parameters and methods are now determined as a result of three years continuous mining. Project viability is highly sensitive to gold price and operating costs.

1.3 Geology and Mineralisation

Mineralisation occurs within Lower Ordovician sandstones, siltstones and mudstones that have been weakly metamorphosed and tightly folded about north-trending axes. The western limbs of the known anticlines dip approximately 70^0W, eastern limbs 85^0W to 85^0E and fold axial planes dip approximately 80^0W. The regional strike of the bedding is northerly. The quartz veins are located predominantly within fold limbs in structurally controlled bodies known as lodes and stockworks. Lodes and stockworks are hosted in west-dipping fault zones (e.g. the Llanberris Mako fault zone). Mineralisation is characterised by notable quantities of coarse gold (>80% +100-micron gold) and very coarse gold (locally >50% +1,000-micron gold) hosted in the quartz veins. A high nugget effect is observed, where grades over a few metres may reach 50 g/t Au or higher, but reduce to a few g/t Au out of the high grade.

1.4 Mine Production

Hard rock 'quartz-mining' commenced in 1858 at Ballarat. Between 1858 and 1917, the goldfield produced over 1.2 Moz Au at a head grade of approximately 9 g/t Au. Recent gold production commenced in 2005 (Table 1.1).

Table 1.1 Gold production history for the Ballarat East goldfield from 2005 to March 2015

Company	Year	Tonnes (t)	Grade (g/t Au)	Ounces Produced (oz Au)
BGF	2005-2009	234,000	3.85	28,965
CGT	2011-2012	57,346	5.00	7,189
CGT	2012-2013	170,663	6.60	29,066
CGT	2013-2014	170,291	8.50	39,962
CGT	2014-2015	250,664	6.78	46,039
Total	**2008-2015**	**882,964**	**6.18**	**151,221**

1.5 Mineral Resources and Ore Reserves

CGT has completed an update of its Mineral Resource with the addition of a maiden Ore Reserve for the Ballarat mine. Resources have been estimated and are reported in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 (The JORC Code 2012).

The estimated Mineral Resource consists of mineralisation within six discreet fault zones referred to as lodes. Each lode is represented by a series of mineralisation wireframes. Tonnage and grade values have been estimated based on 485 diamond drill holes drilled between 2009 and 2015.

Six block models have been created to estimate each of the lodes defined by CGT. Wireframes were constructed of geological domains within each of the lodes and were used to constrain the block model. Blocks that had already been mined were flagged in order to generate results for both unmined and depleted areas. An inverse distance squared estimation algorithm was applied, with composite top-cut grades selected using statistical analysis of the distribution of grade within each domain. The final resource (Table 1.2) is reported at a 0 g/t Au cut-off.

Table 1.2 Mineral Resource summary for the Ballarat East mine as of 31 March 2015. Resources reported at a 0g/t Au cut-off grade.

Category	Mineral type	Gross attributable to licence		Net attributable to issuer (100%)			Contained gold (oz Au)
		Tonnes (t)	Grade (g/t Au)	Tonnes (t)	Grade (g/t Au)	Change from previous update (%)	
Indicated Resources	Gold	79,500	15.9	79,500	15.9	-	40,500
Inferred Resources	Gold	460,000	7.1	460,000	7.1	-22.9%	105,600
Total Resources	**Gold**	**539,500**	**8.4**	**539,500**	**8.4**	**6.7%**	**146,100**

Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t.

Table 1.3 Indicated Mineral Resource estimate, lode by lode for the Ballarat East mine at 0 Au g/t cut-off for 31st March 2015

Lode	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Britannia Mako	4,000	14.2	1,900
Britannia Basking	25,500	12.8	10,500
Llanberris Basking	9,500	7.5	2,300
Sovereign Tiger	19,500	15.6	10,000
Sovereign Gummy	21,000	23.7	15,800
Total	**79,500**	**15.9**	**40,500**

Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

Table 1.4 Inferred Mineral Resource estimate, lode by lode for the Ballarat East mine at 0 Au g/t cut-off for 31st March 2015

Lode	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Britannia Mako	81,000	7.9	20,500
Britannia Basking	182,500	6.8	40,100
Llanberris Basking	36,000	8.1	9,500
Llanberris Mako	49,000	6.3	9,900
Sovereign Tiger	30,500	5.3	5,200
Sovereign Gummy	81,000	7.9	20,500
Total	**460,000**	**7.1**	**105,600**

Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

The company has also calculated a maiden Probable Ore Reserve as summarised in **Table 1.5**. The mine's Reserves are comprised of a number of ore lodes contained within three mine compartments as outlined in **Table 1.6**. The mine is segregated into a series of compartments separated by a series of significant cross-course faults.

Table 1.5 Ore Reserves summary, as of 31 March 2015

Category	Mineral type	Gross attributable to licence		Net attributable to issuer			Remarks
		Tonnes (thousand)	Grade (g/t Au)	Tonnes (thousands)	Grade (g/t Au)	Change from previous update (%)	
Proved	-	-	-	-	-	-	
Probable	Au	129	7.61	129	7.61	100	First report Reserve
Total		**129**	**7.61**	**129**	**7.61**	**100**	**Issuer owns 100% of the company**

Table 1.6 Breakdown of Ore Reserves as of 31 March 2015

Category	Mineral type	Gross attributable to licence		Net attributable to issuer			Remarks
		Tonnes (thousands	Grade (g/t Au)	Tonnes (thousands)	Grade (g/t Au)	Change from previous update (%)	
Proved	-	-	-	-	-	-	
Probable							First report Reserve
Britannia Compartment	AU	41	6.91	41	6.91	-	First report
Llanberris Compartment	AU	9	4.98	9	4.98	-	First report
Sovereign Compartment	Au	79	8.29	79	8.29	-	First report
Total		**129**	**7.61**	**129**	**7.61**	**100**	**Issuer owns 100% of the company**

The Ballarat Mine Indicated and Inferred Resources are based on block models which are constructed using tightly constrained and often quite narrow domain wireframes. Wire-framing is a carried out with an emphasis on constraining the width of the domains to the true widths of the high grade zones within ore lodes. As a result, it is common for the high grade domains to be modelled down to widths between 0.5m and 1.5m.

Current mining methods have a minimum mining width of 2.5m (for up-hole stoping) meaning that a significant amount of planned dilution is required for extraction to occur. This is accounted for during Reserve calculations. In addition to this due to the challenging ground conditions some over-break is included in the Reserve calculations.

The combination of these factors results in a significant increase in tonnes and decrease in gold grade during the conversion from Indicated Resources to Probable Reserves.

1.6 Economic Analysis

All currency values are in Australian dollars (A$) unless otherwise stated and all unit cost references include all operational expenditure associated with the site and exclude all capital related expenditure. Mined ore tonnes for the 2014-2015 year totalled 257,336 t and the site operating cost per tonne of ore mined averaged A$161. Gold ounces sold for the 2014-2015 year totalled 45,503 oz Au, with an associated site cash operating cost per ounce at A$900. The average gold price received per ounce for the 2014-2015 year was A$1,440. The revenue from bullion sales totalled A$66M.

Revenues for the 2015-2016 budget years are calculated assuming US$1,220/oz Au and an exchange rate of 0.81, to give a gold price of A$1,506/oz Au. The plan is to mine 247,000 t at a head grade of 6.9 g/t Au at a site unit operating cost per tonne of ore mined of A$197 for a gross revenue of A$75M. A key objective of the 2015-2016 budget is to ensure sufficient funds are available for the operation to be self-sustaining, including its ability to fund major projects such as the underground diamond drill programme and the sustaining capital expenditure requirements.

The current resource and operation at Ballarat carries a “high” risk. This risk is principally related to high geological and grade variability. To some extent the successful mining operation testifies that a short-term operation is sustainable. At any one time, the mine generally has no more than 12 to 18 months of resource in front of it.

1.7 Risk Assessment

The current Mineral Resource at Ballarat carries an overall “high” risk. The risk is principally relates to geological and grade variability. It is reflected by the predominate use of the Inferred Mineral Resource category.

In-situ sample representivity is likely to be low, given the dominance of coarse-gold present and high-nugget effect – risk rating is thus “high”. Sample type, preparation and assays carry a “medium” risk rating, given the dominance of coarse-gold present. Historically different sample (mass) support, preparation and assaying methods impart some sampling error. Historical and recent QAQC indicates reasonable assay quality; however some results were not to best expectations.

General geological control is based on an approximate 15 m to 30 m grid for diamond drill holes plus mine development. Knowledge of historical mining and recent drilling aids geological interpretation. At the resource stage, there is a lower understanding of small-scale local continuity issues which control variability of tonnes and grade. Best resolution of geological continuity and ore zone complexity is only gained after development. The decision to access an ore zone is generally based on diamond drilling only. The overall geological risk is considered to be “medium-high”.

The grade estimate has a “high” risk due to a high nugget effect. Estimation block size is broadly appropriate to the drill spacing, but does not relate to any Smallest Mineable Unit (SMU) size. The application of cut-off grades on a block by block basis, is considered unreasonable as estimation error will be relatively high. However a cut-off grade has been applied to discrete geological domains within ore lodes based on the expectation that estimation error will be less at the scale of whole geological domains. The current global estimate is reasonable, given that volume is based on a model constrained by drill data and geological interpretation.

Whilst a maiden Probable Ore Reserve has been defined at Ballarat, it is insufficient to underpin a 12 month mine plan, therefore some economic decisions to mine are based on Inferred Mineral Resources – these carry a ‘medium-high” risk. Mine planning and scheduling is carried out with some flexibility built in to allow for change to be implemented efficiently if and when required. The project has established infrastructure and plant in place. Mining costs, parameters and methods are now determined as a result of over 12 months of continuous mining. The processing plant is designed to cope with Ballarat’s typical coarse-gold ore. It can achieve a recovery of around 86-87%. Plant capacity is well within mining rates.

The QPs believe the accuracy of the grade and tonnage estimate for the Inferred Mineral Resources is considered to be within ±35-50% globally based on general experience of this style of mineralisation. Mine reconciliation data over the past two years also supports this range.

Social, legal, political and environmental risks are considered “low”, given the relatively stable and developed nature of Australia.

The current resource and operation at Ballarat carries a “high” risk. This risk is principally related to high geological and grade variability. To some extent the successful mining operation testifies that a short-term operation is sustainable. At any one time, the mine generally has no more than 12 to 18 months of resource in front of it.

1.8 Recommendations

A number of recommendations are made in order to improve the quality of future Mineral Resource estimation. They are as follows:

- Continue on-going geological studies to understand the nature of the mineralisation, in particular controls on grade distribution.
- Implement a formalised management sign-off process for validation of logging and sampling carried out by core logging geologists.
- Undertake a rigorous resource estimation optimisation study to include:
    - Use of de-clustering in statistical analysis of sample grades.
    - Use of variography to determine spatial relationships.
    - Use Qualitative Kriging Neighbourhood Analysis (QKNA) to optimise parent block size and estimation parameters.
    - Investigate the use of kriging (or variant thereof) as an alternative estimation methodology.
- Continue regular collection of density samples and investigate potential to construct a density block model to improve tonnage estimates.
- Continue to refine reconciliation procedures.

In relation to mining:

- On-going review of stoping methods and seek opportunities for improvement where possible.
- Continued rigorous ground control and monitoring, and control of additional mining dilution where possible.
- Reconciliation of mining dilution and over-break by ore style should be implemented in order for over-break and dilution numbers for specific mineralisation styles to be included into scheduling.
- Investigate potential economics of extraction of <2.5m wide zones using alternative mining methods.

2 INTRODUCTION

2.1 Aim and Scope of Report

This Qualified Persons Report ("QPR") has been prepared by Castlemaine Goldfields Propriety Limited ("CGT") for LionGold Corporation Ltd ("LionGold") in compliance with the disclosure requirements of the Singapore Exchange Securities Trading Limited ("SGX"). Mining One Propriety Limited ("Mining One") has verified the contents of this report. Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by LionGold and verified by Mining One.

2.2 Use of Report

The Mineral Resource will be publically reported by LionGold to the SGX and used by CGT to plan mining operations at Ballarat.

2.3 Reporting Standard

The contained Mineral Resource has been reported in accordance with The JORC Code 2012 (JORC, 2012).

2.4 Report Authors and Contributors

Qualified Persons ("QP") for this Qualified Persons Report ("QPR") are listed in Table 2.1.

Table 2.1 QPs for this QPR

Name	Position	Employer	Independent of LionGold	Date of site visit	Professional designation	Contribution to QPR
Mr Peter de Vries	Executive Consultant	(1)Mining One	Yes	Site Visit carried out during February 2015.	MAusIMM & MAIG	All sections. Qualified Person.
Mr Matthew Hernan	Geology Manager	(2)CGT	No	Based on site. Visits mine on a weekly basis	MAusIMM & MAIG	All sections Qualified Person.
Mr Esteban Valle	Resource Geologist	(2)CGT	No	Based on site. Visits mine on a weekly basis	MAIG	Sections 3 to 6, 8 and 16 to 18. Qualified Person.
Mr Philip Petrie	Senior Mining Engineer	(2)CGT	No	Based on site. Visits mine on a weekly basis.	MAusIMM	Sections 9. Section 4 JORC Table 1 Qualified Person.

(1)Address: Level 9, 50 Market Street, Melbourne, VIC 3000, Australia.
(2)Address: 10 Woolshed Gully Drive, Mount Clear, Ballarat, VIC 3350, Australia.

The independent QP, Mr Peter de Vries, has consulted with a wide range of CGT staff and relies on the documentation, reports and other data supplied by CGT in determining the appropriateness of some modifying factors used in determining the metallurgical recovery and throughput, the environmental compliance issues and costs, mining and administration costs, etc. used in this report. Other experts contributed to this QPR under the supervision of the QPs (Table 2.2).

Table 2.2 CGT staff who contributed to this QPR[2]

Name	Position	Employer	Professional designation	Contribution
Mr Jason Fothergill	Principal Geologist/ Tenements Officer	CGT	MAusIMM	Section 3
Mr Matthew Hernan	Geology Manager	Balmaine[1]	MAusIMM	Sections 1 and 2, Compilation and Peer review of report
Mr Darren Watkins	Mine Manager	Balmaine	MAusIMM	Sections 10 to 15
Mr Mark Davies	Processing Manager	Balmaine	-	Section 7 Section 11
Mr Kurtis Noyce	Senior Sustainability Officer	Balmaine	-	Section 3 Section 12 Section 13
Mr Philip Petrie	Senior Mining Engineer	Balmaine	MAusIMM	Section 10
Mr Esteban Valle	Resource Geologist	Balmaine	MAIG	Sections 3 to 6, 8 and 16 to 18.
Ms Toni Griffith	Chief Financial Officer	CGT	Chartered Accountant	Section 15 and Section 16

[1] Balmaine is 100% owned by CGT and operates the Ballarat mine

[2] The staff listed in Table 2.2 are not independent of LionGold

2.5 Qualified Persons Statement

The Qualified Persons ("QP's") responsible for preparation of this QPR are:

- Mr Peter de Vries - Consultant with Mining One, is a member of the AusIMM and the AIG and has 29 years of experience in the mining industry.
- Mr Matthew Hernan – Geology Manager with CGT is a member of the AusIMM and the AIG and has 13 years of experience in the mining industry.
- Mr Philip Petrie – Senior Mining Engineer with CGT is a member of the AusIMM and has 30 years of experience in the mining industry.
- Mr Esteban Valle - Resource Geologist with CGT is a member of the AIG and has 11 years of experience in the mining industry.

All QPs have visited the Ballarat mine within the preceding three months to 31st March 2015.

Mr de Vries is independent of LionGold. Messers Hernan, Petrie and Valle are not independent of LionGold.

The QPR is intended to be read as a whole, and sections or parts thereof should therefore not be read or relied upon out of context. Unless otherwise stated, information and data contained in this report or used in its preparation was provided by CGT.

The effective date of this QPR is 31st March 2015.

2.6 Basis of the Report

This report presents a Mineral Resource estimate undertaken by Mr Valle and Mr Hernan, and a Probable Ore Reserve estimate undertaken by Mr Petrie. The resource is reported in accordance with The JORC Code (2012). The database and geological model used to estimate the resource was compiled by CGT. Mining One has reviewed all data prior to estimation. The resource was estimated using Vulcan software.

Other data has been supplied by members of the Ballarat mine team (Table 2.2).

The QPs have reviewed all input data, models and outputs in this QPR and believe that they are appropriate and permit the Mineral Resource to be reported in accordance with The JORC Code (2012).

3 PROJECT DESCRIPTION

3.1 Project Overview

The Ballarat goldfields have a long history that goes back to the discovery of gold in 1851. Underground mining of quartz veins started in the late 1850's and continued until 1918. The historical quartz mines at Ballarat East occur along a narrow corridor some 400 m wide and approximately four kilometres long with typical mined depth of 350 m (maximum 500 m). Recorded underground gold production totalled 1.6 Moz at an average recovered grade of 9 g/t Au. Some unknown degree of underground hand sorting of ore and waste during mining operations is suspected by most gold historians and geological practitioners studying the gold field.

Ballarat Goldfields NL (BGF) commenced exploration of the Ballarat East goldfield in 1985 with the aim of demonstrating the geology that controlled the deposits of the goldfield extended at depth below the historical mines. The potential for repetition of gold bearing structures with depth, along strike and across a series of parallel fold structures offered highly prospective exploration targets.

Drilling between 1985 and 1991 confirmed the exploration model and an initial resource was delineated. After more than 10 years of exploration, the Woolshed Gully decline was commenced in 1995 to access the underground resource. In 1996 the decline development was suspended and the decline placed under care and maintenance due to declining gold price, and the associated difficulty of raising additional funds for capital mine works. Decline development through the wet and friable weathered portions proved slower and at higher cost than estimated in the initial feasibility studies.

In October 2002 BGF was recapitalised. In late 2003 exploration drilling resumed from both the Woolshed Gully decline and from the surface, and development of the decline recommenced in late 2004. Some trial mining was carried out and ore processed through the gravity plant in late 2005.

Lihir Gold Limited (LGL) acquired the project in 2007 via a merger with BGF costing approximately A$400M and invested in excess of A$290M for the extension of the underground development and processing plant expansions with the aim of developing the project to mine 600,000 tpa for target production of 200,000 oz Au of gold. In late 2008 stope production commenced in the southern end of the deposit, at which point it was evident that the mining resource blocks in these areas were more variable and discontinuous than previously modelled.

During 2008, LGL mined 129,000 t at a grade of 3.5 g/t Au and during 2009, mined 105,000 t at a grade of 4.3 g/t Au.

In July 2009, the project was scaled down after a review was completed that determined the project would not sustain large scale bulk mining techniques as forecast. By mid-2009 total gold production from the Ballarat East operation was approximately 29,000 oz Au.

Castlemaine Goldfields Propriety Limited (CGT) entered into an agreement to acquire the Ballarat tenement package including the mill, various equipment and substantial mine development from LGL in March 2010 for an acquisition cost of A$8.6M (A$4.5M and assuming a A$4.1M rehabilitation bond) plus a 2.5% royalty on future production, capped at A$50M (to Newcrest Mining Ltd). Transfer of ownership occurred in May 2010.

The mineral licences which comprise the Ballarat Gold Project are held by Balmaine Gold Pty Ltd which is a wholly subsidiary of CGT. Licence transfer to Balmaine occurred in May 2010.

CGT underground exploration activities were focussed on the northern exploration targets on the First Chance and Sulieman anticlines, in the Llanberris compartment with 15,000 m of diamond drilling completed in the period between May-December 2010. Exploration success lead to the completion of a feasibility study targeting gold production of 40,000 to 50,000 oz Au per annum. Underground mining activity recommenced in March 2011, with the aim of accessing the Llanberris Mako fault zone, the process plant was recommissioned and first gold production occurred in September 2011.

As well as Ballarat, CGT owns other nearby tenements in Castlemaine (60 km to the north-east), Tarnagulla (80 km to the north) and Sebastian near Bendigo (120 km to the north-east) covering a collective area of approximately 378 km^2.

LionGold became the majority shareholder of CGT in August 2012. LionGold is a Singapore listed mining company with interests in Australia and Ghana. Other LionGold Australian interests include a stake in Signature Metals Ltd, Citigold Corporation, Brimstone Resources Ltd, Unity Mining Ltd and A1 Consolidated Gold Ltd.

The Ballarat gold project site is located to the south of the City of Ballarat, approximately three kilometres from the city centre, and approximately 100 km west of Melbourne.

Figure 3-1 Ballarat mine location

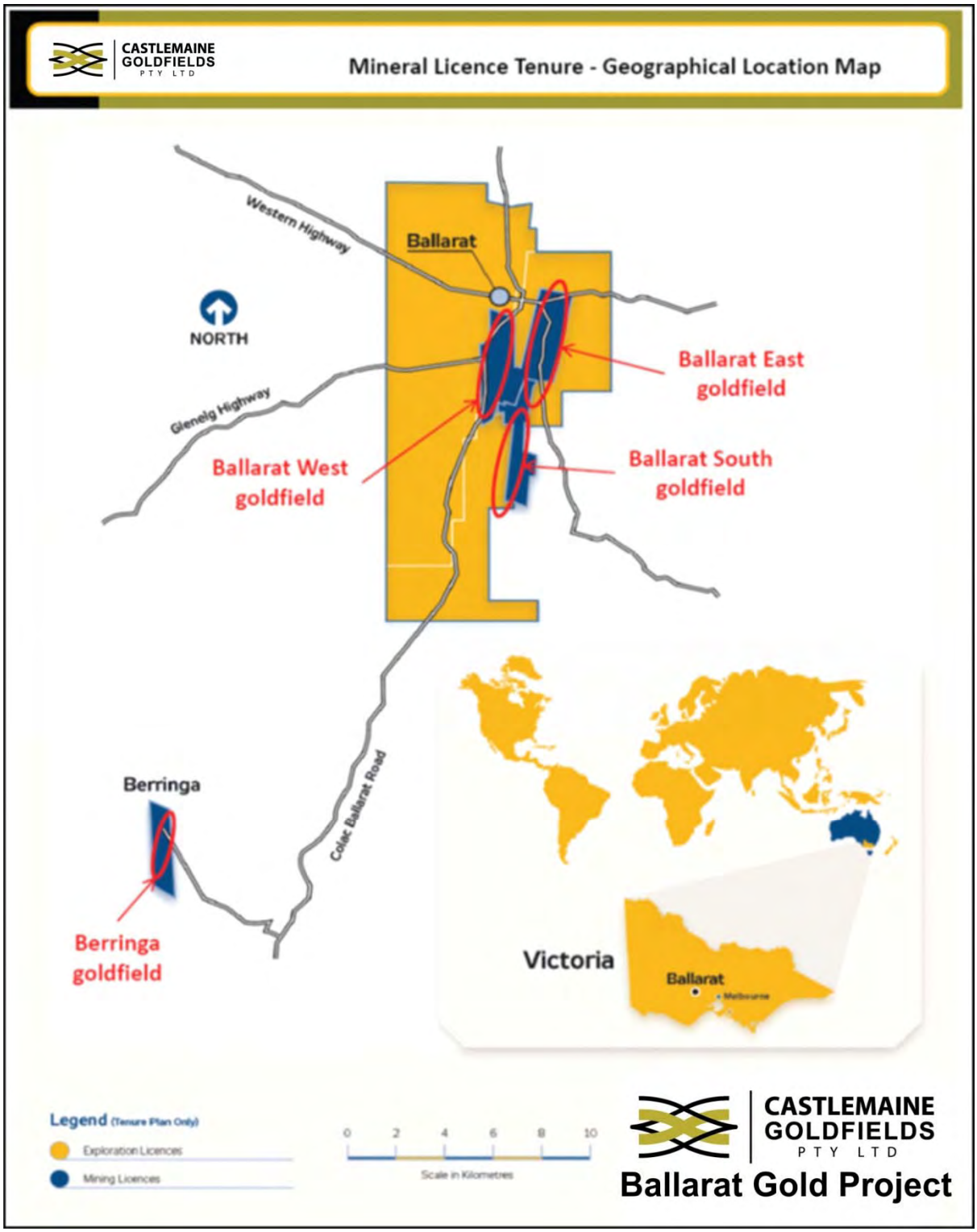
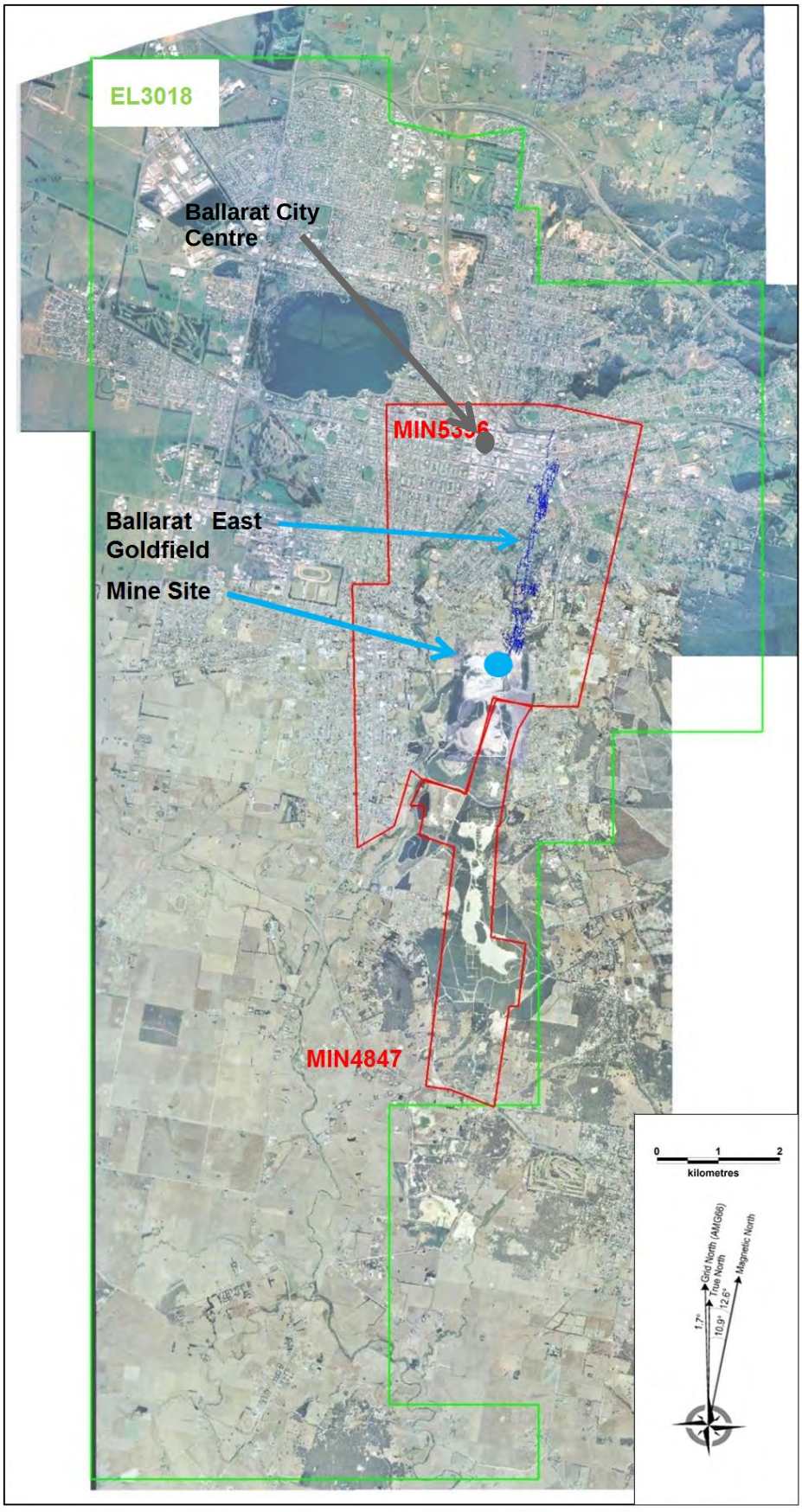


Figure 3-2 Location of Ballarat mine tenements



3.2 Tenure

CGT hold the mining tenements listed below in Table 3.1 through its 100% owned subsidiary Balmaine Gold Pty Ltd (Figure 3-2). The tenements cover the major historic hard rock mining areas of the Ballarat East, Ballarat South and Ballarat West goldfields.

The resources being reported on are located entirely within Mining Licence MIN5396. This Licence is wholly contained within Exploration Licence EL3018. The tenements are in good standing with the regulatory authority, with all required bonds and permits in place to allow mining operations to be carried out.

Table 3.1 Tenure details for Ballarat mine. All tenements held 100% by Balmaine, a wholly owned subsidiary of CGT

Asset name/ Country	Issuer's interest (%)	Development Status	Licence expiry date	Licence Area	Type of mineral, oil or gas deposit	Remarks
Ballarat, Australia	100%	Mining	4/10/2023	14.86 km^2	Gold, platinum, silver	Mining lease (MIN5396)
Ballarat, Australia	100%	Mining	1/11/2019	4.10 km^2	Gold, platinum, silver	Mining lease (MIN4847)
Ballarat, Australia	100%	Exploration	3/10/2015	153 km^2	Gold, platinum, silver	Exploration lease (EL3018)

3.3 Tenure Conditions

The Ballarat gold project consists of the two mining licences MIN5396 and MIN4847, surrounded by the exploration licence EL3018. Conditions for mining licences in Victoria, Australia require that mining activity be current or will not cease for a period of greater than two years. The current operations of CGT satisfy all conditions for the ongoing maintenance of mining leases.

Conditions for tenure of exploration licences in Victoria are based on a combination of exploration activity and expenditure determined by the government under the Mineral Resources (Sustainable Development) Act (MRSDA) rules.

The mining licences of CGT at the Ballarat gold project are administered under the Mineral Resources (Sustainable Development) Act 1990 (MRSDA) along with conditions imposed by other local and State government agencies as discussed below:

An Environment Effects Statement (EES) was approved in September 1988.

A Planning Permit was issued by Shire of Buninyong in September 1993 and subsequently extended by City of Ballarat until September 2027.

The authority to commence work for MIN4621 (one of several licences now amalgamated as MIN5396) was granted on 11 November 1993, and full-scale mining and ore processing now proceeds under that authority.

The Work Plan for the Ballarat East licences was approved in 1993 under the MRSDA for development of the underground access, dewatering, ventilation shafts, process plant (including the use of cyanide), tailings and waste rock storage facilities, services and rehabilitation.

Subsequent variations to the Work Plan were granted for; rehabilitation works near Elsworth Street (1994), the Golden Point ventilation intake shaft (2008, 2009 and 2012), the Terrible Gully tailings storage facility (2005) and a concrete batching plant (2005).

A waste discharge licence (EX258) issued by the Environment Protection Authority allows for discharge of treated mine water to Yarrowee river.

An area Work Plan for exploration was approved in 2008.

The project area covered by Work Plans spans some 153 km^2 in the City of Ballarat. Land tenure within the project area consists of both freehold and Crown Land managed by a range of entities, as could be expected in a regional city. The land managers include; City of Ballarat, Central Highlands Water (CHW), Hancock Victoria Plantations, Sporting Clubs, private land owners and various other Committees of Management.

Crown Land includes that reserved for particular purposes, restricted crown land and unrestricted crown land.

The dominant land use at Ballarat is residential. In the immediate vicinity of the mine site, the land is managed by CGT, CHW and Hancock Victoria Plantations for forestry purposes. Scattered amongst the residential areas are industrial estates, pockets of Crown Land managed for community purposes and small parcels of land used for agriculture.

To the north, urban development is relatively close spaced and Crown Land areas are smaller and closer to residential developments. Towards the south the Crown Land blocks become much larger and the residential development is much sparser.

Table 3.2 outlines each domain's land tenure (CGT internal document - Conceptual Rehabilitation Plan and Bond Review MIN5396–Ballarat East Project, March 2013).

Table 3.2 Specific domain holdings of land tenure

Domain	Property address	Crown allotment/title plan	Current owners
Infrastructure-administration, water ponds, core shed, workshop, warehouse, process plant	10 Woolshed Gully Drive	CA 2A Section 16, Parish of Ballarat	Central Highlands Water-Deed of Assignment to Balmaine Gold Pty Ltd to operate over Crown Land
West side of tailing storage facility	10 Woolshed Gully Drive	CA 2A Section 16, Parish of Ballarat	Crown Land Central Highlands Water-Deed of Assignment to Balmaine Gold Pty Ltd to operate
East side of tailing storage facility and process water tank	10 Woolshed Gully Drive	CA 10K Section 12 Parish of Ballarat	Crown Land - (Forestry) Victorian Plantations Corporations Act – Deed of Assignment to Balmaine Gold Pty Ltd
Process plant ROM Waste rock bund Decline (portal)	10 Woolshed Gully Drive	CA 2B Section 16, Parish of Ballarat	Crown Land - (Forestry) Victorian Plantations Corporations Act – Deed of Assignment to Balmaine Gold Pty Ltd
Golden Point shaft	508 Grant Street, Golden Point	CA 2012, Section 102, East Parish of Ballarat	Freehold - Balmaine Gold Pty Ltd
Llanberris	York Street	16~99\ PP503199	Presumed to be original Work Authority. Presume City of Ballarat holds title for Llanberris Athletics track.
Otway Street	Otway Street, Ballarat East	CA 23, 8A and 4C all of Section 38A – East of Otway Street, Crown Allotment 27 and 23 both of Section 83 – West of Otway Street	Certificate of Title Vol 8386 Folio 827 CA23,CA8A, CA43 Crown Land with a use agreement yet to be discovered. Part of mining lease.
Elsworth Street – substation, North Prince Extended and South Woah Hawp	Elsworth Street	CA 9B, Section 14 Parish of Ballarat	DSE Crown Land – Original Work Authority
North Woah Hawp	Elsworth Street	CA 32, Section 101 Parish of Ballarat 32	Sovereign Hill – Presumed to be included in original Work Authority

3.4 Access

Ballarat is accessible by air, rail and road transport, all link to the capital city of Melbourne, which has both domestic and international air terminals.

Ballarat Aerodrome is located 7.4 km north west of Ballarat in the outer suburb of Mitchell Park in Victoria. This public airport is operated by the City of Ballarat. Standing at 437 m above sea level, Ballarat airport contains three runways; two asphalt and one grass. The aerodrome is a former Royal Australian Air Force (RAAF) base. There are no regular commercial flights available to and from Ballarat at this stage in time. Commercial flights can be taken from Melbourne Tullamarine via a range of domestic and international carriers.

Rail access to Ballarat is well established and rail services to Melbourne have only recently been upgraded to VLOCITY trains. The station is also utilised as a hub for coach services, both private and public.

As a central city in Victoria, roads through Ballarat range from the first grade freeway to Melbourne to lesser roads connecting to all other parts of Australia. Roads within the City of Ballarat are regularly maintained to the highest standards.

3.5 Climate

The Ballarat region has a moderate climate. Its elevation, 435 m above sea level, causes its mean monthly temperatures to be 3 to 4°C (5.4 to 7.2°F) below those of Melbourne. The mean daily maximum temperature for January is 25.1°C while the mean minimum is 10.9°C. In July, the mean maximum is 10.0°C (50°F); the mean July minimum is 3.2°C.

The mean annual rainfall is 693 mm, August being the wettest month with an average of 75 mm. Like much of Australia, Ballarat experiences cyclical drought and heavy rainfall.

Light snowfall typically falls on nearby Mount Buninyong and Mount Warrenheip at least once a year, but only in the urban area during heavy winters. Widespread frosts and fog are common during the cooler months.

Ballarat's highest maximum recorded temperature was 44.1°C on 7 February 2009. The lowest recorded minimum was -6.0°C on 21 July 1982.

The variations in average temperature and rainfall throughout the year for the Ballarat mine are shown in Table 3.3 and shown graphically in Figure 3-3 and Figure 3-4.

Table 3.3 Climate indicators for Ballarat mine

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Mean maximum temperature (°C)	25.1	25.0	22.2	17.7	13.6	10.8	10.0	11.4	13.9	16.6	19.6	22.6
Mean minimum temperature (°C)	10.9	11.5	10.0	7.4	5.7	4.0	3.2	3.7	4.8	6.2	7.8	9.4
Mean rainfall (mm)	39.1	44.6	42.6	51.1	64.5	62.8	66.5	75.0	71.5	67.0	56.2	50.7

Figure 3-3 Mean maximum and minimum temperatures for Ballarat mine

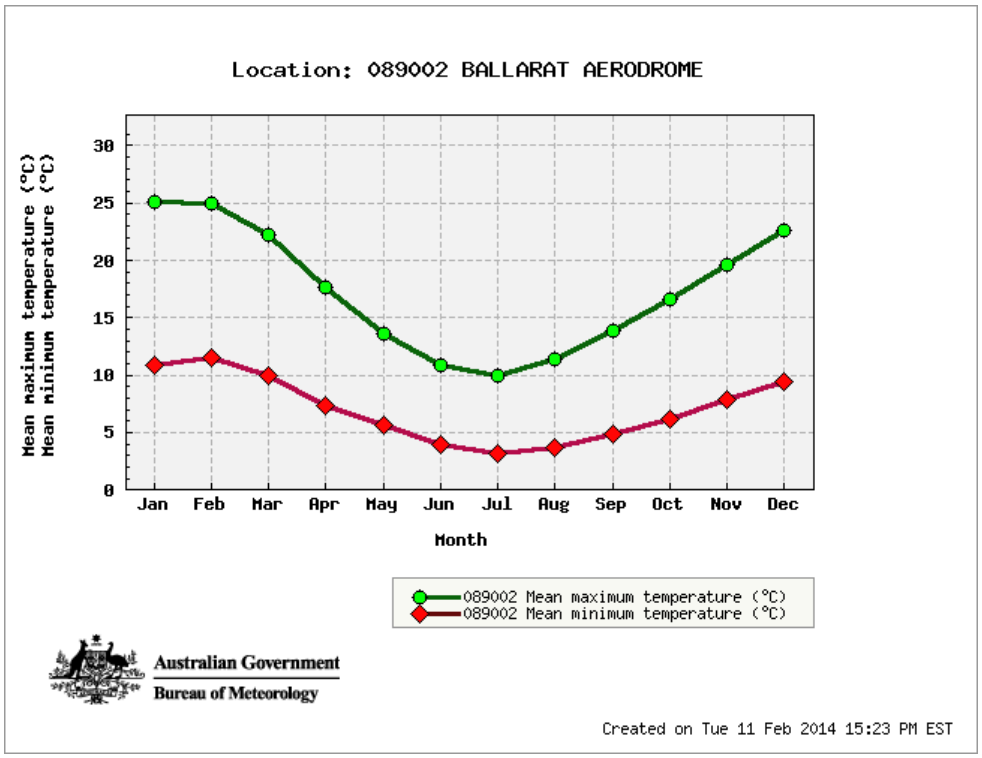
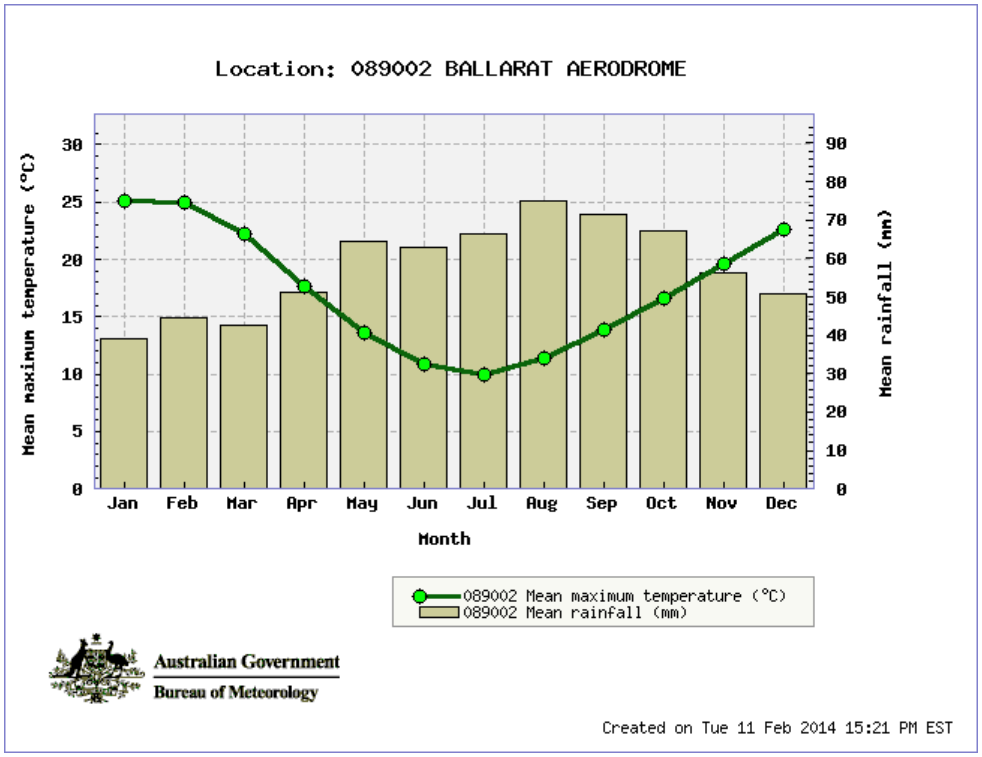


Figure 3-4 Mean maximum and minimum temperatures for Ballarat mine



3.6 Landforms and Soils

Ballarat's location is in the southern foothills of the Great Dividing Range at an elevation of 400 m above sea level. The range runs east-west through the State and divides the southern coastal plains from the northern alluvial plains of Tertiary age Murray basin sediments, which extend north and westwards into the adjoining states.

The city is located in the region known as the Central Highlands, this is due to the mature, undulating topography and the lack of significant mountains. The city itself is centred on the Yarrowee river, a seasonal stream, in a shallow basin surrounded by extinct volcanic cones up to 730 m above sea level and associated basalt flows and outcropping sedimentary rocks. The surrounding areas have been cleared for grazing and cropping with higher areas utilised for commercial pine plantation and the preservation of areas of remnant native forest.

The soils surrounding Ballarat are of two major types, the fertile red soils from the basalt flows have been extensively cleared for agriculture, while the poor siliceous soils generated on the Palaeozoic bedrock hills are covered in stringybark eucalypts with little understory.

3.7 Fauna and Flora

The majority of land occupied by the Ballarat mine and exploration tenements was heavily disturbed during the 1850's gold rush resulting in remnant native vegetation being limited to smaller patches occurring on private property, road and rail reserves and some Crown Land reserves.

No flora studies have been undertaken at the project site since the BGF Environmental Effects Statement (EES) in 1987. The predominant land use of the project site is currently softwood plantation, owned and operated by Central Highlands Water (CHW) and Hancocks Limited. Some of the area is densely covered by noxious weeds particularly gorse, broom and blackberry (Work Plan 1993).

The active mine site is located within a parcel of Crown Land-Forestry gazetted for commercial softwood production. The native vegetation has therefore been heavily modified and primarily consists of pinus radiata. The conservation status of the remnant native vegetation that remains within the mine site area is listed as of "least concern" under the state native vegetation framework.

The Ballarat mine and exploration projects are located within the Central Victoria Uplands, which is dominated by Dry Foothill Forests and seven other Ecological Vegetation Classes (EVC) types including some vulnerable and endangered communities.

The EVC and Bioregional Conservation Status are listed below in Table 3.4 gives indication to the present condition of these communities.

Table 3.4 EVC's occurring within the City of Ballarat on private land or roadsides

EVC code	EVC name	Bioregional conservation status[1]	Area (ha) [2]
20/22	Healthy dry forest/grassy dry forest	Least concern/depleted	3294
47	Valley grassy forest	Vulnerable	203
55	Plains grassy forest	Endangered	83
23	Herb-rich foothill forest	Vulnerable	50
128	Grassy forest	Vulnerable	22
175	Grassy woodland	Endangered	17
164	Creekline herb-rich woodland	Vulnerable	15
83	Swampy riparian woodland	Endangered	6

[1] *Department of Sustainability and Environmental bioregional conservation assessment of EVCs (DSE 2005)*

[2] *Approximate area based on mapping of existing vegetation undertaken for this project overlayed with the DSE pre-1750 Ecological Vegetation Class layer to estimate distribution if vegetation types.*

The exploration project area may contain flora species which are either listed as 'threatened' under the Flora and Fauna Guarantee Act 1988 (Vic), or listed as 'critically endangered', 'endangered', or 'vulnerable' under

the Environmental Protection and Biodiversity Conservation Act 1999 (Commonwealth). The project area may also contain flora species which are shown on Department of Environment and Primary Industry (State) databases as either 'endangered', 'vulnerable', 'rare', or 'poorly known'.

Site selection in areas which may contain such species will include a vegetation survey prior to work commencing. For exploration sites with native vegetation present the governments Native Vegetation Plan "avoid, minimise and offset" approach is adopted.

Preference is always given to locating drill hole collars in areas where the surface has previously been disturbed. As a consequence, drilling will be sited on open parkland, on tracks or on degraded land in preference to areas containing native vegetation or those areas that hold some public value for recreation.

During the development of the Environmental Effects Statement and mine site operations fauna observations and studies have been completed.

These studies have identified over 130 native and exotic reptiles, amphibians, birds and mammals. A web search (28/3/14) of the project area on DEPI-Biodiversity Interactive Map 3.2 indicated twenty nine threatened species (27 birds, 1 mammal and 1 Amphibian) with three listed (1 bird, 1 mammal and 1 amphibian) on the IUCN Red List of threatened species. The IUCN Red List species are not recorded for the mine site but may occur within the exploration lease.

Within the mine site and at surface drill sites there is a possibility of the presence of native fauna including birds, marsupials and reptiles. When selecting sites, consideration will be given to the protection of possible fauna roosts, nests and general habitats and to lifecycles to ensure minimal disturbance to breeding cycles with the aim of having no disruption to native fauna.

3.8 Hydrology

The site is located within the Barwon river basin sub-catchment of the Corangamite Catchment.

Two waterways are located adjacent to the site, the Canadian creek and Yarrowee river. EPA Waste Discharge Licence EX 258/7 provides for discharge of wastewater to the Yarrowee river, and whilst not currently in use, the licence also has provision to allow discharge into the Canadian creek.

Yarrowee river is a small permanent stream that drains the City of Ballarat and surrounding areas, and runs through open grazing country to join the Barwon river which flows into Bass Strait near Geelong. The Yarrowee river is used for stock watering and for some irrigation.

3.8.1 Ground Water

Groundwater in the project area occurs by direct infiltration of rainfall into the aquifer outcrop areas via vertical leakage and lateral flow. The regional groundwater flow pattern is complex and is likely to be locally controlled by topography. Quality of the groundwater is controlled by composition and quality of recharge water, aquifer lithology, depth to water table and residence time within the aquifer. Groundwater residence time is in turn dependent upon hydraulic conductivity of individual aquifers, the hydraulic gradient, flow path length and recharge potential.

Regionally, the groundwater salinity of the Ordovician sediments lies within the range of 3,000 mg/l to 7,000 mg/l. Groundwater quality tends to deteriorate southwards as distance from the recharge areas increases.

The Ordovician rocks have low primary porosity and permeability. The frequency and interconnection of joints, fractures, shears and faults control their capacity to store and transmit groundwater and they are generally regarded as poor aquifers. Recharge into the aquifers is typically less than 50 mm per year.

The main area of potential groundwater impact is around the tailings storage facility (TSF). Groundwater is located in the underlying Ordovician sediments at depths varying from approximately 23 m below ground surface (in the catchment area above the TSF) to approximately 11 m below ground surface (at the toe of the TSF wall). Hydraulic conductivities are generally low and flow direction appears to be westerly towards Yarrowee river, consistent with the local topography. Groundwater is brackish and is classified as Segment C in accordance with the State Environment Protection Policy (SEPP) (Groundwaters of Victoria).

The low hydraulic conductivities and poor quality restrict the abstraction of groundwater for any beneficial uses and this is reflected in the absence of any groundwater bores (with the exception of investigation wells) within a one kilometre radius of the TSF.

As per the TSF Work Plan Variation (2005), potential leakage from the TSF is monitored for by monthly sampling for a range of metals, indicator ions and water quality parameters at nine groundwater-monitoring bores located at the site. Additional bores (over the above nine) will be constructed and monitored if considered necessary to assess changes in the groundwater conditions.

The impact of mine dewatering on the groundwater in the region was addressed in the BGF Environmental Effects Statement prepared in 1987; it was concluded that the resultant lowering of the water table will not have a significant effect on the users in the area.

3.8.2 Surface Water

The Ballarat mine site is located approximately 500 m east of the Yarrowee river. A significant proportion of the environmental flow in Yarrowee river during the summer months is provided by outflows from the CHW's Ballarat South sewage treatment plant (19 ML/day average) and CGT-Ballarat's mine water discharge (1.6 ML/day).

The mine has operated under EPA Licence EX258 since 1989, this licence allows for treated mine water to be discharged to the environment. Initially the mine was dewatered into the Canadian Creek at the Llanberris site, this ceased in the early 1990's when the dewatering commenced at the current mine site location with discharge to the Yarrowee river.

CGT has 17 years of water quality data for the Yarrowee river both upstream and downstream of the mine site and as it makes its way through the treatment process. A range of monitoring is undertaken to ensure changes in water quality as a result of the operation are quickly identified and where required action taken

3.9 Cultural Environment

There are several sites of European and Indigenous significance within the EL3018. Preference will always be given to areas where cultural heritage features have not been identified to carry out work. Consultation will occur with the relevant Registered Aboriginal Party (RAP) to ensure an appropriate assessment is completed prior to work being undertaken.

4 HISTORY

Gold was discovered by James Hiscock in the Ballarat district during August 1851. It was followed by the discovery of gold at Mount Alexander in September and at Bendigo in December of the same year.

Figure 4-1 Yarrowee river, Ballarat goldfield (circa 1860)



The Ballarat goldfield is the second largest gold producer in the state of Victoria as shown in Table 4.1.

Table 4.1 Hard rock and alluvial gold production history for the Central Victorian goldfields (Phillips and Hughes, 1998)

Goldfield	Total Gold (t)
Bendigo	697
Ballarat	**408**
Castlemaine	127
Stawell	82
Creswick	81
Walhalla	68
Maldon	65
Woods Point	52
Clunes	47
Chiltern	46

From the site of the initial discovery, the early diggers commenced a broader search which led to the discovery on 24th August of gold at Golden Point in the heart of what is now known as the Ballarat goldfield. It is also near the centre of the now urban area of Ballarat.

Gold mining commenced in the gravels of the Yarrowee river (Figure 4-1) and its tributary gullies which drained eastwards and westwards of the White Horse range of hills which are the location of the Ballarat East goldfield. By 1854 the search for the continuation of the auriferous gravels had been proven not to follow the modern day drainage of the Yarrowee to the south.

The western side of the Yarrowee river is a basalt capped plateau and while the gullies draining from the plateau contained some gold they were not as rich as those draining from the Whitehorse range. The early miners were of the opinion that the basalt was the summit a deep-seated mass of rock.

However, the lack of a southern outlet for the gold-bearing streams led the diggers to follow the gullies underneath the basalt from the east. As the work progressively moved westwards under the plateau, shaft sinking commenced. The locating of the leads by shafts was a speculative, labour intensive method which was soon superseded by boring. The first being commenced in 1856 and ushered the era of deep lead mining, the mining of gold from alluvial gravel from buried drainage systems underground at depth of up 150m.

The discovery of gold bearing quartz was made at Black Hill, to the north of the Black Hill range in 1851. The first recorded attempts to extract the gold was in early 1853 by the breaking of quartz with hammers while the first use of a stamp battery to break the quartz was in April 1853 by the use of a wind powered machine.

The Deep Lead mines dominated the gold production from their inception to the end of the 1860's when the distance from the sources of the gold increased and the lack of efficient exploration methods lead to the exhaustion of the gold resources within the major lead systems. However, the discovery of gold-bearing quartz in-situ in the bed of the alluvial gutter in a deep lead in Ballarat West led to the hard rock mining era in western part of the Ballarat goldfield.

As the shallow alluvial deposits in the Ballarat East goldfield (White Horse Range) were exhausted the miners commenced developing the auriferous quartz veins which they had encountered during alluvial mining. The first quartz mine had commenced in 1853 at Black Hill at the northern end of the field, where an extensive open cut and system of tunnels was developed. Other small quartz mines were developed on the conspicuous outcrops of quartz on the range. The oldest company reported as working on the Ballarat East field was the Llanberris Quartz Mining Company, which commenced operations in 1858. The development of the field continued through the 1860's and 1870's.

In 1887 a period of intense development on the field was instigated when two diamond holes were put down by the Victorian Mines Department in the northern end of the field. Both holes intersected auriferous quartz at depth, which provided confidence to the miners that gold was present at depths beyond that of the deepest workings.

Figure 4-2 Looking south from Sovereign Hill, Ballarat East goldfield, New Normanby poppet head in foreground (Gregory, 1907)



Mining continued on the Ballarat East goldfield until 1917 when mine workings reached a depth of between 600 m to 700 m. Increasing costs, difficulties with mining at greater depths, flooding of abandoned mines and the outbreak of the First World War led to the closure of the mines. Estimates of gold production from the Ballarat East goldfield are shown in Table 4.1.

No significant gold mining or exploration took place until Ballarat Goldfields NL commenced work in the mid 1980's.

4.1 Prior Ownership and Ownership Changes

The tenements held by CGT cover the areas of the Ballarat East and the Ballarat West goldfields where the whole of the historically significant, hard rock mining at depth was carried out.

In 1983, Kinglake Resources Pty Ltd was granted Mining Lease 1158 which covered the major mines in the Ballarat East goldfield. The lease was then transferred to Ballarat Goldfields Limited which then became Ballarat Goldfields Pty Ltd (BGF).

BGF was granted further mining leases, mining area licences and an exploration licence within and surrounding MIN1158.

In 1998, BGF acquired Lease 4953, EL3391 and EL3714 which covered the major historic mining area of the Ballarat West goldfield by the purchase of Phoenix Resources NL.

In 2003, the Mining Leases and Mining Area Licences were renewed as Mining Licences: MINs 5396 (formerly 1158) 5397, 5398, 5399, 5400, 5401 and 5402. MINs 4621, 5037 and 5038 were granted during this period.

In 2004, BGF acquired MIN4847 adjoining the southern end of MIN5396.

This marked the first time in history that the entire Ballarat goldfield, the East and West fields, had been held under licence by one entity.

On 29th September 2005, MINs 5396, 5397, 5398, 5399, 5400, 5401, 5402, 4621, 5037, 5038 and 4593 were amalgamated into MIN5396.

On 16th November 2005, ELs 3018, 3714 and 3391 were amalgamated as EL3018.

On 4th April 2006, MIN5444 located between the Ballarat East and Ballarat West goldfields was granted to BGF.

In 2007, BGF merged with Lihir Gold Limited.

The tenements were then purchased by CGT from LGL in March 2010.

On 12th January 2011, MIN5444 was amalgamated with MIN5396.

4.2 Previous Exploration and Development Work

MIN5396 held by CGT covers a four kilometre strike length over the most historically productive section of the field. Quartz mining within the present MIN5396 was carried out from the late 1850's until 1917. During this period, the mines within MIN5396 worked to an average depth of 350 m and produced a recorded 1.02 Moz Au.

Prior to 1985 Ballarat Goldfields Ltd undertook detailed studies of the historical records of the Ballarat East mines. It was clear that the mines closed due to economic circumstances and not due to termination of the gold/quartz mineralisation. The company developed an exploration philosophy that if the gold bearing structures worked in the old mines could be shown to extend below the lowest levels of the old mines and that their gold content was likely maintained with depth. A nominal exploration target of 1 Moz Au was considered possible in the zone from 350 m to 700 m below surface.

Between 1985 and 1988, Ballarat Goldfields NL carried out a programme of diamond drilling to test for continuation of mineralisation below the old mines. Approximately 8,000 m of diamond coring was drilled along a strike length of 400 m. The results confirmed the existence of significant gold quartz mineralisation. Data obtained from this drilling programme is presented in Canavan and Hunt (1988).

During 1991 a further 11,000 m of diamond drilling was carried out under a joint venture between Ballarat Goldfields NL and North Broken Hill-Peko. This drilling tested for mineralisation beneath the old mines and extended the tested strike length from 400 m to 2,800 m. Results of this phase of drilling are detailed in O'Neill *et al*. (1992).

In 1994, a decline located at Woolshed Gully was commenced to access a resource delineated by Livingstone and d'Auvergne (1992). In 1996, the decline development was suspended without having reached its target and placed on care and maintenance.

In 2003, exploration drilling resumed from both the Woolshed Gully decline and surface locations. Between 2003 and 2009, 23,108 m of underground development and 246,977 m of drilling was completed. During this time BGF merged with LGL. In February 2010, LGL suspended operations.

4.3 Historical Mineral Resource Estimates

A resource estimate (Olsen and Cox, 2006) for BGF, identified a total resource of 3.9 Mt at 11.3 g/t Au for 1.4 Moz Au including an Indicated Mineral Resource of 0.6 Mt at 13.4 g/t Au for 240 Koz Au and an Inferred Mineral Resource of 3.3 Mt at 10.9 g/t Au.

An updated resource estimate was produced by BGF in 2007 (Carney and Cox, 2007). This contained a total resource of 3.9 Mt at 11.8 g/t Au for 1.5 Moz Au comprising an Indicated Mineral Resource of 0.9 Mt at 10 g/t Au for 310 Koz Au and an Inferred Mineral Resource of 3 Mt at 12.4 g/t Au.

CGT produced an initial Inferred Mineral Resource estimate for the Mako fault zone in the Llanberris compartment in November 2011 of 100,000 t at 10.5 g/t Au for 33,100 oz Au.

An updated Inferred Mineral Resource estimate to include the Britannia compartment, for total of 263,000 t at 8.5 g/t Au for 71,700 oz Au was released in August 2012.

A further resource update to include the Sovereign and Victoria compartments was released in December 2013. The total Inferred Mineral Resource was 411,000 t at 8.5 g/t Au for 112,200 oz Au.

The March 2014 Independent Qualified Persons report released on the SGX stated the Mines Inferred Mineral Resource to be 370,000 t at 11.5 g/t for 137,000 oz Au.

4.4 Reliability of Historical Estimates

Since commencement of operations at Ballarat, CGT has carried out a continuous drilling programme to delineate resources not accessed by the previously. This has increased the sample density and geological knowledge, resulting in changes to the interpretation of grade and geological continuity.

Compared to CGTs interpretation of lode continuity, previous interpretations were based on excessive extrapolation. The resource estimates were audited by a number of independent parties and found to be appropriate. However the assumptions of lode continuity were not valid. As a result estimates made by either Lihir or BGF have not been utilised by CGT.

4.5 Production History

Gold production from the commencement of production in 2005 is tabulated below in Table 4.2. Gold production recommenced in 2005 and is tabulated to the end of 2013 in Table 4.3.

Table 4.2 Gold production history for the Ballarat East goldfield to 1917

Source	Alluvial (Moz Au)	Hard rock (Moz Au)
Baragwanath [1]	4.0	1.18
Fothergill [2]	-	1.70
d'Auvergne 2007 [3]	0.8	1.38

[1] – Memoir 14 (1923). Hard rock data taken from 46 mines extending from Black Hill to the Woah Hawp No.1 mine. Alluvial data estimated by BGF based on production figures from 1851 to 1862.
[2] – Unpublished BGF report. Data collated from Baragwanath (1923), DNRE GIS data (2000), Canavan (1988) and Bowen (1974). Includes estimates of unaccounted gold e.g. theft, incomplete records etc.
[3] – Unpublished report. Extensive analysis of hard rock gold production across the Ballarat East field incorporating data from 244 mines. Does not take into consideration gold theft, etc. Alluvial estimate is incomplete and does not include gold escort returns between 1851 and 1862.

Table 4.3 Gold production history for the Ballarat East goldfield from 2005 to March 2015

Company	Year	Tonnes (t)	Grade (g/t Au)	Ounces Produced (oz Au)
BGF	2008-2009	234,000	3.85	28,965
CGT	2011-2012	57,346	5.00	7,189
CGT	2012-2013	170,663	6.60	29,066
CGT	2013-2014	170,291	8.50	39,962
CGT	2014-2015	250,664	6.78	46,039
Total	**2008-2015**	**882,964**	**6.18**	**151,221**

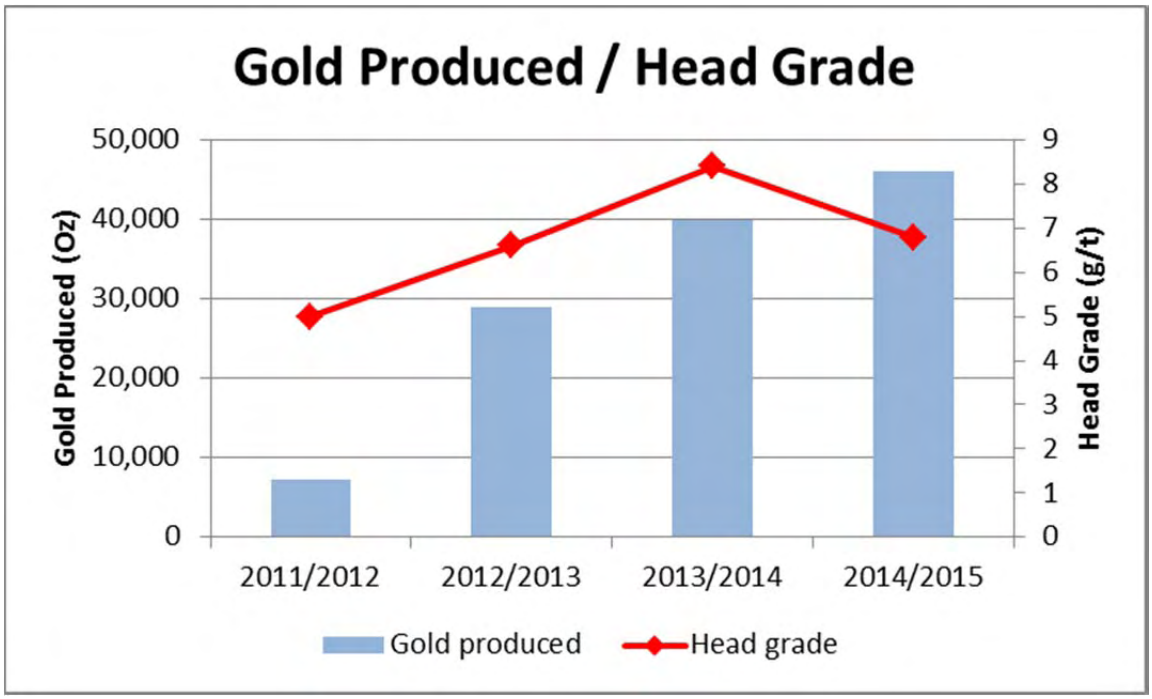
A summary of CGT production from the Ballarat Mine in the period from August 2011 to March 2015 is included in Table 4.3 and Table 4.4.

Table 4.4 Summary of Ballarat Production

Activity	Unit	Year			
		2011/2012	2012/2013	2013/2014	2014/2015
Ore tonnes mined	t	57,346	170,663	170,271	257,336
Ore milled	t	56,771	167,996	170,392	250,664
Head grade	g/t Au	5.0	6.6	8.4	6.8
Total gold recovery	%	83%	85%	88%	83.6%
Gold produced	oz Au	7,189	29,066	39,962	46,039
Gold sold	oz Au	6,010	29,065	39,528	45,503

Note: Production Years in Table 4.4 match the LionGold Financial year which runs from April to March.

Figure 4-3 Annual CGT gold production from the Ballarat mine



5 GEOLOGICAL SETTING

The Ballarat gold field is located in the south-western part of the Lachlan Fold Belt (LFB) in the subdivision called the Bendigo–Ballarat zone. The location of which is shown in Figure 5-1

5.1 Regional Geological Setting

Summaries of regional and local geology are found in Taylor *et al*. (1996) and Vandenberg *et al*. (2000) and the references contained therein. The geology of the Ballarat East goldfield and the forms and control of the mineralisation are described at length in Gregory and Baragwanath (1907), Baragwanath (1923), Canavan and Hunt (1988), d'Avergne (1990), Osborne (2008) and Fairmaid *et al.* (2011).

Ballarat is located in the south-western part of the LFB in Victoria, within the Palaeozoic sedimentary rocks of the Bendigo-Ballarat subdivision. The outcropping bedrocks of the region are graptolite-bearing Ordovician age turbidites of the Castlemaine Super Group which comprises the majority of the bedrock of the Bendigo-Ballarat zone of the LFB in Victoria.

To the north of the region the Ordovician rocks are covered by Tertiary-age shallow water sediments of the Murray Basin, to the south they are overlain by Miocene marine sediments. East and West of Ballarat, Quaternary age basalt flows cover the Ordovician rocks.

The Ordovician turbidites comprise clastic sediments ranging in grain size from hemi-pelagic shales to coarse sandstones and rare grits. They were deposited in submarine turbidite fans by large and rapid from influx of sediment from the west into a deep water environment. They have undergone regional low-grade metamorphism to lower green-schist facies.

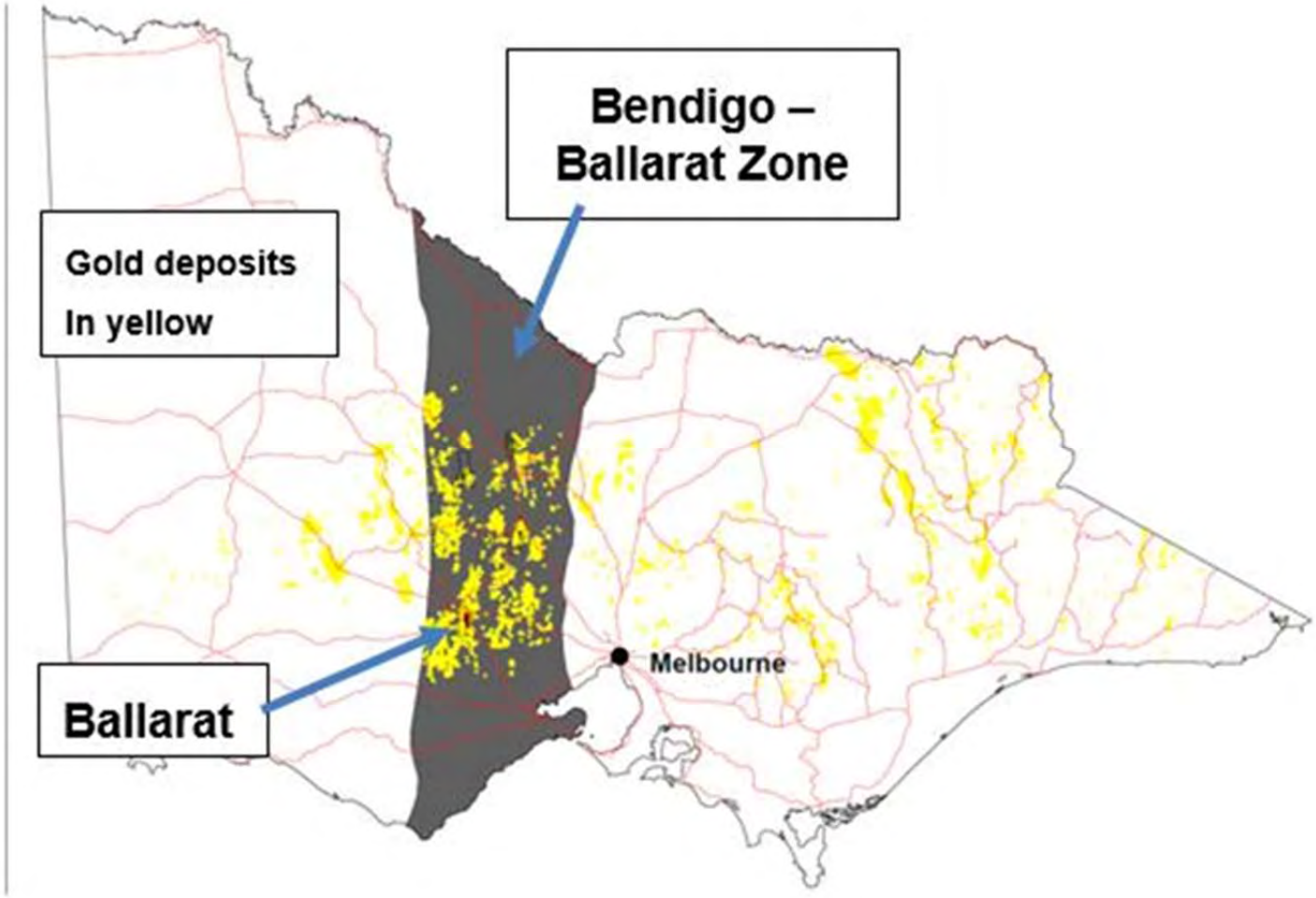
The Quaternary volcanics consist of basaltic lavas and pyroclastics which are part of the volcanic flows spread across south western Victoria. Four flows have been identified in the Ballarat area. No mineralisation has been recorded to occur within them.

The sediments have been folded into north-south striking of anticlinoriums and synclinoriums during the Benambran Orogeny, with wavelengths of individual folds ranging across all scales up to 500 m. They have been intruded by a suite of Devonian age granites and, locally, by Jurassic lamprophyre dykes.

Regional scale, north-south striking, steeply west-dipping reverse faults occur across the zone resulting from the compressional events during the Benambran orogeny and have been interpreted to be related to the formation and the distribution of the numerous gold deposits in the region, Bendigo and Ballarat being the largest.

The gold occurs in quartz veins associated with second and third order faults in the hanging wall of the first-order regional scale faults. The gold occurs in the quartz primarily as free particles in close association with the sulphide mineral suite of pyrite, arsenopyrite, sphalerite and galena.

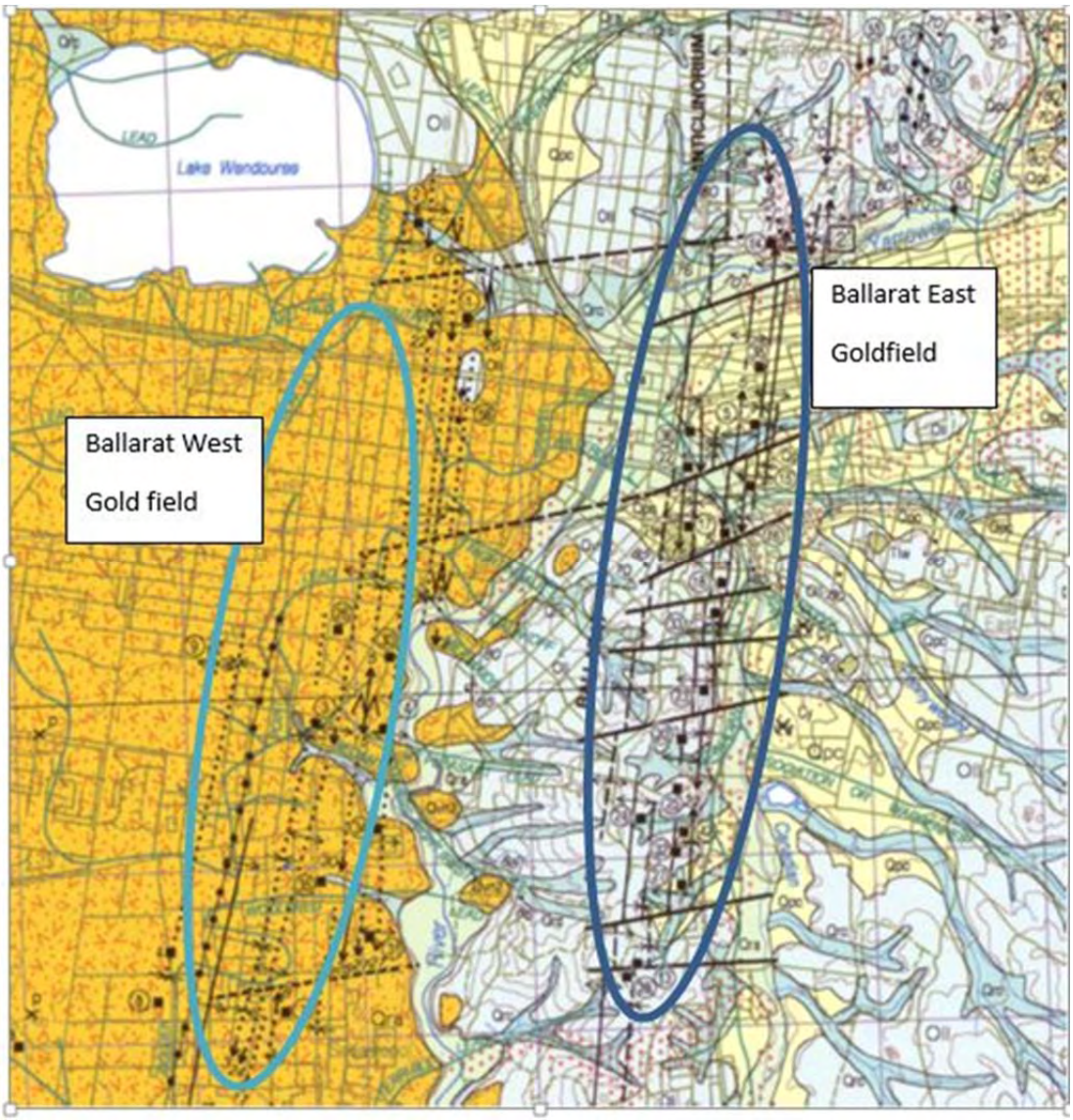
Figure 5-1 Plan of Victoria showing location of the Bendigo-Ballarat zone and gold deposits in yellow



5.2 Local Geological Setting

The Ballarat goldfield is located in the hanging of the regional north-south striking, west-dipping reverse, Williamson Creek fault. Although covered by basalt flows in much of the area it is exposed to the south and has been inferred by deformation and fold patterns in the Neerina area.

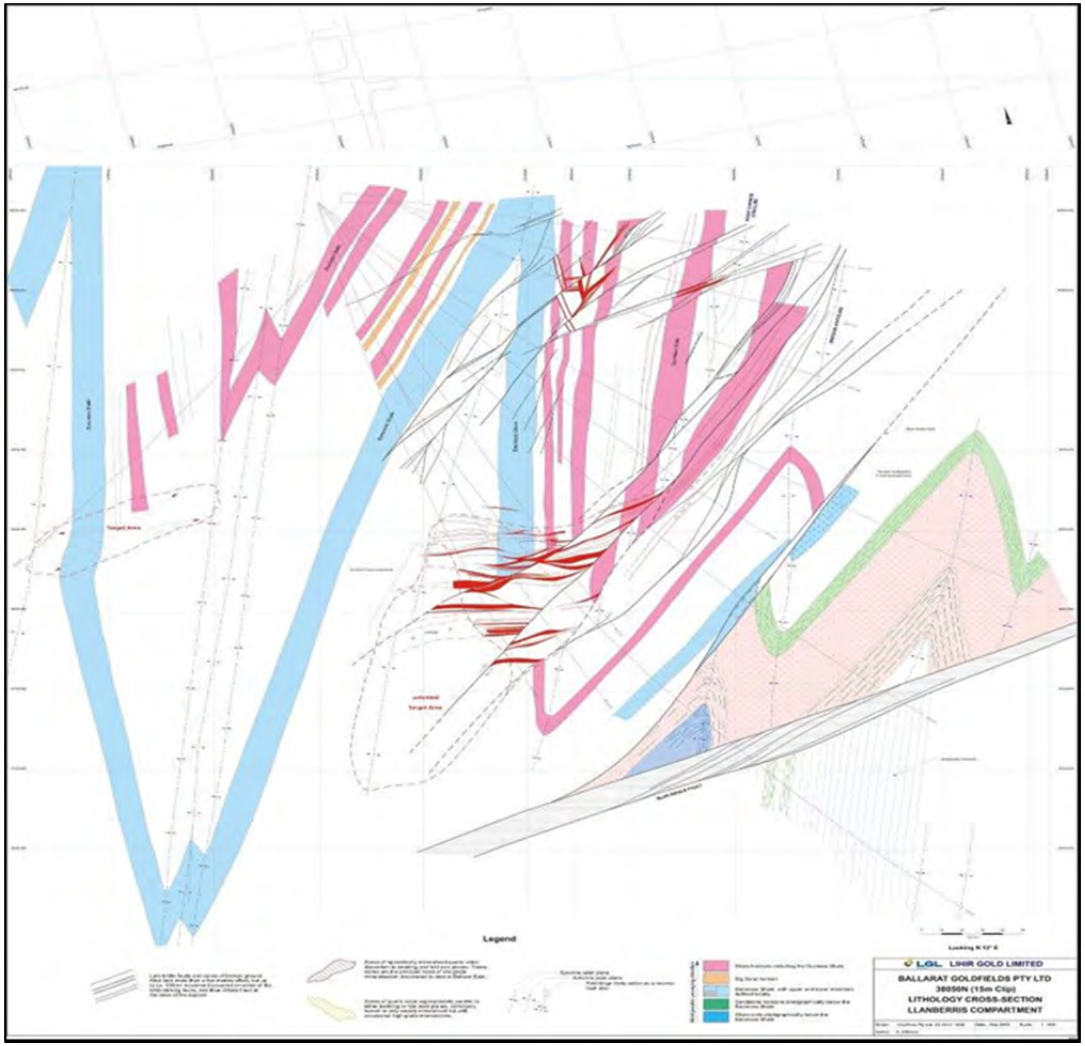
Figure 5-2 Extract from 1:50,000 Geological Map, Geological Survey of Victoria (1995) (Not to Scale)



In Figure 5-2 features on the plan are as follows Basalt flows - Dark yellow, alluvial deposits - Light yellow, Ordovician sediments – light purple. The black squares are historic mine shaft locations and the thick solid black lines are the cross course faults which are used to subdivide the goldfield into areas called compartments.

The bedrocks of the goldfield are strongly weathered and possibly metasomatised Ordovician sedimentary rocks which range in grain size from pelagic black shale to coarse grain sandstones. The rocks have been folded into a series of upright chevron-style anticlines with wavelengths ranging from 50 m to 300 m with numerous parasitic folds occurring around the hinge zone of the larger folds (Figure 5-3).

Figure 5-3 Geological Interpretation of the First Chance anticline on the Ballarat East goldfield at the 38050 mN section (Allibone, 2009)



To the east of the goldfield, the rocks have been intruded by the unmineralised Gong Gong granodiorite which has a contact metamorphic aureole of hornfelsed sediment up to 200 m.

The sediments are covered by Quaternary basalt to the south, east and west of the goldfield and at Ballarat consist of four flows with a total thickness of up 150 m. The flows have filled in the pre-existing drainage forming the Deep Lead deposits exploited early in Ballarat's mining history. They are part of the extensive basaltic plains to the south and west of Ballarat.

The Ballarat goldfield is comprised of three spatially separate areas, Ballarat East, Ballarat West and Neerina. The mineralisation occurs in the same manner in each area with gold particles in quartz veins with minor sulphides. Each has a distinct structural style and the relationship between the areas remains to be resolved. The work reported on here has been carried out solely in the Ballarat East goldfield.

5.3 Mineralisation

Mineralisation at Ballarat is *orogenic* in character. The vein systems can generally be described as those forming at temperatures of between 200°C and 300°C at 1,500 m and 4,500 m crustal depth. They are associated with regional tectonic features and dominated by grossly tabular ore bodies (e.g. veins and stockworks). They bear ore minerals of native gold, chalcopyrite, arsenopyrite, galena, sphalerite, etc.; and are associated with variable intensities of wallrock alteration.

An alternative classification of *turbidite-hosted* (often black shale) *quartz-carbonate veins* is also suggested. The Central Victorian goldfields of Australia are considered to be classic examples of black-shale/turbidite-hosted quartz-carbonate vein systems.

The Ballarat mineralisation shows many similarities to other "slate-belt" hosted deposits such as those in Wales, UK (Dolgellau gold-belt), Bolivia (Liphichi-Maria Louisa belt), Canada (Nova Scotian Meguma Terrain) and China (Guizhou province).

5.3.1 Evaluation Style of Mineralisation

The gold-quartz veins at Ballarat are characterised by a high-nugget effect and the presence of coarse, often visible gold particles (>100 µm in size). This is typical of most Central Victorian gold deposits (e.g. Bendigo, Castlemaine, etc.). They rank amongst the most difficult of ore deposits types, in terms of producing an accurate and precise resource estimate (Dominy, 2014). Their effective sampling is generally difficult because of the relatively low concentrations involved and the erratic and dispersed nature of the gold particles. Resource risk in these systems comprises (1) grade, (2) geological and (3) estimation risk. Significant risk relates to tonnage ("geological risk") and grade ("grade risk").

Grade risk is often greater than geological risk in high nugget systems, though the effect of the latter should not be understated. Grade risk is related to information that should be based on quality sampling and assaying data from drilling and/or underground development. In coarse gold systems, this may be difficult without specialised protocols.

Geological risk is related to the identification of economic volumes from both geological and grade data (i.e. drilling and/or underground development) and must consider continuity of both geology and grade at various scales. Challenges relate to the presence of the host structure with no mineralisation through to barren zones within mineralisation.

Estimation risk includes additional factors such as database quality, survey data, data density, bulk density and estimation methods.

In high-nugget gold veins, the following resource evaluation characteristics are often observed (Dominy, 2014):

- relatively long geostatistical range and low-moderate nugget effect for the background gold mineralisation population;
- short geostatistical range and high-extreme nugget effect for the high-grade (coarse-gold) mineralisation population;
- relatively wide-spaced drilling (>30 m) likely to understate grade whereas dense close-spaced sample data (<15 m) may approximate grade;

- grade estimates are highly sensitive to grade distribution, sample support and type (e.g. volume-variance effect); data density (e.g. information effect) and estimation approach (e.g. sensitivity to top-cuts, block size, search ellipse, etc.);
- it may only be possible to define a global grade for each zone of mineralisation dependent upon data spacing: and
- confidence in tonnage is usually higher than confidence in the grade estimate.

These lead to challenges such as (Dominy, 2014):

- a vein and/or ore shoot may not have the contained gold in the mineable bodies with the shapes, sizes, grades and distributions expected; and
- the boundaries and internal grade distribution of the defined bodies may not be correctly assigned ahead of mining, resulting in either/or excessive dilution and/or misclassification of ore as waste.

Within the Central Victorian high-nugget environment, diamond drilling alone is generally only able to define global Inferred Mineral Resources unless drill spacing is unrealistically tight. Underground development, closely spaced in-fill drilling (<15 m) and/or bulk sampling are potentially needed to define local Indicated Mineral Resources and Probable Ore Reserves. The definition of Measured Mineral Resources and Proven Ore Reserves is effectively impossible, though ultimately dependent upon the nature of the deposit, and data quality and density.

5.3.2 Ore Shoots and Grade Distribution

The term “ore shoot” in the context of the mineral resources at Ballarat East can be correlated with the ore domains which are components of the individual resources and are described in the sections for each resource below.

The domains are determined by the geological interpretation of faulting, quartz veining and assay grades from the logging of drill core. The quartz veining is associated with the west-dipping reverse faults in the eastern limb of the anticlines however the controls on the formation of the veins are yet to be determined. It is possible the localisation veining may be controlled by either the formation of dilation zones due to faults cutting across bedding, the interaction of splay faults oblique to the strike of the main trends or the competency contrast between sandstone dominant and shale dominant lithologies within the host lithological sequence or the interaction of some or all of the above.

No variography has been carried out on the samples contributing to the resource however the observations made on the distribution of grade throughout the resource suggest the Llanberris and Sovereign compartments are more highly endowed than other compartments and the mineralised zones located on the Sulieman anticline are comparable in grade to those on the First Chance anticline. For individual resources the grade tends to be concentrated on the margins of larger quartz bodies, particularly where the margins are faulted. Vein arrays in the hanging walls of the west-dipping reverse faults have an erratic grade distribution. A plan to analyse, confirm and quantify these observations is to be developed in the coming year.

Major west-dipping faults have good geological continuity with the Mako fault identified in four compartments over two kilometre of strike length, mineable lengths are limited by offsets on major cross-course faults, with continuity varying between 50 m and 250 m between faults. East-dipping vein arrays are observed to have less strike continuity, varying from 20 m to 150 m in strike extent. Grade continuity within the major west- and east-dipping structures is variable, however zones of high grade (above 10 g/t Au average grade) have been observed to persist for strike extents between 20 m and 85 m.

5.3.3 Local Mineralisation

The gold-bearing quartz reefs of the Ballarat East goldfield occur entirely within Ordovician sandstones, siltstones and shales that are weakly metamorphosed to lower green-schist facies and are tightly folded about north trending axes. Western limbs of anticlines dip approximately 70^{o}W, eastern limbs 85^{o}W to 75^{o}E and fold axial planes dip approximately 80^{o}W. The regional strike of the bedding is northerly. The auriferous quartz veins are located in fault zones predominantly within the eastern limbs of the folds.

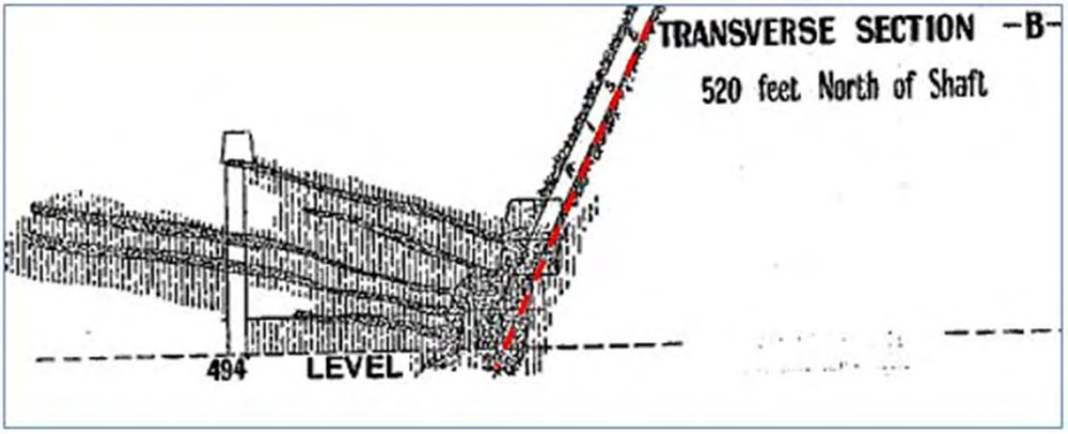
The Ballarat East goldfield contains two known productive lines of reef, located on anticlines of the same name, the (western) Sulieman Line and the (eastern) First Chance Line (Figure 5-3).

Each line of reef (a zone that contains the lodes) has been divided into local geological domains called “compartments” which are defined by a series of major sub-vertical brittle faults which obliquely cross cut the goldfield at semi-regular intervals.

The majority of gold mined occurred in a semi-continuous series of lodes associated with 45^{o} west-dipping reverse faults that cross-cut the vertical to overturned eastern limb of the First Chance anticline (Figure 5-3). The major fold is continuous along the length of the goldfield with a gentle northerly plunge. More than 90% of the gold produced came from the 90 m to 110 m wide eastern limb of this fold.

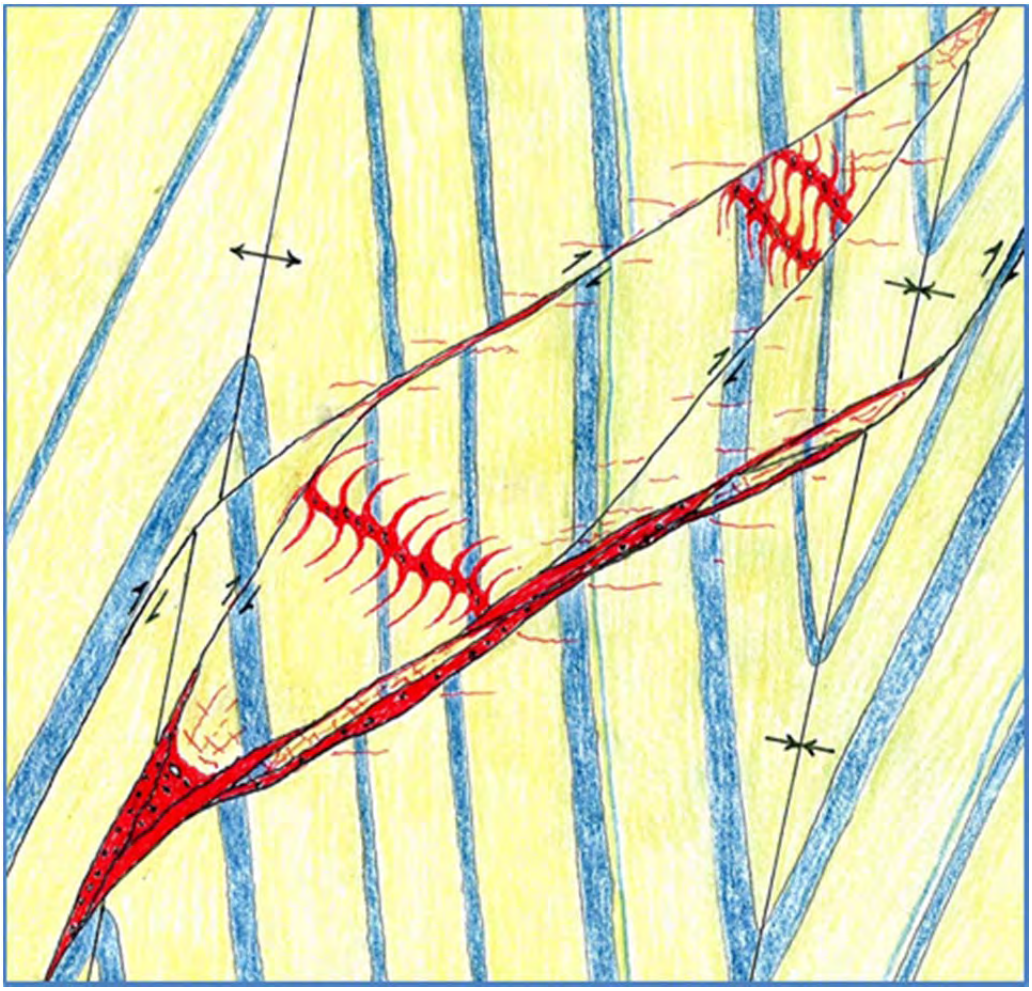
Analysis of historical mining showed that 80% of the gold was produced from the west-dipping fault zones or from sub-horizontal to east-dipping tension veins associated with discrete vertical faults. Gold was also produced from shale hosted “indicator” beds (11%) or from fold hinge zones. The size and dimensions of the fault zones varied significantly. Comparison of historic mapping with recent mining experience confirmed that all mineralised quartz veins within the fault zones are highly variable in geometry and size (Figure 5-4 and Figure 5-5).

Figure 5-4 Sketch of geology demonstrating historical geological model



Flat spurs ("makes") are distributed along the hanging wall of west-dipping fault (or leather jacket) highlighted by red dashed line, (from Baragwanath,1923)

Figure 5-5 Cross section looking north showing relationship of west-dipping fault lodes



Cross section looking north showing relationship of west-dipping fault lodes and east-dipping fault veins. (Sandstone beds in yellow, shale beds in blue, quartz veins in red)

East-dipping faults and veins

Steep to moderately east-dipping vein arrays developed between the major west-dipping faults have been observed, suggesting they are conjugate structures to the faults, although the arrays may not be synchronous with a single phase of faulting. The east-dipping vein arrays do not extend more than 50 m along strike. They occur at discrete intervals adjacent to the west-dipping faults, which by contrast have mineralisation which extends up to 250 m along strike.

Figure 5-6 shows a stacked array of flat-dipping quartz veins which across a 3 m distance increase in dip from shallowly west-dipping to steeper sigmoidal shaped veins centred on a 50^{o} to 65^{o} east-dipping shear zone, with 1 m to 2 m of dislocation of the veins. Figure 5-6b shows an example of post-veining bedding-parallel fault-slip which is interpreted as dextral strike-slip (the veins are shallowly north plunging).

The volume of narrow quartz veins developed as vein arrays may have occurred prior or close to the initiation of the west-dipping reverse faulting across the field (Allibone, 2009). The veined areas may have relieved initial stress and facilitated pathways for the reverse faults, which continued to modify (brecciate and over print) proximal veins to the faults. Later fault and veining events are likely to have a dominant control on the deposition of gold.

Tension veins

Sub-horizontal to east-dipping quartz vein arrays comprise a significant proportion of the veining in Ballarat East, and are well-documented in the literature. An example is shown in Figure 5-7. They are interpreted as being generated by reverse movement on the major west-dipping faults.

En-echelon vein arrays and cleavage parallel veins

Particular stratigraphic horizons appear to control quartz veins as either cleavage or bedding-parallel veins and vein arrays (Figure 5-8). The pervasive cleavage is sub-parallel to the bedding across the First Chance east limb and displays a control on some generations of veining.

Major west-dipping fault zones

The Mako Fault Zone (MFZ) in the Llanberris compartment is an example of a west-dipping reverse fault zone (Figure 5-9). The fault zones dip between 20^{o} and 50^{o} degrees, extend up to 250 m along strike (north-south), 90 m down dip and range in thickness from 0.5 m to 6 m. Veining comprises a combination of massive quartz, weakly laminated quartz, brecciated quartz and stockwork veins. Later faults offsetting early stage veining has been observed amongst a complex zone of shearing and fault gouge development.

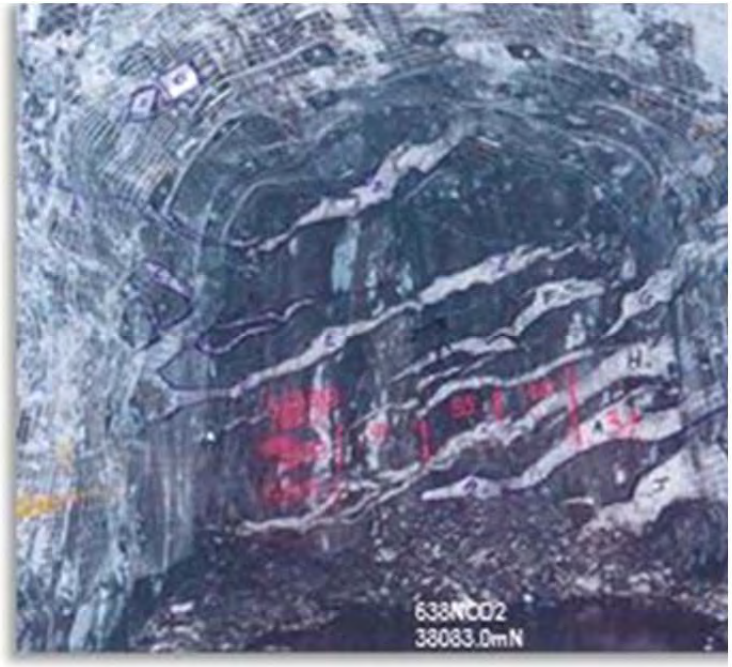
The faults, however, can potentially extend throughout the length of the goldfield. The MFZ has been traced for 2,500 m and may extend beyond the limits of current testing.

Mineralisation along the faults is intermittent, with the controls of formation still to be determined.

An example of the type and extent of the veining which occurs in the hanging wall of the major west-dipping faults is shown in Figure 5-10.

Figure 5-6 Face photos 3 m apart from the lode mined in the LLB638 level north ore drive 2 (NOD2). View to north





Figures 5.6a (left) and 5.6b (right). Face photos 3 m apart.

Figure 5-7 “Chinese Dragon” tension vein extends 50 m across the strike of bedding in the WHD567 access drive

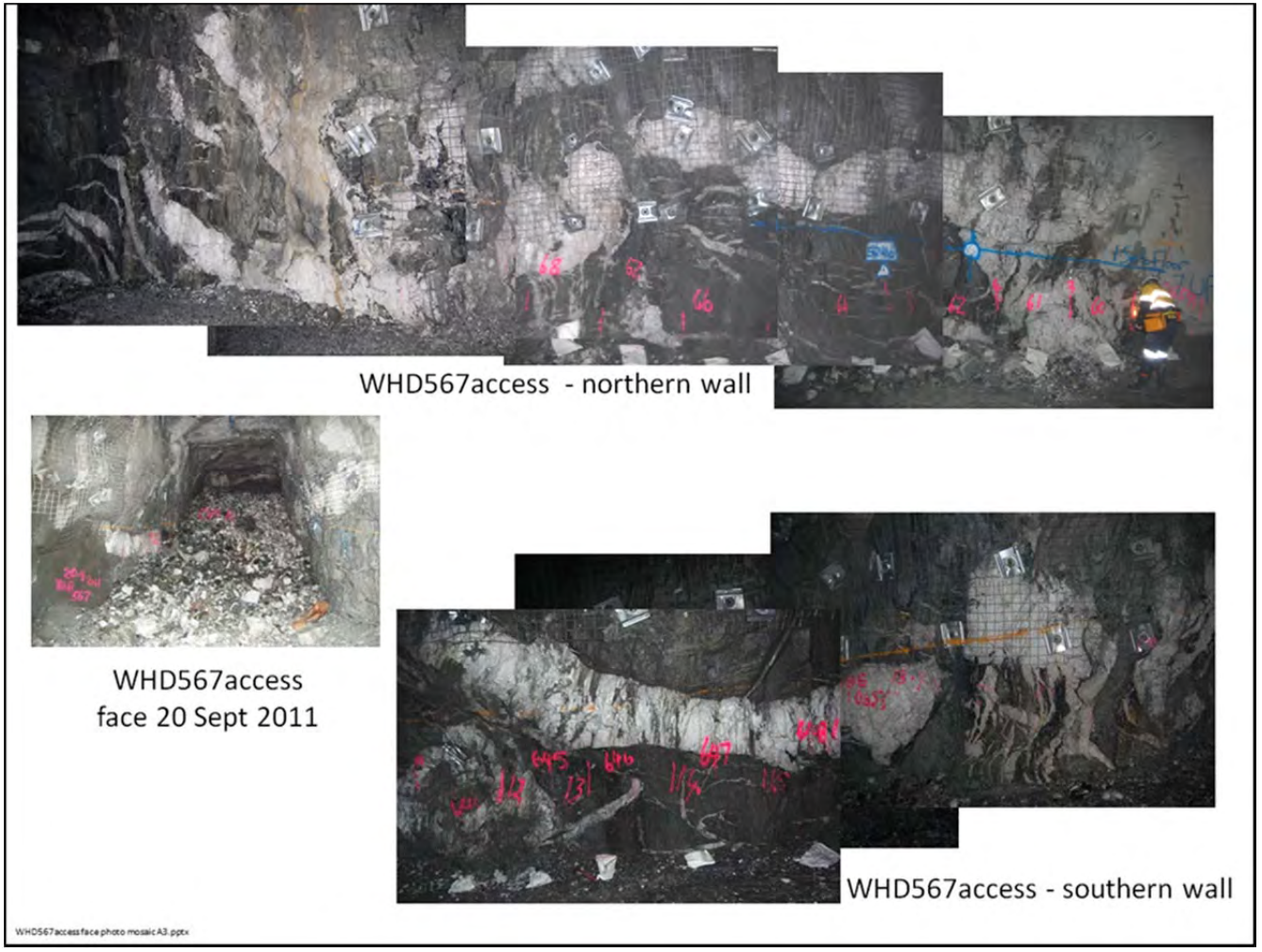
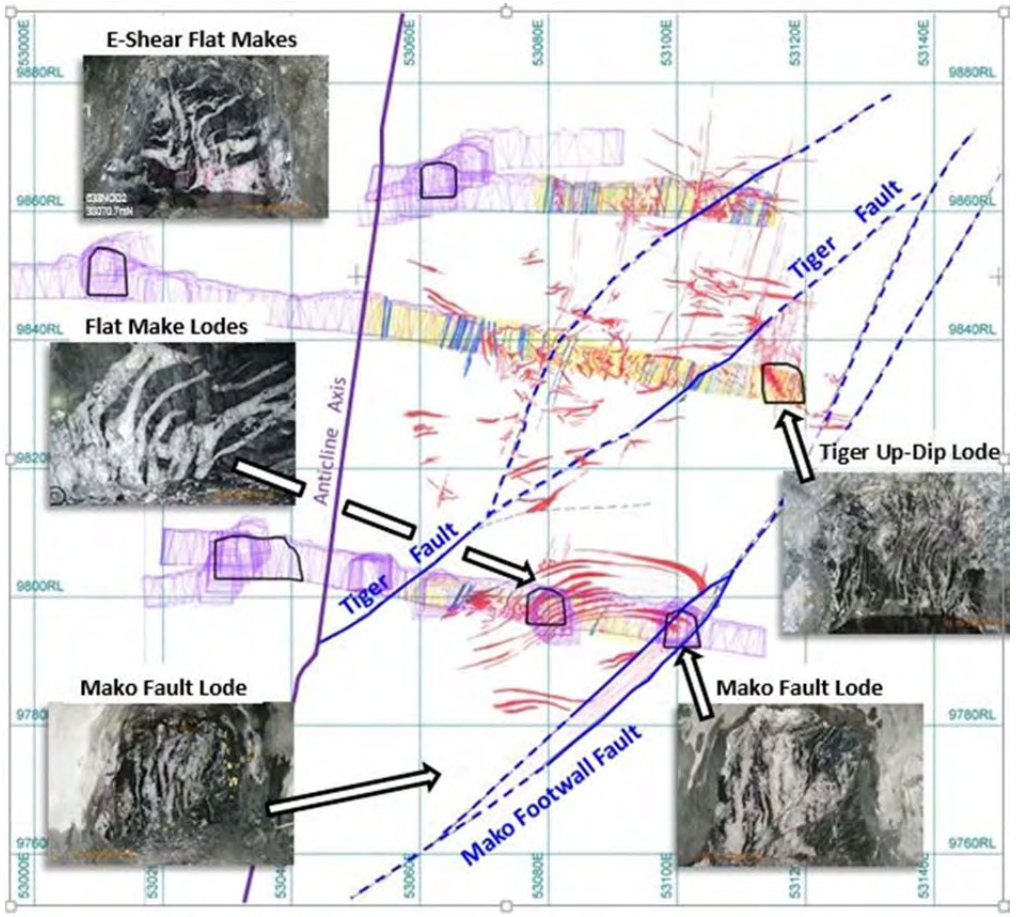


Figure 5-8 North facing photos taken from the Tiger Up-Dip lode sill drive (LLB596NOD1)



Figure 5.8a (left) and 5.8b (right). North facing photos taken from the Tiger Up-Dip lode sill drive (LLB596NOD1). West-dipping fault planes and veined shears (shallow & steep) are evident in 4.8a, with 4.8b showing a shallow east-dipping shear of veins amongst cleavage parallel veins.

Figure 5-9 Composite cross section for the MFZ in the Llanberris



Composite cross section for the MFZ in the Llanberris compartment with representative photos for different lode styles. Mapped quartz veins are extrapolated in between mining levels using drill hole information. Shale beds are coloured blue, Sandstone beds are coloured yellow and quartz veins are coloured red.

Figure 5-10 The Mako fault in the Llanberris compartment (LLB648SOD1 turnout)



Vein mineralisation

The vein mineral assemblage includes several generations of quartz with chlorite, sericite, albite and carbonate minerals. Arsenopyrite and pyrite are the dominant sulphide minerals with galena, sphalerite, chalcopyrite and pyrrhotite also commonly observed. The estimated percentage of sulphide minerals in the veins is 2%.

The sulphides occur in a spatial relationship with gold. Observation has shown that gold may occur within fractures within sulphide minerals or be deposited on the margins of sulphide grains, indicating that gold was deposited last (Figure 5-11 and Figure 5-13).

Figure 5-11 Gold particles in drill hole CBU526, MFZ, Britannia compartment



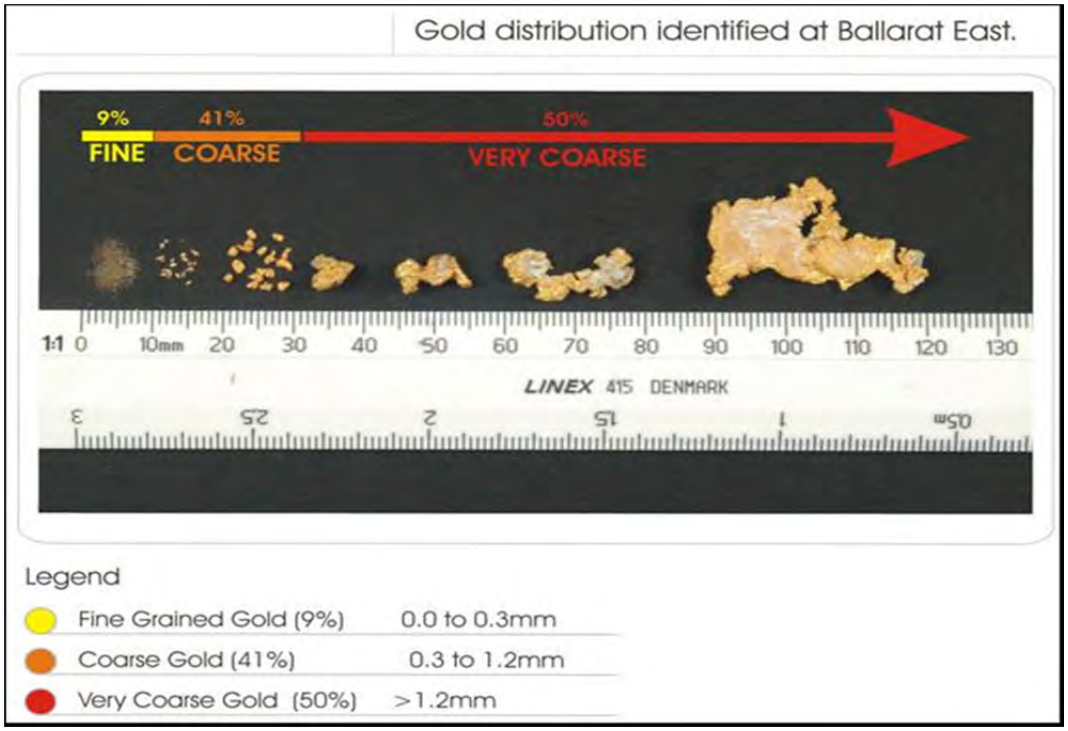
The host rocks show bleaching, carbonate aggregates, disseminated pyrite and arsenopyrite and pervasive sericitic alteration as a halo around quartz veining. There has not been a correlation established with the presence or grade of gold.

The alteration also includes the carbonate minerals calcite, dolomite, ankerite and paragonite. Sericite, kaolin, albite and chlorite are also observed within veins and mineralised zones.

Figure 5-12 Gold and sulphide mineralisation CBU523A, MFZ, Victoria compartment



Figure 5-13 Gold distribution as recovered from a metallurgical test sample



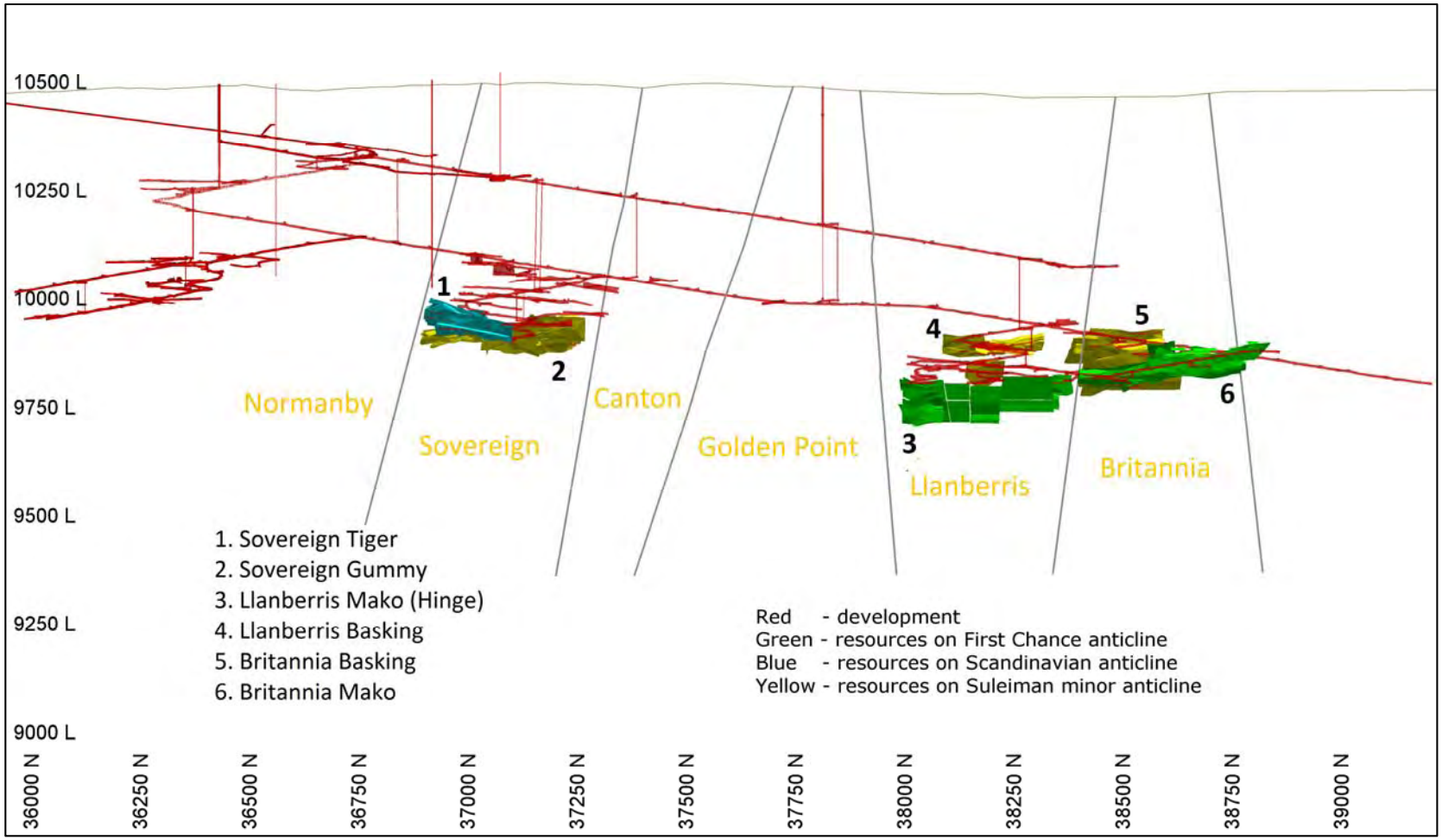
Gold has been observed as native particles which range in size from microns up to 30 mm in length (Figure 5-13).

The historical reports of occurrence of the gold (Baragwanath, 1923) and the observations made during the current mining operations have shown no change in the style and nature of gold occurrence throughout the Ballarat East goldfield.

5.3.4 Resource mineralisation

The mineralisation of each of the resources estimated in this report is described in the following section. The resource locations are shown in Figure 5-14.

Figure 5-14 Resource location, Ballarat East. Long section looking west



5.3.4.1 Sovereign Gummy

The Gummy Fault resource is a west-dipping fault zone located in the eastern limb of the Sulieman anticline in the Sovereign compartment (Figure 5-15 and **Error! Reference source not found.**). It consists of quartz veining associated with two prominent sets of west-dipping faults, east-dipping en-echelon vein arrays between the two sets of faults and stockwork zones at the upper and lower ends of the zone at the fold hinges where the west- dipping faults converge. Occasional mineralised veining has been observed in the footwall of the fault zone, but only minor veining has been intercepted above the in the hanging wall of the upper faults.

The fault zone commences as the faults begin to transgress bedding from the Sulieman syncline, through the steeply dipping sediments of the east limb and continues to the hinge of the anticline. The dip of the zone commences steeply and then flattens as it reaches the midpoint of the limb and then steepens again as the faults approach the hinge of the fold. The dip ranges between 45o and 70o degrees.

In the middle part of the fault zone, i.e. at the midpoint of the fold limb, the quartz veining diminishes zero. This is interpreted as due to the increase in separation between the hanging wall and fault wall sets and as faults die out along strike and up and down dip, possibly as an en-echelon pattern, reducing the deformation of the intervening rock (Figure 5-15).

Gold was seen throughout the zone and was observed more frequently on the hanging wall of the zone. High grade assays were also returned from east-dipping vein arrays.

The lower stockwork zone was often observed to consist of abundant quartz, which in many intersections returned very low grade assays with better grades concentrated on the margins.

The hanging wall or “bounding” fault was often defined, especially in the lower southern part of the zone, by a laminated quartz vein up to 15 cm wide.

In long section, the fault zone is appears horizontal but mineralisation in the northern half of the compartment plunges to the north while the southern half seems to have a least two distinct south plunging zones.

Figure 5-15 Plan view of the position of the Sovereign Sulieman mineralisation in purple relative to current development

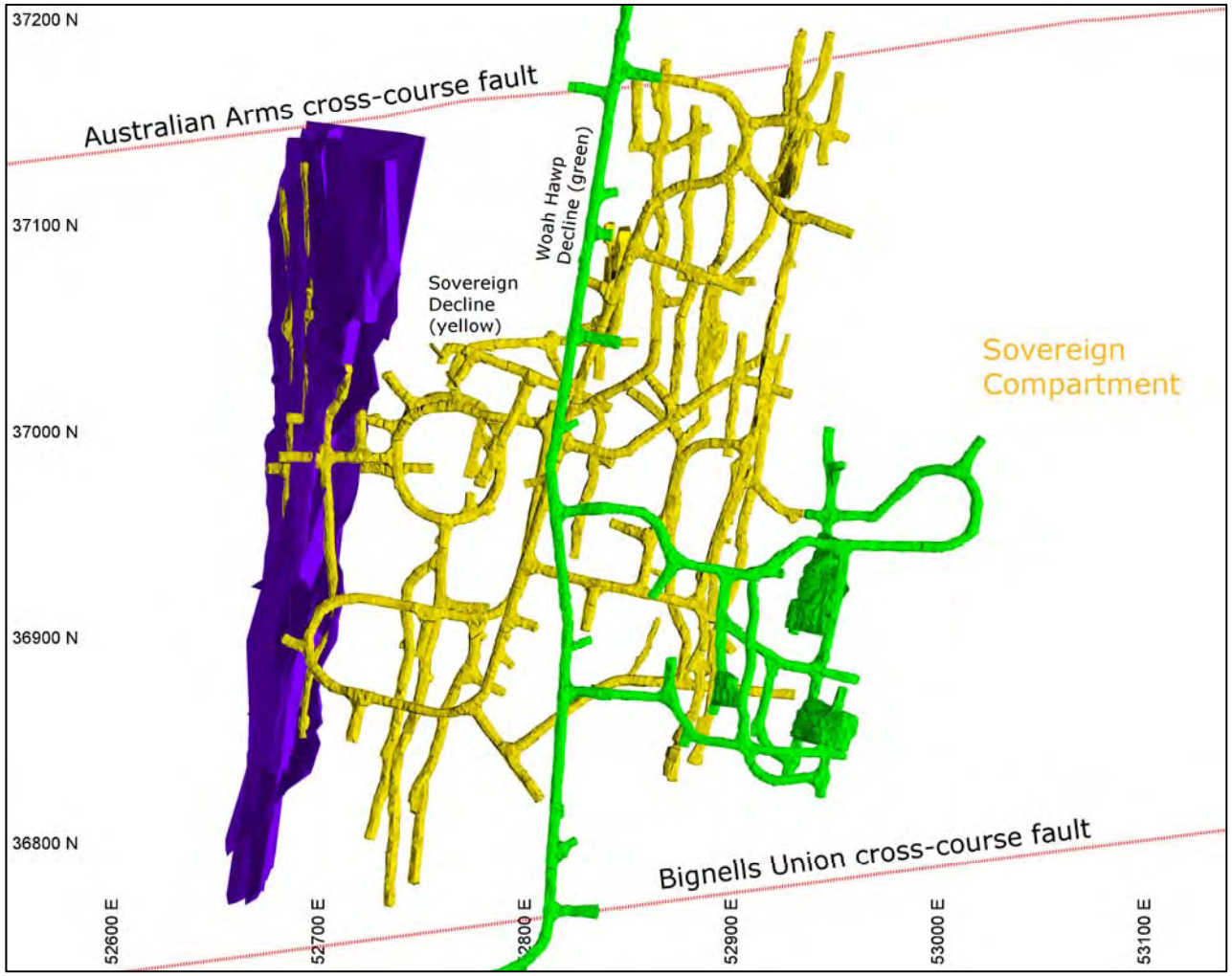
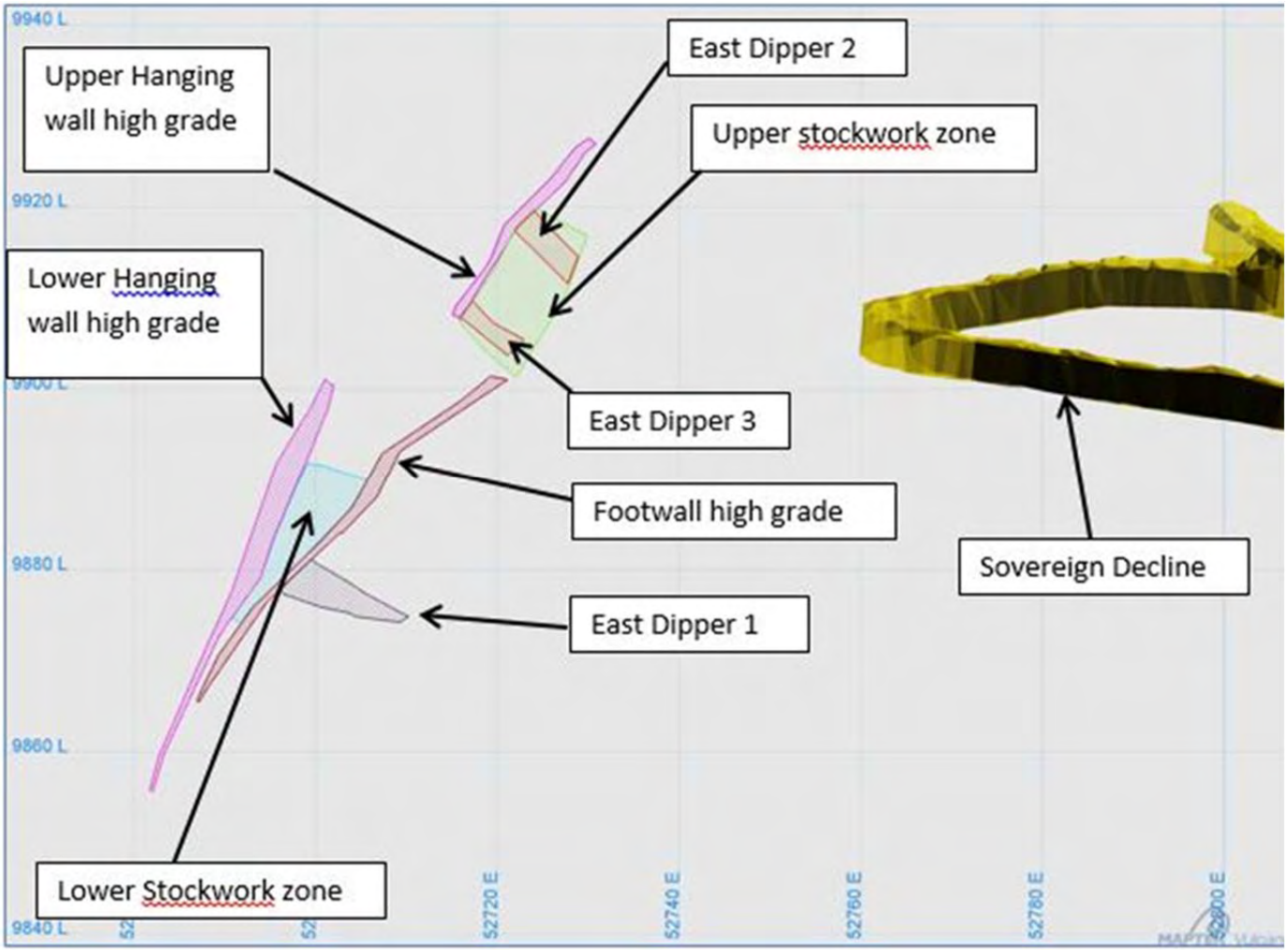


Figure 5-16 Cross section of Sovereign Sulieman resource looking north at 37,060 mN



5.3.4.2 Llanberris Basking

The Llanberris Basking resource is located on the eastern limb of the Sulieman minor anticline within the Llanberris compartment (Figure 5-17 and Figure 5-18). The host lithological sequence is thinly-bedded sandstone, shale and graded fine grained sand beds, typical of the turbiditic depositional environment throughout Central Victoria. An unmineralised mafic dyke (<1 m wide), has been intruded along bedding and fault planes on the limb of the fold.

The Basking fault is major steep west-dipping reverse fault with subsidiary parallel reverse faults, which has an estimated width of 3 m to 15 m. The displacement along each fault is difficult to determine, but the overall movement has been interpreted to be in the range of 10 m to 20 m.

The fault zone extends across the eastern limb of the fold, from the syncline in the east to the axis of the anticline; the fault extends beyond the fold axes but is not significantly mineralised in the west limb. It is 410 m in length,60 m high and up to 17 m wide.

The deformation within the fault zones has allowed the formation of quartz deposits in the form of massive quartz veins, which generally occur along the dominant faults and arrays of orientated quartz veins which occur in the hanging wall of faults and occasionally extend between adjacent faults. In addition the deformation has formed zones of veining, parallel to bedding, within shale beds which may extend up to 5 m in width.

The quartz which comprises the veins has a variety of textures including massive, stylolitic, brecciated and occasionally laminated. The quartz veining can be strongly fractured due to deformation after emplacement.

Mineralisation is hosted solely within the quartz veining associated with the faulting. Gold is present as discrete particles in quartz and within fractures of sulphide minerals, notably arsenopyrite and pyrite. The other sulphide minerals which occur in proximity to gold mineralisation are galena, sphalerite and chalcopyrite.

The Basking fault resource is interpreted to be associated with the intersection of the fault zone with a minor anticline-syncline pair on the eastern limb of the main Sulieman anticline with the majority of quartz veining located in the vicinity of the intersection and extending up dip of the fault zone.

Figure 5-17 Plan view of Llanberris Basking resource. Resource in red, mine development in blue

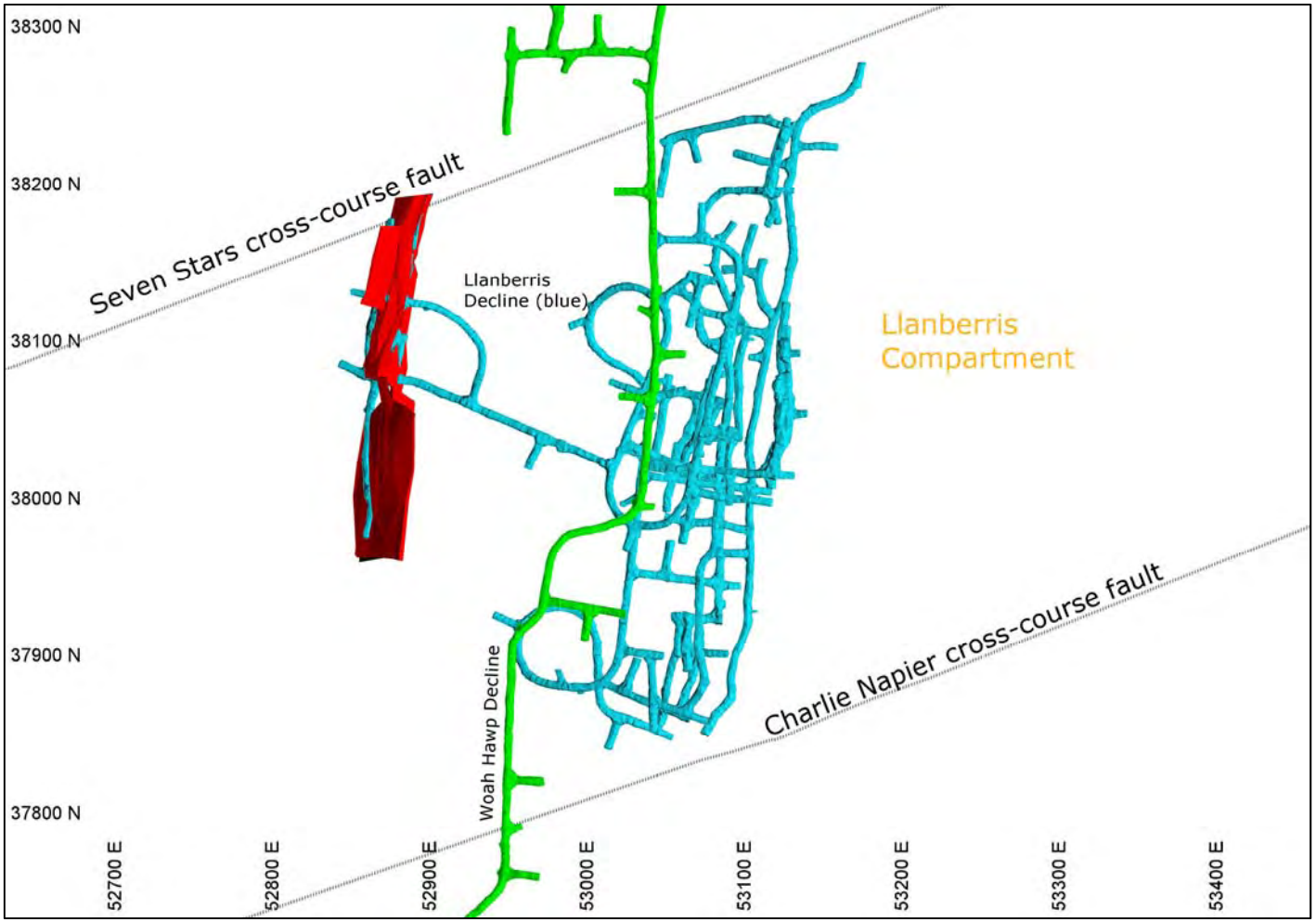
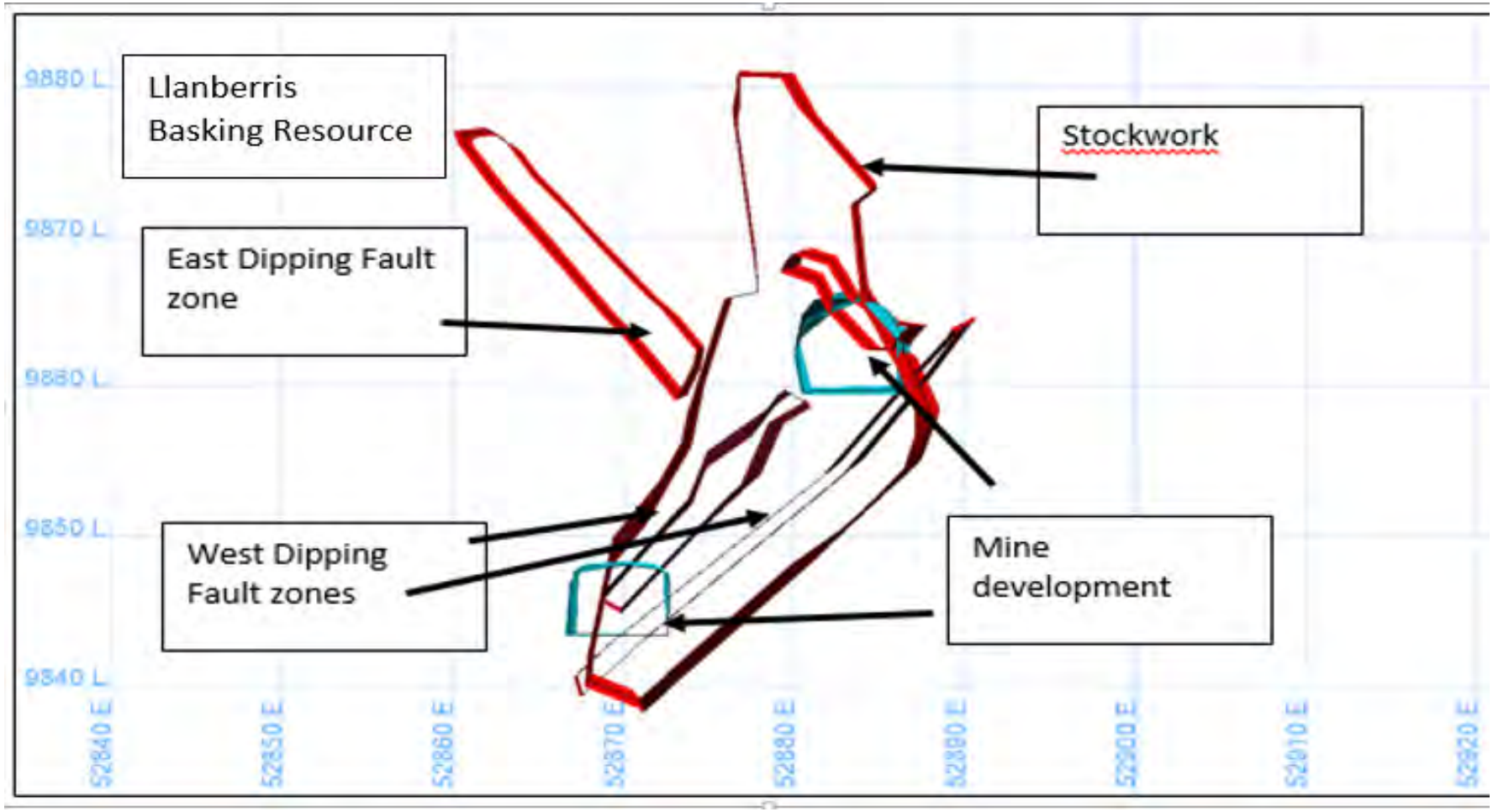


Figure 5-18 Cross section of Llanberris Basking Resource at 38,160 mN, Resource in Red, mine development in blue



5.3.4.3 Llanberris Mako Hinge

The Llanberris Mako resource is located on lower part of the east limb of the First Chance anticline, within the Llanberris compartment (Figure 5-19). The host lithological sequence is thinly-bedded sandstone, shale and graded fine grained sand beds, typical of the turbidite depositional environment throughout Central Victoria.

The Mako fault zone is a major steep west-dipping reverse fault with subsidiary shallower sub parallel reverse faults, which has an estimated width across the zone of 30 m to 40 m. The displacement along each fault is difficult to determine but the overall movement has been interpreted to be in the range of 10 to 20 m. The total length of the resource is 385 m, which extends up to100 m in height and across a combined width of 70 m.

The fault zone extends across the eastern limb of the fold, from the syncline in the east to the axis of the anticline. The extension of the faults beyond the axis has not been resolved due to limited exposure.

The deformation within the fault zones has allowed the formation of quartz deposits in the form of massive quartz veins, which generally occur along the dominant faults and arrays of orientated quartz veins which occur in the hanging wall of faults and occasionally extend between adjacent faults.

The quartz veins bear a variety of textures including massive, stylolitic, brecciated and occasionally, laminated. The quartz veining can be strongly fractured due to deformation after emplacement.

Mineralisation is hosted solely within the quartz veining. Gold is present as discrete particles in quartz and within fractures of sulphide minerals, notably arsenopyrite and pyrite. The other sulphide minerals, which occur in proximity to gold mineralisation are galena, sphalerite and chalcopyrite.

The Mako resource comprises several discrete zones of quartz veins associated with individual faults within the overall fault zone (Figure 5-20). They consist of massive quartz veins up to 3 m true width and tension vein arrays sub-parallel to the faults and also vein arrays parallel to the steeply-dipping cleavage and bedding which also have an overall east-dipping orientation. The percentage of quartz veining within each zone varies between 30% and 90%. The zones terminate due to a decrease in the amount of quartz which relates to a decrease in the activity of the faults along strike.

Figure 5-19 Plan view of Llanberris Mako resource. Resource in yellow, mine development in blue

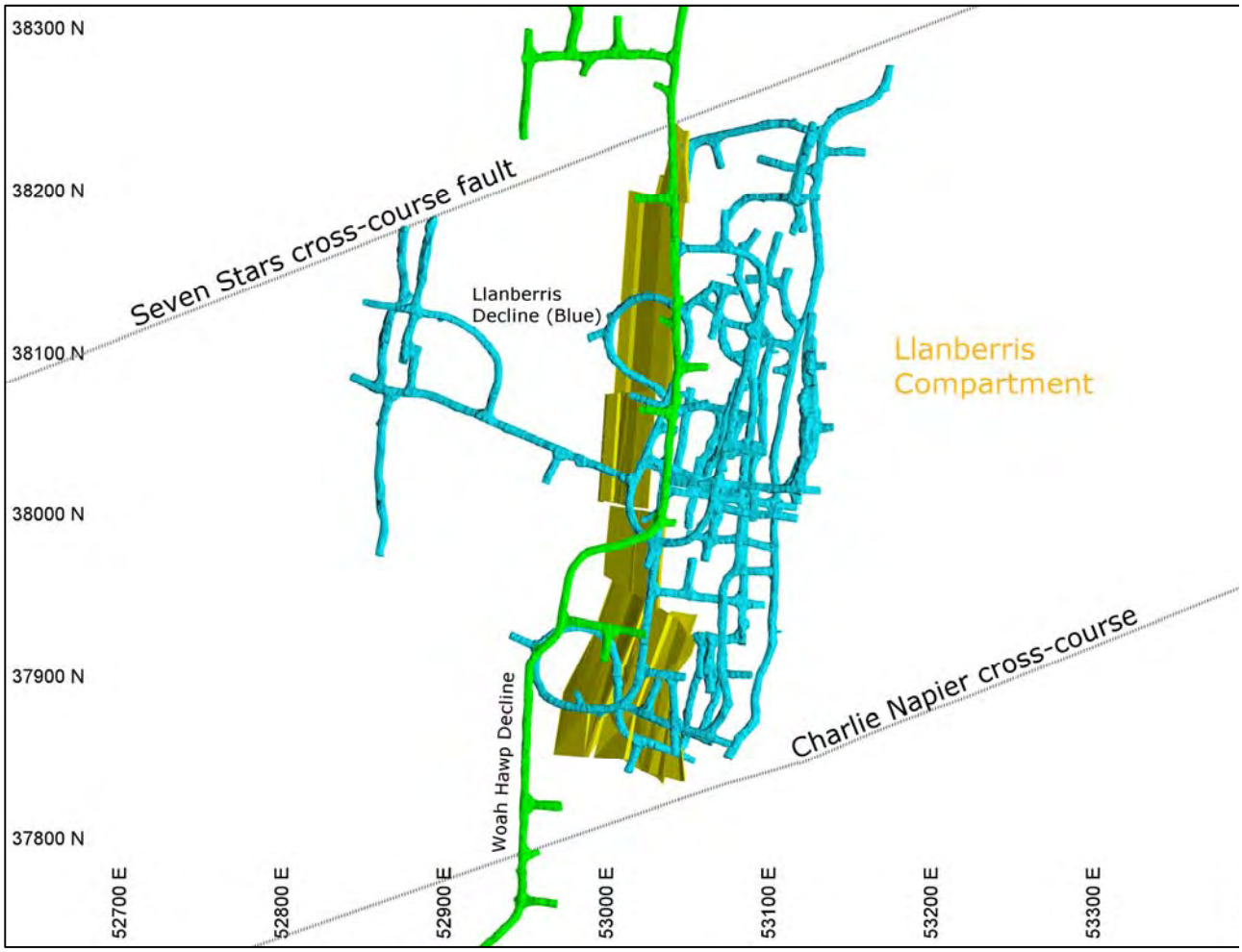
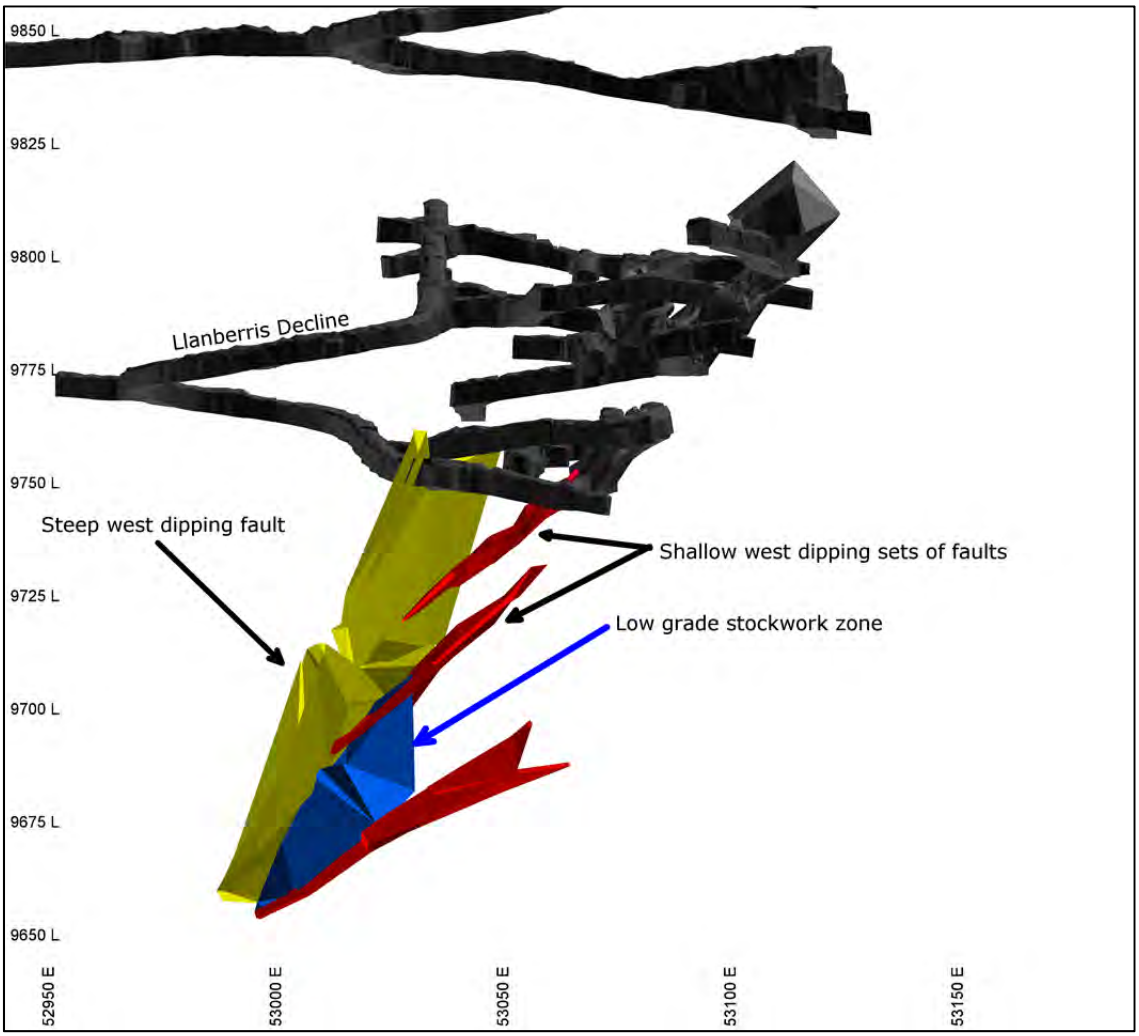


Figure 5-20 Llanberris Mako resource. Oblique view looking north, mine development in grey, resource in coloured shapes



5.3.4.4 Britannia Mako

The Britannia Mako resource is located on the east limb of the First Chance anticline, within the Britannia compartment (Figure 5-21). The host lithological sequence is thinly-bedded sandstone, shale and graded fine grained sand beds, typical of the turbidite depositional environment throughout Central Victoria.

The MFZ is a major west-dipping reverse fault with subsidiary parallel reverse faults, which has an estimated width of 30 m to 40 m. The displacement along each fault is difficult to determine but the overall movement has been interpreted to be in the range of 10 to 20 m. The total length of the resource is 400 m, which extends up to 100 m in combined height and across a maximum width of 100 m.

The fault zone extends across the eastern limb of the fold, from the syncline in the east to the axis of the anticline. The extension of the faults beyond the axis has not been resolved due to limited exposure. There are multiple faults recognised within the zone that occur in the hanging wall of the MFZ with varying amount of quartz veining and mineralisation associated with them.

The most prominent subsidiary fault is the Siberian which has been observed to extend upwards from the intersection of the MFZ and the anticlinal hinge, and transgress bedding at a steeper angle (Figure 5-22). It extends to 30 m above the Mako when it ceases to be mineralised. The intersection point with the anticline is the location of a zone of stockwork and east-dipping en-echelon vein arrays which extends up to 20 m in height and up to 15 m in width. The western boundary of this zone is the Siberian fault, occasionally

observed to have a laminated quartz vein 15 cm on the fault, no mineralisation has been observed in the hanging wall of this fault.

East-dipping vein arrays occur around the west-dipping faults in the hanging wall of the Mako often terminating up and down dip against parallel faults. They have been observed to extend up to 40 m along strike and up to 10 m in height.

The quartz shows a variety of textures including massive, stylolitic, brecciated and occasionally laminated. The quartz veining can be strongly fractured due to deformation after emplacement.

Mineralisation is hosted solely within the quartz veining associated with the faulting. Gold is present as discrete particles in quartz and within fractures of sulphide minerals, notably arsenopyrite and pyrite. The other sulphide minerals which occur in proximity to gold mineralisation are galena, sphalerite and chalcopyrite.

Figure 5-21 Britannia Mako resource. Resource in orange, mine development in purple and green Llanberris Mako resource

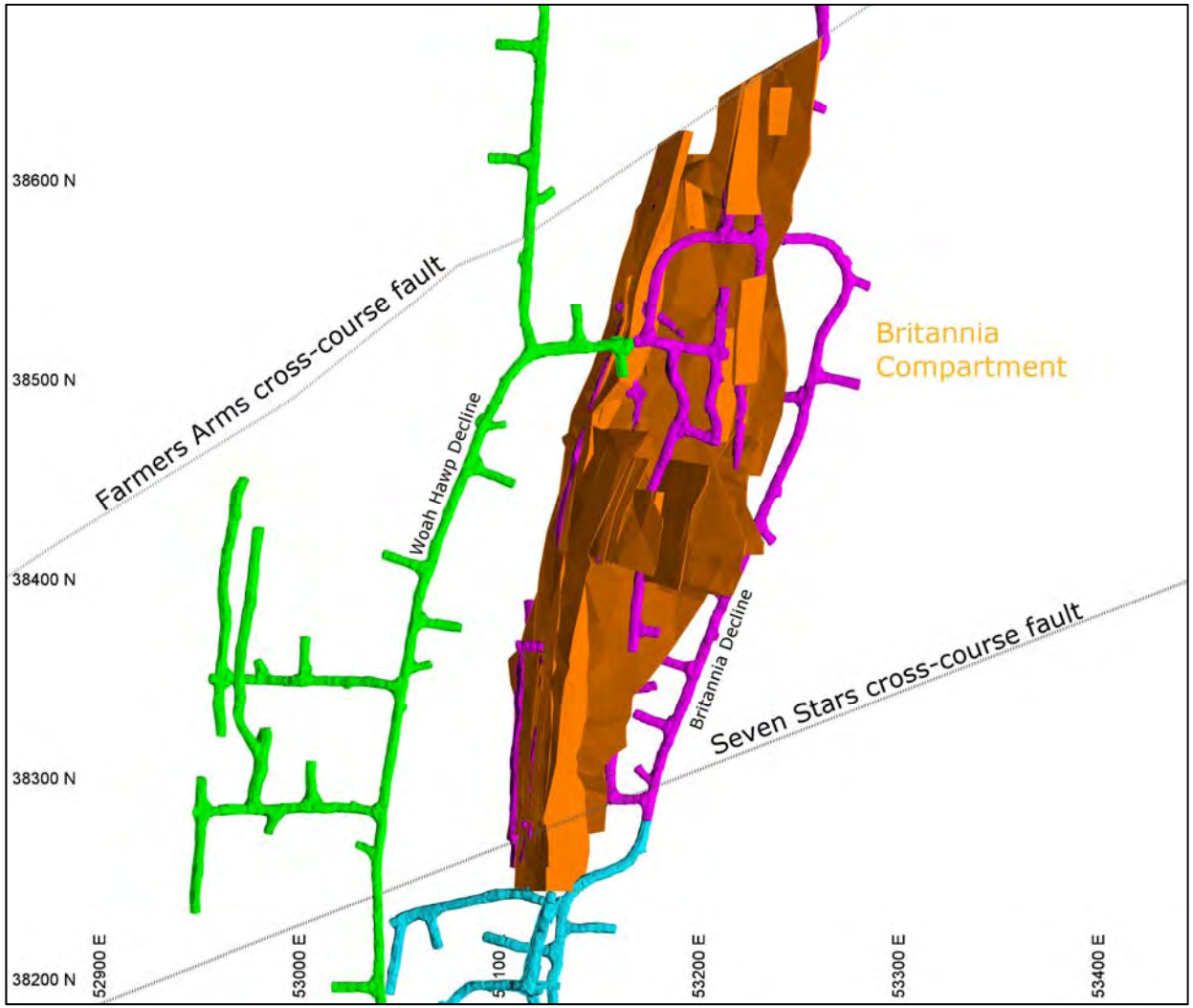
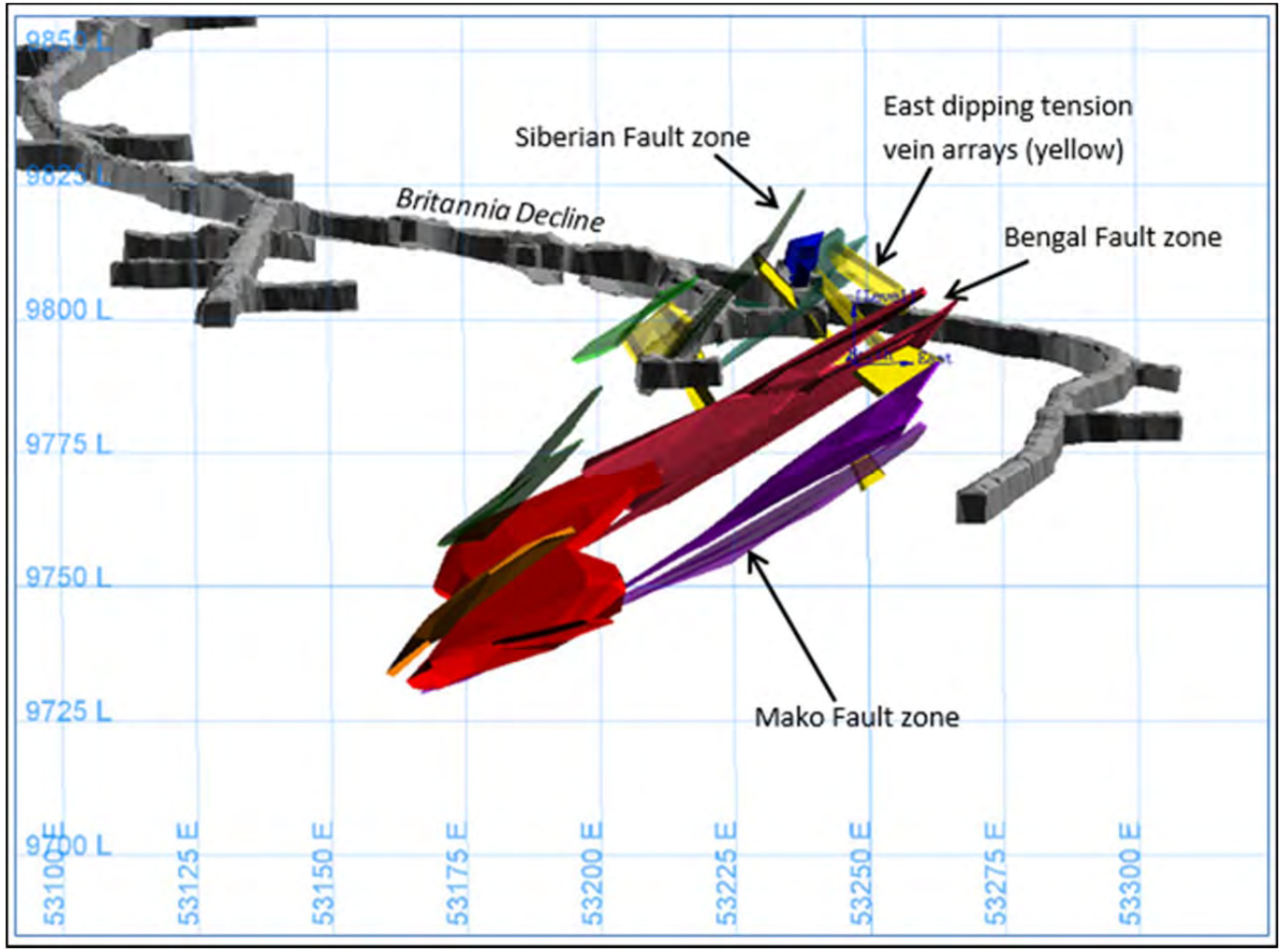


Figure 5-22 Llanberris Mako resource. Oblique view looking north, mine development in grey, resource in coloured shapes



5.3.4.5 Britannia Basking

The Britannia Basking resource is located on the east limb of the Suleiman minor anticline, within the Britannia compartment (Figure 5-23). The host lithological sequence is thinly-bedded sandstone, shale and graded fine grained sand beds, typical of the turbidite depositional environment throughout Central Victoria.

The mineralisation associated with the BFZ and HHFZ has a steeper west dipping orientation than seen in the First Chance line which in part seems to be related to the lack of overturning of the Suleiman minor anticline hinge. Localised changes in orientation and mineralisation extent are observed in the 30-50m adjacent to the major cross-course faults that define the compartment boundaries.

The major west-dipping reverse fault and associated subsidiary sub-parallel faults, have an estimated width across the zone of 35 m to 50 m. The displacement along each fault is difficult to determine but the overall movement has been interpreted to be in the range of 10 m to 20 m. The total length of the zone is 255 m, which extends up to 80 m in combined height.

The fault zone extends across the eastern limb of the fold, from the syncline in the east to the axis of the anticline. The major faults extend beyond the axis but do not have significant mineralisation on the west limb of the Suleiman minor anticline. There are multiple faults recognised within the zone that occur in the hanging wall of the BFZ with varying amount of quartz veining associated with them.

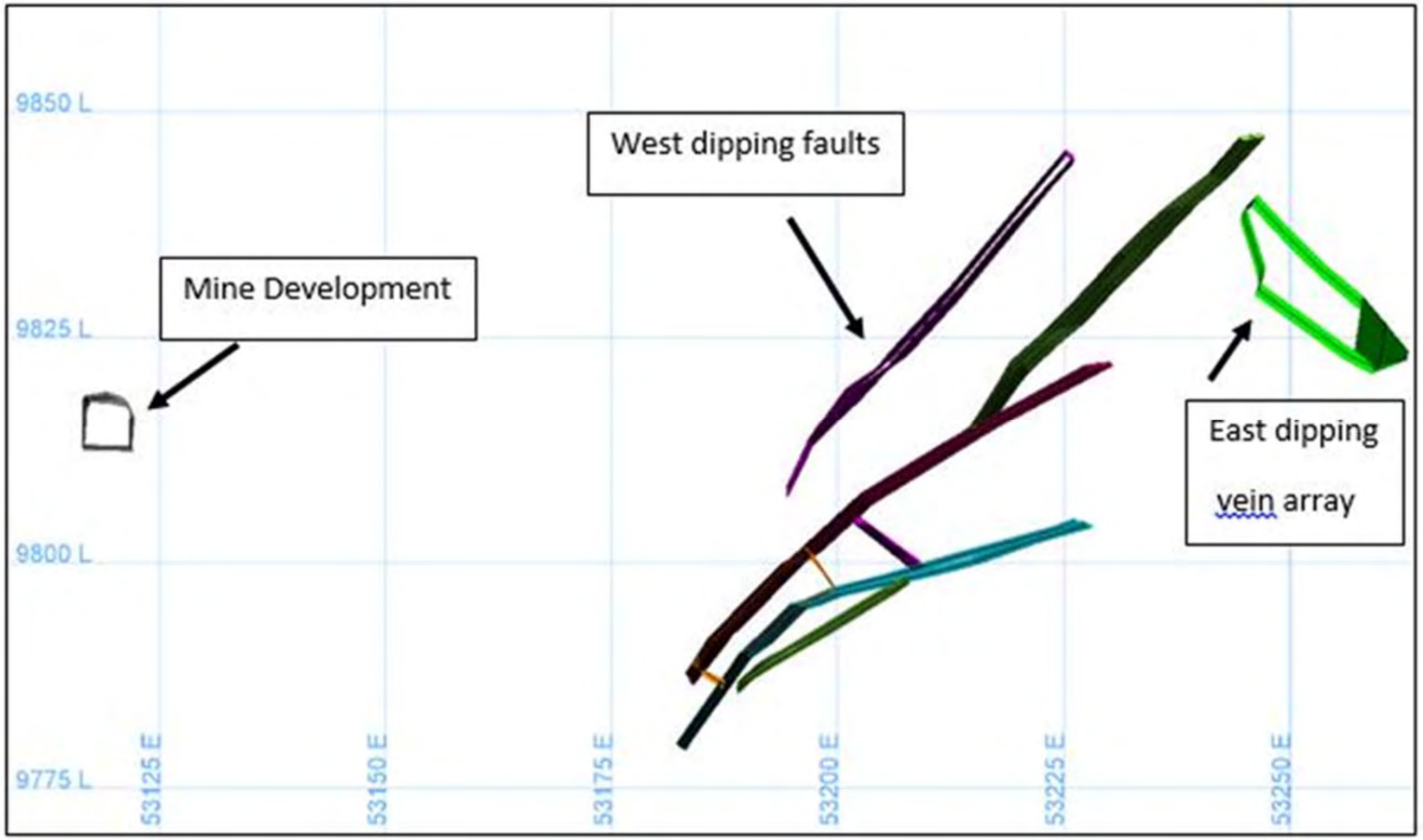
East-dipping vein arrays are interpreted to occur in the upper part of the zone between the BFZ hanging wall, BFZ footwall and the HHFZ hanging wall faults. In the southern most end of the compartment, rotation contemporaneous with mineralisation caused by cross-course drag has re-orientated the east dippers to a vertical aspect. The west dippers have also been rotated and are shallower than in the rest of the compartment. The quartz which comprises the veins has a variety of textures including massive, stylolitic, brecciated and occasionally laminated. The quartz veining can be strongly fractured due to deformation after emplacement.

Mineralisation is hosted solely within the quartz veining associated with the faulting. Gold is present as discrete particles in quartz and within fractures of sulphide minerals, notably arsenopyrite and pyrite. The other sulphide minerals which occur in proximity to gold mineralisation are galena, sphalerite and chalcopyrite.

Figure 5-23 Plan view of Britannia Basking resource. Mine development in green & purple, resource is yellow.



Figure 5-24 Section view of Britannia Basking resource. Mine development in grey, resource coloured



5.3.4.6 Sovereign Tiger

The Sovereign Tiger resource is located on the east limb of the Suleiman minor anticline, within the Sovereign compartment (Figure 5-23). The host lithological sequence is thinly-bedded sandstone, shale and graded fine grained sand beds, typical of the turbidite depositional environment throughout Central Victoria.

The Tiger lode in the Sovereign compartment plunges north at -23 degrees with deformation, quartz veining and mineralisation declining the north as the fault arrays cross the hinge axis into the west limb of the anticline. The major steeper west-dipping reverse fault with subsidiary shallower sub parallel west dipping faults, which have an estimated width across the zone of 15 m to 20 m. The displacement along each fault is difficult to determine but the overall movement has been interpreted to be in the range of 5 m to 15 m. The total length of the zone is 200 m, which extends up to 70 m in combined height and across a combined width of45 m.

The fault zone extends across the eastern limb of the fold, from the syncline in the east to the axis of the anticline. The faults extend beyond the axis with the mineralisation diminishing in the west limb of the anticline. The quartz which comprises the veins has a variety of textures including massive, stylolitic, brecciated and occasionally laminated. The quartz veining can be strongly fractured due to deformation after emplacement.

Mineralisation is hosted solely within the quartz veining associated with the faulting. Gold is present as discrete particles in quartz and within fractures of sulphide minerals, notably arsenopyrite and pyrite. The other sulphide minerals which occur in proximity to gold mineralisation are galena, sphalerite and chalcopyrite.

Figure 5-25 Plan view of Sovereign Tiger resource. Mine development in yellow, resource is red.

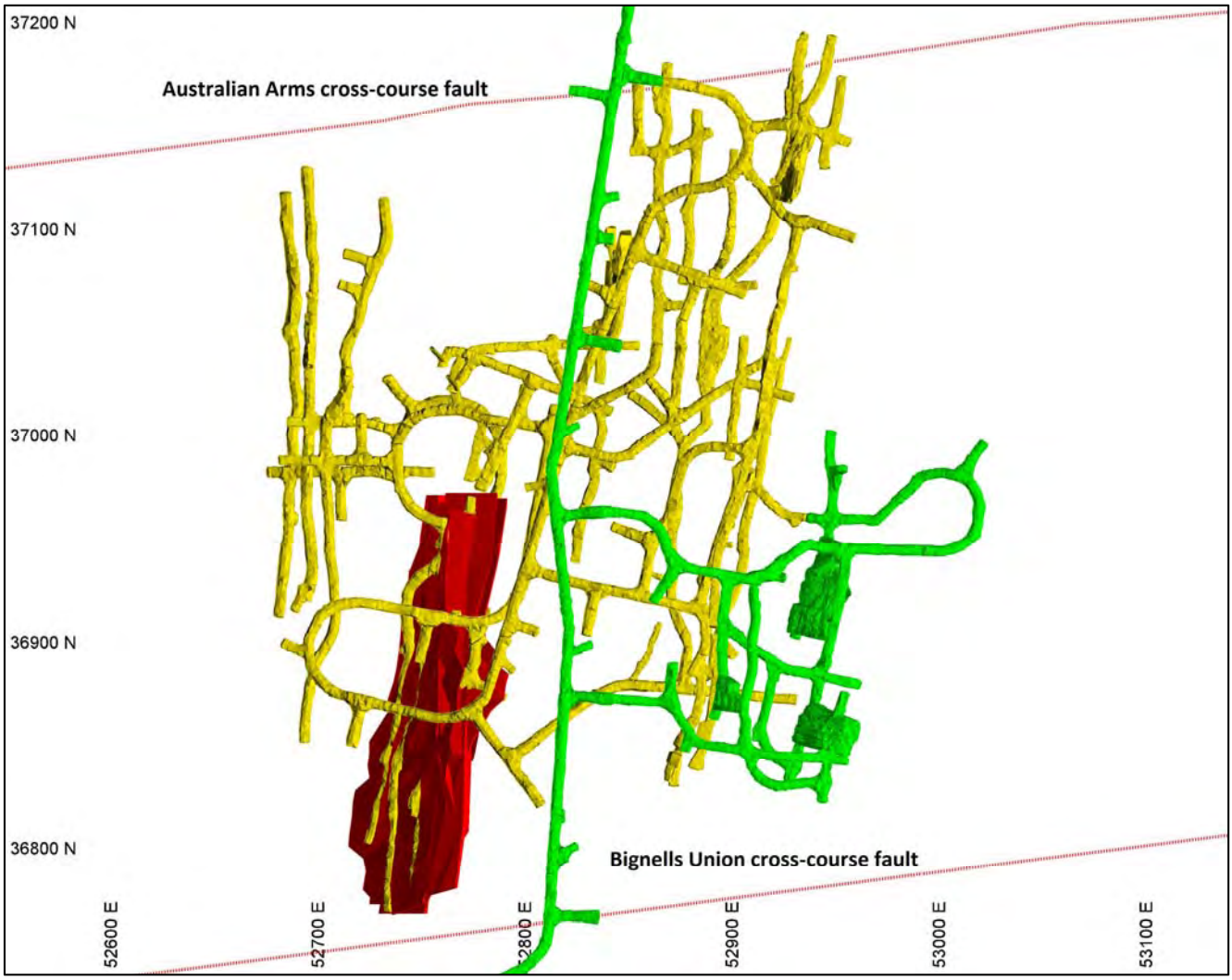
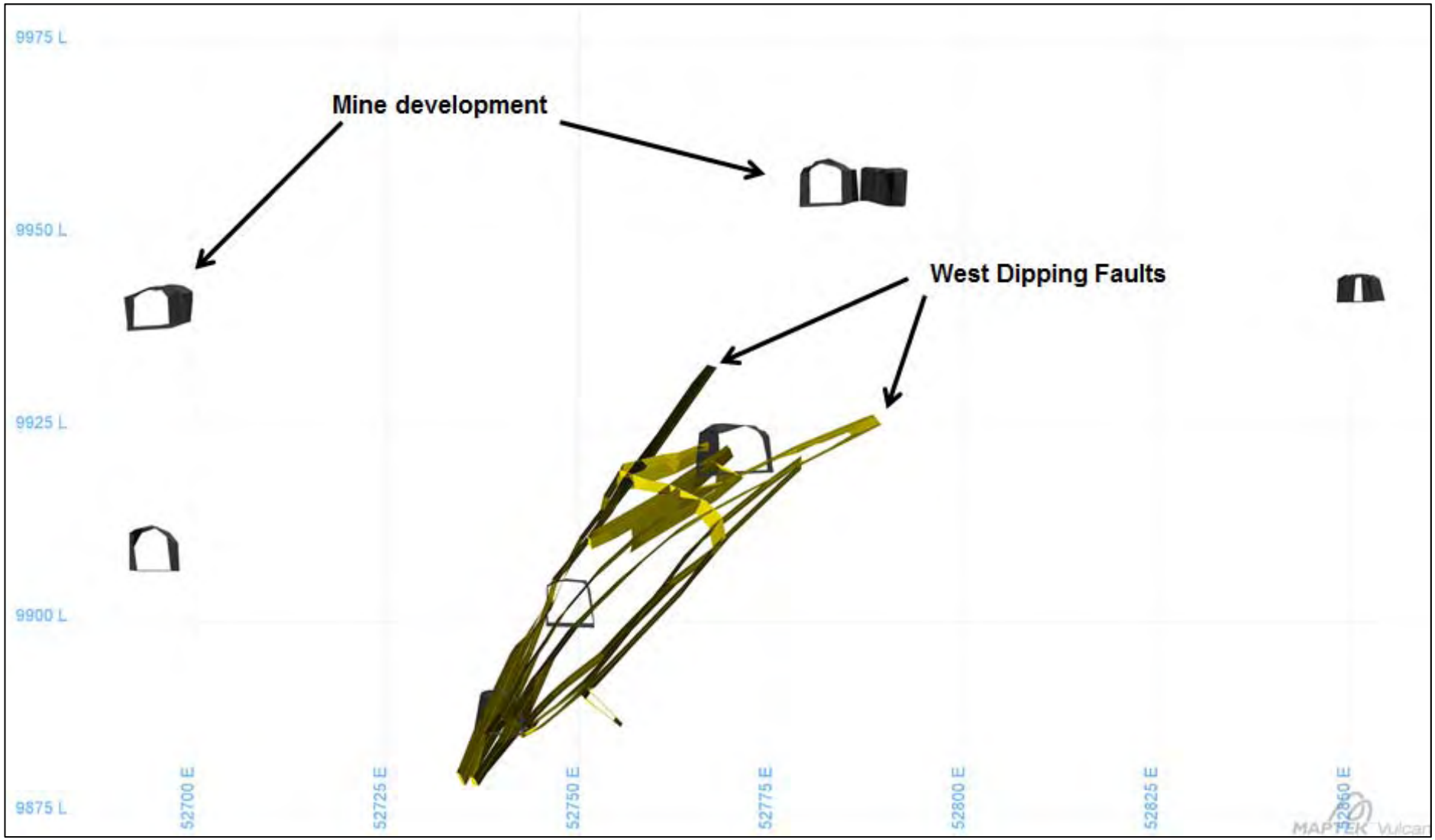


Figure 5-26 Section view of the Sovereign Tiger Resource. Mine development in grey, resource is coloured



6 EXPLORATION ACTIVITIES

6.1 Exploration Overview

Previous exploration activities are summarised in Section 4.2.

6.2 Exploration Methods

Both historically and currently, exploration is dominated by diamond core drilling and development.

6.2.1 Geology

Ballarat regional and local geology is presented in Sections 5.1 and 5.2.

6.2.2 Geophysics and Remote Sensing

No geophysical exploration has been undertaken at Ballarat.

6.2.3 Geochemistry

No geochemical exploration has been undertaken at Ballarat.

6.2.4 Drilling

Grid coordinate system

All references are in mine grid. The local grid utilised at the Ballarat East site is a plane co-ordinate system given the name “mine grid”. All surface survey data is stored using this grid with the vertical control being Australian Height Datum 1971 (AHD). All underground survey data is stored using mine grid with vertical control being Australian Height Datum 1971 (AHD) plus 10,000 m. The relationship between the national grid systems that have been used and the mine grid since it was established are shown below (Table 6.1 and Table 6.2).

The mine grid was established prior to CGT taking ownership of the Ballarat mine in 2010. The mine grid was established early in the mine’s life to suit the mining software used at the time. The vertical control (AHD plus 10,000 m) was implemented to prevent the occurrence of negative numbers.

The declination of the Ballarat area to magnetic north is shown in Figure 6-1.

Relationship between mine grid and Map Grid of Australia (MGA94)

Scale	1.000310271
Rotation	00deg 00min 00sec
Shift North	5800177.789
Shift East	700120.707

Table 6.1 Relationship between mine grid and Map Grid of Australia (MGA94)

	Mine Grid			MGA94		
Point #	Easting	Northing	AHD Elevation	Easting	Northing	AHD Elevation
BGF003	52401.537	35638.516	452.951	752522.244	5835816.305	452.951
BGF004	52150.073	35776.976	435.426	752270.702	5835954.808	435.426

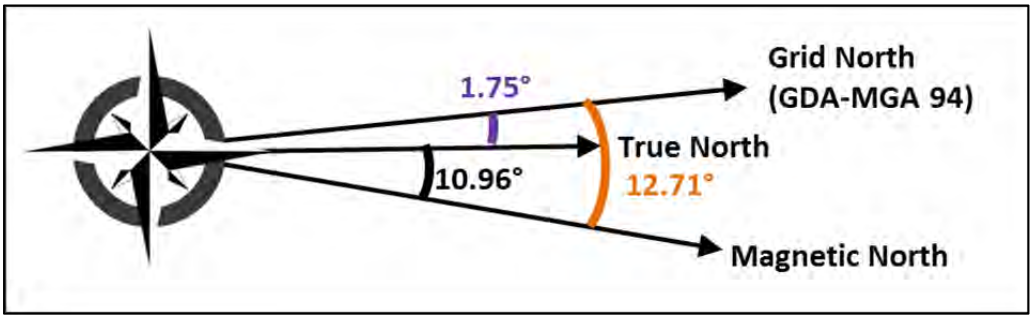
Relationship between mine grid and Australian Map Grid (AMG66)

Scale	1.000
Rotation	00deg 00min 00sec
Shift North	5800000.000
Shift East	700000.000

Table 6.2 Relationship between mine grid and Australian Map Grid (AMG66)

	Mine Grid			AMG66		
Point #	Easting	Northing	AHD Elevation	Easting	Northing	AHD Elevation
BGF003	52401.537	35638.516	452.951	752401.537	5835638.516	452.951
BGF004	52150.073	35776.976	435.426	752150.073	5835776.976	435.426

Figure 6-1 Relationship between mine grid north, true north and magnetic north



Drilling

All drilling data utilised in the resource estimate was collected from diamond drill core recovered from underground in the period 2003 to the present day. The core sizes drilled have been NQ (45 mm), NQ2 (47.6 mm), BQ (35 mm) and LTK 60 (44.1 mm).

In the period 2003-2009 drilling was carried out by Boart Longyear Pty Ltd, BGF using company owned drill rigs and company staff, and also rigs provided by Deepcore Drilling Pty Ltd (Deepcore).

Since 2010, when CGT purchased the tenements, all drilling has been carried out from underground by Deepcore. Deepcore has used Boart Longyear LM75 and LM90 drill rigs. The core sizes drilled during this time have been HQ, NQ2 and LTK60. All drilling completed since January 2014 has been NQ2 size apart from CBU1076A which was drilled in HQ. The average support of HQ core is 8.55 kg/m, NQ2 core is 5.43 kg/m and of LTK60 core is 4.09 kg/m using the expected non-ore SG of 2.7 g/cm^3.

Drillhole collar locations

Drillhole collars have been surveyed by CGT surveyors, using a one man total station and downloaded electronically. Seven holes informing this resource estimate were not collar surveyed, all these holes were within fans of drilling and had their positions estimated based on the position of adjacent surveyed collars. These holes are considered acceptable for inclusion in the estimate as they are deemed accurate to within 300 mm. Further information regarding collar survey validation is outlined in the downhole survey data validation section.

Drillhole downhole surveys

Downhole surveys were carried out using Globaltech Pathfinder® downhole multi-shot cameras up to January 2015 when they were replaced with Reflex EZ-Trac 6393 cameras. These cameras are used to carry out single-shot surveys every 30 m during drilling and multi-shot surveys every three metres upon completion of the hole. A total of 15 holes out of the 513 holes informing the estimates did not have a final multi-shot survey carried out. Whilst these holes have a lower level of confidence in the downhole surveys, they are considered adequate for inclusion in the estimate. Further information regarding downhole survey validation is outlined in Section 8.3.1.

Deepcore Drilling leases the downhole cameras and has a six month replacement schedule in place, whereby cameras are returned to the point of hire and replaced with calibrated cameras. In addition, from September 2013 CGT has conducted monthly checks of the calibration of Deepcore's cameras, using a calibration cradle installed on site.

6.2.5 Sampling

Primary samples

Sampling during the LGL period was nominally at 1 m intervals for both half core and full core. Exploration level drilling was half core sampled while production level was full core sampled. CGT drilling between 2011 and July 2014 was whole core sampled with the maximum sample length of 0.4m based on a 2 kg sample weight limit to avoid splitting or sub-sampling the primary sample at the laboratory. From August 2014 sampling length was extended to a nominal 0.7m with a minimum of 0.30m when required. This is based on the maximum mass that can efficiently pulverised by commercial labs, which is 3.5 kg, the sample is then rotary split into a 2 to 2.3 kg sub-sample and the remaining material is bagged as reject.

Both CGT and LGL used the logged geology to define expected ore zone, sampling being extended at least 1 m into waste zones to avoid missing any contact mineralisation. The LGL and CGT sampling procedures are attached in Appendix B of the Ballarat March 2015 JORC Report.

Field duplicates

The LGL process for field duplicates varied between exploration and production level diamond drill holes. For exploration level holes (target spacing of 100 m by 20 m) only half core was sampled unless visible gold was present. In that situation the other half of the core was also sent for assaying as a duplicate. For production level holes (target spacing of 35 m by 10 m to 20 m by 5 m), whole core sampling was undertaken with the larger sample routinely being split using a Boyd crusher. The reject split was also assayed if visible gold was identified at the logging stage and entered into the system as a field duplicates.

CGT undertook a limited half core campaign that included the following holes, CBP0013, CBP0015, CBP0019, CBP0022, CBP0023, CBP0024, CBP0025, CBP0045, CBP0046, CBP0047, CBP0048, CBP0049, CBP0050, CBP0051, CBP0052, CBP0053, CBP230 and CBP231. After this campaign whole core sampling was adopted as the standard procedure for all drilling at the Ballarat mine.

Laboratory preparation

The primary Laboratories used during between 2007 and 2014 are listed in

Table 6.3.

Table 6.3 Primary assaying laboratories

Period	Laboratory	Location
September 2007 to April 2008	Amdel	Adelaide and Kalgoorlie
April 2008 to August 2009	Ballarat Goldfields (BGF) owned by LGL	On site at Ballarat mine
June 2011 to March 2015	Gekko Systems Laboratory	On site at Ballarat mine

An on-site (BGF) laboratory was commissioned by LGL in March 2008 which then replaced Amdel Kalgoorlie for the processing of all geological samples. The BGF laboratory provided the following services:

- Sample preparation (crushing and pulverisation).
- LeachWELL bottle rolling and gold analysis by AAS.
- Metallurgical test work.

The BGF laboratory commissioned fire assaying on site in late 2009 for quality control checks on leach residues.

Upon CGT's purchase of the mine the BGF laboratory was sold to Gekko Systems Pty Ltd (Gekko) who continue to operate with the mine as a client.

Table 6.4 and Table 6.5 summarises the laboratory processes for the laboratories

Table 6.4 Summary of laboratory processes, September 2007 to March 2014 at Amdel Laboratory

	AMDEL (exploration)	AMDEL (production)
Location	Kalgoorlie/Adelaide	Kalgoorlie/Adelaide
Sample Type	Half-Core	Full-Core
Nominal Sample Length (m)	1	1
Nominal weight (kg)	2.5	5
Drying temperature		
Drying time (hr)	Crushed in Ballarat to 5-10mm before being bagged and sent to Amdel	Crushed in Ballarat to 5-10mm before being bagged and sent to Amdel
Crushing		
crush size (mm)		
Boyd Crusher Splitting/sub-sample	No	Split at Amdel
Target size	NA	Not defined
Sample to reject ratio <1.5kg sample	NA	Not defined
sample to reject ratio for 1.5 to 6kg sample	NA	Not defined
sample to reject ratio >6kg sample	NA	Not defined
Pulveriser	LM5	LM5
Target grind	grind passes 75µm	grind passes 75µm
Method	LeachWELL	LeachWELL
maximum weight	2000 g	2000 g
Leach solution	2000 g pre mixed aqueous solution of Sodium Cyanide, later changed to two LeachWELL tablets.	2000 g pre mixed aqueous solution of Sodium Cyanide, later changed to two LeachWELL tablets.
Roll time	24hr	24hr
Au Measurement Method	AAS	AAS
Filter and press bottle roll residue and FA50	1 in 20	1 in 20
Turn around	4 weeks	4 weeks

Table 6.5 Summary of laboratory processes, September 2007 to March 2015 at the BGF and Gekko laboratories

	BGF (exploration)	BGF (production)	Gekko
Location	on site (Ballarat)	on site (Ballarat)	on site (Ballarat)
Sample Type	Half-Core	Full-Core	Full Core
Nominal Sample Length (m)	1	1	variable
Nominal weight (kg)	2.5	5	variable
Drying temperature	80-100	80-100	110
Drying time (hr)	6-12	6-12	24
Crushing	Jaw Crusher	Jaw Crusher	Jaw Crusher
crush size (mm)	5 to 10	5 to 10	5 to 10
Boyd Crusher Splitting/sub-sample	No	Yes	No
Target size	NA	95% passing 3mm	NA
Sample to reject ratio <1.5kg sample	NA	no splitting	NA
sample to reject ratio for 1.5 to 6kg sample	NA	50:50	NA
sample to reject ratio >6kg sample	NA	60:40	NA
Pulveriser	LM5	LM5	LM5
Target grind	95% passing 75µm	95% passing 75µm	95% passing 75µm
sub sample using rotary splitter	No	No	Yes
Method	LeachWELL	LeachWELL	LeachWELL
maximum weight	2000 g (sub-sample)	2000 g (sub-sample)	2000 g
Leach solution	2 LeachWELL tablets per bottle	2 LeachWELL tablets per bottle	2 LeachWELL tablets per bottle
Roll time	24hr	24hr	24hr
Au Measurement Method	AAS	AAS	AAS
Filter and press bottle roll residue and FA50	1 in 20	1 in 20	1-June 2010 to 18-July 2013 All primary samples returning results greater than 5 g/t Au, 19-July-2013 to 31-March-2014 Limited to selected zones.
Turn around	2-6 days	2-6 days	3-10 days

6.2.6 Analysis

Drill hole sample assay

Drill hole sample assays were analysed by one of three methods. Some 10,814 samples were analysed using the Leachwell 2000 bottle-roll technique on full core. During re-commencement of the Ballarat mine in 2010 on-site facilities for Leachwell 2000 analysis were not available. As a result of this the first 62 diamond drill holes carried out by CGT in 2010 were analysed using either a 50 g fire assay or 1 kg screen fire assay. Some late 2010 CGT holes had small sections of full core sampling where visible gold was recorded and the rest as half core, other holes had full core sampling with small sections of half core each half assayed using a different method, these were either fire assay and/or leachwell and/or screen fire assaying. Some 249 samples informing this estimate were analysed using half core Leachwell assaying, another 128 were half core fire assaying. Table 6.6 details the use of these two methods on samples informing the estimate.

Table 6.6 Analysis methods used on Ballarat drill holes

Lode	Analysis type	No. Collars	No. Samples
Britannia_First_Chance_Mako	Full Core LeachWell2000	103	2,344
	Half Core LeachWell2000	7	82
	Half Core Fire Assay	9	64
Britannia_First_Chance_Mako Total		118	2,490
Britannia_Suleiman_Basking	Full Core LeachWell2000	88	1,541
	Half Core LeachWell2000	9	63
Britannia_Suleiman_Basking Total		96	1,604
Llanberris_Mako_Hinge	Full Core LeachWell2000	47	1,068
Llanberris_Mako_Hinge Total		47	1,068
Llanberris_Suleiman_Basking	Full Core LeachWell2000	68	1,154
	Half Core LeachWell2000	8	104
	Half Core Fire Assay	5	64
Llanberris_Suleiman_Basking Total		79	1,322
Sovereign_Suleiman_Tiger	Full Core LeachWell2000	56	1,483
Sovereign_Suleiman_Tiger Total		56	1,483
Sovereign_Suleiman_Gummy	Full Core LeachWell2000	91	3,224
Sovereign_Suleiman_Gummy Total		91	3,224
Grand Total		**485**	**11,191**

Note: The sum of the collars listed in the table above does not match the grand total; this is because some holes have two or three methods of assaying, several drill holes also intersect two lodes.

Apparent relative density

Apparent relative density was carried out by BGF in December 2007. Density was determined by the water immersion technique with a total of 134 drill core samples tested. Details of the method used to determine density values can be found in the procedure in Appendix G of the Ballarat March 2014 JORC Report.

A summary of the densities determined is given in Table 6.7. Block model densities have been assigned based on this data. Ore domains are assigned a density of 2.65 g/cm^3 whilst the surrounding blocks are assigned a density of 2.72 g/cm^3, the 2015 results indicate a slightly higher density (2.73 g/cm^3) for the non-quartz component, and further data will be collected to confirm this. Full details of the bulk density data informing this resource can be found in Appendix H of the Ballarat March 2014 JORC Report.

Two of the BGF samples have been omitted from the data-set used to determine bulk density as they returned relatively low densities. Annotation associated with these samples suggests they were samples of vuggy quartz. The proportion of vuggy quartz within the resource is not well understood, to ensure these samples do not bias the determined bulk density they have been omitted. Between February 2014 and December 2014 CGT submitted bulk density samples for analysis by Gekko using the water immersion

method. From January 2015 all relative density testing has been completed by CGT in house using the water immersion method.

Table 6.7 Apparent relative densities attributed to the Ballarat resource

Source	Lithology	No. samples	Average bulk density (t/m^3)
BGF	Quartz	40	2.65
BGF	Shale	43	2.75
BGF	Sandstone	49	2.71
BGF Sample Total		132	
GEKKO 2014	Quartz	73	2.64
GEKKO 2014	Shale	13	2.73
GEKKO 2014	Sandstone	17	2.71
Gekko Sample Total		103	
CGT 2015	Quartz	37	2.65
CGT 2015	Shale	3	2.73
CGT 2015	Sandstone	1	2.71
CGT Sample Total		41	
Combined	Quartz	150	2.65
Combined	Shale	59	2.76
Combined	Sandstone	49	2.71
Combined Sample Total		276	

6.2.7 Quality Assurance and Quality Control

Assay certificates

The period between January 2009 and February 2015, covers the drill holes included in the resource. Two drill holes with the BEU prefix processed after December 2008 by the on-site LGL owned lab and no certificates were issued for these.

Table 6.8 Summary of drill hole with assays for which no certificates were issued

Drill hole with assays for which no certificates were issued (LGL on-site lab)						
BEP1271	BEP1680	BEP1768B	BEP1811	BEU225A	BEU226	BEU252
BEU257A	BEU259A	BEU262	BEU263	BEU265	BEU269	

The majority of CB prefixed holes were processed by Gekko although a limited number (CBU001, CBU002, CBU005, CBU006, CBU010, CBU011, CBU012, CBU016 and CBU019) were processed by Genalysis, Adelaide, of these only CBU011 and CBU012 are used in this resource.

A review of thirteen Genalysis certificates including 5 associated with CBU011 and CBU012 (38 in total) found all certificates matched the results received.

On-going checks of Gekko certificates as they are issued, the review rate is 43% of all sample batches used for this report (209 out of 483) with only three certificates re-issued for minor errors prior to samples being loaded.

Drillhole QA/QC

Deepcore has a procedure outlining the correct core presentation and handling of drill core and core trays. This requires that core is laid out in a neat organised fashion. Where core is broken to fit in the tray, breaks are marked as “driller’s breaks”. Core is re-arranged to ensure it is as representative as possible of how it entered the core tube.

Core is prepared by field assistants with core rotated to ensure cleavage is oriented consistently. Metre intervals marked up are marked up based on core block downhole measurements. Run lengths and core loss interval lengths are checked against the measurements recorded on driller’s core blocks.

CGT has procedures in place outlining core preparation, cutting and sampling procedures. This procedure involves cross-checking of sample mark up and collection by the logging geologist and the sampler.

6.2.8 Sample Security

Core trays are brought directly from the underground drill sites to the site core shed, located within 500 metres of the mine portal and within the fenced perimeter of the mine site, which is not accessible to the general public. After core logging and sampling the prepared samples are packed into pods and delivered to the assay laboratory located 50 metres from the core shed and within the mine site compound. Access to mine site is restricted to employees and authorised visitors.

Results are sent electronically to the geology staff and validated through the acQuire software program. Only files validated for format and QAQC are loaded into the database.

The SQL database is backed up twice a day to disc and daily to tape. The tapes are stored in three separate secure sites, two of which are offsite. As of May 2014 all changes in the acQuire software program are recorded with the user login details and the date and time. The program includes the ability to reverse any changes made. Access to the site IT network is by a unique user login and password and is controlled by the site IT manager. To avoid unauthorised use of another person’s login all computers on site automatically lock down after a pre-set period of inactivity.

6.3 Exploration Results

Exploration results are used to support the Mineral Resource estimate. Further details are provided in Section 8.

6.4 QA/QC Results

6.4.1 Blanks

LGL and CGT both undertook QA/QC programmes using standards and blanks.

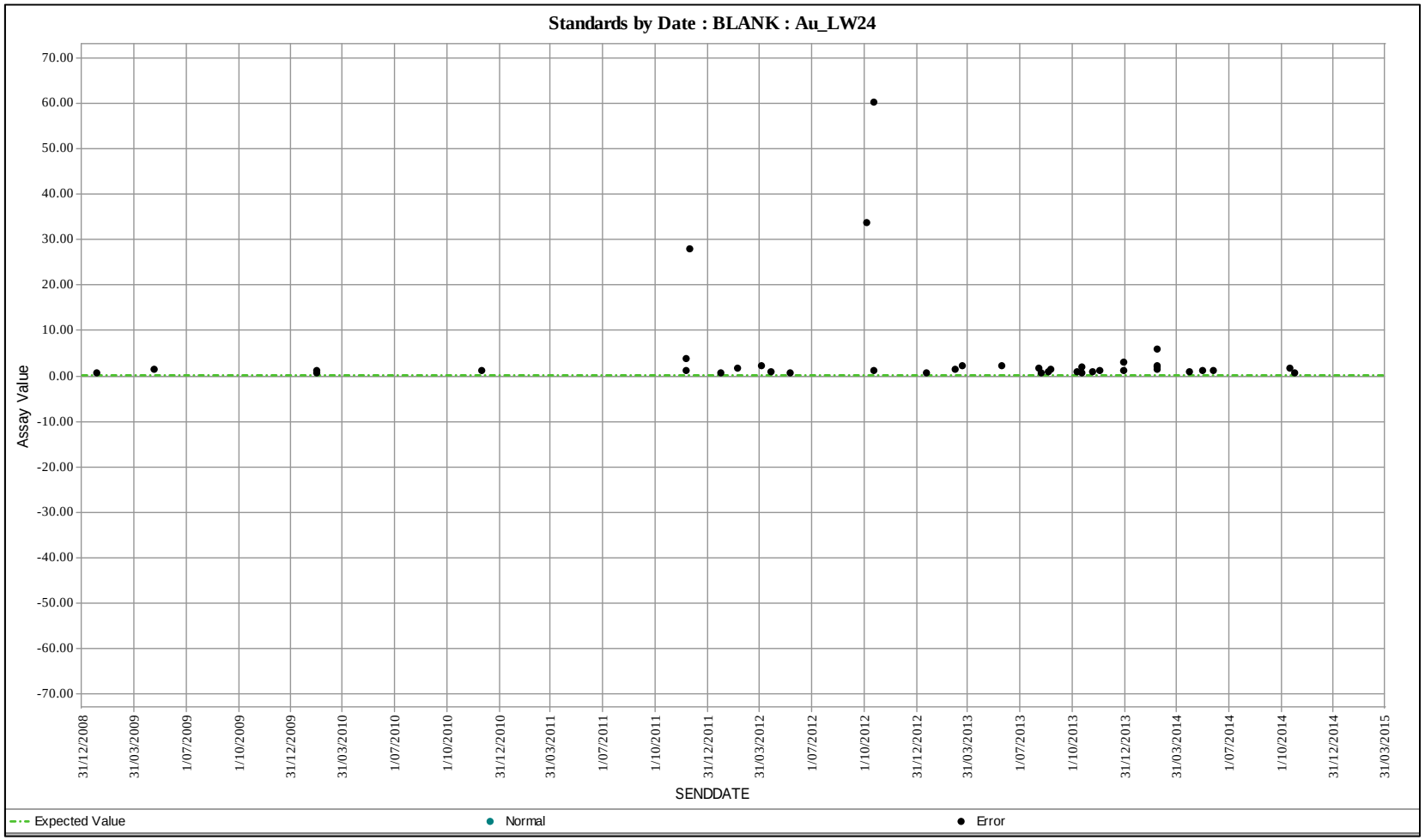
A total of 5998 blanks were processed between January 2009 and March 2015 of which 119 (1.98%) returned results above 0.2 g/t Au (three times the minimum reporting limit) (Figure 6-2). The reasons determined for each of the anomalous results is summarised in Table 6.9.

The reporting limit (RL) of 0.06 g/t Au differs from the detection limit (DL) in that it is meant to represents the lower limit that can be reliably measured and reported. Results close to the DL have a higher probability of representing false positives. The Gekko DL is set at three times the standard deviation of the blank. The RL is set at three and one-third times the DL. The highest reported Gekko DL has been 0.02 g/t Au.

Table 6.9 Summary of blanks with anomalous results

Reason	No.	Percentage of all blanks	Blanks average g/t Au	Preceding sample average g/t Au
Preceded by >9 g/t Au result (range 9.23 - 2,404 g/t Au)	101	1.68%	1.57	391.06
Followed by significant result or results mix up suspected	11	0.18%	0.96	3.22
No adjacent samples with significant results	7	0.12%	0.85	-

Figure 6-2 Chart of assay values returned on Blank standards.



6.4.2 Certified Reference Materials

CRMS and Standards

A total of 5,592 standards have been processed comprising of 33 different certified reference materials listed in Table 6.10 of these STD308GAN and STD905 are of low confidence due to poor or no supporting documentation. In addition, STD308 and STD308GAN were confused with each other and for a period mixed together. The data collected for these is deemed to be compromised. CGT transitioned away from the historical standards in September 2012 and has an on-going policy of rotating out a percentage of the standards which are then replaced by new sets. The use of LeachWELL certified standards has also been discussed with the suppliers in effort to improve QAQC. As of March 2015 none of the existing suppliers will commit to expanding their certification to include Leachwell.

Two 2014 standards (G908-3 and G308-6) had a significant proportion of their results outside of two standard deviations, 24.5% and 35.6% respectively. This was not believed to be due to issues with Gekko as other new standards introduced at the same time were behaving within expectations. The data for these two standards was relayed back to Geostats in November 2013. Geostats analysed the data and advised that the results were due to matrix effects; these two standards are no longer used.

Standard G907-2 which previously has had no issues has had a high rate of failures for a recently delivered batch. Other standards used during the same period have not shown the same trend. Geostats has been made aware of the issue and once sufficient data has been collected analysis to pinpoint the cause will be undertaken.

LeachWELL has been used extensively by LGL and CGT it is a bulk leach extractable gold (BLEG) method that can process larger sample volumes better suited for a nuggetty gold environment. The leaching process may not necessarily extract all the gold though particularly if tied up in refractory minerals or bound by carbon rich sediments; this may result in under calling the total amount of contained gold. The Geostats standards are supplied with both fire assay (total Au and small volume) and aqua regia (partial leach and small volume) certified values. Of these, only the aqua regia is referred to in Table 6.10 as it is expected to better match the LeachWELL results.

Table 6.10 Summary statistics for gold assay standards

Standard	Number of assays	Expected value (g/t Au)	Standard deviation (g/t Au)	Certified method
G308-3 (A)	93	2.47	0.13	Aqua Regia
G308-5 (B)	78	13.07	0.71	Aqua Regia
G900-2 (C)	189	1.45	0.11	Aqua Regia
G310-1 (D)	20	4.84	0.32	Aqua Regia
G307-8 (E)	176	1.97	0.12	Aqua Regia
G310-9 (F)	194	3.25	0.18	Aqua Regia
G310-8 (G)	68	7.92	0.45	Aqua Regia
G907-2 (H)	166	0.86	0.06	Aqua Regia
G908-3 (I)	49	1.00	0.05	Aqua Regia
G301-3 (J)	116	1.89	0.16	Aqua Regia
G910-6 (K)	178	3.05	0.18	Aqua Regia
G311-2 (L)	181	4.82	0.29	Aqua Regia
G908-8 (M)	141137	9.41	0.45	Aqua Regia
G307-7 (N)	39	7.75	0.45	Aqua Regia
G308-6 (O)	45	1.23	0.06	Aqua Regia
G910-5 (P)	34	5.21	0.23	Aqua Regia
G308-4 (Q)	197	6.65	0.44	Aqua Regia
G313-6 (R)	233230	4.91	0.25	Aqua Regia
G903-9 (S)	198197	11.15	0.77	Aqua Regia
G913-8 (U)	99	4.93	0.29	Aqua Regia
G310-10 (W)	19	47.74	2.03	Aqua Regia
G312-10 (X)	20	24.67	1.35	Aqua Regia
G907-7 (Y)	29	1.53	0.08	Aqua Regia
STD228	141	1.47	0.08	Aqua Regia
STD279	142	7.18	0.31	Aqua Regia
STD308	183	1.52	0.06	Aqua Regia
STD308GAN	180	Unknown	Unknown	Unknown
STD335	550	13.65	0.62	Fire Assay
STD383	762	7.24	0.27	Fire Assay
STD431	873	1.54	0.06	Fire Assay
STD483	96	13.16	0.47	Fire Assay
STD904 (G904-1)	81	12.53	0.74	Fire Assay
STD905	19	Unknown	Unknown	Unknown

6.4.3 Duplicates

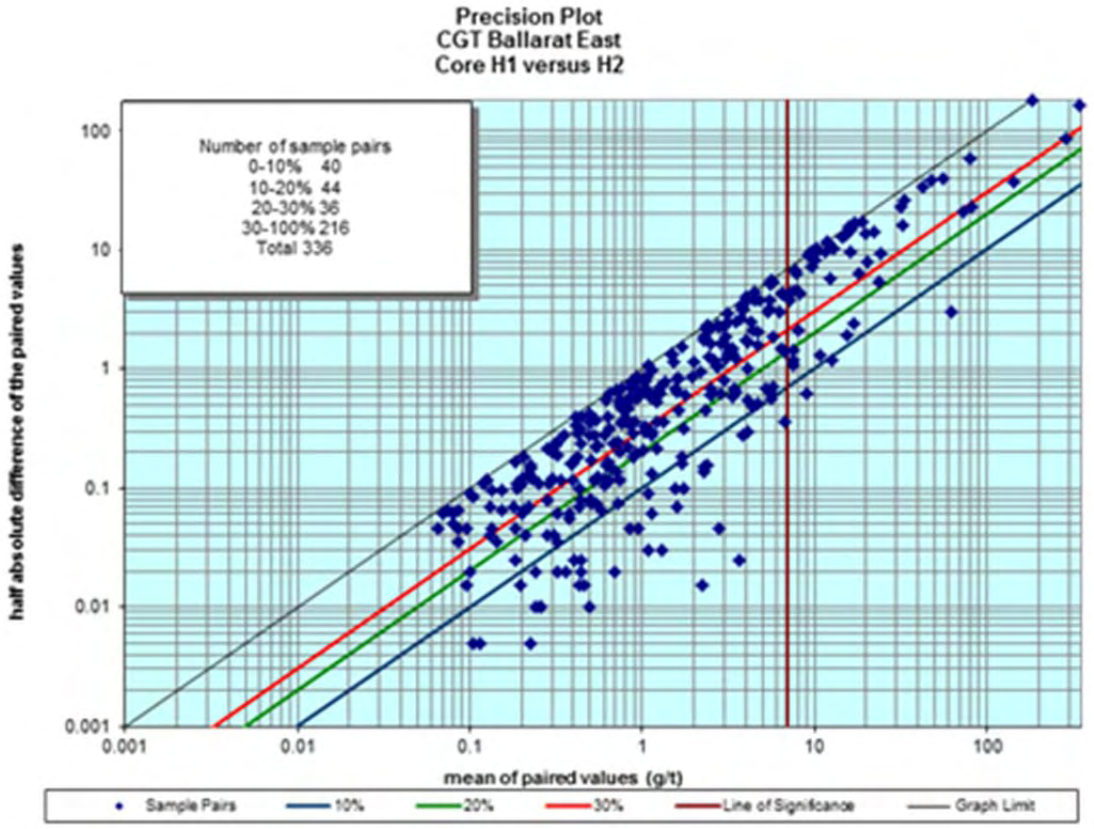
Large volume pulp duplicates

True field duplicates (half core vs half core) are available in limited numbers (seventeen records) during the LGL period, of which only five are within the current resource.

Some duplicate sample records were recorded as being field duplicates during the LGL period. However on closer inspection these were found to be assay duplicates, where whole core samples were pulverised and then split into two separate samples for assay. Bias may exist in these sample results, as this practise was only carried out on sections of core containing visible gold. For the purpose of this report, these duplicate are termed large volume pulp duplicates as each sample is around 1 kg.

The large volume pulp duplicates are limited to the Llanberris Mako resource, with a total of 336 records analysed. A reasonable measure of precision is the HARD plot – Half Absolute Relative Difference (= [ABS(H1-H2)/MEAN]*0.5) (Figure 6-3). This shows poor sample repeatability, with only 25% of sample pairs within ±20%. Such a result is to be expected in a coarse gold-dominated system such as Ballarat. The gold does not pulverise effectively and remains in the pulp. This validates the whole core assaying approach used by CGT.

Figure 6-3 Precision plot for duplicate samples collected by CGT from the Llanberris Mako lode



6.4.4 Check Analyses

Independent assay laboratory checks

In January 2014, the first campaign was completed where a CRM/standard would be run simultaneously through Gekko (internal standards blind testing), ALS (blind testing) and randomly through CGT sample batches. The standard selected by Gekko had already been used previously by CGT, so an additional historic comparison became available. All assaying was processed using the same weight (200 g) and apart from the historic data, the same production batch from Geostats. The method of analysis selected for use by ALS was equivalent to the Gekko LeachWELL 24 hour bottle roll.

The certified Aqua Regia value for G908-8 is 9.41g/t Au with a SD of 0.45 (acceptable range 8.51 g/t Au to 10.31 g/t Au). The results seen in Table 6.11 below indicate that all four sample populations are within one standard deviation of the certified value. Figure 6-4 show that overall the ALS results distribution is slightly higher than the Gekko results. In comparison Figure 6-5 shows that there is a closer match between the CGT and Gekko results as would be expected with both sets of CRM's being processed in the same

laboratory. Figure 6-6 indicates that the historical CGT data very closely matches the current CGT data confirming a level of consistency across time of the Gekko results.

Table 6.11 Summary statistics of independent assay laboratories

	ALS	CGT-current	CGT-historic	GEKKO
Mean (g/t Au)	9.69	9.42	9.42	9.49
Standard Error	0.04	0.03	0.03	0.03
Median	9.72	9.43	9.40	9.46
Mode	#N/A	9.40	9.30	9.45
Standard Deviation	0.18	0.22	0.32	0.15
Sample Variance	0.03	0.05	0.10	0.02
CV	0.018	0.023	0.033	0.016
Skewness	-0.37	-0.46	-0.39	0.17
Range	0.63	1.25	2.81	0.55
Minimum	9.34	8.65	7.94	9.25
Maximum	9.97	9.90	10.75	9.80
Sum	193.79	631.46	895.34	227.78
Count	20.00	67.00	95.00	24.00

Figure 6-4 QQ plot comparing analytical results from standard G908-8 between the Gekko laboratory and the ALS Laboratory

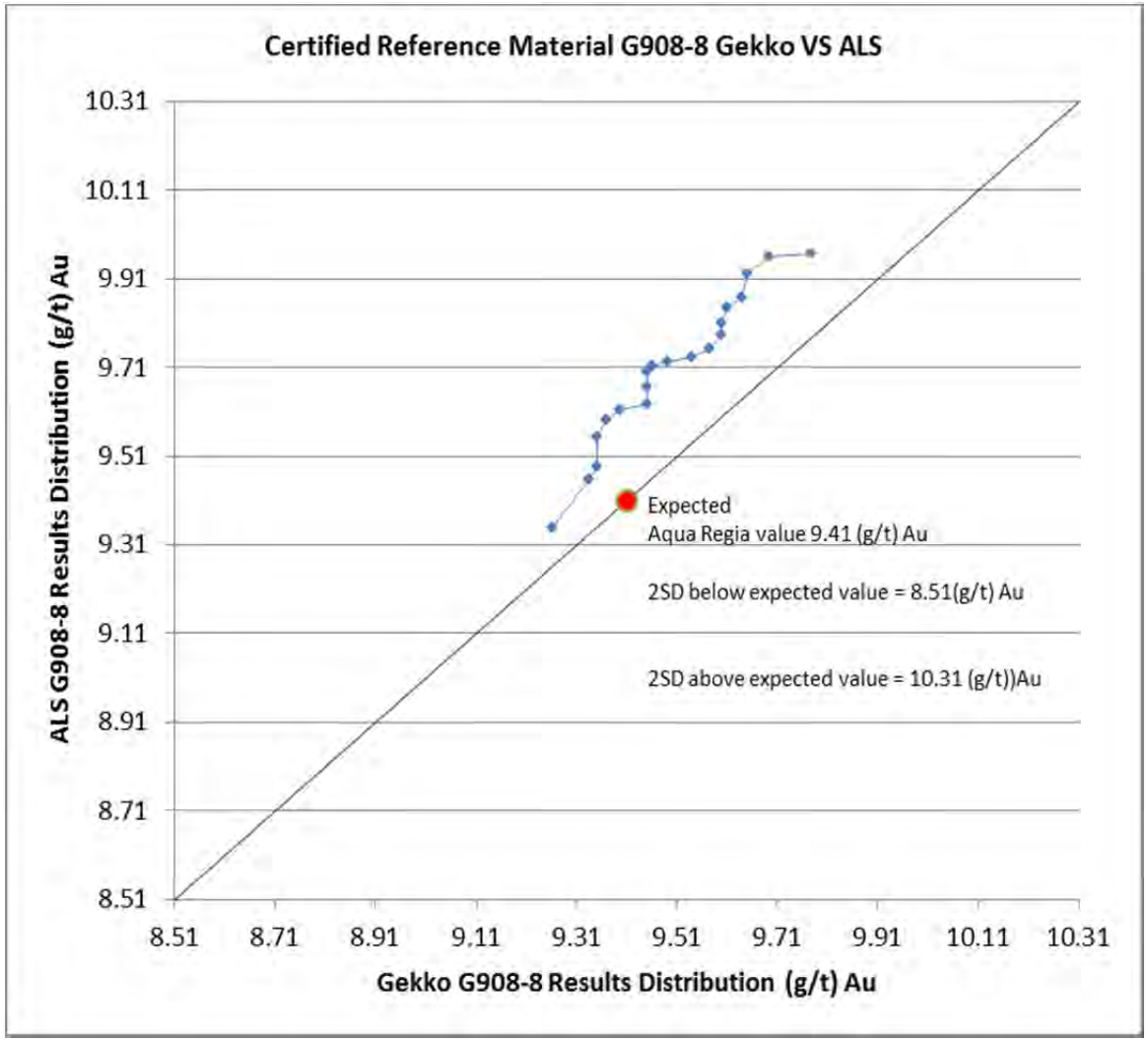


Figure 6-5 QQ plot comparing analytical results from the Gekko laboratory and CGT Batch data for standard G908-8

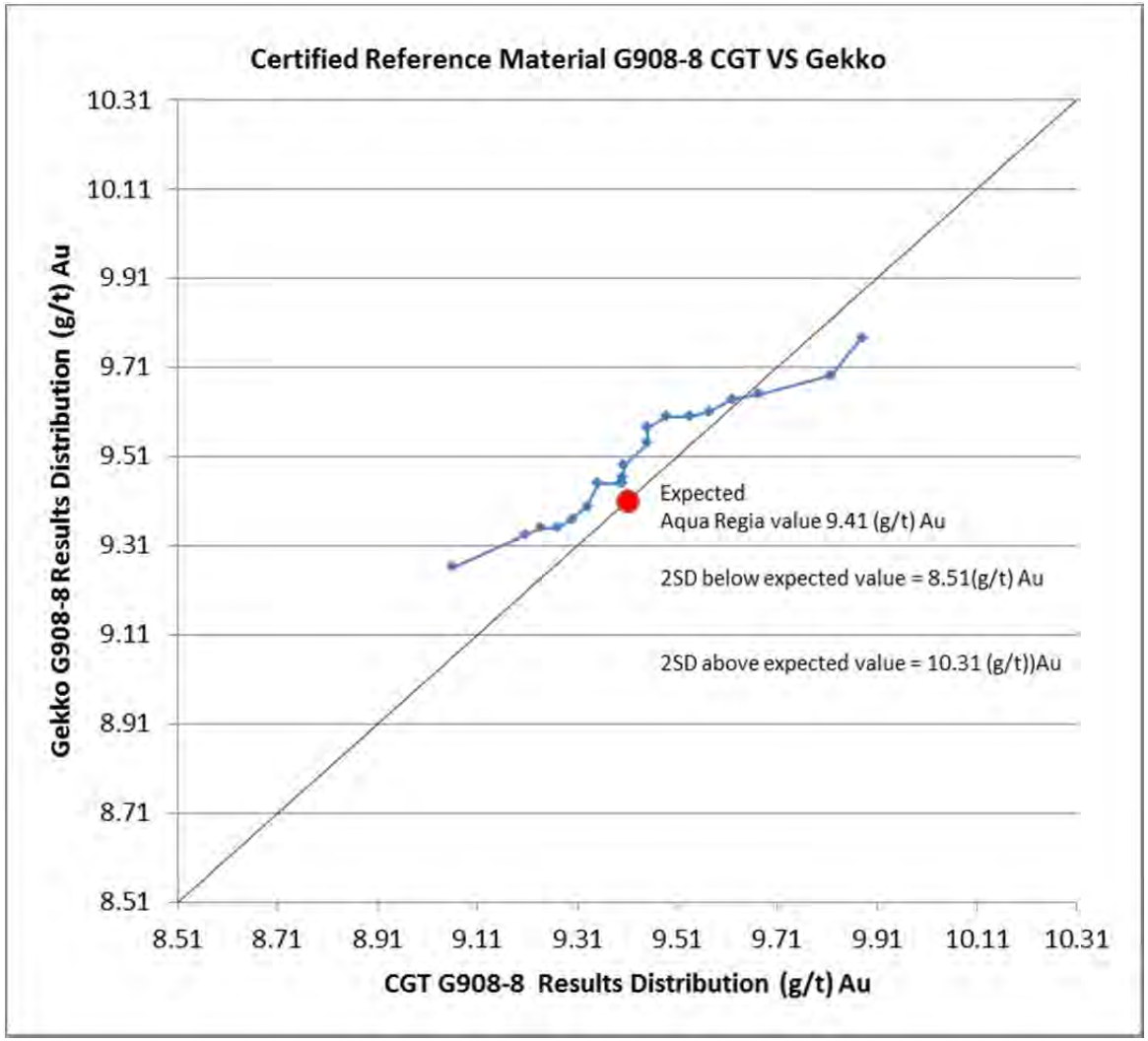
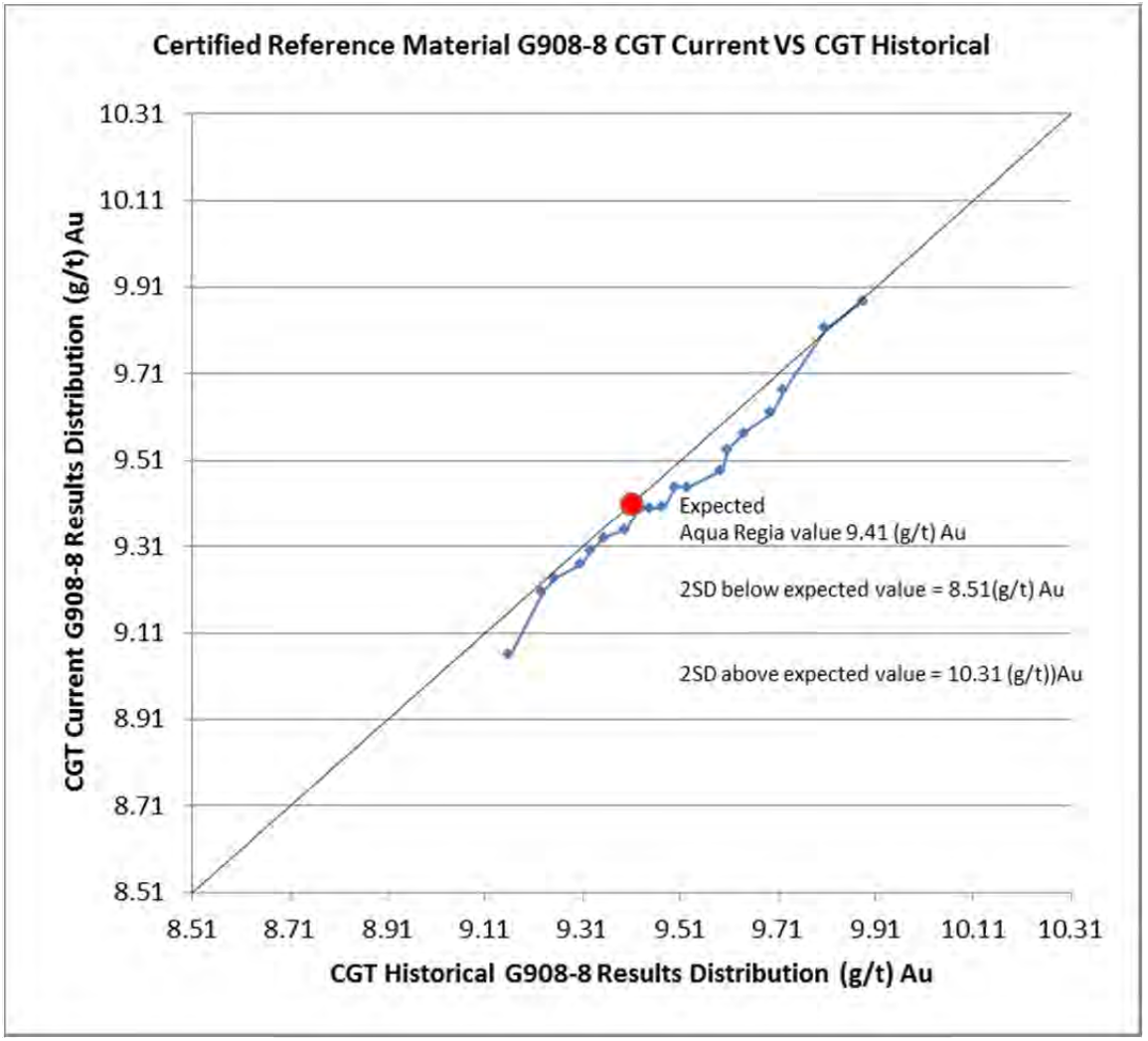


Figure 6-6 QQ plot comparing analytical results from CGT historical data and the current campaign data for standard G908-8



Comparison of drilling campaigns

The majority of drilling between January 2009 and March 2015 was completed by Deepcore Drilling (78%) and the remainder by LGL owned rigs (up to December 2010). Not all records were kept up to date in regards to drilling company so a comparison may be flawed, some records attributed to Deepcore or LGL may actually have been drilled by the other company.

A more complete record is the breakdown of assaying undertaken by different laboratories during this period particularly LGL vs CGT. For whole core samples all assaying during the LGL period covered by this resource were processed by the in house BGF laboratory. During the CGT period, all whole core samples were processed by Gekko (Table 6.12; Figure 6-7 and Figure 6-8). The results indicate a substantial difference in the 0 to 5 g/t Au range and this is may indicate of a material difference in the areas drilled by the respective companies such as increased coarse gold that may account for the increased variability. No valid field duplicates were taken of whole core samples.

Table 6.12 Comparison of summary statistics for whole core sample grades by LeachWELL

BGF_LW24		[CGT] GEKKO_LW24	
Mean (g/t Au)	2.18	Mean (g/t Au)	2.76
Median	0.19	Median	0.01
Mode	0.01	Mode	0.01
Standard Deviation	11.99	Standard Deviation	26.95
Sample Variance	143.61	Sample Variance	726.01
CV	5.50	CV	9.76
Skewness	17.00	Skewness	38.28
Range	427.57	Range	2,404.05
Minimum	0.01	Minimum	0.01
Maximum	427.58	Maximum	2,404.06
Sum	33,895.8	Sum	189558.5
Count	15,560	Count	68,670

Figure 6-7 QQ plot comparing analytical results from the BGF laboratory and the Gekko laboratory for grades between 0 and 5 g/t Au

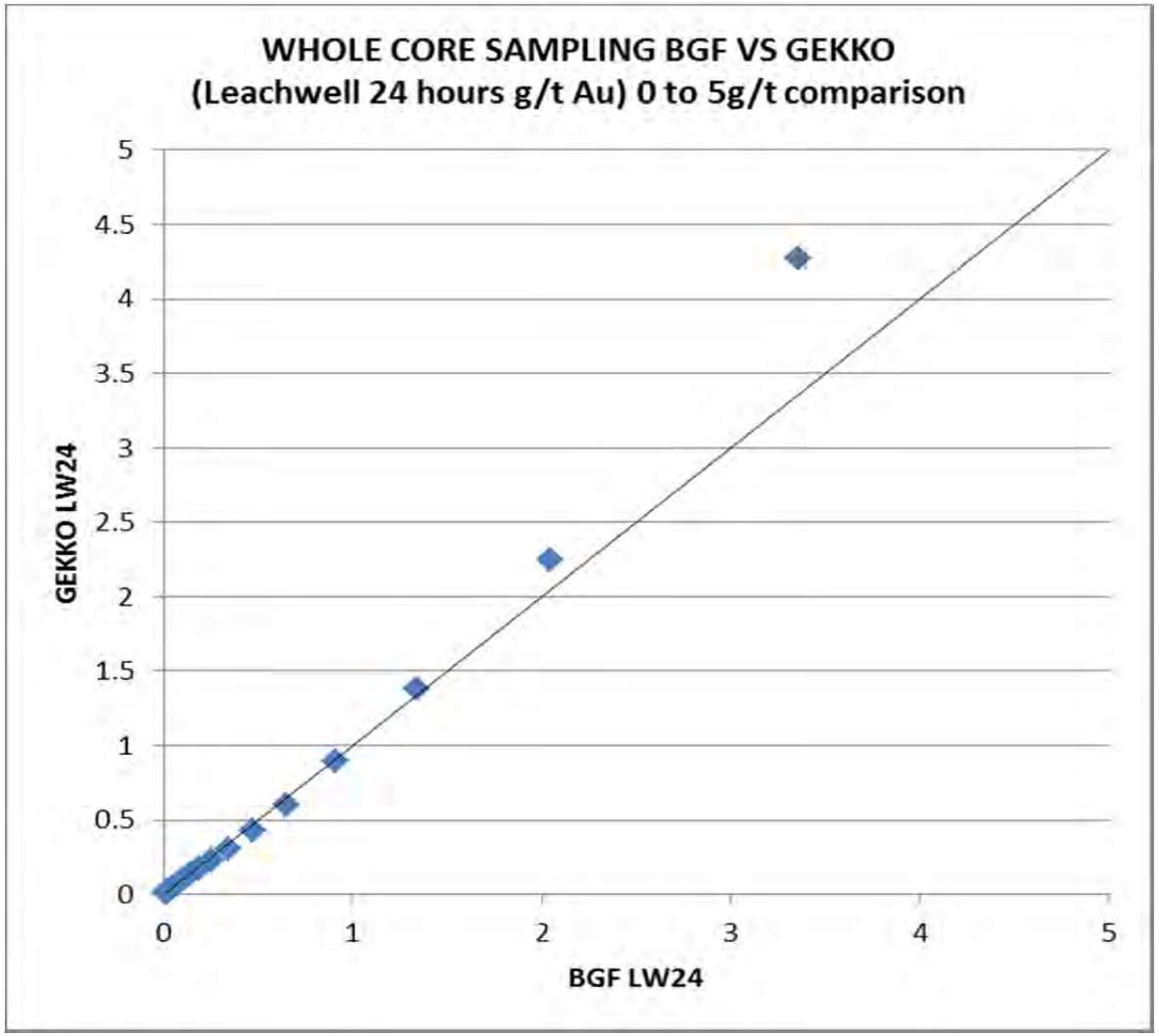
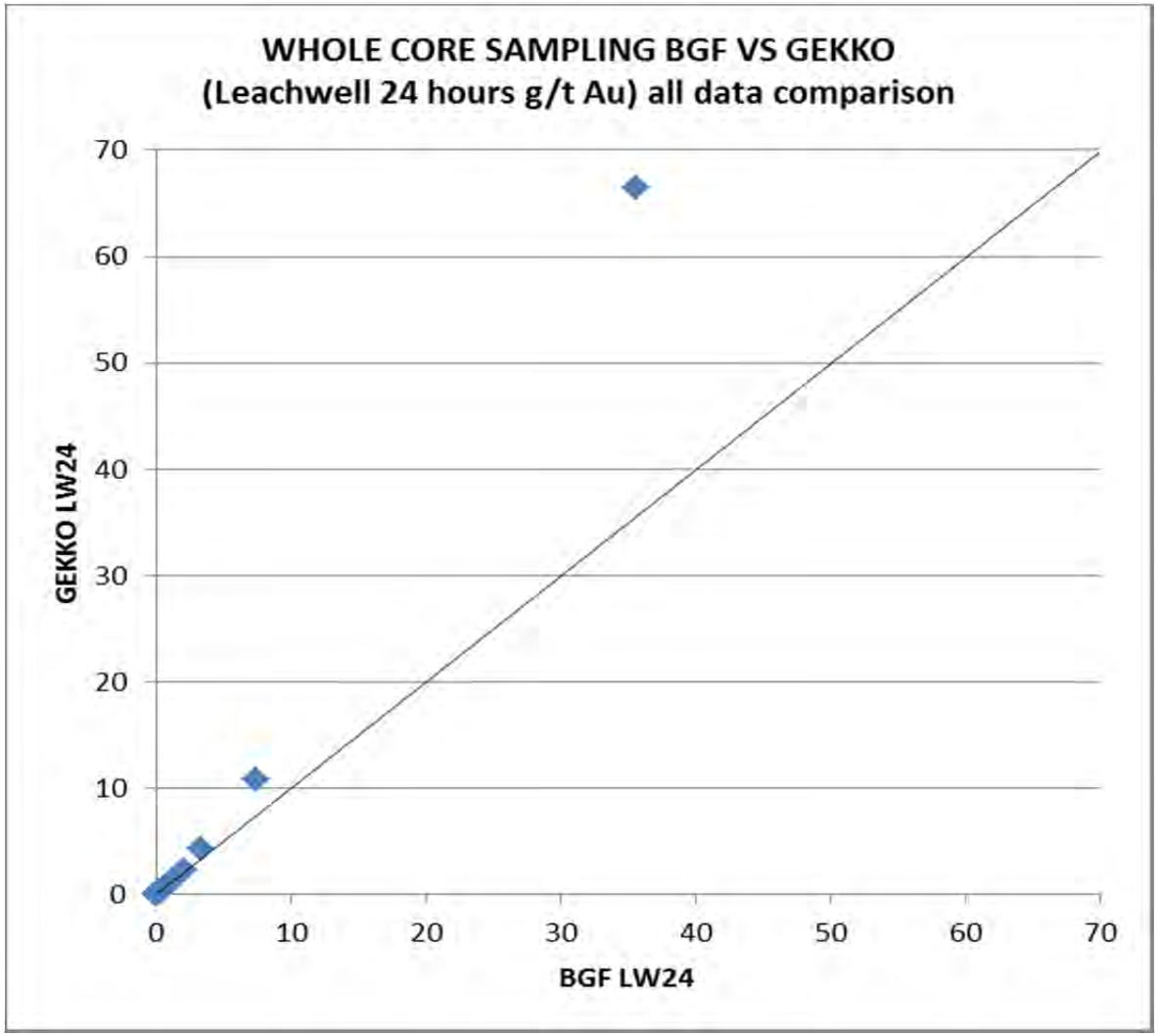


Figure 6-8 QQ plot comparing analytical results from the BGF laboratory and the Gekko laboratory



6.5 Data Entry and Validation

Geological data is entered into and uploaded electronically from laptop computers into the SQL database via an acQuire software program front end. Internal validations restrict the codes that can be entered with additional safeguards including automated overlap and interval gap checks. For hole ID's and sample numbers only unique values can be entered in to the database. Data entry is limited to geology logging staff with access permission set by the site IT manager.

Survey data validation

DDH collars are surveyed by the mine surveyor on completion of drilling and compared against planned position and previous pickups of nearby drill hole collars. If valid the collar co-ordinates are updated in the database. The data file is returned to survey for storage. Where a collar is unable to be located by the mine surveyor a calculated collar position may be generated if multiple collars exist from the same drilling location and it can be confirmed that the drill rig has not moved position.

Geological logging validation

Geological core logging is entered directly into the acQuire database using laptop computers or "tough-books" with live links to acQuire. Logging data entry requires geologists to select logging codes from a list of approved codes, rather than manually typed in to avoid typing errors.

During data entry any overlapping intervals entered by the logging geologist are flagged by an error message as soon as they occur, prompting the geologist to correct the error. Upon completion of logging, it is procedure for a macro in acQuire to be run by the logging geologist to flag and correct any gaps or un-logged intervals within the log.

Upon completion of core photography, prior to sampling, logging geologists are required to review their logging against the photographs taken. Any discrepancies found between the photography and the logging, are corrected prior to sampling commencing.

Upon receipt of assays, significant intersections are reviewed against logging and core photos to ensure that the assays received are consistent with expectations. This check is initially carried out by the responsible logging geologist.

Core recovery validation

Core recovery is recorded in the lithology logging field of AcQuire database as "core-loss". Core loss is initially located by diamond drilling staff during core layout underground, with wooden core blocks placed in the core tray to demark the position of the lost core. They are marked "core loss" with the length of lost core written underneath.

During core orientation and mark up by field assistants, the position of core loss is reviewed, run lengths are checked to ensure the claimed lost core interval is consistent with the recovered core. Breaks in the core either side of the lost core interval are checked to ensure they don't match. Where a core loss interval is considered to be questionable, the geologist responsible for logging the core is consulted to determine the correct position of the interval. Where necessary, diamond drilling staff is consulted to determine the final position.

7 MINERAL PROCESSING AND METALLURGICAL TESTING

7.1 Overview

The processing plant relies solely on gravity separation for primary recovery of gold. Optimisation of the gravity circuit has therefore been a focus for metallurgical improvement and in particular, the potential for flotation of gravity tailings to raise gold recovery. In addition, improvement to the gravity circuit to reduce the carryover of free gold into leach has also been a key area of work during the past year.

From a leaching perspective, the focus of metallurgical test work has centred on the resin column and stripping circuit with the aim of increasing gold throughput without impacting recovery.

A full review of the plant is provided in Section **Error! Reference source not found.**.

7.2 Metallurgical Test Work

The key areas of metallurgical test work and plant optimisation over the 2013/2014 year have included:

- Gravity circuit surveys to better understand individual mineral responses to the gravity separation process and provide information on which to optimise the circuit operation. This included size by size analysis of the gravity tailings to understand the nature of gold losses to tailings.
- Improving the separation of free gold from sulphides to reduce the load on the gold room and minimise the carryover of free gold into the leaching circuit. Free gold carryover into the leaching circuit involves a recovery penalty. Improvements implemented involved reconfiguration of the recleaner jig circuit and the installation of a nugget trap and Gekko Spinner (ISP30).
- Laboratory based test work to investigate the merits of flotation to recover fine gold and sulphides currently being lost to gravity tailings and hence to raise overall gold recovery. The majority of the gold in gravity tailings was found to be recoverable using grinding and flotation
- Numerous plant trials to optimise the performance of the resin stripping and electrowinning circuit, namely to decrease stripping times, raise the amount of gold per strip and in turn increase the gold extraction capacity. At present, concentrate yields in the gravity circuit exceed the concentrate leach capability due to higher contained gold grades. Double stripping trials (double the resin) are currently in progress.

7.3 Metallurgical Accounting

- Metallurgical accounting is performed based on gold produced, gold in tailings discharge and gold in circuit (GIC) - including concentrate stockpiled.
- Samples are taken by hand sampling of solid and slurry streams.
- Monthly plant recovery calculations are reconciled against indicated gold in feed compared to actual gold in feed (gold produced, gold in tail and GIC).

7.4 Mineral Processing Design

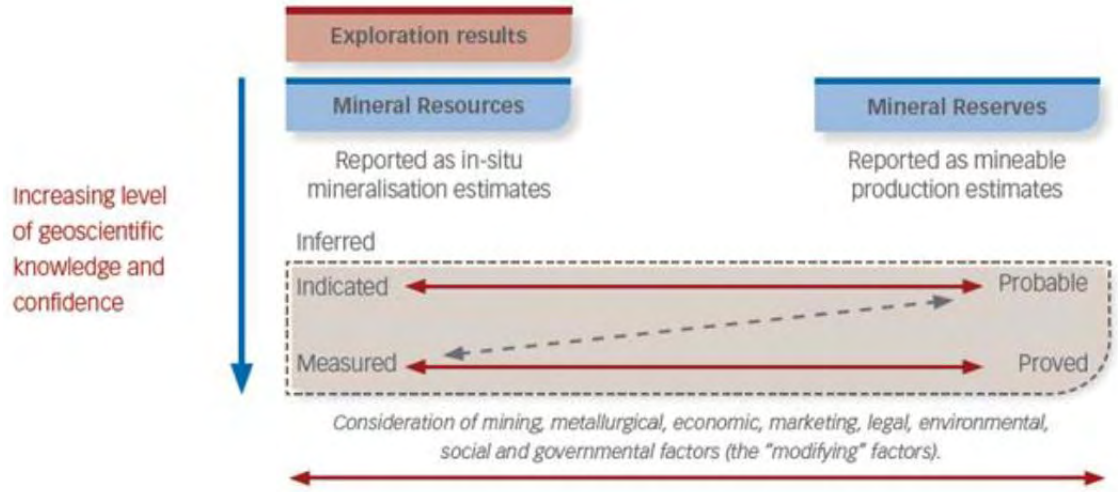
- Following promising laboratory scale test work, further work is planned to develop a suitable design for the addition of a ball mill and flotation circuit into the existing processing plant flow sheet.
- Going forward, the leaching circuit may require the addition of a second resin column to increase the gold extraction capability and allow higher concentrate feed rates as dictated by the gravity circuit mass recovery. However this will ultimately depend on the outcome of the double stripping trials outlined above. The original design for the leaching circuit included two resin columns but due to budgetary constraints at the time, the previous owners decided to install just the one column initially.

8 MINERAL RESOURCES

8.1 Summary of Mineral Resources

This Mineral Resource has been classified in accordance with The JORC Code (2012). When following the guidelines of The JORC Code (2012), tonnage and grade estimates are classified so as to reflect different levels of geological confidence and different degrees of technical and economic evaluation. A geologist will estimate the Mineral Resource using geoscientific information such as drill hole cores, sample assay values and QA/QC data (Figure 8-1).

Figure 8-1 General relationship between Exploration Results, Mineral Resources and Ore Reserves



The Mineral Resource of the six Ballarat lodes as of March 2015 is given in Table 8.1.

Table 8.1 Mineral Resource summary as of 31 March 2015. All resources reported at 0g/t Au cut-off

Category	Mineral type	Gross attributable to licence		Net attributable to issuer (100%)			Remarks
		Tonnes	Grade (g/t Au)	Tonnes	Grade (g/t Au)	Change (tonnes) from previous update (%)	
Indicated Mineral Resource*	Gold	4,000	14.2	4,000	14.2	-	Britannia Mako
	Gold	25,500	12.8	25,500	12.8	-	Britannia Basking
	Gold	9,500	7.5	9,500	7.5	-	Llanberris Basking
	Gold	19,500	15.6	19,500	15.6	-	Sovereign Tiger
	Gold	21,000	23.7	21,000	23.7	-	Sovereign Gummy
Inferred Mineral Resource*	Gold	81,000	7.9	81,000	7.9	-29.9	Britannia Mako
	Gold	182,500	6.8	182,500	6.8	-	Britannia Basking
	Gold	36,000	8.1	36,000	8.1	46.9	Llanberris Basking
	Gold	49,000	6.3	49,000	6.3	-55.9	Llanberris Mako
	Gold	30,500	5.3	30,500	5.3	-	Sovereign Tiger
	Gold	81,000	7.9	81,000	7.9	-26.7	Sovereign Gummy
Total	Gold	**539,500**	**8.4**	**539,500**	**8.4**	**45.8%**	**-**

** The reserves reported in Section 9 of this report are based on Gold contained within the Resources listed above, therefore, this Mineral Resource estimate is reported inclusive of Mineral Reserves.*
Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t.

8.2 General Description of Mineral Resource Estimation Process

CGT has completed an update of its Mineral Resource estimate for the Ballarat mine. The estimate consists of mineralisation within six discreet fault zones referred to as lodes. Each lode is represented by a series of mineralisation wireframes with a combined volume of 986,698 m^3. Tonnage and grade values have been estimated based on 524 diamond drill holes drilled between 2009 and 2015.

Six block models have been created to estimate each of the lodes defined by CGT. Wireframes were constructed of geological domains within each of the lodes and were used to constrain the block model. Blocks that had already been mined were flagged in order to generate results for both unmined and depleted areas. An inverse distance squared estimation algorithm was applied, with composite top-cut grades selected using statistical analysis of the distribution of grade within each domain.

The model was classified as Indicated and Inferred Mineral Resources according to the definitions in The JORC Code (2012). After all items specified within The JORC Code (2012) [see JORC Table 1 in Appendix A of this report] such as sampling techniques, data quality and estimation techniques were considered, the resources were classified according to drill hole density and spacing, as well as taking into account the number of samples and search ranges used to inform block estimates. The interpolated block model was validated through visual checks, a comparison of the mean composite and block grades, and through the generation of section validation slices.

8.3 Mineral Resource Estimate

8.3.1 Mineral Resource Input Data

Drillhole data

The total drillhole database covers a region spanning from 35,400 mN to 39,150 mN and 51,700 mE to 53,800 mE (mine grid). Since modern exploration commenced in 1991, over 4,500 diamond drill holes have been drilled into the Ballarat East goldfield.

The dataset used for this resource estimate has been restricted to drill holes which penetrate the six lodes relevant to the current resource estimate and only considers holes drilled between 2007 and 2015 (Table 8.2). This consists of 485 unique diamond drill holes representing, 62,959 m of diamond drill core and a total of 37,373 assay data records.

A total of 23 holes have been omitted from the data set.

Drill-holes with a “BDD” prefix were drilled in 1991, and were sampled using 50 g fire assay.

Drilling is carried out in east-west trending vertical fans spaced approximately 25 m apart. Hole spacing within fans varies between 7 m and 20 m. Placement of diamond drill holes within the current mineral resources is shown in Figure 8-2 and Figure 8-3.

Table 8.2 Summary of drill hole data informing the Ballarat resource

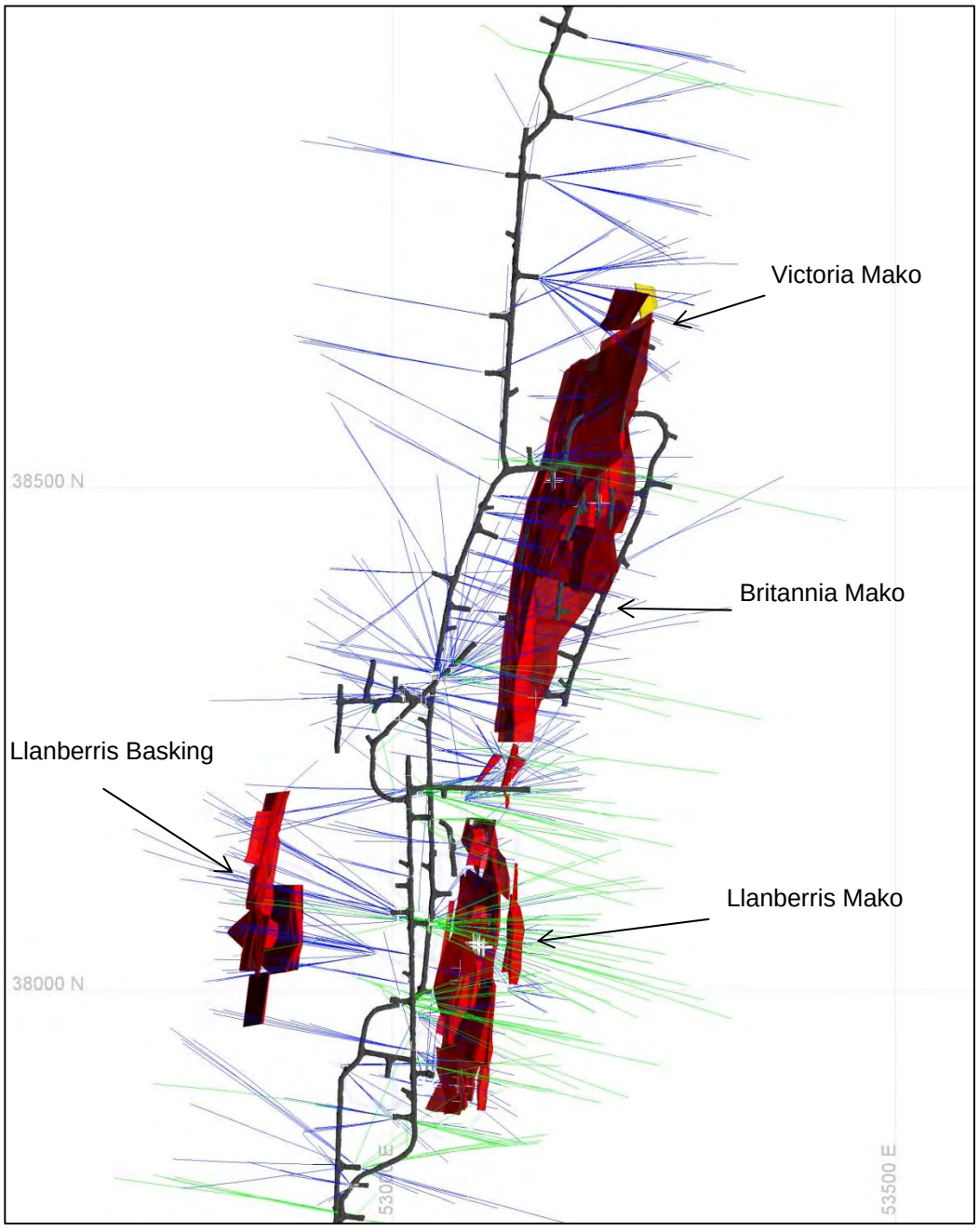
Resource	Diamond drill holes	Samples	Metres drilled (m)
Britannia Mako	120	12,239	18,389
Britannia Basking	101	6,646	12,751
Llanberris Mako	56	5,700	7,173
Llanberris Basking	80	4,303	11,420
Sovereign Gummy	95	7,901	11,128
Sovereign Tiger	72	8,504	7,158
Grand Total	**513**	**40,888**	**67,027**

Note: The sum of the collars and samples listed in the table above does not match the grand total, this is because of multiple drill holes were used to inform more than one lode. Note not all drill holes informing the resource end up within the final modelled domains.

Table 8.3 Drill holes excluded from the Ballarat dataset

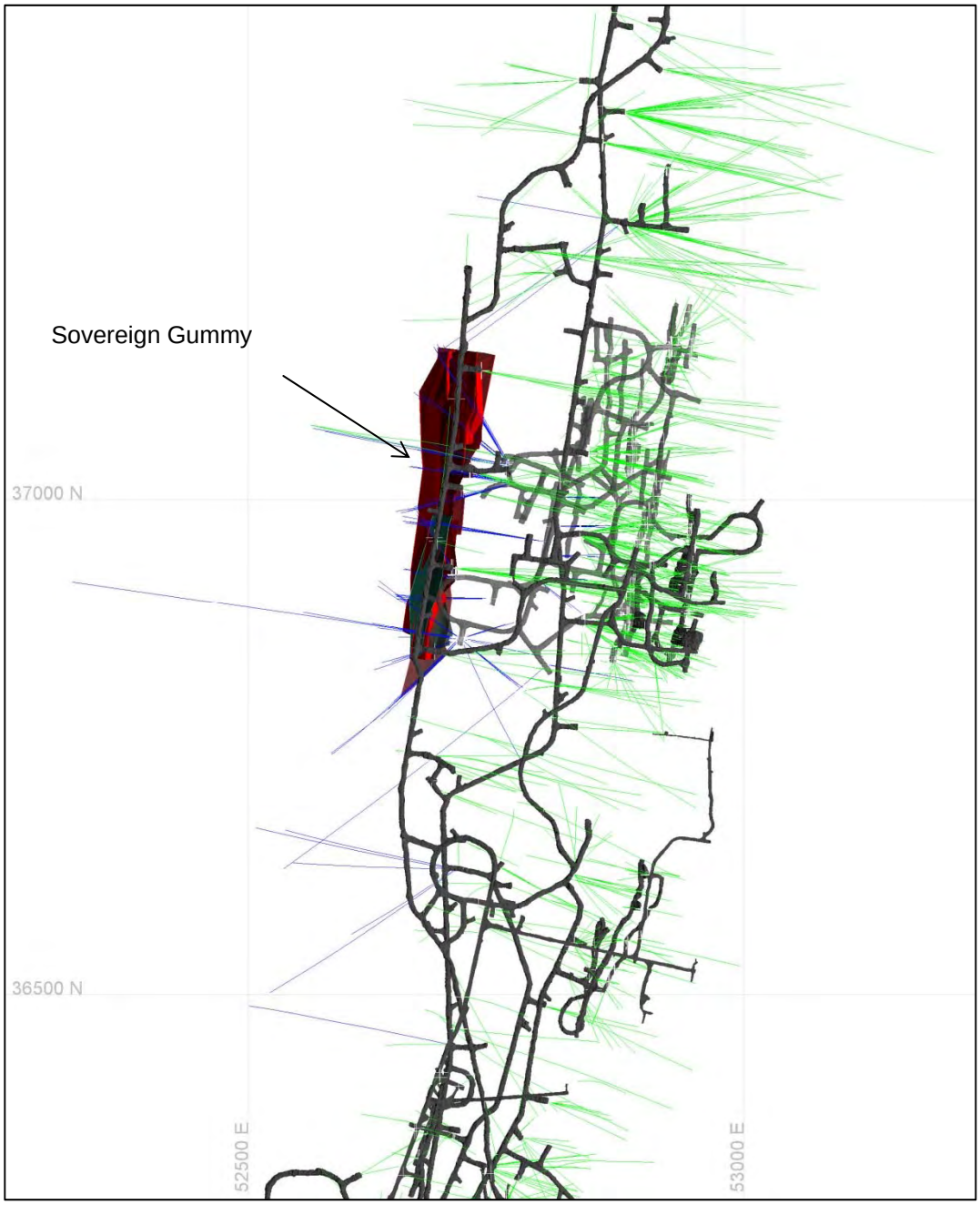
Hole Id	Collar Verification	Assay type
BDD001C	Not verifiable at this time	Fire assay 50 g charge
BDD002A	Not verifiable at this time	Fire assay 50 g charge
BDD002B	Not verifiable at this time	Fire assay 50 g charge
BDD003	Not verifiable at this time	Fire assay 50 g charge
BDD004	Not verifiable at this time	Fire assay 50 g charge
BDD004A	Not verifiable at this time	Fire assay 50 g charge
BDD006	Not verifiable at this time	Fire assay 50 g charge
BDD008	Not verifiable at this time	Fire assay 50 g charge
BDD008A	Not verifiable at this time	Fire assay 50 g charge
BDD008B	Not verifiable at this time	Fire assay 50 g charge
BDD008D	Not verifiable at this time	Fire assay 50 g charge
BDD008E2	Not verifiable at this time	Fire assay 50 g charge
CBG013	Not verifiable at this time	Leachwell 2000
BEU265	Not verifiable at this time	Leachwell 2000
CBU143	Not verifiable at this time	Leachwell 2000
CBU1379	Not verifiable at this time	Leachwell 2000
CBU109	Not verifiable at this time	Leachwell 2000
CBU1290	Not verifiable at this time	Leachwell 2000
CBU839	Not verifiable at this time	Leachwell 2000
CBU834	Not verifiable at this time	Leachwell 2000
CBU837	Not verifiable at this time	Leachwell 2000
CBU838A	Not verifiable at this time	Leachwell 2000
CBU826	Not verifiable at this time	Leachwell 2000

Figure 8-2 Plan view of location of the Llanberris Mako, Britannia Mako, Victoria Mako and Llanberris Basking drill holes (mine grid)



Note: Green drill hole traces indicate drill holes completed prior to CGT ownership, blue drill hole traces indicate drilling carried out by CGT.

Figure 8-3 Plan view of location of the Sovereign Gummy drill holes (mine grid)



Note: Green drill hole traces indicate drill holes completed prior to CGT ownership, blue drill hole traces indicate drilling carried out by CGT.

Drill hole logging

Qualitative code logging was undertaken for lithology, alteration, veining and geotechnical rock quality. Structural measurements of bedding, cleavage and fault planes were taken where possible to aid in the interpretation of the ore body orientation. The core is oriented against the north-south trending cleavage which is pervasive throughout the goldfield. This has been confirmed by geological mapping to be consistent throughout the underground mine workings. The intersection of geological structures logged from drill core and subsequently intersected by underground development has verified the means of core orientation.

A trial of the use of a core orientation tool was carried out in September 2012. The trial found that for holes drilled perpendicular to the strike of cleavage, when compared against the cleavage method (considered to be reliable), only 45% of the orientation lines marked on core were within 20^{o} of expectations. In addition, mineralised fault zones are often associated with very broken core; as a result it was uncommon for core orientation marks to be marked on core through mineralised fault zones. Using cleavage as the main orientation method was considered a more practical approach.

Geological logging was carried out on all drill holes informing the estimate. An outline of the codes used during core logging is given in Table 8.4 to Table 8.7.

Table 8.4 Core logging lithology codes used at the Ballarat mine

Sedimentary data codes (Sed_Log)	Description/definition
	SAND LITHOLOGY CODES
CSD	Coarse sand
MSD	Medium sand
FSD	Fine sand
VFS	Very fine sand
SAI	Interbedded sediments within a dominant sand lithology
	SHALE LITHOLOGY CODES
ISH	Interbedded sediments within a dominant shale lithology
RBS	Thinly bedded shale and silt (~1-3cm beds)
LSH	Laminated shale (<1cm beds)
MSH	Massive shale
BSH	Black shale
	VEIN LITHOLOGY CODES
QVM	Quartz vein massive
QVS	Quartz vein stylolitic
QVL	Quartz vein laminated
QVC	Quartz vein crustiform
QVB	Quartz vein brecciated
QVA	Quartz vein breccia sealed
QVU	Quartz vein undifferentiated
	OTHER LITHOLOGY CODES
FLT	Fault zone
DYK	Dyke
NAV	Navi run
VOD	Void
NOC	No core/core loss
PRE	Pre-collar
XXX	Not geologically logged

Table 8.5 Core logging mineralisation codes used at the Ballarat mine

Sedimentary data codes (Sed_Log)	Description/definition
VISIBLE AU CODES	
AU1	1 spec of Au
AU2	2 to 9 specs of Au
AU3	1 spot or/and 10+ specs of Au
AU4	2 to 10 spots and/or 15+ specs of Au
AU5	1 slug and/or 10+ spots of Au
MIN1 to MIN5 CODES	
SA	Arsenopyrite
SPO	Pyrrhotite
SGN	Galena
SCP	Chalcopyrite
SSP	Sphalerite
SPY	Pyrite
Y	Clay mineral

Table 8.6 Core logging alteration codes used at the Ballarat mine

Alteration data codes	Description/definition
ALT_INTENSITY CODES	
W	Weak
M	Moderate
S	Strong
ALT_MIN1 to ALT_MIN_3 CODES	
MS	Sericite
B	Carbonate
C	Chlorite
SPY	Pyrite
SA	Arsenopyrite

Table 8.7 Core logging structure codes used at the Ballarat mine

Structure data codes	Description/definition
	ORIENTATION CODES
ASM	Assumed from logging knowledge
CLE	Estimated bottom of hole (Pseudo orientation line) using cleavage plane
BED	Estimated bottom of hole (Pseudo orientation line) using bedding plane
ORI	Measured from orientation line derived from down-hole “ori” tool
	STRUCTURE CODES
cv	Cleavage
bd	Bedding
fa	Fault plane
fg	Fault Gouge
ac	Anticline hinge
sn	Syncline hinge
DYK	Dyke Contact
	VEIN ORIENTATION CODES
VB	Bedded
VC	Axial planar/Cleavage
VP	Perpendicular/Cross cutting
VU	Undifferentiated
	VEIN TEXTURE CODES
M	Massive
B	Brecciated
L	Laminated
S	Stylolitic
	FACING CODES
U	Younging (facing) Up hole
D	Younging (facing) down hole
	FACING EVIDENCE CODES
SL	Slumping
SC	Scours
XB	Cross Bedding
GB	Graded Bedding
FL	Flame Structures
SB	Sharp Bedding Contact
VG	Vergence

Core photos are taken of each core tray throughout all holes informing the resource. Over the period during which the drilling was carried out, a number of changes have been made to the core logging procedure to streamline and improve the process. These changes did not affect the way mineralisation domains are identified and interpreted.

Drillhole sampling

Core sample intervals were selected to represent mineralised zones. Mineralised zones are identified based on lithology and structural features i.e. such as faulting, percentage quartz and quartz textures. Sample start and finish points were adjusted so as not to cross lithological boundaries where possible. This practise allows for statistical analysis of grade distributions within specific zones of mineralisation. However samples must be composited to a constant length to ensure consistent sample support for estimation purposes.

Two sample collection methods were used to collect samples from drill holes informing the estimate. Samples collected prior to 2011 were half diamond saw cut and sampled to nominal 1 m lengths. In 2011, the sampling method was changed to whole core sampling on nominal 0.4 m lengths for NQ2 core and 0.5 m lengths for LTK60 core. The nominal lengths were selected to generate approximately 2 kg of sample material as required for Leachwell 2000 analysis. Table 8.8 provides detail of the sampling methods used in each of the lodes included in this estimate.

The change to full core sampling in 2011 was made to increase the volume of samples collected from diamond drill holes. This required a reduction in maximum sample length to provide the requisite 2 kg of sample required for Leachwell 2000 analysis. The maximum sampling length was increased to 70cm in August 2014 which increased the sample size to 3.5 kg, the sample is now pulverised using LM5's and once homogenised is then split using a rotary splitter. Approximately 2 to 2.3 kg is then analysed using Leachwell 2000 and the reject kept until the results are verified as acceptable via included certified reference materials.

Table 8.8 Sampling methods used on Ballarat drill holes within modelled domains

Lode	Sample type	No. Collars	No. Samples	Total sample length (m)
Britannia_Mako	full core	103	2,344	817.76
	half core	16	146	86.68
	Total	119	2,490	904.44
Britannia_Basking	full core	88	1,541	673.15
	half core	9	63	40.48
	Total	97	1,604	713.63
Llanberris_Mako	full core	47	1,069	385.74
	Total	47	1,069	385.74
Llanberris_Basking	full core	68	1,154	423.26
	half core	13	168	108.76
	Total	81	1,322	532.52
Sovereign_Tiger	full core	56	1,483	472.26
	Total	56	1,483	472.26
Sovereign_Gummy	full core	91	3,224	1,188.39
	Total	91	3,224	1,188.39
Grand total		**485**	**11,192**	**4,196.97**

Note: The sum of the collars listed in the table above does not match the grand total; this is because two holes in the estimate intersected mineralisation in both the Sovereign Gummy and Tiger lodes as well earlier CGT holes having two or three types of samples thus getting counted multiple times.

Topography

CGT's Topographical GIS layers have been supplied by Spatial Vision (August 2012) under licence through the Victorian Government Department of Sustainability and Environment Spatial Information Infrastructure.

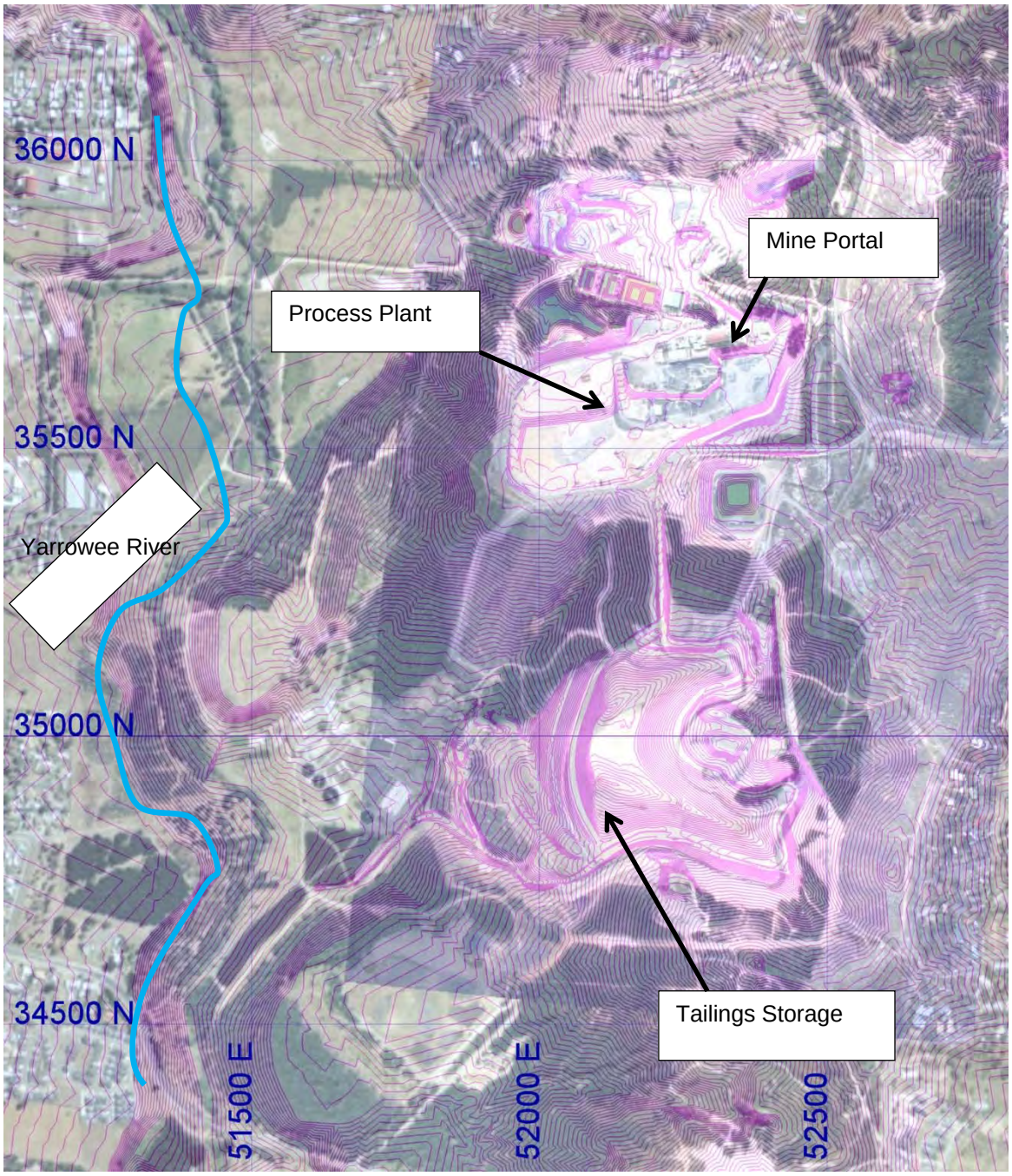
Since all holes used in this resource estimate were drilled from underground, accuracy of topography is not a primary concern. Details regarding the lineage and accuracy of the topographic layer are outlined in Table 8.9.

Topography can have a significant impact on the ability for mining to take place. Figure 8-4 shows the position of the Ballarat mine facilities with 1 m contours overlaid. The Ballarat mine is positioned to take advantage of the local topography, with the process plant and portal positioned on a topographic high. The tailings storage facility is positioned down-hill from the process plant.

Table 8.9 Topography elevation layer data quality summary

Data set source	
Lineage	Data has been derived from Melbourne Water base maps and converted to Microstation .DGN format.
Processing steps	
Positional Accuracy	Varies with scale of capture and the contour interval. e.g., 1 m contours from aerial photos +/- 0.5 m 0.2 m contours from survey +/- 0.1 m
Attribute Accuracy	Varies with scale of capture and the contour interval. e.g., 1 m contours from aerial photos +/- 0.5 m 0.2 m contours from survey +/- 0.1 m

Figure 8-4 DTM over the Ballarat mine site (1 m contours – not to scale)



Data validation

Validation of the drill hole data was performed before commencing statistical analysis and estimation. These validation checks were;

- checks for duplicate collar location records
- overlapping assay intervals
- negative assay values
- drillhole depth vs. final “To” depth

There were no errors found in the final data-set for duplicate collar locations, overlapping assay intervals, negative assay intervals or drill hole depth vs. final “To” depth check.

Collar validation

As the collars for all holes informing this resource are all drilled from underground positions, validation was limited to a comparison of the collars against the triangulation of the underground workings. This validation was performed in Vulcan. No significant discrepancies between collar positions and the surveyed underground workings were identified.

Drill hole collars are regularly picked up by survey and that data is integrated into the drill hole database, in addition since May 2014 the walls and floors around the collars are also picked as the profile of the development can change significantly overtime. In situations where a collar is lost before the survey pick up a calculated collar may be used.

The process involves using the position of other drill holes drilled at the same location and time to calculate the centre point of the fan of drill holes. The down-hole surveys are used to confirm the angles of each hole. Using the common centre point as the start point and the drilled dip and azimuth for direction, a line is projected to the surveyed surface giving the calculated position for the collar.

Calculated collars are only used where a collar could not be found, but was within a fan drilled in one campaign without a rig move. In this instance, the hole is assumed to be half way between the collars adjacent to it in the fan. Calculated collars are not definable for single hole sites or sites where multiple holes are drilled at different times or different rigs due to variations in rig placement and/or rig height.

Drill hole validation identified eight holes having no survey pick up for the collar location (Table 8.10). Drill holes with only planned collar locations have been removed from the resource estimate. Drill holes with calculated collar locations have been included in the resource estimate as they are estimated to a level of accuracy within 300 mm.

Table 8.10 Summary of drill holes without collar location survey pickups

Hole_ID	Collar location source	Down hole survey type	Depth of hole (metres)	Resource
CBU143	Planned	Multi-shot	268.8	Britannia Mako
CBU1379	Planned	Multi-shot	110.1	Britannia Basking
CBU109	Planned	Multi-shot	300	Britannia Basking
CBU1290	Planned	Multi-shot	119.6	Britannia Basking
CBU839	Planned	Multi-shot	131.7	Llanberris Mako
CBU834	Planned	Multi-shot	91.6	Llanberris Mako
CBU837	Planned	Multi-shot	109.9	Llanberris Mako
CBU838A	Planned	Multi-shot	130.6	Llanberris Mako
CBU826	Planned	Multi-shot	90.6	Llanberris Mako
CBU497	Calculated	Multi-shot	35.3	Britannia Mako
CBU508	Calculated	Single shot	27.3	Britannia Mako
CBU512	Calculated	Multi-shot	89.1	Britannia Mako
CBU514	Calculated	Single shot	60.7	Britannia Mako
CBU927	Calculated	Multi-shot	155.7	Llanberris Mako
CBU369	Calculated	Multi-shot	149.8	Llanberris Basking

CBU1599	Calculated	Multi-shot	119.86	Llanberris Basking

Downhole survey data validation

As mentioned previously, downhole surveys were carried out within 90% of the drill holes informing this resource. Holes which lacked down-hole surveys have been reviewed to validate that the geology intersected was consistent with adjacent drill holes which were surveyed.

A total of 12 holes out 485 only had single shot downhole surveys (Table 8.11).

Table 8.11 Drill holes with only single shot down hole data

Hole ID	Resource	Depth (m)
CBU100	Britannia Basking	301.9
CBU1086	Britannia Basking	134
CBU1179	Britannia Basking	66.7
CBU364	Llanberris Basking	182.1
CBU912	Llanberris Mako	60.6

The initial orientation for each drill hole is based on a survey set out matching the drilling plans. The drill hole pathway is initially monitored using single shot surveys done at 15 m, 30 m and then every 30 m interval to monitor drill hole path. If the 15 m survey deviates by more than a pre-set amount, drilling halts and the deviation is investigated. For significant differences additional checks using different cameras are done. If the issue is not resolved by this the rig orientation is surveyed.

Once a drill hole is completed a multi-shot survey (measurements every 3 m) is undertaken. The multi-shot data is then compared to the single shot data. Anomalous data points are downgraded (based on magnetic data and visual inspection) so that they do not influence the drill hole path. Where a single shot coincides with a multi-shot survey, the multi-shot takes precedence unless there is evidence the data is compromised.

Near the collar the downhole surveys are less reliable and the azimuth is often affected by the presence of metal (e.g. rig, cable bolts, split sets). Using the multi-shot data beyond the affected area the azimuth can be back calculated and inserted as a modified 0 depth survey.

Assay validation

The assay data was validated for negatives, text values, significant figures, overlapping intervals and above and below detection limits. Assay intervals were compared against lithology logs to ensure no assays were attributed to intervals of lost core. This highlighted 2.9m (0.07%) of sample length from a total of 4,197m which had grade for invalid lithological codes (lost core, pre-collar, not logged), these have been reviewed to ensure they do not have a material impact on the resource and have been corrected where possible. Procedures and systems will be put in place to prevent these errors from occurring. Raw assay data statistics are shown in Table 8.12.

Table 8.12 Raw assay data statistics (not declustered)

Resource	Measure	Samples	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Standard deviation (g/t Au)	Coefficient of variation (CV)
Britannia Mako	Length	2490	-	1.1	0.36	0.15	0.42
	AU_1ST	2490	0.01	807.69	7.46	33.57	4.5
Britannia Basking	Length	1,604	-	1	0.45	0.17	0.39
	AU_1ST	1,604	0.01	884.55	6.35	35.49	5.59
Llanberris Mako	Length	1,069	0.001	0.7	0.36	0.10	0.27
	AU_1ST	1,069	0.01	1,027.5	5.84	41.3	7.08
Llanberris Basking	Length	1,322	0.001	1.1	0.40	0.19	0.47
	AU_1ST	1,322	0.01	1,031.18	5.92	40.64	6.87
Sovereign Gummy	Length	3,224	-	0.75	0.37	0.11	0.30
	AU_1ST	3,224	0.01	1,017	7.02	41.90	5.97
Sovereign Tiger	Length	1,483	-	0.5	0.32	0.11	0.36
	AU_1ST	1,483	0.01	533.06	6.07	30.94	5.10
Combined	Length	11,192	-	1.1	0.38	0.15	0.37
	AU_1ST	11,192	0.01	1031.18	6.65	37.71	5.85

8.3.2 Geological Interpretation

Interpretation

Geological interpretation is initially carried out on paper sections at a 1:100 and/or 1:250 scale. Depending upon the level of detail required to illustrate the complexity of the geology intersected. Interpretations are carried out on sections plotted on drill fan centres.

Interpretations are carried out in two passes. A working section interpretation is carried out by the core logging geologist, as the logging for each hole is completed the working section is updated. When the fan of drilling is complete, and all surveys and assays have been received, validated and updated in the database, a final interpretation is carried out by the project exploration geologist supervising the drill rig.

Geological interpretations take into consideration lithological units identified, the orientation of major faults intersected, the orientation of individual quartz veins and the position of fold hinges. Interpretations are based on recognition of mineralisation styles based on characteristics observed during mining. Where available, face, wall and backs mapping of exposures underground were also incorporated into geological interpretations. Figure 8-6 provides an example of a working geological interpretation. The geology observed in the two faces photographed and registered in Vulcan has influenced the interpretation.

All geological interpretations are peer reviewed by the Geology Manager and the Senior Mine Resource Geologist. Daily meetings are held between senior geology staff and logging geologists. The logging completed on the previous day, and the preliminary interpretations formed by the logging geologists are reviewed and discussed.

Expected mineralisation and structures intersected on adjacent sections are marked onto new drill sections so as to guide geologist's interpretations, where discrepancies arise, alternative interpretations and requirements for additional drill hole testing are considered.

Upon completion of a drill fan, interpretations are finalised and compared against adjacent sections where available to assess continuity of the structures and mineralisation interpreted.

This estimate considers mineralisation within five separate lodes. These lodes are separated by a combination of cross-course faults and major thrust faults. In general major cross-course faults, have divided the goldfield into compartments. The major thrust faults have been offset by these cross-courses (Figure 8-5). Descriptions of the mineralisation styles specific to each of the lodes can be found in Section 5.3.

Figure 8-5 Long-section looking east showing position of the six lodes relative to the major cross-course faults which separate mining compartments (not to scale)

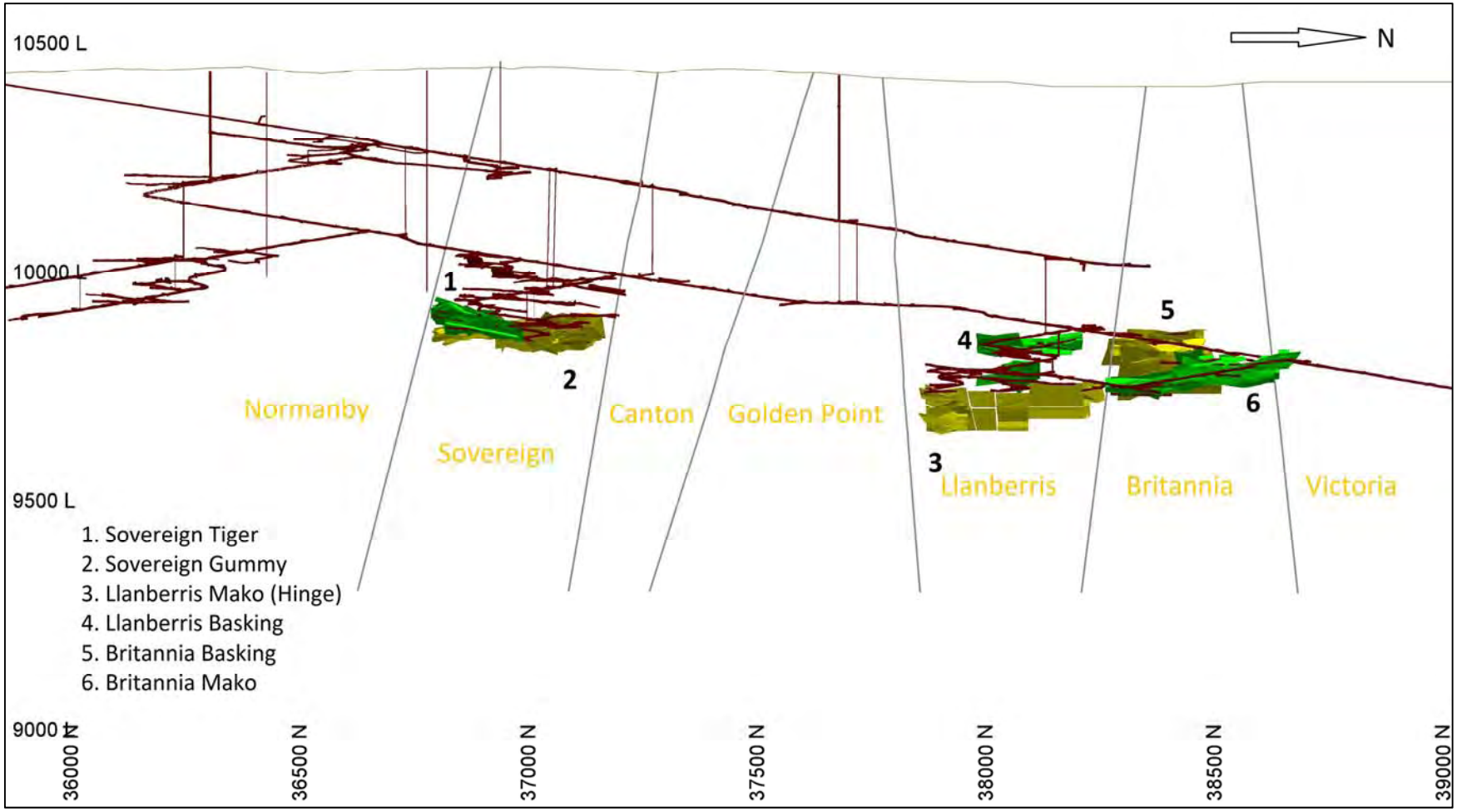


Figure 8-6 Example of a geological interpretation working section 38,485 mN looking north (not to scale)

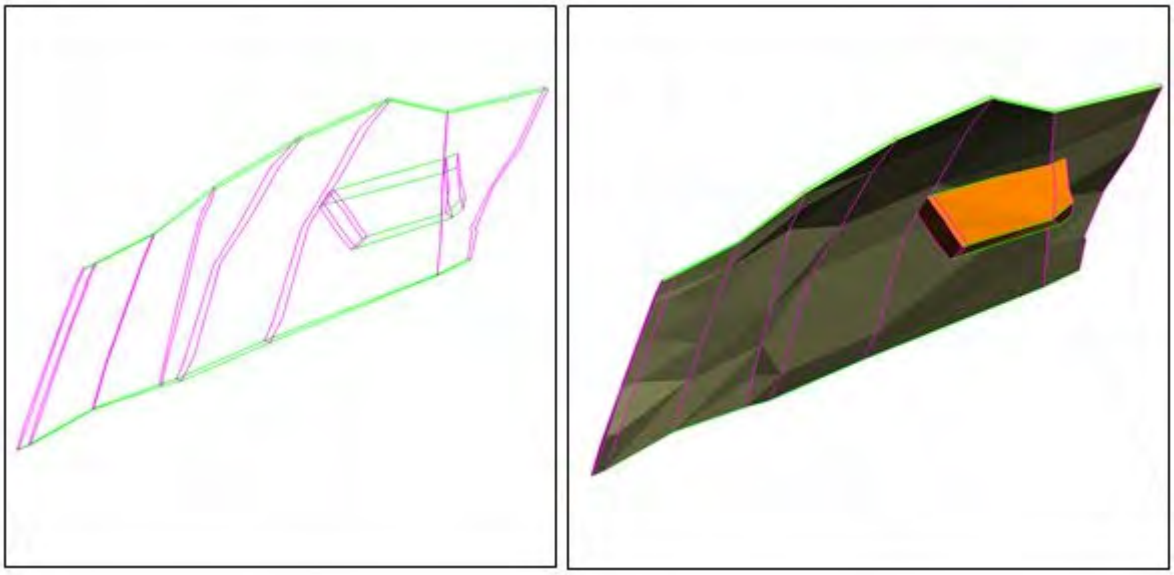


Modelling

Sectional strings are digitised by the Senior Resource Geologist using Vulcan software V9.1. Ore domains are defined based on the geological interpretations carried out by logging geologists. These strings are used to generate solid wireframes. Where necessary tie-strings are utilised on domain corners to ensure domain boundaries are triangulated correctly. Figure 8-7 demonstrates the construction of solid wireframes from strings in Vulcan.

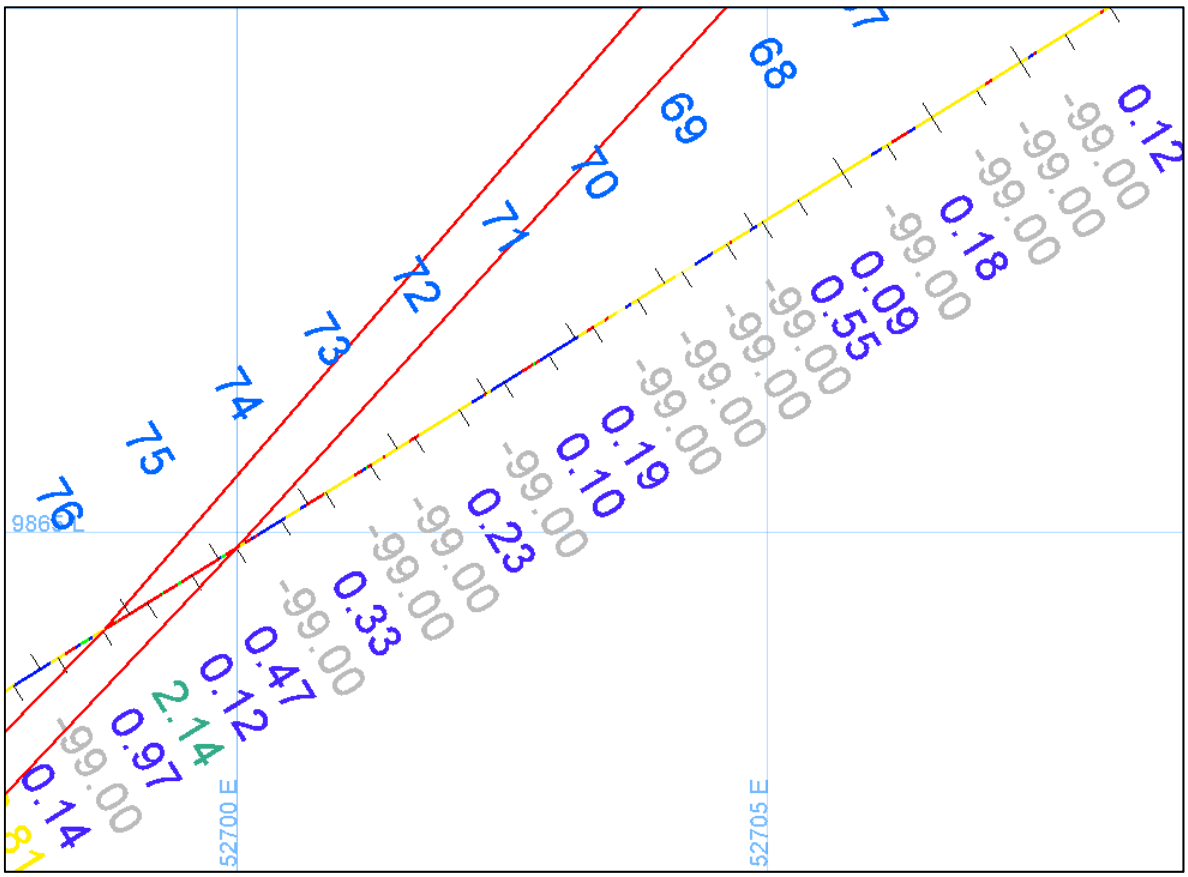
Wireframes are checked for closure, consistency, crossing triangles, small triangles, small angles and coincident points prior to estimation. All wireframes informing this estimate passed these tests.

Figure 8-7 Wireframe construction: sectional strings and triangulated surfaces



String points have been snapped to assay intervals to ensure that assay data is correctly allocated during estimation. Assay intervals are selected during sampling to, wherever possible, honour lithological boundaries. This is the case for the main west-dipping and east-dipping structures. However in the stockwork zones adjacent the main structural features it is common for mineralisation to consist of a number of very narrow (less than 5 cm thick) veins spaced between 5 cm and 20 cm apart. In this instance it is not practical for sampling to represent each of the veins present, so sample intervals are selected to represent the stockwork zone rather than each of the veins. Extrapolation of wireframes is limited to half drill spacing. Figure 8-8 provides an example of sample interval allocation within a west-dipping fault zone and the stockwork zone adjacent on the footwall of the fault.

Figure 8-8 Example of sample interval selection relative to lithological boundaries in diamond drill hole CBU633 at 37027mN



Down hole depths are shown above drill trace, sample intervals and assays are shown below the trace. Quartz veining is coloured red on trace, sandstone is yellow and slate is blue. The red lines indicate the margins of a west-dipping fault. The west-dipping fault is represented by three assays with intervals conforming to the boundaries of the fault zone. On the footwall of the main fault,there are a number of narrow quartz veins, interpreted to be part of a stockwork zone. These veins are too narrow to be sampled individually, instead samples have been taken to represent the stockwork zone rather than the individual veins.

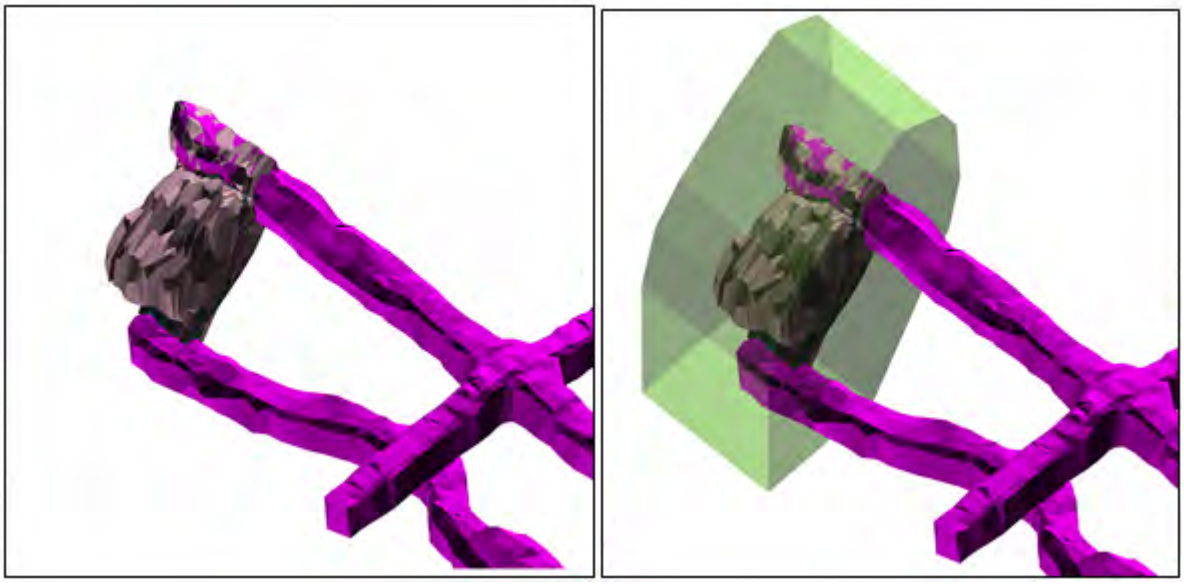
Mining depletion models

Mining depletion shapes are generated from strings created in Surpac software by the mine surveyor. These strings are collected underground using a total station for ore drives, and a CMS for stope voids.

The models created to deplete resource block models represent both mined voids and any zones of sterilisation around unstable voids such as un-filled stopes. A single triangulation is created for each lode; this triangulation consists of the surveyed void, with areas of sterilisation modelled to include the specified sterilisation in each case.

Geotechnical staffs are consulted regarding the size of exclusion zones around unstable voids during the modelling process. Stopes stabilised with cemented rock fill (CRF) require a 5 m sterilisation halo to be applied, based on 3D geotechnical modelling open stopes require a 8 m sterilisation halo. Where significant ground failures have occurred during stoping, or there are concerns around the impact of major fault zones on ground stability, 3D geotechnical modelling is undertaken, in which case larger sterilisation halos are specified on a case by case basis. Figure 8-9 provides an example of the application of an 8 m exclusion zone around an open stope. This zone is considered sterilised and has been depleted from the block model.

Figure 8-9 Mining depletion wireframe construction and sterilisation around unstable void



8.3.3 Data Analysis and Geostatistics

Statistical analysis of raw assay data

Sample lengths

The raw assay results were exported from Vulcan as a straight composite file and imported into Snowden Supervisor for statistical analysis. Sample lengths can be categorised into two groups due to a change in sampling procedures implemented in 2011. Holes prior to the change were cut in half using a diamond saw, with one half sent for assay on 1 m intervals. Holes sampled after the change were no longer cut in half, but whole core sampled to 0.4 m lengths for NQ2 diameter core and 0.5 m lengths for LTK60 sized core.

Variation in the distribution of sample lengths for each of the lodes estimated is summarised in

Table 8.13 and histograms for each lode are shown in Figure 8-14 to Figure 8-19.

The Llanberris Mako, Llanberris Basking, Britannia Mako and Britannia Basking were tested with both half core and full core sampling. These lodes include a wider range of sample lengths than the Sovereign Gummy and Basking lodes which were tested using only full core sampling.

No samples were filtered as having lengths less than or equal to zero. Sample lengths for each sample type and lode are summarised in

Table 8.13. No intervals were identified that were longer than expected, from August 2014 the maximum NQ2 sample length was increased to 0.7m and the minimum to a nominal 0.3m.

Table 8.13 Summary statistics for raw sample lengths

Sample type and lode	No. collars	No. samples	Sample length (m)		
			Mean	Min	Max
Full core					
Britannia Mako	104	11,001	0.38	0.08	0.7
Britannia Basking	92	5,933	0.47	0.15	1.4
Llanberris Mako	56	5,568	0.39	0.15	1.25
Llanberris Basking	69	3,671	0.4	0.1	1.7
Sovereign Gummy	94	7,893	0.4	0.1	1.3
Sovereign Tiger	72	3,537	0.39	0.1	0.86
Total	*485*	*37,603*	*0.40*	*0.1*	*1.7*
Half core					
Britannia Mako	18	761	0.72	0.2	1.1
Britannia Basking	10	386	0.67	0.3	1.4
Llanberris Basking	13	428	0.76	0.1	1.7
Total	*41*	*1,575*	*0.72*	*0.1*	*1.7*
Grand Total	**513**	**39,178**	**0.42**	**0.1**	**1.7**

Note: The sum of the collars listed in the table above does not match the grand total; this is because several holes are counted twice having been sections of half core and full core, two holes intersect both the Sovereign Gummy and Tiger lodes.

Some variation in diamond drill core sample length has been noted. This is, however, to be expected when core is correctly sampled to lithological boundaries. Figure 8-10 is a scatter plot of sample length versus grade. Based on this comparison a relationship between sample length and maximum grade does exist, with the maximum grade generally decreasing with increasing sample length as expected.

A more important metric is the average grade for each sample interval. This relationship is complicated by several factors including the historical sampling practice of only half coring sections with visible gold and using full core for the rest of the sample. This has caused the half core data to have with significant peak around the 0.7m length range due to a greater proportion of samples with visible gold and the inverse in the full core data.

Another complicating factor is the uneven distribution of visible gold within the range of sample lengths, there is a slight bias towards the smaller sample lengths as seen in Figure 8-13. This due to the presence of gold in narrow veins which are sampled to the lithological boundaries, this practice avoids grade smearing into non-mineralised lithological units.

Figure 8-11 shows more variability in the smaller lengths intervals, this due to the large differences in data density between the sample lengths as seen in Figure 8-13 and Table 8.14.

Table 8.14 Number of sample length categories classified by sample support.

Sample count for discrete sample lengths	No. of full core lengths	No. of half core lengths
1 to 29	45	21
30 to 99	10	2
100 to 999	11	7
1,000 to 9,999	6	-
10,000 to 30,000	1	-

Figure 8-10 Scatter plot sample length versus raw gold grade

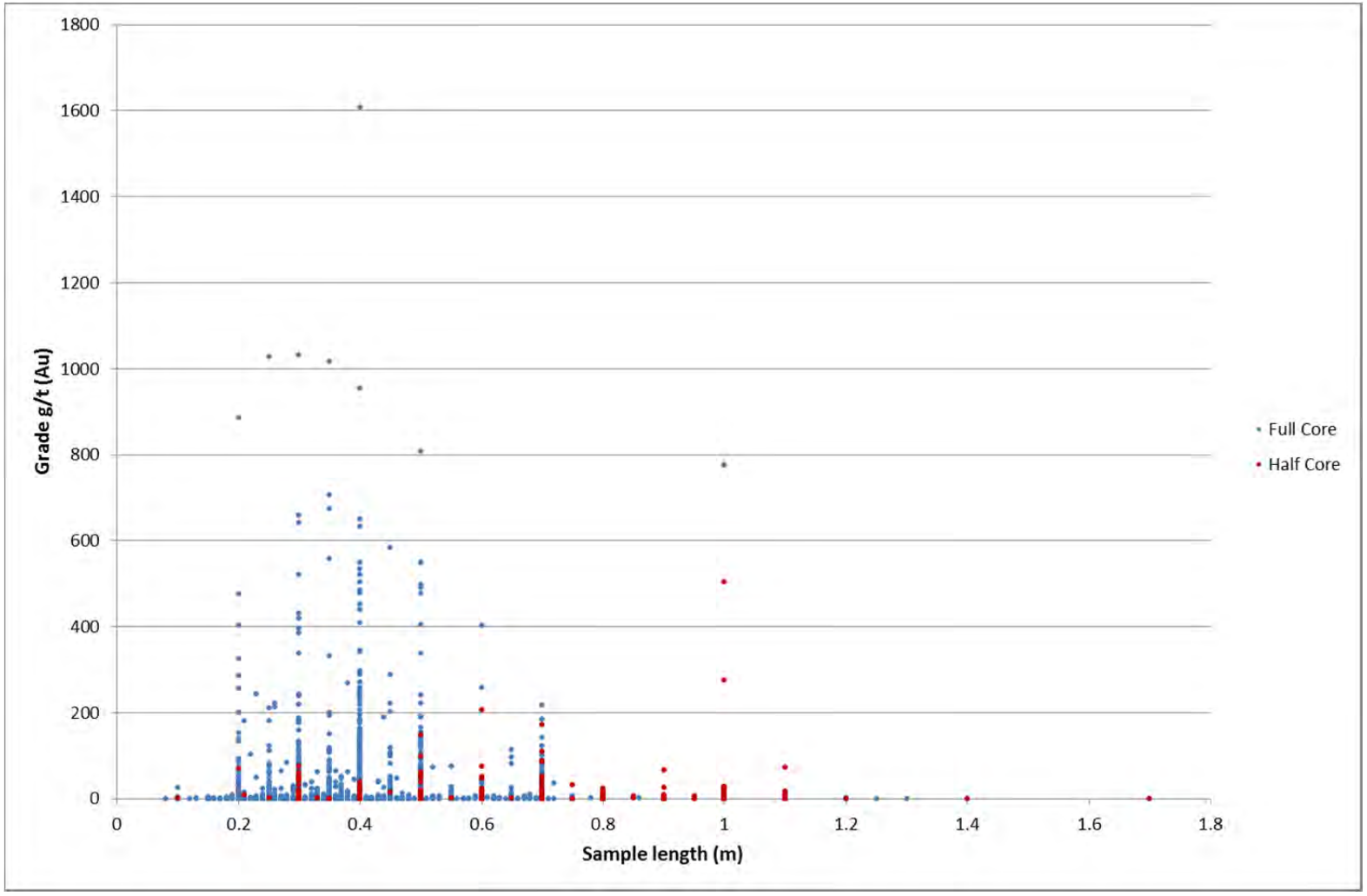


Figure 8-11 Scatter plot sample length versus average grade for full core and half core data.

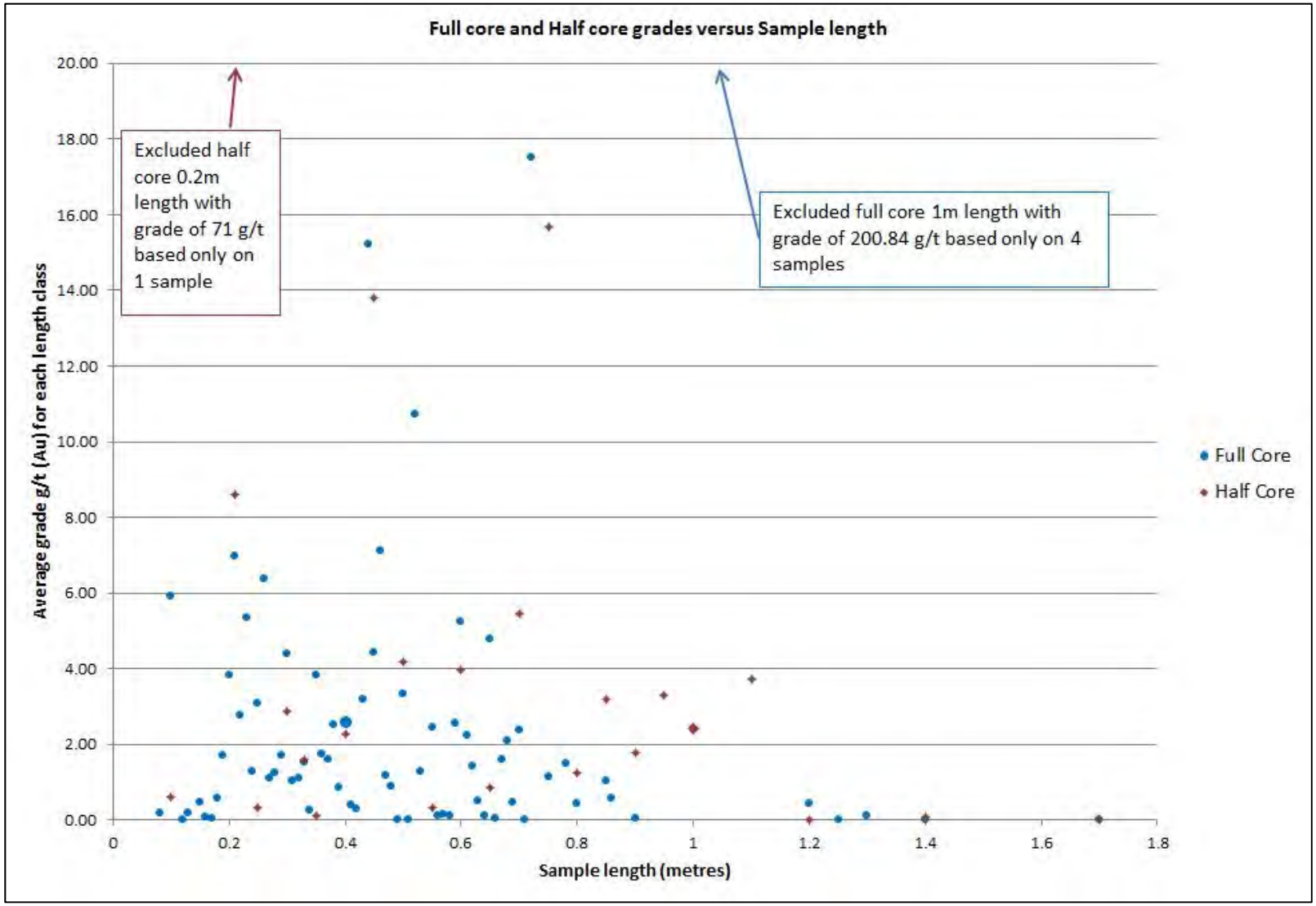


Figure 8-12 Scatter plot full core and half core sample length versus count of sample with visible gold.

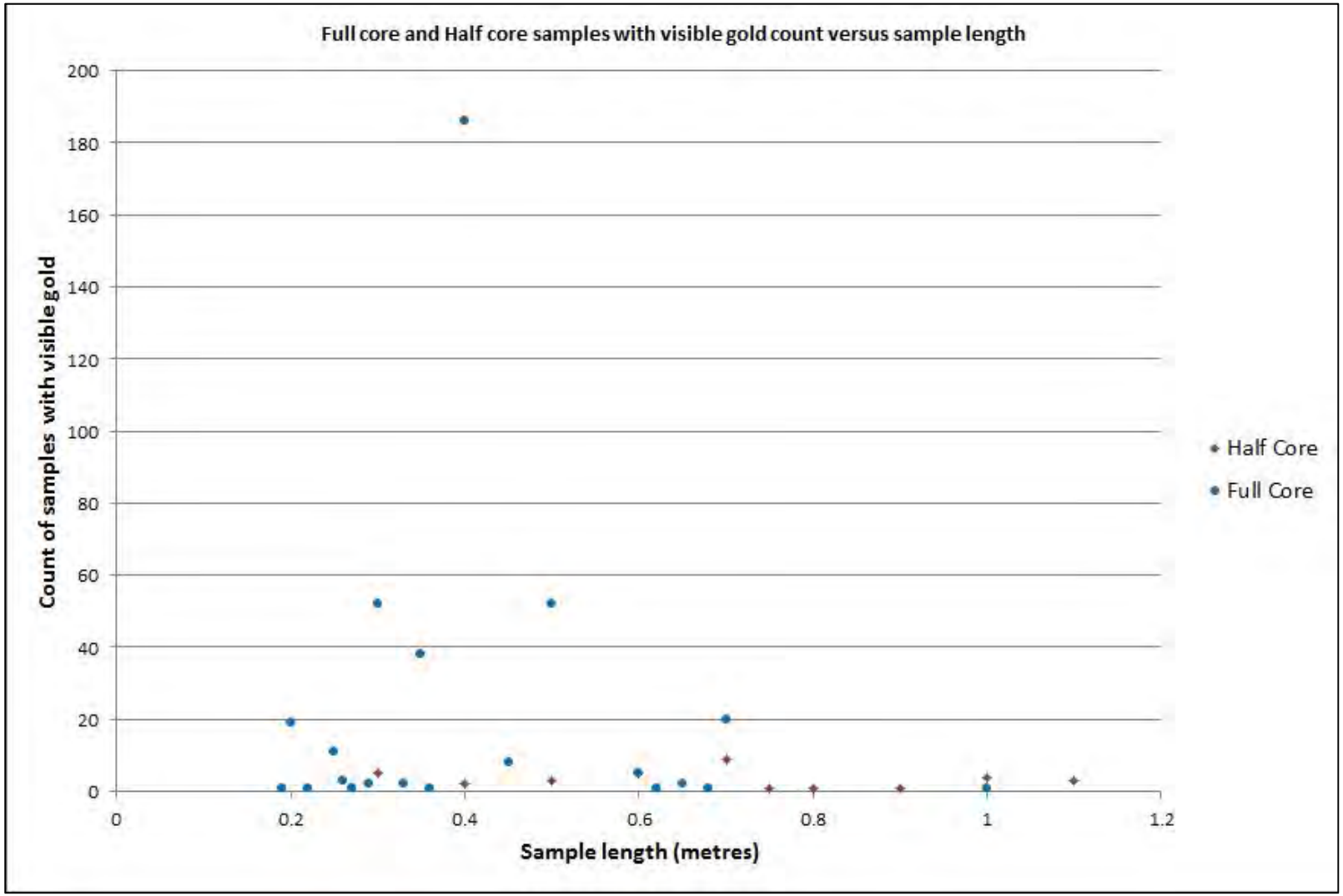


Figure 8-13 Scatter plot Full core and Half core sample length versus sample count.

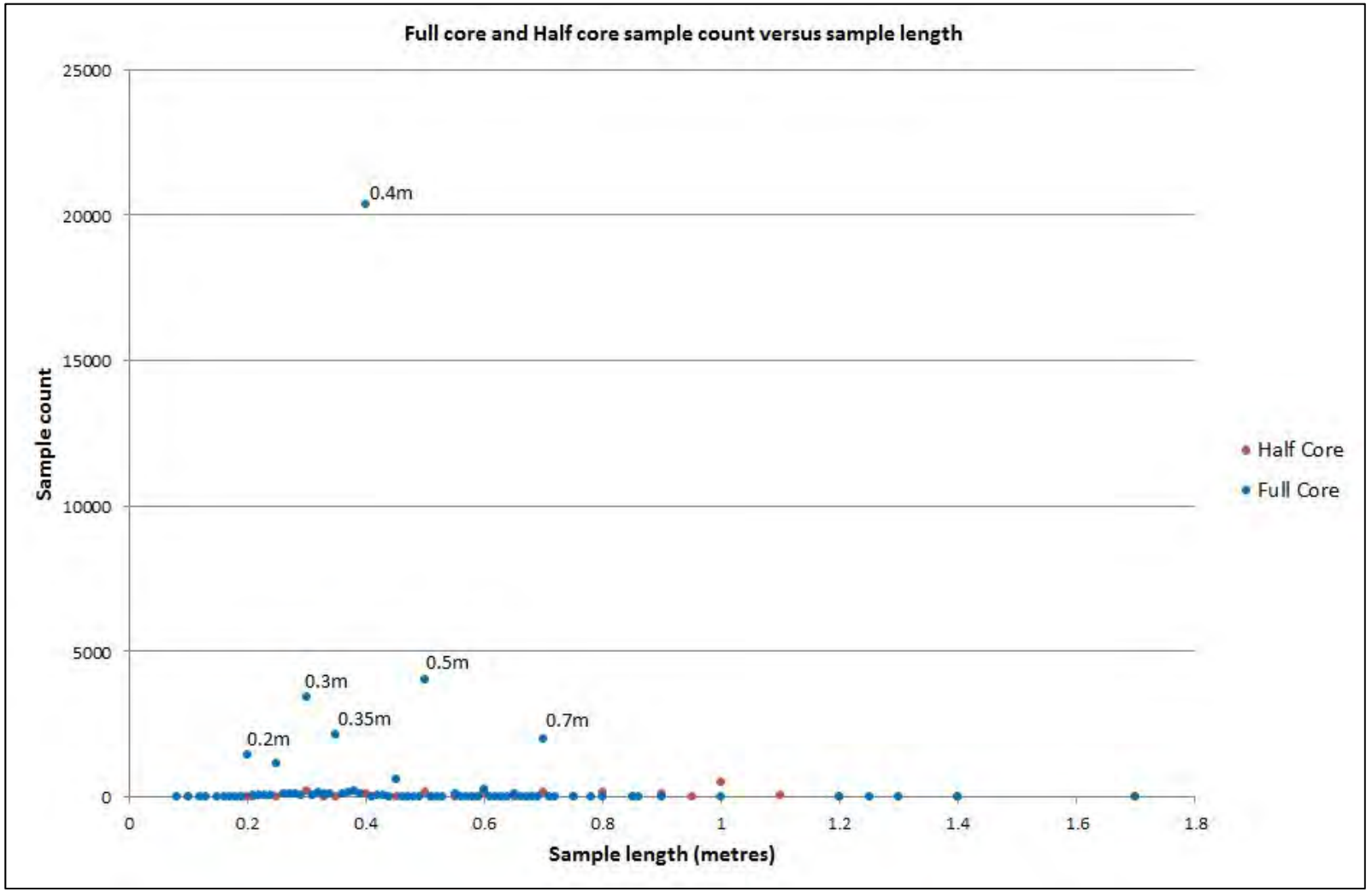


Figure 8-14 Histogram of sample length (m) for Britannia Mako assayed intervals

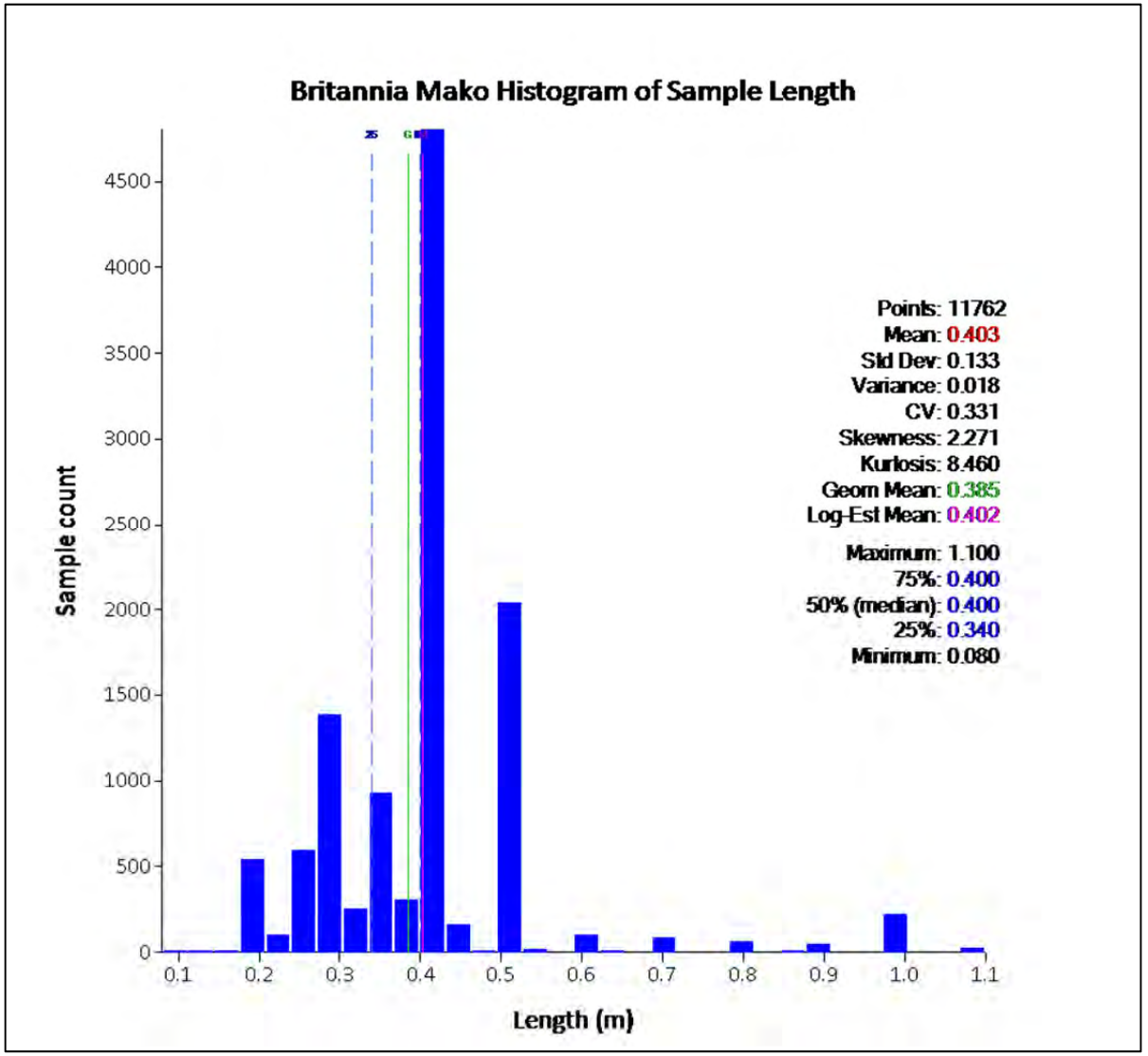


Figure 8-15 Histogram of sample length (m) for Britannia Basking fault zone assayed intervals

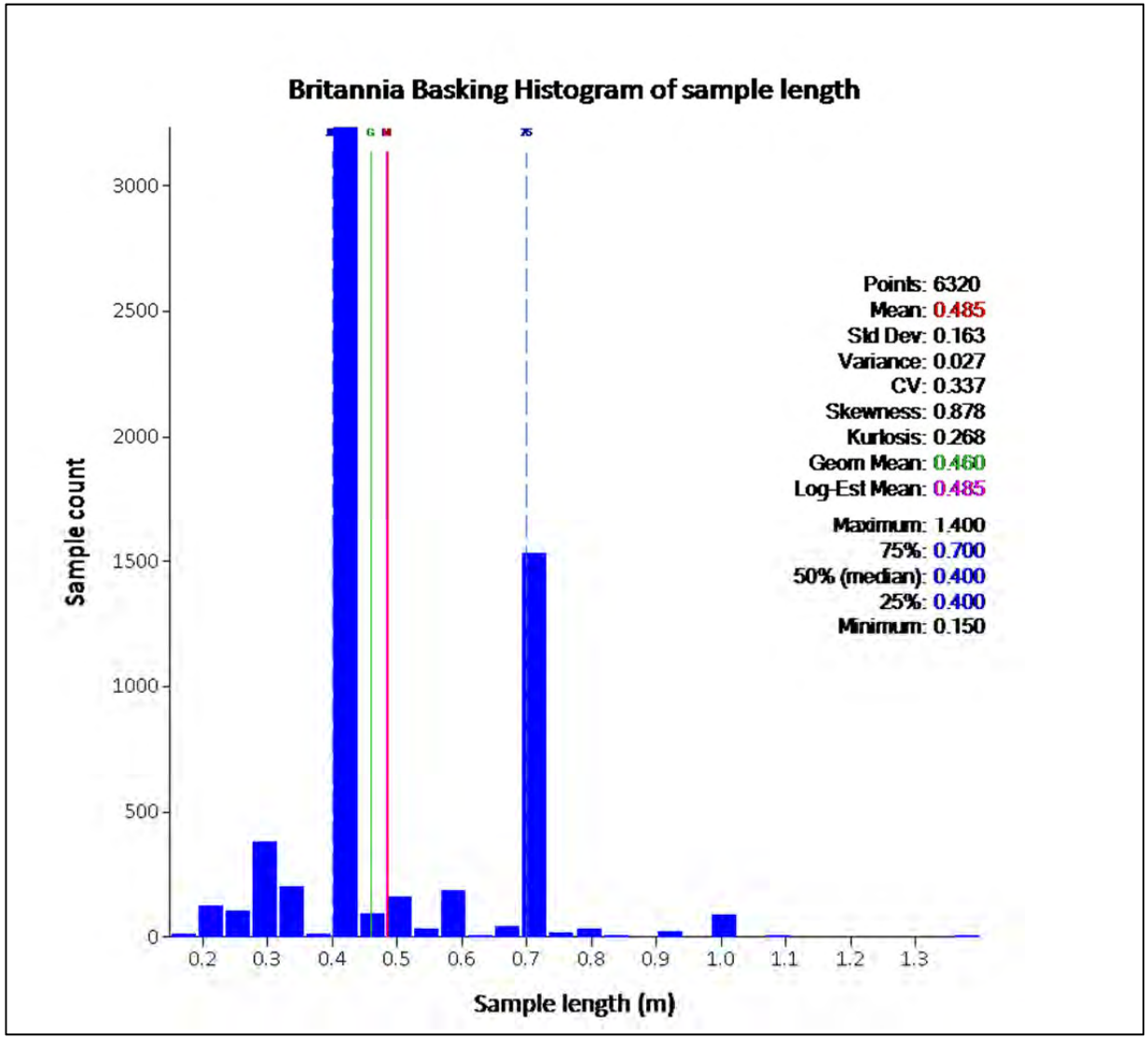


Figure 8-16 Histogram of sample length (m) for Llanberris Mako assayed intervals

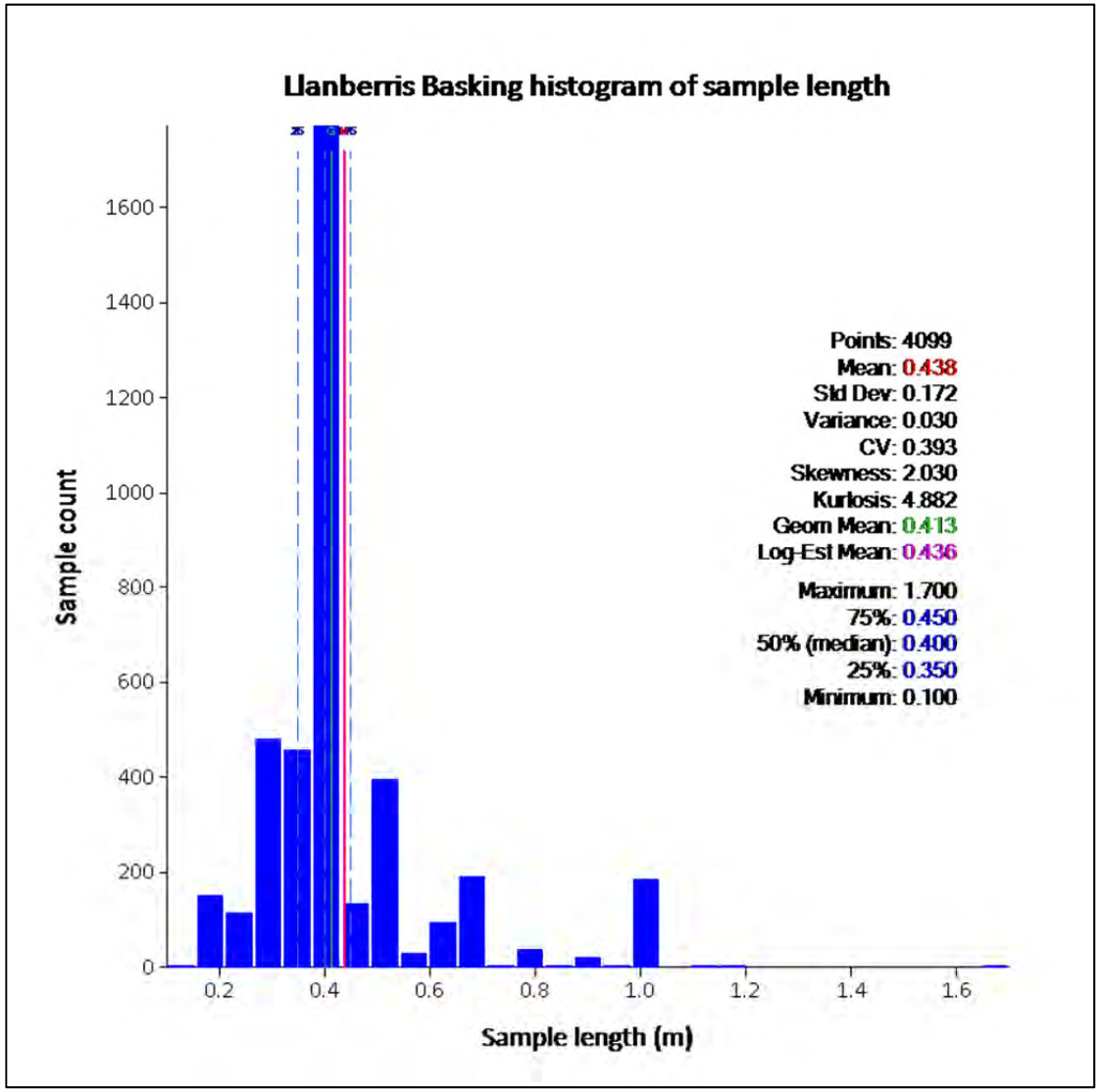


Figure 8-17 Histogram of sample length (m) for Llanberris Basking assayed intervals

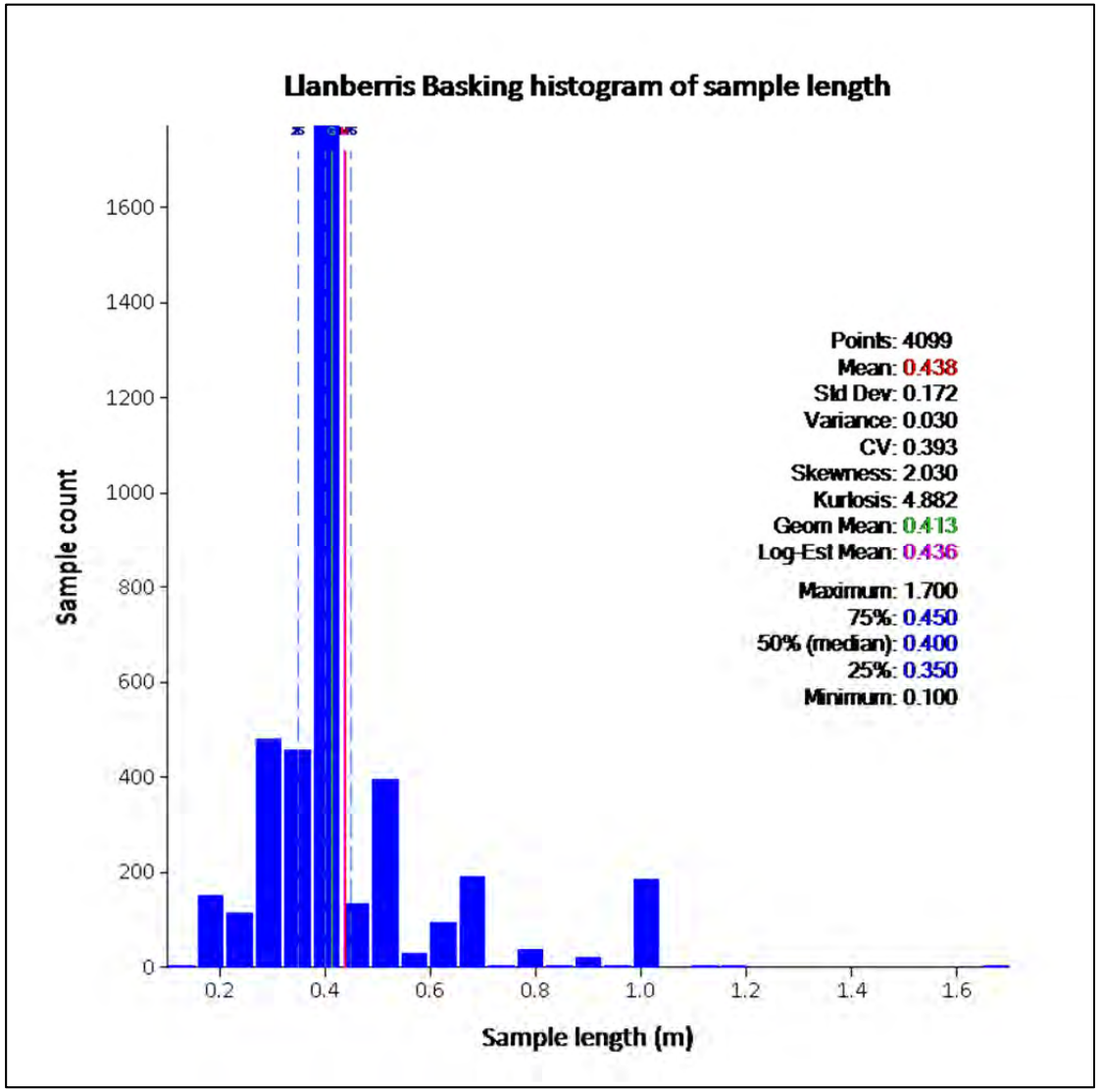


Figure 8-18 Histogram of sample length (m) for Sovereign Gummy assayed intervals

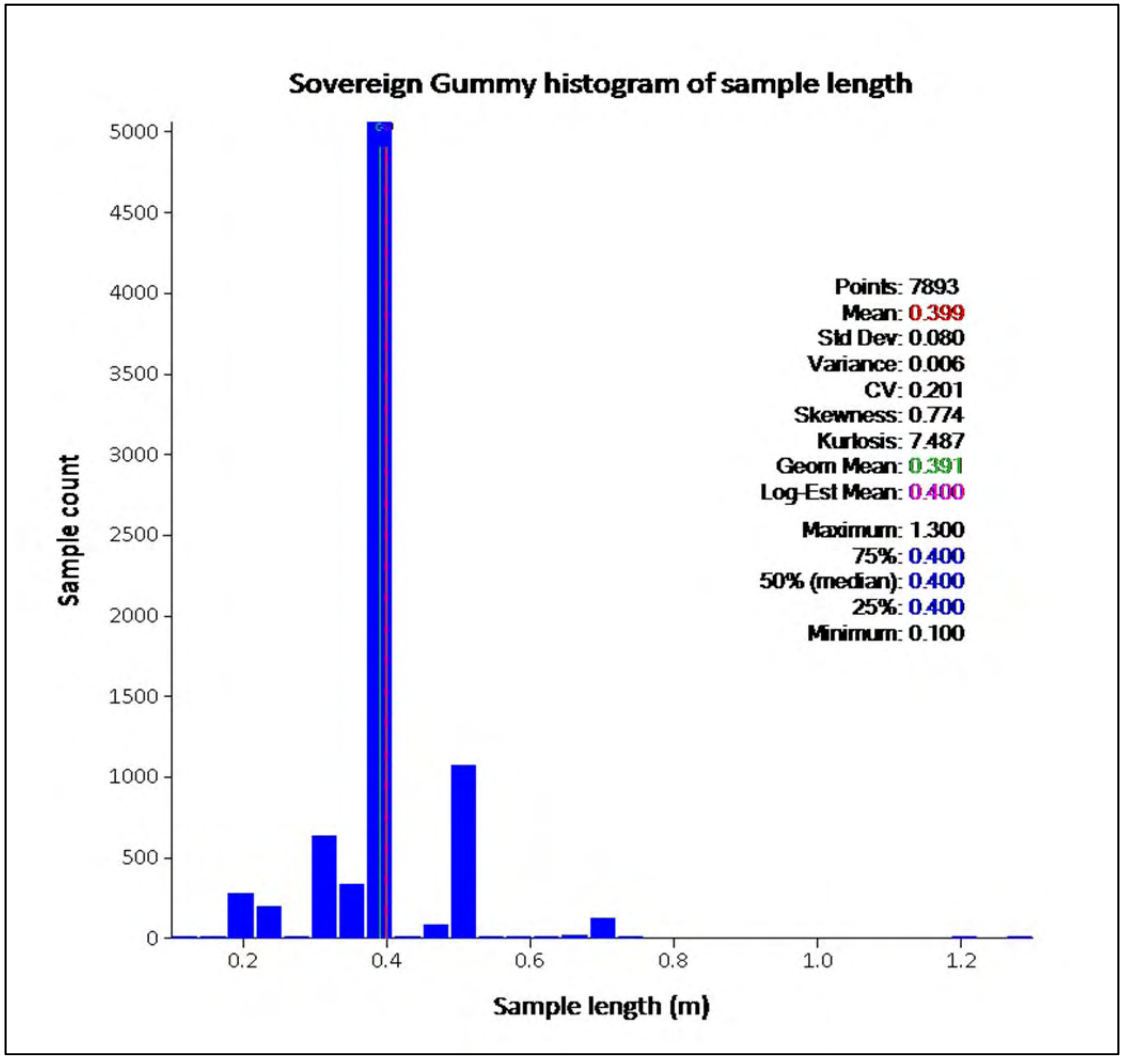
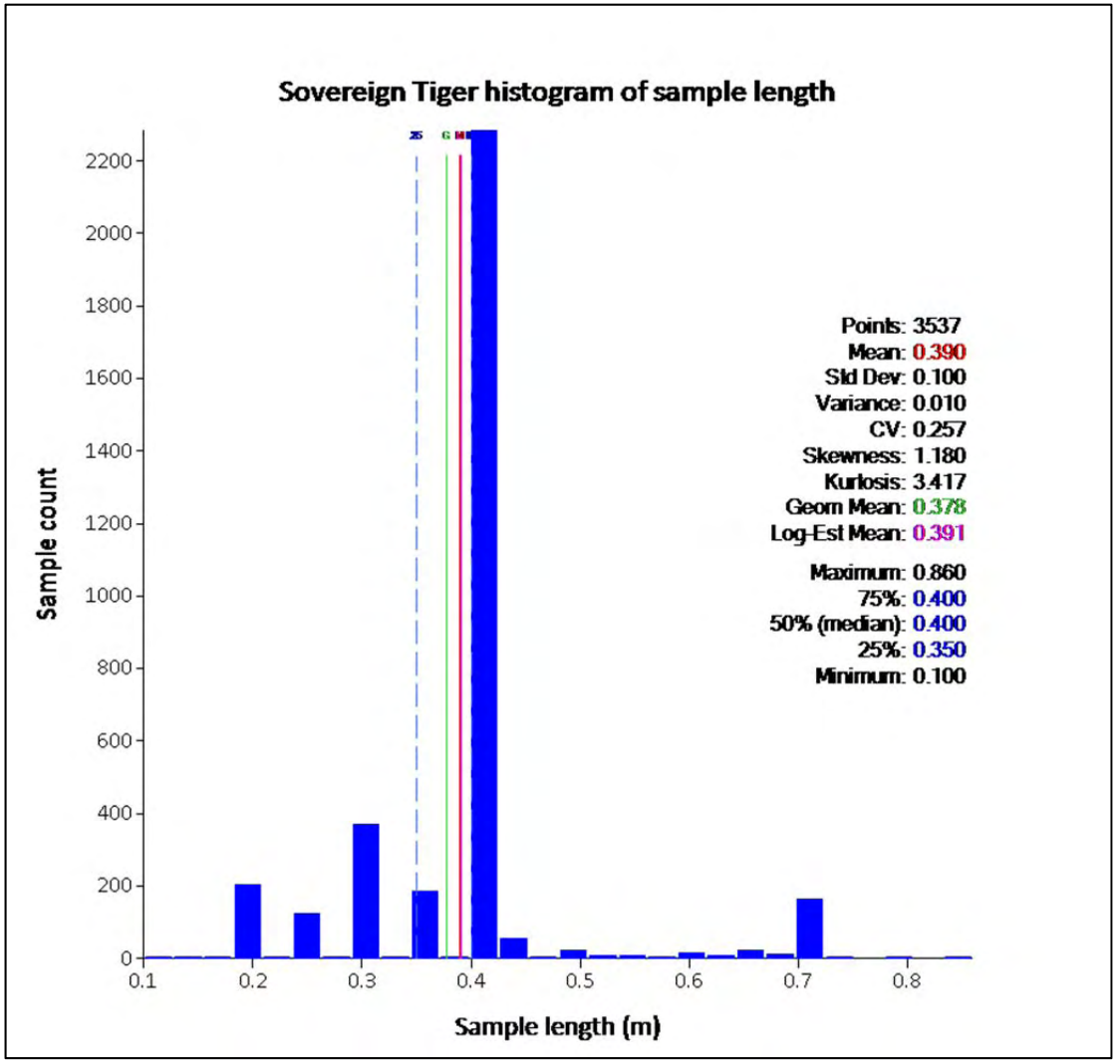


Figure 8-19 Histogram of sample length (m) for Sovereign Tiger assayed intervals



Sample compositing

Sample compositing is performed to obtain samples of equal length. Samples with different lengths or volumes may result in grade bias. The selection of an appropriate composite length needs to take into consideration the sample lengths present and should honour geological or domain boundaries. The greater the composite length chosen, the greater the extent of grade smoothing that occurs.

There are six different methods of compositing available in Vulcan. CGT has selected the “run-length” method. This method produces composites of equal length (except for end of hole, geological and triangulation boundaries). “Short” composites (those less than 0.2 m length) were merged with the preceding composite where possible.

Sample composite lengths were chosen based on the raw sample lengths informing the composites. Lodes which include a combination of half core (1 m lengths) and full core (0.7, 0.5 and 0.4 m lengths) have been composited to 1 m lengths. Lodes which were sampled entirely with full core sampling have been composited to largest significant length interval. The distribution of composite lengths for each of the lodes estimated is outlined in Figure 8-25 to Figure 8-26

Composites are assigned unique codes in the “bound” field of the composite file for each of the domains modelled (Figure 8-20). Where domains overlap, the "priority" function in Vulcan is used to ensure that composites are allocated to the preferred domain. Priorities are assigned based on interpretation of the structural setting of each domain. West-dipping faults are considered to be the most dominant structures, so are assigned the highest priority. East-dipping fault zones have been observed through mining to be truncated by west-dipping faults and are assigned a lower priority than the west-dipping faults. Stockwork zones adjacent to west- and east-dipping fault zones are assigned the lowest priority.

Compositing resulted in 4,916 composites based on 11,192 samples within the domain wireframes. The drill hole composites were created in Vulcan software, based on an ISIS database extracted from the CGT acQuire database. Only the drill hole paths and the domain wireframes were used.

Figure 8-20 Comparison of drill holes passing through the "fhg1" domain (left) and the resultant composites coded "fhg1" in the composite file (right) in the Sovereign Gummy compartment (oblique view, not to scale)

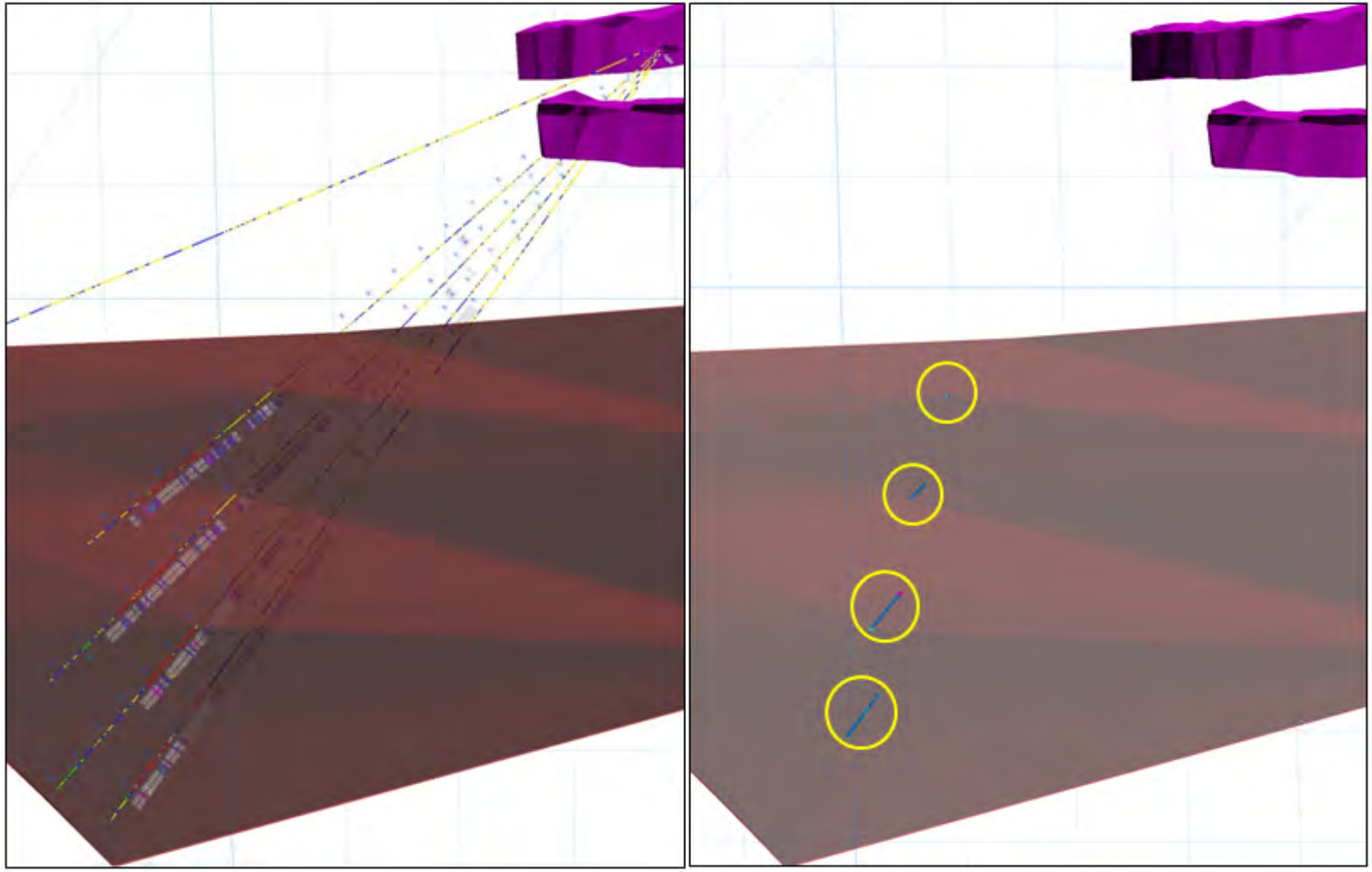


Figure 8-21 Histogram of composite sample length (m) for Britannia Mako assayed intervals

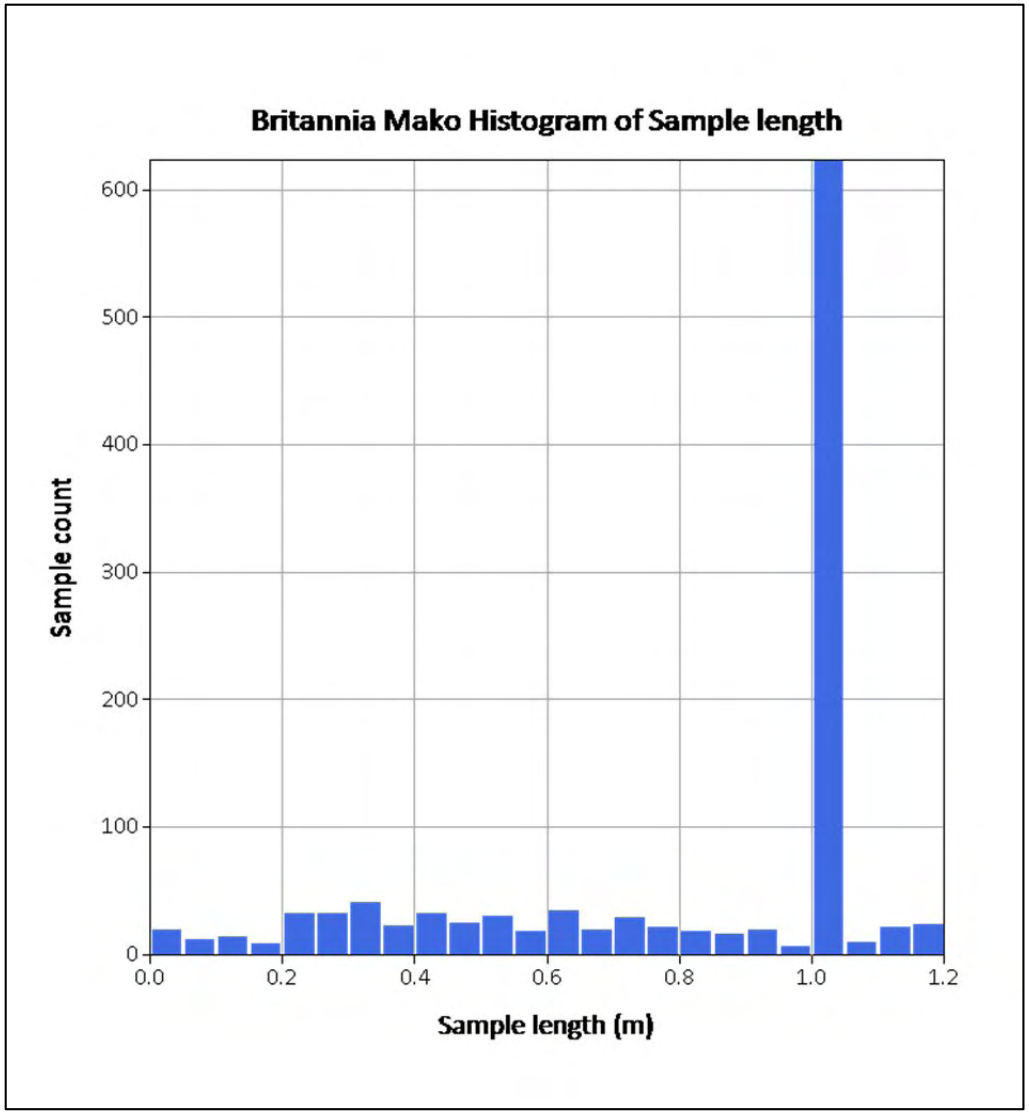


Figure 8-22 Histogram of composite sample length (m) for Britannia Basking assayed intervals

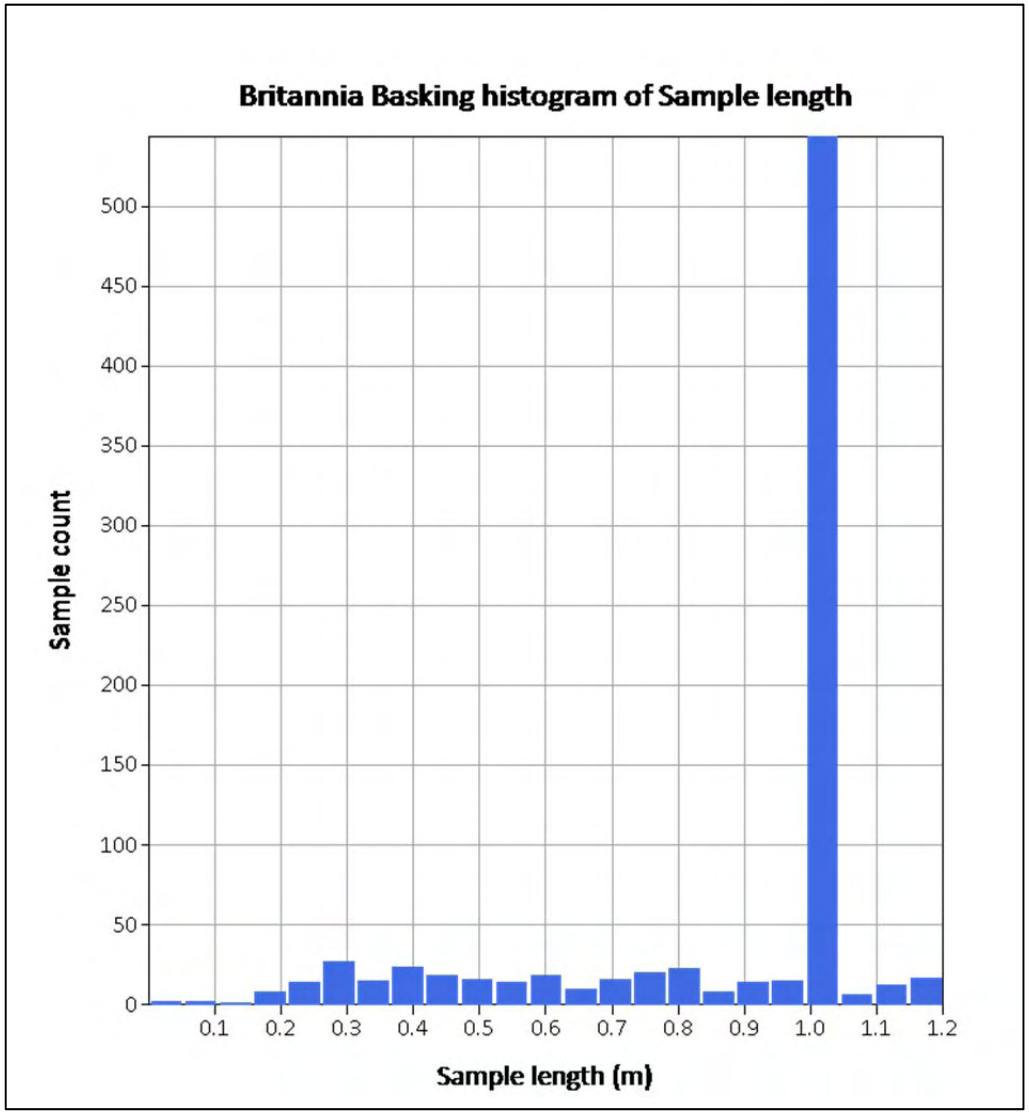


Figure 8-23 Histogram of composite sample length (m) for Llanberris Mako assayed intervals

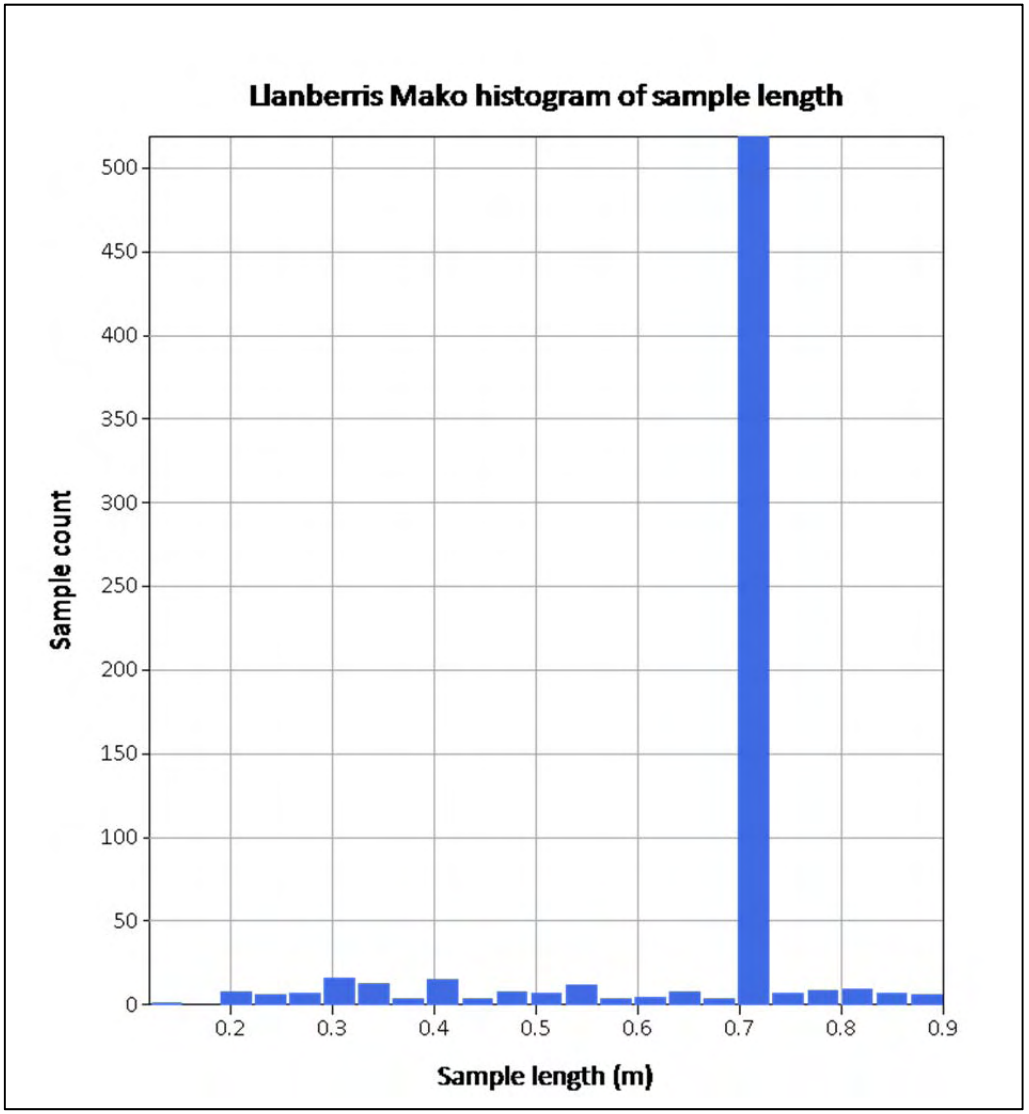


Figure 8-24 Histogram of composite sample length (m) for Llanberris Basking assayed intervals

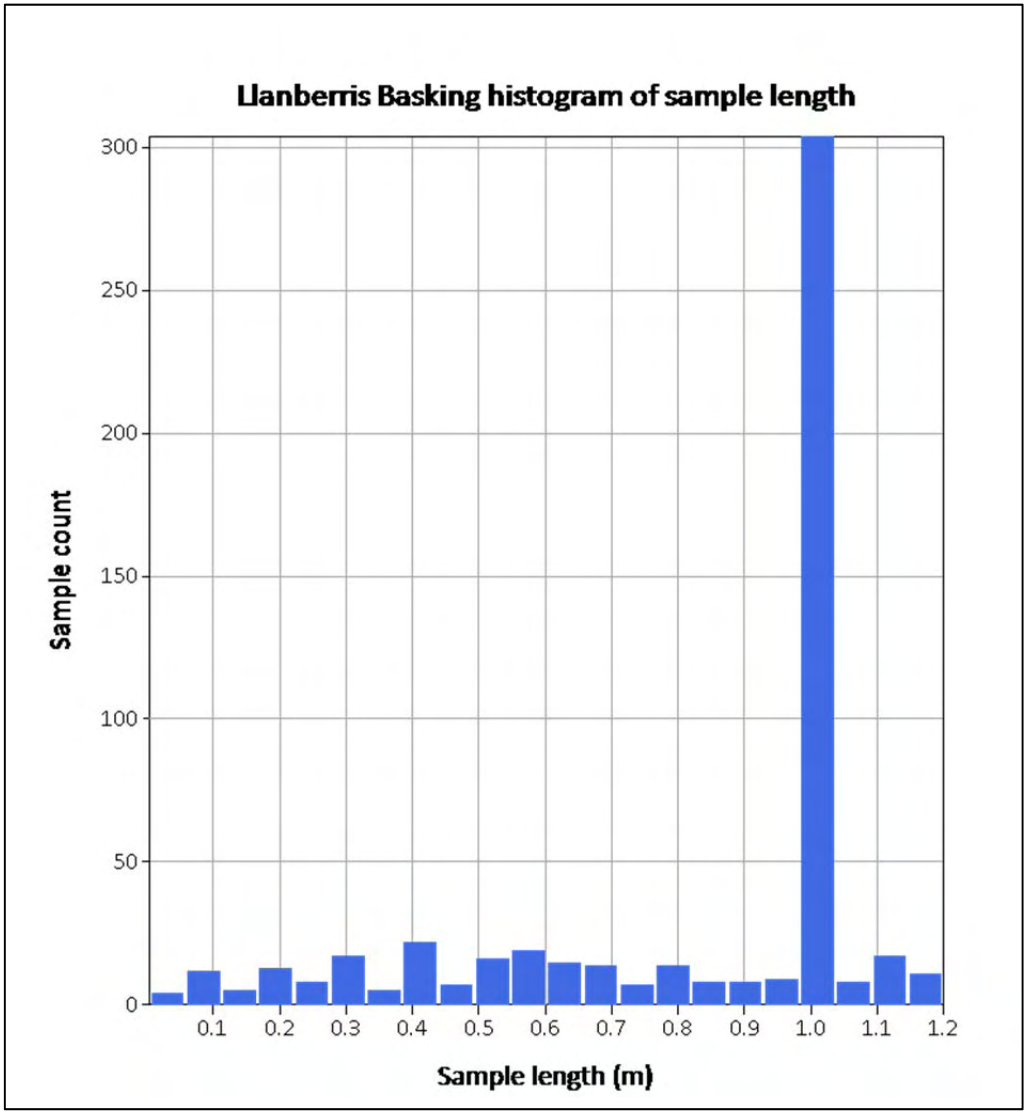


Figure 8-25 Histogram of composite sample length (m) for Sovereign Gummy assayed intervals

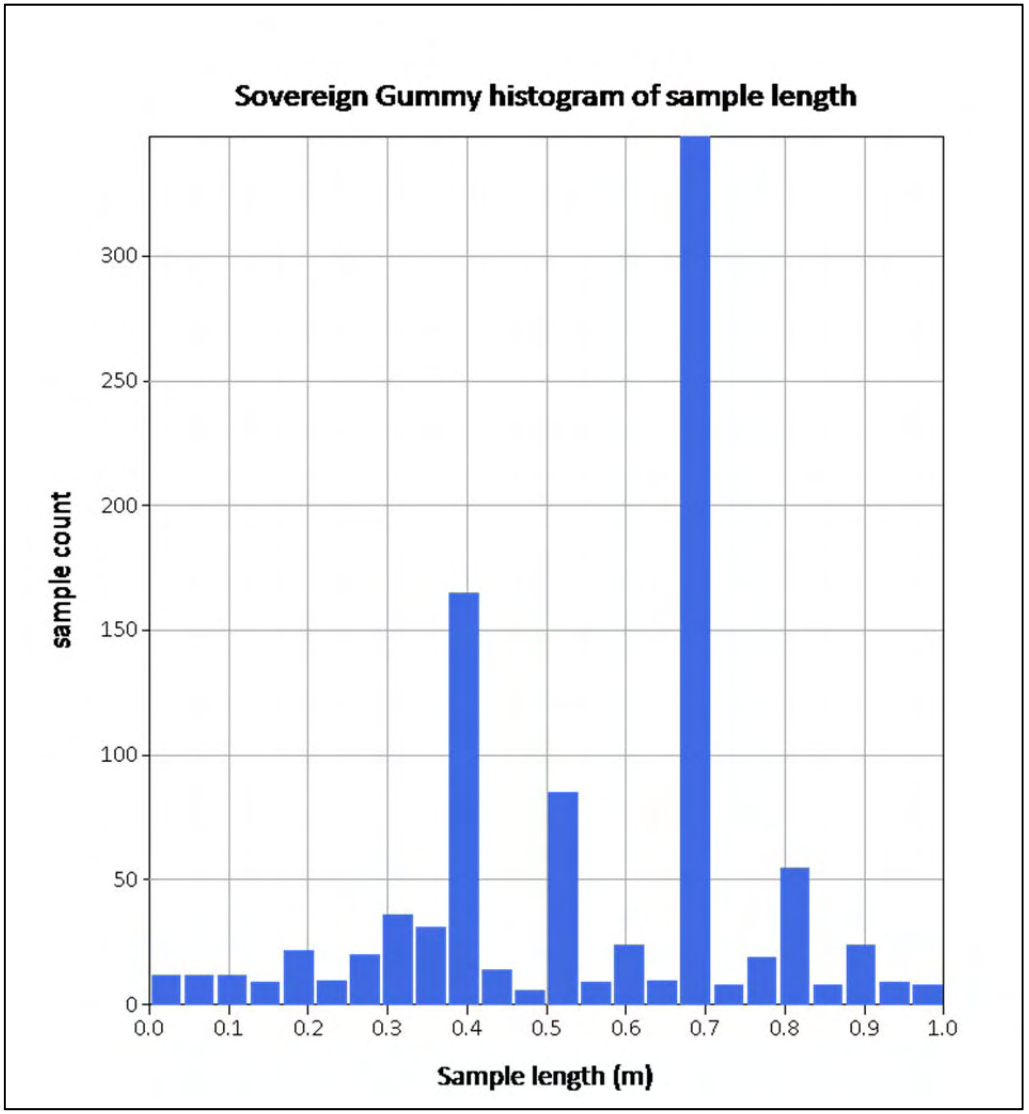
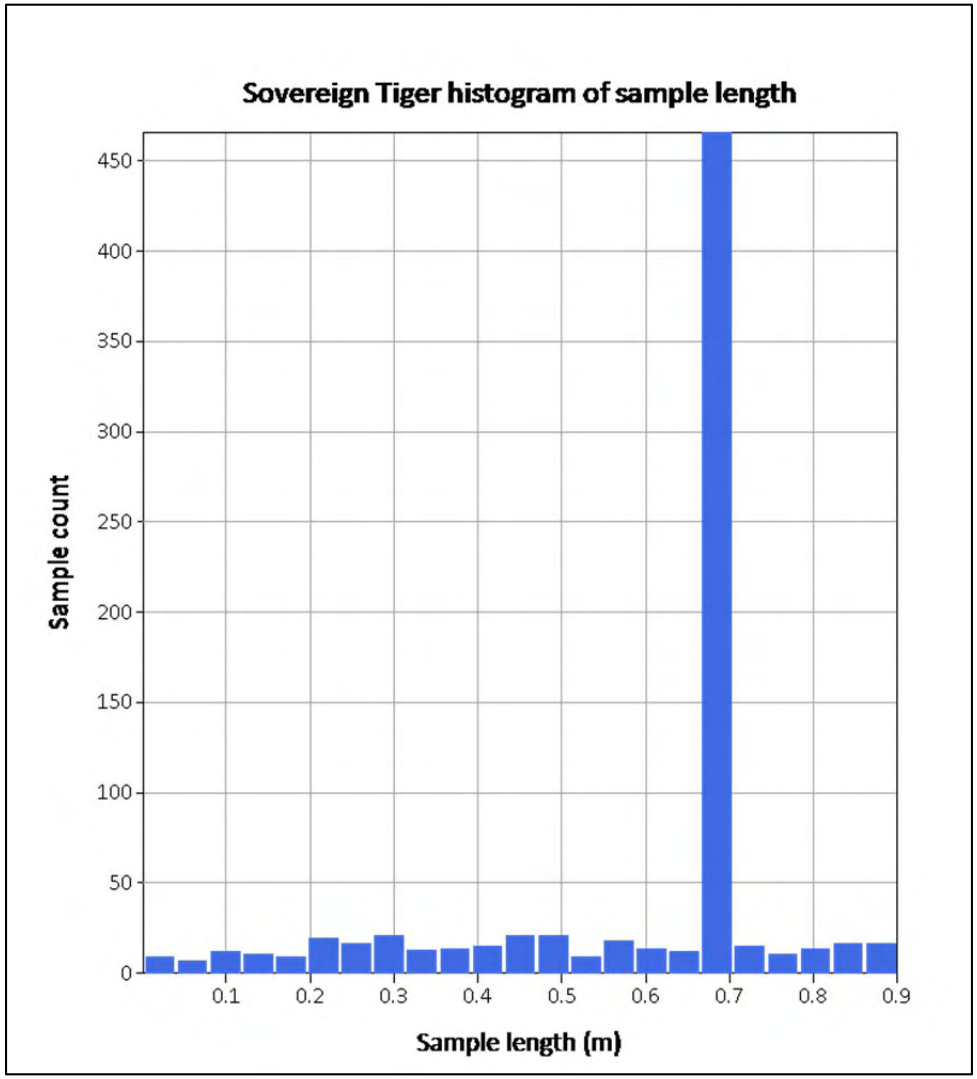


Figure 8-26 Histogram of composite sample length (m) for Sovereign Tiger assayed intervals



Statistical analysis

The effect of compositing within the domain wireframes changes statistical measures, the coefficient of variance (CV) for gold is reduced from 5.8567 to 3.97.2. Summary statistics for the composited samples are presented in Table 8.15.

Table 8.15 Summary statistics for composite samples (not declustered)

Sample type and lode	No. collars	No. composites	Gold grade (g/t Au)			
			Mean	Min	Max	CV
Full core						
Britannia Mako	103	1006	7.80	0.010	445.96	3.58
Britannia Basking	88	775	6.70	0.010	296.56	3.21
Llanberris Mako	47	680	5.91	0.010	451.24	4.55
Llanberris Basking	68	406	8.31	0.010	774.75	5.75
Sovereign Tiger	56	783	7.07	0.010	452.74	3.99
Sovereign Gummy	91	956	13.27	0.010	640.95	3.85
Total	485					
Half core						
Britannia Mako	16	106	12.87	0.010	275.6	3.02
Britannia Basking	9	67	4.37	0.028	66.09	2.34
Llanberris Basking	13	137	7.54	0.010	467.45	5.4
Total	41					
Grand Total	**485**	**4,916**	**8.37**	**0.010**	**774.75**	**4.20**

Note: The sum of the collars in the table above does not match the grand total; this is because two holes in the estimate intersected mineralisation in both the Sovereign Gummy and Tiger lodes as well as earlier CGT holes having two or three types of samples thus getting counted multiple times.

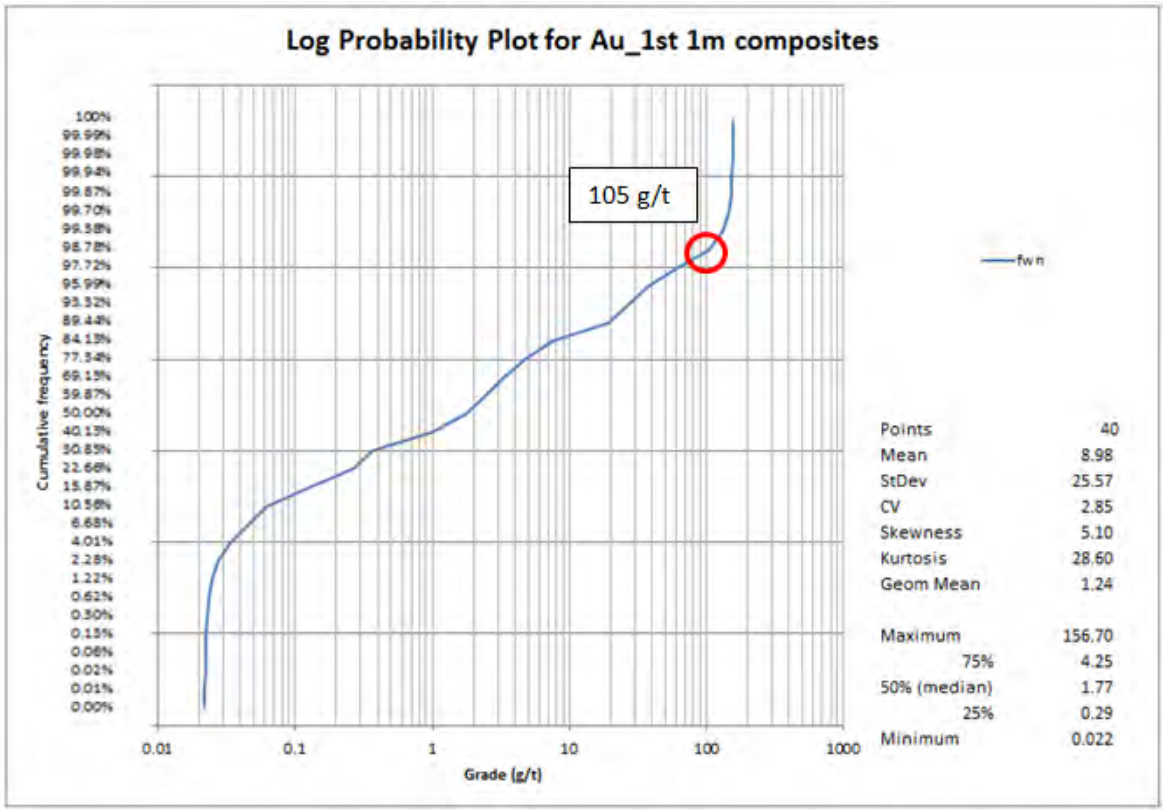
Table 8.16 Summary statistics for global composite samples (not declustered) within domains.

	Composite Samples	Sample length (m)
Samples	4916	4916
Minimum	0.000	06
Maximum	774.75	1.20
Mean	8.37	0.72
Standard Deviation	35.17	0.26
CV	4.20	0.36
Variance	1,236.98	0.07
Skewness	10.65	-0.50
10%	0.04	0.33
20%	0.15	0.48
30%	0.30	0.70
40%	0.56	0.70
50%	0.95	0.70
60%	1.62	0.78
70%	2.95	0.80
80%	5.55	1.00
90%	14.74	1.00
95%	32.18	1.00
97.50%	65.68	1.00
99%	129.08	1.15

Top-cut analysis of composite data

Top-cutting is applied to all domains estimated at Ballarat. This is done to prevent extreme grades resulting in over-estimation of the resource. All composite samples were considered when analysing domain data-sets for top-cut selection. Log probability plots were generated for all domains, with top-cuts selected by identifying inflection points in the grade distribution. Figure 8-27 provides an example of top-cut selection for the “footwall north” domain in the Llanberris Mako lode. The top-cut of 105 g/t Au was selected, based on the inflection in the grade distribution.

Figure 8-27 Example of top-cut selection from a log probability plot of grade distribution in the Llanberris Mako "footwall north" domain



The top-cut grades applied to each of the domains in each of the lodes estimate vary dependant on the distribution of composite grades within each domain. Where multiple inflection points on the grade distribution curve are observed, the selection of top-cut grades is influenced by the level of confidence held in the domain estimated. For example, a zone of stockwork veining between two faults would have its top-cut selected more conservatively than a west-dipping fault zone. This is because major west-dipping fault zones exhibit greater geological and grade consistency than stockwork zones where grade variability is exacerbated by erratic quartz vein distribution. A summary of the top-cuts selected for the domains in each of the lodes estimated is given in Table 8.17.

Table 8.17 Summary of top-cuts used for each of the domains estimated

Domain	Mineralisation style	Top-cut (g/t) Au
Britannia Mako Lode		
Anticline east dipper 1	East dipper	11
Anticline east dipper 2	East dipper	65
Anticline east dipper 3	East dipper	67
Anticline east dipper 4	East dipper	16
Anticline stockwork	Stockwork	22
Bengal inter fault zone	West dipping fault	15
Bengal fault 1	West dipping fault	15
Bengal fault 2	West dipping fault	24
Bengal east dippers	East dipper	49
Bengal north	West dipping fault	3
Bengal vertical spurs	Vertical spurs	69

Central east dipper 1	East dipper	24
Central east dipper 2	East dipper	17
Central east dipper 3	East dipper	30
Central east dipper 4	East dipper	31
Central east dipper 6	East dipper	6
Mid-east dipper 5	East dipper	84
Mako Anticline HG	West dipping fault	207
Mako lower fault zone	West dipping fault	71
Mako south fault zone	West dipping fault	22
Mako upper fault zone	West dipping fault	16
North east dipper 1	East dipper	8
North east dipper 2	East dipper	19
North east dipper 4	East dipper	8
North east dipper 5	East dipper	96
North east dipper 6	East dipper	38
Siberian fault zone	West dipping fault	80
Siberian Anticline HG	West dipping fault	24
Thresher fault zone	West dipping fault	1
Britannia Basking Lode		
Basking hangingwall main	West Dipper	105
Basking footwall 1 main	West Dipper	101
Hammerhead hangingwall 1	West Dipper	28
Upper east dipper 1	East dipper	9
Mid-east dipper north	East dipper	22
Basking hangingwall splay north	West Dipper	1
Basking hangingwall splay central	West Dipper	28
Basking hangingwall splay south	West Dipper	7
Hammerhead footwall	West Dipper	9
Mid-east dipper far south 1	East dipper	14
Mid-east dipper far south 2	East dipper	47
Basking hangingwall south	West Dipper	11
Basking footwall 2 main	West Dipper	17
Basking footwall 2 south	West Dipper	46
Hammerhead hangingwall 2	West Dipper	13
Hammerhead footwall 2	West Dipper	102
Gummy fault	West Dipper	6
Llanberris Mako Lode		
Siberian block 1 upper	Steep west dipper	8
Siberian block 1 lower	Steep west dipper	12
Siberian block 2 upper	Steep west dipper	1
Siberian block 2 lower	Steep west dipper	11
Siberian block 3 upper	Steep west dipper	80
Siberian block 3 lower	Steep west dipper	20
Siberian block 4 upper	Steep west dipper	2
Siberian block 4 lower	Steep west dipper	45

Siberian block 5 upper	Steep west dipper	1
Siberian block 5 lower	Steep west dipper	114
Mako 1 block 5 upper	Shallow west dipper	21
Mako 1 block 5 lower	Shallow west dipper	36
Mako 4 block 5 lower	Shallow west dipper	52
Anticline stockwork block 5 lower	Stockwork	42
Llanberris Basking Lode		
Basking fault down dip	West dipper	7
Basking fault south	West dipper	114
Basking main quartz	West dipper	5
Basking shallow HG zone A	West dipper	28
Basking shallow HG zone B	West dipper	50
Basking steep HG zone	West dipper	15
Basking vertical spurs	Vertical spurs	12
East dipper 1	East dipper	23
East dipper 2	East dipper	5
East dipper 3	East dipper	7
East dipper 4	East dipper	10
East dipper 5	East dipper	4
Hammerhead footwall HG zone	West dipper	21
Hammerhead hangingwall HG zone A	West dipper	10
Hammerhead hangingwall HG zone B	West dipper	13
Hammerhead inner HG zone	West dipper	21
Hammerhead main quartz	West dipper	3
Sovereign Gummy Lode		
East dipper 2	East dipper	21
East dipper 3	East dipper	128
East dipper 4	East dipper	50
Gummy footwall 1	West dipping fault	72
Gummy footwall 2 north	West dipping fault	77
Gummy hangingwall 1 main	West dipping fault	355
Lower stockwork	Stockwork	35
North east dipper	East dipper	5
South east dipper	East dipper	17
Gummy footwall 2 south	Stockwork	4
South stockwork	Stockwork	66
Gummy footwall 3	West dipping fault	201
Gummy footwall 2 centre	West dipping fault	34
Upper stockwork	Stockwork	4
Top HG Spurs	Stockwork	57
Gummy mid main fault	West dipping fault	115
East dipper 6	East dipper	5
Gummy footwall 1 south	West dipping fault	1
Gummy footwall 4 south	West dipping fault	80
South upper massive quartz	Stockwork	4
Gummy hangingwall 1 south	West dipping fault	66

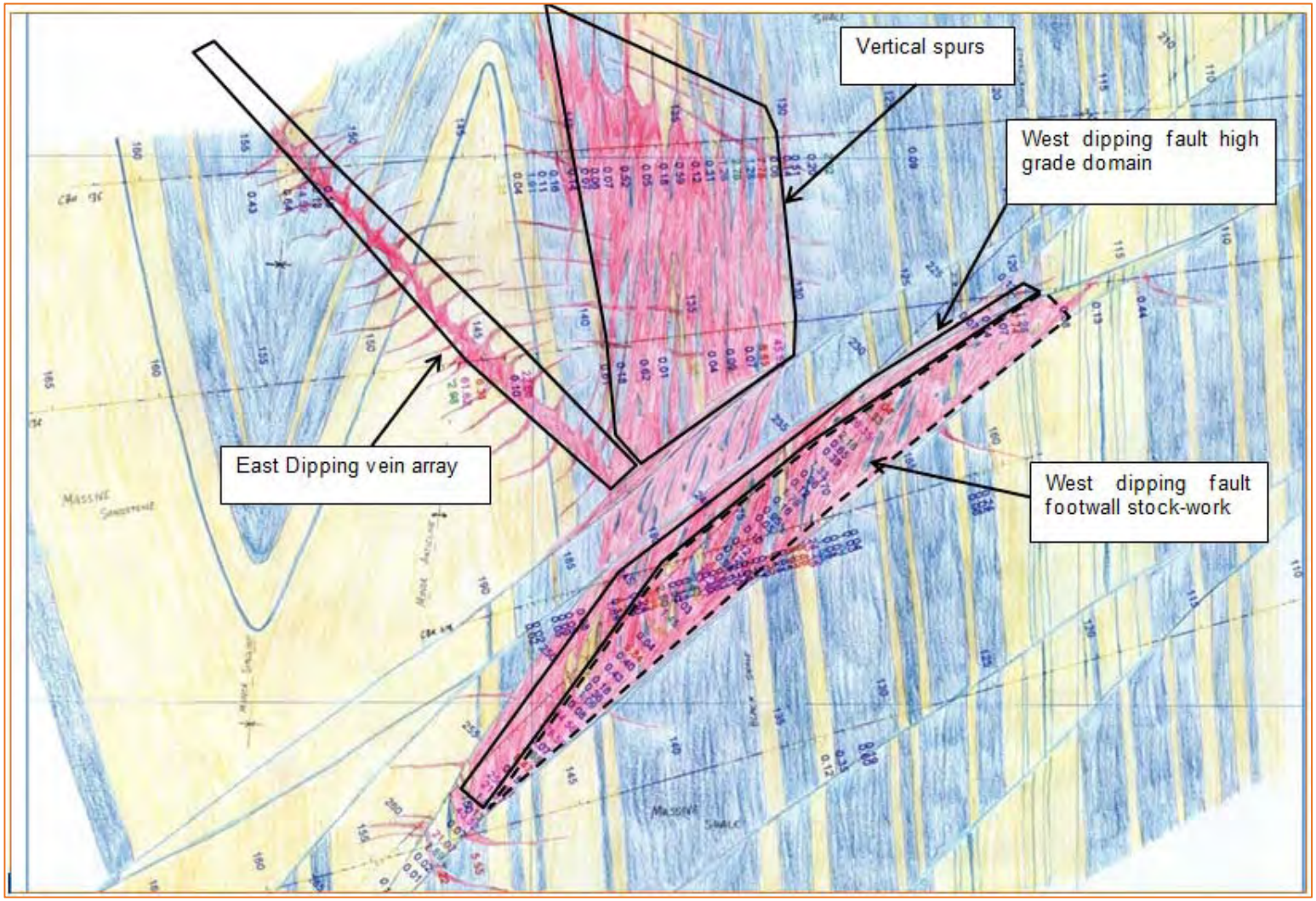
East dipper 7	East dipper	11
Sovereign Tiger Lode		
Main footwall fault	West dipping fault	180
Lower footwall Fault	West dipping fault	62
Main hangingwall fault	West dipping fault	60
lower hangingwall fault	West dipping fault	17
Upper flat west dipping fault	West dipping fault	3
Lower east dipping fault	East dipper	30
stockwork zone	Stockwork zone	7
Main peripheral stockwork zone	Stockwork zone	180

8.3.4 Domaining

Geological domains

Geological domaining is carried out based on the interpretations as discussed in Sections 5.3.3 and 8.3.2. Domains are constructed to delineate zones of quartz mineralisation with consistent geological and grade characteristics within specific structural horizons identified during interpretation. It is common for large volumes of quartz mineralisation to be associated with the major west- and east-dipping fault zones, with elevated gold grades more frequently occurring immediately adjacent the major structures. Figure 8-28 shows west- and east-dipping fault zones domained separately from a stockwork zone (see vertical spurs in Figure 8-28). Each of the domains identified are interpreted to have reasonably consistent quartz textures and grade characteristics.

Figure 8-28 Example of mineralisation domains based on detailed geological interpretation in the Llanberris Basking fault zone (38175 mN) – not to scale



Domains are modelled in order of structural importance. Major west-dipping fault zones are modelled first, these are considered to be the most dominant and continuous mineralised structures in the goldfield. They have been observed during mining to constrain the extents of secondary structures such as east-dipping vein arrays and flat-makes. East-dipping vein arrays are modelled second, once the major west-dipping zones have been delineated and are limited by the position of west-dipping faults. Stockwork and vertical vein zones are modelled third. In the example given in Figure 8-28, the west-dipping fault zone is modelled first, followed by the east-dipping vein array, with the stockwork zone modelled last.

Wireframes of geological domains are based on geological interpretations, however are often refined by the distribution of gold grades. Figure 8-28 gives an example of geological domains based on geological interpretation. The mineralisation associated with the Basking fault has been separated into two domains based on gold grade distribution. This reflects drill hole assays which suggest there is a narrow zone of high-grade (frequently above 10 g/t Au) mineralisation on the hangingwall of the fault, with low to moderate (predominantly below 5 g/t Au) grades on the footwall.

It is common for zones of weak stockwork veining (less than 20% quartz veins relative to sediments) to be associated with the major structures modelled. These zones often contain sporadic low to moderate grades. These zones have been modelled and estimated to better assess the impact of dilution from these areas, but are not reported as part of the resource. A summary of the domains estimated for each of the lodes is given in Table 8.18 to Table 8.23.

Table 8.18 Summary of domains estimated for the Britannia Mako lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au)		
				Min	Max	Average
Bengal intermediate fault zone	West dipping fault	13	19	0.08	25.11	6.09
Mako upper fault zone	West dipping fault	8	17	0.1	63.5	9.26
Siberian fault zone	West dipping fault	6	13	0.26	44.44	9.21
Mako south fault zone	West dipping fault	30	67	0.045	75.22	9.18
Thresher fault zone	West dipping fault	5	12	0.01	1.62	0.4
Mako Anticline HG	West dipping fault	88	263	0.01	100.16	5.79
Bengal east dippers	East dipper	5	27	0.17	57.25	10.86
Central east dipper 1	East dipper	4	13	0.52	30.95	9.09
Central east dipper 2	East dipper	2	11	0.02	75.35	15.64
Central east dipper 3	East dipper	3	12	0.03	43.89	8.2
Central east dipper 4	East dipper	3	18	0.04	351.47	28.72
Central east dipper 6	East dipper	2	5	0.01	135.65	28.64
Bengal vertical spurs	Vertical spurs	2	7	2.07	370.92	67.34
Mid-east dipper 5	East dipper	2	8	2.19	17.19	8.85
Anticline East dipper 1	East dipper	6	21	0.01	46.63	6.19
Anticline East dipper 2	East dipper	13	47	0.04	151.23	9.37
Anticline East dipper 3	East dipper	13	44	0.08	87.7	10.5
Anticline East dipper 4	East dipper	4	14	0.25	23.26	6.05
Anticline stockwork	Stockwork	38	313	0.01	98.19	1.835
Siberian Anticline HG	West dipping fault	35	51	0.01	104.05	18.27
Bengal fault 1	West dipping fault	7	12	0.02	126.46	16.45
Bengal fault 2	West dipping fault	7	13	0.16	28.27	7.09
Bengal north	West dipping fault	1	17	0.01	4.04	1
Mako lower fault zone	West dipping fault	29	50	0.01	275.6	12.97
North east dipper 1	East dipper	5	16	0.05	13.8	2.86
North east dipper 2	East dipper	3	22	0.01	35.31	7.73
North east dipper 4	East dipper	2	7	0.1	9.9	5.4
North east dipper 5	East dipper	5	12	0.25	445.96	49.54
North east dipper 6	East dipper	1	6	0.2	368.94	71.69

Table 8.19 Summary of domains estimated for the Britannia Basking lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au)		
				Min	Max	Average
Basking hangingwall main	West Dipper	32	87	0.005	255.62	11.91
Basking footwall 1 main	West Dipper	27	125	0.005	296.56	9.82
Hammerhead hangingwall 1	West Dipper	28	71	-	55.10	4.03
Upper east dipper 1	East dipper	5	24	0.005	19.06	3.00
Mid-east dipper north	East dipper	24	71	0.005	27.70	3.4
Basking hangingwall splay north	West Dipper	2	9	0.005	1.49	0.75
Basking hangingwall splay central	West Dipper	6	17	0.14	63.33	13.97
Basking hangingwall splay south	West Dipper	5	15	0.005	9.90	2.30
Hammerhead footwall	West Dipper	5	12	-	18.2	3.32
Mid-east dipper far south 1	East dipper	20	53	0.005	20.5	2.54
Mid-east dipper far south 2	East dipper	5	17	0.09	63.9	7.70
Basking hangingwall south	West Dipper	3	17	0.20	21.26	3.73
Basking footwall 2 main	West Dipper	5	22	0.005	25.00	6.82
Basking footwall 2 south	West Dipper	21	66	0.005	229.84	9.82
Hammerhead hangingwall 2	West Dipper	6	17	-	15.80	3.55
Hammerhead footwall 2	West Dipper	28	129	-	185.98	7.70
Gummy fault	West Dipper	19	74	0.001	9.70	1.73
Basking hangingwall main	West Dipper	32	87	0.005	255.62	11.91

Table 8.20 Summary of domains estimated for the Llanberris Mako lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au) Min	Max	Average
Siberian block 1 upper	steep west	10	34	0.01	89.82	5.58
Siberian block 1 lower	steep west	2	10	0.84	316.71	35.84
Siberian block 2 upper	steep west	1	13	0.01	1.95	0.42
Siberian block 2 lower	steep west	3	16	0.01	12.79	2.84
Siberian block 3 upper	steep west	5	21	0.01	172.07	17.75
Siberian block 3 lower	steep west	3	51	0.01	30.14	4.06
Siberian block 4 upper	steep west	4	24	0.01	2.9	0.55
Siberian block 4 lower	steep west	2	69	0.01	86.21	4.96
Siberian block 5 upper	steep west	1	11	0.01	0.17	0.04
Siberian block 5 lower	steep west	13	190	0.01	451.24	6.63
Mako 1 block 5 upper	shallow west	7	25	0.01	49.24	6.66
Mako 1 block 5 lower	shallow west	10	33	0.07	40.54	9.10
Mako 4 block 5 lower	shallow west	13	45	0.09	185.97	8.50
Anticline stockwork block 5 lower	stockwork	6	138	0.01	129.13	2.69
Siberian block 1 upper	steep west	10	34	0.01	89.82	5.58
Siberian block 1 lower	steep west	2	10	0.84	316.71	35.84
Siberian block 2 upper	steep west	1	13	0.01	1.95	0.42
Siberian block 2 lower	steep west	3	16	0.01	12.79	2.84
Siberian block 3 upper	steep west	5	21	0.01	172.07	17.75
Siberian block 3 lower	steep west	3	51	0.01	30.14	4.06
Siberian block 4 upper	steep west	4	24	0.01	2.9	0.55
Siberian block 4 lower	steep west	2	69	0.01	86.21	4.96
Siberian block 5 upper	steep west	1	11	0.01	0.17	0.04
Siberian block 5 lower	steep west	13	190	0.01	451.24	6.63
Mako 1 block 5 upper	shallow west	7	25	0.01	49.24	6.66
Mako 1 block 5 lower	shallow west	10	33	0.07	40.54	9.10
Mako 4 block 5 lower	shallow west	13	45	0.09	185.97	8.50
Anticline stockwork block 5 lower	stockwork	6	138	0.01	129.13	2.69
Siberian block 1 upper	steep west	10	34	0.01	89.82	5.58
Siberian block 1 lower	steep west	2	10	0.84	316.71	35.84
Siberian block 2 upper	steep west	1	13	0.01	1.95	0.42
Siberian block 2 lower	steep west	3	16	0.01	12.79	2.84

Table 8.21 Summary of domains estimated for the Llanberris Basking lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au) Min	Max	Average
Basking vertical spurs	Vertical spurs	5	10	0.05	21.53	4.42
East dipper 3	East dipper	8	27	0.01	55.81	3.94
East dipper 1	East dipper	5	12	0.05	32.3	6.66
Basking main quartz	West dipper	18	126	0.01	59.2	1.54
East dipper 2	East dipper	4	9	0.02	19.14	7.1
Hammerhead main quartz	West dipper	7	112	0.01	5.52	0.71
Hammerhead footwall HG zone	West dipper	5	23	0.35	41.47	6.02
Hammerhead inner HG zone	West dipper	5	18	0.1	505.66	35.38
Hammerhead hangingwall HG zone A	West dipper	7	17	0.13	29.25	7.16
Hammerhead hangingwall HG zone B	West dipper	6	10	0.42	37.56	7.3
Basking fault south	West dipper	26	70	0.02	159.41	11.642
Basking fault down dip	West dipper	5	17	0.01	47.9	4.11
Basking steep HG zone	West dipper	12	26	0.07	46.6	9.1
East dipper 4	East dipper	5	18	0.02	35.9	6.7
East dipper 5	East dipper	2	9	0.1	14.2	3.41
Basking shallow HG zone A	West dipper	6	16	0.01	467.45	34.58
Basking shallow HG zone B	West dipper	3	12	2.32	774.75	89.59

Table 8.22 Summary of domains estimated for the Sovereign Tiger lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au) Min	Max	Average
Main footwall fault	West dipping fault	50	187	0.01	452.74	15.81
Lower footwall Fault	West dipping fault	22	66	0.01	189.6	8.00
Main hangingwall fault	West dipping fault	48	69	0.01	240.19	6.64
lower hangingwall fault	West dipping fault	14	41	0.01	23.85	5.06
Upper flat west dipping fault	West dipping fault	6	33	0.01	11.44	1.00
Lower east dipping fault	East dipper	5	35	0.01	40.39	3.40
Stockwork zone	Stockwork	1	6	1.04	58.51	12.75
Main peripheral stockwork zone	Stockwork	44	346	0.01	240.19	3.49

Table 8.23 Summary of domains estimated for the Sovereign Gummy lode

Domain	Mineralisation style	No. DDH's	No. Samples	Assay grade range (g/t Au)		
				Min	Max	Average
East dipper 2	East dipper	2	18	0.1	47.01	11.04
East dipper 3	East dipper	3	29	0.07	371.96	40.39
East dipper 4	East dipper	1	5	8.22	251.28	73.39
Gummy footwall 1	West dipping fault	14	25	0.15	120.51	16.66
Gummy footwall 2 north	West dipping fault	10	26	0.22	559.89	44.68
Gummy hangingwall 1 main	West dipping fault	52	113	0.04	520.4	17.8
Lower stockwork	Stockwork	26	258	0.06	457.4	5.34
North east dipper	East dipper	2	1	74.71	74.71	74.7
South east dipper	East dipper	4	15	0.08	29.25	4.65
Gummy footwall 2 south	Stockwork	7	14	0.06	5.7	1.34
South stockwork	Stockwork	14	157	0.01	195.46	4.54
Gummy footwall 3	West dipping fault	43	106	0.01	338.48	19.57
Gummy footwall 2 centre	West dipping fault	18	39	0.11	640.95	18.94
Upper stockwork	Stockwork	6	39	0.06	10.23	1.4
Top HG Spurs	Stockwork	2	8	2	90.35	26.31
Gummy mid main fault	West dipping fault	32	78	0.12	333.17	21.16
Gummy footwall 1 south	West dipping fault	3	4	0.06	7.85	2.11
South upper massive quartz	Stockwork	8	9	0.18	11.57	2.01
Gummy hangingwall 1 south	West dipping fault	10	39	0.1	65.71	6.14
East dipper 7	East dipper	3	10	0.09	87.27	17.9

8.3.5 Variography

Preliminary assessment of spatial grade distribution using variography was commenced during 2014, however at the time of reporting has not been undertaken in sufficient detail for consideration during this estimate. Further work is planned for 2015 to integrate the use of variography in the mines estimation process.

8.3.6 Block Modelling and Estimation

Volume model construction

Independent block models were constructed for each of the five lodes included in this mineral resource. The models were constructed using Vulcan software. The volume cell model was constructed using the interpreted mineralisation solid wireframes as described in Section 7.1. All block models were rotated 5^{o} to align the blocks to approximately the same strike as the west-dipping faults to improve the way sub-blocks represent the domain wireframes. Table 8.24 provides a summary of the block model parameters used for each of the lodes estimated.

The parent block size of 15 mN by 5 mE by 5 mRL reflects half the drill spacing. Variable sized sub-blocking down to a minimum of 0.2 mN by 0.2 mE by 0.2 mRL was utilised to enable blocks to fit the constraints of the wireframes more closely. The selection of the parent block size is based on the “rule of thumb” of half drill fan spacing. It has not been derived from kriging neighbourhood analysis (KNA). It is anticipated that KNA test work will be undertaken during the coming year.

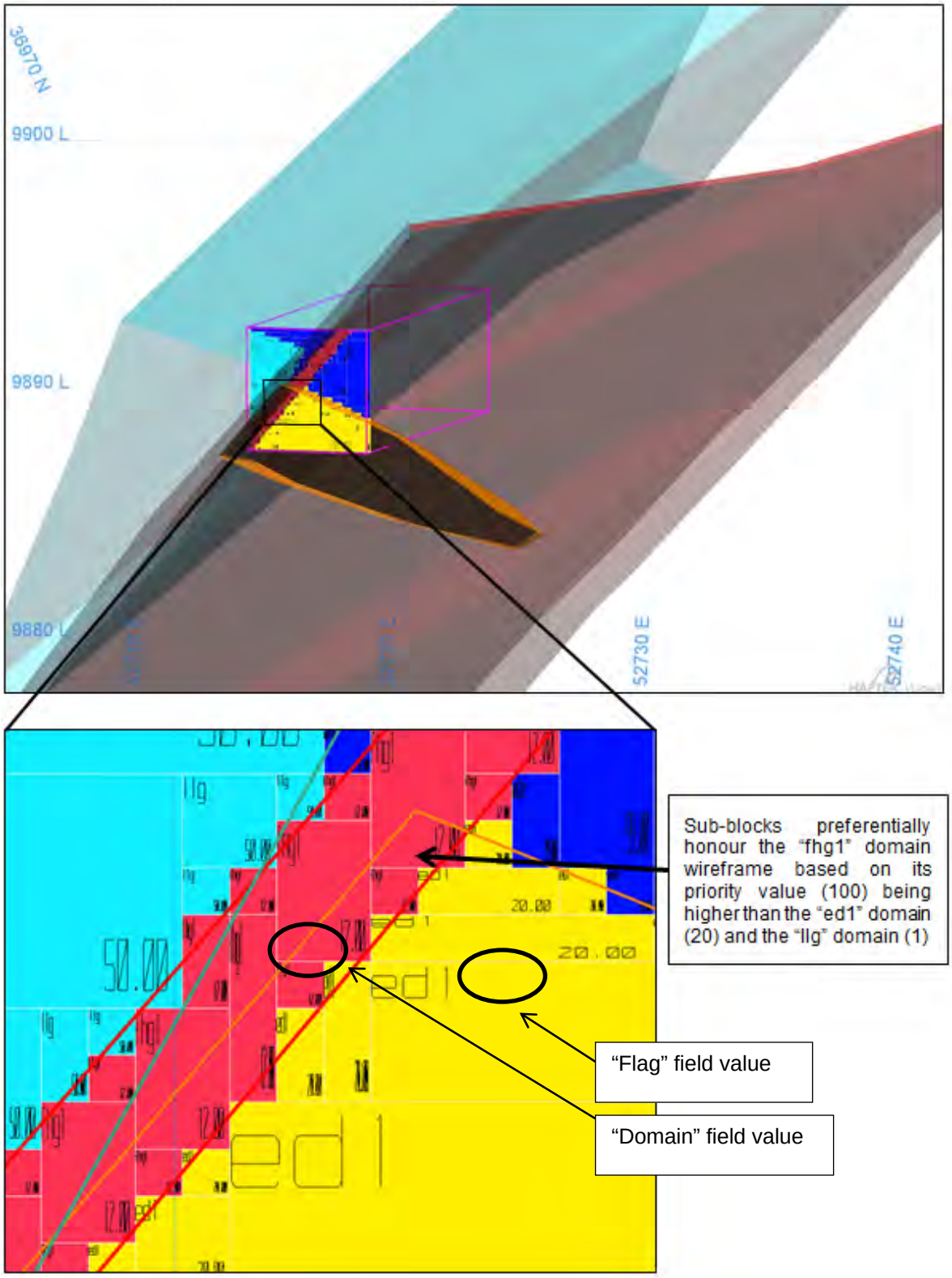
In a number of instances, domain wireframes have been constructed to overlap one another. Where this occurs, the "priority" function in Vulcan is used to ensure that sub-blocking is allocated to the preferred domain. Priorities are assigned based on interpretation of the structural setting of each domain. West-dipping faults are considered to be the most dominant structures, so are assigned the highest priority. East-dipping fault zones have been observed through mining to be truncated by west-dipping faults and are assigned a lower priority than the west-dipping faults. Stockwork zones adjacent the west-dipping and east-dipping fault zones are assigned the lowest priority.

Most domains modelled during this resource intersect domains adjacent them, attempting to make triangulations which don't overlap, but fit neatly together is time consuming, and difficult to achieve with perfectly adjoining surfaces. The risk is that some overlap or gaps are inevitable at domain boundaries. Deliberately overlapping wireframes and using priorities to assign sub-blocks to the preferred triangulation is simpler, and ensures no gaps occur providing better block assignment at domain intersection boundaries than constructing adjacent wireframes.

Sub-blocks within each domain are assigned unique alpha-numeric codes in the "domain" field built into the block model; they are also assigned unique numerical codes (ranging between 1 and 99) in the "flag" field built into the block model. Figure 8-29 demonstrates sub-blocking of separate domains within a single parent block including the assignment of "domain" and "flag" values (top left and bottom corner of each block respectively) and the use of priorities to preferentially assign sub-blocks to selected domain wireframes.

Importantly, the "domain" field within each sub-block is assigned the same alpha-numeric code assigned to the "bound" field in the composite sample .map file. This is used to ensure blocks within each domain are only estimated using composite samples from the specific domain being identified (refer to Section 8.3.6).

Figure 8-29 Section demonstrating effect of domain blocking priorities applied to overlapping domains in the Sovereign Gummy fault zone



Close up view of block allocation within domain wireframe boundaries and the allocation of "domain" and "flag" field values within sub-blocks. The solid blue, red and orange lines indicate the domain wireframe margins. 37030 mN – Orthogonal view, not to scale

Table 8.24 Block Model Construction parameters

Dimension	Origin (m)	Parent block size (m)	Minimum sub-block size (m)	Maximum sub-block size (m)	Extents (m)
Llanberris Mako					
Northing	37800	15	0.2	15	510
Easting	53231	5	0.2	5	300
Elevation	9675	5	0.2	5	300
Britannia Mako					
Northing	38200	15	0.2	15	600
Easting	53300	5	0.2	5	255
Elevation	9725	5	0.2	5	200
Llanberris Basking					
Northing	37750	15	0.2	15	510
Easting	53000	5	0.2	5	250
Elevation	9500	5	0.2	5	750
Victoria Mako					
Northing	38575	15	0.2	15	720
Easting	53300	5	0.2	5	300
Elevation	9700	5	0.2	5	300
Sovereign Gummy					
Northing	36750	15	0.2	15	510
Easting	52800	5	0.2	5	200
Elevation	9800	5	0.2	5	200

Apparent relative density

Density was assigned to each block in the resource model based on the apparent relative densities discussed in Section 6.2.6**.** A density of 2.65 g/cm^3 is applied to all mineralisation domains whilst, a density of 2.72 g/cm^3 is applied to all remaining blocks outside of mineralised domains. There is some variation (13%) in the relative densities used to determine the designated densities applied to this resource. This variance can be attributed to variation in voids and lithology.

Commencing February 2014, CGT has been submitting samples for density analysis as part of its sampling routine. Incorporating the estimation of density values into a block model from measured sample density data is expected to deliver a more accurate estimate of tonnage. This will be implemented in future estimations when sufficient density sample data has been collected.

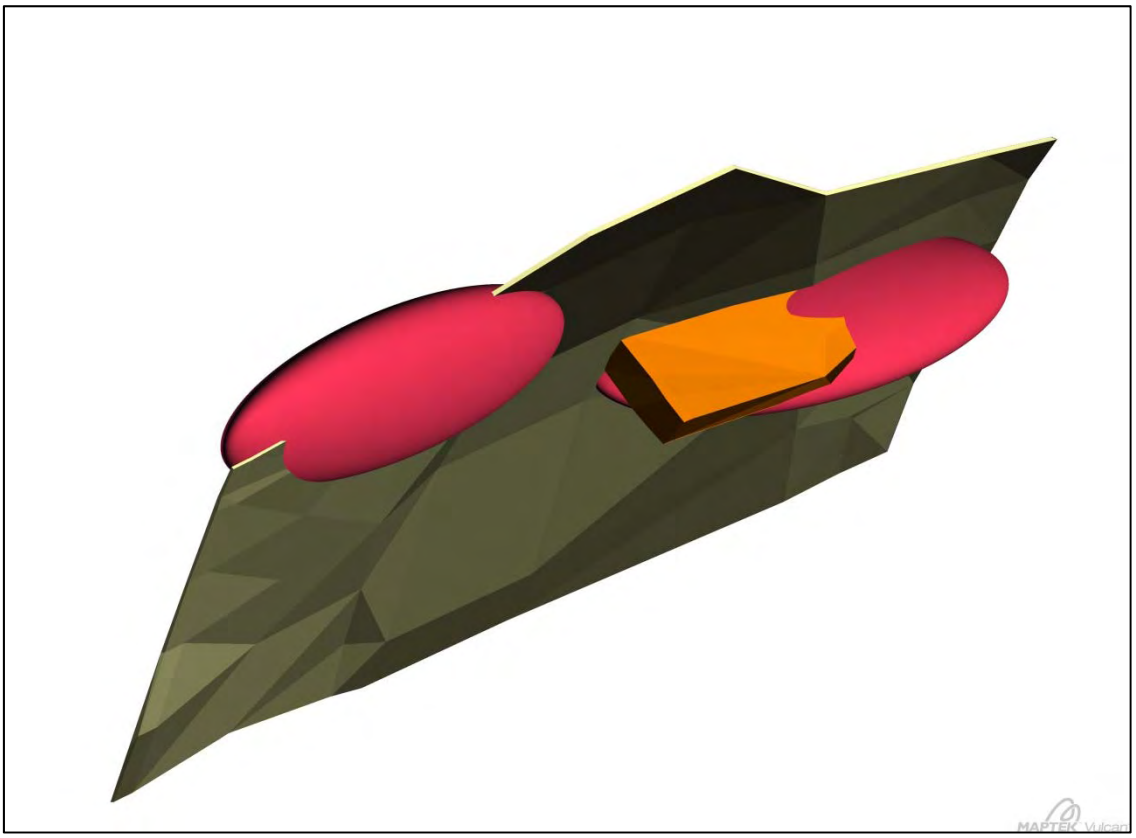
Search neighbourhood parameters

Search parameters were based on drill fan spacing and the orientation of the domain being estimated, with blocks estimated in two separate passes. All search ellipses were oriented to match the geometry of the domain being estimated, this was carried out in lieu of any analysis of the spatial distribution of gold grades (variography). It is anticipated that variography (planned for the coming year) will provide a better understanding of the spatial distribution of grades within domains, leading to refinement of search ellipses.

The extents of the first pass search ellipse were constrained to 60 m along strike; 20 m down dip and 10 m across strike (approximately twice drill spacing). The second pass, low confidence search ellipse was extended to 90 m along strike; 30 m down dip and 20 m across strike (approximately three times drill spacing).

Visual checks of the search ellipses used for each domain were completed using Vulcan software to validate the search orientation and extent (Figure 8-30)

Figure 8-30 Example of primary search ellipsoids for estimation in the Sovereign Gummy fault zone – orthogonal view, not to scale



Grade estimation

Grade interpolation method

Gold grade is estimated using inverse distance weighting (IDW) estimation. IDW is a linear interpolator where data is smoothed without reference to the spatial variability of the data. In each case a search area or volume is erected around each block centre (or discretization point) in turn. Any sample values (grade and thickness) captured by these areas/volumes are weighted by the inverse of the distance of each of these samples from this point raised to a power 'n'.

The weighting power for the inverse of the distance may vary between 1 and 5; the former may result in excessive smoothing while the latter effectively only selects the nearest sample as this receives a dominant weighting. Beyond n = 5 the estimate effectively becomes very similar to a polygonal result. IDW cubed is suited to erratic high-grade domained data that will be selectively mined.

The problems related to the application of the IDW method include the selection of unsuitable search and weighting parameters that fail to reflect the nature and variability of the mineralisation. There is rarely any relationship between the search area and the geostatistical range. The technique results in the excessive smoothing and smearing of high-grades by use of large search areas combined with low values of the weighting power 'n' resulting in lower global and local grade estimates. Grades are smeared into barren/low-grade areas generally resulting in excessive tonnage estimates.

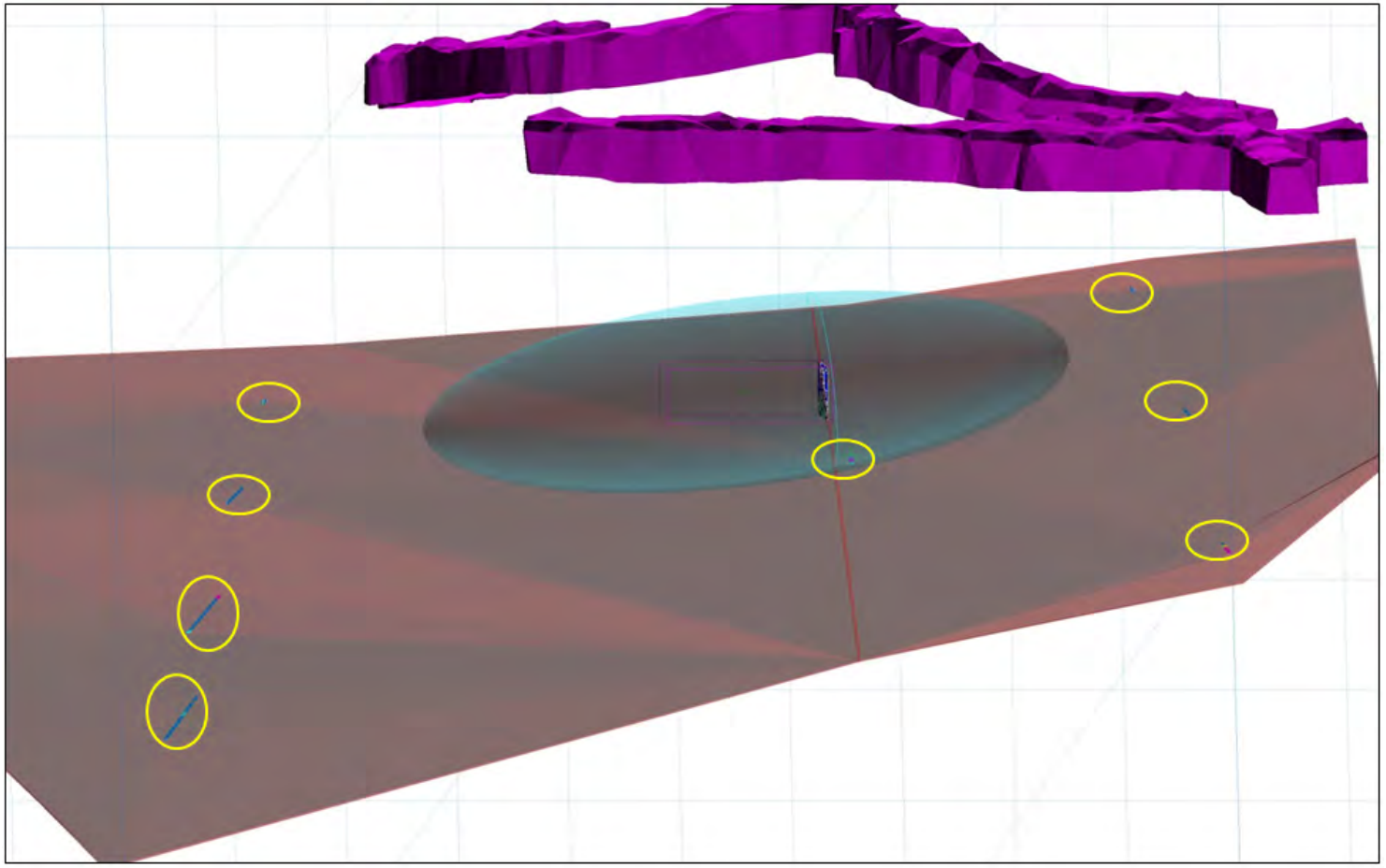
As with all non-geostatistical estimators, IDW makes no distinction between the nugget and spatial components of the total variation. Only spatial variation is assumed, although a nugget component is an observable fact in all gold ore bodies. The smaller the nugget component the better the method works and vice versa. As the nugget component increases, IDW becomes more biased and there is an increasing and negative linear correlation between the 'nugget effect' (e.g. high nugget variance in relation to the total variance) and the regression slope of the true block value on its estimated value

Ballarat methodology

For this estimate an IDW to the power of two was used. Top-cutting was applied to composite grades prior to block estimation with grades above the top-cut threshold re-assigned to the top-cut grade before being used in the estimation algorithm. Top-cuts were selected based on statistical analysis of the composite grades contained within each domain (Section 8.3.4).

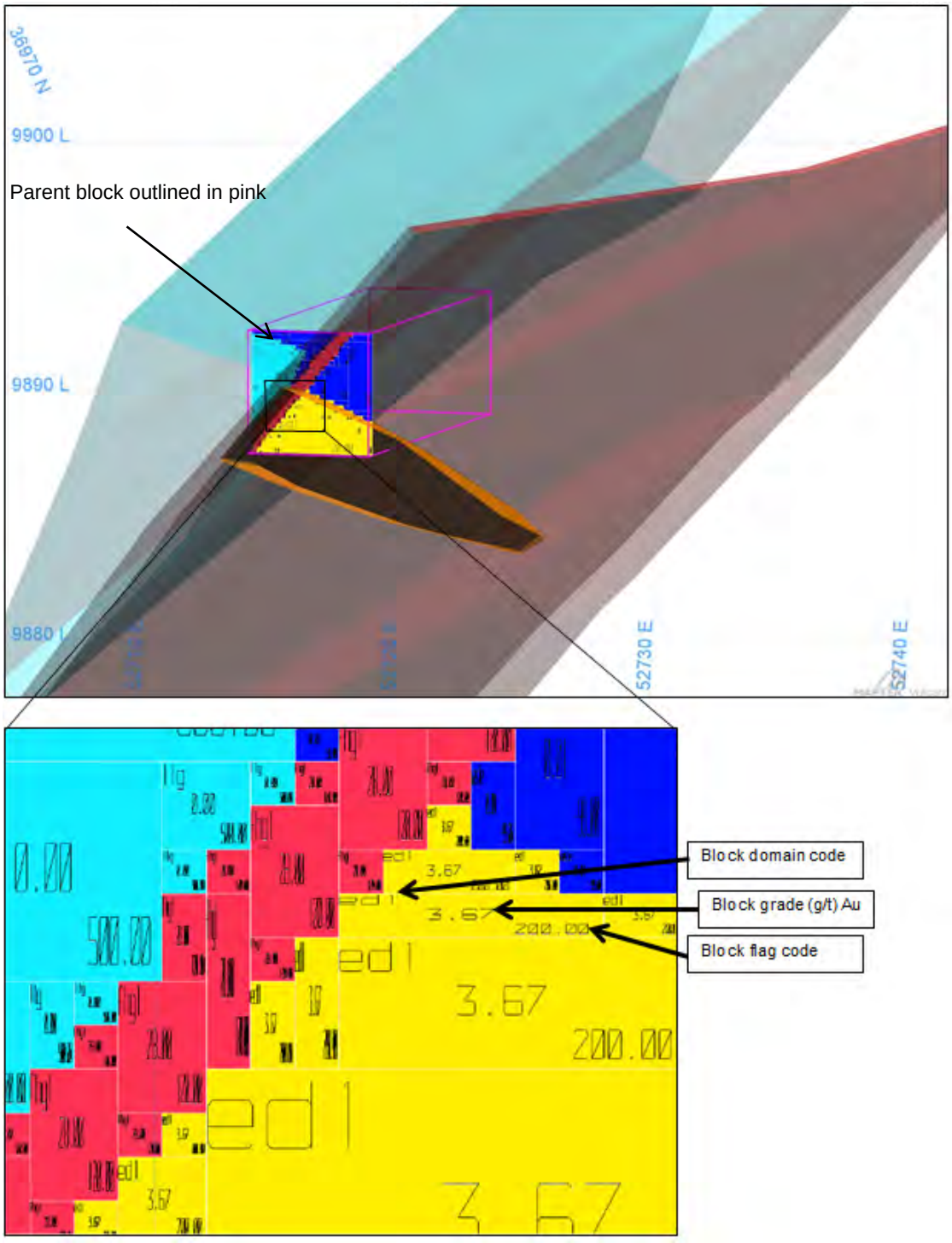
Block estimation is carried out one domain at a time. Sub-blocks are selected for estimation based on the unique flag field assigned to them during block construction. Composite samples used for estimation are selected using the unique "bound" field codes assigned during compositing. This ensures that sub-blocks are estimated using only composites which fall within the specific domain being estimated (Figure 8-31).

Figure 8-31 Example of a search ellipse for estimation of sub-blocks with the domain code “fhg1”, relative to composites samples with a “bound” field code of “fhg1



The parent block outline is shown in pink, in this instance, four composites (circled in yellow) from one drill hole fall within the search ellipse (shown in blue).

Figure 8-32 Example of sub-block grade allocation where multiple domains intersect a single parent block in the Sovereign Gummy fault zone. 37,030 mN – orthogonal view, not to scale



Close up view of the allocation of domain, flag and grade field values within sub-blocks. Note that flag values have increased by a factor of 10 (when compared to Figure 8-29) as they have been estimated on the first pass. Sub-blocks within each domain in the parent block are allocated the same grade value based on the parent block grade, in this example, all sub-blocks within the "fhg1" domain have a grade of 28 g/t Au, whilst in the "ed1" domain all sub-blocks have a grade of 3.7 g/t Au.

Table 8.25 Block model parameters summary

Attribute name	Type	Default	Description
domain	Name (Translation Table)	waste	bound code
pass	Name (Translation Table)	not estimated	Identifies which estimation pass was used to estimate grade
Flag	Float (Real * 4)	-99	Estimation Flag records if and which estimation has been applied
nholes	Integer (Integer * 4)	-99	number of holes used to estimate block grade
nsamp	Integer (Integer * 4)	-99	number samples used to estimate block grade
block_var	Integer (Integer * 4)	-99	Stores block variance
density	Float (Real * 4)	2.72	default - set at 2.72 g/cm^3
au2_tcstat	Float (Real * 4)	-99	Inverse Distance squared top-cut to statistically chosen grade.
au_ave	Float (Real * 4)	-99	Average grade of all assays informing block estimate
mined	Float (Real * 4)	100	Percentage of block available for mining 100 = not mined, 0 = completely mined
classification	Byte	0	Classification as per JORC reporting 0 = unclassified, 1= Inferred

8.3.7 Validation

Block models were validated by on-screen inspection and visual comparison of block and sample grades for gold. Additionally, a comparison of the mean input sample grades and the mean output block grades was also conducted.

A comparison of wireframe volumes against block models was carried out. As some wireframes overlap others not all wireframe volumes are comparable with block volumes (dependent on the manner in which block priorities have been applied). Instead a comparison was made on a selection of wireframes which were not overlapped in order to obtain a true assessment of the efficiency with which blocks estimate the wireframe volumes. Two domains were selected from each of the lodes estimated, the comparison showed less than 1% difference between wireframe and block volumes (Table 8.26).

Block construction in areas of overlapping domains were visually assessed in Vulcan to ensure that block construction “priorities” have assigned blocks to the correct wireframes. All overlapping wireframes were reviewed visually in Vulcan and found to have blocks assigned to the correct wireframes.

Table 8.26 Comparison of wireframe and block model volumes

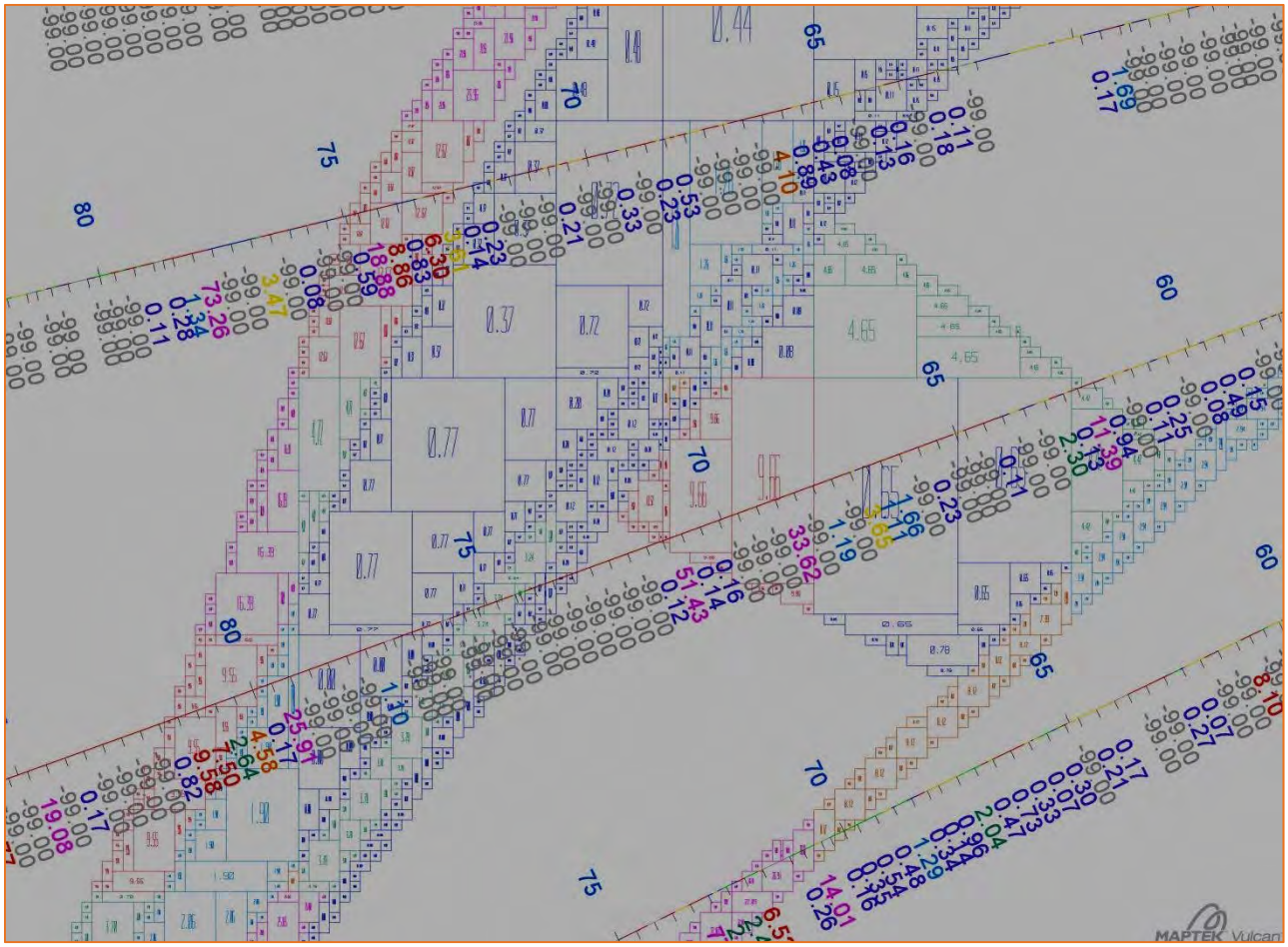
Lode	Domain	Block volume (m^2)	Wireframe volume (m^2)	% diff
Britannia Mako	bm1	1,734	1,734	0.01%
Britannia Mako	shg	18,524	18,527	-0.01%
Llanberris Mako	fwc	33,223	33,222	0.00%
Llanberris Mako	fws	7,578	7,579	-0.01%
Sovereign Gummy	hg1	23,419	23,418	0.01%
Sovereign Gummy	fhg1	5,498	5,449	0.90%
Llanberris Basking	hg3	2,702	2,705	-0.11%
Llanberris Basking	hg2a	1,500	1,498	0.10%
Total		94,178	94,132	0.05%

Visual validation

The estimated block grades were validated visually by comparing raw diamond drill hole grades (not de-clustered) against block estimates. The estimated grades show moderate variation from adjacent drill hole grades. This is due to sub-blocks for each domain being assigned the grade of the parent block for the

specific domain (Section 8.3.6). Sub-blocks grades are unable to reflect the grade variability observed in drilling data. Visual assessment concluded that sub-block grades do not reflect well, the grade variability observed within drill holes, this is due to the practise of assigning sub-block grades the grade of the parent block for the specific domain in question. Figure 8-33 gives an example of validation of block grades against drill hole assay grades.

Figure 8-33 Estimated gold grades versus drill hole gold grades in Sovereign Gummy lode at 37,000 mN - section looking north, not to scale



Input and output means

The mean grade of the uncut sample composites (not declustered) and the estimated block grades were compared for all domains estimated. Table 8.27 summarises the differences for each of the lodes in the resource.

This analysis shows that the estimation has produced blocks with a mean grade 33% lower than the average of the uncut composite grades. The difference is attributed largely to composite top-cutting prior to estimation; however geological domaining and the estimation method used are also likely to have an effect.

Table 8.27 Mean grade comparison between the uncut input drill hole composites (not de-clustered) and block model – total deposit

Lode	Composites (g/t Au)	Block model (g/t Au)	Difference (%)
Britannia Mako	8.16	4.57	-44.0
Britannia Basking	6.39	5.62	-12.1
Llanberris Mako	5.91	3.73	-36.9
Llanberris Basking	8.20	3.02	-63.1
Sovereign Gummy	13.27	6.87	-48.2
Sovereign Tiger	7.07	4.92	-30.4
Combined	8.27	4.93	-40.4

Moving window statistics

Sectional validation graphs were created comparing the average of the estimated grades to the top-cut and uncut composite grades. Comparisons were made in three directions (e.g. easting, northing and elevation) within model slices (bins). This was undertaken to assess the re-production of local means and to validate the grade trends within the model. The graphs also chart the number of samples within each slice. Block estimates in well-drilled areas should compare well with the average grade of the respective composites. Areas that show significant discrepancy between the average composite grades and the estimated grades are areas of concern and require further investigation.

Figure 8-34 gives an example of a sectional swath plot comparing uncut and top-cut composite grades against estimated grades on 30 m north-south oriented intervals in the Sovereign Gummy fault zone. North-south oriented sections are given the greatest importance as the lodes strike north south and are considerably longer in the north-south direction than any other.

The north-south plots show a reasonable correlation between the estimated block grades and input composited drill hole grades within well drilled regions. Significant discrepancies were found in sections of all lodes analysed, however in each case this could be explained by a combination of top-cutting of composite grades and/or tight domaining of narrow high grade zones. The most severe example of this is shown in Britannia MFZ (Figure 8-35).

Between 38,420 mN and 38,570 mN a considerable difference is observed between composite grades and block estimates. The chart demonstrates that top-cutting is partially responsible for this discrepancy; however geological domaining also contributes to this difference.

By constructing tightly constrained domain wireframes of the zones of elevated grades associated with narrow west-dipping and east-dipping faults, and using large stock-work wireframes to capture the remaining lower grades surrounding these structures, the way in which grades are apportioned is affected as a result. The domains associated with fault structures have a higher sample density per tonne than the, broad stock-work domains surrounding them, so when volume weighting is taken into consideration, the narrow zones of elevated grade, have less impact on the average when volume weighted, than when they are arithmetically averaged.

Comparison of the distribution of grades as a proportion of total sample length for sample composites against grades as a proportion of total block volume, reveals that a higher proportion of the block model volume is associated with low grade (less than 5 g/t Au) estimates, than the proportion of samples associated with low grade composites within the composite dataset. Figure 8-36 outlines the relative proportions of block volumes compared to composite sample volumes by geological domain. In this zone of the Britannia Mako, over 50% of the block volume is represented by the low grade “alg” and “mdf” domains. This comparison does not take into consideration composite sample or block grades outside of modelled domains (waste).

Figure 8-34 Moving window sectional swath plot showing both uncut and top-cut composite gold grades versus estimated block grades for the Sovereign Gummy fault zone

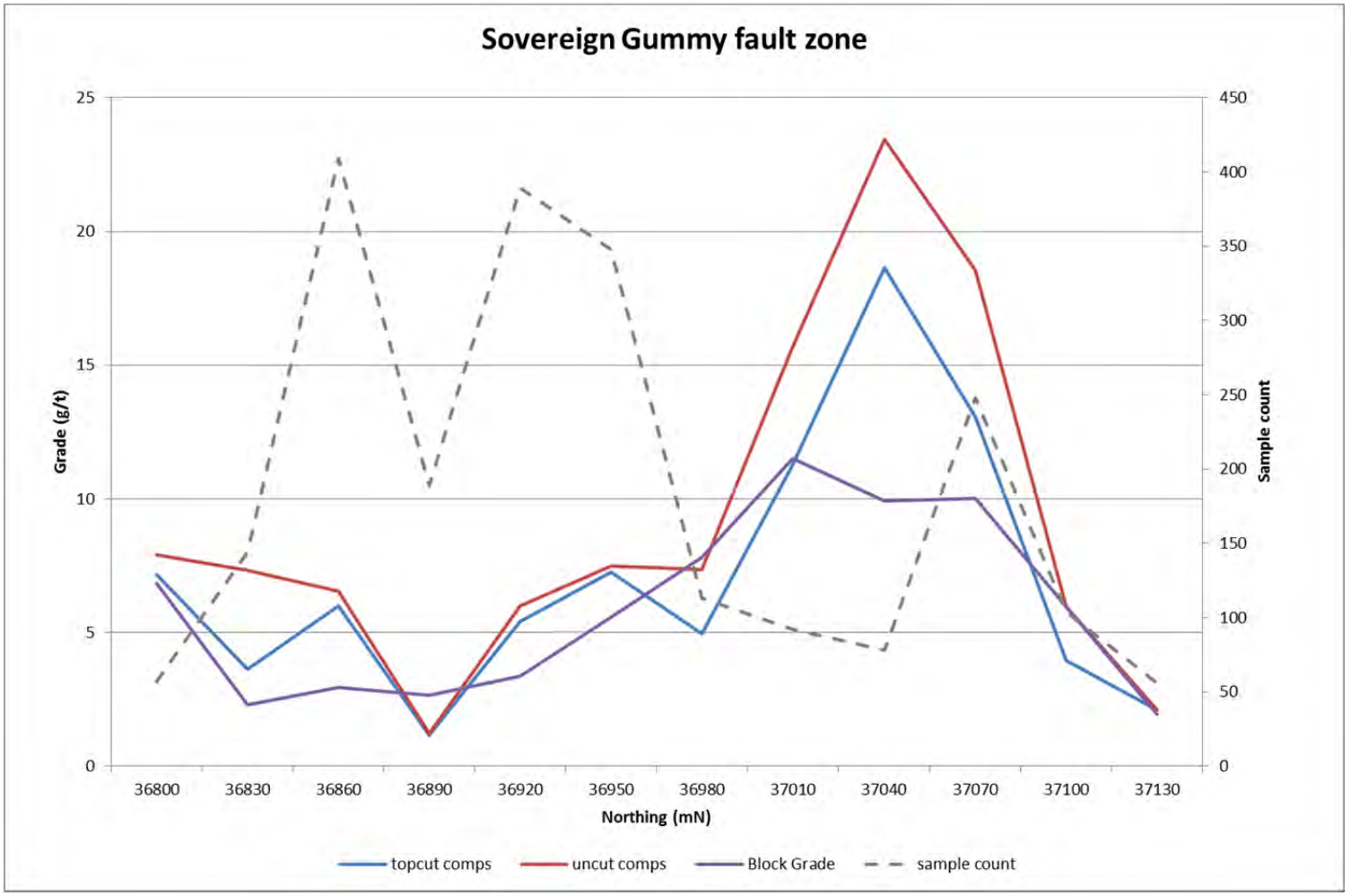


Figure 8-35 Moving window sectional swath plot showing both uncut and top-cut composite gold grades versus estimated block grades for the Britannia Mako fault zone

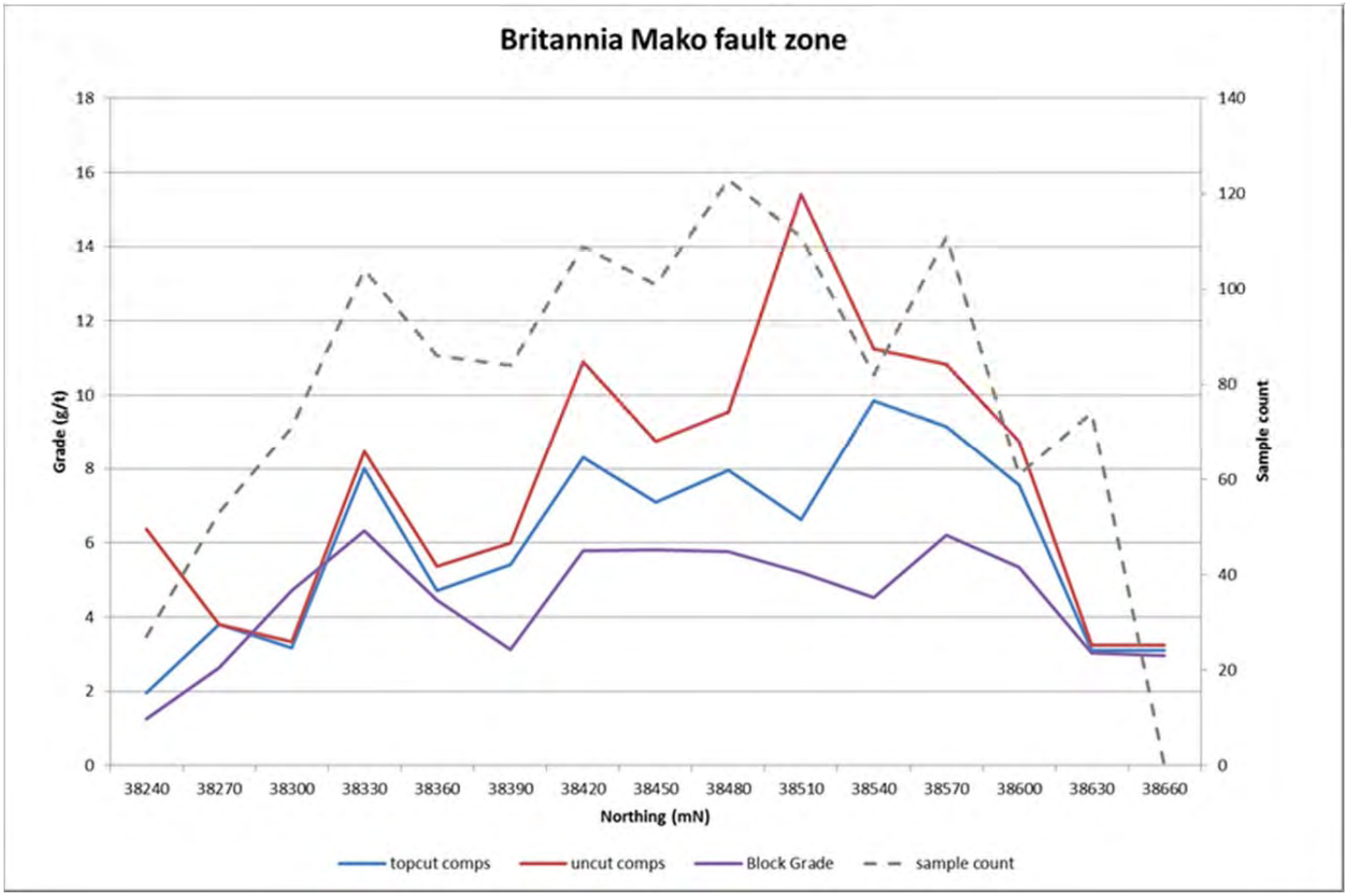
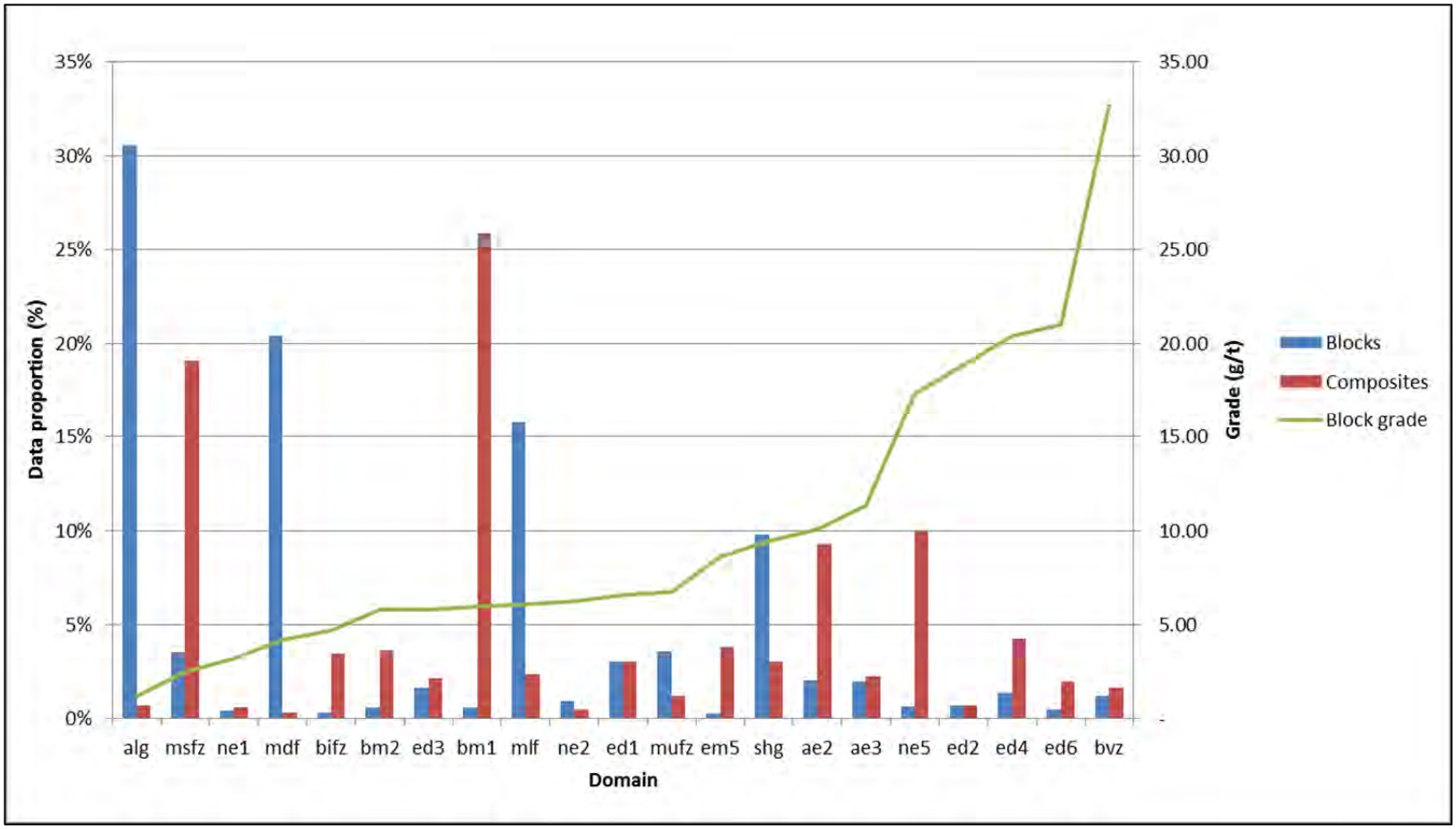


Figure 8-36 Comparison of the relative proportions of composite samples against block volumes in the Britannia Mako fault zone between 38,420 mN and 38,570 mN



Search pass comparison

Block estimations were carried out in two passes, whereby the second pass used a larger search ellipse than the first. Analysis of the proportion of blocks estimated on each pass can highlight errors in domain codes and search ellipse orientations. In general the current estimation parameters result in 80% or greater of all blocks to be estimated on the first pass, with the remainder either estimated on the second pass, or not estimated if insufficient samples are found within the search ellipse. All domains estimated were reviewed and found to meet these expectations. Table 8.28 summarises the proportion of blocks estimated.

The Llanberris Mako estimation resulted in 6% of the blocks not being estimated. The majority of these blocks fall in domains within the Tiger fault zone which have been intersected by mining and modelled, but lack the drill hole sample support for estimation to be carried out. These domains were omitted from the final resource. The remainder of blocks not estimated were reviewed in Vulcan and found to be on the margins of the resource. They are omitted from the final resource.

Table 8.28 Summary of proportion of blocks estimated by each search pass for each lode

Lode	Pass 1	Pass 2	Not estimated
Britannia Mako	80%	17%	3%
Llanberris Basking	84%	15%	1%
Llanberris Mako	87%	6%	6%
Sovereign Gummy	89%	10%	1%
Victoria Mako	89%	9%	2%

Comment on validation

The validation procedures undertaken show that the model is a reasonable approximation of the input data, but it is not best practice. This is to a major extent reflected in the resource classification.

Over the next year, a review of estimation will be undertaken covering:

- Variography to define spatial variability
- QKNA to optimise estimation block size
- Kriging or variant thereof to interpolate grade

8.3.8 Classification

In high-nugget narrow-vein gold deposits such as Ballarat, proving continuity of both mineralisation (geology) and grade can be economically prohibitive. Generally, such deposits remain high risk even during mining operations (Dominy, 2014).

The drilling carried out into this resource is considered sufficient to verify geological continuity, however, due to the high grade variability observed it is only considered sufficient to imply grade continuity, and not to verify it. As such, this estimation has been classified as an **Inferred Mineral Resource** as defined by the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore reserves (The JORC Code, 2012).

Whilst all geological domains were estimated, they were only included into the resource if they met a set of criteria outlined in Table 8.29.

Table 8.29 Inferred Mineral Resource classification criteria

Criteria	Minimum requirement
No. drillholes	>3 drillholes per domain
Spatial distribution	Must be intersected on two or more drill fans
No. Samples	>8
Estimated domain grade	>4 g/t Au
Mining depletion/sterilisation	>500 t remaining

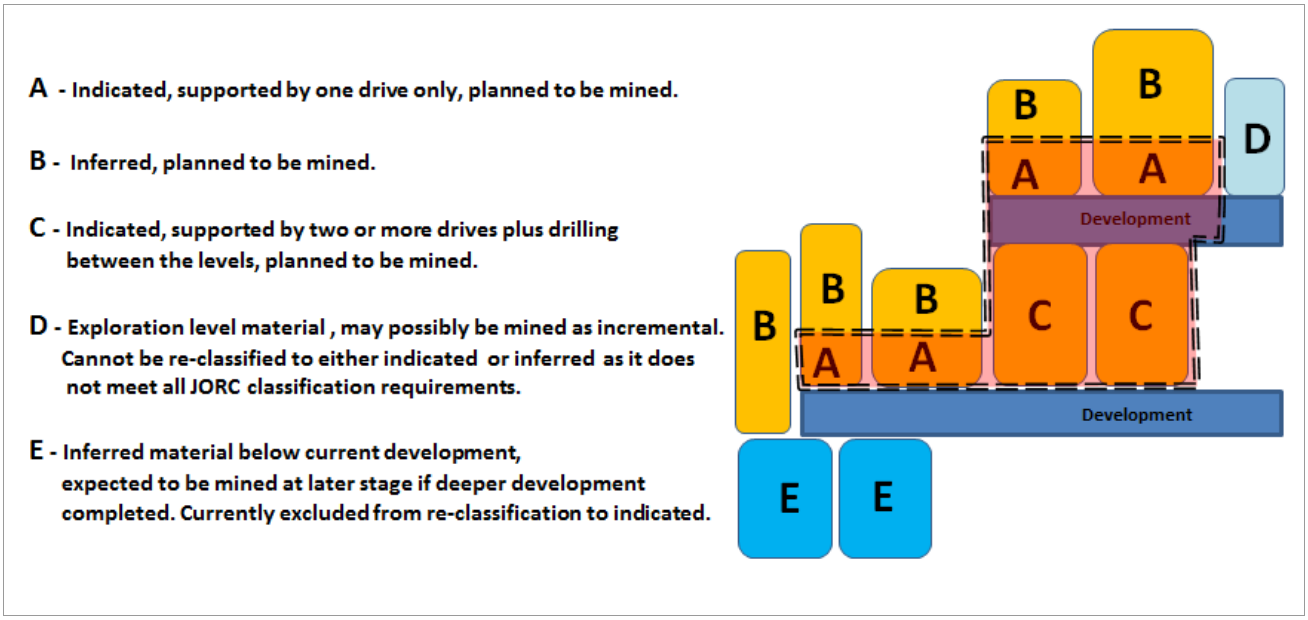
The number of drill holes required for a domain to be included in the resource is three holes. This is considered the minimum requirement to verify geological continuity. A minimum of eight composites was required for a domain to be included. Whilst these numbers are quite low, they are considered adequate to meet the requirements for classification as an Inferred Mineral Resource.

Indicated Resources

The delineation of indicated resources is based on several conditions;

1. Existing inferred level resources that are likely to be mined and have been verified in the development and drilling as matching the modelled orientation and expected structural/geological setting. Mineralised material that does not meet the required grade or support may still be mined as incremental ore but is not classified as either inferred or indicated.
2. The development to access the mineralisation must be in place, where there is only one drive and there is no expectation of another below it, then only a zone up to **10m** above the drive is reclassified as indicated as long as condition 1 is met.
3. Where two or more levels of development exist on the same mineralised structure and sufficient support exists in the drilling between them, then spans up to 30m can be reclassified as long as condition 1 is met.
4. Once initial indicative resources are delineated the shapes are forwarded to engineering for evaluation. Mining shapes generated by engineering on the initial indicated resources are then used to convert any remaining geologically supported inferred material within the mining shapes to indicated status. Condition 1 and either of 2 or 3 must still be met; non-inferred mineralised materials do not change status.

Figure 8-37 Diagram of inferred and indicated resource material relative to development.



Based on the predicted 2014-2014 budget combined mining and processing cost of A$184 per t (excluding capital development costs), a gold price of A$1,395 per oz Au and mill recovery of 87%, a breakeven cut-off grade of 4.5 g/t Au is estimated. Accordingly, for the resource those domains whose average grade is less than 4 g/t Au are excluded from the estimate.

8.3.9 Reported Mineral Resources

This Mineral Resource estimate comprises mineralisation from within five separate lodes (Table 8.30 and Table 8.31) within the Ballarat mine.

The estimation block size used (15 m by 5 m by 5 m; approximately 1,010 t equivalent) is larger than the expected selective mining unit (SMU). A development SMU may reasonably be expected to be 200 t to 270 t and a stope SMU (single ring) between 50 t and 200 t. As a result, selective mining above a cut-off on an estimation block by block (e.g. 1,010 t) basis is unlikely to be achievable. The resource is thus reported at a 0 g/t Au cut-off and is global in nature.

For completeness, an assessment of the application of cut-off grades to this resource is given in Figure 8-38.

Table 8.30 Indicated Mineral Resource estimate for the Ballarat mine at 0 g/t Au cut-off for 31st March 2015

Lode	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Britannia Mako	4,000	14.2	1,900
Britannia Basking	25,500	12.8	10,500
Llanberris Basking	9,500	7.5	2,300
Sovereign Tiger	19,500	15.6	10,000
Sovereign Gummy	21,000	23.7	15,800
Total	**79,500**	**15.9**	**40,500**

Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

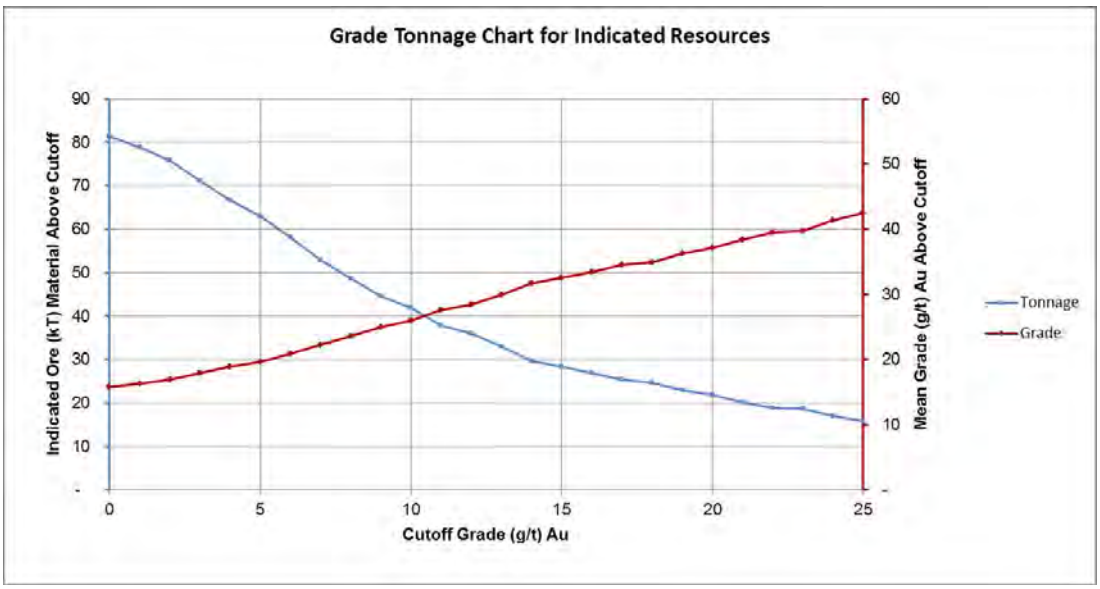
For completeness, an assessment of the application of cut-off grades to this resource is given in Figure 8-38 and Figure 8-39.

Table 8.31 Inferred Mineral Resource estimate for the Ballarat mine at 0 g/t Au cut-off for 31st March 2015

Lode	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Britannia Mako	81,000	7.9	20,500
Britannia Basking	182,500	6.8	40,100
Llanberris Basking	36,000	8.1	9,500
Llanberris Mako	49,000	6.3	9,900
Sovereign Tiger	30,500	5.3	5,200
Sovereign Gummy	81,000	7.9	20,500
Total	**460,000**	**7.1**	**105,600**

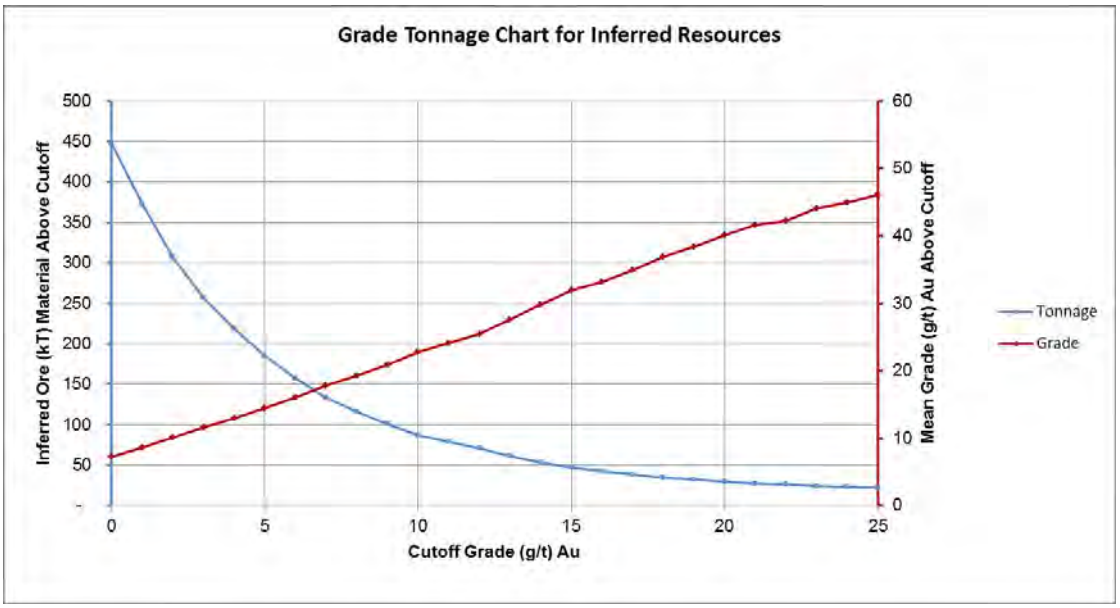
Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

Figure 8-38 Grade-tonnage curve for the Ballarat Indicated Resource as at 31st March 2015



Note for the reasons provided above, this grade-tonnage curve should be treated with caution

Figure 8-39 Grade-tonnage curve for the Ballarat Inferred Resource as at 31st March 2015



Note for the reasons provided above, this grade-tonnage curve should be treated with caution

The JORC Code (2012) requires that a resource must have "*reasonable prospects for eventual economic extraction*". The Ballarat gold mine is currently operational, based on decline access and fully mechanised mining methods. Stoping is via a combination of conventional drive development and open stoping. The on-site processing plant achieves a recovery of 87%. The Mineral Resource is deemed to have reasonable prospects for eventual economic extraction.

Comparisons with previous mineral resource estimate

This Mineral Resource represents a 7% increase in the contained gold (oz) when compared against the March 2014 estimate. As outlined in Table 8.32.

Table 8.32 Comparison between current and previous Mineral Resource estimates at Ballarat mine. All resources reported at a 0 g/t Au cut-off

Classification	31st March 2014 Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)	31st March 2015 Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Indicated	-	-	-	79,500	15.9	40,500
Inferred	370,000	11.5	137,000	460,000	7.1	105,600
Total	**370,000**	**11.5**	**137,000**	**539,500**	**8.4**	**146,100**

Note: Mineral Resources which are not Ore Reserves do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

Table 8.33 Comparison between current and previous Inferred Resource estimates at Ballarat mine. All resources reported at a 0 g/t Au cut-off

Lode	31st March 2014 Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)	31st March 2015 Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
Britannia Mako	115,500	8.7	32,200	81,000	7.9	20,500
Britannia Basking				182,500	6.8	40,100
Llanberris Mako	111,000	10.8	38,600	49,000	6.3	9,900
Llanberris Basking	24,500	9.5	7,500	36,000	8.1	9,400
Victoria Mako	8,000	11.8	3,100	-	-	-
Sovereign Tiger	-	-	-	31,000	5.3	5,300
Sovereign Gummy	110,500	15.5	55,200	81,000	7.9	20,500
Total	**370,000**	**11.5**	**137,000**	**460,500**	**7.1**	**105,700**

Note: Mineral Resources which are not Ore Reserves and do not have demonstrated economic viability. Tonnage is reported in metric tonnes (t), grade as grams per tonne gold (g/t Au) and contained gold in troy ounces (oz Au). Tonnages rounded to the nearest 500 t. Ounces rounded to the nearest 100 oz Au.

Whilst the delineation of two new ore lodes (Britannia Basking and Sovereign tiger) has added to the global Resource tonnages, the relatively low grade of these lodes combined with the mining of high grade zones from within existing ore lodes, has resulted in an over-all reduction in grade. The end result is a modest (7%) increase in ounces in Indicated and Inferred Resources. Figure 8-40 to Figure 8-42 provide detail of the cumulative changes to the tonnes, grade and contained ounces reported.

Figure 8-40 Waterfall chart showing cumulative differences in tonnage between current and previous Mineral Resource estimate

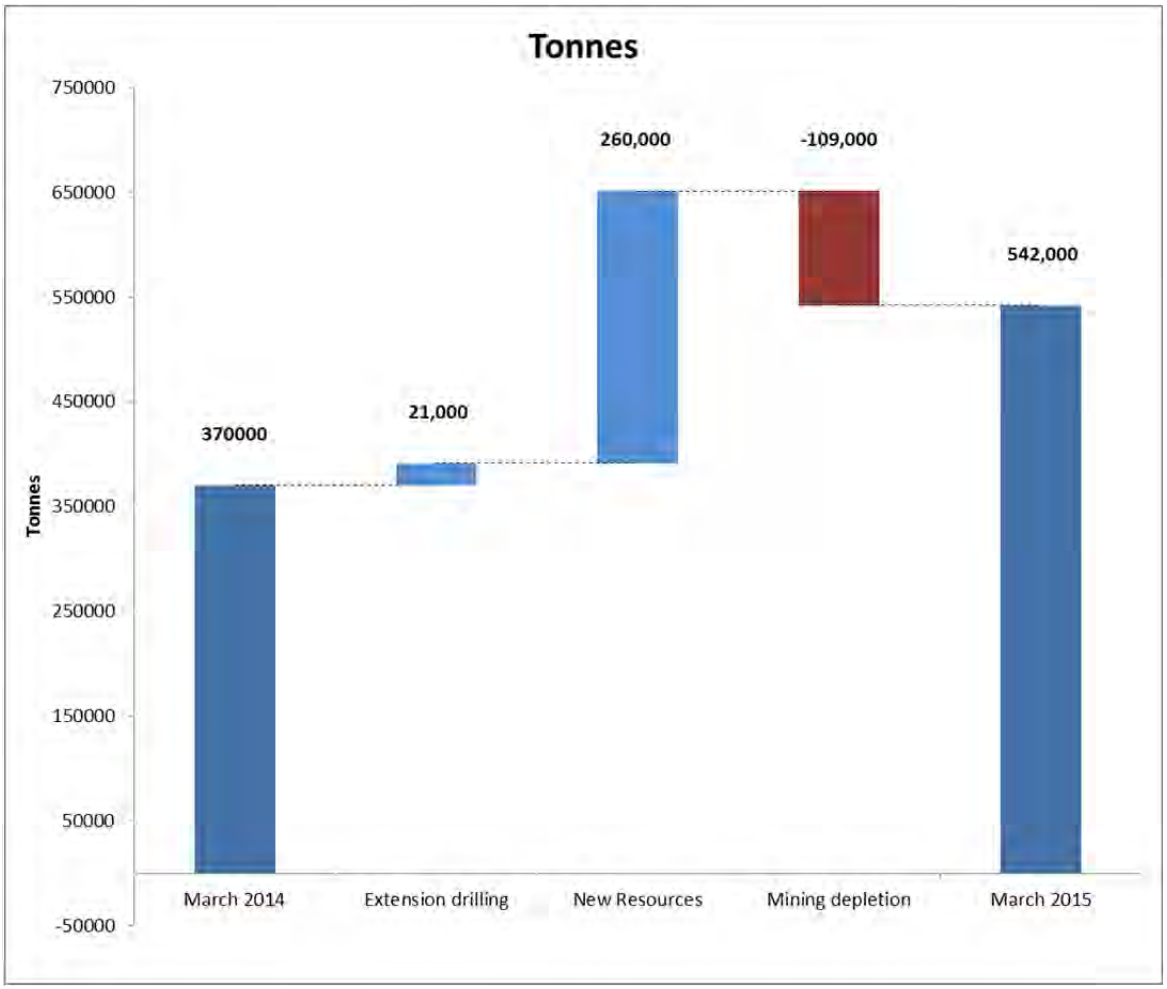


Figure 8-41 Waterfall chart showing cumulative differences in gold grade between current and previous Mineral Resource estimate

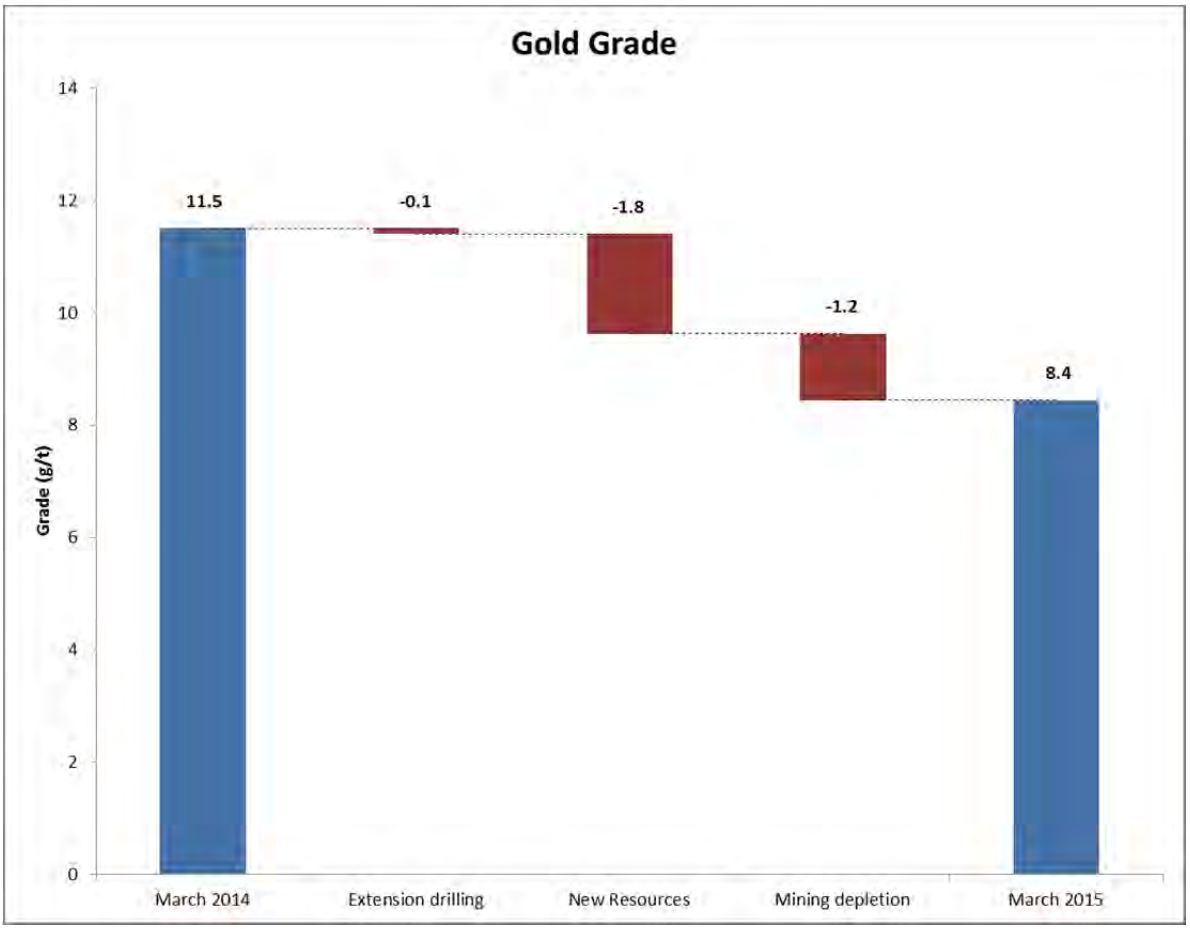
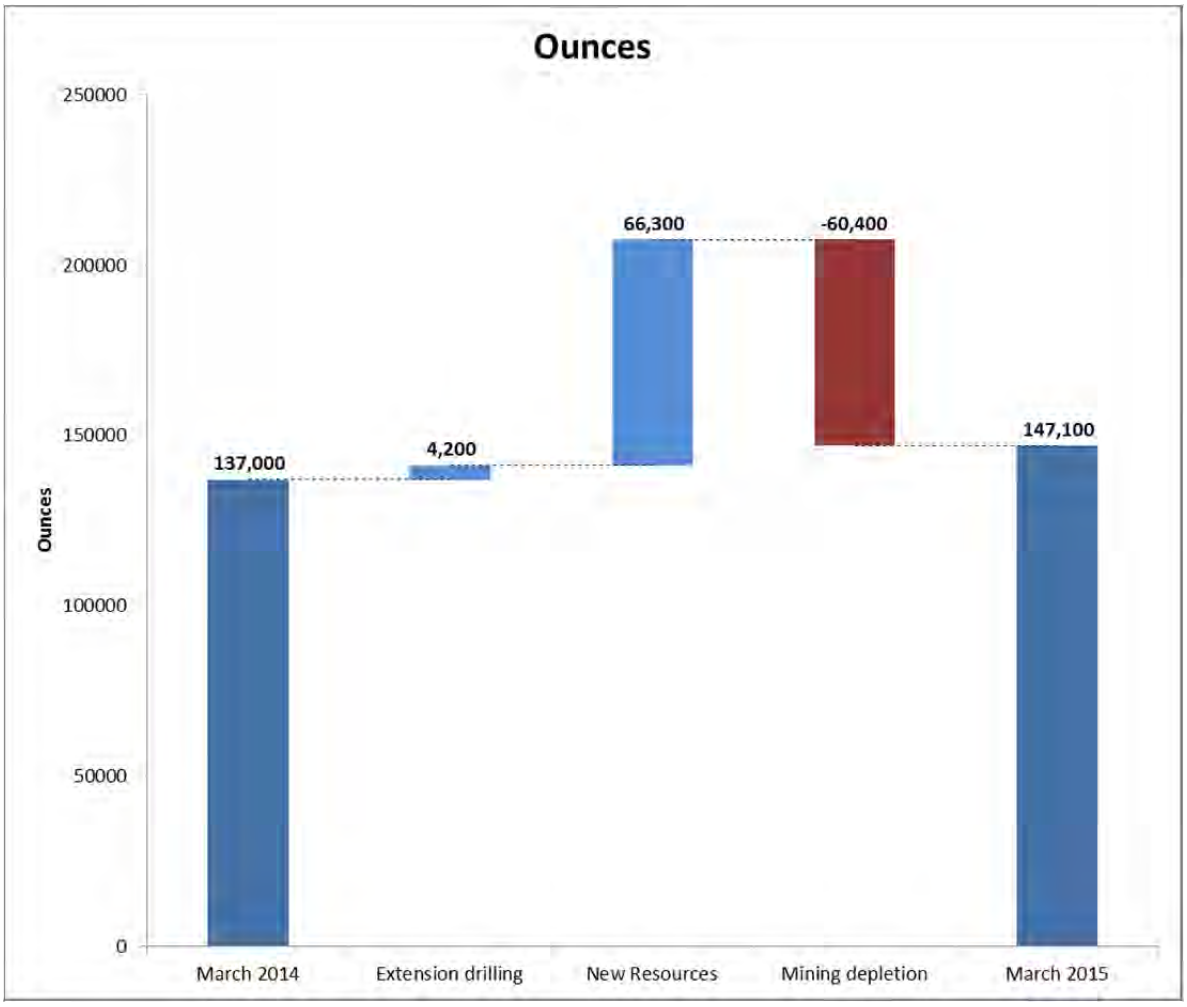


Figure 8-42 Waterfall chart showing cumulative differences in gold troy ounces between current and previous Mineral Resource estimate



9 ORE RESERVES

9.1 Summary of Ore Reserves

The Ore Reserve for the Ballarat Gold Project has been estimated and reported in Table 9.1 in accordance with the JORC Code.

Table 9.1 Ore Reserve summary, as of 31 March 2015

Category	Mineral type	Gross attributable to licence		Net attributable to issuer			Remarks
		Tonnes (thousand)	Grade (g/t Au)	Tonnes (thousands)	Grade (g/t Au)	Change from previous update (%)	
Proved	-	-	-	-	-	-	
Probable	Au	129	7.61	129	7.61	100	First report Reserve
Total		**129**	**7.61**	**129**	**7.61**	**100**	**Issuer owns 100% of the company**

This is the first ore reserve estimate published by Castlemaine Goldfields Pty Ltd.

9.2 General Description of Ore Reserve Estimation Process

The Probable Ore Reserve is derived from the Indicated Mineral Resource, in accordance with the JORC Code 2012. Reported Indicated Mineral Resources are within 10 metres of established development.

The Indicated Mineral Resource is defined by 5 block models which are spatially defined by generally east-west striking locally termed *"cross-course"* faults and vertically by north-south mineralised bedding or anticlinal parallel faults. Refer section 8.

The underground Probable Ore Reserve is based on portions of the Indicated Mineral Resource model which are considered to be mineable based on historic unit cost, established and operating mining parameters and a processing recovery (refer Section 7) of 87%. The mining shapes are based in Indicated Mineral Resource material and are projected to provide a minimum break-even margin within incremental (where development exists) stoping panels.

9.3 Ore Reserve Assumptions

Ballarat Gold Project is an established operating mine.

The underground Probable Ore Reserve is based on several assumptions which include:

- Reserves lie within 10m of established development
- Current minimum mining widths
- Geological and geotechnical similarities to current mining areas
- Historical cost base for estimation of operating and capital costs
- Historical and budgeted metallurgical performance

The Probable Ore Reserve is not based on a fixed cut-off grade. It is costed on historical unit cost data, modified for changing activity levels and location within the mine.

9.3.1 Mining Method

Mining of the Ballarat Gold Project ore bodies by Castlemaine Goldfields Pty Ltd commenced in March 2011 with the first gold doré poured in September 2011. The principle mining method adopted is retreat longhole bench stoping, retreat blind uphole stoping and occasional cut & fill or modified drift & fill mechanised stoping.

The longhole bench stopes are extracted between levels based on geotechnical parameters for stope lengths, then backfilled with loose or consolidated (cemented rock fill ("CRF")) fill before the next retreating stope is extracted. Retreat blind uphole stoping, extracts panels of ore with no backfill horizon, pillars are left between the individual panels. Limited backfilling is completed. Cut & Fill and Modified Drift & Fill is mechanised production by the drill jumbo completing lifts above or adjacent to the previous development.

Subsequent lifts are backfilled with loose fill. These methods have been used extensively at Ballarat over the last three years.

All Reserves estimated in this report are amenable to these mining methods.

A minimum mining width for stoping of 2.5m is used at Ballarat East. This is based on the mine plan and existing production drilling equipment on site.

Cut-off Probable Ore Reserves are not based on a fixed cut-off grade. They are costed on historical unit cost data, modified for changing activity levels and location within the mine.

9.3.2 Cut-off Grade

Cut-off grades are not used to estimate Ore Reserves, they are more a generalisation of economic areas. There are numerous cut-off values dependent on the cost structures applied. A fully costed break even stoping cut-off grade of 2.90g/t is representative of a mine cut-off grade in an area of established development (incremental stope).

All reserves are fully costed within an economic model (Ore Decision Model) and based on the proportion of operation and/or capital development required for extraction. Thus the cut-off grade varies dependent on these factors, and no one cut-off grade has been used for the ore.

9.3.3 Exchange Rate and Gold Price Factors

Based on an internal review of 10 economic analysts the AUD/USD exchange rate has been set at $0.81 and a gold price of US$1,220 per troy ounce. This represents an Australian Gold price of $1,506 per troy ounce. This price is seen as representative of economic forecast for the period and Castlemaine Goldfields Pty Ltd has used these assumptions in the 2015-16 mine site budget.

9.3.4 Processing Method and Recovery

At the Ballarat Gold Project ore is trucked to the Woolshed Gully processing plant. The plant is located within 300 metres of the main access portal of the mine. The Ballarat East Mill consists of a primary crushing circuit with ore separation/treatment via primary gravity circuit with a secondary cyanide leach of the sulphide mineral tail. Probable Reserve ore mineralogy is similar to that already being treated in the process plant. The mill has been operating in current configuration since 2011.

In the 2015-16 budget there are plans to incorporate a ball mill into the primary circuit and a flotation cell into the secondary circuit of the mill to improve recoveries.

The metallurgical process is well tested technology.

Under the existing mill configuration the 2015 year to date (April14 to Jan15) recovery is 84%. Recovery is variable and is related to ore head grade combined with ore source location. The figure used is based on current plant performance with an allowance made for the projected improvement in recovery associated with the installation of a flotation circuit, which is currently being commissioned.

Metallurgical recovery of 87% has been applied for the Probable Ore Reserve.

No assumptions or allowances have been made for deleterious elements.
Current resource has a history of operational experience.

For detail of processing data refer section 11.

9.3.5 Sale of Product

Castlemaine Goldfields Pty Ltd sells to a gold refiner at “Australian spot market” prices. The company is paid on the refined weight of gold by the refiner at the “Australian spot market” price on the day of sale.

9.3.6 Hedging Program

No hedging program is in place.

9.3.7 Right to Mine

Refer to section 3.2 and 3.3.

9.3.8 Royalties

No gold mineral royalties are payable to the State, in Victoria, Australia.

However, as part of the acquisition negotiated in 2010, there is a 2.5% royalty on gold production payable to Newcrest Mining Ltd, capped at A$50M.

9.3.9 Company Tax

The current Australian Company Tax rate of 30% on net profit, payable to the Australian Federal Government is applicable.

9.3.10 Staff, Plant and Equipment

Ballarat Gold Project is an operating gold mine. All plant and equipment required for the mining and processing of the Ore Reserve is in place and operational. They are located on Castlemaine Goldfields Pty Ltd (Balmaine Gold, 100% owned subsidiary) held tenements and include but not limited to:

- Ballarat Goldfields Pty Ltd (BGF) electrical sub-station connecting the mine to the State power grid.
- Secure water supply for mining and processing.
- Processing Plant.
- Tailing dam facilities
- Mine development
- Underground power and dewatering infrastructure.
- Workshop facilities on surface.
- Ventilation fans.
- Administration complex
- Mining fleet to support a 250,000 tonnes per annum underground mining operation.
- Established labour force – Jan 2015, 142 permanent and 36 contractors
- Permanent all weather access to public roads.

9.4 Ore Reserve Estimate

9.4.1 Ore Reserve Input Data

Probable Ore Reserves are derived from the Indicated Mineral Resources, in accordance with the JORC Code. Reported Indicated Mineral Resources are within 10 metres of established development.

The indicated Mineral Resource is defined by 5 block models which are spatially defined by generally east-west striking locally termed "*crosscourse"* faults and vertically by north-south mineralised bedding or anticlinal parallel faults. Refer section 8.

9.4.2 Estimation

The underground Probable Ore Reserve is based on portions of the Indicated Mineral Resource model which are considered to be mineable based on historic unit cost, established and operating mining parameters and year to date mill recovery (refer Section 7 & 11). The mining shapes are based in Indicated Mineral Resource material that is projected to provide a notional breakeven margin on total costs.

Table 9.2 Ore Reserves summary, as of 31 March 2015

Ore Reserve

Category	Gross attributable to licence		Net attributable to issuer			Remarks
	Tonnes (thousands	Grade (g/t Au)	Tonnes (thousands)	Grade (g/t Au)	Change from previous update (%)	
Proved	-	-	-	-	-	
Probable						First report Reserve
Britannia Compartment	41	6.91	41	6.91	-	First report
Llanberris Compartment	9	4.98	9	4.98	-	First report
Sovereign Compartment	79	8.29	79	8.29	-	First report
Total	**129**	**7.61**	**129**	**7.61**	**100**	**Issuer owns 100% of the company**

9.4.3 Validation

The estimated tonnes and grade of individual Probable Reserve stoping shapes generated from the Indicated Mineral resource were validated by company peer review and external consultant.

The estimates were validated using

- The Vulcan computer program has an automatic check for validating wireframed triangulations that checks for closure, consistency and crossings.
- Tonnes and grade calculations have been replicated and confirmed by peer review and external consultant.
- The mine void model was checked against Probable Reserve stoping shapes to ensure pre-March 31 mined resources were not included in the estimation.
- Visual comparison of the model grades and corresponding drill hole grades show a reasonable correlation.
- Wireframe triangulations have been checked, including that the final geometric shapes looked sensible with respect to mining method.

9.4.4 Classification

The reported Reserve is for Probable Ore Reserves. The Probable Ore Reserves are derived from the Indicated Mineral Resources, and are not in addition to the resource.

9.4.5 Reported Ore Reserves

Table 9.3 Ore Reserves summary, as of 31 March 2015

Category	Gross attributable to licence		Net attributable to issuer			Remarks
	Tonnes (thousands)	Grade (g/t Au)	Tonnes thousands)	Grade (g/t Au)	Change from previous update (%)	
Proved	-	-	-	-	-	
Probable	129	7.61	129	7.61	-	First report Reserve
Total	**129**	**7.61**	**129**	**7.61**	**-**	**Issuer owns 100% of the company**

9.4.6 Production Reconciliation

CGT commenced gold production from the Ballarat East goldfield in late 2011. Initial production was based on exploration results, whereby length weighted drill hole intercept grades were assigned to modelled zones of mineralisation. Block models were not used for estimation of resources until the commencement of mining in the Llanberris Mako lode in March 2012.

Reconciliation of resource estimates with gold production

The Ballarat mine has reconciled gold production with resource estimates on a monthly basis. The amount of gold poured, the calculated tailings grade and the estimated change of the amount of gold retained within the processing circuit is compared with the estimated tonnage and grade of material mined.

The size of individual ore zones and the mining sequence does not allow sufficient material from any one source to be processed as an individual batch. Some material has been mined and processed from development outside the bounds of the resource block models. Over the past 11 months this material has contributed 4,980t (2.1% of declared ore mined) at an estimated grade (based on rock chip sampling) of 6.2 g/t Au for a total of 993oz Au (1.9% of declared ounces mined) delivered to the ROM. The amount of material mined outside the bounds of resource block models has decreased from previous years due to the increased delineation drilling.

Reconciliation process

The process for determining the allocation of tonnes and grades for gold production during a month is outlined below.

The estimated tonnes and grade mined are derived from the resource block model which has been diluted with mining parameters to give a “block model“ grade and tonnage for each mining area.

Sampling is carried out during the mining of both in-situ and broken material, this sampling is referred to as “grade control” sampling. This is used to monitor grade during mining operations and is not used for resource estimation.

The volume of material mined is measured by survey at the end of each month or at cessation of mining in an area. The volume multiplied by the relative density of 2.7 t/m^3 is used to determine the tonnage mined during the month. This relative density has been selected based on a combination of ore (2.65 t/m^3) and sediments (2.72 t/m^3) within material mined. The surface ROM stockpiles are either added or subtracted as the case may be, to the tonnage derived from survey data to generate the tonnage for the month which is referred to as the “declared ore mined” (DOM).

An estimate of the tonnage mined from each ore source during the month is generated by multiplying the number of truck loads from each source by a tonnage specific for each type of truck, the estimate is referred to as “hauled” tonnes.

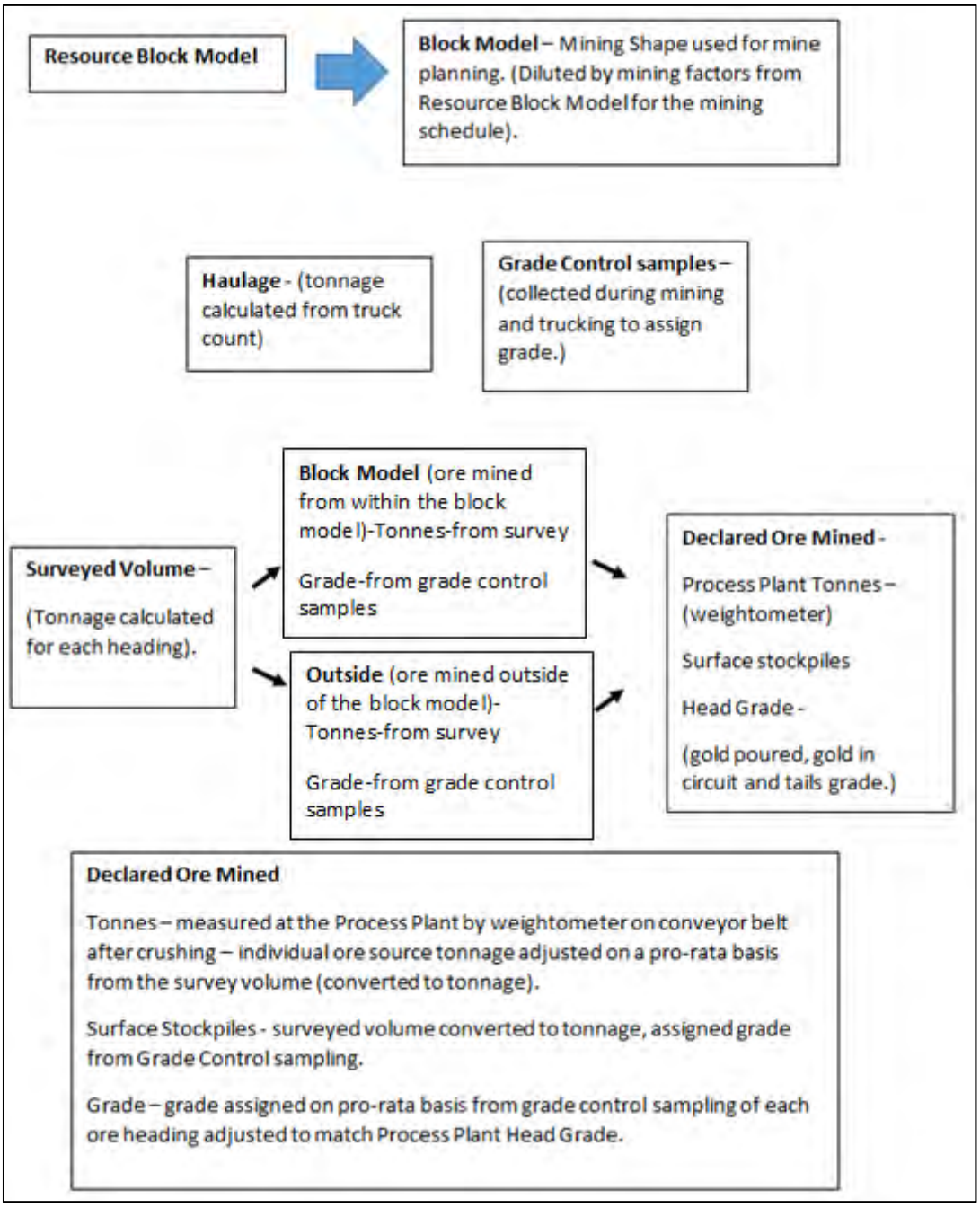
The “hauled” tonnes are used to allocate the proportion of the monthly DOM tonnages processed on a pro-rated basis to each ore source.

Differences in tonnage are relatively minor and are attributed to the inability to accurately survey some completed stoping areas due to ground failure and the inconsistencies between individual truck loads.

The “grade control” sample grades are used to allocate the grade to each ore source on a pro-rated basis to determine the DOM grade.

A flow sheet summarising the reconciliation procedure is given in Figure 9-1.

Figure 9-1 Flow sheet outlining the reconciliation process



Process plant sampling of crushed material

Sampling of material within the process plant is not used in the reconciliation process. A daily feed grade for the process plant is estimated from samples of material collected from a conveyor belt after crushing.

The samples are collected by hand, by taking a scoop of 0.5 kg to 0.7 kg of material from across the belt on an hourly basis. This is done three times and each scoop is placed into one of three separate sample bags, resulting in the collection of three composite samples for each shift. The samples results are averaged to give a grade for the shift.

It is noted that this approach to sampling is poor, with strong potential for low precision through high fundamental sampling and grouping and segregation errors, and high bias through delimitation and extraction errors. The results can only be regarded as indicative and are not used as part of the reconciliation process.

Comparison of resource estimates with process plant results

A comparison of the material mined from within the resource block model titled “block model” with the “declared ore mined”, described above, for the period April 2014 to March 2015 is shown in Table 9.2 below. Material mined from sources outside the block model has been excluded from this analysis so as to make a more direct comparison between estimated block grades and mined grades.

Table 9.2 Comparison of tonnes and grade mined from within the resource model “block model” and the DOM tonnes and grade. Figures exclude ‘not in resource’ mined tonnes

2014-2015	Block model		Declared ore mined (DOM)	
	Tonnes (t)	Gold (g/t Au)	Tonnes (t)	Gold (g/t Au)
April	14,030	7.10	14,857	5.55
May	15,659	5.59	17,336	6.69
June	15,914	7.2	17,730	5.04
July	20,920	6.05	22,347	5.39
August	26,980	5.86	26,018	6.24
September	20,794	6.85	21,313	8.92
October	18,370	6.63	18,891	9.33
November	27,786	5.03	27,576	7.45
December	19,818	5.23	24,387	6.58
January	20,845	6.35	21,027	5.91
February	15,131	6.01	15,848	7.61
March	20,773	7.06	24,226	5.37
Total	**237,020**	**6.18**	**251,556**	**6.67**

Note that there is a 14,536 tonne (6%) difference between the tonnage estimates in Table 9.2 for mined voids using the block model and the “Declared ore mined” (Mill reconciled) for development carried out during 2014-2015. This difference is likely due to a combination of factors including variability in survey pickups underground and on the surface ROM pad, in-accuracies in the Process plants throughput estimates including weightometer calibration issues and moisture content calculations and/or variability in the apparent relative densities of the mines ore sources. The 6% difference in estimates is considered acceptable given the range or variables involved.

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10 MINING

10.1 Mining Overview

The current mine covers a relatively narrow area approximately 400 m in width and five kilometre in length extending to a depth of around 700 m below the surface; beneath the historic Ballarat East goldfield workings. Much of the mine extends under the Ballarat residential area with operating restrictions placed around noise, dust and blasting vibration.

Primary access underground is via the Woolshed Gully decline, nominal dimensions of 4.6 m high and 4.6 m wide at a gradient of 1:6.5 down to 130 metre vertical depth, with the portal located at the southern end of the mine (Figure 10.1).

The decline system below the Woolshed Gully decline has been developed at nominal dimensions of 5.3 m high by 5.0 m wide and a gradient of 1:6.5. At approximately 1,200 m from the portal twin declines splitting into the upper Sulieman decline (approximately 1,900 m long) and the lower Woah Hawp decline (approximately 3,700 m long) that extends north to within 300 m of the Mining Lease boundary.

A number of internal declines (Prince, Sovereign, Llanberris, Britannia and Britannia West) are developed off the Woah Hawp decline to access the ore zones within each compartment.

Fresh “intake” air enters by two main routes into the mine, via the main haulage decline and by the 6.1 metre diameter concrete lined 318 metre deep Golden Point intake ventilation shaft. The mine operates on a through flow ventilation principal with air returning to surface through a series of internal return airways which connect to the Sulieman decline and then exit the mine via the 6.1 metre diameter concrete lined 129 m deep North Prince Extended shaft.

The mine for production and development is heavily dependent on auxiliary ventilation provided by forcing fan and duct ventilation systems. This provides flexibility to the operation but requires constant management of the auxiliary systems. The key issues being proper duct installation, leakage management, ensuring delivered airflows are at or above requirements for safe operation and ensuring adequate bypass airflow past the fans to ensure that adequate anti-recirculation requirements are met.

The underground mining operations including development drilling and ground support, blasting, excavation and haulage are carried out by CGT as an “owner operator”.

Production drilling is carried out by a separate contactor (Macmahon) that supplies underground production drilling services.

Exploration drilling is carried out by separate contractor (Deepcore Drilling) using mobile and skid mounted diamond drill rigs.

10.2 Mining Operations

Development of underground excavations is carried out using conventional drill and blast techniques with twin boom 1000V electric hydraulic drill jumbos used to drill blast holes in development faces and for the subsequent installation of ground support in the walls and backs of the excavation once the blasted rock has been mucked. Both jumbos operated underground are also used to drill longer holes for the installation and grouting of cable bolts in intersections or where structural wedges are identified.

The ground support design for mine development considers the expected prevailing ground conditions and service life of the excavation. For example the minimum support requirements for:

- Capital infrastructure/permanent access with a life span greater than two years includes galvanized primary support (split sets) and secondary support (full encapsulated rock bolts or cable bolts) and floor to floor surface support (typically 50 mm shotcrete)

- Waste access or ore development with a life span less than 12 months includes black or galvanized primary and secondary support and surface support less than 0.5 m from floor (typically mesh)

Rubber tyred diesel powered loaders and trucks are used to move broken rock (ore and waste) from development drives or stopes.

Development waste is preferentially placed in underground voids (development or stope) as backfill or trucked via the decline to the surface waste dump.

Ore from development or stopes is trucked via the decline to the surface ROM pad.

The current mine production plan is based on a combination of ore generated from the development along the strike of the ore zone, mechanised drift and fill (DAF) and longhole bench stoping. Geotechnical conditions and geometry of the ore bodies are highly variable and the mining method is selected to suit. Long hole stoping will be a combination of “up hole retreat” stopes with no backfill and stopes where a top and bottom access is present allowing the stope void to be backfilled. The bulk of future production is scheduled from three main areas - Llanberris, Sovereign and Britannia compartments.

Figure **10-1** **Mine plan view**



10.2.1 Backfill

Unconsolidated rock fill

Waste rock produced directly from waste development headings is used to support the hanging wall in stopes where crown access is available. Loose rock fill is used in preference to cemented rock fill when the length of the entire stoping panel is less than or equal to the maximum unsupported span.

Cemented rock fill (CRF)

CRF is a mixture of blasted waste development that has been mixed with a cement slurry to bind it.

The aim of backfill is to provide sufficient support to the surrounding rock mass once orebody extraction has been completed enabling continuous mining of the adjacent stope and ensuring full (100%) recovery of the

orebody. However, in order to fulfil its role as a passive support element and prevent excessive ore dilution, CRF must not undergo excessive failure during exposure.

The backfill support expectations can be defined by four critical responses.

- To provide regional stability (confinement) to the surrounding rock mass and in doing so, prevent unravelling of the sediments and quartz.
- To provide enough strength to allow exposure of both horizontal and vertical faces.
- To provide enough strength to retain unconsolidated fill in larger stopes

Be resilient to slot firing activities within close proximity and adjacent stopes.

At the present time, two CRF mixes have been specified that include:

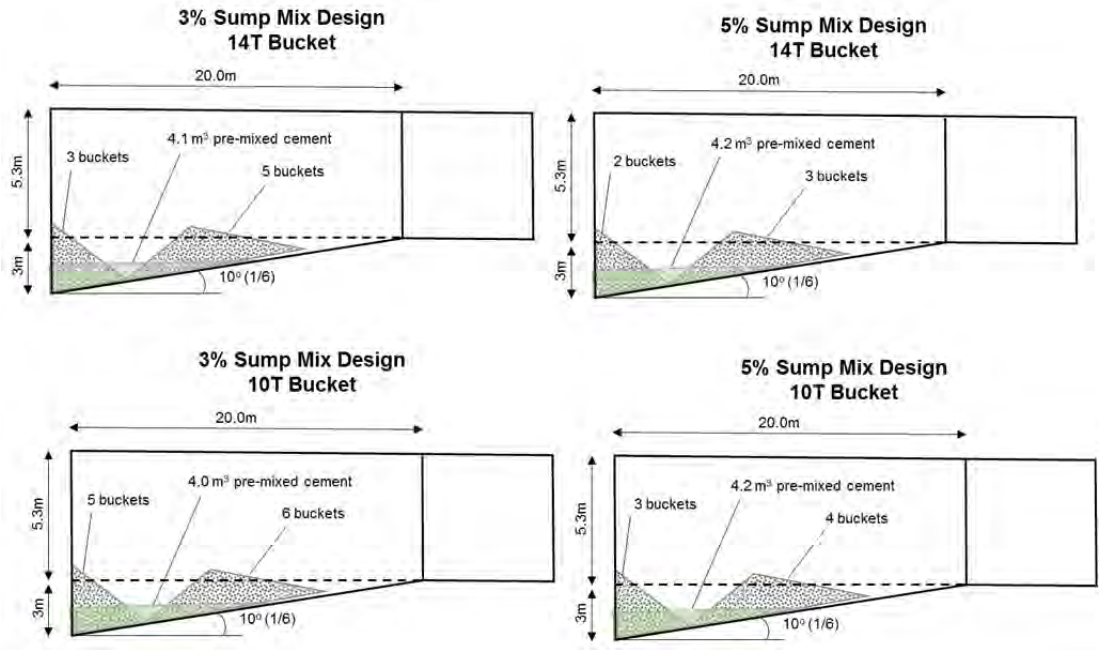
- 3% cement (weight) content when vertical stope exposure is required.
- 5% cement (weight) content when horizontal stope exposure is required.

Each of these mixes has been developed for use with both a 10 t and 14 t loader bucket capacity.

The cement slurry pre-mix is mixed with the waste rock underground at mixing bays located in close tramming distance to the stope requiring backfilling. The cement slurry, with a water cement ratio of 0.8, is pre-mixed by a batch plant on the surface and transferred to the CRF mixing bay by an underground agitator.

Using both the 10t and 14t loaders with bucket capacities of 7 m^3 and 5 m^3 respectively, the sump mix designs illustrated in Figure 10-2 and have been developed for 3% and 5% CRF product.

Figure 10-2 Design of sump mixing system for 3% and 5% mix of CRF product



The backfill delivery requirements at Ballarat have been estimated to be 224 m^3 per shift. This rate is consistent with current operational efficiencies, equipment and labour availability. Waste headings typically generate 93 m^3 of waste each cut. This equates to approximately 2 ¼ waste headings per shift required for CRF backfilling. In order to minimise any backfilling delay, a stockpile of loose rockfill is maintained underground in close proximity to each stope.

10.2.2 Mining fleet and machinery

Mobile equipment used underground are listed in Table 10.1.

Table 10.1 Current underground fleet

Type	Make/model
Loaders	Toro 1400 (LOS Remote)
	Sandvik LH307
	Toro 1400
	Toro 1400 (Tele Remote)
	Sandvik LH514
	Volvo L60E IT
	Volvo L120E IT
Trucks	Toro 45
	Toro 50+ ejector tray
	Sandvik TH550
	Toro 50D
Twin boom Jumbos	Tamrock Axera 7
	Tamrock Mini-Matic
Telehandlers	Dieci Zeus 33.11TA
Grader	CAT 140H
Agitator trucks	Toro 40D Agi
	Getman Agi
Water truck	Scania
Shotcreter	Jacon Midjet MK3

10.3 Mine Schedule

The 2015/16 ore production mine plan and associated development schedule is discussed in the following subsections.

10.3.1 Development

The planned development occurs in the three main mining areas – Sovereign, Llanberris and Britannia.

Development advance rates (capital waste, operating waste and ore development) targets to achieve between 310m and 320m per month.

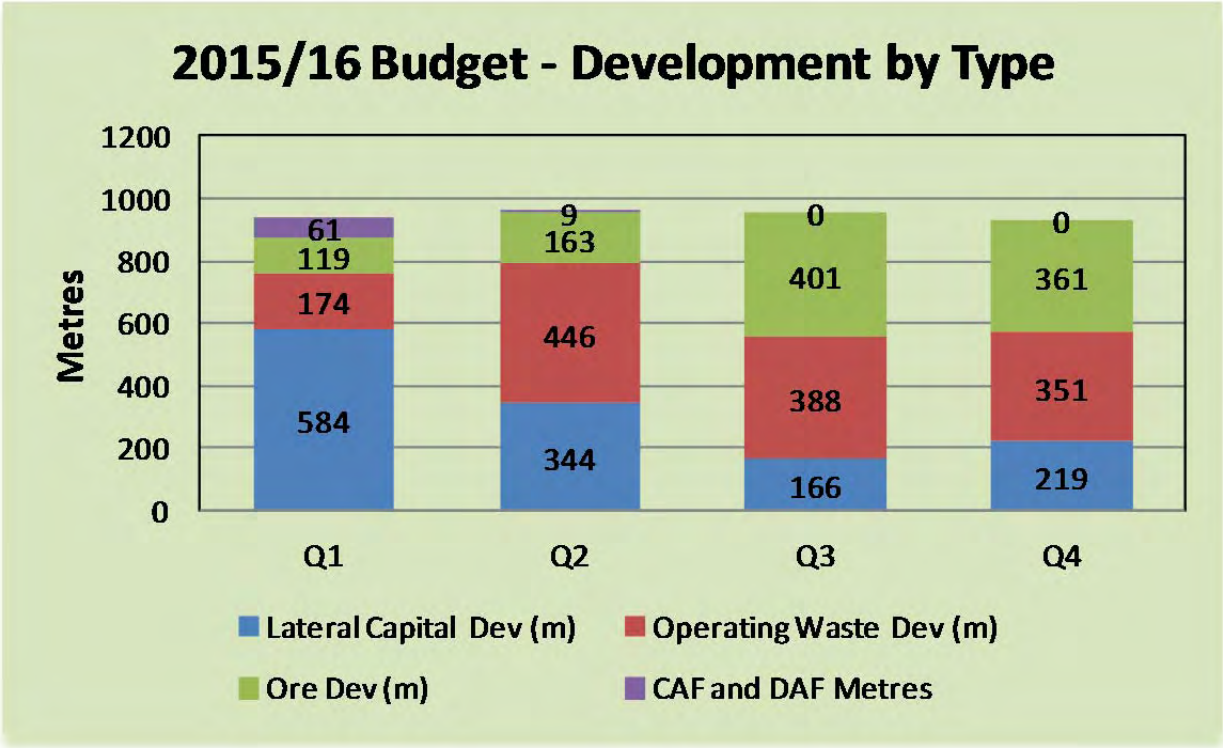
The 12 months 2014/15 year-to-date average has been 276.5m per month.

The anticipated improvement is expected to come from additional manning and a replacement jumbo.

Table 10.2 Development physicals by quarter during 2015/16

	Year 2015/16				
ACTIVITY	**Q1**	**Q2**	**Q3**	**Q4**	**Total**
Capital development (m)	584	344	166	219	1,313
Operating Waste (m)	174	446	388	351	1,359
Ore development (m)	119	163	401	361	1,044
Cut and fill development (m)	61	9	0	0	70
Total development (m)	**938**	**962**	**955**	**931**	**3,786**

Figure 10-3 Quarterly development break-down



10.3.2 Ore Production

The planned ore production occurs in the three main mining areas – Sovereign, Llanberris and Britannia.

The 2015/16 production schedule has 76% of the ore tonnes mined (188,000 tonnes at 6.91 g/t Au) from the March 2015 resource.

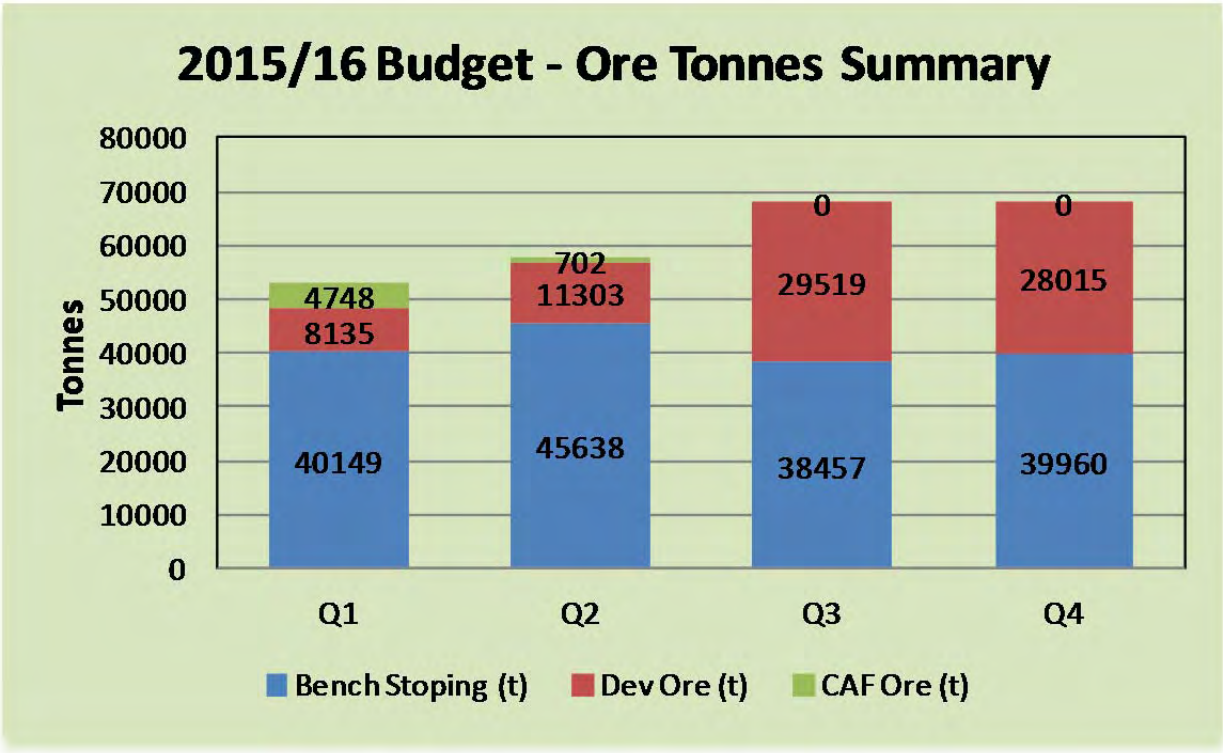
However, the resource is depleted during the 2016/17 forecast year, such that only 19% (50,000 tonnes at 6.26 g/t Au) of the forecast total of 246,000 tonnes comes from the resource; the remainder are conceptual targets that require drilling and exploration success to delineate.

The recent introduction of cemented rock fill into the stoping cycle is seen as a major improvement initiative to increase ore recovery (less pillars left) and to reduce lost production time and re-work required that has resulted from stope/hangingwall failures to date. The latter has also resulted in ore lost within stopes.

Table 10.3 Mine production physicals by quarter during 2015/16

	Year 2015/16				
Activity	**Q1**	**Q2**	**Q3**	**Q4**	**Total**
Development ore (t)	8,135	11,303	29,519	28,015	76,972
Cut & fill stoping (t)	4,748	702	0	0	5,450
Bench stoping (t)	40,149	45,638	38,457	39,960	164,204
Total ore mined (t)	53,032	57,643	67,976	67,975	246,626
Grade mined (g/t Au)	5.9	6.1	7.6	7.4	6.8

Figure 10-4 Ore tonnes by mining method



Bench stoping: 67% tonnes, Development: 31% tonnes, CAF ore: 2% tonnes.
Based on March 2015 resource estimate

10.4 Geotechnical and Hydrological Inputs

10.4.1 Geological Structures

Large scale (10m+) structures

The goldfield is structurally complex where numerous deformation events have created folds and major faults. There are four major fault types exposed in workings, which are described below.

West dipping faults

Major west-dipping faults are spaced between 50 m to 70 m apart and strike approximately north-south and dip at low to steep angles. Minor faults or splays have been observed between the major fault zones. The faults are typically identified by quartz veining and associated carbonate veining, arsenopyrite alteration and rotated cleavage. The fault gouge ranges from 1 cm to 50 cm thick with the quartz veining associated with the fault extending up to 10 m either side in the form of tension veins. The mineralisation occurs around the intersection of the west-dipping faults and vertical structures in different quartz vein configurations. Colloquially these faults are known as Leatherjackets because of their graphitic fault gouge.

Recent observations in the Llanberris and Britannia compartments have identified that in the western limb of the fold axes, the west-dipping faults appear as relatively minor bedding plane faults. They are commonly found on the boundary between large sandstone and shale lithological sequences.

Cross course faults

Cross-course faults crosscut the bedding and typically strike in a conjugate northwest, northeast or east-west orientation. The northeast set is dominant and displays a combination of normal dip-slip and dextral strike-slip movement. The majority of the faults are sub-vertical, and are identifiable by their puggy brecciated appearance. The faults typically tend to be more oxidised and water saturated than other fault types. Any quartz associated with these faults has been heavily fractured. The faults can be up to 40 cm thick with a zone of broken and weathered rock surrounding the fault from 1 m to 10 m wide. The weathering can extend well below the normal surface weathered domain and the fault conditions remain very weak at depth, typical cross-course fault characteristics are discussed later.

Bedding parallel faults

Bedding parallel faults typically occur within a siltstone unit or along the contact between sandstone and siltstone units. The faults can be up to 10 cm thick and appear as pug along the bedding, a thin clay veneer, or a thin film of slickenside quartz.

Axial planar faults

Fold axes related faults are typically distinguished by an intense axial planar spaced cleavage, quartz veining and minor faulting.

Minor (drive scale: 5m) structures

Joint set data has been collected from core logging and face mapping data over time, the table below summarises the historical structural set data. Note that the measurements refer to the mine grid and joint set data only, i.e. no fault measurements.

Table 10.4 Minor structure orientation

ID	Dip/dip direction
Bedding/cleavage	80°/270°
“Flat dipper”	20°/090°
Joint 2	80°/180°
Joint 3	60°/150°

Rock mass classification

Rock mass classification is completed on an as needs basis and follows the industry accepted methods of determining Barton's Q and/or Bieniawski's RMR values. This data is used for geotechnical design and stope stability assessments.

Table 10.5 and Table 10.6 show typical rock mass classification values for the various rock types and geological features found at Ballarat East.

Table 10.5 Typical dock properties – Q system

	RQD (%)	Jn	Jr	Ja	Jw	SRF	Q range	Category
Sandstone	60-100	6-9	1-1.5	1-4	1	2.5	0.7-10.0	Poor-Fair
Siltstone	20-40	9	1	3	1	2.5	0.3-0.6	Very poor
Ore	20	9-12	1	8	1	2.5	0.08-0.1	Extremely poor-Very poor
Faults	10	12	1	8	0.66	2.5	0.03	Extremely poor

Table 10.6 Typical rock properties – RMR[89] system

	Strength (MPa)	RQD (%)	Spacing (m)	Jt surface	Water	Ori	RMR	Description
Sandstone	7	17	15	20	10	-5	64	Good Rock
Siltstone	4	8	8	10	10	-5	35	Poor Rock
Ore	4	3	8	10	10	-5	30	Poor Rock
Faults	2	3	8	0	7	-5	15	Very Poor

Intact characterisation

Geomechanical laboratory testing is completed on an as needs basis. Several campaigns have been completed in recent years that include:

- 2003-2005: UCS testing at the University of Ballarat;
- 2004: University of Ballarat student (M Chaplin) completed work to determine the correlation between point load and UCS.
- 2011: suite of rock strength testing completed by Trilab.

The results are summarised in Table 10.7.

Table 10.7 Intact Rock Properties

Rock type	UCS (MPa)			Young's modulus (GPa)			Tensile strength (MPa)			Dry density (t/m^3)	Porosity (%)
	Min	Max	Avg	Min	Max	Avg	Min	Max	Avg		
Sandstone	51	98	76	62	88	78	9.1	21.7	13.9	2.7	1.2
Shale	8	25	18	62	118	82	-	-	-	2.8	1.8
Quartz	16	38	30	6	91	56	2.5	10.5	5.6	2.7	0.5

Historically, all of the UCS test work undertaken on siltstone/shale samples has failed along either cleavage or bedding that exhibit significant strength anisotropy. As a result of this, the estimated UCS of siltstone from experience is expected to be greater than that measured. Work is currently being conducted to characterise this variability/anisotropy.

Chaplin (2004) converted the average point load data for each rock type to a UCS. The result of this study gave conversion factors of 24 x $Is_{(50)}$ for sandstone; 22 x $Is_{(50)}$ for siltstone/mudstone; and 28 x $Is_{(50)}$ for shale. These figures are still considered preliminary.

Pre-mining stress

Several *in situ* stress testing campaigns have been completed in recent years and are summarised as follows:

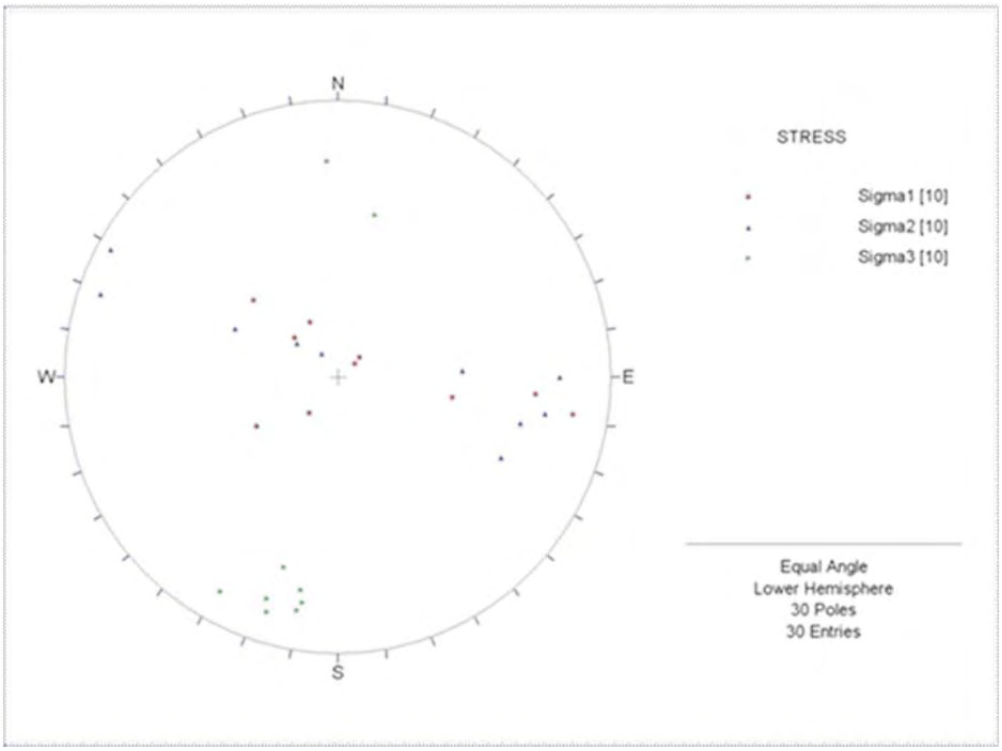
- Acoustic Emission (AE) test work at a depth of 450 m and 745 m below surface drilled from the Sulieman Decline. Completed in 2005 by the WA School of Mines;
- CSIRO Hollow Inclusion (HI) Cell test work in 2007, 2008 and 2009 was completed by Coffey Mining.

Results from the various test methods differ significantly. The HI Cell is considered to be the most accurate and acceptable test method. As a result of this only the HI results are used in numerical modelling analysis at the mine. The HI Cell stress measurement pole plot is shown in Figure 10-5.

The vertical to horizontal stress ratio appears higher at Ballarat than that typically experienced in other Australian mines. Stress magnitudes are typically very low and variable in magnitude and orientation throughout varying fault blocks due to the geological and structural complexity. In such an environment pre-mining stresses are likely to be locally variable and affected by nearby structures.

Squeezing ground conditions have been observed and monitored down to a vertical depth of approximately 670 m, predominantly in the north-south oriented development and where major structures (i.e. west-dipping faults, cross-course faults or bedding parallel faults) are in close proximity to, or intersect the development.

Figure 10-5 HI cell stress measurement pole plot (all tests)



Domains

There are five main geotechnical domains that are commonly observed underground, they are defined in the following sections.

Sandstone

Sandstone units are typically sparsely bedded. Typical uniaxial compressive strengths approach 100 MPa. The most common failure mode observed in drives developed in sandstone is sidewall slabbing on bedding plane contacts where sandstone is running sub-parallel to development drive walls. Discrete block/wedge loosening can also occur.

Siltstone

Siltstone units are more closely bedded and inherently weaker than the sandstone units. The most common failure mode observed in drives developed in siltstone is sidewall slabbing due to bedding planes and weak contacts running sub-parallel to development drive walls. Siltstones can also exhibit quasi-squeezing ground characteristics with sidewall convergence in weaker units due to rock mass dilation and shearing of bedding planes under vertical loads.

Cross-course faults

Cross-course fault material in the first 300 m below surface is normally extremely weathered and very weak. Below 300 m the faults are characterised by a relatively small pug zone, generally between 10 cm and 40 cm thick. This pug zone can be surrounded by a zone of highly jointed rock up to 10 m wide. When intersected, cross course faults generally unravel along the fault boundaries due to the unconsolidated nature of the clay and rock contained within the fault. The presence of groundwater is common in the fault zones, and tends to promote unravelling. Once supported, the cross course faults can increase sidewall convergence due to the very weak nature of the fault.

West dipping faults (including ore zones)

The west-dipping fault domain is typical in ore drives. The ground in and around the west-dipping faults within ore horizons will be made up of quartz veins and rotated sandstone and siltstone beds. Away from the ore zones these faults can be pug zones of 5 cm to 20 cm thick with a deformed zone up to 5 m thick. The west-dipping fault, when intersected perpendicular to the drive, commonly forms wedge failures in the backs of the drives. If driving parallel to the fault the rock mass can either unravel or create a long wedge in the drive back. The faults are typically weak and commonly trigger excessive sidewall convergence.

Weathered domain

The interbedded sandstone and siltstones are weathered to a minimum of 100m below surface. This weathering process significantly decreases the rock mass strength and convergence of the sidewalls is common in the extremely weathered to moderately weathered rock mass. The most common failure mechanism in the weathered domain is sidewall degradation due to wall creep.

Ground behaviour and failure mechanisms

The rock mass is considered to be generally weak when compared to most Australian underground metalliferous mines. The moderate to heavily jointed host rock (cyclic sandstones, siltstones and mudstones), an abundance of significant geological structures, and relatively low in situ stress conditions often provide challenging geotechnical conditions at Ballarat.

The most common ground behaviour and failure mechanisms observed at the mine include gravity induced unravelling of weak, faulted ground associated with structures – predominantly west-dipping faults (ore hosting) and cross course faults (Figure 10-6).

Dilation of weak laminated siltstones/shales and faulted ground predominantly in excavation sidewalls, typically resulting in wall squeezing (converging conditions) and stress-induced buckling, is also common (Figure 10-7).

Major geological structures can often have a significant impact on ground behaviour and the failure mechanisms mentioned above. Therefore knowing the location and expected severity (in geotechnical terms) of such structures is of particular importance to ensuring effective control of the ground.

Figure 10-6 Schematic of unravelling along faults

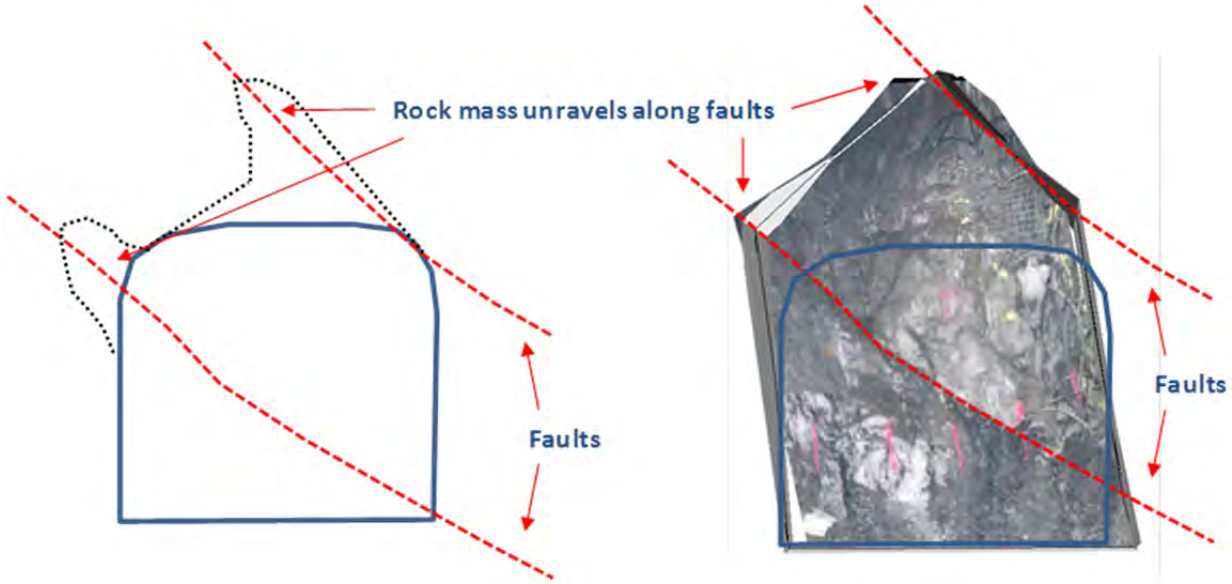
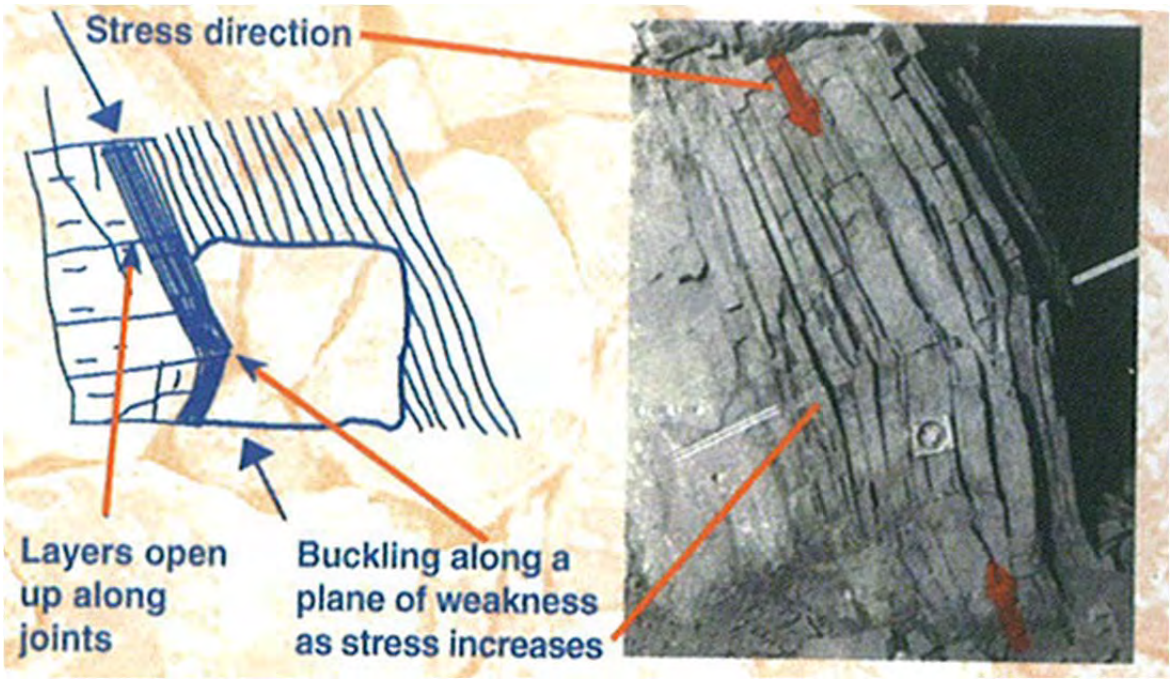


Figure 10-7 Schematic of buckling (after Nedin and Potvin 2000)



10.4.2 Hydrological Inputs

The following aquifer systems have been identified in the mine area:

Palaeozoic basement

The Palaeozoic basement aquifer is comprised of Ordovician rocks that generally have low primary porosity and permeability. The frequency and interconnection of joints, fractures, shears and faults control their capacity to store and transmit groundwater. The aquifer is either unconfined or can be semi-confined where overlain by low permeability sediments. Packer test work was undertaken in 2006 on one diamond drill hole to determine hydraulic conductivity. The rock mass surrounding the borehole had a moderately low hydraulic conductivity. These results were considered surprising considering the apparent fracturing in the core

photographs, indicating that rock mass permeability cannot be adequately judged purely from core logging. Test data and results are presented in Table 10.8.

Table 10.8 Hydraulic conductivity results

Test No.	Test section depth range (m)			Hydraulic conductivity ($x10^{-8}$ m/s)	Comment
	From	To	Length		
1	331	350	19	2 - 20	Possible 'reservoir' filling
			20 hour delay between tests		
2	278	350	72	2 - 9	Possible 'reservoir' filling
3	226	350	124	0.9 - 2	
4	173	350	177	0.3 - 0.7	'Reservoir' full
			3 hour delay between tests		
5	128	350	222	0.1 - 0.5	'Reservoir' full
6	76	350	274	0.3 - 0.4	'Reservoir' full
7	31	350	319	0.1 - 0.7	'Reservoir' full
			3 hour delay between tests		
8	31	350	319	1 - 4	Representative result

Post-Ordovician aquifers

Overlaying the Ordovician aquifers are varying thicknesses of alluvial sediments primarily associated with the valleys of the Yarrowee river, which drains the western side of the mine area and Canadian creek on the eastern side. Both drainage lines are approximately parallel to the north-south strike of the fold axes for much of their respective courses.

Water levels from previous dewatering of the underground workings indicate that there is little connection between this aquifer and the basement aquifer; therefore there is the possibility of perched water within the alluvials that must be considered during excavation and drilling.

10.5 Future Plans

The 2014-2015 budget aims to schedule ore from the March 2014 resource estimate. This is achieved such that 91% (217,000 t at 7.9 g/t Au) of the tonnes scheduled to be mined in the budget year are from the resource.

However, the resource is depleted during the 2015-2016 forecast year, such that only 20% (43,000 t at 8.6 g/t Au) of the forecast total of 218,000 t comes from the resource. This is scheduled to be mined in the period from April 2015 to October 2015.

The remaining 80% is a conceptual, based on the fundamental assumption that on-going exploration success will be achieved from drilling the exploration targets from within the existing narrow mine footprint and this will identify further ore sources to allow economic extraction in 2015-2016 at production rates, grades and costs similar to the 2014-2015 budget year.

11 PROCESSING

11.1 Processing Overview

The gold processing plant was constructed in 2005 and was purposely designed to suit the coarse grained nuggetty Ballarat ore with the aim of capturing gold and sulphides at the point of liberation without over-grinding. The gold and sulphide minerals are separated away from the waste using the difference in density.

Approximately 70% of the recovered gold is 'free' and is direct smelted into bars, with the other 30% present as sulphide bound gold which must be leached first.

The processing plant consists of a three stage crushing and screening plant, a gravity separation circuit with pressure jig separators, falcon concentrator and tables to recover both direct smeltable gold as well as sulphide concentrate, the latter requiring further processing via the Intensive Leach Plant (ILR).

At the end of 2014, a flotation circuit was being constructed with the aim of recovering fine gold and fine sulphides which are below the recoverable size range of the gravity circuit. Previously these losses were reporting to the tailings dam.

The gold processing facility has a capacity of around 250,000 t of ore per annum (at 50% rostered availability).

The processing plant can be split into two main stages, Crushing, Gravity & Flotation (Figure 11-1) and Leaching (Figure 11-2).

11.1.1 Crushing, Gravity and Flotation Separation

Three stages of crushing are used to liberate the gold and sulphide minerals prior to gravity recovery. The primary and secondary crushing stages are in a separate part of the circuit and operate on a batch basis. The crushing plant capacity is around 250 t per hour, shutting down at 2200hrs, which allows the crushed product to be stored in bins providing approximately 12 hours of feed supply to the downstream tertiary crushing and screening circuit.

The tertiary crushing and screening circuit operates on a continuous basis at a nominal rate of 70 t per hour and consists of two crushers (one duty and one standby) and two wet vibrating screens. The purpose of this circuit is to control the feed size of ore presented to the gravity jigs.

The high specific gravity of the gold containing minerals and the coarse grain size makes the ore particularly well suited to separation by gravity. Free gold particles and sulphide minerals which are liberated in the crushing and screening circuit are pumped to the jigs, where the mineral bed is fluidized with pulsated water. The high density gold and sulphides settle through the bed to form a concentrate whilst the lighter materials remain on top of the bed and are removed as tailings. There are three parallel trains of jigs, with two jigs in each train, and each capable of processing 25 t per hour. The jig tailings are processed through a Falcon concentrator to scavenge fine gold and then over a Sieve Bend Screen to separate the fine portion for Flotation and divert the oversize for tailings disposal.

The flotation circuit (under construction) aims to recover the fine liberated native gold and sulphides that the gravity circuit misses. Collector and frothing reagents are added to render the gold and sulphides hydrophobic such that they are collected on air bubbles and rise to the surface of the flotation cell to effect a separation. This gold containing froth (concentrate) is thickened to remove water before joining the sulphide component of the jig concentrate for leaching.

The jig concentrate is cleaned in two additional jig stages with the final concentrate delivered to the gold room for processing over Wilfley and Gemini tables. The sulphide component of the concentrate cannot be smelted directly and is tabled away from the free gold and sent to the leaching circuit.

11.1.2 Leaching

The gold associated with the sulphides is not refractory and can be leached directly with cyanide. The sulphide concentrates are first ground in a small ball mill to a size of 130 microns and sent to the cyanide leaching circuit. Only the sulphide concentrate which equates to approximately 5% of the total ore mass is leached. Hence the leaching plant differs from many gold processing facilities that employ CIP/CIL to leach the entire volume of ore.

Leaching occurs in two rotating drum leach reactors (Gekko ILR's) to ensure maximum contact between cyanide and the gold. The gold is dissolved into solution and then separated from the barren solids by thickening. The solution is pumped across a resin column where the gold is transferred onto an ion exchange

resin. The resin performs a similar role to carbon in a conventional CIP/CIL circuit. The resin is periodically stripped of its gold into a concentrated gold solution which forms the electrolyte feed to the electrowinning circuit. The gold is plated out of the electrolyte using an electrical current and deposited onto stainless steel cathode wool. The wool is periodically stripped of its gold and the gold is smelted in a gas fired furnace to form gold doré.

The residual cyanide remaining in the leach tailings is destroyed prior to disposal in the tailings storage facility. The cyanide destruction process is known as the INCO method and uses sodium metabisulphite and copper sulphate for the destruction of the cyanide complexes.

Figure 11-1 Simplified separation circuit flow diagram

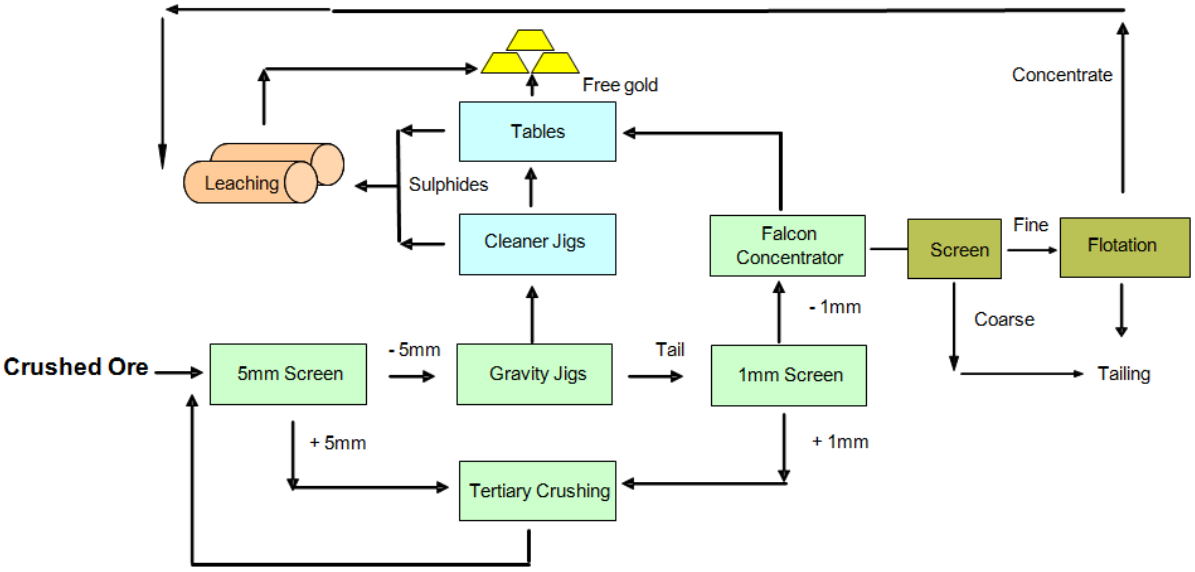
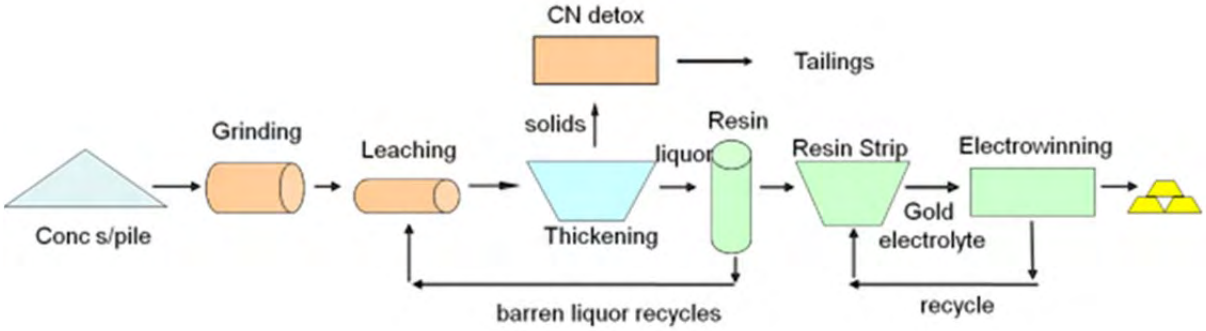


Figure 11-2 Simplified leach circuit flow diagram



11.1.3 Gold room

Free gold produced from the Wilfley and Gemini tables is smelted with fluxes in a gas fired furnace and poured as doré gold.

The gold sludge from the electrowinning cathodes are separately fluxed and smelted to produce doré gold.

11.2 Plant Operations

Nameplate capacity of plant: 500,000 tpa – 70tph at 92% availability

Crushing plant

The coarse grained nature of the Ballarat ore allows liberation of minerals at 1 mm to 5mm in size; hence no grinding mills are required for ore comminution. Instead the plant utilises three stages of crushing.

- Front end loader from ROM (run of mine) stockpiles to grizzly
- 14 inch [35 cm] static grizzly to ROM bin
- Hawk vibrating feeder discharging from ROM bin into primary crusher. Two thirds pan one third 75 mm grizzly. Variable speed to control feed rate to circuit
- Primary crusher – Jaques 42 by 30 single toggle jaw crusher
- Electromagnetic separator for tramp metal removal – M&O Equipment RCDF8
- Back up tramp metal detector interlocked with conveyor if tramp is detected
- Vibrating incline screen – Hawk, single deck with 30 mm screen mats
- Secondary crusher taking screen oversize. Allis Chalmers 60 inch [1.5 m] Hydrocone crusher, with gap set at ~20 mm
- Secondary crusher product and screen undersize combine and conveyed to fine ore storage bins. Secondary crusher in open circuit (no size control on product)
- Fine ore storage consists of two bins each of approximately 450 t working capacity
- Typical crushing rates 200 to 250 t per hour. Batch crushing as required with sufficient crushed ore storage to satisfy night time gravity circuit operation without having to crush (noise minimisation)

Tertiary crushing and screening

- Two fine ore bins each have two Hawk vibrating pan feeders to control feed of ore from fine ore bin – only one bin discharges at a time. Speed is variable and driven off belt weightometer output to control feed rate to circuit
- Milltronics Accumass BW500 belt weigher
- Feed material sampled here to provide head grade and moisture
- Static belt magnet to remove ferrous tramp metal
- Tertiary screening – two Joest wet horizontal vibrating screens, one with 5 mm aperture polyurethane screen panels and one with 1 mm aperture polyurethane screen panels. The 5 mm screen sets the entry size into the jig recovery circuit and is 2,450 mm by 6,100 mm in size and the 1.6 mm screen sets the exit size from the jig recovery circuit and is 2,450 mm by 7,320 mm in size. Both screens operate in parallel and in closed circuit with the VSI crushers
- Combined screen oversize (> 5 mm and >1.6 mm) conveyed to Vertical Shaft Impact crushers
- Static belt magnet protecting VSI from tramp metal
- Tertiary crushers – Duty Crusher is Auspactor VS300RR DD Vertical Shaft Impact crusher with 600KW of installed power. Standby Crusher is Auspactor VS200RR DD Vertical Shaft Impact crusher with 300 KW of installed power. Feed rate is 55 to 70 t per hour of new feed with 250% to 300% circulating load. Final product size P80 approximately 800 microns
- VSI discharge conveyed back to 5 mm vibrating screen – closed circuit
- Jig tailings also returns to Tertiary/screening circuit via the 1.6 mm screen
- Gravity jigs are in closed circuit with the VSI – jig tailings are re-crushed and re-jigged until particle size is <1.6 mm before they leave the circuit. This is to maximise gravity gold recovery and minimise over-grinding

Gravity circuit

- Minus 5 mm material reports to pressure jigs for co-recovery of coarse gold and sulphides (pyrite and arsenopyrite).
- Six Gekko IPJ 2400 In line Pressure Jigs on rougher scavenging duty (three trains of two) followed by single Gekko IPJ 1500 pressure jig on cleaning duty and IPJ 1000 on re-cleaning duty. Recleaner Jig concentrate is upgraded in a Gekko Spinner (ISP30) and again in the gold room by Wilfley and Gemini tabling to produce a direct smeltable gold concentrate away from the sulphides
- Jig circuit tail returns to VSI circuit via 1.6 mm screen. Oversize is recrushed to further liberate any composite gold/sulphides and returns to jigs (jigs in closed circuit with VSI's) and undersize reports via a cluster of four dewatering cyclones – Cavex 250CVX to Falcon concentrator to recover fine sulphide gold.
- Falcon concentrator is Model SB1350 batch machine. Designed for <1% mass pull to concentrate. Approximately 40 minute cycle time between concentrate dumps
- Falcon concentrate reports to gold room. Tailings to flotation.
- Cyclone overflow (dirty water) to dirty water storage tank for redistribution back through plant

Flotation circuit (under construction)

- Gravity tailings screened over DSM (sieve bend) to produce a 300 micron cut. Oversize to final tails (future ball milling) and undersize to flotation
- Three banks of 2 cell OK16 mechanically agitated flotation cells (96m3 total) equivalent to 45 minutes residence time at 70 tph ore (20 mins required so capacity for future upgrade)
- Flotation reagents: Collector Sodium Di-butyl-dithio-phosphate (non-xanthate) and Frother Polyfroth W34.
- Approx. 1% mass recovery to concentrate which is thickened and stored in an agitated tank ready for leach feed

Concentrate leaching circuit

- All sulphide concentrate from the recleaner jig tail, flotation circuit and gold room table tails contains fine gold which is non-refractory and readily leachable
- Concentrate can be stored in concrete bunkers, thereby divorcing gravity circuit and leaching circuit operations if required
- Concentrate is ground in a small ball mill to 130 microns and leached in a Gekko ILR10000 Intensive Leach Reactor to dissolve the gold, followed by two stages of Counter Current Decantation to generate a clean solution for the resin column feed. The leaching circuit runs at pH 10.5-11 as controlled by caustic addition
- The resin column uses Aurix100 ion exchange resin for gold adsorption
- Loaded resin is batch stripped with 40 g per litre caustic at 50 C into a concentrated solution from which the gold is electrowon via conventional means
- Barren resin returns to the resin column. In every sixth strip cycle, the resin is regenerated with a dilute acid wash before returning to the column
- Leach circuit tailings are detoxified in an INCO cyanide destruction circuit to produce cyanide code compliant tailings prior to co-disposal with the gravity circuit tailings. Concentrate production (leach tailings) only represent 5% to 7% of the total tailings mass produced

Process pumps

All slurry pumps in both the gravity and leaching plants are Warman pumps of various sizes

Chemicals/reagents used

The intensive leaching process uses sodium cyanide (Orica mini-sparge unit on site) and Caustic as a pH regulator. The cyanide destruction circuit uses sodium metabisulphite and copper sulphate. A small amount of flocculent is used in the CCD circuit. The flotation circuit uses a collector and a frother reagent as detailed above

Process control system

Allen Bradley PLC system and SCADA based operator control interface

11.3 Performance

The Ballarat Processing plant for the previous year 2014-2015 is detailed in Table 11.1

Table 11.1 Process plant performance

Month	Milled ore (t)	Head grade (g/t Au)	Recovery overall (%)
Apr 2014	17,640	5.5	81.2%
May 2014	16,931	5.8	81.0%
Jun 2014	18,588	5.3	83.2%
Jul 2014	21,055	5.5	82.1%
Aug 2014	19,518	6.7	84.8%
Sep 2014	21,784	8.1	80.6%
Oct 2014	19,693	8.6	82.7%
Nov 2014	23,620	8.6	82.3%
Dec 2014	21,798	7.2	86.4%
Jan 2015	21,657	7.0	85.6%
Feb 2015	24,149	5.9	83.5%
Mar 2015	24,231	7.2	84.9%
Total	250,664	6.8	83.6%

11.4 Metallurgical Test Work

The primary focus areas of metallurgical test work and plant optimisation over the 2014/2015 year were:

- Laboratory test work to understand the response of gravity tailings to flotation. This involved both screened gravity tailings for flotation feed as well as full tailings grinds under a variety of reagent regimes. From this work, a design for a large scale flotation circuit was derived which ultimately led to project approval and construction of a flotation circuit using redundant equipment from the RAV8 nickel mine in WA. Recovery improvements of approx. 4% are expected.
- Laboratory flotation test work was also conducted to determine whether additional recovery was possible by grinding the coarse fraction of gravity tailings to liberate locked gold prior to flotation. This work was positive with further recovery gains of 2-3% possible. As a result, a commitment was made to purchase a redundant ball mill from the Wodgina mine site in WA with a view to reinstalling the mill at Ballarat. This work will be a focus of the coming year.
- Size by grade analysis of gravity tailings to understand the nature of the gold losses. The increasing supply of ore from the Sovereign areas of the mine appeared to contain finer gold which had a detrimental effect on tailings grades. The information received from this work provided the impetus for the flotation and ball mill (future) projects.
- Numerous optimisation works were carried out to improve the fine gold recovery performance of the jigs – increasing pulsation rates, ragging optimisation, installation of tailings samplers – all to improve gravity gold recovery and combat increasing losses of fine gold to tailings.

- The trial and subsequent purchase of a Wave Table to assist with the gravity tabling of ultra-fine (-150 micron) concentrates to yield additional gold previously considered un-cleanable and un-smeltable.
- The trial of higher strip solution flowrates in the resin stripping circuit to increase the gold transfer capacity and hence throughput of the leaching circuit. Double strips have since been adopted as being the new norm.

11.5 Metallurgical Accounting

- Metallurgical accounting is performed based on gold produced, gold in tailings discharge and gold in circuit (GIC) - including concentrate stockpiled.
- Samples are taken by hand sampling of solid and slurry streams.
- Monthly plant recovery calculations are reconciled against indicated gold in feed compared to actual gold in feed (gold produced, gold in tail and GIC).

11.6 Future Plans

There is opportunity for ore from other mines, either CGT or externally operated, to be processed at Ballarat given the name plate capacity of the processing plant and the forecast production schedule from the mine.

There is consideration being given to installing a ball mill circuit to liberate fine gold still locked within the quartz particles and thereby increase gold recovery

The installation of a flotation circuit and ball mill paves the way towards reprocessing of the tailings already stored within the TSF – approx. 1million tonnes – approx. 17,000 oz of recoverable gold. This is contingent on permits and approvals being granted, a wall lift on the new tailings cell being in place and a suitable reclaim methodology being adopted.

12 INFRASTRUCTURE

12.1 Mine Infrastructure

Site Infrastructure includes the following:

- Administration buildings
- Process plant administration building
- Amenities building
- Operations building
- Heavy vehicle workshop
- Light vehicle and electrical workshop
- Stores building
- Core shed
- Gold room

The process plant comprises crushing and screening, gravity separation, concentrate leaching, and gold electrowinning and smelting circuits.

Other infrastructure includes:

- Shotcrete batching plant (owned and operated by others)
- Reverse Osmosis water treatment plant (not presently used)
- Laboratory (equipment owned and operated by others)
- Chemical storage compound

Electrical infrastructure includes:

- Off-site Elsworth Street substation
- Four substations for surface operations
- Seven underground substations
- MCC’s and distribution boards

Underground infrastructure includes:

- Woolshed Gully underground including a total of approximately 13 km of roadways extending up to approximately 700 m below the surface with
- Approximately 13 dewatering pump stations and associated pipework
- Ventilation systems
- Explosives magazine
- Refuge chambers.

Water storage and distribution:

- Process water pond
- Tailings storage facility (TSF)
- Mine Water treatment comprising aeration tanks, settling and polishing ponds
- Fire system including dedicated hydrant system and hose reels and underground explosives magazine sprinkler system

12.2 Power

CGT purchases electricity directly from the national electricity grid under a contracted supply agreement with ERM Power. This agreement is due to expire at the end of July 2017 and is for the supply of 22 GWh pa of electricity along with associated services such as metering.

Power is supplied from the local 66kV grid to the Company owned Elsworth Street substation (commissioned in 2008) which consists of incoming gas filled circuit breakers, 66kV/11kV, 5MVA transformer and 11kV switchroom.

From there, power is fed underground to the nearby North Prince Extended ventilation shaft leading to the RMU No 1 (ring main unit) situated in the First Chance decline approximately 150 m directly below the surface.

RMU No 1 feeds a total of seven underground substations each consisting of incoming protection fuses, 11kV/1000V, 1.5MVA transformers and switchboards located in the First Chance, Sulieman, Sovereign and Woah Hawp declines.

RMU No 1 also feeds Substation 1 situated at the surface.

Substation 1 feeds:

- The Process Plant main substation (Sub 3) via a 11kVA/433 V, 2MVA transformer
- Surface mine substation (Sub 2) which supplies the mine surface infrastructure via 500kVA step-down transformer including workshops and offices
- Substation 6 which then feeds via 500kVA and 750kVA step-down transformers:
    - RO plant (not used)
    - Workshops
    - Laboratory building
    - Concrete batch plant

12.3 Water

Ballarat has a positive water balance due to the dewatering of the historic mine voids and groundwater entering the underground mine. This water is either used on site for dust suppression or the processing plant with the remainder being discharged to the environment under strict EPA discharge licence conditions

Recycled process water from the TSF flows into the lined process water dam which is topped up from the mine dewatering system. This is a zero release closed water circuit between the TSF and the process plant.

The mine dewatering system comprises approximately 13 “Mono” pump stations, which are fed by submersible Flygt pumps in decline face and settling sumps and handles approximately 1.6 ML per day. Mine water passes through two parallel trains of aeration tanks where blowers force air bubbles to help form iron, arsenic and manganese precipitates which separate into the first of three settling ponds. The treated water then passes through wetland/polishing ponds before discharge to the nearby Yarrowee river.

12.3.1 Potable and waste water

The main mine operation is connected to a reticulated potable water supply managed by Central Highlands Region Water Authority (CHW). CHW is a regional water corporation providing high quality drinking water, sewerage, trade waste and recycled water services to customers in Ballarat and surrounding towns; it is one of the state-owned water businesses operating under the guidance of the Victorian Water Act.

Potable water is primarily used within the administration and change house for drinking and cleaning. A small amount of potable water supplements the concrete batching plants rain water supply for the manufacturing of the shotcrete used for ground support.

The mine is a relatively small user of potable water with an average annual use of 6 ML.

Water from within the mine is currently pumped from the active mine areas to the surface via a rising main in the Main decline. As the water flows through the pipeline at a pump rate of ~18-27 L/sec, poly-ferric sulphate is added to the water at a rate of ~3 L/hr to aid in coagulation and settlement of iron, arsenic and manganese. The mine water being anaerobic is then directed to an aeration tower where it is saturated with

oxygen that results in the formation of ferric ions and the precipitation of ferric hydroxide. The arsenic ions are oxidised from a valency state of predominantly As^{3+} to As^{5+} assisting their adsorption onto the Fe-oxyhydroxides. Manganese ions undergo a similar oxidation, but with varying rates of precipitation. Settling of the iron/ arsenic rich precipitate occurs in three HDPE plastic lined ponds with a total volume of 9 ML. The ponds are located down a slope utilising gravity to maintain water flow. Mine water is cascaded through each pond to provide the maximum resonance time before the water is directed to the 36 ML main mine water dam.

All storm-water run-off generated at the mine site is directed to siltation dams. The 36 ML main mine water dam receives the greater majority of this runoff, in which it mixes with treated mine dewater. A 20 ML surge dam and an eleven cell linear wetland and 6 ML sediments dams manages storm-water from the TSF intercept drain. A lesser amount collects in a 3 ML settling pond located below the waste rock dump. This dam's second purpose is to catch seepage from the waste rock dump. The 36 ML siltation dam discharge is directed via a cascade and gully to a 12 ML constructed wetland for final polishing and biological removal of a percentage of remaining metals. This wetland system is of particular importance for the final removal of manganese through biological activity. Final discharge occurs at the downstream end of the wetland, where the water flows through a monitored V-notch weir before entering Yarrowee river.

12.4 Transport

The site located in the suburbs of Ballarat is easily accessible via sealed public roads.

The City of Ballarat Planning Permit requires Heavy Vehicles (those being in excess of 10 t) shall only be permitted to enter and leave the site between 0700 and 1800 from Monday to Friday (except where emergency repair works are required to be undertaken to maintain the on-site operation).

A combination of sealed and graded roads provides good light vehicle access from the main gate to buildings and plant areas throughout the site. Separate haul roads for underground haul trucks and light vehicles provide access to the underground portal and ROM pad.

12.5 Staffing

The workforce for the mine operation is predominately sourced from Ballarat and the surrounding areas.

The mine operates 24 hours per day, seven days per week working an even time roster based on two 12 hour shifts with a seven days on/ seven days off.

The processing plant operates 24 hours per day, three and a half days per week.

CGT employs 143 mine staff (as of end of March 2015) as shown in Table 12.1, and a number of part time and full time contractors that totals 55 (as of end of March 2015) as shown in Table 12.2.

Table 12.1 Ballarat mine staff personnel numbers

Department	Position	Number
Geology	Geologist	10
	Field assistants	4
Mining	Operations	58
	Fixed plant maintenance	7
	Mobile plant maintenance	19
	Technical services	7
Processing	Operations	17
	Maintenance	3
	Support	3
Shared services	Environment and community	2
	Administration	8
	Safety and security	5

Table 12.2 Ballarat mine contract personnel numbers

Position	Number
Diamond drilling	24
Production drilling	4
Mining operations	5
Fixed plant maintenance	18
Mobile plant maintenance	2
Cleaning	2

12.6 Accommodation

The mine is residential based and no accommodation for employees is required.

13 SOCIAL, ENVIRONMENTAL, HERITAGE AND HEALTH AND SAFETY MANAGEMENT

13.1 Social Management

All exploration and mining conducted by CGT is undertaken in a manner to ensure minimal impact on the existing land use, environment and community and there is comprehensive Environmental Management and Community Engagement Plans in place.

Strong historic links and relationships built on open transparent communication, mutual trust, respect and commitment with CGT's community and stakeholders has allowed the Ballarat project to operate alongside the community and beneath the City of Ballarat since 1993.

CGT's commitment to the community is demonstrated in company policies, engagement framework and procedures. The community engagement strategy involves a number of techniques including; open days, newsletters, opportunistic face to face, door knocks, public meetings including an Environmental Review Committee and by participating in community partnerships through a sponsorships and in-kind donations programme.

Methods used for monitoring and evaluation of the community engagement performance include community perception surveys, consultation, face-to-face discussions, engaging an Environmental Review Committee and analysing community feedback and concerns on a regular basis.

13.2 Environmental Management

CGT has a strong commitment to providing responsible stewardship of the environment over which it has control or influence over.

CGT has Environmental and Community Policies, an Environmental Management Plan (EMP) and is developing an Environmental Management System (EMS) framework.

Operational and environmental risks associated with the exploration and mining licences are addressed in the EMP. An Environmental Risk Register has been developed to identify the broad aspects/hazards and impacts associated with the various activities that are either currently undertaken, or planned to be undertaken. The register is reviewed regularly.

A summary of the conditions associated with the various permits and licences related to the project has also been completed.

The environmental monitoring programme has been designed and implemented to address environmental risks, Work Plan commitments and Work Plan conditions. Environmental monitoring results are compared against regulatory limits and reported to the various state and federal regulatory authorities and the Ballarat Environmental Review Committee (ERC).

Annual environmental performance is included in the CGT annual report with copies provided to state regulatory authorities and the ERC.

The key environmental management issues for the Ballarat mine site are noise, ground and surface water, air quality, waste rock and tailings disposal, blast vibration, land management and traffic management.

13.2.1 Noise

Noise control has been an integral part of the design of the Ballarat mine site including locating all infrastructure away from residences and below the natural surface within a sound shell to minimise the noise impact of the operation.

Noise limits specified within the Work Plan are that levels within 10 m of any residence shall not exceed:

- 54dB(A) between 0700 and 1800 and at specific monitoring sites 50dB(A)
- 48dB(A) between 1800 and 2200 and at specific monitoring sites 44dB(A)
- 43dB(A) between 2200 and 0700 and at specific monitoring sites 42dB(A)

The process plant design included significant cut back to lower the crushing circuit, as well as earth bunds and sea containers to meet all noise limits.

The maintenance and improvement of a tree buffer around the site is also designed to reduce noise as was the placement of waste rock to provide a solid sound barrier.

13.2.2 Blast vibration

Regular review of blast performance allows for any potential improvements of blasting practices to be implemented as the underlying geology may change as underground mining proceeds.

The site operates well within the licence limits set out in the Work Plan;

- Day time Maximum 10 mm per second
- Annual Average 5 mm per second – 95% of all firings to be below
- Night time limit 3 mm per second

The community is informed of current and planned mining activities as well as likely impacts of blast vibration via the Community newsletter, with more specific notices sent to residences in areas likely to be affected by blasting.

A timely response to complaints, installation of a ground vibration monitor on the property and communication of the recorded data, independent inspections for property damage if appropriate, changing firing times for specific locations are part of the measures used to respond to any community concerns.

13.2.3 Air quality

Air emissions from the mine activity primarily consist of particulates (PM_{10} and total suspended particles) and to a lesser extent carbon dioxide, carbon monoxide, nitrous oxides, sulphur dioxide and hydrogen sulphide.

Dust (and its components) resulting from surface activity has also been identified as an issue that may affect air quality. Modified work practices along with the use of water carts and polymers have been successful mitigation tools.

Monthly depositional dust monitoring occurs at 8 locations surrounding the mine site and monitoring of the North Prince Extended ventilation shaft emissions occurs annually.

13.2.4 Waste rock

The chemical nature of the waste rock generated at the Ballarat site has been analysed for acid mine drainage (AMD) generating potential. Tests indicated that most of the rock is inert and will not pose a risk of producing AMD when exposed to air and water.

13.3 Heritage Management

Mining has defined the character of the township of Ballarat and the works within the mine site represent the on-going development of an area founded upon a famous mining heritage.

Potential heritage management issues and impacts that arise as a result of the mine site operations include:

- disturbance of sites of European heritage
- disturbance of sites of cultural significance to Indigenous people

Heritage sites have been identified and documented and the site management processes are in place to ensure there is no future disturbance.

13.4 Health and Safety Management

CGT operates a Safety Management System that provides a framework for the management and continual improvement of Health and Safety in all activities including the integration of Health and Safety standards and practises into day to day operations.

This includes specific policies, procedures and plans for:

- Inductions, training and supervision
- Hazard identification, risk assessment and the control of risk
- Consultation and communication
- Contractor management
- Injury and incident reporting and investigation
- Emergency response and crisis management

Inductions, training and supervision

All employees and contractors undergo the general Site Introduction and area specific inductions.

Operational level training is competency based with assessed skills and competency profiles documented in an individual's training matrix.

On the job supervision and the training of supervisors is focused on the active motivation of people to work in a safe and responsible manner.

Hazard identification and control of risk

Risk management systems include:

- Major mining hazards (fatal risk) safety assessments
- Formal risk assessments
- Management of change
- Hazard and incident reporting
- Incident investigation
- Step-backs and job safety analysis

All employees are responsible for using hazard and risk management tools to assist with ensuring the effective day to day management of hazards.

The risk ranking of hazards is used to priorities corrective actions.

Major mining hazard safety assessments are reviewed internally by subject matter experts and senior management, externally by contract high hazard industry safety consultant and audited by WorkSafe Victoria.

Consultation and communication

Consultation and communication occurs via a number of formal and informal forums including:

- Roster start up meetings
- Cross-shift handover discussion between operators
- Pre-shift (muster) meetings
- Safety toolbox meetings
- Employee consultation group (ECG)
- Participation in major mining hazard safety assessments, risk assessments, investigations and workplace inspections

Contractor management

CGT is firmly committed to the provision of a safe and healthy workplace for contractors and subcontractors in accordance with its occupational health and safety policy; this includes:

- Ensuring that contractors and subcontractors work in a healthy and safe manner and are not harmed, or do not cause harm to others, while working on CGT work sites
- Promoting measures to prevent injury and illnesses by insisting on safe systems of work, safe equipment, use of trained and competent personnel and the maintenance of Health and Safety supervision and records
- Selecting contractors for work in terms of their ability to meet or exceed CGT's health and safety standards

Injury and incident reporting

Hazard and incident reporting protocols are continually reinforced at safety and operational meetings.

Corrective actions are recorded and tracked through to "close out" in the incident/hazard database.

Emergency Response

The Ballarat site has a documented emergency response and crisis management plan in place, with roles and responsibilities designated to staff. The plan is subject to document control policies and is reviewed regularly.

All employees, contractors and visitors are made aware of the site emergency response arrangements during inductions. Periodic emergency response drills involving both desktop and evacuation exercises are held to maintain familiarity with emergency response arrangements.

The site has 24 hour incident management capability.

Training for mine rescue team members includes training in manual firefighting equipment, breathing apparatus, rope rescue, search and rescue. Some members of the team are also members of the local Country Fire Authority (CFA).

There is an incident control centre and an emergency response training room on-site that the external emergency service organisations (e.g. Police, CFA and Ambulance) are familiar with.

14 MARKET STUDIES AND CONTRACTS

14.1 Market Overview

Assumptions on gold price used in the 2015-2016 budget for the A$:US$ exchange rate and US$ gold price to determine the resultant A$ gold price were based on 10 broker consensus gold and foreign exchange forecasts.

A$:US$ 2015 average = 0.81

US$ per ounce 2015 average = $1,220 /oz

Figures of exchange rate (Figure 14-1), US$ gold price (Figure 14-2) and A$ gold price (Figure 14-3) for the last five years are shown in following pages.

Figure 14-1 Exchange rate (AUD/USD) for last 5 years (2010-2015)



Figure 14-2 Gold price 2010-2015 US$/troy ounce

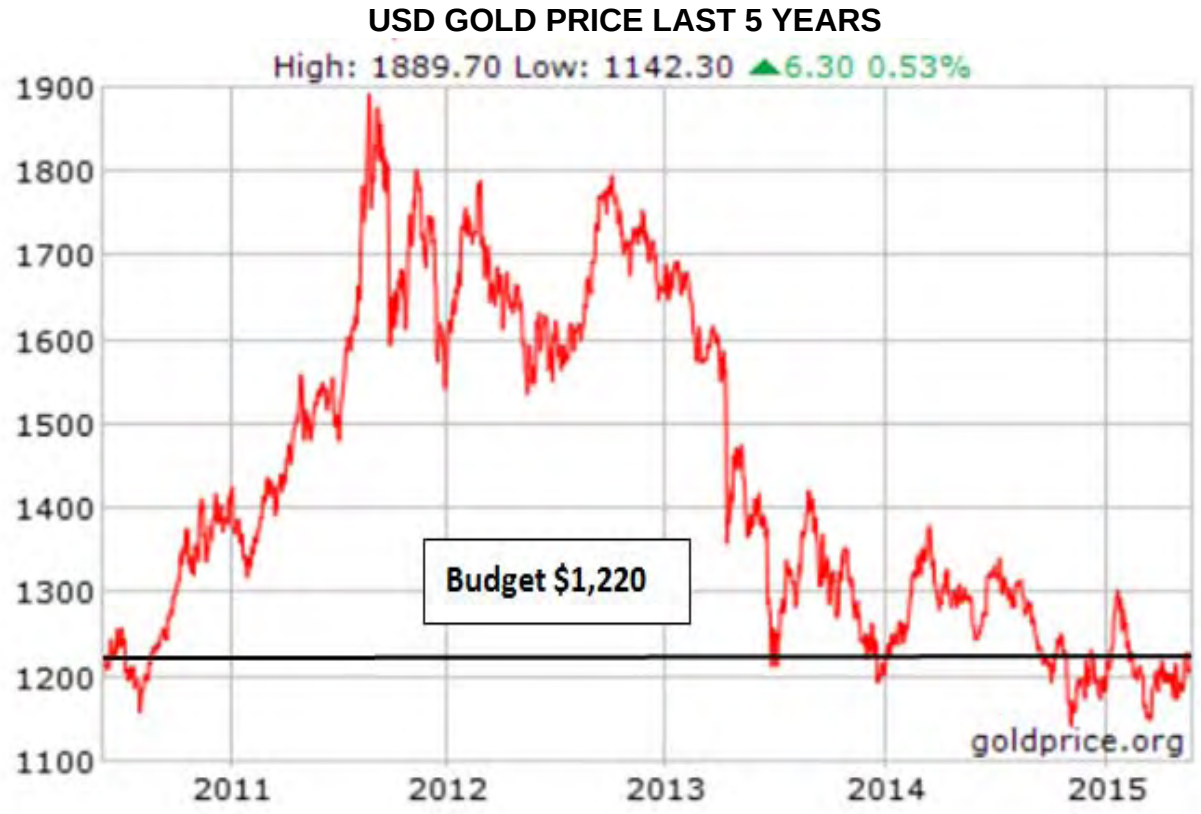


Figure 14-3 Gold price 2010-2015 A$/troy ounce



14.2 Sales Contracts

Contracts are in place for the refining and transportation of gold doré.

The gold doré produced on-site is shipped to the West Australian Mint for refining and the resultant outturn is credited to the CGT metal account.

The transportation of gold doré to the West Australian Mint is carried out by a Security firm.

Gold sales are based on the spot Reuters Inter Bank quoted price.

15 FINANCIAL ANALYSIS

15.1 Historical Financial Analysis

All currency values are in Australian Dollars unless otherwise denoted. The actual 2014-2015 operating expenditure by department is detailed in Table 15.1.

The mined ore tonnes for the 2014-2015 year totalled 257,336 t and the operating cost per tonne mined averaged A$161. The unit cost by department per tonne of ore mined is shown in Table 15.2.

Gold ounces sold for 2014-2015 totalled 45,503 oz Au, and the site actual Ballarat mine cash operating cost per ounce sold averaged A$894. The operating cost per ounce sold is given in Table 15.3.

The average gold price received per ounces for the 2014-2015 year was A$1,440 and revenue from bullion sales totalled A$65.5M. Operating statistics for the 2014/15 year are included in Table 15.4.

Table 15.1 Ballarat mine actual operating costs by department. Currency A$

	Total
Geology (excluding UG exploration)	7,046,029
Mining (excluding capital development)	24,327,085
Processing	6,239,826
HSE, Admin & Security	3,828,040
Total	**41,440,980**

Table 15.2 Ballarat mine actual unit operating cost per tonne mined. Currency A$

	Total
Geology (excluding UG exploration)	27
Mining (excluding capital development)	95
Processing	24
HSE, Admin & security	15
Total	**161**

Table 15.3 Ballarat mine operating cost per ounce sold. Currency A$

	Total
Operating cost per ounce sold	894

Table 15.4 **Operating statistics for the Ballarat mine and process plant for the 2014-2015 year**

	Unit	Total
Ballarat mining production and mining costs		
Lateral development - capital waste	Metres	583
Lateral development - operating waste	Metres	771
Lateral development - ore	Metres	1,816
Lateral development - total	Metres	3,170
Cut and fill stoping	Metres	149
Total development metres		3,319
Ore mined – total	Tonnes	257,336
Mined grade	g/t	6.7
Operating cost - mining	A$'000	24,326
Unit cost mining per tonne ore mined	A$/t	95
Ballarat processing and costs		
Ore milled	Tonnes	250,664
Head grade	g/t	6.8
Total gold recovery	%	84
Operating cost – processing	A$'000	6,240
Unit cost processing per tonne ore milled	A$/t	25
Ballarat gold production		
Gold produced	oz	46,039
Gold sold	oz	45,503

15.2 Forecast Capital Costs

Capital mine development totals A$6.7M in the 2015-2016 budget year to support the development to and extraction of the scheduled ore sources, at a budgeted cost of A$5,099/m of advance.

Site sustaining capital and productivity improvements total A$6.8M, with the larger items including:

- Maintaining (through replacement or rebuilds) some of the underground mobile equipment
  - Three trucks (A$1,536,000)
  - Jumbo (A$350,000)
  - Loader (A$300,000)
  - Shotcreter and Agitator Truck (A$250,000)
  - Replacement and additional Integrated Tool Carrier (A$190,000)
- Pumping, electrical and ventilation infrastructure (A$492,000)
- Ball Mill dismantling, transportation to site and commencement of civil and mechanical design (A$1,420,000)
- Commence preparations for the tailings storage facility cell 2B lift including stockpiling of waste (A$475,000)

The capital expenditure for the gold recovery improvement project involving the construction of a small flotation circuit has been completed in the 2014-2015 financial period. Wet commissioning of the flotation circuit was ongoing at the close of the financial year.

The purchase of a second hand ball mill was completed in the 2014-2015 financial year with dismantling and transportation to site of the ball mill scheduled to occur during the 2015-2016 budget year. Engagement of a project cost estimation group to determine a brief scope of work and obtain quotes so as to collate information and generate a full project costing will also occur over the first half of the 2015-2016 budget period. It is envisaged that the installation of a ball mill into the existing processing plant will better assist in the recovery of the fine gold. The installation of a ball mill will also provide the opportunity to analyse the potential retreatment of tailings from within the existing Tailings Storage Facility.

15.3 Forecast Operating Costs

The 2015-2016 budget expenditure across all departments has been worked up from cost element/first principles basis (Table 15.5). Current costs have been used where known (salaries and wages, and key consumables – power, cyanide, diesel, explosives, ground support, tyres etc.). The operating and capital development cost by expense element is summarised in Figure 15-1.

Table 15.5 Ballarat mine operating costs by department

	Total (A$)
Geology (excluding UG exploration)	8,357,390
Mining (excluding capital development)	27,670,457
Processing	7,775,830
HSE, Admin & Security	4,713,249
Total	**48,516,926**

Figure 15-1 Ballarat mine cost breakdown

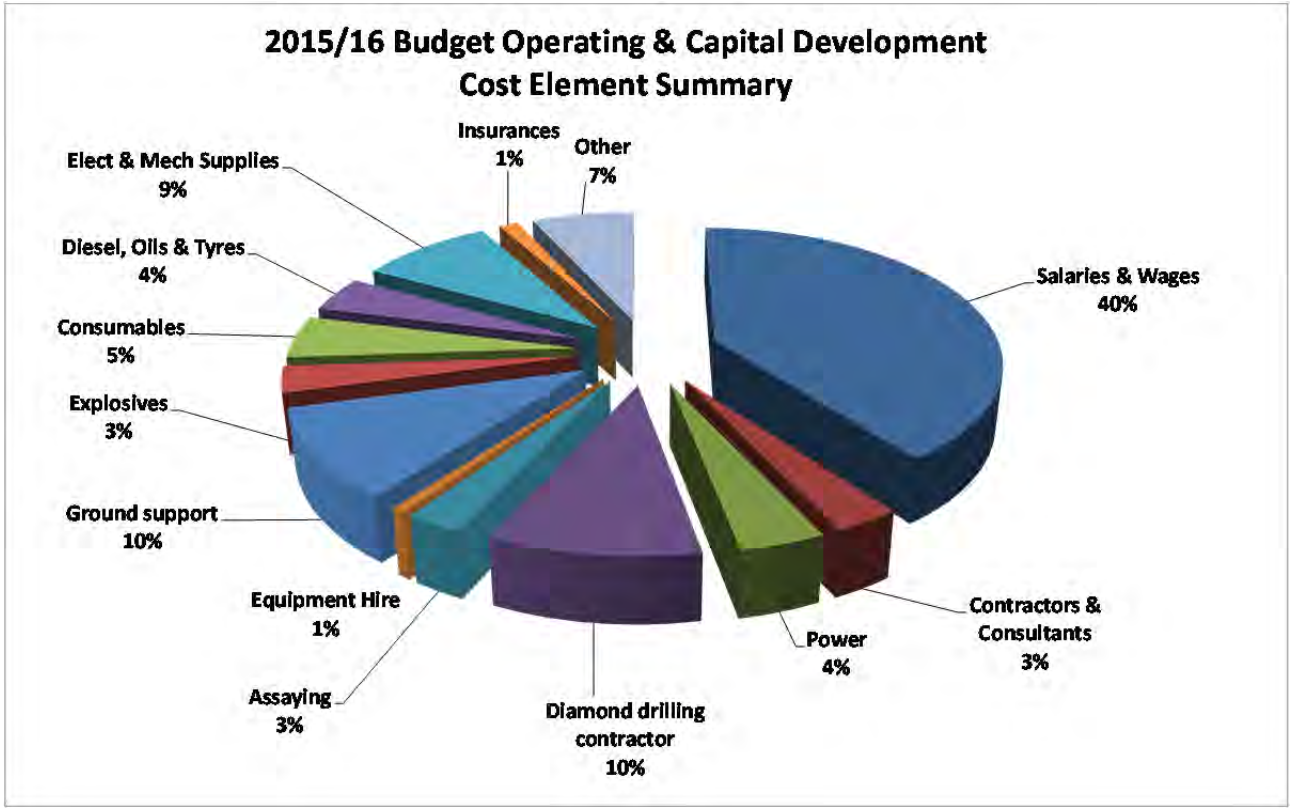


Table 15.6 below details the respective unit operating costs per tonne of ore mined.

Table 15.6 Unit operating cost per tonne mined by department

	Total (A$)
Geology (excluding UG exploration)	34
Mining (excluding capital development)	112
Processing	32
HSE, Admin & Security	19
Total	197

16 RISK ASSESSMENT

16.1 Risk Rating Definitions

Project risks have been assessed on the basis of likelihood of occurrence, and on the consequence of an event occurring, resulting in a risk matrix which is used to define the level of management responsibility. The tables below define the categories used in this report to assess likelihood, consequence, and risk rating within the context of the group (Table 16.1 to Table 16.3).

Table 16.1 Categories and definitions used to assess likelihood

Likelihood	Definition
Almost Certain	Event is expected to occur in most circumstances (easily); more than 1 event every year
Likely	Event will probably occur in most circumstances (should); about or less than 1 event per year but more than 1 event per 5 years
Possible	Event might occur at some time (conceivably); less than 1 event per 5 years but more than 1 event per 10 years
Unlikely	Event could occur at some time (conceivable but rare); about or less than 1event every10 years
Remote	Event might occur only in exceptional circumstances (theoretical) or is unlikely to occur

Table 16.2 Categories and definitions used to assess consequence

Consequence	*Definition*
Severe	Very large financial loss (>A$50M) of total assets; death or serious injury to multiple persons; major loss of plant resulting in >6 months loss of production capability; toxic environmental release off-site with serious detrimental effect
Major	Major financial loss (A$20M- 50M) of total assets; death or serious injury to multiple persons; extensive loss of plant resulting in 3–6 months loss of production capability; off-site environmental release with detrimental effect or on-site release with detrimental effect
Moderate	High financial loss (A$10M- 20M) of total assets; serious injury to multiple persons; moderate loss of plant resulting in 1 week to 3 month loss of production capability; on-site environmental release contained with assistance without causing long-term detrimental effect
Minor	Medium financial loss (A$1M- 10M) of total assets; minor injury to one or two persons; minor loss of plant resulting in 1 day to 1 week loss of production capability; on-site environmental release immediately contained without long-term detrimental effect
Insignificant	Low financial loss (<A$1M) of total assets; no injuries; less than one day loss of production capability; no environmental impact

Table 16.3 Risk rating

	Severe	Major	Moderate	Minor	Insignificant
Almost Certain	High Risk	High Risk	High Risk	Medium Risk	Medium Risk
Likely	High Risk	High Risk	Medium Risk	Medium Risk	Low Risk
Possible	High Risk	High Risk	Medium Risk	Low Risk	Low Risk
Unlikely	High Risk	Medium Risk	Medium Risk	Low Risk	Low Risk
Remote	Medium Risk	Medium Risk	Low Risk	Low Risk	Low Risk

16.2 Risk Assessment

The categories used to assess risk for this project reflect the parameters defined in the JORC Code to assess mineral resources and ore reserves.

CGT has undertaken a semi-quantitative risk assessment of risks identified for the Ballarat mine Mineral Resource estimate (Table 16.4).

Table 16.4 Ballarat East mineral risk profile

Factor	Risk	Comment
Bulk density	Low	The current value of 2.65 t/m^3 is reasonable and based on core measurements. Some local bias may exist where the proportions of host rock versus quartz change and minor effects of sulphides. Variability is unlikely to be greater than ±5%. Areas of vuggy quartz will cause a local drop in density, but are believed to be <5% of total quartz.
Sample representivity	High	In-situ sample representivity is likely to be low given the coarse-gold high-nugget nature of the mineralisation. Samples (e.g. drill holes and face samples) likely represent the low-grade fine-gold population relatively well, with poorer representation of the coarse gold population.
Sample collection, preparation and assaying	Medium	Poor core recovery and quality is the notable issue at Ballarat, resulting in core loss and sub-sampling challenges (e.g. core splitting). Thus different sample support and assaying methods, together with the effects of core loss impart sampling error. The coarse-gold nature of the ore exacerbates potential sampling error, particularly through Preparation Error and Grouping and Segregation Error. Application of whole core LeachWELL assaying is good practice.
QAQC	Low	Historical and recent QAQC indicates reasonable assay quality, though this does not ameliorate representivity issues. Current practices and procedures in the Gekko Laboratory are appropriate, up-coming NATA certification will allow higher level of external confidence in the laboratory. Minor CRM and blank issues are prevalent from time to time and ongoing checking needs to be continued.
Geological data and model	Medium-High	General geological control is reasonable on 15-30 m drill sections. Knowledge of historical and recent development aids interpretation. Understanding of small-scale local continuity issues which control variability of tonnes and grade is improving but continuous modelling of exposures will improve future predictions. Best resolution of geological continuity and ore zone complexity is only gained after development. On-going detailed mapping is also improving the quality of the structural and computer models, Detailed reconciliation from production stopes will enable determination of geological models.
Grade estimate	High	The grade estimate bears a high uncertainty due to a high-nugget effect, sampling and data uncertainties. The current estimate relies on a global grade for each domain based on relatively wide-spaced data. No local estimate is possible. Estimation block size is broadly appropriate to the drill spacing, but does not relate to any SMU size. The application of cut-off grades is highly problematic. On a block by block basis estimation error will be high. The IDW squared estimation method is outdated practice. The estimate is not rigorously spatially controlled as search ellipses are not controlled by variography. The current estimation approach under-calls grade.
Tonnage estimate	High	The current global estimate is reasonable, given that volume is based on a Vulcan model constrained by drill data and geological interpretation. Estimation block size is broadly appropriate to the drill spacing, but does not relate to any SMU size. The application of cut-off grades is highly problematic. On a block by block basis estimation error will be high.
Resource up-rating and addition	Medium	This year is the first to include an uprated resource. The methodology and logic behind the determination is consistent with JORC (2012). Due to the overall nature of the Ballarat goldfield and the parameters employed to determine an Indicated Resource the total amount of Indicated as a proportion of the overall resource will remain relatively low. A two-year plan is highly

Factor	Risk	Comment
		dependent upon exploration success.
Economic factors/reasonable prospects for economic extraction	Medium-High	Economic decisions based on Inferred Mineral Resources carry a high uncertainty. The project has appropriate infrastructure and plant in place. Mining costs, parameters and methods are now determined as a result of two years continuous mining. Project viability is sensitive to gold price and operating costs.
Metallurgy/mineral processing	Low	The plant is designed to cope with Ballarat coarse-gold ore. It can achieve a recovery of around 84-87%. Plant capacity is well within mining rates. There is an opportunity to increase plant recovery with the addition of a flotation circuit. Introduction of new circuit should reduce risk associated with recovery
Accuracy of the resource estimate	High	On a global basis, the CPs believes the accuracy of the grade and tonnage estimate should be within ±25% annually based on general experience of this style of mineralisation. Tonnage and grade reconciliations show a higher level of accuracy compared to previous years. Resource risk would be lowered for a closer drill spacing (potentially <10 m by 10 m) to inform the model, though this will be very costly. Current accuracy is considered to be in the range ± 5-20%.
Stope extraction	Medium	Global utilisation of average historic percentages for stope dilution and stope recovery should be avoided. The mine contains a number of structurally different mineralised shapes, and a database containing the over-break and recovery figures for each style of mineralisation mined will allow more accurate scheduling and planning going forward.
Geotechnical Ground Conditions	Medium	Moderate to poor geotechnical ground conditions can have potentially significant effected on production rates in the both development and extraction cycles.
Social, political and environmental risk	Low	Given the project location in Australia, these matters are considered to be low risk. CGT however needs to ensure that the local community are kept on-board given the location of the operation under a city.
Overall rating	**High**	**The current resource estimate carries high uncertainty and risk. This risk is principally related to high geological and grade variability. This rating is marginally reduced by the inaugural inclusion of Indicated Mineral Resources and Probable Reserves along with the previous "Inferred Mineral Resource" category.**

The Ballarat gold project carries an overall high risk. This risk principally relates to geological and grade variability, however the inaugural introduction of Indicated Mineral Resource and Probable Reserves adds a higher confidence level to the forward planning process. Ongoing production reconciliations will be important in determining degree of accuracy of the resource model at smaller 'stope' scale. This rating is reflected in the fact that 76% of the total ore feed for the 2015/16 period is derived from material within the March 2015 Indicated and Inferred Resources, whilst during the 2016/17 forecast year, only 19% of the forecast tonnage comes from the resource; the remainder are conceptual targets that require drilling and exploration success to delineate.

17 INTERPRETATION AND CONCLUSIONS

The Ballarat underground gold mine is owned and operated by Castlemaine Goldfields Propriety Limited, a wholly owned subsidiary of LionGold Corporation Ltd. CGT holds an exploration licence which covers the historic Ballarat East, Ballarat West and Ballarat South goldfields. This area includes two mining licenses which covers the Ballarat mine site, process plant and tailings storage facility, and the Ballarat South goldfield. The Ballarat mine is located beneath the city of Ballarat.

Gold mineralisation is found within narrow (less than 2 m thick) quartz veins associated with a series of major west-dipping faults which traverse the goldfield. The distribution of gold within these quartz veins exhibits a high- to extreme-nugget effect and the presence of coarse, often visible gold particles (>1 mm in size).

CGT has completed an update of its Mineral Resource estimate for the Ballarat mine. Resources have been estimated and are reported in compliance with The JORC Code 2012. The resource consists of mineralisation within six discreet lodes. Each lode is represented by a series of mineralisation wireframes. Tonnage and grade values have been estimated based on 513 diamond drill holes drilled between 2009 and 2015. Six block models have been created to estimate each of the lodes defined by CGT. Wireframes were constructed of geological domains within each of the lodes and were used to constrain the block model. An inverse distance squared estimation algorithm was applied, with composite top-cut grades selected using statistical analysis of the distribution of grade within each domain. Continuous selective mining may not be achievable due to the high-nugget effect and the resources are therefore reported at a 0 g/t Au cut-off. Domains containing estimated gold grades of less than 4 g/t Au are excluded from the resource as they are considered unlikely to have a reasonable chance of eventual economic extraction based on costs and gold price at the time of estimation.

The project has excellent infrastructure, including surface buildings, a fully operating plant, a fleet of mining vehicles (e.g. light vehicles, trucks, jumbos, etc.) and underground decline access to development. Production areas are accessed via the 1,205 m long Woolshed Gully decline and the 3,715 m long Woah Hawp decline, which has reach a point about 690 m below the portal and 300 m south of the mining lease. Overall, the current mine extends 3,422 m from the portal to the end of the decline. The entire underground network comprises some 19 km of tunnels.

The 2015-2016 budget aims to schedule ore from the current resource (Table 1.2). This is achieved such that 76% (188,000 t at 6.9 g/t Au) of the tonnes scheduled to be mined are from the current resource. The resource is depleted during the 2015-2016 forecast year, such that only 19% (50,000 t at 6.3 g/t Au) of the forecast total of 246,000 t come from the resource. The remaining 81% is based on the assumption that on-going exploration success will be achieved from drilling the exploration targets from within the existing mine footprint and this will identify further ore sources to allow economic extraction in 2016-2017 at production rates, grades and costs similar to the 2015-2016 budget year.

Three diamond drill rigs operate underground on a 24/7 basis, producing around 5,600 m of drill core per month. CGT has, over the last two years, demonstrated its capacity to replace resources depleted for mining. The existing infrastructure allows quick exploitation of areas identified during drilling and over the next 12 months.

Probable Ore Reserves have been defined at Ballarat, as a result of the establishing of the inaugural Indicated Mineral Resource. The presence of additional resources to support the reserves is the result of better understanding of the grade distribution and structural setting of mineralisation as well as close-spaced drilling to continue to resolve geological and grade continuity, in particular a high- to extreme-nugget effect of gold grade. In addition, localised variations in lode geometry are present. The project has appropriate infrastructure and plant in place. Mining costs, parameters and methods are now determined as a result of two years continuous mining. Project viability is highly sensitive to gold price and operating costs.

18 RECOMMENDATIONS

A number of recommendations are made in order to improve the quality of future Mineral Resource estimation. They are as follows:

- Continue on-going geological studies to understand the nature of the mineralisation, in particular controls on grade distribution.
- Implement a formalised management sign-off process for validation of logging and sampling carried out by core logging geologists.
- Determine a methodology to automate the validation of diamond drill hole collar surveys using Vulcan software.
- Undertake a rigorous resource estimation optimisation study to include:
    - Use of de-clustering in statistical analysis of sample grades.
    - Use of variography to determine spatial relationships.
    - Use QKNA to optimise parent block size and estimation parameters.
    - Investigate the use of kriging (or variant thereof) as an alternative estimation methodology.
- Increase the volume of density samples and investigate potential to construct a density block model to improve tonnage estimates.
- Continue to refine reconciliation procedures.

In relation to mining:

- On-going review of stoping methods and seek opportunities for improvement where possible.
- Continued rigorous ground control and monitoring, and control of additional mining dilution where possible.
- Reconciliation of mining dilution and over-break by ore style should be implemented in order for over-break and dilution numbers for specific mineralisation styles to be included into scheduling.
- Investigate potential economics of extraction of <2.5m wide zones using alternative mining methods.

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20 DATE AND SIGNATURE PAGES

I, Mr Peter de Vries, do hereby consent to the public reporting of the Ballarat gold project Mineral Resource and release of the Qualified Persons Report entitled "*Annual QPR for the Ballarat Gold Project, Australia for the Year Ended 31 March 2015*". I have given and have not withdrawn prior to lodgement, my written consent to be named in any Announcement as a person responsible for this Mineral Resources statement and to the inclusion of this statement in the form and context in which it appears.

I certify that I have read the Qualified Persons Report and that it fairly and accurately represents the work for which I am responsible.

Based on the requirements of the Singapore Exchange Securities Trading Limited Practice Note 4C, I am a Qualified Person. I am also a Competent Person as defined by the JORC Code (2012), having five years of experience that is relevant to the style of mineralisation and type of deposit described in the report, and to the activity for which I am accepting responsibility.

Dated: 31st May 2015

Peter de Vries

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Mr Peter de Vries
BAppSc (Geol), MSc (Min. Econ), MAusIMM, MAIG

I, Philip Petrie, do hereby consent to the public reporting of the Ballarat gold project Mineral Resource and release of the Qualified Persons Report entitled "*Annual QPR for the Ballarat Gold Project, Australia for the Year Ended 31 March 2015*". I have given and have not withdrawn prior to lodgement, my written consent to be named in any Announcement as a person responsible for this Mineral Resources statement and to the inclusion of this statement in the form and context in which it appears.

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Dated: 31st May 2015

Philip Petrie

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Philip Petrie
MAusIMM

I, Matthew J. Hernan, do hereby consent to the public reporting of the Ballarat gold project Mineral Resource and release of the Qualified Persons Report entitled "*Annual QPR for the Ballarat Gold Project, Australia for the Year Ended 31 March 2015*". I have given and have not withdrawn prior to lodgement, my written consent to be named in any Announcement as a person responsible for this Mineral Resources statement and to the inclusion of this statement in the form and context in which it appears.

I certify that I have read the Qualified Persons Report and that it fairly and accurately represents the work for which I am responsible.

Based on the requirements of the Singapore Exchange Securities Trading Limited Practice Note 4C, I am a Qualified Person. I am also a Competent Person as defined by the JORC Code (2012), having five years of experience that is relevant to the style of mineralisation and type of deposit described in the report, and to the activity for which I am accepting responsibility.

Dated: 31st May 2015

Matthew J Hernan.

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Matthew J Hernan
MAusIMM

I, Esteban Valle, do hereby consent to the public reporting of the Ballarat gold project Mineral Resource and release of the Qualified Persons Report entitled "*Annual QPR for the Ballarat Gold Project, Australia for the Year Ended 31 March 2015*". I have given and have not withdrawn prior to lodgement, my written consent to be named in any Announcement as a person responsible for this Mineral Resources statement and to the inclusion of this statement in the form and context in which it appears.

I certify that I have read the Qualified Persons Report and that it fairly and accurately represents the work for which I am responsible.

Based on the requirements of the Singapore Exchange Securities Trading Limited Practice Note 4C, I am a Qualified Person. I am also a Competent Person as defined by the JORC Code (2012), having five years of experience that is relevant to the style of mineralisation and type of deposit described in the report, and to the activity for which I am accepting responsibility.

Dated: 31st May 2015

Esteban Valle

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Esteban Valle
MAIG

21 GLOSSARY OF TERMS

Alteration	A change in mineralogical composition of a rock commonly brought about by reactions with hydrothermal solutions or by pressure changes.
Au	The chemical element gold
Breccia	A rock mass composed of large, angular fragments of pre-existing rocks
Cambrian	Period of geological time between 542 Ma and 488 Ma
Carbonates	Any carbonate mineral, compound composed of carbonate ions and metal such as calcium, magnesium or iron
Carboniferous	Period of geological time between 359 Ma and 299 Ma
Chalcopyrite	The mineral copper iron sulphide
Cleavage	A regular parting in rock formed as a result of compression. Typically seen in slate
Development	Underground activity to access an orebody (vein) for evaluation and mining
Devonian	Period of geological time between 416 Ma and 359 Ma
Diamond (core) drilling	Method of obtaining a cylindrical core of rock by drilling with a diamond impregnated bit. Produces a high quality sample
Dip/dipping	Angle and direction of steepest slope on a planar surface
Fault	A fracture plane in rocks showing significant movement between the two sides
Galena	The mineral lead sulphide
Grade	The relative quantity or percentage of mineral content. Gold grade is commonly expressed in the terms: g/t - grams per tonne, ppb – parts per billion, ppm – parts per million
Group	A major sequence of sedimentary rocks forming a distinctive unit by virtue of rocks and/or fossils present
g/t	Grams per tonne, used to express concentration of rare metals in rock. 1 g/t is equivalent to 1 ppm and 1,000 ppb
Indicated Mineral Resource	An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical, characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and or grade continuity but are spaced closely

	enough for continuity to be assumed
Inferred Mineral Resource	An 'Inferred Mineral Resource' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered though appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability
JORC / the JORC Code	The Reporting Code of the Joint Ore Reserves Committee (of the Australian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Minerals Council of Australia). The JORC Code 2012.
Ma	Millions of years
Measured Mineral Resource	A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaces closely enough to confirm geological and grade continuity
Metamorphism	The process of recrystallisation of rock as result of increased temperature and pressure
Micron (µm)	A measurement of distance – 1,000 µm is equivalent to 1 mm. A µm is 1×10^{-6} of a m
Mineral Resource	A technical term which is controlled in its use by the 2004 JORC Code. A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are subdivided, in order of increasing confidence, into Inferred, Indicated and Measured categories. The words 'ore' and 'reserves' must not be used in describing Mineral Resources as the terms imply technical feasibility and economic viability and are only appropriate when all relevant Modifying factors have been considered
Nugget effect	A term that describes grade variability for samples at small distances apart (less than a few cm). A low nugget effect (<20%) indicates minimal grade variation, whereas a high nugget effect (>70%) indicates that grade is highly variable and potentially relatively unpredictable. Pure nugget effect (100%) indicates an almost random grade distribution.
Ordovician	Period of geological time between 488 Ma and 443 Ma.

Ore Reserve	A technical term which is controlled in its use by the 2012 JORC Code. 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. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could be reasonably justified. Ore Reserves are sub-divided in order of increasing confidence into Probable Ore Reserves and Proved Ore Reserves
Ore shoot / shoot	A high grade zone within a mineral vein
Pyrite	The mineral iron disulphide
QA/QC (for sampling and assaying)	There are two components to a QA/QC system – quality assurance and quality control. Quality assurance (QA) refers to the protocols and procedures, which ensure that sampling and assaying is completed to the required quality. Quality control (QC), however, is the use of control samples and statistical analysis to ensure that the assay results are reliable
QKNA	Qualitative Kriging Neighbourhood Analysis
Quartz	The mineral silicon dioxide
Strike	Trend of an horizontal line on any geological plane
Strike slip	Movement parallel to the strike of a fault plane
Sulphides	Minerals composed of metals combined with sulphur
Variogram	A graphic representation of spatial correlation between samples in a given orebody. The variogram allows the calculation of the nugget effect and the sphere of influence of samples (the range)
Vein	A relative thin (millimetres to 10 m scale) sheet of quartz or other minerals cutting across pre-existing rocks

Appendix A
Checklist of assessment and reporting criteria, based on Table 1 of the 2012 JORC Code

Section 1 Sampling techniques and data

(Criteria in this section apply to all succeeding sections)

Criteria	JORC Code explanation	Commentary
Sampling techniques	Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling. Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. '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 (e.g. submarine nodules) may warrant disclosure of detailed information.	• Diamond drilling was used to obtain either nominal 1 m lengths of halved drill core, or full core sampling on nominal 0.4 m lengths of drill core, from which between 2 kg and 2.5 kg of material was pulverised for analysis using either Fire Assay (50 g) analysis or the LeachWELL 2000 g Cyanide leaching technique. For further details see section 6.2.6. • The mineralisation contains coarse particles of gold, up to 10 mm. The sampling method has been selected to accommodate the coarse nature of the gold particles. • The sample size preparation and the assay method is regarded as suitable for the style of mineralization. • Sample start and finish points and sample lengths were adjusted to match the boundaries of ore zones in order to maximise sample representivity. Sample compositing was utilised during the estimation process to compensate for any change of sample support occurring as a result of sample length variation incurred by these adjustments.
Drilling techniques	Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. 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.).	• Diamond drilling was used for all holes within the resource comprising NQ2 (50.6 mm), LTK60 (43.9 mm) and HQ (63.5 mm) sized core. • Core orientation was carried out by one of two methods; either using the Globaltech Orifinder® Orientation tool, or by using the pervasive north south trending upright cleavage as a reference plane.
Drill sample recovery	Method of recording and assessing core and chip sample recoveries and results assessed. Measures taken to maximise sample recovery and ensure representative nature of the samples. Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Intervals of lost core are identified using core blocks by drilling staff as core is recovered underground. During geological logging, intervals of lost core are verified by inspecting the core either side of the interval to ensure the breaks do not fit neatly together, if necessary drilling staff are consulted to determine the most likely position of the lost core. The final position is recorded within the lithological log as "lost core". • During core sampling, sample intervals are terminated at the edge of the lost core intervals to ensure that no assays are attributed to intervals of lost core. • During sample compositing, intervals of lost core are ignored. The result is that an intercept with a section of lost core will have a run of composites which stop precisely at the start of the lost core interval, and re-commence at the end of the interval of lost core. This ensures that block model estimates will only utilise composite data where assay data has been collected. • Core recovery can be poor in faulted zones often associated with gold mineralisation. It is anticipated that core loss as a result of faulted ground may result in under-reporting the true grade of the intersection. This is not

Criteria	JORC Code explanation	Commentary
		anticipated to have a material impact on the resource estimation as core loss accounts for less than 6% of the ore intersected.
Logging	Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography. The total length and percentage of the relevant intersections logged.	• Qualitative code logging was undertaken for lithology, alteration, veining and geotechnical rock quality. Structural measurements of bedding, cleavage and fault planes were taken where possible to aid in the interpretation of the ore body orientation. • Geological logging was carried out on all drill holes informing the estimate. • Core photos were taken of each core tray throughout all holes informing this resource. • Over the time during which the drilling was carried out a number of changes have been made to the core logging procedure to streamline and improve the logging process. These changes did not affect the way mineralisation domains are identified and interpreted.
Sub-sampling techniques and sample preparation	If core, whether cut or sawn and whether quarter, half or all core taken. If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry. For all sample types, the nature, quality and appropriateness of the sample preparation technique. Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. Whether sample sizes are appropriate to the grain size of the material being sampled.	• Core sampling has collected half diamond saw cut drill core on nominal 1.0 m lengths of drill core, and full core samples on nominal 0.4 m lengths of drill core. Approximately 2 kg to 2.5 kg of sample was used for assaying. • Samples were pulverised for 4 minutes using an LM5 pulveriser. 1 in every 10 samples have a 2g sub sample taken and tested using laser sizing analysis to ensure that >95% of the sample passes 75µm. • Second-half sampling was carried out on samples during 2010 to assess sample representivity. 336 samples with lengths between 0.5m and 0.9m, greater than 20% quartz content and greater than 0.1 g/t were analysed. As expected extreme variability was observed, with a 12% difference between the average grade of the LHS of the core and the RHS of the core. The change to full core sampling was made to improve sample representivity.
Quality of assay data and laboratory tests	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	• From November 2010 samples have been assayed by the Gekko Laboratory at the CGT Ballarat mine site. Samples prior to this date were processed in house at the BGF laboratory at the CGT Ballarat mine site or at Genalysis laboratory in Adelaide. • LeachWELL is not a total assay method, this technique generally recovers 98% of gold at Ballarat on a 24 hour leach. • QA/QC Procedures include the submission of standards and blanks. A campaign of duplicate sampling was carried out in 2010 whilst half core sampling was carried out. No duplicate samples have been submitted with full core samples. Internal laboratory standards were analysed within all submitted batches. • Drill hole samples have been supported by the submission of certified reference standards, details of which are given in Section 6.2.7.
Verification of	The verification of significant intersections by either independent or alternative company	• Significant intersections were identified and modeled during detailed

Criteria	JORC Code explanation	Commentary
sampling and assaying	personnel. The use of twinned holes. Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. Discuss any adjustment to assay data.	geological interpretation by either the Resource Geologist (Mr. Esteban Valle) or the Geology Manager (Mr. Matthew Hernan). All significant intersections modeled are reviewed by the Geology Manager. • Sample intervals are allocated unique sample identification numbers and entered directly into the company's AcQuire™ database. Analytical results are received from the Gekko assay laboratory as .CSV files and imported directly into the database. Data validation functions built into the AcQuire™ database data entry and importing forms reduce the potential of importing incorrect data. • CGT regularly audits the assay laboratory and routinely submits and monitors a series of Certified Reference standards and blanks in accordance with the company's sampling QA/QC procedure.
Location of data points	Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. Specification of the grid system used. Quality and adequacy of topographic control.	• All diamond drill holes are located relative to a local mine grid. The mine grid is based on a modified AMG66 grid whereby northing's are AMG66 minus 5,800,000 m and easting's AMG66 minus 700,000 m. Relative levels are based on the Australian height datum 1971 (AHD), whereby relative levels are AHD plus 10,000 m. • Drill hole collars have been surveyed by Castlemaine Goldfields surveyors. Down hole surveys were carried out using a Globaltech Pathfinder® down hole multi shot camera. Holes which lacked collar surveys and/or downhole surveys have been discussed in sections 6. • Topographic surface level has been surveyed for the mine, however is not considered material to this estimate due to the depth of the mineralization being between 550m and 700m below the surface.
Data spacing and distribution	Data spacing for reporting of Exploration Results. Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. Whether sample compositing has been applied.	• Diamond drilling within the resource was completed on 20m to 30m spaced east-west oriented drill fans. Hole spacing within fans varies between 7m and 15m. • The drill hole spacing used in this estimate is considered adequate to test the geological continuity of the domains identified. The spatial variability of gold grades observed within ore domains indicates it is unlikely that the drill spacing will enable an accurate estimation of grades on a local scale. • The drill spacing is regarded as typical of that used to define resources that have been mined during the past year. Grade estimates have been validated against processed grades over the past year and found to be within acceptable tolerances, given the Inferred Mineral Resource classification previously applied to all resources at Ballarat. • The drill spacing used for this resource is considered adequate to qualify it as an Inferred Mineral Resource as defined by the 2012 JORC code.

Criteria	JORC Code explanation	Commentary
		• Where drilling is supported by the inclusion of development levels showing acceptable reconciliation between face and block model grade, then this resource is considered adequate to qualify it as an Indicated Mineral Resource. • Sample intervals were adjusted to ensure sampling was not carried out across mineralisation boundaries, as a result there is some variation in the lengths of the sample intervals informing this estimate. • Sample compositing was undertaken in an effort to attain equal sample support as described in section 8.3.3
Orientation of data in relation to geological structure	Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• The drill orientation is variable within the deposit; however most holes are drilled at angles approaching perpendicular to the orientation of the main west-dipping fault zones. • As the mineralisation is comprised of a combination of west-dipping fault zones and east-dipping vein arrays, it is common for west-dipping fault zones to be well delineated by drilling perpendicular to their orientation, but for east-dipping vein arrays to be poorly represented due to holes being almost parallel to their orientation. The drill intersection angle common for east-dipping vein arrays may cause bias whereby they are under-represented by volume due to conservative wire-framing commonly applied to domains of low geological confidence.
Sample security	The measures taken to ensure sample security.	• Samples from drilling used in the estimate were retained at the Ballarat mine site at all times. The assay laboratory is located on the mine site and subject to the same security monitoring as the mine site.
Audits or reviews	The results of any audits or reviews of sampling techniques and data.	• No independent audit or review has been carried out on the sampling techniques or data • Sampling techniques and data have been internally reviewed by the Ballarat Mine Geology Manager, Mr Hernan • The Competent Persons, Mr Valle and Mr Hernan, continually review sampling techniques and data as part of the QAQC programme.

Section 2 Reporting of exploration results

(Criteria listed in the preceding section also apply to this section)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	• Details of the tenement and all related material issues , and the security of the tenure as reported in the relevant section, have been independently verified by the Competent Persons, Mr de Vries and Mr Hernan, via the DSDBI web site.
Exploration done by other parties	Acknowledgment and appraisal of exploration by other parties.	• Acknowledgment and appraisal of exploration by other parties is contained within the historical section of the report.
Geology	Deposit type, geological setting and style of mineralisation.	The deposit type, geological setting and style of mineralisation are all detailed within the relevant sections of the report.
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.	• A summary of all information material to the understanding of the exploration results is contained within relevant sections of the report and has been verified by the Competent Persons, Mr de Vries Mr Valle and Mr Hernan.
Data aggregation methods	In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated. Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. The assumptions used for any reporting of metal equivalent values should be clearly stated.	• All data aggregation methods are detailed within the appropriate sections of the report.
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 (e.g. 'down hole length, true width not known').	• All relationships between mineralisation widths and intercept lengths are detailed within the appropriate sections of the report.
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.	• All maps and sections (with scales) and tabulations of intercepts that are considered appropriate have been included in the report.
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	• Comprehensive reporting of all Exploration Results is not practicable, however a representative reporting of both low and high grades and/or widths

Criteria	JORC Code explanation	Commentary
	reporting of Exploration Results.	has been provided within the report as required by the JORC2012 to ensure balanced reporting.
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.	• The Competent Persons, Mr de Vries Mr Valle and Mr Hernan are unaware of any substantive exploration data not included within the report.
Further work	The nature and scale of planned further work (e.g. 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.	• The nature and scale of planned further work has been discussed in the relevant sections of the report. Diagrams and detailed discussions have been restricted due to the commercial sensitivity of such items.

Section 3 Estimation and reporting of Mineral Resources

(Criteria listed in Section 1, and where relevant in Section 2, also apply to this section)

Criteria	JORC Code explanation	Commentary
Database integrity	Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. Data validation procedures used.	• Geological logging is entered directly into the company's AcQuire™ Database. The data entry module will not allow invalid logging codes to be entered, nor will it allow overlapping intervals. • Geological logging has been validated visually against drill core photos. • Drill hole collars are visually inspected in Vulcan to validate their position is consistent with the position of development. • Before any assays are imported into the database, the results of standards and blanks submitted are reviewed. Any inconsistencies identified are addressed with the assay laboratory before being imported. • Access to the CGT drilling database used for resource estimation is restricted to geological and selected technical staff. The database is managed by Mr Esteban Valle, Mr Valle is a qualified geologist, with over 8 years' experience in mine geology. Mr Valle is a member of the geological team who has administration rights for the database. • The database, together with all data on the company's computer network is backed up on a daily, weekly and monthly basis by CGT's IT co-ordinator.
Site visits	Comment on any site visits undertaken by the Competent Person and the outcome of those visits. If no site visits have been undertaken indicate why this is the case.	• Mr Matthew Hernan and Mr Esteban Valle who have compiled and prepared this Mineral Resource estimate have regularly inspected the underground workings and diamond drill core as part of their duties. Mr Hernan and Mr Valle are co-CP's with Mr de Vries. • The competent person Mr Peter de Vries is a consultant with Mining One. Mr de Vries has visited the Ballarat site as part of this role during the preparation and compilation of this resource estimate. Mr de Vries inspected geological database and core logging operations.
Geological interpretation	Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. Nature of the data used and of any assumptions made. The effect, if any, of alternative interpretations on Mineral Resource estimation. The use of geology in guiding and controlling Mineral Resource estimation. The factors affecting continuity both of grade and geology.	• This resource estimate is based on detailed geological interpretations carried out by CGT geologists Ms Sarah Cochrane, Ms Jacinta Holland, Mr Tom Cochrane, Mr Karl McNamara, Mr Daniel Braunsteins, Mr Jesse Coates-Marnane, Mr Rod Fraser and Mr Matthew Hernan. A broad description of the geology has been given in section 6.2.1. • Geological interpretation is based primarily on hand drawn detailed paper sections of drill fans on individual cross-sections. As described in section 8.3.2. • Geological wire-framing is based on the detailed interpretations as described in section 8.3.4.

Criteria	JORC Code explanation	Commentary
Dimensions	The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The resource is comprised of discrete mineralised zones associated with the First Chance and Sulieman anticlines within the Ballarat East goldfield. • The six zones estimated occur within an area 600 m in strike (north-south), 500 m in width (east-west) and 200 m in height (elevation). The base of the resource is located approximately 700 m below the surface, with the upper-most portion terminating approximately 550 m below the surface.
Estimation and modelling techniques	The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. The assumptions made regarding recovery of by-products. Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation). In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. Any assumptions behind modelling of selective mining units. Any assumptions about correlation between variables. Description of how the geological interpretation was used to control the resource estimates. Discussion of basis for using or not using grade cutting or capping. The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Wireframes of geological domains based on detailed hand drawn interpretations were constructed using Vulcan Version 9 Software. • Wireframes were extrapolated no more than 15m beyond the limit of drilling data (approximately half drill fan spacing). • Block model construction, Sample compositing and grade estimations were all carried out using Vulcan version 9 Software. • Geological domaining is carried out to reduce the potential for grade smearing. Geological domains are constructed to constrain high grade assays within high grade domain wireframes. • No variography has been performed on the assays informing this resource, however statistical analysis was undertaken as described in section 8.1 • Top cutting was carried out on all domains estimated. Refer to section 8.1.4 for details. • Geological domains were estimated independently of one another. Sample selection for each domain honoured the boundaries of the domain. • Block models were constructed for each of the five lodes estimated, the construction parameters for which are described in section 9.1 • Inverse Distance squared estimation was used for estimation of gold grade within the modelled geological domains, further detail regarding estimation parameters can be found in section 9.3 and 9.4. • Each of the block models created had checks and validations carried out on them as described in Section 10.
Moisture	**Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.**	• **The estimation is based upon dry tonnages. Moisture content has not been included**
Cut-off parameters	The basis of the adopted cut-off grade(s) or quality parameters applied.	• Due to the highly variable grade distribution within this resource, there is a lower level of confidence in estimations of individual mining blocks, than there is in the overall resource. As a result selective mining above a grade threshold, on a block by block basis, may not be achievable. The resource reported is global in nature and reported at a 0g/t cut-off. • In cases where whole domains were estimated to contain gold grades less than 4 g/t, these domains were omitted from the Inferred Resource as they are considered unlikely to have reasonable prospects of eventual economic

Criteria	JORC Code explanation	Commentary
		extraction based on mining costs and the gold price at the time of estimation.
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.	• Mining at Ballarat is via a combination of conventional drive development and open stoping. • Based on current all in operating costs the mineralisation estimated is considered to have reasonable prospects for economic extraction. • This assumes (for the FY 2015-2016) a gold price of A$1,506 per ounce and combined mining and processing costs of $197 per tonne (based on mining operations carried out to date).
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.	• Based on recent upgrades to the processing plant and its recent performance, processing recovery is assumed to be 87%.
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 greenfield 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.	• Mining activity is being carried out on MIN5396 and MIN4847. • The Ballarat mine has sufficient waste and tailings storage facilities in place to store any by-products generated as a result of processing the ore contained in this resource. • All required permits are in place. • All required monitoring is undertaken to ensure compliance with licences.
Bulk density	Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit. Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Bulk density was determined by the water immersion technique, details of which can be found in Section 6.2.6. • A bulk density of 2.65 g/cm3 was determined and applied to all estimations in this resource.
Classification	The basis for the classification of the Mineral Resources into varying confidence categories. Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). Whether the result appropriately reflects the Competent Person's view of the deposit.	• Whilst the drilling carried out into this resource is considered sufficient to verify geological continuity of fault zones, due to the high grade variability observed, the assay data informing this resource is only considered sufficient to imply grade continuity, and not to verify it. • Where mine development has accessed and exposed ore lodes the additional information gained by geologists during underground mapping and sampling is considered sufficient to verify grade continuity locally. • This estimation has been classified as containing **Indicated** and **Inferred Mineral Resources** as defined by the Australasian code for reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC code

Criteria	JORC Code explanation	Commentary
		2012) on the basis of extrapolation which has been kept to a minimum.
Audits or reviews	The results of any audits or reviews of Mineral Resource estimates.	• An independent audit and review has been carried out on this resource by Mr de Vries.
Discussion of relative accuracy/ confidence	Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The tonnages estimated in this resource estimate are reported with varying levels of confidence, with models created to emulate geological structures observed during recent mining at the Ballarat mine. • Estimates of grade at a global scale within this resource are reported with a moderate level of confidence. This is based on review of reconciliation data discussed in Section 9.4.6. • Due to grade variability observed within the assay data set used in this estimate, grade estimates of discrete blocks are considered to be indicative only and insufficient to be used as the basis for selective mining practices. • The Competent Persons believe that a global precision of ±20% to ±30% is reasonable for the Ballarat Gold Mine resources and is reflected by their classification as **Indicated** and **Inferred Mineral Resources.**

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code 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 underground Ore Reserve estimate is based on the Mineral Resource estimate prepared in March 2015 by Castlemaine Goldfields Pty Ltd (“CGT”) in accordance with the reporting guidelines of the 2012 JORC code. • The Mineral Resources are reported inclusive of the Ore Reserve.
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 Persons work at the site. In addition, site visits have been completed by the external review organisation and time spent with each person involved in the estimation.
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 Ballarat Gold Project is a current and operating mine. Historic costs and operating parameters have been used in determining the Ore Reserve estimate. • As historical operating data has been utilised it is considered to be more accurate than a feasibility study. As such, no material Modifying Factors have been considered.
Cut-off	• *The basis of the cut-off grade(s) or quality parameters applied.*	• The Probable Ore Reserve estimate lies within 10 metres of existing development. All

Criteria	JORC Code explanation	Commentary
parameters		stopes were evaluated on an incremental basis, with a fully costed notional break even cut-off grade of approximately 2.9 g/t.
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.* • *The assumptions made regarding geotechnical parameters (e.g. pit slopes, stope sizes, etc.), grade control and pre-production drilling.* • *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.* • *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* • *The infrastructure requirements of the selected mining methods.*	• The Ballarat Gold Project Ore Reserve has been estimated by generating detailed mining shapes based on existing development and stopes. Individual factors for dilution and mining recovery have been completed post-geological interrogation to generate the final diluted and recovered ore reserve. • The Ballarat Gold Project is in production with all planned mining methods currently practiced on site. Production history demonstrates these mining methods to be successful. • Stope size, development placement and ground support strategies are designed in accordance with recommendations from professional geotechnical personnel during several phases of mine design. The existing mine plan approval process and Stope Note documentation ensure that individual stope parameters are considered. • Grade control Block Models are generated utilising a combination of diamond drilling results and information gathered during underground development by mine geologists. These are updated, as required, with additional drilling once the development for the stope is in place. • A minimum stope width of 2.5m is used based on the current mine plan (levels 14 to 20 metres vertically apart) and the production equipment utilised (64mm or 76mm production holes). The Mineral Resource model used has been prepared under the supervision of the geological Competent Person. • The Ballarat Gold Project is a complex orebody with mineralisation closely associated with faulting. Dilution factors have been applied according to historical dilution data from past stoping and development. Mining dilution has been applied at 5% in Floor Stripping, through to an upper range for Long Hole Stoping of 50% in areas of poor ground conditions. 72 to 95% for blind uphole stoping, where stope pillars have not been incorporated into the design and 95% for detail design where pillars have been taken into account. 95% for longhole stoping. • The minimum mining width for stopes is 2.5m. • Inferred Mineral Resources are included within the mine plan to allow for well-informed strategic planning. Historically, Ballarat Gold Project has mined an Inferred Mineral Resource. • Mining infrastructure will comprise ventilation, and escape raises, typical underground operating and capital development such as stockpiles, electrical substations, and pump stations, As an operating mine the infrastructure requirements of the stoping and development methods used are already in place or are an integrated part of development design when development in new areas commences.
Metallurgical factors or	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.*	• At Ballarat Gold Project the ore is trucked to the processing plant which is located within 300 metres of the main access portal of the mine. The mill consists of a crushing circuit with ore separation/treatment via a primary gravity circuit that co-recovers both

Criteria	JORC Code explanation	Commentary
assumptions	• *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?*	free gold (70%) and "sulphide gold". The free gold component is smelted to doré and the sulphide component is cyanide leached, electrowon, and smelted. Probable Reserve ore mineralogy is similar to that historically and currently treated through the processing plant. Note that the processing plant has been operating in its current configuration since 2011. A flotation circuit is currently being commissioned and will be incorporated into the processing plant in the near term. • The metallurgical process is well tested technology. • Recovery is variable to ore head grade and is based on current plant performance with an allowance made for flotation improvement. The Metallurgical recovery factor applied is 87%. • No assumptions or allowances have been made for deleterious elements. • The current resource has a history of operational processing experience. • N/A
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.*	• Ballarat Gold Project currently possesses all necessary government permits, licences and statutory approvals and is compliant with all legislative and regulatory requirements. The mining the Probable Mineral Reserve will have no further environmental impact except to increase the height of the tailings within the approved storage facility and possibly increase the footprint of the permanent waste rock storage facility. Where appropriate underground voids resulting from stoping will be filled with waste rock from underground access development.
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 mine is currently in operation and therefore has adequate infrastructure to support current and future mining.
Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.* • *The methodology used to estimate operating costs.*	• The Ore Reserve lies within 10 metres of established operational and capital development. All capital costs have been estimated based upon the Mine Plan and experience of costs incurred through past mining and processing activities in the past. Infrastructure capital costs have already been expended. • The operating cost estimates are based upon historical costs incurred over previous periods and the internal budgeting process.

Criteria	JORC Code explanation	Commentary
	• *Allowances made for the content of deleterious elements.* • *The source of exchange rates used 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.*	• No deleterious elements are expected. • Exchange rates are based upon internal technical and economic analysis. • Mining and Haulage costs are based on historical costs incurred during previous operating periods. • Processing costs are based on historical data from the process plant at the Ballarat Gold Project. • No Victorian State royalty. 2.5% royalty on gold production payable to Newcrest Mining Ltd, capped at A$50M.
Revenue factors	• *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.*	• N/A • Revenue is calculated using a gold price of A$1,506/oz. This is based on a gold price of US$1,220 and a AUD/USD exchange rate of $0.81. The basis of this forecast is a consensus of 10 economic analyst groups' forecast for the period. All product is sold at "Australian dollar spot market" prices.
Market assessment	• *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.*	• The Ballarat Gold Project Ore Reserve will produce a revenue stream from the sale of gold doré. All product is sold at "Australian spot market" prices. • N/A • N/A • N/A
Economic	• *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.*	• N/A. The Mineral Reserve represents less than one year's production so that no discount rate or inflation modifiers have been applied to the cash flow estimate.
Social	• *The status of agreements with key stakeholders and matters leading to social license to operate.*	• CGT maintains its social license to operate by engaging with neighbours to the mine and the local community to foster a close relationship and actively seek and provide feedback and dialogue. To the best of the Competent Person's knowledge all agreements are in place and are current with all the key stakeholders.
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*	• None • Supply and service contracts are in place for all critical goods and services required to operate the mine. • The Ballarat Gold Project is currently in operation with all government and third party

Criteria	JORC Code explanation	Commentary
	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.	approvals in place for the stated reserves.
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 Ore Reserve estimate is based on the Mineral Resource estimate contained within the designed stopes and classified as "Indicated" after consideration of all drilling, geological validation, the orebody experience, mining method, metallurgical, social, environmental, and financial aspects of the project. The Ore Reserves include Probable Ore derived from the Indicated Mineral Resource. • The Ore Reserve classification appropriately reflects the Competent Person's view of the deposit. • There is no Measured Mineral Resource estimated. The Probable Ore Reserves are not derived from nor do they include a Measured Mineral resource.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• The Ballarat East Ore Reserve estimate was subject to an internal peer review and was reviewed by the Competent Person and is considered to be reasonable, and adequately supported. • Mining One Pty Ltd ("Mining One") was commissioned to conduct an independent review of the Ore Reserve estimation process and results. Mining One did not identify any material issues with the Ore Reserve estimate.
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 Ore Reserve estimate is prepared within the guidelines of the 2012 JORC code. The relative confidence of the estimate falls within the criteria of Probable Reserves. Significant operating history supports the Mineral Resource model, metallurgical factors and operating unit costs. • This statement relates to global estimated tonnes and grade. • Not applicable as the Ballarat Gold Project is in operation and historic data has been used. • Reconciliation results from past mining at Ballarat Gold Project has been considered and factored into the reserve assumptions where appropriate.