NI 43-101 Technical Report for Osborne Copper-Gold Project located in northwest Queensland Region of Australia

Prepared by

srk consulting

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Date of Report:		**2 November 2012**
Effective Date of Mineral Resources:		
	Osborne	**27 October 2011**
	Kulthor	**5 September 2012**
Effective Date of Mineral Reserves:		**01 June 2012**
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1 Executive Summary

Introduction

Ivanhoe Australia Limited (Ivanhoe) commissioned SRK Consulting (Australasia) Pty Ltd (SRK) to prepare a Technical Report in line with Canadian and internationally recognised National Instrument 43-101 reporting standards and Form 43-101F1, for mining activities at the Osborne Open Pit, Osborne Underground and Kulthor Underground deposits.

Ivanhoe is an Australian-based Resource company listed on both the Australian Securities Exchange and Toronto Stock Exchange as IVA. It has assembled a significant package of Mineral Resources in the highly mineralised Cloncurry district, near Mt Isa, Queensland. Exploration success so far has revealed a portfolio of copper, molybdenum, rhenium and gold Mineral Resources. This report considers only the copper and gold assets which include the Osborne deposits and Kulthor deposit.

Ivanhoe purchased the Osborne copper-gold assets from Barrick in 2010.

Property Description and Location

The Osborne copper-gold deposits are located at latitude 22° 04' south, longitude 140° 23' east. The deposits are situated some 195 km southeast of Mt Isa in North West Queensland, Australia. The Kulthor copper-gold deposit and underground mine are located approximately 2 km west of the Osborne Underground mine and the two mines are connected via a decline.

The Osborne mine is located within mining lease (ML) 90040 (Osborne). The Kulthor deposit is on the adjacent ML90158 (Kulthor). Both mining leases are located within exploration lease EPM9624 (Trough Tank) which is held 100% by Ivanhoe (Osborne) Pty Ltd. The tenement corner points were surveyed in by an accredited surveyor from M H Lodewyk Pty Ltd of Mt Isa.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Osborne copper-gold project area can be accessed by chartered aircraft via an all-weather airstrip from Townsville or Mt Isa. Osborne can also be accessed by predominantly sealed roads from Mt Isa via Dajarra, or partly sealed roads from Mt Isa via Duchess.

There are several river and creek crossings through causeways which can become impassable for relatively short periods of time (days) during the wet season.

The population of Mt Isa and surrounding area is approximately 22,000 and Cloncurry is 2,400.

Vegetation consists of arid spinifex and sparse eucalypt trees. The area has a semi-arid climate with temperatures varying between 10°C and 25°C in winter and from 25°C to 40°C in summer. Average rainfall is 350 mm, most of it occurring during the summer months of December to March. The weather is acceptable for exploration and mining operations year-round.

History

The Ivanhoe assets at Osborne were acquired from Barrick (PD) Australia Limited (Barrick). Exploration, mining and processing operations have been undertaken in the Osborne region since the mid-1980s. Between 1995 and 2010, 24.2 million tonnes (Mt) @ 2.68% copper, 0.96 grams per tonne (g/t) of gold were processed from the Osborne open pit and subsequent underground operations yielding 601 kilo tonnes (kt) copper and 566 kilo ounces (koz) gold. At the time of closing the operations, underground development toward the Kulthor underground mine was underway. The Kulthor deposit is now being developed and production has commenced.

The Osborne deposits have been the subject of exploration and mining activities by a number of companies over the past 30 years.

Geological Setting and Mineralisation

The Osborne copper-gold deposit lies within palaeo-Proterozoic metasediments assigned to the Mt Norna Quartzite of the Soldiers Cap Group. The host sequence of sandstone, siltstone and ironstone is cut by dolerite dykes and has undergone partial melting to produce granofels, migmatites and gneiss. Pegmatite dykes and related alteration and mineralisation are concentrated in a pod of lower-grade metamorphic rocks surrounded by partial melt rocks. At least four phases of deformation are recognised in the Osborne region with the second thrusting event producing the dominant foliation. The strike slip faulting of the third event is believed to have produced dilations that now host the magnetite-rich Osborne ore bodies.

The mineralisation at Kulthor is shear and replacement sulphide lodes that overprint a series of mineralogically zoned pegmatitic veins and shears. The veins and shears are contained within a shear bound block of altered psammite and amphibolite that is up to 150 m wide and at least 900 m long. There is a general increase in the degree of partial melting on either side of the central altered and mineralised zone at Kulthor.

Osborne, Kulthor, and most of Osborne's exploration targets are characterised as belonging to the iron oxide copper-gold (IOCG) class of deposits.

Exploration, Drilling, Sampling and Assaying

The Osborne and Kulthor deposits are completely covered by later barren Mesozoic sediments, making surface exploration ineffective. The Kulthor deposit lacks the strong magnetic signature of the nearby Osborne deposits so was found much later. Kulthor was eventually found by geochemical sampling at the base of the Mesozoic sediments using air core drilling. The Osborne and Kulthor deposits have been explored mainly by diamond drilling.

All core samples were cut using a diamond saw at the Osborne Mine core processing facility. Samples were dispatched to Mt Isa and Townsville for further preparation and assay by commercial laboratories. Assays were received electronically from the laboratory and merged into the site acQuire database under the supervision of the senior site geologist or database manager.

Ivanhoe and the previous owners of the Osborne Copper Gold Project (Barrick and PDAP) used industry standard QA/QC protocols for sample preparation and assaying, including check sampling of duplicates, use of standards and blanks and use of a second commercial laboratory for further checking.

The resource database is of adequate quality for use in resource estimation.

Mineral Resource Estimates

The combined Mineral Resource estimate for the Osborne copper-gold project is summarised in Table ES-1. Table ES-2 summarises the additional Inferred material. The tables show the equivalent copper (eCu) cut-off grade as well as the individual copper and gold grades, where eCu is defined as eCu = copper (%) + gold (g/t) x 0.6. This equivalence is based on assumed metals prices of 3.75 USD/lb copper and 1400 USD/oz gold at Osborne with comparable concentrator recoveries of 85%. Recovery of gold in copper concentrates has previously been demonstrated for mining operations at Osborne. The Kulthor resources have been depleted for production.

Table ES-1: Summary of Measured and Indicated Mineral Resource Estimate

Category	Cut-off Grade eCu %	Tonnes (Mt)	Copper (%)	Gold (g/t)	Contained Copper (000' t)	Contained Gold (000' oz)
Osborne Open Pit[2]						
Measured	0.5% eCu	2.2	0.7	0.6	16.5	40.8
Indicated		0.2	0.7	0.6	1.5	4.1
Osborne Underground [2]						
Measured	1.2% eCu	2.1	1.5	0.9	31.7	57.5
Indicated		0.8	1.2	0.9	9.7	22.1
Kulthor Underground [3]						
Measured	1.2% eCu	2.9	1.7	1.0	48.9	96.7
Indicated		4.5	1.5	1.0	67.6	137.7
Subtotal Osborne-Kulthor Deposits						
Measured		**7.2**	**1.3**	**0.8**	**97.1**	**195.0**
Indicated		**5.5**	**1.4**	**0.9**	**78.9**	**163.9**
Total Mineral Resource		**12.7**	**1.4**	**0.9**	**176.0**	**358.9**

1 eCu = copper (%) + gold (g/t) x 0.6.

2 The Mineral Resource Estimate is effective as at 27 October 2011.

3 The Mineral Resource Estimate is effective as at 5 September 2012.

4 The Mineral Resource Estimates have been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

5 Some totals may not add due to the effects of rounding.

Table ES-2: Summary of Inferred Mineral Resource Estimate

Category	Cut-off Grade eCu %	Tonnes (Mt)	Copper (%)	Gold (g/t)	Contained Copper (000' t)	Contained Gold (000' oz)
Osborne Open Pit[2]	0.5% eCu	0.1	0.6	0.6	0.4	1.3
Osborne Underground[2]	1.2% eCu	0.5	1.2	0.9	5.6	13.4
Kulthor Underground[3]	1.2% eCu	5.4	1.3	0.9	72.8	148.2
Total Inferred Mineral Resources		**5.9**	**1.3**	**0.9**	**78.9**	**162.8**

1 eCu = copper (%) + gold (g/t) x 0.6

2 The Mineral Resource Estimate is effective as at 27 October 2011.

3 The Mineral Resource Estimate is effective as at 5 September 2012.

4 The Mineral Resource Estimate s have been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

5 Some totals may not add due to the effects of rounding

Osborne Deposits

The Osborne deposits Mineral Resource estimates included the partly mined 1SS Deeps and the Open pit Zones, together with the more recently discovered Kulthor zone. The total Mineral Resources for the Osborne deposits are shown in Table ES-3. The cut-offs used were 1.2% eCu for underground Mineral Resources and 0.5% eCu for open pit Mineral Resources.

The Measured and Indicated Mineral Resources, presented in Tables ES-3 to ES-5, are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1SS Deeps

The primary criteria used to generate the classified Mineral Resource estimate below 125 metres reduced level (mRL) in Table ES-3 are summarised below:

- The 1SS Deeps mineralisation was modelled as a single ore zone and the enclosing dilution zone;
- An external cut-off of 1.2% eCu was used for modelling the 1SS ore zone and 0.6% eCu for modelling the enclosing dilution zone;
- The area of previous mining was delineated and depleted from the Mineral Resources;
- Top-cuts were selected and the zones estimated using Ordinary Kriging. Copper, gold and density were estimated;
- Dynamic Anisotropy[1] modelling was used to handle changes in the orientation of the zones;
- The estimates were validated for grade bias and also for local and global variability; and
- The estimates were classified into Measured, Indicated and Inferred Mineral Resources using Kriging variance.

Table ES-3: 1SS Mineral Resources below 125 mRL (eCu>=1.2%)

Category	Quantity (Mt)	Grade			Metal	
		eCu (%)	Copper (%)	Gold (g/t)	Copper (000' t)	Gold (000' oz)
Measured	2.1	2.1	1.5	0.9	31.7	57.5
Indicated	0.8	1.7	1.2	0.9	9.7	22.1
Measured + Indicated	**2.9**	**2.0**	**1.4**	**0.9**	**41.5**	**79.6**
Inferred	0.5	1.7	1.2	0.9	5.6	13.4

(eCu = copper (%) + gold (g/t) x 0.6)

Note: some totals may not add due to the effects of rounding

Open Pit Area

The primary criteria used to generate the classified Mineral Resource estimate in Table ES-4 are summarised below.

- The mineralisation was modelled as two main ore zones and two small subsidiary zones using an external cut-off of 0.5% eCu. An enclosing dilution zone was modelled using a cut-off of 0.25% eCu;
- There was a small proportion of Reverse Circulation (RC) drilling in the database. This was compared to nearby diamond drilling and found to be conservative;
- Top-cuts were selected and the zones estimated using Ordinary Kriging. Copper, gold and density were estimated;
- Dynamic Anisotropy[1] modelling was used to handle changes in the orientation of the zones;

[1] Dynamic Anisotropy is a modelling technique that allows for the rotation of geological blocks to follow mineralisation trends.

- The estimates were validated for grade bias and for local and global variability;
- Classification of the estimates into Measured, Indicated and Inferred Mineral Resources was done using Kriging variance (the same nominal variogram was used for all domains);
- The use of limiting pit design removed the necessity to deplete the Mineral Resources by previous underground mining;
- The final block model was fully populated with blocks and DENSITY values out to the model limits to assist pit design; and
- The oxidation surfaces are important for metallurgy recovery.

Table ES-4: Open Pit Classified Mineral Resources (eCu >= 0.5%)

Category	Quantity (Mt)	Grade			Metal	
		eCu (%)	Copper (%)	Gold (g/t)	Copper (000' t)	Gold (000' oz)
Measured	2.2	1.1	0.7	0.6	16.5	40.8
Indicated	0.2	1.1	0.7	0.6	1.5	4.1
Measured + Indicated	**2.4**	**1.1**	**0.7**	**0.6**	**18.0**	**44.9**
Inferred	0.1	0.9	0.6	0.6	0.4	1.3

(eCu = copper (%) + gold (g/t) x 0.6)

Note: some totals may not add due to the effects of rounding

Kulthor Mineral Resource Estimate

The primary criteria used to generate the classified Mineral Resource estimate in Table ES-5 are summarised below:

- The Kulthor mineralisation was modelled as three easterly-trending steeply-dipping mineralised zones. These were surrounded by a broad mineralised envelope that was based on the presence abundant quartz-dolomite veining and scattered mineralisation;
- A small portion of the upper part of some of these zones was affected by oxidation. This was not important for resource estimation, but will affect metallurgical recovery;
- The drillhole data were composited to 1 m;
- Top-cuts were selected and the zones estimated using Ordinary Kriging. Copper, gold and density were estimated;
- Dynamic Anisotropy[2] modelling was used to vary the search and variogram parameters throughout the mineralised zones;
- The estimates were validated by comparison to the other estimation methods and to the drillhole data. Local and global variability was also checked;
- The estimates were classified using the Kriging Variance of the kriged copper estimates. There were a high proportion of Indicated and Inferred Mineral Resources due to the wide drillhole spacing except in the area of underground drilling; and
- The estimates were depleted for production as of the Effective Date 5th September, 2012 (mainly development).

[2] Dynamic Anisotropy is a modelling technique that allows for the rotation of geological blocks to follow mineralisation trends.

Table ES-5: Summary of Kulthor Mineral Resource Estimates

Item	Cut-off Grade eCu (%)	Tonnes (Mt)	Cu (%)	Au (g/t)	Cu (000' t)	Au (000' oz)
Measured Resource	1.2	2.9	1.7	1.0	48.9	96.7
Indicated Resource		4.5	1.5	1.0	67.6	137.7
Total Measured and Indicated Resource	**1.2**	**7.4**	**1.6**	**1.0**	**116.5**	**234.4**
Inferred Resource	1.2	5.4	1.3	0.9	72.8	148.2

1 eCu = copper (%) + gold (g/t) x 0.6.
2 The Mineral Resource Estimate is effective as at 5 September 2012.
3 The Mineral Resource Estimate has been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.
4 Some totals may not add due to the effects of rounding.

Mineral Reserve Estimates

The combined Mineral Reserve Estimate for the Osborne and Kulthor deposits is summarised in Table ES-6. This is a subset of the available Mineral Resource.

The Kulthor Mineral Reserve Estimate is based on the previous Mineral Resource (effective date 27 October 2011). Kulthor Mineral Resource described in this Technical Report has not been used for the Kulthor Mineral Reserves Estimate.

Table ES-6: Summary of Mineral Reserve Estimate [1,2,5]

Classification	Tonnes (Mt)	eCu [3] (%)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (t)	Contained Gold (ozs)
Osborne Open Pit						
Proven	2.4	1.17	0.83	0.57	19,920	43,982
Probable	0.1	1.04	0.72	0.54	720	1,736
Total	**2.5**	**1.16**	**0.82**	**0.57**	**20,640**	**45,718**
Osborne Underground						
Proven	0.5	2.47	1.93	0.90	9,742	14,602
Probable	-	-	-	-	-	-
Total	**0.5**	**2.47**	**1.93**	**0.90**	**9,742**	**14,602**
Kulthor Underground[4]						
Proven	-	-	-	-	-	-
Probable	2.58	2.04	1.47	0.94	37,787	77,706
Total	**2.58**	**2.04**	**1.47**	**0.94**	**37,787**	**77,706**
Total Mineral Reserve	**5.58**	**1.69**	**1.22**	**0.77**	**68,169**	**138,026**

1 The Mineral Reserve is as at 1 June 2012.
2 The Mineral Reserve has been prepared by Ms Anne-Marie Ebbels, MAusIMM (CP), an employee of SRK Consulting (Australasia) Pty Ltd, who is a qualified person as defined by NI43-101.
3 eCu = copper (%) + gold (g/t) x 0.6
4 Based on 2011 Mineral Resource Estimate.
5 Some totals may not add due to the effects of rounding.

Mining Methods

The proposed mining operations all utilise standard mining methods and have utilised design methodology, design criteria and scheduling assumptions consistent with good practice and are applicable for a study of this nature.

Osborne Open Pit

Ivanhoe are proposing to extend the current Osborne open pit via a cut back to south-west of the current workings. The open pit deposit is a copper-gold mineralisation of approximately 2.5 Mt ore grading 0.8% copper and 0.6 g/t gold.

Open pit mining at Osborne is to be completed using standard methods - drill and blast followed by load and haul. This was previously undertaken at Osborne with the use of contractors from 1995 to 1996.

Current open pit operations lie to the northeast of the underground operations, with a portal accessing the underground operation currently at the 1200 mRL within the open pit operation. As part of the open pit expansion, underground access is required to be maintained while the underground operations continue. For this reason, the open pit will be scheduled in two stages. This will allow an initial cutback, maintaining the underground access with a second final cutback to complete the pit once underground access is no longer required. The underground operation can also be accessed by personnel and smaller materials via the main shaft.

The Open Pit design has applied conventional design criteria as has been successfully utilised in previous mining. The design was constrained to the top of the Underground workings. The design is shown in Figure ES-1.

From the run-of-mine (ROM) pad, ore material will be blended and hauled to the crusher via wheel loader. The production schedule indicates the material movement will vary over the life of the project (approximately 2 years) and have a maximum rate of 310,000 bcm per month. The mine production rate tapers down as haul lengths increase as the depth of the pit increases and the ratio of ore to waste increases.

Figure ES-1: Final Design with current Underground workings (purple)

Osborne Underground

It is proposed to mine the extension to the Osborne Underground Mineral Resource between 135 mRL and 60 mRL at a rate of 60 kilo tonnes per month (ktpmth) for 3 months and reducing to 25 ktpmth by the 6th month. Osborne was previously mined at 1.5 Mtpa by longhole open stoping to a depth of over 1,100 m below surface. The mine is serviced by a surface decline and haulage shaft to 700 m depth.

The mine will employ longhole open stoping methods with longitudinal uphole retreat consistent with previous activities during 2009 and 2010 at Osborne. Development mullock will be placed in open voids after mining is complete in the block. No other backfill will be used.

Numerical modelling work completed, by AMC Consultants (AMC), in 2012 for the stopes between 135 mRL and 60 mRL recommended that the rib pillars are modified as follows:

- The 110D/E and 60D/E rib pillars were increased by 3.0 m in width. This resulted in optimised pillar width increasing from 6.0 m to 9.0 m;
- The 85D/E rib pillar was increased by 4.0 m in width. This resulted in optimised pillar width increasing from 6.0 m to 10.0 m width; and
- The 110E/F, 85E/F and 60E/F rib pillars were increased by 3.5 m, 6.0 m and 1.0 m in width respectively. This resulted in optimised pillar width being 11.0 m, 8.5 m and 7.0 m respectively.

AMC considered that the sill pillars recommended by Barrick geotechnical staff were adequate.

An overall dilution rate of 10% has been applied based on an empirical assessment of the hangingwall and footwall rockmass quality.

A schematic of the designed stopes is presented in Figure ES-2.

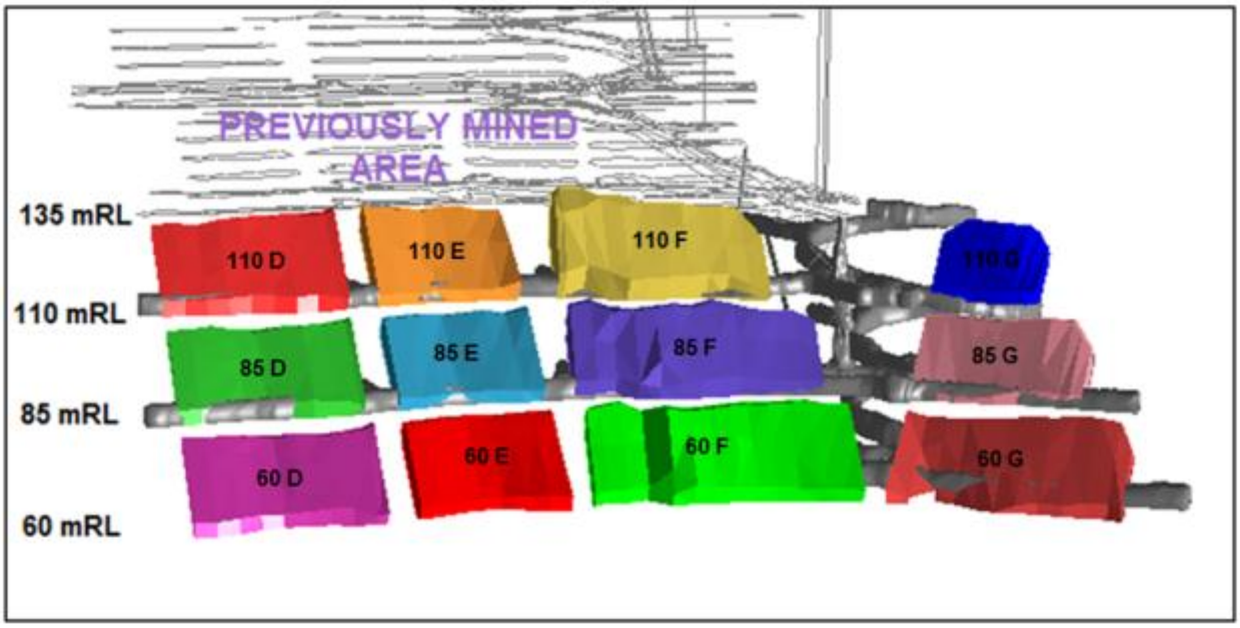

Figure ES-2: Long section of proposed stopes between 135 and 60 levels

Kulthor Underground

The mine is accessed through the existing Osborne Underground workings and will utilise the existing infrastructure. Material is planned to be truck hauled to the 1000 mRL ore pass for crushing and hoisting to surface via the existing Osborne shaft.

Preliminary geotechnical studies, by Northwind, have determined maximum stable open stope sizes to be no bigger than 20 m x 35 m. The geotechnical conditions in the three shoots varies, resulting in the use of longhole open stoping and longhole bench and fill. It is recommended that further geotechnical studies are undertaken to finalise the mining method and stope design criteria.

As part of Kulthor Feasibility Study (2007), a series of trade-off studies were conducted to determine the optimum mining method.

The mining methods selected for the Kulthor deposit are:

- Longhole open stoping (LHOS); and
- Longhole bench and fill (LHBF).

Longitudinal sub-level caving has been considered for the deposit, but further geotechnical studies have determined that this mining method is not suitable for the deposit.

Figure ES-3 shows the long section of the Kulthor Mine design.

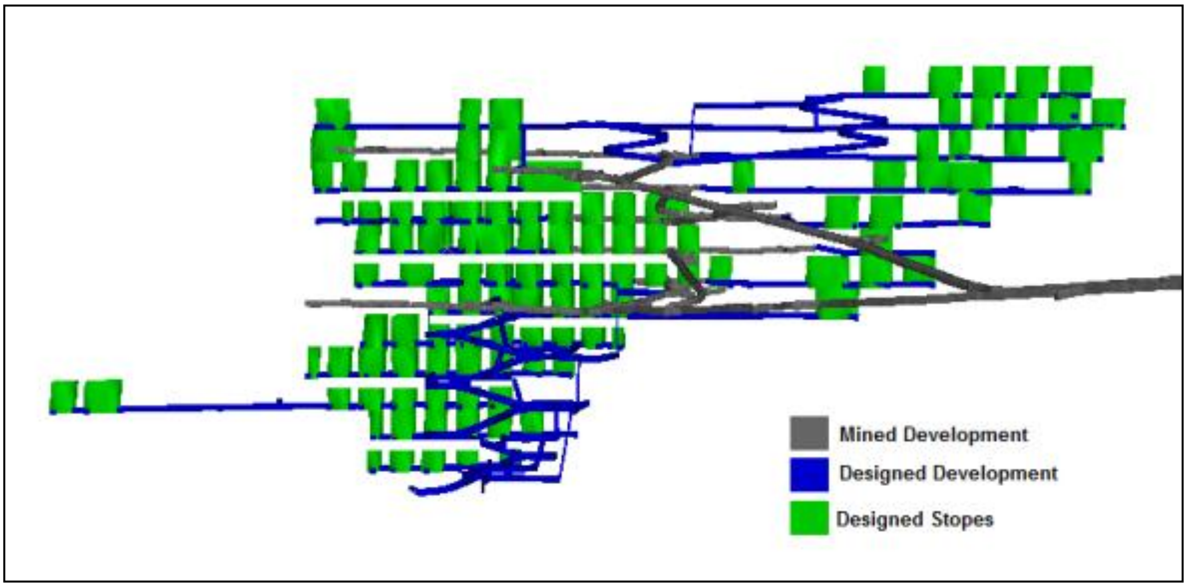

Figure ES-3: Long section of Kulthor stoping areas

Metallurgy and Recovery Methods

All ore is planned to be processed at the existing Osborne processing facility. This is a conventional sulphide flotation concentrator plant commissioned in 1995 and operated continuously until 2010 initially treating ore from the Osborne open pit, then from the Osborne underground mine and finally material from the Osborne underground and the Trekelano deposit 95 km to the northwest. The original flowsheet included a carbon-in-pulp gold recovery circuit, intended to recover around 50% of the gold as doré from a pyrite flotation concentrate. However, this system was abandoned after it was discovered that 60–70% of the contained gold reports to the copper concentrate, and a new gravity circuit, designed around a Knelson concentrator, was installed in its place.

Design capacity of the plant was 119 tph at a flotation feed density of 35% w/w solids but by 2008 actual throughput had reached ~265 tph at a flotation feed density of 50% w/w solids after a series of upgrades.

It is proposed to operate the plant at a nominal rate of 215 tph / 160,000 tpmth.

A summary of the metallurgical recoveries is presented in Table ES-7.

Table ES-7: Assumed metallurgical recoveries

Category	Copper Recovery (%)	Gold Recovery (%)	Copper Concentrate Grade (%)
Osborne Pit – Low Grade	60	45	23.5
Osborne Pit – High Grade	85	75	23.5
Osborne Underground	90	80	23.5
Kulthor Underground	85	75	23

Osborne Underground

Ore from the Osborne underground mine was the dominant feed to the Osborne concentrator and its metallurgical performance has been demonstrated over nearly 15 years of operation. The major minerals in the Osborne underground ore were iron oxides (principally magnetite), quartz and feldspar with minor amounts of chalcopyrite, amphibole, chlorite and iron sulphides. Minerals affecting flotation performance were pyrite, pyrrhotite, silica and talc.

The Metallurgical recoveries for Osborne Underground presented in Table ES-7 have three caveats that should be considered.

1 If the head grade is lower for the remaining material underground than that previously treated, then the flotation + gravity recoveries for copper and gold are likely to be lower as well;

2 Sales terms for copper concentrates have changed dramatically since last sales from the Osborne concentrator; and

3 Exposure of sulphides in underground workings could lead to “tarnishing” and oxidation adversely affecting flotation performance until sufficient new material has been stoped.

Kulthor Underground

The Kulthor Feasibility Study (Buxton 2007) had the following points on the processing of Kulthor material:

- “Time of grind” tests on Kulthor composites showed that the hardness was less than that for Osborne ore so specific grindability tests were not done; and
- Used metallurgical performance of 85% copper recovery and 75% gold recovery (gravity + flotation) into a concentrate assaying 23% copper.

The mineralogy of Kulthor material has some important differences to that of Osborne ore previously treated in the Osborne concentrator:

- Iron sulphide (pyrite + pyrrhotite): copper ratio varies from ~11:1 for Main Lode High Pyrrhotite to ~5.5:1 for Main Lode Low Pyrrhotite and North Lode Low Pyrrhotite compared with ~3:1 for Osborne Underground material previously treated in the concentrator. Osborne material had more magnetite and less iron sulphides; and
- Pyrrhotite to chalcopyrite ratio in western vein material from Kulthor is 0.6:1 compared with 0.2:1 for Osborne.

As mentioned for other sources, the recoveries presented in Table ES-7, will have to be adjusted downwards if a higher copper concentrate grade is targeted.

A programme of metallurgical test work will be required to better define the flotation performance (+ gravity for gold if applicable) for a definitive feasibility study, which should include grinding and flotation tests, preferably supported by quantitative mineralogy to determine both the metallurgical performance of the material and the suitability of the current flowsheet configuration and equipment in the Osborne concentrator to treat it. This should include mixed milling of the various ore types.

Osborne Open Pit

The Osborne copper concentrator should be capable of treating material with primary copper sulphide mineralisation.

Recoveries for the "Osborne Open Pit High Grade (Red)" material in Table ES-7 look optimistic considering the head grade of the open pit material is less than half that of ore treated to date from the Osborne underground mine.

Actual performance data from previous treatment of open pit material should be examined critically considering the much coarser flotation feed sizing practised in the Osborne concentrator at the end of its operating life, compared with initial operation when it processed material from the open pit.

A conservative assumption of a "constant tailing" hypothesis was applied to estimate the metallurgical performance for the "Osborne Open Pit Low Grade (Yellow)" material in Table ES-7.

An important component of the metallurgical performance is in the definition of the mineralisation. When Osborne started in 1995, the copper concentrate produced from the open pit contained excessive chlorine due to the presence of the mineral atacamite $Cu_2(OH)_3Cl$ in the oxide and transition zone material. The Osborne concentrator will not recover non sulphide copper minerals such as azurite, malachite, cuprite, tenorite and chrysocolla.

While azurite + malachite can readily be recovered by sulphidisation flotation, cuprite and tenorite are less amenable; chrysocolla is not recovered by this technique. Processing material with non-sulphide copper minerals means a loss of copper recovery or a reduction in throughput to provide the additional flotation capacity.

Hence, while the metallurgical performance data could be tentatively used for a scoping study, a programme of metallurgical test work will be required to better define the flotation performance (and gravity for gold if applicable) for a definitive feasibility study.

Aspects that should be examined include the following:

- Definition of the zone of primary copper sulphide mineralisation; and
- Grinding and flotation tests, to determine both the metallurgical performance of the material and the suitability of the current flowsheet configuration and equipment in the Osborne concentrator – this should include mixed milling of the various ore types.

Project Infrastructure

Most of the infrastructure required for the Osborne copper-gold project is already in-place has been successfully operated previously for a number of years and has recently been recommissioned having been on care and maintenance.

A detailed review of the fixed plant associated with the Osborne processing facility and Osborne Underground was completed by AECOM in 2011. Generally, this infrastructure appears to be in reasonable condition. However, most of the significant components are 16+ years of age. Therefore it would be prudent to allow for increased operating costs in the future.

The conclusions on the gyratory crusher warrant further analysis because it is nominated by SRK as the weakest link of this system. The duty of the crusher / feed source is different (reduced) to what was in place at the time of closing the Osborne operations.

Rotation of the mantle of the surface cone crusher through 90 degrees is highlighted by SRK as a significant risk, and in need of remedial works it was acknowledged by Ivanhoe as requiring work but no firm plans are in place. Rotation will enable the “inefficient” axis to be lined up to suit the most suitable ore flow from underground or the open pit. It is noted that the scheduled throughput of underground ore is 43% of the total feed; as such any inefficiencies are less likely to be material.

A forecast site power balance is recommended for future studies but based on previous operations, it is appropriate to assume sufficient power can be generated to meet ongoing requirements.

It is recommended a site wide water balance be undertaken to demonstrate the capacity of the existing system to deliver the water requirements. It is understood that work on a site-wide water balance has commenced.

Market Studies and Contracts

Concentrate products will be trucked 980 km to and shipped through the port of Townsville via 30 t containers tipped directly into the ships-hold.

A contract is in place with Northern Stevedoring Services Pty Ltd (NSS) of AUD110/dry metric tonne (dmt). This price includes all transport and handling of concentrate from site and ship loading at the Townsville port.

Loading of ships with half-containers is currently being done at Townsville port. The volume of concentrate produced in 2012 is estimated to be 50,000 t, or four parcels. The key export market for Osborne copper concentrate will be either China or Japan depending on deliveries by traders.

For the purposes of this study, it is expected that all product will be sold on the spot market, with the potential to negotiate fixed term contracts available at a later stage of project development.

Environmental Studies, Permitting and Social Impacts

A summary of permits relating to the project is provided in Table ES-8.

Table ES-8: Osborne (M90040), Kulthor (ML90158) and Borefield (ML90057)

Permit	Expiry Date	Description	Comment
Environmental Authority (EA) MIN100459006	-	Environmental approval for a Level 1 mining project,	A revised EA was issued in June 2011 A Plan of Operations (revised in May 2011) and Environmental Management Plan linked to EA MIN100459006 have been approved by DEHP The PoO also applies to proposed activities at the Lucky Luke deposit.
Groundwater Licence 90311J	30 Nov 2011	Groundwater licence authorising abstraction of up to 947 ML per annum from the ‘Longsight Sandstone’ borefield, 25 km southwest of the site	The bore field and access to and from Osborne is covered by ML90057
Groundwater Licence 69987J	30 Nov 2011	Groundwater licence authorising abstraction of up to 35 ML per annum from the ‘Longsight Sandstone’ borefield, 25 km southwest of the site	This is understood to be for the ‘Carbo Bore’
Groundwater Licence 404161	31 Aug 2011	Allows for the abstraction of 365 ML per annum from the Kulthor Fault Shear Zone	This licence is for mining and dewatering, and therefore allows for extraction of whatever is required to dewater the mine.

From conversations with Ivanhoe staff, it is understood that the following changes to the permits will be required as part of the restart and expansion:

- Access road Osborne to Merlin (and Starra 276) – registered access to ML90187 covers 38 km of 54 km and variation of access to ML2961 and minor Environmental Authority amendment and Plan of Operations (PoO). It is understood that this is now in place; but SRK has not sighted this;
- Associated borrow pits (Quarry permits) ERA16, dredging permit and Riverine Protection Permits. It is understood that these are now in place; but SRK has not sighted these permits; and
- EA amendments and PoO to authorise proposed mining activities and tailings strategy.

As Ivanhoe are also currently investigating treatment of molybdenum and rhenium, including roasting, it is recommended that approval for such activities be delayed to avoid triggering an EIS prior to copper-gold amendments being approved.

SRK considers that there remains a possibility of amendments to the Osborne Project EA to trigger an EIS process. SRK considers that it is less likely that a requirement to prepare an EIS would be triggered if the applications for approvals to establish a molybdenum / rhenium roaster at Osborne are sought separately to the amendments to existing EAs for the proposed copper-gold project.

Tenure

Mining leases are issued and administered by the Queensland Department of Natural Resources, and Mines (DNRM) A lease is granted under the Mining Resources Act 1989 (MR Act) and is required for mining activities to be carried out.

The Osborne operation, Kulthor deposit and Bore Field are located on ML90040 (1770.9 ha), ML90158 (2152 ha) and ML90057 (64.4 ha) respectively. These leases are contiguous and ML90040, 90158 and part of 90057 are within exploration lease EPM9624 which was originally granted in 1993. Environmental approval for EPM9624 was granted for environmentally relevant activity (ERA) Number 20 (Mineral exploration or mining – exploring for or mining for minerals under a mining authority). The Osborne Kulthor and Borefield leases are now covered by Environmental Authority (EA) MIN100459006 which is discussed in detail in Section 20. Both mining leases are fully owned by Ivanhoe.

Public reports covering the above leases were generated via the ‘Minesonline’ website maintained by DNRM and the following observations are noted in relation to these reports

- ML90040 covers the following minerals/purpose; silver ore, gold, cobalt ore, copper ore and iron (magnetite);
- ML90158 covers the following minerals/purpose; silver ore, gold-silver ore, gold, cobalt ore, copper ore, iron (magnetite) and treatment plant/site; and
- ML90057 covers the following minerals/purpose; silver ore, gold, cobalt ore, iron-magnetite, water supply.

ML90215 and ML90217 cover the following purposes; pipeline – water only, power line/aerials, transport – vehicular haul road. No comment on the native title process is given.

Capital and Operating Cost Estimates

Capital Cost Estimates

The total capital requirement for the Osborne and Kulthor deposits is summarised in Table ES-9. These costs are associated with fixed plant infrastructure and capital development.

Table ES-9: Osborne copper-gold project - Capital Cost Estimate

Item	Total (AUD M)
Osborne Open Pit	29.02
Osborne Underground	0
Kulthor Underground	43.57
Total	**72.59**

Operating Cost Estimates

The total operating mining costs for the Osborne Pit cutback associated with the continued excavation by contractor from commencement of production have been included as operating costs and are summarised in Table ES-10.

Table ES-10: Osborne Open Pit Total Operating Cost Estimate

Item	Unit Cost (AUD / ore t)	Total Cost (AUD M)
Mining	14.97	37.41
Processing	10.60	26.49
General and Site Administration	4.24	10.59
Off Site Costs	5.99	14.96
Total	**35.79**	**89.45**

The total operational costs for Osborne and Kulthor underground operation(s) are summarised in Table ES-11 and Table ES-12.

Table ES-11: Osborne Underground - Operating Cost Estimate

Item	Unit Cost (AUD /t)	Total Cost (AUD M)
Mining	32.11	15.78
Processing	10.60	5.21
General and Site Administration	7.30	3.59
Off Site Costs	15.34	7.54
Total	**65.35**	**32.10**

Table ES-12: Kulthor Underground - Operating Cost Estimate

Item	Unit Cost (AUD /t)	Total Cost (AUD M)
Mining	37.89	97.58
Processing	10.60	27.30
General and Site Administration	7.30	18.80
Off Site Costs	11.30	29.09
Total	**67.09**	**172.76**

The processing unit cost of AUD10.60 /t per tonne has been estimated from Osborne operating budget.

General and Site Administration Costs

The costs that have not been included elsewhere in the operating costs inclusive of maintenance, site management, occupational health and safety, accounting and finance, warehousing and logistics and information services equates to AUD9.08 M for Osborne Open Pit, AUD3.59 M for Osborne Underground and AUD18.8 M for Kulthor.

Offsite Costs

Concentrate handling costs of AUD119.90 /dmt of concentrate, and third party shipping broker costs of AUD43.60 /dmt shipped to overseas smelters, allowing for 9% moisture, has been applied.

Forecast treatment and refining charges in line with market reports of USD55 / concentrate tonne and USD0.055 / lb copper and USD5 / oz gold have been used.

Costs associated with Marketing and Assays have been estimated using historical data from Osborne operation prior to suspending shipping and sales in February 2011. An allowance of AUD5.5 /dmt of concentrate has been applied.

The following royalty rates have been applied for the Osborne project:

- Queensland Government Copper Royalty: 4.8% sales; and
- Queensland Government Gold Royalty: 5.0% of sales.

Economic Analysis

It has been assumed that sufficient mill feed will be available to maintain the milling rate and cost profile when each deposit is mined. The analysis is based on the Mineral Reserve estimates.

Osborne Open Pit

The key metrics for the Osborne Open Pit are summarised in Table ES-13. No discounted cashflow, payback period calculations or sensitivities were undertaken because the Osborne Open Pit mine life is under two years.

Table ES-13: Summary of Key Financial Parameters

Parameter	Units	Value
Tonnes Milled	t	2,499,389
Total OPEX	AUD M	89.45
Total CAPEX	AUD M	12.77
Royalty	AUD M	7.33
Total Cost	AUD M	109.55
Copper Produced	Mlb	33.14
Gold Produced	ozs	28,950
Total Revenue	AUD M	148.2
Cashflow	AUD M	21.23
IRR	%	17.6

Osborne Underground

The key metrics for the Osborne Underground are summarised in Table ES-14. No discounted cashflow, IRR, payback period calculations or sensitivities were undertaken because the Osborne Underground mine life is under one year.

Table ES-14: Summary of Key Financial Parameters

Parameter	Units	Value
Tonnes Milled	t	491,304
Total OPEX	AUD M	32.10
Total CAPEX	AUD	0
Royalty	AUD M	3.35
Total Cost	AUD M	35.45
Copper Produced	Mlb	16.58
Gold Produced	ozs	9,779
Total Revenue	AUD M	67.59
Cashflow	AUD M	32.13

Kulthor Underground

The key metrics for the Kulthor Underground are summarised in Table ES-15.

Table ES-15: Summary of Key Financial Parameters

Parameter	Units	Value
Tonnes Milled	t	2,575,058
Total OPEX	AUD M	172.76
Total CAPEX	AUD M	43.57
Royalty	AUD M	13.82
Total Cost	AUD M	230.14
Copper Produced	Mlb	63.83
Gold Produced	ozs	55,116
Total Revenue	AUD M	284.60
Cashflow	AUD M	54.46
Discounted Cashflow (EBIT) (8.6%)	AUD M	38.43
IRR	%	71
Payback period	Year	1.8

Several sensitivities were analysed for the Kulthor underground cashflow model. The sensitivities were applied at ± 10% to determine which changes have the highest impact on the project. Figure ES-4 shows the results from the sensitivity analysis. Commodity prices and metal recovery have largest impact on the project financial results.

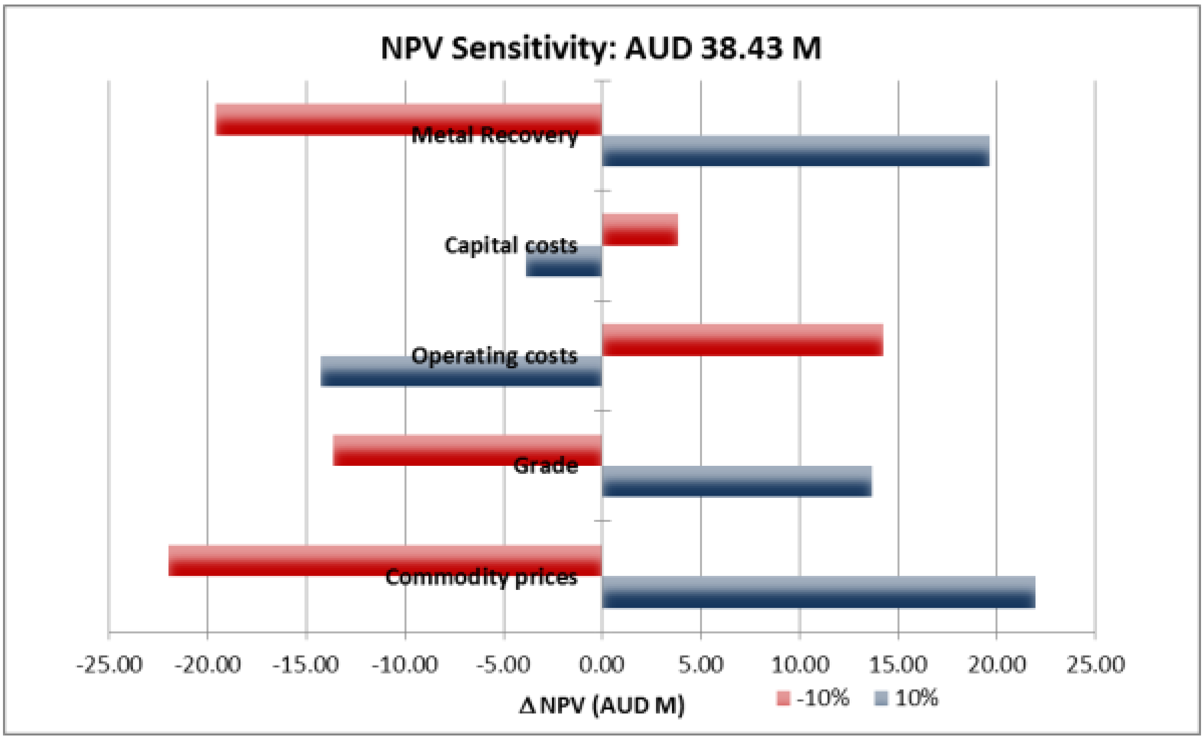

Figure ES-4: Impact of Sensitivities on the Project

Adjacent Properties

There are a number of projects and Resources that are owned by Ivanhoe within the Cloncurry district in proximity to those discussed in this study. The Starra Line, Merlin, Mt Dore and Lady Elliott deposits are under evaluation by Ivanhoe. While these deposits may not be directly related to / influenced by the each other in terms of the mineralisation, the proximity of the deposits presents the opportunity to share infrastructure.

While other deposits such as Lucky Luke and Houdini are referred to in this report, they are not sufficiently advanced to be considered material to this Technical Report.

Other Relevant Data and Information

SRK and LMRC consider that all data and information relevant to the Osborne copper-gold project has been disclosed by Ivanhoe and discussed appropriately in this Technical Report.

Interpretation and Conclusions

From the sensitivity analysis, the Kulthor Underground deposit has been shown to be sensitive to commodity price and metallurgical recovery. Any variance to these items would have a significant impact, positively or negatively, to the overall financial performance of the project as shown in Figure ES-4.

The Osborne Open Pit, Osborne Underground and Kulthor deposits are part of an overall mining strategy for the Osborne copper-gold project to provide mill feed to the Osborne processing plant. Mining of the deposits contribute to the overall mill feed and ensures that the processing plant is utilised to capacity. If the processing plant is not fully utilised, this has an impact on the operating costs of the project and potentially makes the remaining deposits uneconomical to mine.

The existing surface infrastructure will operate at below its historical capacity. This reduces the project risk to inefficiencies and provides potential for an increase in throughput without significant injections of capital costs.

Cost estimates for the concentrate handling and power generation are based on current base cases. Ivanhoe is currently engaging in discussions that have the potential to reduce both operating and capital costs.

The technical and financial aspects for each of the deposits in the Osborne project have been shown to be robust at this level of study. The Mineral Reserve based on a NI 43-101 compliant Mineral Resource estimate is at a pre-feasibility study level of detail and supports the reporting of Mineral Reserves.

Comments and Recommendations

There has not been a work programme recommended because the deposits have been incorporated into the Osborne Mine Development Plan.

Osborne Open Pit

The Open Pit Resources are well drilled already. The current Open Pit design is considered conservative as it avoids interaction with previously-mined stopes and underground development. There is potential for a considerable increase in Resources, depending on mining and economic constraints.

LMRC considers that the blocks located within the conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a Mineral Resource.

Osborne Underground Mine

The Resources included within the Mineral Reserve are well defined. Establishment of a Resource definition drilling programme has the potential to convert Inferred Resources at depth thereby extending the known Mineral Resource.

The grade of the Resources decreases with depth, so it will be important to increase the amount of drilling in the lower parts of the 1SS Zone.

Kulthor Underground Mine

The Kulthor mineralisation is now exposed in development and has received additional underground and surface drilling.

The Resource definition drilling programme should be continued to increase the proportion of Measured Resources and convert the Inferred Resources to Measured and Indicated Resources.

Reconciliation and assessment of mined stopes should be undertaken to understand the impact of the shear zone on the stope performance and for revision of the modifying factors.

The Kulthor Mineral Reserves in this Technical Report are based on the 2011 Kulthor Mineral Resource Model. The Kulthor Mineral Reserve should be re-estimated with the Kulthor Mineral Resource reported in the Technical Report.

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List of Abbreviations

Abbreviation	Meaning
AC	Air core
AAS	atomic absorption spectroscopy
AECOM	AECOM Pty Ltd
AMC	AMC Consultants Pty Ltd
AMG	Australian Map Grid
ANFO	Ammonium nitrate-fuel oil
ASX	Australian Securities Exchange (www.asx.com.au)
AUD	Australian Dollar
Barrick	Barrick (PD) Australia Pty Ltd
bcm	bank cubic metre
BCom	Bachelor of Commerce
BEng	Bachelor of Engineering
BSc	Bachelor of Science
CASA	Civil Aviation Safety Authority
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CV	coefficient of variation
DEHP	Department of Environment and Heritage Protection
DNRM	Department of Natural Resources and Mines
dmt	dry metric tonne
DTH	down-the-hole
EA	Environmental Authority
eCu	Copper equivalent (eCu = copper + 0.6 gold)
ED	Environmental Dam
EDTA	a crystalline acid, ethylenediaminetetraacetic acid
EIL	environmental investigation levels
EIS	Environmental Impact Study
EMP	Environmental Management (Plan)
EM	electromagnetic
EMI	Electromagnetic induction
EMS	Environmental management system
EPA	Environmental Protection Agency
EP Act	Environmental Protection Act 1994
EPBC	Environment Protection and Biodiversity Conservation (Act 1999)
EP Reg	Environmental Protection Regulation 2008
FAusIMM (CP)	Chartered Professional Fellow of the Australian Institute of Mining and Metallurgy
FAusIMM	Fellow of the Australian Institute of Mining and Metallurgy
FEL	front end loader
FIFO	fly in / fly out
FW	footwall
geologs	electronic drill logs
g/t	grams per tonne
HIL E	Health investigation levels
HW	hangingwall
HV	high voltage

Abbreviation	Meaning
ICM	Ivanhoe Cloncurry Mines Pty Ltd
ICP-AES	ICP-atomic emission spectroscopy
ID^2	inverse distance squared
ID^3	inverse distance cubed
IOCG	iron oxide copper-gold
Ivanhoe	Ivanhoe Australia Ltd (ASX and TSX listing code - IVA)
JORC	Joint Ore Reserves Committee
KL	kilolitre
koz	kilo ounces
kt	kilo tonne
ktpa	kilo tonnes per annum
kV	kilo volt
kVA	kilo volt ampere
kW	kilo watt
kWh	kilo watt hour
L/s	litres per second
LHBF	longhole bench and fill
LHOS	longhole open stoping
LMRC	Lewis Mineral Resource Consulting Pty Ltd
M	million
Ma	million years
MAusIMM (CP)	Chartered Professional Member of the Australian Institute of Mining and Metallurgy
MBS	MBS Environmental Pty Ltd (Martin Bosch Sell)
mg/kg	milligram per kilogram
mg/L	milligrams per litre
mH	metres high
ML	mining lease
ML	million litres
mm	millimetres
Mm^3	Million cubic metres
MPa	Mega pascals
MR Act	Mining Resources Act 1989
mRL	metres reduced level
m/s	metres per second
m^3/s	cubic metre per second
mS/cm	millisiemens per centimetre
MSc	Master of Science
$m^3/s/KW$	cubic metre per second per kilowatt
Mtpa	million tonnes per annum
mW	metres wide
MW	megawatt
NAF	non acid-forming
NGER	National Greenhouse and Energy Reporting Act 2007
NI 43-101	National Instrument 43-101
NN	Nearest Neighbour estimation method
Northwind	Northwind Pty Ltd

Abbreviation	Meaning
NPI	National Pollution Inventory
NRM	Natural Resource Management
NSS	Northern Stevedoring Services Pty Ltd
NW	Northwest
ozs	ounces
PAF	potentially acid-forming
PDAP	Placer Dome Asia Pacific Limited
PEA	Preliminary Economic Assessment
PFS	pre-feasibility study
PoO	plan of operation/s
PORX	porphyroblastic alteration / recrystallization
POX	partially oxidised
PSM	Pells Sullivan Meynink Pty Ltd
QA/QC	quality assurance/quality control
QG	Quantitative Group Pty Ltd
QP	Qualified Person
RAR	return air raise
RC	reverse circulation
RF	Revenue factors
RLP	relative by pair
RO	reverse osmosis
ROM	run-of-mine
RQD	rock quality designation
SFA	Screen fire assays
SG	Specific Gravity
SRK	SRK Consulting (Australasia) Pty Ltd
STP	Sewer treatment plant
t	tonnes
TDS	total dissolved solids
TOX	totally oxidised
tph	tonnes per hour
TSF	tailings storage facility
TSX	Toronto Stock Exchange
UGKM	Underground Kulthor mine
USD	US Dollars
w / w	weight / weight

2 Introduction and Terms of Reference

Ivanhoe Australia Limited (Ivanhoe) is an Australian-based Resource company listed on both the Australian Securities Exchange and Toronto Stock Exchange as IVA. It has assembled a significant package of Mineral Resources in the highly mineralised Cloncurry district, near Mt Isa, Queensland. Exploration success so far has revealed a portfolio of copper, molybdenum, rhenium and gold Mineral Resources. This report considers only the copper and gold assets, which include the Osborne and Kulthor deposits.

In October 2012, Ivanhoe commissioned SRK Consulting (Australasia) Pty Ltd (SRK) to prepare a Technical Report in line with Canadian and internationally recognised National Instrument 43-101 reporting standards (NI 43-101) and Form 43-101F1. The Mineral Resource and Reserve Statements reported herein were prepared in conformity with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines."

Ivanhoe Cloncurry Mines Pty Ltd (ICM) owns the leases for each of the deposits reviewed. ICM is a wholly owned subsidiary of Ivanhoe.

Ivanhoe separately engaged Lewis Mineral Resource Consulting Pty (LMRC) to prepare a Mineral Resource Report for the Osborne copper-gold project deposits, which included the Osborne Underground, Osborne Open Pit and Kulthor Underground deposits.

Richard Lewis from LMRC will act as Qualified Persons for reporting of the Mineral Resources. SRK reviewed the Osborne Project Mineral Resource report.

SRK was engaged to prepare the consolidated Technical Report for the Osborne Open Pit, Osborne Underground and Kulthor deposits, including a review of mining and processing operations, infrastructure requirements and environmental aspects.

This Technical Report summarises the technical information available on the Osborne copper-gold assets and demonstrate that the property clearly qualify as an "Advanced Exploration Property" as defined by the Toronto Stock Exchange.

2.1 Scope of Work

The scope of work includes the compilation of a Technical Report on the Osborne Open Pit and Underground and the Kulthor deposit in compliance with NI 43-101 and Form 43-101F1 guidelines. This work consisted of updating the Osborne NI43-101 Technical Report with the inclusion of the updated Mineral Resource for Kulthor.

2.1.1 Mineral Resource Report – Osborne Deposits

Ivanhoe engaged LMRC in April 2011 to complete an Independent Mineral Resource Estimate and report for the Osborne copper-gold project, suitable for NI 43-101 and JORC compliant reporting.

Ivanhoe provided the exploration data used by LMRC and undertook the data collation, interpretation and preliminary modelling. Most of the exploration data are historical information which LMRC has reviewed. Geological interpretation and modelling were reviewed by LMRC and the Mineral Resource estimation completed by LMRC.

Ivanhoe engaged LMRC to undertake and update of the Kulthor Mineral Resource during August and September 2012.

2.2 Work Programme

The Osborne copper-gold deposits Mineral Resource and Reserve Statement reported herein is a collaborative effort between SRK, LMRC and Ivanhoe personnel. The exploration database was compiled and maintained by Ivanhoe, and was audited by LMRC. The geological model and outlines for the copper-gold mineralisation were constructed by LMRC from a two-dimensional geological interpretation provided by Ivanhoe. In the opinion of LMRC, the geological model is a reasonable representation of the distribution of the targeted mineralisation at the current level of sampling. The geostatistical analysis, variography and grade models were completed by LMRC during May, June and July 2011 (Osborne), and August and September 2012 (Kulthor).

The Mineral Resource Statement reported herein was prepared in conformity with generally accepted CIM "Exploration Best Practices" and "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. This Technical Report was prepared following the guidelines of the NI 43-101 and Form 43-101F1.

This Technical Report was assembled by SRK in October and November 2012.

2.3 Basis of Technical Report

This report is based on information collected by SRK during a site visit undertaken between 22 and 27 April 2011, and any additional information provided by Ivanhoe throughout the course of SRK's investigations. A further site visit was undertaken between 11 and 15 June 2012. Other information was obtained from the public domain. SRK has no reason to doubt the reliability of the information provided by Ivanhoe.

This Technical Report is based on the following sources of information:

- Discussions with Ivanhoe personnel;
- Inspection of the Ivanhoe's Osborne copper-gold assets;
- Review of exploration data collected by Ivanhoe; and
- Additional information and studies provided by Ivanhoe.

2.4 Qualifications of SRK and SRK Team

The SRK Group comprises 1,500 professionals, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This fact permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

The compilation of this Technical Report was completed by Anne-Marie Ebbels, Principal Consultant (Mining), MAusIMM (CP), BEng. By virtue of her education, membership to a recognised professional association and relevant work experience, Anne-Marie Ebbels is an independent Qualified Person as this term is defined by NI 43-101.

Dale Sims, SRK Associate Principal Geologist conducted a site visit to verify and validate geological interpretation and data used in this report. Dale Sims, FAusIMM (CP), is by virtue of his education, membership to a recognised professional association and relevant work experience, an independent Qualified Person as this term is defined by NI 43-101.

Peter Munro, SRK Associate Principal Metallurgist, undertook a review of the metallurgical and mineral processing aspects of the project. Peter Munro, Principal Consulting Engineer (Metallurgy), BAppSc, BEc, BCom, FAusIMM, SRK Associate, is by virtue of his education, membership to a recognised professional association and relevant work experience, an independent Qualified Person as this term is defined by NI 43-101.

The lead author of this Technical Report is Anne-Marie Ebbels.

Internal SRK Peer Review of this Technical Report was completed by **Peter Fairfield**, Principal Consultant (Project Evaluations), who conducted a peer review of non-geological aspects of this Technical Report. Peter Fairfield, BEng, FAusIMM, is by virtue of his education, membership to a recognised professional association and relevant work experience, an independent Qualified Person as this term is defined by NI 43-101.

Table 2-1 lists the individuals who, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in NI 43-101 for this Technical Report. The table defines the areas of responsibility for the QP who all meet the requirements of independence as defined in NI 43-101.

Other experts, whose contributions the QP have relied on, are included in Table 2-2.

Table 2-1: List of Qualified Persons

Qualified Person	Position	Employer	Last Site Visit Date	Professional Designations	Area of Responsibility and Report Sections
Anne-Marie Ebbels	Principal Consultant (Mining)	SRK	13 - 14 June 2012	BEng (Mining), GDip (Computer Studies), MAusIMM (CP)	Lead Author and author of all other undesignated sections
Peter Fairfield	Principal Consultant (Project Evaluations)	SRK	16 - 17 May 2011	BEng (Mining), FAusIMM	SRK Peer Review
Richard Lewis	Principal Consultant	Lewis Mineral Resource Consulting	15 - 17 March 2011	MSc (Geol), FAusIMM	Osborne Geology 5 to 12, 14, 23.1,26.1
Peter Munro	Senior Principal Consulting Engineer	Mineralurgy Pty Ltd	1 - 2 August 2004[1]	BAppSc, BEc, BCom, FAusIMM	Metallurgy and Processing 13, 17

1 There have been no material changes to the concentrator since August 2004.

Table 2-2: List of Contributing Authors

Expert	Position	Employer	Last Site Visit Date	Professional Designations	Report Sections
Duncan Pratt	Senior Consultant (Mining)	SRK	N/A	BEng (Hons), MAusIMM (CP)	Sections16.1,21,22
Hugh Thompson	Principal Mining Engineer	GHD	16 - 17 May 2011	BEng (Mining), MAusIMM	Project Infrastructure Section 18
Lisa Chandler	Principal Consultant (Geoenvironmental)	SRK	N/A	MEng (Civil), MEIANZ, RABQSA	Environmental Section 20
Troy Hindmarsh	Consultant (Environmental)	SRK	16 - 7 May 2011	BSc Hons (Env Sc), MEIANZ	Environmental Section 20

2.5 Site Visit

In accordance with NI 43-101 guidelines, Peter Fairfield, Jeff Price, Troy Hindmarsh and Hugh Thompson visited the Osborne operations on 16 and 17 May 2011, accompanied by Timothy Fisher of Ivanhoe.

The purpose of the site visit was to review the site infrastructure, understand proposed mine plans, material transport, interview project personnel and to collect all relevant information for the preparation of the Mineral Resource reports and the compilation of a technical report.

In accordance with NI 43-101 guidelines, Anne-Marie Ebbels visited the Osborne operations on 13 June and 14 June 2012, accompanied by Adrian Pratt and Mike McCracken of Ivanhoe.

The purpose of the site visit was to review the proposed mine plans, material transport, interview project personnel and to collect all relevant information for the compilation of a technical report.

SRK was given full access to relevant data and conducted interviews of Ivanhoe personnel to obtain information on the past work, to understand procedures used to collect, record, store and analyse historical and current data.

2.5.1 Osborne Copper-Gold Project Mineral Resource Report

Richard Lewis, MSc (Geology), FAusIMM, Principal of LMRC visited the Osborne site on 15 to 17 March 2011, and 28 to 30 May 2012 accompanied by D Crimeen of Ivanhoe. Mr Lewis made several previous site visits between 1998 and 2009 to supervise exploration and complete Mineral Resource Estimates. Richard Lewis, by virtue of his education, membership to a recognised professional association and relevant work experience, is an independent Qualified Person as this term is defined by NI 43-101.

The purpose of the site visits were to meet Ivanhoe staff and obtain the full scope of the work required. The drillhole and other Mineral Resource data and reports needed to complete new Mineral Resource estimates and a Technical Report were also collected. A selection of the drill core for Kulthor was also viewed. Previous Osborne and Kulthor geological modelling and Mineral Resource estimation work was discussed with D Crimeen.

This document references the Summary memo, “LMRC Osborne Resource July 2011”, dated July 21, 2011 and described in the Technical Report: Osborne NI 43-101 Technical Report located in northwest Queensland Region of Australia, 29th August 2012. SRK Consulting (Australasia) Pty Ltd, IVA005.

2.6 Acknowledgement

SRK and LMRC would like to acknowledge the support and collaboration provided by Ivanhoe and personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

2.7 Declaration

SRK’s opinion contained herein and effective 06 July 2012 is based on information collected by SRK throughout the course of SRK’s investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages.

Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Ivanhoe, and neither SRK nor any affiliate has acted as advisor to Ivanhoe, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 Reliance on Other Experts

Exploration has been completed by Ivanhoe and previous owners of the project (Placer Dome Asia Pacific Limited and Barrick Gold Limited). LMRC reviewed all exploration field processes and data used for the Mineral Resource estimate and undertook all Mineral Resource estimation. Richard Lewis from LMRC has overseen or completed all tasks and is the QP nominated for reporting of Mineral Resources associated with the Osborne Project.

LMRC has not verified the title of the Osborne Deposits mineral tenements which was compiled by SRK from advice provided by Ivanhoe.

SRK has not performed an independent verification of land title and tenure as summarised in Section 4 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties.

SRK was informed by Ivanhoe that there are no known litigations potentially affecting the Osborne copper-gold assets, and SRK relied on Ivanhoe for advice on the mineral tenure currency and liabilities.

4 Property Description and Location

4.1 Location

The Osborne copper-gold project is located in northwest Queensland, Australia, approximately 195 km southeast of Mt Isa and 700 km west-southwest of Townsville. Figure 4-1 presents the Osborne site location in relation to Australia and Queensland respectively.

The Osborne copper-gold project encompasses the Osborne Open Pit and Underground mines and the Kulthor Underground deposit.

The Osborne copper-gold underground mine is located at latitude 22° 04' south, longitude 140° 23' east. The deposit is situated some 195 km southeast of Mt Isa in North West Queensland, Australia.

The Kulthor copper-gold deposit is located approximately 2 km west of Osborne and is connected via decline.

Figure 4-1: Location of Osborne in relation to Australia

4.2 Tenure

The following summary was prepared by SRK (SRK Consulting, 2012) and LMRC based on information provided by Ivanhoe and historical reports. In preparing the summary, SRK and LMRC have relied on information provided by Ivanhoe.

The Osborne 1SS and open pit deposits are located on ML90040. The Kulthor deposit is on ML90158 (Kulthor) and exploration lease EPM9624 (Trough Tank) which is held 100% by Ivanhoe (Osborne) Pty Ltd. The tenement corner points were surveyed-in by an accredited surveyor from M H Lodewyk Pty Ltd of Mt Isa (Lodewyk, 2010). Details of these and other tenements owned by Ivanhoe are outlined in Table 4-1.

Table 4-1: Mineral Tenure Table

Tenement	Name	Status	Company	Area (ha)	Grant Date	Expiry Date
ML90040	Osborne	Granted	Ivanhoe (Osborne) Pty Limited	2,152	9/06/1994	30/06/2014
ML90057	Osborne Borefield & Services	Granted	Ivanhoe (Osborne) Pty Limited	64.4	15/12/1994	31/12/2014
ML90068	Osborne Concentrate Loading	Granted	Ivanhoe (Osborne) Pty Limited	18	29/06/1995	30/06/2014
ML90158	Kulthor	Granted	Ivanhoe (Osborne) Pty Limited	1,770.9	28/06/2007	30/06/2027
EPM9624	Trough Tank	Granted	Ivanhoe (Osborne) Pty Limited	333.9	10/11/1993	9/11/2015

Figure 4-2 and Figure 4-3 show the location of the tenements, the Osborne and Kulthor deposits and mining and processing infrastructure located at the northern end of ML90040. ML90057 is very narrow as it covers the bore field pipeline. The Kulthor deposit is accessed by a drive from the Osborne underground workings and will have little infrastructure at the surface.

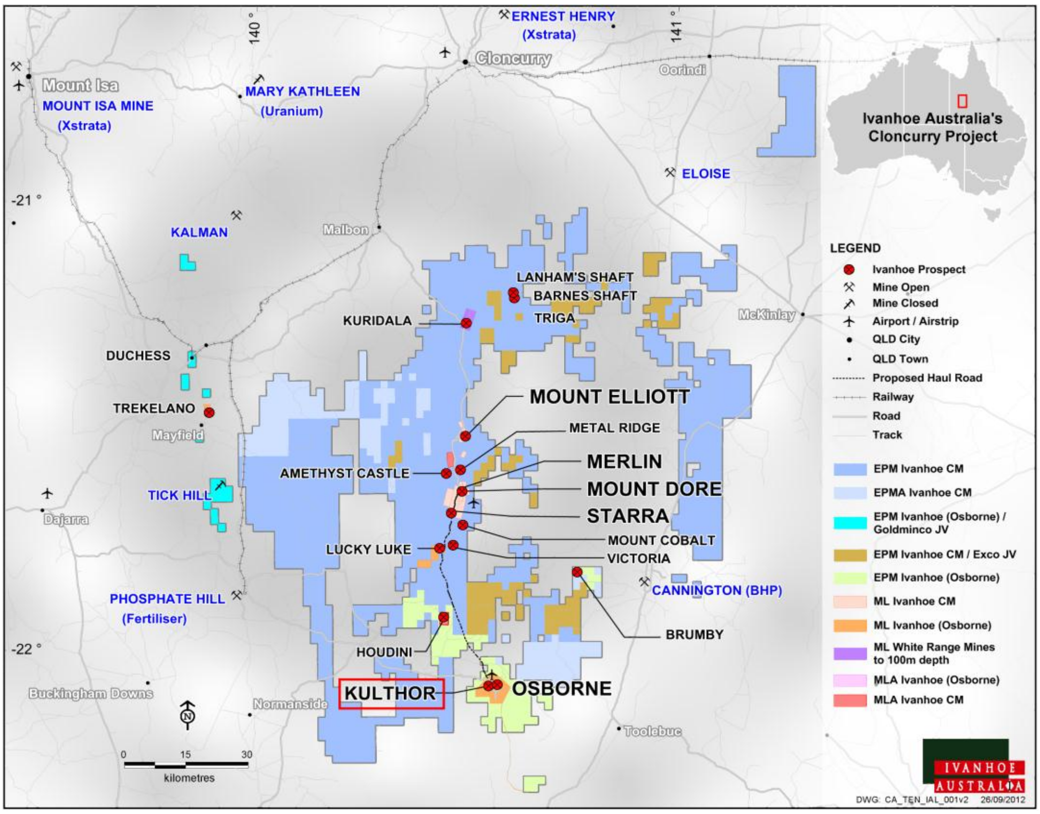


Figure 4-2: Location of the Osborne and Kulthor Tenements

Source: Ivanhoe Quarterly Report, June 2011

Figure 4-3: Land Tenure Map

Source: Ivanhoe, 2011

4.3 Property Tenure and Agreements

ML 90040, ML 90057, ML 90068 and ML 90158 are 100% owned by Ivanhoe (Osborne) Pty Limited, a wholly-owned subsidiary of Ivanhoe.

The Osborne operation, Kulthor deposit and Bore Field are located on ML90040 (1,770.9 ha), ML90158 (2,152 ha) and ML90057 (64.4 ha) respectively. These leases are contiguous and ML90158 and part of ML90040 and ML90057 are within exploration permit EPM9624 which was originally granted in 1993. Environmental approval for EPM9624 was granted for environmentally relevant activity (ERA) Number 20 (Mineral exploration or mining – exploring for or mining for minerals under a mining authority). These MLs provide sufficient rights for the mining of the Osborne and Kulthor deposits.

Public reports covering the above leases were generated via the 'Minesonline' website maintained by DNRM and the following observations are noted in relation to these reports:

- ML90040 covers the following minerals/purpose; silver ore, gold, cobalt ore, copper ore and iron (magnetite);
- ML90158 covers the following minerals/purpose; silver ore, gold, cobalt ore, copper ore, iron (magnetite) and treatment plant/mill site; and
- ML90057 covers the following minerals/purpose; silver ore, gold, cobalt ore, copper ore, iron-magnetite, water supply.

Where a Mining Lease or the registered access to a Mining Lease is situated over property owned by a third party, the Mineral Resources Act 1989 requires that compensation is determined between the mining tenement applicant and the relevant owner of land before a Mining Lease will be granted or renewed. Confidential compensation agreements are in place for Mining Leases situated over property owned by a third party for ML 90040, ML 90057 and ML 90158. No compensation agreement is required for ML90068.

A royalty equivalent to 2% on net smelter returns on all products produced from ML 90040, ML 90057, ML 90068 and ML 90158 is payable to Barrick (PD) Australia Limited.

Further information regarding the environmental liabilities for the MLs is discussed in Section 20. There are no other known factors and risks that may affect access, title or the right or ability to perform work of the MLs for the Osborne copper-gold project.

4.4 Mining Lease Costs

The general conditions of a Mining Lease are stated in the Mineral Resources Act 1989 under Section 276 and include payment of rents and rates for granted Mining Leases. The annual rental for the Mining Leases is detailed in Table 4-2.

Table 4-2: Mining Lease Costs

Lease	Rent (AUD)
Ivanhoe (Osborne) Mining Leases ML 90040, ML 90057, ML 90068 and ML 90158 for the year beginning 1 September 2012	203,304.50

Local council rates for Cloncurry Shire Council for all 28 granted ICM and Ivanhoe (Osborne) Mining Leases and two pastoral holdings leased by ICM and Ivanhoe (Osborne) will total approximately AUD 860,000 for the year beginning 1 July 2012. (Rates notices are usually issued for groups of contiguous Mining Leases and it is not possible to separate the rates for the four granted Mining Leases in Figure 4-3 from rates for adjacent Mining Leases.)

Any minerals mined under the Mineral Resources Act 1989 will require a royalty to be paid to the Crown at the rate set for each. Royalty payments are at a floating scale depending on the metal produced, determined by the floating quarterly metal prices.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Road access to the Osborne copper-gold project is via predominantly sealed roads to Mt Isa via Dajarra, or partly sealed roads to Mt Isa via Duchess. Copper concentrate is trucked in sealed containers from Osborne to the port of Townsville. There are several river and creek crossings through causeways which can become impassable for relatively short periods of time (days) during the wet season. This has little impact on shipping of concentrate as each ship load comprises the delivery of 4 to 6 weeks trucking.

Kulthor is accessed by decline from the existing workings.

5.2 Local Resources and Infrastructure

The nearest significant population to Osborne is Mt Isa with a population of around 22,000 people, approximately 195 km to the northwest. Osborne is a fly-in fly-out operation with its employment base at Townsville, population 185,000, located some 800 km east of Osborne. Ivanhoe has chartered flights between site and Townsville on most week days. Personnel are also flown to and from site from Brisbane. Supplies are transported by road haulage from Mt Isa. Services are sourced from Mt Isa or Townsville. The Osborne mine and mill are shown in Figure 5-1 to Figure 5-3.

Figure 5-1: Osborne mine site

Figure 5-2: Osborne open pit and mine site

Figure 5-3: Osborne mine infrastructure

5.3 Climate

The climate at Osborne copper-gold project is typical of the inland arid zones of sub-tropical North West Queensland. The majority of rainfall comes from summer thunderstorms and decaying depressions drifting down from the northern coast. The average annual rainfall is 320 mm. Temperatures range from extremely hot (average maximum 38°C) in summer to mild (average minimum 7.6°C) in winter.

The weather is amenable for year round mining operations.

5.4 Physiography

The Osborne mine is located on a large residual plateau (mesa) that is common in old weathered terrains in which ephemeral creeks incise the landform. The topography varies from around 280 m on the plateau to 220 m in the surrounding alluvial catchment areas in Figure 5-4. The Osborne copper-gold project workers accommodation is shown in the foreground.

Spinifex and Snappy Gum are the dominant species on the plateau, with shrubby open-hummock grassland or mixed scrubland on the flatter areas.

Figure 5-4: Typical landscape in the Project area

Note: Osborne camp is in middle distance

5.5 Surface Rights, Power, Water, Mining Personnel, Tailings and Waste Storage and Plant Site

The Osborne and Kulthor deposits are currently in production with processing being carried out on site. There is a sufficiency of surface rights for the plant and for tailings and waste disposal. Power is generated on site and water is sourced from the underground workings and a licensed bore field. Personnel are sourced from Mt Isa, Townsville and Brisbane for this fly-in / fly-out operation.

6 History

6.1 Historical Exploration Summary

6.1.1 Osborne

Early exploration by various companies (from the mid-1980s onward) targeted uranium initially, and then Pegmont-style lead-zinc deposits. In 1985, CSR Limited entered the area exploring for copper-gold hosted within ironstone similar to that at Starra 60 km north of Osborne. Initial exploration involved airborne and ground geophysics and limited drilling. By the end of 1987, a total of 74 reverse circulation (RC) holes and 23 diamond drillholes had been drilled into the Osborne magnetic anomaly without encountering any significant economic intersections. Elsewhere, exploration drilling had outlined 20 km of ironstone prospective for hosting copper-gold mineralisation.

In 1988, Placer Exploration Limited acquired CSR Limited and progressively worked towards earning full ownership of the project from various joint venture partners. The discovery hole containing a significant economic intersection of 32 m @ 5.8% copper and 3.2 g/t gold in TTHQ0029 was drilled in late 1989. Based on these results, Placer undertook an intensive drilling programme during 1990 to establish a Mineral Resource in early 1991 of 27 Mt @ 1.4% copper and 0.8 g/t gold. The mining of the Osborne deposit commenced in August 1995. Barrick (PD) Australia acquired the Osborne assets as part of a takeover of Placer Dome Asia Pacific Limited (PDAP) in early 2006. Production ceased in October 2010 with the sale of the Osborne assets (including Kulthor) to Ivanhoe Australia Limited.

6.1.2 Kulthor

The Kulthor prospect was originally detected by PDAP in 1994, as a low-level basement anomaly on the margin of an air core (AC) basement-sampling grid designed to test the western ironstones. In 1995 and 1996, 388 AC drillholes were completed to define the Kulthor basement copper geochemical anomaly within an area of non-magnetic basement. This anomaly was initially tested in June 1996 with 22 RC and diamond drillholes. These holes were inclined to the northeast, and intercepted extensive sub-economic oxidised mineralisation. Surface and downhole electromagnetic (EM) failed to identify a potential conductive source to the oxidised mineralisation. The prospect was interpreted to be the product of supergene enrichment of weakly mineralised basement metasediments.

In April 2001, following a programme of digital data capture and review, hole SUNQ023 was drilled to test for a sulphide source beneath the most intense section of the basement copper anomaly. This drillhole intercepted 16 m @ 0.96% Cu and 0.43 g/t Au in a easterly dipping structure. This intercept led to the testing of 800 m strike length of the anomaly on 200 m-spaced sections and resulted in the first significant intercepts (SUNQ0028 5 m @ 3% Cu, 5 g/t Au and SUNQ0031 9 m @ 5.57% Cu, 9.95 g/t Au) in June 2001. A total of 914 drillholes for 152,071 m have now been completed.

Barrick extended a decline towards Kulthor from the Osborne underground mine, which by 2010 was within 200 m of the Kulthor deposit. Fourteen underground holes were drilled into Kulthor before Barrick placed the Osborne copper-gold Project into care and maintenance in 2010. Ivanhoe completed the decline and recommenced surface and underground drilling in 2011. Some production has taken place (mainly development material).

6.2 Previous Mineral Reserve and Mineral Resource Estimates

The Osborne deposit has been the mainstay of Mineral Resources and Mineral Reserve statements until inclusion of the Kulthor Mineral Resource in 2003. The Osborne Mineral Resources and Mineral Reserves (NI 43-101 compliant) were previously reported in 2011 and 2012 (SRK Consulting 2011, 2012) and are unchanged in this report.

The Mineral Resources for the Kulthor project were previously estimated in compliance with NI 43-101 by LMRC in 2011 (SRK Consulting 2011, 2012). Additional drilling completed since these resources were estimated has allowed the geological interpretation and these Mineral Resources to be updated.

6.3 Osborne

Exploration during 1996-1997 saw the discovery of the 1SS orebody, which increased the reserve tonnes by 17%. Since the delineation of the 1SS orebody in 1997, underground production has steadily depleted the total reserves by 9-10% per year in Figure 6-1, until 2001 when total reserves increased by 21%, compared to 2000. The most significant increase was in Proven Mineral Reserves, by 36%, which reflected the increase in Measured Mineral Resource by 68%, compared to 2000 (Crimeen et al., 2009).

This trend has since changed with a consistent decrease in both Mineral Resources (15%) and reserves (8%) in 2002 and 2003. In 2004, another increase in both Mineral Resources (48%) and reserves (6%) was seen, with the introduction of satellite deposits such as the Inheritance open pit. A change of economic cut-off grade from 2% eCu to 1.5 % eCu for Mineral Resources and 1.73% eCu for reserves has contributed to this increase. Unfortunately, a proportion of this gain due to change in cut-off grade has been offset by the loss of unrecoverable pillars and ore lost to mining at the extremities of the orebody.

In 2005, a minor addition of 192,000 t was made to the Probable Mineral Reserves with the delineation of the 200 Block. Further increases in the copper price and subsequent lowering of the cut-off grade to 1.35% eCu for Mineral Reserves and 1.22% eCu for Mineral Resources led to increased Mineral Resource and Mineral Reserve tonnes at the end of 2006. From 2007 onwards, Mineral Rreserves have continued to decline due to the mining of the Mineral Resource, increased costs due to mining at increased depths and global economic conditions.

The Osborne 1S–1SS–1M underground Mineral Reserve decreased by 751 kt, due to depletion by mining. Lower blocks are impacted by the high stress regime and poor ground conditions. Consequently, larger pillars have been left to provide support, leading to further loss of Mineral Reserves, although as mining nears completion, recovery of pillar Mineral Resources has occurred. Mineral Resources in the 1 orebody have decreased, partly due to upper level blocks being left intact to provide support for the Kulthor access drive. Lower level blocks are impacted by a reduction in the orebody dimensions from infill drilling.

They are also depleted due to the high stress regime and poor ground conditions.

The Osborne 2S–2M underground Mineral Reserve increased by 11 kt with depletion by mining offset by the inclusion of remnant mining blocks not previously included in Mineral Reserves.

The Osborne Mineral Resource wireframes and ore block models in the 1SS 200-500 Blocks and the 1SS 010-100 Blocks have been remodelled.

The eCu formula has been adjusted to a new value in line with predicted metal prices, and varies according to metal prices used, and whether the ore is from Osborne or Trekelano.

An underground drill programme from the drill drive in the 290 zone was completed in the second half of 2008. This allowed for the updating of the Osborne 1SS ore block model down to 0 Level. Ore down to 110 Level was included in the reserves, and ore below 110 Level has been included in the Mineral Resources.

The Proven Mineral Reserve copper grade steadily decreased at approximately 10% per year from 1999 to 2003, while the Measured Mineral Resource copper grade remained fairly constant from 2000 to 2004 until the introduction of lower cut-off grades in Figure 6-3 and Figure 6-5. In drive delineation, diamond drilling along the 480 and 455 levels in 2005 significantly increased the copper grade of the 1SS orebody in both Mineral Reserves and the Mineral Resource. In 2006, lowering of cut-off grade has led to decreased Mineral Resource and Mineral Reserve copper grades. Grades for 2008 are similar to those quoted in 2006 and 2007 in Figure 6-2 and Figure 6-5.

Up to 2001, the average Mineral Reserve gold grade has decreased by approximately 3% per year since 1997 in Figure 6-3 and Figure 6-6. Both Probable Mineral Reserve and Indicated Mineral Resource gold grades indicated relatively no change between 2002 and 2001. The 20% decrease in the Indicated Mineral Resource grade in 2003 was offset by the 7.5% increase in the Measured Mineral Resource due to the tonnage difference between the two categories in Figure 6-4.

In 2004, the grade for both Mineral Reserve categories decreased by about 18% in response to the decrease in the cut-off grade to 1.73% eCu. The Mineral Resource models have indicated a similar decrease of 11% in both categories since 2003. Mineral Resource models for 2004 were based on a combination of 1.5% eCu and 2% eCu cut-offs, depending on how committed the area was to the mining phase.

In response to the significant increase in the copper grade in 2005 with the delineation of 480 and 455 levels, the contained gold has also increased in both the Mineral Resource and the Mineral Reserve. In 2006, lowering of cut-off grade led to decreased Mineral Resource and Mineral Reserve gold grades. While grades for 2008 are slightly higher than 2007 due to higher cut-offs required at the deeper mining levels, grades in 2009 are slightly lower again, due to operations maximising extraction from remaining stopes.

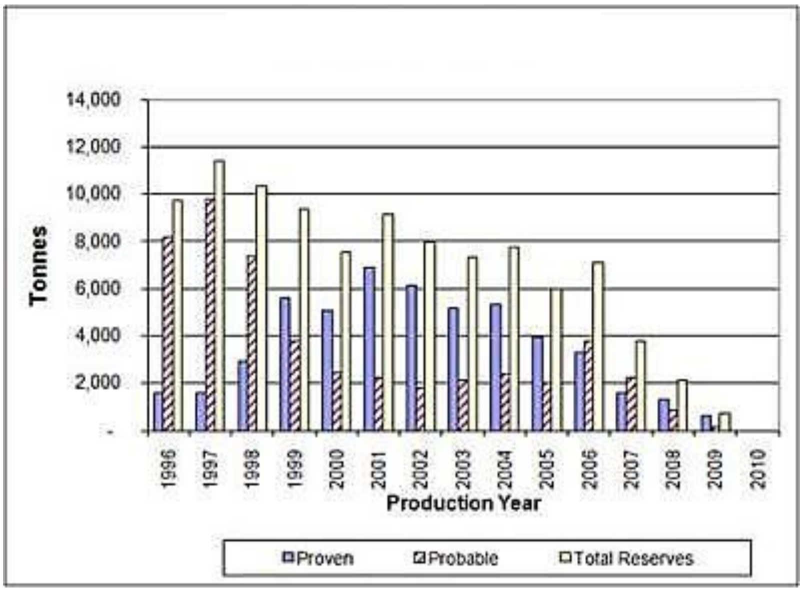


Figure 6-1: Yearly comparison of Mineral Reserves

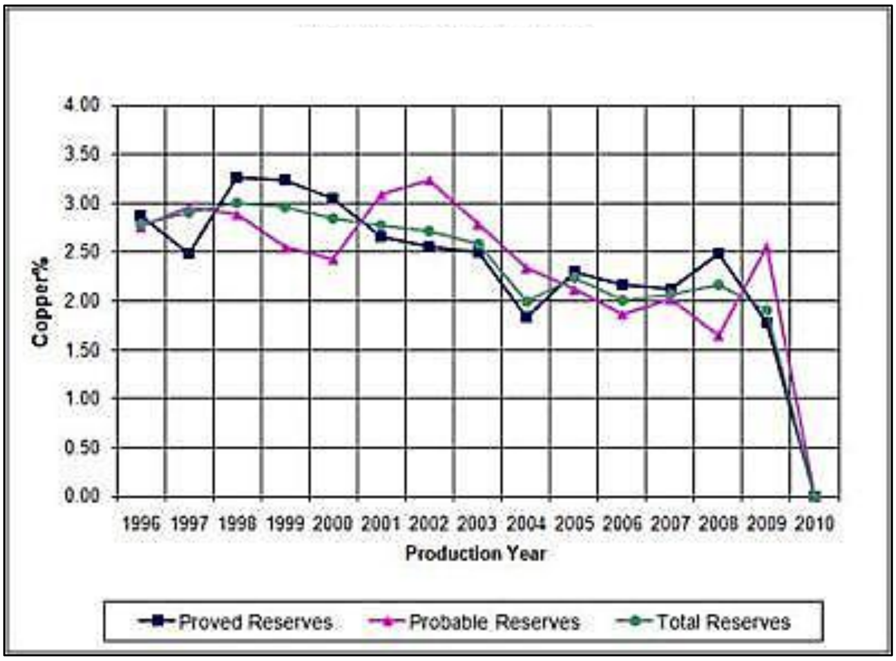


Figure 6-2: Comparison of Copper grade in Mineral Reserves

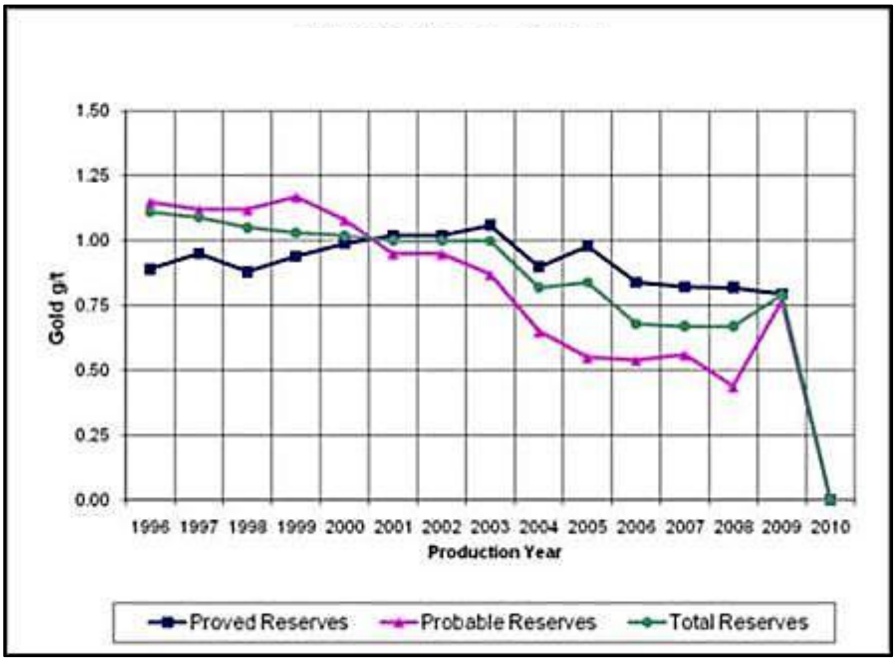


Figure 6-3: Comparison of Gold grade in Mineral Reserves

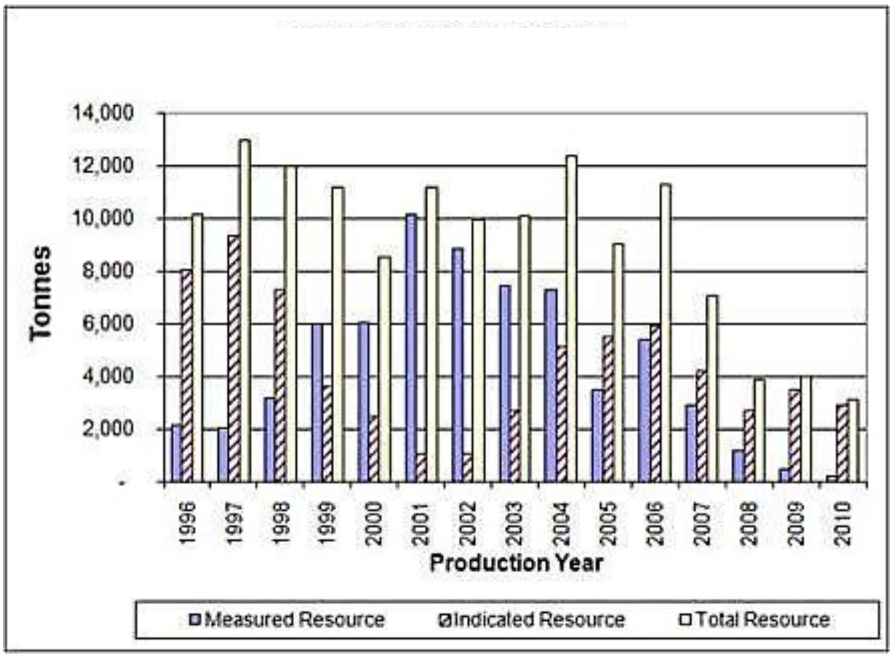


Figure 6-4: Yearly comparison of Mineral Resources

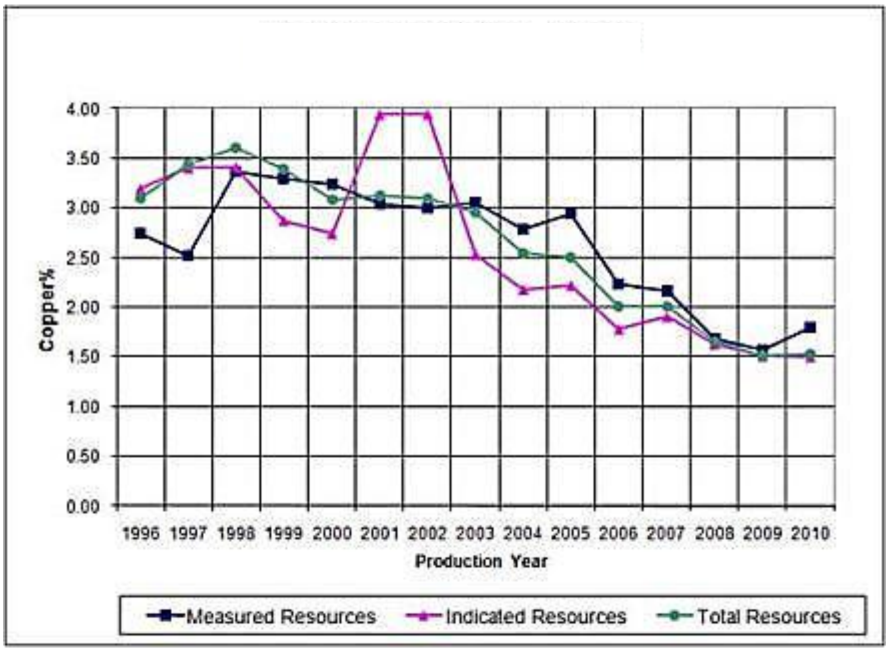


Figure 6-5: Comparison of Copper grade in Mineral Resources

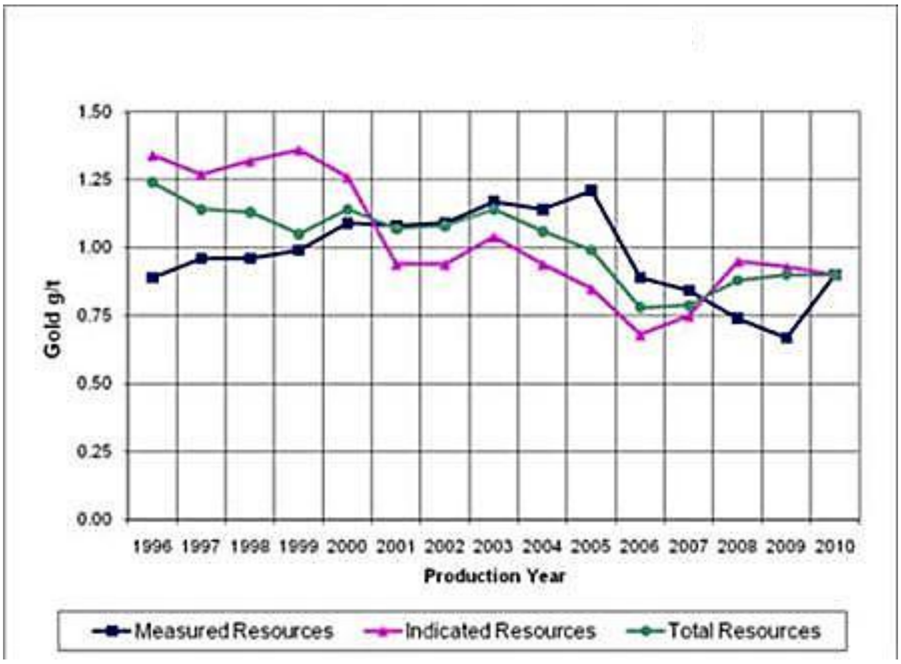


Figure 6-6: Comparison of Gold grade in Mineral Resources

6.4 Kulthor

Ivanhoe have completed the decline connecting the Kulthor deposit to the Osborne mine, allowing underground development and production of the Kulthor deposit to commence. At the Effective Date (5 September 2012) only development mineralisation has been mined.

6.5 Production from Property

Yearly production from Osborne has been consistent at around 1.5 million tonnes per annum (Mtpa) until 2005 when production targets were increased to offset falling grades in Figure 6-7. The introduction of ore from Inheritance (Trekelano), which ceased in 2009, helped to achieve 2.0 Mt plant throughput in 2007, 2008 and 2009. Production ceased in September 2010 with the sale of the Osborne copper-gold project assets to Ivanhoe. Production recommenced in March 2012.

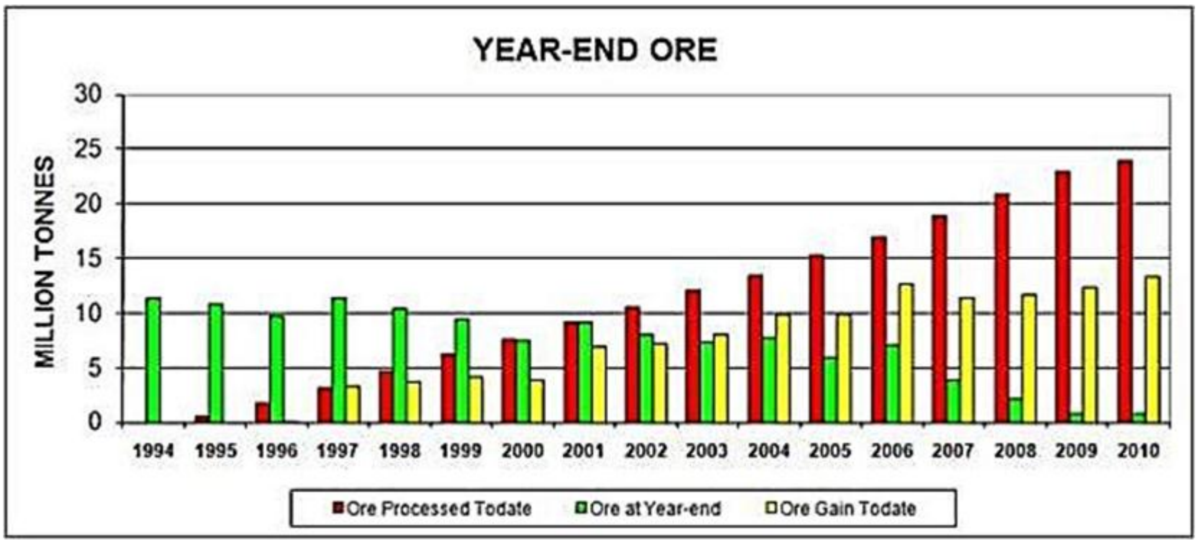


Figure 6-7: Year-End Ore

7 Geological Setting and Mineralisation

7.1 Regional Geology

All Osborne copper-gold project tenements are located in the Proterozoic Mt Isa Block (Figure 7-1). The Mt Isa Inlier is a multiple deformed and metamorphosed terrain subdivided into three blocks – the Western Fold Belt, the Kalkadoon-Leichhardt Block and the Eastern Fold Belt, which are bounded by major north-striking fault zones. The Eastern Fold Belt of the Mt Isa Inlier comprises an Archean-Proterozoic basement of metamorphic rocks variably overlain by three cover sequences of sediments and volcanics which range in age from 1850 Ma to 1670 Ma.

Two major tectonic events have been identified in the inlier – the Barramundi Orogeny and the Isan Orogeny. The second of two major tectonostratigraphic cycles occurred between these two events and is represented by three cover sequences.

The Osborne district is at the southern end of the Cloncurry-Selwyn Zone of the Eastern Fold Belt. The stratigraphic units are part of the Soldiers Cap Group, which is part of regional Cover Sequence three (3). The depositional age for the Soldiers Cap Group is 1712-1654 Ma. The mapped units are psammite-quartzite-ironstone-amphibolite of the Mt Norna Quartzite in the north and west and psammite-pelite of the Llewellyn Creek Formation in the southeast. The metamorphic grade is described as sillimanite zone of the amphibolite facies for the Mt Norna Quartzite and sillimanite-K feldspar zone for the Llewellyn Creek Formation. The maps of the region show that Osborne lies in the eastern limb and that Kulthor lies in the western limb of a fold nose that is within the Mt Norna Quartzite, but close to a core of Llewellyn Creek Formation.

The Isan Orogeny occurred between 1590 and 1500 Ma and is divided into four deformation events and two metamorphic events. The peak of metamorphism, a low pressure-high temperature type coincided with D_2 deformation. The present structural pattern is evident in steeply dipping, north-trending folds and faults developed during the Isan Orogeny. There has also been extensive mafic to felsic granitoid emplacement during this time. These granites are believed to be temporally and spatially associated with the IOCG deposits in the district.

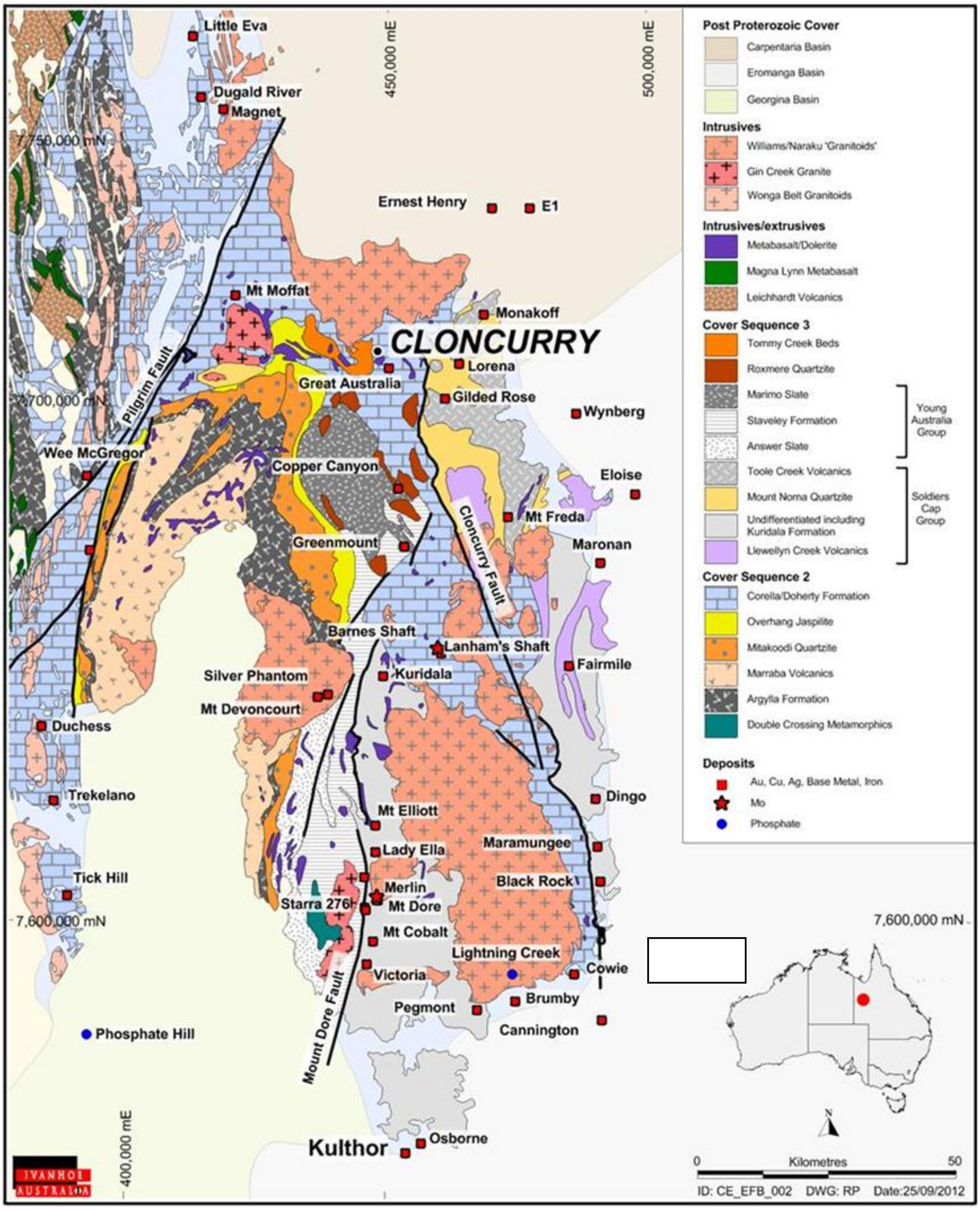


Figure 7-1: Geology of the Mt Isa Inlier

7.2 Local Geology

The Osborne and Kulthor copper-gold deposits lie within Palaeoproterozoic metasediments assigned to the Mt Norna Quartzite of the Soldiers Cap Group in the Eastern Fold Belt of the Mt Isa Inlier (Figure 7-2, Crimeen, et al., 2009).

The host sequence of sandstone, siltstone and ironstone is cut by dolerite dykes and has undergone partial melting to produce granofels, migmatites and gneiss. Pegmatite dykes and related alteration and mineralisation are concentrated in a pod of lower grade metamorphic rocks surrounded by partial melt rocks. At least four phases of deformation are recognised in the Osborne region with the second thrusting event producing the dominant foliation (Banvill, 1998). The subsequent strike slip of the third event is believed to have produced dilations that now host the Osborne deposits (King, 2001).

The Kulthor veins and shears are contained within a shear-bound block of altered psammite and amphibolite that is up to 150 m wide and at least 900 m long. The zone boundaries and the internal fabrics are all steeply dipping and have a NE strike. The zone, as well as the veins and mineralisation, is open to the SW and NE and seems to change to a more northerly orientation at its northern end.

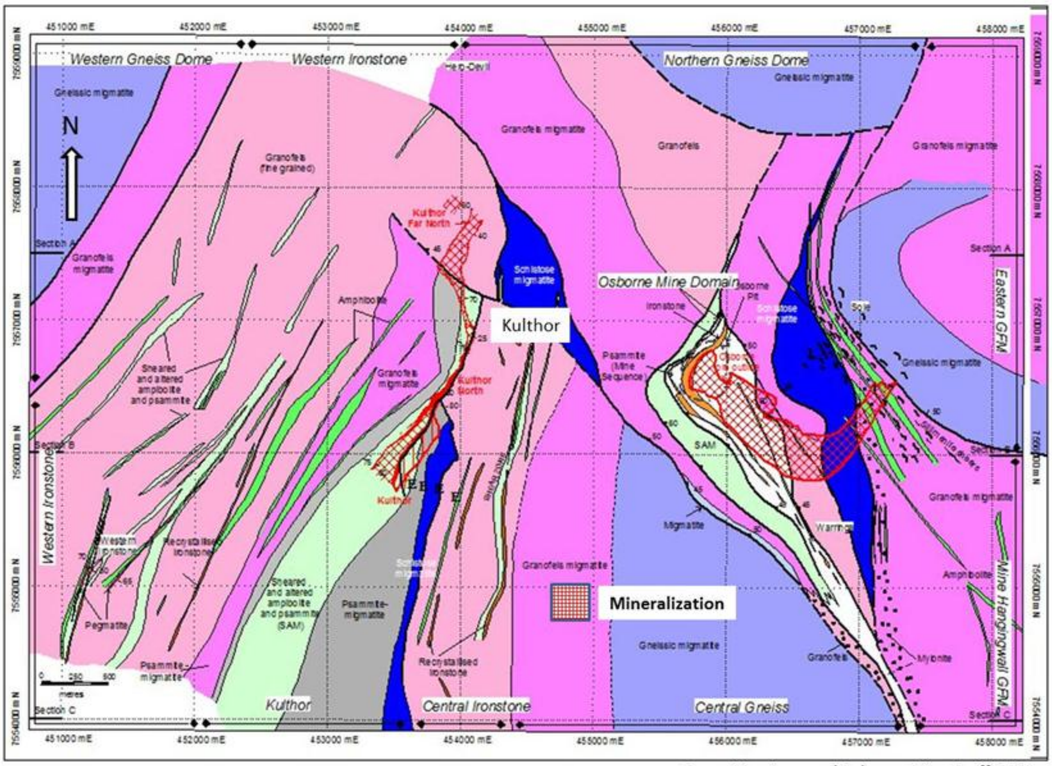


Figure 7-2: District Geology of Osborne and Kulthor

7.3 Property Geology

7.3.1 Osborne

According to Voulgaris and Tullemans (1998), Osborne can be divided into two mineralised domains, the Eastern and Western, based on host lithology. In the Western Domain, the economic mineralisation is hosted in the Upper and Lower Ironstones in Figure 7-3, which have altered to grey, massive, coarse-grained quartz, colloquially known as 'silica flooding'. The silica flooding is interpreted as alteration related to pegmatites that have intruded in dilational sites in the pod. The Eastern high-grade lode occurs in a dilational biotite shear, while two sub-parallel ironstone units contain mineralisation in the Western Domain. The two ironstones are separated by some 5 m to 40 m of quartz-albite and calc-silicate altered sandstone rock. The ironstones are composed of magnetite and quartz with minor apatite and specular haematite.

The majority of the structural deformation recorded in the Osborne Mine can be attributed to the D_2 deformation event. This is a prolonged event involving the rotation of stresses leading to the formation of several shear sets. According to King (2001), the D_2 event is divided into two main stages. The first involves the east south-east to west-northwest compression of the entire sequence, creating north to south trending shears. The ironstone units of Osborne occur in a low strain domain within this stress regime, and are surrounded by high strain zones where mylonitic fabrics have developed. Shears related to this initial compression are generally north to south trending, and occur along unit and orebody (1S and 2S) boundaries. Rotation of the compressional stresses to southeast to northwest resulted in units within the low strain domain displaying 'pinch and swell' geometries, with dextral shear on northern structures and sinistral shear on southern structures. The plan of the 755 Level illustrates the 1S orebody with sheared N/S-trending boundaries which are formed by the initial compression of D_2 in Figure 7-4. The 1S in Figure 7-5 does not display the 'pinch and swell' textures typical of the 2S orebody, as the 1S is located on the major boundary of the low and high strain domains. Instead, the 1S remains north to south striking, with sinistral shear. The 2S occurs in the low strain domain and is thus affected by shortening resulting from the rotation of the stress regime. The original north to south trending 2S shears are 'pinched' towards the 1S orebody, thus becoming northeast to southwest trending structures with dextral shear.

The second stage of D_2 involves further rotation of the compressional stresses. The result is a strong sinistral strike slip motion developed along north to south trending structures. This movement develops anastomosing east-northeast to west-southwest shears splaying from the 1S and linking to the 2S Footwall shears, thus reactivating the 'pinched' 2S shears. The shear sense along the 2S shears is now sinistral, causing localised extension which is related to silica flooding controlled by 2S-3E dilational shears.

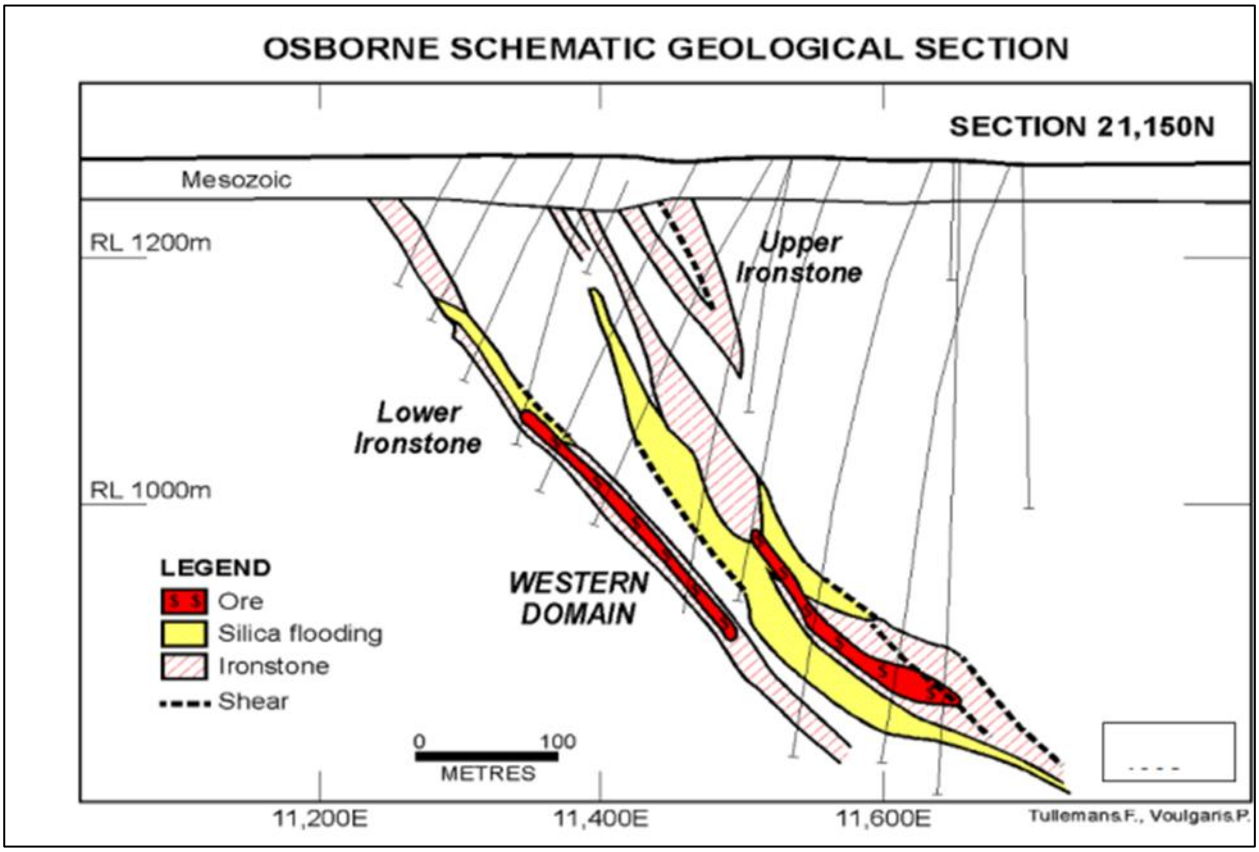


Figure 7-3: Upper and Lower Ironstones

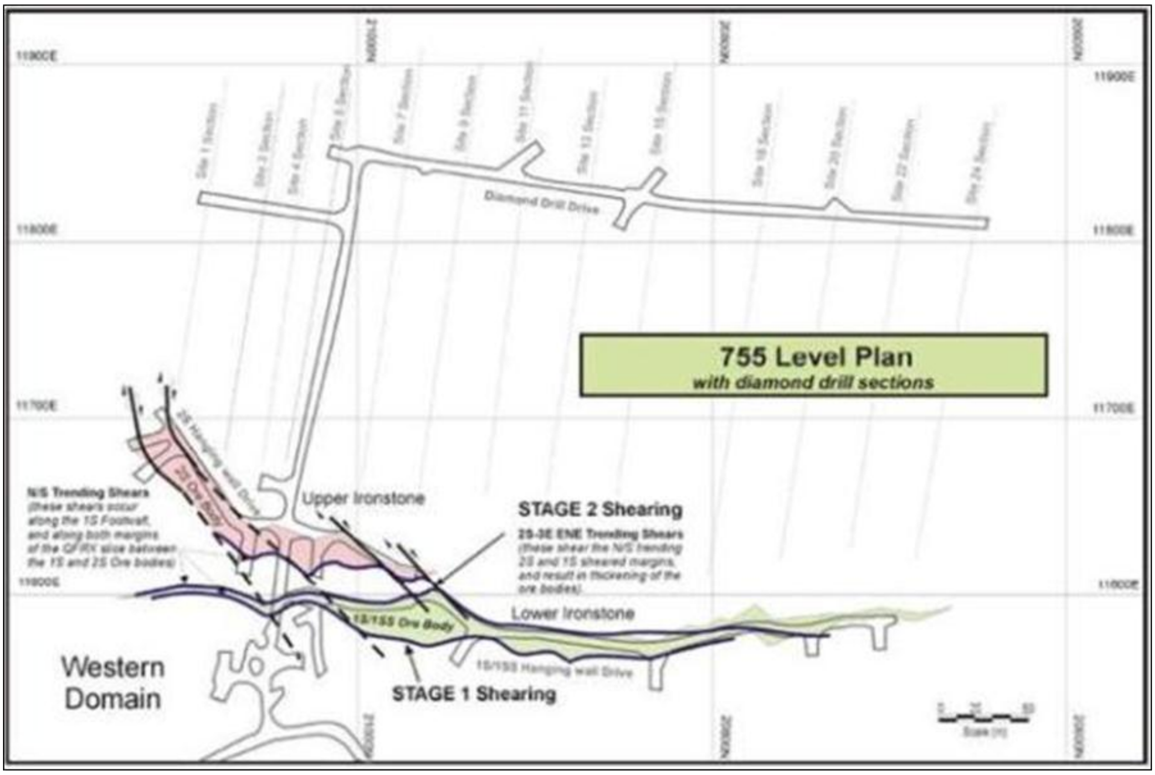


Figure 7-4: 755 Level Plan

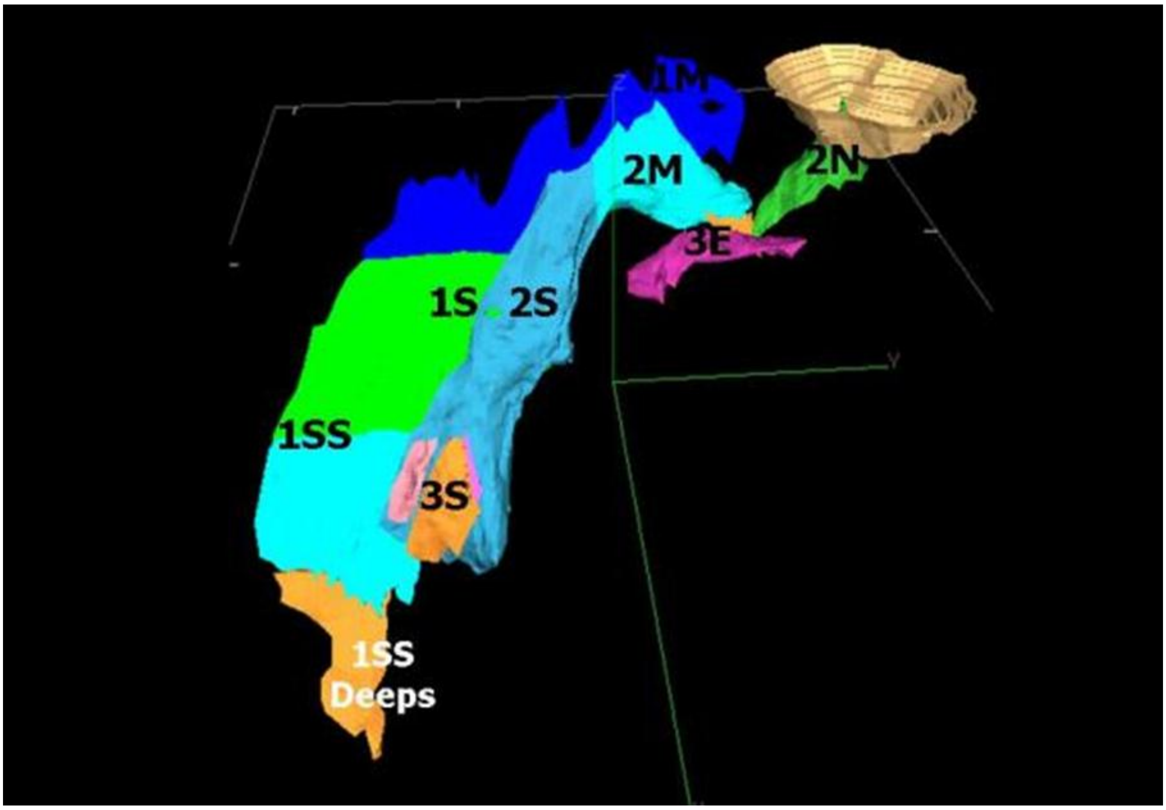


Figure 7-5: Osborne Orebodies

7.3.2 Kulthor

The mineralisation at Kulthor consists of shear and replacement sulphide lodes that overprint a series of mineralogically zoned pegmatitic veins. Hinman (2012) divided the Kulthor into three distinct packages (Figure 7-3). The hangingwall (HW) package consists of high-grade metamorphic psammitic and pelitic metasediments showing significant partial melting. There are concordant and discordant pegmatites. The footwall (FW) package consists of lower metamorphic grade amphibolites and psammitic metasediments. There are no pegmatites. The central package consists of low metamorphic grade amphibolite and siliceous, sulphidic fine sediments with minor psammite. The bulk of the better grade mineralisation is hosted by dolomite-quartz pegmatite. The Kulthor mineralisation has a strike of at least 2,300 m and a known vertical extent of approximately 750 m.

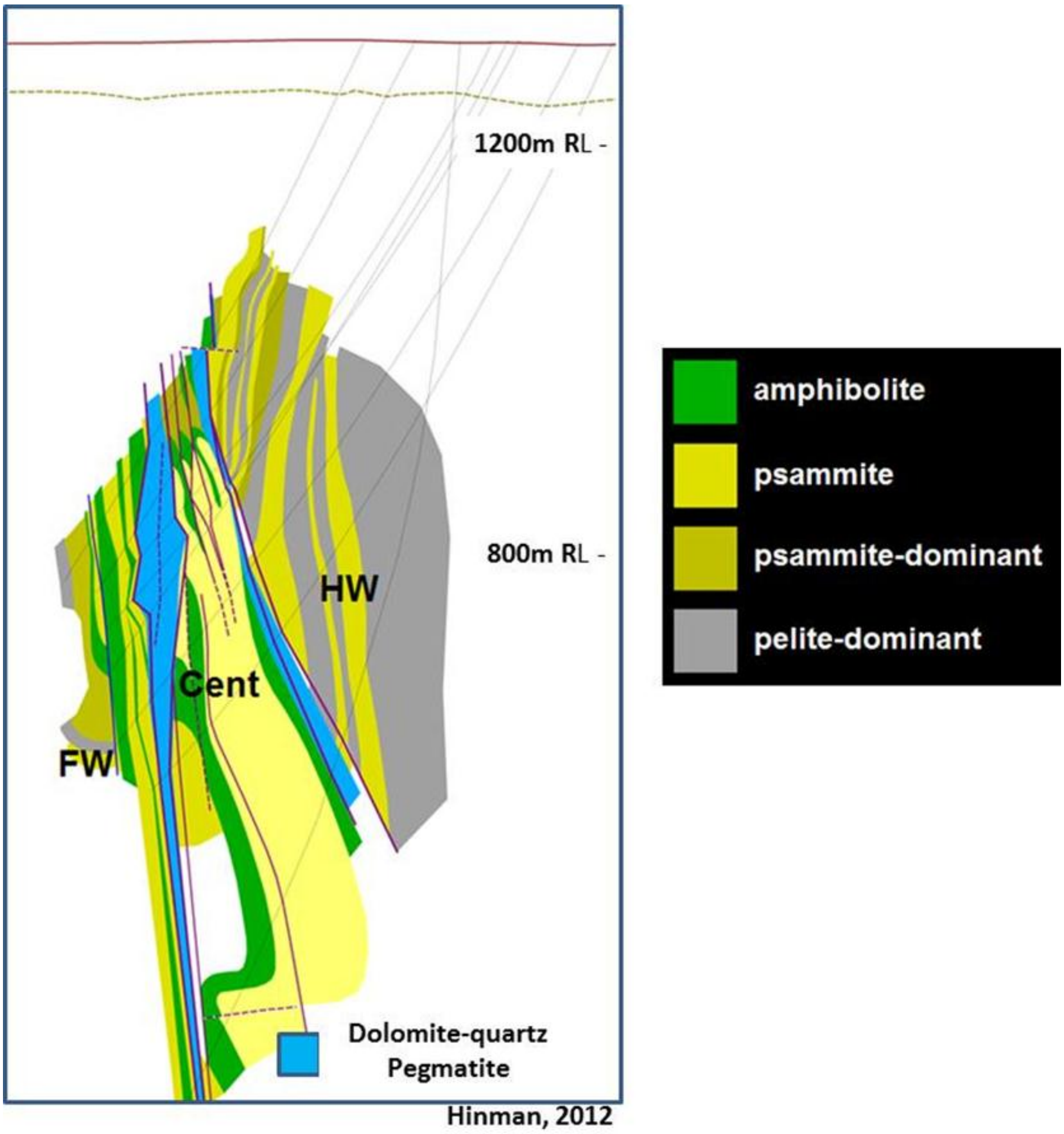


Figure 7-6: Kulthor cross section (looking northeast)

8 Deposit Types

Osborne, Kulthor, and most of Ivanhoe's exploration targets are characterised as belonging to the IOCG class of deposits. IOCG deposits are characterised by abundant iron oxides, both magnetite and haematite, and association of the characteristic copper and gold and enrichment with substantial hydrothermal alteration surrounding the orebodies. The amount of magnetite at Kulthor is less than at Osborne.

The following geological models and concepts are being applied by Ivanhoe in its investigation of, and form the basis for, Ivanhoe's exploration, Mineral Resource estimation and reserve estimation.

8.1 Mineralisation and Occurrence

8.1.1 Osborne

Mineralisation in the Western Domain occurs in the 2M, 2S, 1S and 1SS orebodies. The 2S orebody consists of a folded tabular lens, 10 to 20 m thick, which plunges 50 – 60° to grid southeast and is located on the silica flooded margins of the Upper Ironstone. The 1S and its southerly extension, the 1SS orebody, are located in the Lower Ironstone and vary from 5 to 20 m in true thickness. There is some slight indication of a plunge to the southeast at 50°. The mineralisation is semi-continuous for over 2 km down-dip and is still continuing (at reduced grade) at the lowest level drilled. The maximum strike length is 1100 m and the maximum thickness is approximately 30 m.

8.1.2 Kulthor

The mineralisation is in the form of veins and minor replacement lodes exploiting brittle fracture and breccia zones developed along the margins and within the dolomite-quartz pegmatite. Sparse mineralisation outside the lodes is localised in shears and is almost exclusively in the form of centimetre-scale dolomite veins with local quartz, pyrite, chalcopyrite, chlorite, and calcite overprint. The mineralisation occurs in three main lenses which have a horizontal extent of 2,300 m and a vertical extent of at least 700 m. The width of the mineralisation is variable (maximum thickness is approximately 40 m but usually less than 16 m).

The majority of the material to be mined at Kulthor will be sourced from the Western Lode (the "M" zone). This is an 85° east-southeast dipping sulphide-rich shear-hosted lode that extends throughout the central sections and beyond for at least 500 m. The lode is dominated by stylo-shears with sulphide and stylo-breccias with near massive sulphide that seem to replace fragments of dolomite vein caught up in the structure. High grade intercepts are chalcopyrite blebs and stylo-networks that replace dolomite that has not already been replaced by other sulphides. The intercepts of mineralisation and the higher-grade shoots are localised at the intersections of cross vein sets.

8.2 Precious Metals

In all deposits, primary copper mineralisation occurs solely as chalcopyrite, with gold occurring as an accessory phase in pyrite and chalcopyrite.

8.3 Oxide and Sulphide Mineralisation

The bulk of the Osborne and Kulthor mineralisation is fresh sulphide. Most of the Osborne oxidised mineralisation was previously mined in the open pit, though some still remains. A small amount of the Kulthor mineralisation is oxidised.

9 Exploration

9.1 Introduction

Only limited exploration (other than drilling) has been carried out on the Osborne and Kulthor deposits since they were purchased from Barrick in 2010. Geophysical surveys have been carried out in the Osborne-Kulthor and Houdini areas.

The Osborne and Kulthor deposits are completely covered by later barren Mesozoic sediment, preventing use of outcrop/trench sampling.

9.2 Geophysics

9.2.1 Osborne / Kulthor

The Osborne and Kulthor systems have recently had an initial geophysical evaluation completed by Ivanhoe's newly established Geophysics group. Following the Ivanhoe Geological Conference held at Osborne in September 2010, geophysical data inherited from previous owners of Osborne was used to begin assessing the potential for ore extensions below the current mining levels at 1400 m.

At Osborne, the geophysical survey has indicated extension to the known mineralisation by as much as 1,000 m. EM conductor orientations showed substantial conductors that are likely caused by sulphides. Drilling to define the depth extension will be undertaken following further geophysical work defining target areas. The maximum depth extent is expected to be determined when Ivanhoe's 2009 high-resolution aeromagnetic survey is inverted.

The host southwest-striking fault zone at Kulthor dips to the east and there is potential that this zone could continue for a further 2,000 m to the south before being terminated by the same structure that terminates the Osborne mineralisation. This indicates that there is potentially a magnetic structure linking the Osborne and Kulthor environments. A 1,000 m drillhole is proposed from the Kulthor decline access to test this hypothesis.

One of the most striking new features is a newly identified 5 km-long magnetic structure striking parallel, and to the west of the existing Kulthor mineralisation. This target, known as Avalon, in which previous drilling intersected mineralisation at the very top of the system extends to approximately 1,500 m depth, and will be drilled after further geophysical evaluation.

9.2.2 Houdini

A sub-audio magnetics survey was completed to identify conductivity anomalies associated with the magnetic anomalies. This survey showed a linear conductivity anomaly associated with the linear copper anomaly which was named the Houdini anomaly.

9.3 Geochemistry

No AC or surface based geochemical surveys were completed in the Osborne copper-gold project area since the purchase of the tenements from Barrick in 2010.

10 Drilling

The drilling used for Mineral Resource estimation for the Osborne mine was carried out by CSR Minerals, Placer Dome Asia Pacific Limited and Barrick Gold Corporation between 1987 and 2009.

A total of 116 surface and underground diamond drillholes have been completed by Ivanhoe at Kulthor since the 2011 Mineral Resource estimation. All samples through the mineralisation were diamond core.

The only drilling in the Osborne area away from the Osborne Deposits was at the Houdini Project, 20 km to the northwest. Exploration successes in the 2009 drilling campaign indicated the potential for further Mineral Resources at Houdini. Following a sub-audio magnetic survey, a 500 m long near-surface mineralised structure was identified following a series of significant high-grade copper-gold intercepts in this zone. Follow-up diamond drilling in late 2010 returned economic grade and width intercepts on a single section down to around 300 m from surface. Mineralisation at Houdini is structurally controlled. Chalcocite exists closer to the surface and within the main structure, with chalcopyrite at depth (100 m) and in the footwall of the Houdini shear.

10.1 Drilling Methods

Ivanhoe uses a variety of drilling methods. Underground drilling is exclusively by diamond drill. Surface drilling relies on establishing a pre-collar by RC drilling followed by a diamond drill tail through the zone of interest.

10.2 Logging Procedures

Geological data is logged directly into the acQuire database via entry screens on portable computers. Geological staff are provided with procedures and training in order to provide accurate and consistent results.

10.3 Surface Outcrop/Trench Sampling

Surface outcrop or trench sampling was not carried out in relation to Mineral Resource delineation for any of the Osborne-Kulthor deposits.

10.4 Underground Channel Sampling

Channel sampling is conducted underground at the Osborne mine. Continuous face samples are collected at regular intervals along the drive, depending on the production cycle. These samples are collected by chipping across the face in a continuous line with the jumbo collecting the 1 m sample (5 to 10 kg) on a mat. This procedure is then represented as a small drillhole, even though it has a different support to the in-drive diamond drillholes. In the Mineral Resource estimation process, these samples are given equal weight to the diamond samples. While this is not strictly correct, it is done in order to obtain sufficient data for modelling.

The direction of the in-drive drillholes and the channel samples conflicts with other indicated drilling between and around these levels. This severe directional change affects the variograms, which has some effect on the ranges and Kriging process. The in-drive drillholes and channel samples were used to guide past production, and have no direct bearing on the new Mineral Resource estimates.

10.5 RC Drill Sampling

10.5.1 Osborne / Kulthor

RC drilling has been used exclusively for establishing pre-collars for deeper diamond drillholes. As such, almost no RC samples are likely to have been used in the determination of Mineral Resources, except in the open pit area where some holes RC holes drilled pre-mine were available.

10.6 Diamond Drill Core Sampling

10.6.1 Osborne

The diamond core drilling used for Osborne deposit resource estimation was carried out by Barrick and proceeding companies (mainly PDAP). Initially 2 m samples were cut but after production commenced, all sampling was on 1m intervals. Whole core was used for infill drillholes, in order to increase the sample size. Details of the drilling used for Mineral Resource estimation are provided in Section 14.

10.6.2 Kulthor

The first 13 underground diamond core cores into Kulthor were drilled by Barrick in 2009. Ivanhoe recommenced underground and surface diamond drilling in 2011.

An underground NQ2 diamond drillhole programme is continuing at Kulthor providing the detailed information needed for final stope design. There were 84 underground drillholes used for Mineral Resource estimation. This is supplemented by a surface diamond drilling program designed to further explore the Kulthor mineralisation along strike and at depth. Samples are taken at 1 m and 2 m intervals using a core saw. Further details are provided in Section 14.

10.7 Drilling Pattern and Density

The Osborne open pit Mineral Resources are based on close-spaced surface and underground drilling. A small proportion of the drilling was by RC (pre-mine).

The Osborne 1SS Mineral Resources are mainly based on underground drilling. Channel sampling and short diamond holes in the mined-out area up-dip have some relevance. The drillhole spacing for 1SS is variable, but is 20 to 30 m in the better-drilled areas, increasing with depth.

The drilling used for estimation of the Osborne open pit and 1SS underground Mineral Resources is shown in Figure 10-1. None of this drilling was carried out by Ivanhoe. The Kulthor drilling shown in this figure was pre-Ivanhoe.

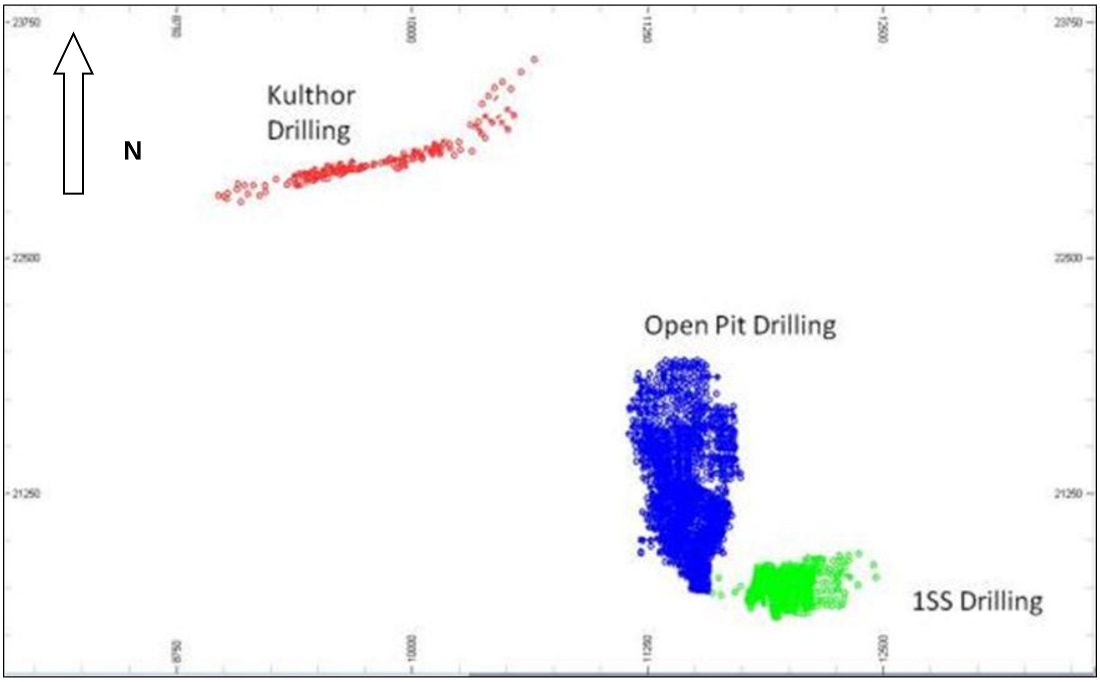


Figure 10-1: Drilling used for Mineral Resource Estimation

Source: LMRC, July 2011

The drilling at Kulthor has been limited firstly by the considerable depth of the deposit below the surface, and secondly, by the lack of underground access for drilling until recently. The drillhole spacing at Kulthor varies from 30 to 200 m. The Kulthor drillholes are shown in oblique view, looking north in Figure 10-2. The Ivanhoe drilling is distinguished from that performed by previous owners.

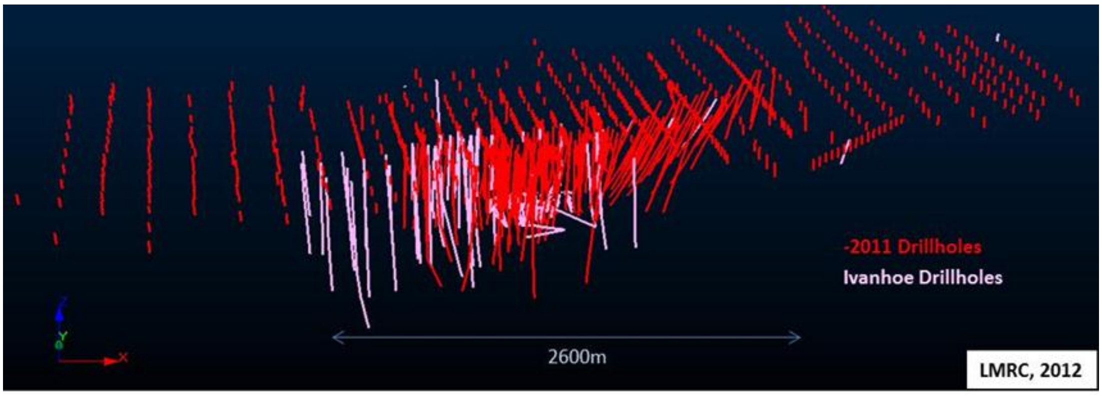


Figure 10-2: Kulthor Drillholes (oblique view looking north)

The only drilling by Ivanhoe in the Osborne area away from the Osborne Deposits was at the Houdini Project, 20 km to the NW. Exploration successes in the 2009 drilling campaign indicated the potential for further Mineral Resources at Houdini. Following a sub-audio magnetic survey, a 500 m-long near-surface mineralised structure was identified following a series of significant high-grade copper-gold intercepts in this zone. Follow-up diamond drilling in late 2010 returned economic grade and width intercepts on a single section down to around 300 m from surface. Mineralisation at Houdini is structurally controlled. Chalcocite exists closer to the surface and within the main structure, with chalcopyrite at depth (100 m) and in the footwall of the Houdini shear.

10.8 Interpretation of Drilling Results

Drilling at Osborne and Kulthor has taken place from 1998 to 2012. As this drilling is used for Mineral Resource estimation (discussed later in this report), no specific discussion of the numerous individual holes is included here. See Section 14 for further information. The quality of the drilling, surveying, sample recovery, sample preparation and assaying is good, and the amount of drilling is adequate for reliable Mineral Resource estimation.

11 Sample Preparation, Analyses, and Security

11.1 Sample Preparation and Analyses

Most of the sample preparation and analysis for the drillhole data used for Mineral Resource estimation was done at independent laboratories. Due to production pressures, a limited amount was done at the Osborne mine laboratory in 2009. The basic methodology has remained unchanged for many years. The basic methodology has remained unchanged for many years under Placer / Barrick. Modifications were made when ownership changed to Ivanhoe. Only the Kulthor deposit was estimated using Ivanhoe samples in addition to pre-Ivanhoe data.

11.2 Sample Preparation

11.2.1 Historical (Pre-Ivanhoe)

Diamond drill core samples from surface and underground drilling at Osborne and Kulthor were sent to an external laboratory. ALS Chemex Townsville processed the samples and conducted the gold analyses while ALS Chemex Brisbane determined the base metal assays. Some samples were processed at the ALS Chemex Mt Isa Laboratory in 2008.

The RC holes drilled pre-mine were sampled using a riffle splitter. In the Kulthor area they were used only for pre-collar drilling. They form a very small part of the total drilling database.

Drill core samples are processed and bagged by Osborne field technicians and transported to the laboratory by road haulage.

Standard sample preparation at ALS Chemex involves the following steps:

- Whole sample dried at 110° Celsius for 12 hours;
- Whole sample crushed to 70% passing 6 mm through jaw crusher;
- Whole sample crushed to 70% passing 2 mm through rotary crusher;
- Sample passed through rotary splitter to achieve a sample size of less than 3 kg;
- Coarse residue and pulp reject stored until advised; and
- Split sample pulverised to 85% passing -75 micron through LM5 pulveriser.

11.2.2 Current (Ivanhoe)

Diamond drill core samples from surface and underground drilling at Kulthor are sent to an external laboratory. ALS Chemex Mt Isa processes the samples and determines the multi-element analyses while ALS Chemex Townsville determines the gold assays.

The RC holes drilled pre-mine were sampled using a riffle splitter. No RC samples have been submitted for assaying by Ivanhoe's Resource Geology department. Limited sampling of RC pre-collars by the Ivanhoe Exploration department did not intersect mineralisation.

Drill core samples are processed and bagged by Ivanhoe field technicians and transported to the laboratory by road haulage.

Standard sample preparation at ALS Chemex involves the following steps:

- Whole sample dried at 100° Celsius for 12 hours;
- Whole sample crushed to 90% passing 9 mm through jaw crusher;
- Whole sample crushed to 90% passing 2 mm through rotary crusher;
- Sample passed through rotary splitter to achieve a sample size of less than 1 kg;

- Coarse residue and pulp reject returned to site; and
- Split sample pulverised to 90% passing -75 micron through an LM2 pulveriser.

11.3 Sample Analysis

11.3.1 Historical (Pre-Ivanhoe)

Almost all of the assaying of the underground and exploration core has been performed off site by ALS Chemex in Townsville and Brisbane. A limited number of samples were assayed at the ALS laboratory in Mt Isa in 2008. All ALS Chemex laboratories operate in compliance with ISO17025.

Gold assays were analysed by fire assay and atomic absorption spectroscopy (AAS) on 30 g charges (method Au-AA25). Screen fire assays (method Au-SCR22AA) to determine the influence of the coarse gold particles were used in samples containing more than 1 g/t gold for exploration and 3.5 g/t gold for underground.

Drill core samples were treated by four acid digest (method ME-ICP41s). Copper was analysed by ICP-atomic emission spectroscopy (ICP-AES). All copper assays registering over 10,000 ppm were re-assayed by the ore grade technique which involves an aqua regia digest with ICP-AES analysis optimised for high grades (method CU-OG46).

11.3.2 Current (Ivanhoe)

All of the assaying of the underground and surface core has been performed off site by ALS Chemex in Mt Isa, Townsville and Brisbane. All ALS Chemex laboratories operate in compliance with ISO17025.

Gold assays were analysed by fire assay and AAS on 30 g charges (method Au-AA25). Screen fire assays (method Au-SCR22AA) to determine the influence of the coarse gold particles were used in samples containing more than 2 g/t gold.

Drill core samples were treated by four acid digest (method ME-ICP41s). Copper was analysed by ICP-atomic emission spectroscopy (ICP-AES). All copper assays registering over 3,000 ppm were re-assayed by the ore grade technique which involves an aqua regia digest with ICP-AES analysis optimised for high grades (method CU-OG46).

11.4 Sample Security

11.4.1 Historical (Pre-Ivanhoe)

Samples were transported to the laboratory by a commercial haulage company. The samples were bagged individually and then grouped into poly-woven bags with a Ziploc closing the mouth of the closed bag. The poly-woven bags were then placed in a steel cage for transport. Shipping instructions and sample submission forms were attached to the top poly-woven bag and issued electronically to the warehouse, the laboratory manager, the database manager and senior geology staff. Samples were collected in the cage and taken to the laboratory by the same carrier. A sample receipt advice was issued by the laboratory when processing of the samples commenced.

With secure access to the ALS Chemex website, it was possible to monitor the progress of sample batches through the laboratory. When assay results were returned, the laboratory electronically issued a signed certificate, a data file and a quality control report.

The laboratory received basic sample information only – sample numbers and the overall project: e.g. Kulthor.

11.4.2 Current (Ivanhoe)

Samples are transported to the laboratory by a commercial haulage company. The samples are bagged individually and then grouped into poly-woven bags with a Ziploc closing the mouth of the closed bag. The poly-woven bags are then placed in a Bulka bag for transport. Shipping instructions and sample submission forms are attached to the Bulka bag and issued electronically to the warehouse, the laboratory manager, the database manager and senior geology staff. Samples are taken to the laboratory by the same carrier. A sample receipt advice is issued by the laboratory when processing of the samples commenced.

With secure access to the ALS Chemex website, it is possible to monitor the progress of sample batches through the laboratory. When assay results are returned, the laboratory electronically issues a signed certificate, a data file and a quality control report.

The laboratory receives basic sample information only – sample numbers and the overall project: e.g. Kulthor.

11.5 Bulk Density Data

Due to the high magnetite content of the mineralisation, density is very important at Osborne. There are many density data stored along with metal grades in the acQuire database.

Bulk densities are determined by weighing a representative 100-200 mm sample of diamond core in air, followed by weighing the sample totally submerged in water. It was assumed that the sample was impermeable where:

$$\gamma = \text{wt. in air}/(\text{wt. in air} - \text{wt. in water})$$

Bulk density measurements made pre-mine used all the core pieces from a 1 m or 2 m run for a single measurement. This was reduced to one selected piece after mine production commenced. Tests performed at the time found that results were equivalent.

The densities of model blocks are estimated by kriging along with copper and gold.

11.6 Database

Drillhole collars, downhole surveys, geological logs, data and bulk density data are stored in the acQuire database. This database uses SQL server tables and views. The advantage of this is that both Datamine and other MS Windows-based packages can connect directly to the tables using the ODBC links. Validation of the data entry is at the cell level, and is controlled by predetermined validation tables.

A number of checks have been incorporated into both SQL scripts and Datamine macros to ensure the integrity of the data. If everything passes this check, the de-surveyed data are ready for use in Datamine for Mineral Resource estimation.

11.7 Quality Assurance and Quality Control (QA/QC) Programmes

11.7.1 Historical (Pre-Ivanhoe)

Routine quality control was conducted at various stages throughout the sample preparation and analytical stages. These were done in the form of reference standards, replicate and duplicate samples and blanks.

The performance of blind standards in each batch is assessed prior to inclusion into the assay database. Duplicates are also assessed at this stage. Umpire samples are sent to SGS Analabs in Townsville to provide further external controls on assay QA/QC:

- **Reference standards** obtained from Ore Research & Exploration Pty Ltd (ORE) were used to measure the potential analytical accuracy of the Osborne laboratory. A review of exploration samples submitted over the same time period, as well as internal standard assays, does not indicate any systematic bias, precision, accuracy or contamination issues;
- **Duplicate sampling** was performed on the coarse reject after the sample had been crushed prior to pulverising in the LM5 or LM2; and
- **Replicate sampling** was also performed on the pulps.

Osborne Mine produced annual QA/QC summary reports for the period 2004 to 2009 that included the Kulthor samples.

A number of assaying audits and round-robin surveys were carried out by the former owners of Osborne:

- The Osborne Mine laboratory had no failures in the 2009 round-robin study for gold (Bloom, 2009);
- There were performance issues identified with the Osborne Mine laboratory in the Barrick Annual Quality Control and Quality Assurance Report for 2009 (Kuhneman et al., 2010). The mine laboratory was biased low for copper and gold. Although most exploration assaying was carried out at ALS Townsville, the thirteen Kulthor underground holes drilled in 2009 were assayed at the Osborne mine laboratory. Check samples from this programme were assayed at ALS Chemex, Townsville. The Townville assays are now used in place of the Osborne Mine assays;
- The 2010 assay round-robin (Hayes, 2010) reported generally satisfactory results for gold at the Osborne mine laboratory and the ALS Townsville laboratory. Copper was also satisfactory for the ALS Brisbane laboratory where most Osborne exploration copper assays are done; and
- The 2007 audit of mine and commercial laboratories in Australia (Smee, 2007) identified a number of shortcomings in sample preparation and assays at the Osborne mine laboratory and at the ALS Townsville laboratory. The most serious of these (inappropriate flux mixture) would result in under-estimation of gold grade.

11.7.2 Current (Ivanhoe)

QA/QC is managed by a dedicated and permanent group of technicians overseen by a geological manager. QA/QC processes are reviewed quarterly on site by the parent company's QA/QC Manager (Sketchley, 2012). This review process included all aspects of drilling data collection including collar and down-hole surveys, core logging, sample preparation and assaying.

Ivanhoe's core sampling within mineralised zones is generally taken on continuous one-metre intervals down each drillhole, or on smaller lengths over narrow geological units, for large disseminated or weakly mineralised zones sample lengths may increase to a maximum of two metres. The core is marked with a continuous cutting line along the middle, parallel to the long axis for the purpose of preventing a sampling bias during splitting. Core is cut with a rock saw flushed continually with fresh water and one-half of NQ/HQ core or one-quarter of PQ core is taken for analysis. RC samples are taken on continuous one- or two-metre intervals down each drillhole and collected from a rig-based cone splitter.

Sample dispatches include Certified Reference Materials (CRMs), Field Blanks, Field Duplicates, Crushed Duplicates, and Pulp Duplicates. The CRMs, Field Duplicates, and Field Blanks are randomly inserted during sampling, whereas the Crushed and Pulp Duplicates are inserted at the laboratory. CRMs are certified for gold, copper, molybdenum, and/or rhenium.

Samples are placed in plastic bags, sealed, and collected in large, labelled shipping bags that are secured and sealed with numbered tamper-proof security tags. Samples are shipped to ALS Laboratory Group's Mineral Division at Mount Isa for preparation. Gold, copper, molybdenum and rhenium assays, and multi-element geochemical analyses are conducted at ALS Mount Isa, Townsville, and Brisbane laboratories. ALS operates in accordance with ISO/IEC 17025.

Reference material assay values are tabulated and compared to those from established Round Robin programs. Values outside of pre-set tolerance limits are rejected and samples subject to re-assay. A reference material assay fails when the value is beyond the 3SD limit and any two consecutive assays fail when the values are beyond the 2SD limit on the same side of the mean. A Field Blank fails if the assay is over a pre-set limit.

Ivanhoe also regularly performs check assays at an independent third party laboratory, conducts onsite internal QA/QC reviews, and laboratory reviews to ensure procedural compliance for maintaining industry standard best practices.

11.8 LMRC Comments

In LMRC's opinion, the sampling preparation, security and analytical procedures used for Osborne and Kulthor drilling during the long exploration/development phase are consistent with generally accepted industry best practices and are therefore adequate. Any inadequacies are likely to result in slight under-estimation of grade.

The Osborne Mine had standard procedures in place to review and summarise QA/QC results before data were stored in the acQuire database.

The reliability of the assay data are further discussed in Section 12.

12 Data Verification

12.1 Data Verification of the acQuire Database

12.1.1 Historic (Pre-Ivanhoe)

All the data used for Mineral Resource estimation was collected by the previous owners of Osborne and Kulthor – PDAP and later Barrick. The Osborne Mineral Resource data have allowed the successful exploitation of the Osborne deposit from start of mining operations in 1995 till the deposit was sold to Ivanhoe in 2010. Approximately 2,550 holes have been drilled in the immediate Osborne-Kulthor area since exploration commenced in 1985. All the companies had formal specified QA/QC programmes.

The performance of the QA/QC for Osborne and Kulthor was reviewed by the senior Osborne mine geologists during the assay loading stage, prior to importing. Reports on standard, repeat and duplicate performance are generated by the acQuire Import object. These reports are used in the compilation of the batch QA/QC report. The final stage ensuring assay data is loaded into the database requires the acceptance of the QA/QC results. Refusal to accept the QA/QC performance prevents primary assay data from entering the database.

The Kulthor deposit has not been mined, so it does not have the “validation by production” available for the Osborne mineralisation. The Kulthor mineralisation is defined by 228 drillholes that actually intersect the main modelled mineralised zones (3,774 samples in the mineralised zones).

The entry of the data into the database was checked by LMRC in 2011 for 10% of these samples:

- 322 samples selected in the ore zones on a 1-in-10 basis;
- 30 samples of the 49 samples not located were assayed at the Osborne Mine Laboratory and hence had no electronic assay file - these samples were from the underground drilling programme; and
- No discrepancies were found - the data have been correctly loaded into acQuire.

12.1.2 Current (Ivanhoe)

Approximately 116 additional surface and underground holes have been drilled by Ivanhoe in the immediate Kulthor area since purchasing the Osborne copper-gold project assets from Barrick in 2010.

Ivanhoe has formal specified QA/QC programmes. The performance of the QA/QC is reviewed by the Ivanhoe QA/QC team during the assay loading stage, prior to importing. Reports on standard, repeat and duplicate performance are generated by the Import object and other reporting objects in acQuire. These reports are used in the compilation of the batch QA/QC report. The final stage ensuring assay data is loaded into the database requires the acceptance of the QA/QC results. Refusal to accept the QA/QC performance prevents primary assay data from entering the database.

12.2 Verification by LMRC

LMRC have not carried out any specific verification sampling or assaying for the Osborne copper-gold project, as:

- Until 2010 all of the sampling and assaying for the Osborne and Kulthor drillholes was done by PDAP and Barrick, two of the largest and most experienced gold mining companies in the world. Both had clearly-specified QA/QC procedures in place. The author was aware of these procedures, firstly as a senior employee of PDAP since the inception of the Osborne exploration, and later, as a consultant to Barrick. Both companies used reputable commercial laboratories for their sample preparation and assaying;

- No problems with drillhole sampling and assaying were identified throughout the extensive exploration and production history;
- The Kulthor drill core remaining from the pre-Ivanhoe drilling has been stored in the open and most of the depth marker information has been lost. It would be difficult to obtain quarter-core for check sampling and assaying over exactly the same intervals as were assayed originally; and
- The economic value of the Osborne and Kulthor deposits depends more on copper than on gold; copper has fewer assaying issues than gold.

As part of the data loading procedure for the updated Kulthor Mineral Resource estimate, the assays received in July 2012 were compared to those that were also available for the 2011 Mineral Resource estimation. The 2011 data were direct from the Barrick acQuire database. A number of differences were found, especially with gold assays. There were 298 samples where the gold assays differed by more than +/- 0.2 g/t; the mean difference was 0.92 g/t. Only four copper assays were found to be different. The differences with the gold assays were found to be mainly due to whether FA (fire assay) of SFA (screen fire assay) data were exported from acQuire; in some cases the more reliable SFA assay was not exported. For one drillhole, tellurium assays were mixed up with the gold assays. These problems were resolved by Ivanhoe staff and the database corrected before Mineral Resource estimation commenced.

The statistics of the Kulthor density data showed that a number of samples had anomalously low or high density values. The high values (> 5.0 t/m^3) were due to calibration values being included. Low values (< 1.5 t/m^3) were excluded if occurring more than 200 m down-hole from the surface as such low values are unlikely for fresh material.

LMRC is of the opinion that the Mineral Resource database is of adequate quality for updating the Osborne and Kulthor Mineral Resources.

12.2.1 Site Visit

The core storage and core sampling facilities at the Osborne mine site were visited for three days in March 2011 and again in May 2012). At the time of the 2011 visit, the core logging and core–cutting facility used by Barrick and PDAP was inspected and found to be of adequate standard (the facility was inactive at that time). Drill core storage was poor as the core boxes were in a core farm exposed to the weather.

A new core logging, cutting and storage facility was constructed by Ivanhoe in 2012. The new facility was found to be modern and well organised. Drill core is logged directly into portable computers before being split by diamond sawing using an automatic cutting facility.

12.2.2 Verifications of Analytical Quality Control Data

The Osborne Mine staff produced annual summaries of the QA/QC results for samples submitted to ALS Chemex and of Umpire assaying carried out at SGS Analabs. A total of 13 of these annual summary reports from 2004 to 2009 were reviewed by LMRC (SRK Consulting, 2011). These reports showed that the ALS Chemex assaying for gold and copper was quite satisfactory over this time period.

The control charts for the assays of copper and gold in five standards used during 2012 (the main period of Ivanhoe drilling) were reviewed by LMRC. The results were generally satisfactory but there may some be slight over-estimation of low copper assays (<= 0.5%) and slight under-estimation of high assays (> 1%). Gold assays may be overestimated by about 0.05 g/t. Analysis of sample blanks returned very low values, indicating that sample cross-contamination is not a problem.

13 Mineral Processing and Metallurgical Testing

13.1 Osborne Underground

Ore from Osborne underground has been the dominant feed to the Osborne concentrator and its metallurgical performance has been demonstrated over nearly 15 years of operation. The major minerals in the Osborne underground ore were iron oxides (principally magnetite), quartz and feldspar with minor amounts of chalcopyrite, amphibole, chlorite and iron sulphides. Minerals affecting flotation performance were pyrite, pyrrhotite, silica and talc.

The metallurgical outcome of treating material from the Osborne Underground Mine through the Osborne concentrator should be predictable considering the many years of actual production data available. This is based on the assumption that the material remaining underground has the same mineralogical characteristics.

Metallurgical performance for the "Kulthor Osborne Underground" material with a head grade of 1.57% copper and 0.94 g/t gold is predicted to be 85% copper recovery and 80% gold recovery into concentrate presumably grading ~23.5% copper. While individual recoveries have not been shown for "Osborne Underground" or "Kulthor Underground", the Osborne Underground material should give a copper recovery of 90% and gold recovery approaching 80% to a 23.5% copper concentrate.

However, three caveats should be borne in mind as follows:

- If the head grade is lower for the remaining material underground than that previously treated, then the flotation + gravity recoveries for copper and gold are likely to be lower as well;
- Sales terms for copper concentrates may have changed since the Osborne concentrator ceased operation in 2010; when these favour the smelters it suggests that the previous operating goal of a comparatively low concentrate grade of ~23.5% copper may not be the optimum. If economics favour a higher copper concentrate grade, then recovery would have to be traded off against concentrate grade; and
- Exposure of sulphides in underground workings could lead to "tarnishing" and oxidation adversely affecting flotation performance until sufficient new material has been stoped.

13.2 Kulthor Underground

As mentioned above, metallurgical performance for the "Kulthor Osborne Underground" material with a head grade of 1.57% copper and 0.94 g/t gold is predicted to be 85% copper recovery and 80% gold recovery into concentrate presumably grading ~23.5% copper.

The "Kulthor Feasibility Study" (Buxton 2007) had the following points on the processing of Kulthor material:

- "Time of grind" tests on Kulthor composites showed that the hardness was less than that for Osborne ore, so specific grindability tests were not done; and
- Used metallurgical performance of 85% copper recovery and 75% gold recovery (gravity + flotation) into a concentrate assaying 23% copper.

The mineralogy of Kulthor material has some important differences to that of Osborne ore previously treated in the Osborne concentrator:

- Iron sulphide (pyrite + pyrrhotite): copper ratio varies from ~11:1 for Main Lode High Pyrrhotite to ~5.5:1 for Main Lode Low Pyrrhotite and North Lode Low Pyrrhotite compared with ~3:1 for Osborne Underground material previously treated in the concentrator. Osborne material had more magnetite and less iron sulphides; and

- Pyrrhotite to chalcopyrite ratio in western vein material from Kulthor is 0.6:1 compared with 0.2:1 for Osborne.

Test work on Kulthor composites has given a wide range of metallurgical results:

- Main Lode Low Pyrrhotite: 73.5% to 79.4% copper recovery “normalised” to 26.4% copper concentrate grade;
- North Lode Low Pyrrhotite: 73.3% to 73.3% copper recovery “normalised” to 26.4% copper concentrate grade; and
- Main Lode High Pyrrhotite: 41% to 77.8% copper recovery “normalised” to 26.4% copper concentrate grade.

A blend of one third each of Kulthor Main Lode High Pyrrhotite, Osborne and Trekelano have 88% copper recovery “normalised” to a 26.4% copper concentrate grade.

The proportions of each Kulthor ore zone are then given as follows:

- 32.5% Main Lode Low Pyrrhotite;
- 45.4% Main Lode High Pyrrhotite;
- 20.1% North Lode Low Pyrrhotite; and
- 2% Central Lode.

The approach taken in the feasibility study was to generate two graphs using the weighted proportions of the individual composites which made up the Kulthor deposit as shown in Figure 13-1. The “best case” is the original optimal performance while the “worst case” represents the results from the repeat tests.

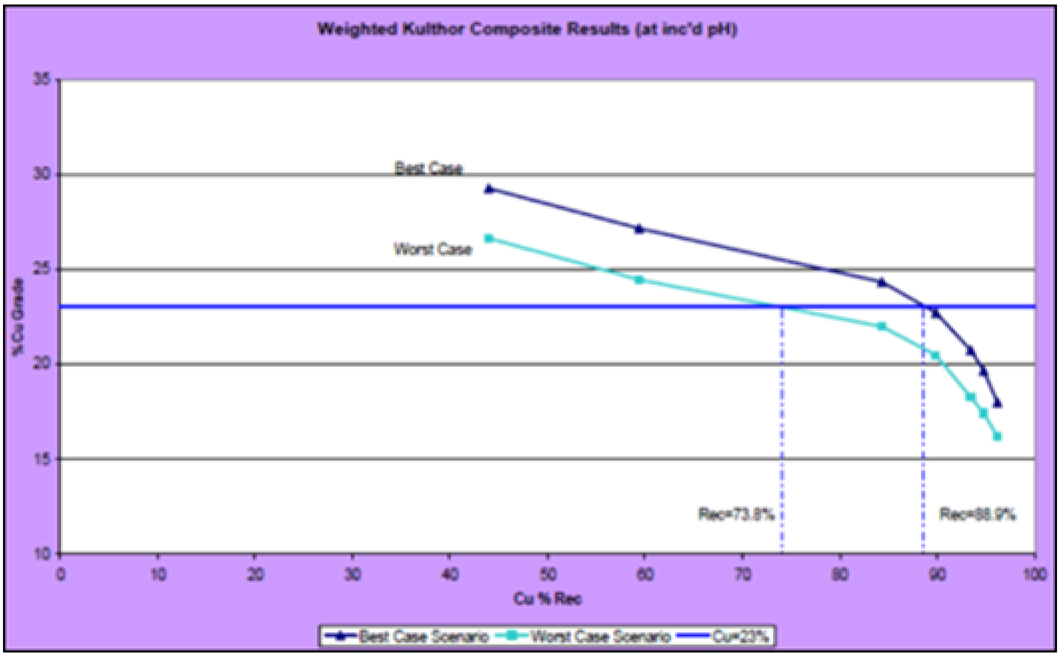


Figure 13-1: Kulthor metallurgical test work weighted by Composite

“Normalising” the results to concentrate grade at 23% copper gave 73.8% copper recovery for the “worst case” and 88.9% for the “best case” with a mean of 81.4%. Based on the proposition that previous laboratory test work on Osborne material gave recoveries 3-5% in absolute terms lower than those achieved in the concentrator a copper recovery of 85% was selected.

Two issues arise with this optimistic approach:

1 Experience relating the metallurgical performance of laboratory test work to that actually achieved in the Osborne Concentrator relates to Osborne ore only. The significant difference in mineral composition for Kulthor material, particularly its high iron sulphide: copper ratio compared to Osborne requiring stronger depressant conditions, strongly suggests that it will not be valid; and

2 Osborne Underground" material is less than 12% of total "Kulthor Osborne Underground" material to be processed, strongly suggesting that the better results indicated from treating a blend are not likely to be replicated as the performance of the Kulthor material will be dominant.

For the study, a copper recovery of 80% and gold recovery of 75% to a 23% copper concentrate should be used. As mentioned for other sources these recoveries will have to be adjusted downwards if a higher copper concentrate grade is targeted.

A programme of metallurgical test work will be required to better define the flotation performance (+ gravity for gold if applicable) for a definitive feasibility study, this will include:

- Grinding and flotation tests, preferable supported by quantitative mineralogy to determine both the metallurgical performance of the material and the suitability of the current flowsheet configuration and equipment in the Osborne concentrator to treat it. This should include mixed milling of the various ore types.

13.3 Osborne Open Pit High Grade (Red)

The Osborne copper concentrator should be capable of treating material with primary copper sulphide mineralisation. Metallurgical performance for the "Osborne Open Pit High Grade (Red)" material with a head grade of 0.89% copper and 0.67 g/t gold is predicted to be 85% copper recovery and 75% gold recovery into concentrate presumably grading ~23.5% copper concentrate which gives a tailing assaying of ~0.14% copper. While these recoveries may look optimistic, considering that the head grade of the material open pit is less than half that of ore treated to date from the Osborne underground mine, actual performance data from recent treatment of1125 1M stopes material supports these figures. This material is reported to be identical to the open pit material to be treated.

The Osborne concentrator will not recover non-sulphide copper minerals such as azurite, malachite, cuprite, tenorite and chrysocolla. While azurite + malachite can readily be recovered by sulphidisation flotation (also possibly using collectors of the hydroxamate type), and while cuprite + tenorite are less amenable but sulphidisation can be done with more difficulty, chrysocolla is not recovered by this technique. Sulphidisation has to be done after recovery of the sulphide minerals, this will require additional rougher flotation capacity and separate cleaner flotation for which the plant is not currently configured. Processing material with non-sulphide copper minerals means a loss of copper recovery or a reduction in throughput to provide the additional flotation capacity. When Osborne started in 1995, the copper concentrate produced from the open pit contained excessive chlorine due to the presence of the mineral atacamite $Cu_2(OH)_3Cl$ in the oxide and transition zone material.

Hence, while the metallurgical performance data could be tentatively used for a scoping study, a programme of metallurgical test work will be required to better define the flotation performance (+ gravity for gold if applicable) for a definitive feasibility study.

Aspects that should be examined include the following:

- Definition of the zone of primary copper sulphide mineralisation, this will involve a combination of geological logging, quantitative mineralogy and/or sequential copper assaying and possibly chemical tests such as ethylenediaminetetraacetic acid (EDTA) extraction if heavy metal ion species are present; and
- Grinding and flotation tests, preferably supported by quantitative mineralogy to determine both the metallurgical performance of the material and the suitability of the current flowsheet configuration and equipment in the Osborne concentrator to treat it. This should include mixed milling of the various ore types if they cannot be treated in separate campaigns.

13.4 Osborne Open Pit Low Grade (Yellow)

Metallurgical performance of the low grade (yellow) material is anticipated to be less than the 85% copper recovery and 75% gold recovery predicted for the higher grade (red) material.

Metallurgical performance for the “Osborne Open Pit Low Grade (Yellow)” material with a head grade of 0.45% copper and 0.35 g/t gold is estimated based on a conservative assumption of a “constant tailing” hypothesis for the open pit materials both high grade and low grade so using the 0.18% copper value derived from the metallurgical performance of the “Osborne Open Pit High Grade (Red)” material in Section 0 gives a copper recovery of ~60%. Applying similar logic to gold gives a gold recovery of ~45%.

As for the “Osborne Open Pit High Grade (Red)” material, future metallurgical test work required for a definitive feasibility study should include the following:

- Definition of the zone of primary copper sulphide mineralisation, this will involve a combination of geological logging, quantitative mineralogy and/or sequential copper assaying and possibly chemical tests such as EDTA extraction if heavy metal ion species are present; and
- Grinding and flotation tests, preferably supported by quantitative mineralogy to determine both the metallurgical performance of the material and the suitability of the current flowsheet configuration and equipment in the Osborne concentrator to treat it. This should include mixed milling of the various ore types.

14 Mineral Resource Estimates

14.1 Introduction

The three Mineral Resource Statements prepared in accordance with the Canadian Securities Administrators' NI 43-101 are presented in this Technical Report: The Osborne 1SS, the Open Pit area and the Kulthor zone (Osborne copper-goldproject).

The Mineral Resource models prepared by LMRC use all drilling and sampling carried out over the life of the project (from 1998). The data were collected by PDAP, Barrick and Ivanhoe. The Mineral Resource estimation work was completed by R W Lewis, FAusIMM (No 100799), and an appropriate independent Qualified Person as this term is defined in NI 43-101.

This section describes the Mineral Resource estimation methodology and summarises the key assumptions considered by LMRC. In the opinion of LMRC, the Mineral Resource evaluation reported herein is a reasonable representation of the global copper-gold Mineral Resources found in the Osborne Project at the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

The database used to estimate the Osborne copper-gold project Mineral Resources was checked for consistency by LMRC but not audited in the strictest sense. The database has been in use for several years during successful copper-gold production at the Osborne Mine by PDAP and Barrick. LMRC is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for the Osborne 1SS, Open Pit and Kulthor mineralisation and that the assay data are sufficiently reliable to support Mineral Resource estimation.

Datamine Studio Versions 3.19 and 3.21 were used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate Mineral Resources. The Datamine and the Geostatistical Software Library (GSLib) family of software were used for geostatistical analysis.

14.2 Mineral Resource Estimation Procedures (1SS, Open Pit and Kulthor)

The Mineral Resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the mineralisation;
- Definition of Mineral Resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Mineral Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades; and
- Preparation of the Mineral Resource Statement.

14.3 Cut-off Grades

The assessment of “reasonable prospects for economic extraction” is dependent on the cut-off grades used. As both copper and gold contribute to revenue, a “copper equivalent” cut-off grade is used (eCu). The cut-off grades and the eCu factor were calculated using the following:

- Metal Prices copper: USD3.75/lb, gold: USD1400/oz;
- The USD/AUD exchange rate was 1.0;
- The eCu factor was calculated from the ratio of the value of 1 g/t gold in mill feed to that of 1% copper in mill feed, taking into account metallurgical recovery, transport, smelting and refining charges and deductions and the Queensland Government Royalty. The calculated eCu factor (eCu=copper% + gold g/t * Factor) was 0.6;
- The break-even and Incremental cut-off grades were calculated from the ratio of the total site costs plus mining costs per tonne to the value per percent copper in mill feed;
  - The total site costs in AUD/t (excluding mining costs were the same for all three Mineral Resources. These included processing costs, asset management costs, commercial unit costs, Human Resources unit costs, Environmental, Safety and Security and Sustainability;
  - The mining costs for Kulthor and 1SS included Crushing and Hoisting, Trucking, Loading, Drill and Blasting, Filling, Development and Support, and Services and Administration. These were similar for both these Mineral Resources;
  - The mining costs for the Open Pit included, Drill and Blasting costs, Load and Haul, run-of-mine (ROM) Haulage, Road maintenance and Site Management; and
- The calculated break-even cut-offs were:
  - 1SS 1.15%
  - Kulthor 1.22%
  - Open Pit 0.53%
- The incremental cut-offs were approximately 50% of the break-even cut-offs;
- On the basis of these calculations, it is reasonable to use 1.2% eCu as a cut-off for underground mining and 0.5% eCu for open cut mining; and
- It is also reasonable to use the incremental cut-offs for the limit of the dilution envelopes.

The eCu factor of 0.6 reasonably reflects the relative value of copper and gold, taking the metal prices, recoveries, transport costs, smelting charges and losses into account.

14.4 Mineral Resource Estimation 1SS

14.4.1 Mineral Resource Database 1SS

The Mineral Resource database for 1SS Deeps contains data for 723 drillholes and channel sample lines. There were 567 drillholes with 15,704 copper assays and 156 channel samples and short stab holes with 881 copper assays. Not all of these data were in the ore zones. The channel samples and stab holes make up a small proportion of the database, and occur only in the previously mined portion of 1SS. Their influence is therefore small.

There were some minor problems with the drillhole database. Ten holes had surveys but no collars; only two of these had assays. There were some bad specific gravity (SG) data in the database for TT prefix drillholes (old data pre-mine). There were a large number of SG values equal to 1.9 in these holes that were culled when the data were loaded into Datamine.

Further trimming was necessary, as there were also other low SG data. All SG values less than 2.6 were set to missing; such low values are quite unlikely at Osborne below the zone of oxidation.

Data are dumped from the Osborne acQuire database using a “project” field. It was found that this project field can be misleading because some holes may intersect mineralisation in more than one project area. In order to ensure that all the 1SS data were available, a complete dump of data from acQuire was made.

14.4.2 Solid Body Modelling

The 1SS Deep zone is the down-dip extension of the 1SS Zone that has been mined over a vertical extent of more than 650 m. The mining does not extend below 125 mRL but mineralisation (at lower grades) continues down to at least -200 mRL, the lower limit of drilling. In addition to the unmined mineralisation below 125 mRL, there is also mineralisation lateral to stopes mined above. Mineralisation wireframes at 0.6% eCu (dilution zone) and 1.2% eCu (ore zone) were modelled from 790 mRL down to -265 mRL, even though much of the zone above 125 mRL has been mined previously. This allowed drillhole data in the mined-out area to be used to inform blocks both below and lateral to the stopes to be used for estimation of the related mineralisation. As the 1.2% eCu zone is an island domain in the 0.6% eCu domain, a priority list was used for drillhole data tagging and block model construction. This prevented drillholes data being used in more than one domain, and the low-grade dilution zone overprinting the included high-grade ore zone. Figure 14-1 shows a plan projection of the mined-out area (stopes) along with the hull (maximum extent) of the area modelled. Figure 14-2 shows the drillholes available to estimate the 1SS Zone. The lower drillhole density at depth is very clear. There are not a lot of drillholes north and south of the east-dipping mineralisation in Figure 14-3. The tagging process resulted in addition of a DOM field in the drillhole file with a value of “1” for the Inner Ore Zone and “2” for the Dilution Zone. All other samples received “0”.

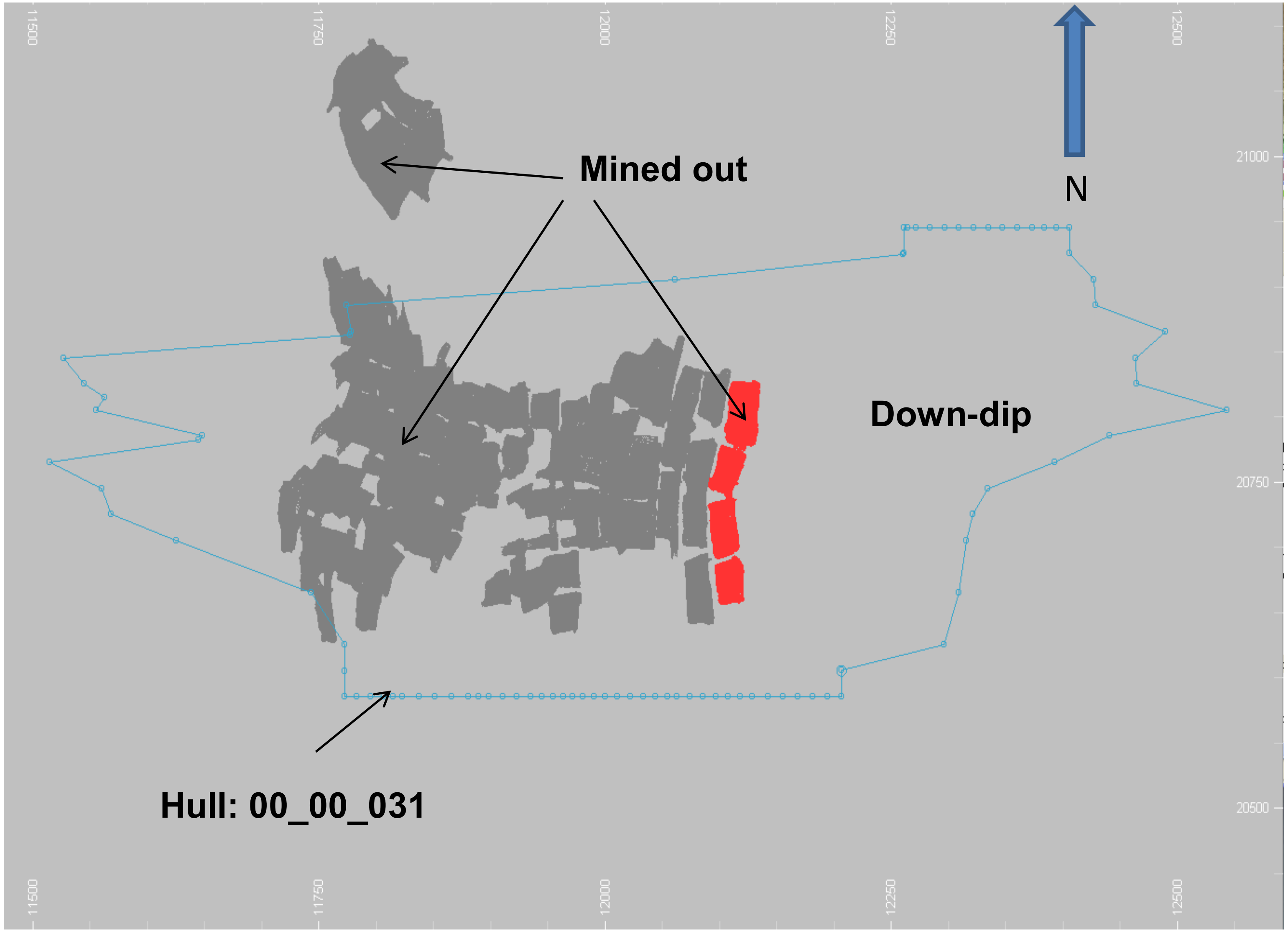


Figure 14-1: Extent of stoping in the 1SS Zone compared to the 2011 limits of modelling plan

Source: LMRC, June 2011

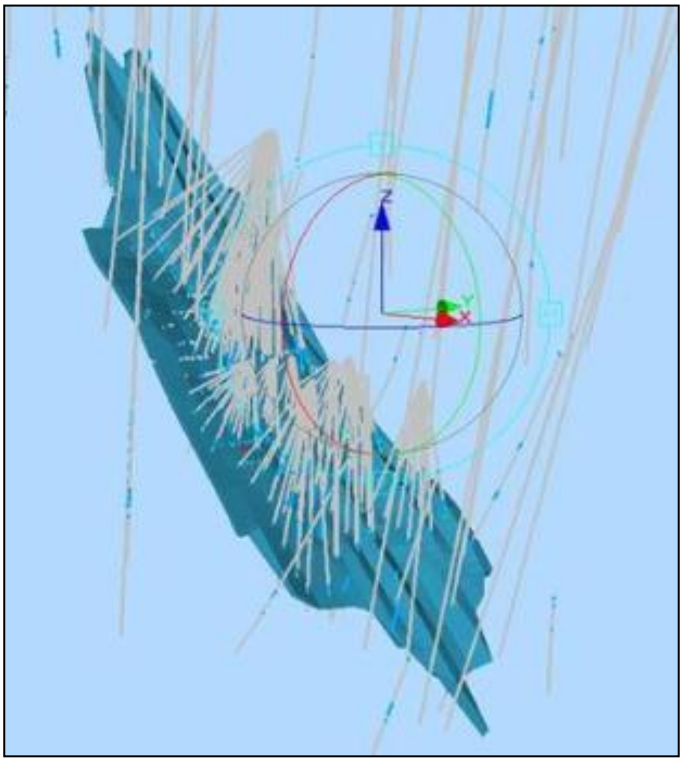


Figure 14-2: Drillholes through the 1SS Zone

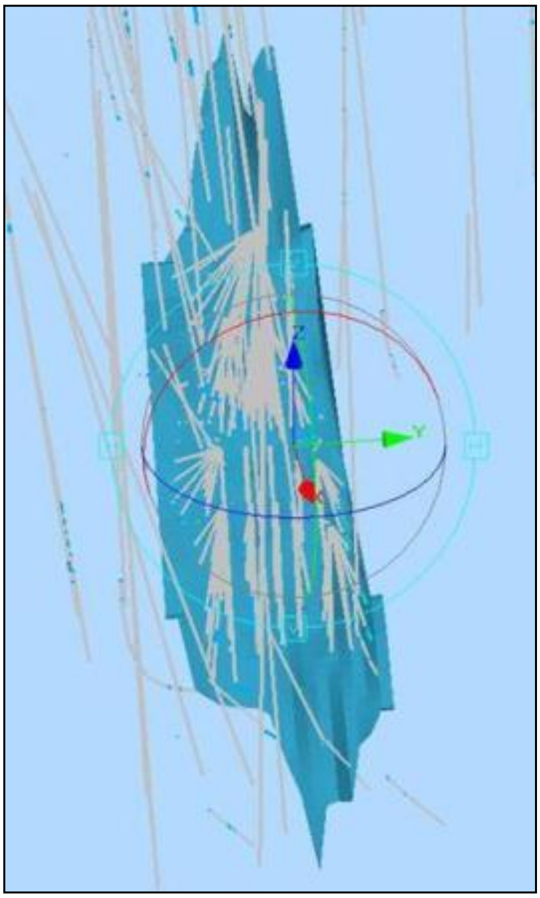


Figure 14-3: 1SS Dilution zone looking west

A section through the 1SS is shown in Figure 14-4. The relatively narrow higher grade ore zone is surrounded by lower grade mineralisation that is captured in the dilution zone. The use of an estimated dilution zone allows non-zero grades to be used when adjusting reserves for expected dilution.

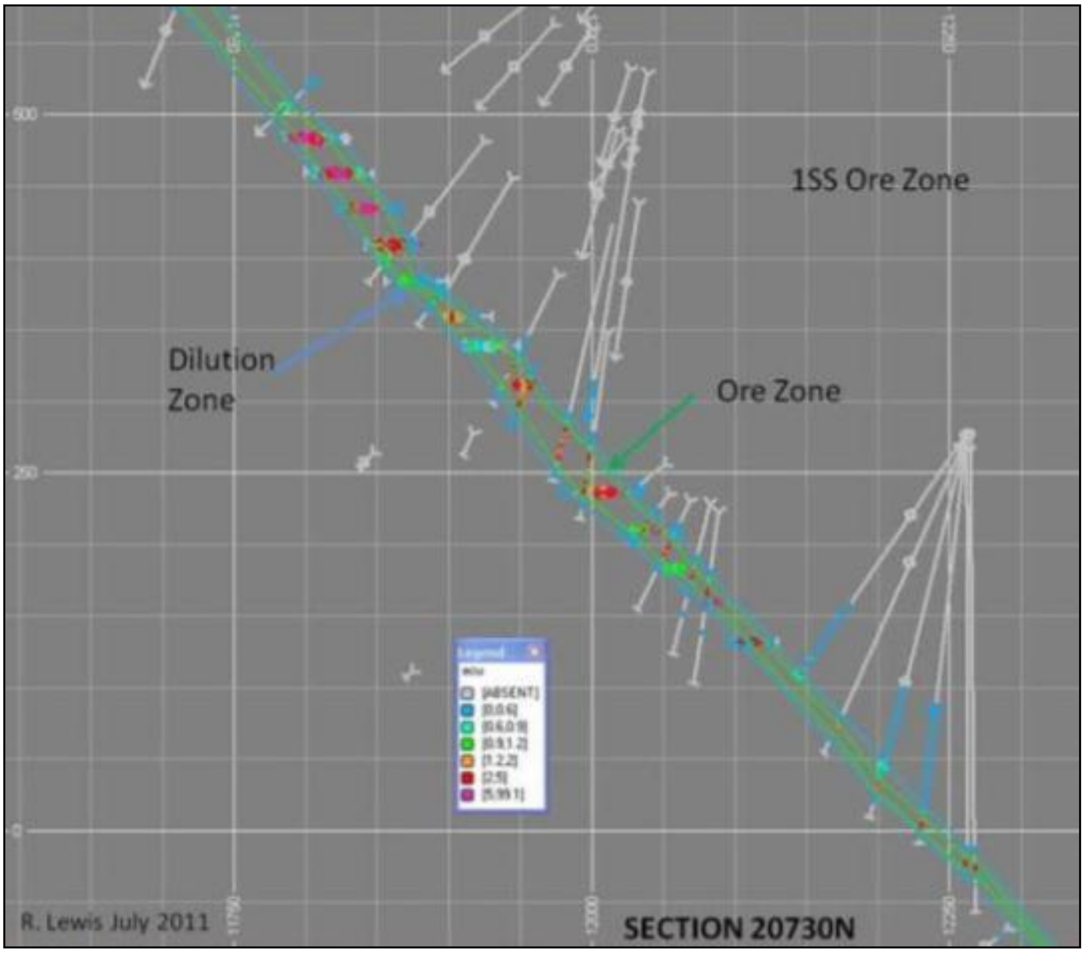


Figure 14-4: Typical section through the 1SS Zone at 20730N

After the drillhole data were tagged by the wireframes (in priority order), they were further processed to adjust the tagging. Any samples immediately outside the main ore zone that were greater than 0.6% eCu were re-tagged to belong to the main ore zone. Any samples immediately outside the dilution zone that were >=0.6% eCu were retagged to belong in the dilution zone. Only a single additional sample was re-tagged for each drillhole at each ore zone contact (if meeting the requirement).

This re-tagging had two purposes:

1 When snapping strings to drillholes during wireframe construction, it is easy to snap to the midpoint of assays instead of the end of assays when using 1 m samples. This may result in mis-selection of data by the wireframe; and

2 Although the main 1SS ore zone has naturally sharp boundaries in most places, there are places where the boundary is more gradual. In such places, the use of an assay cut-off can result in over-estimation of grade. Adding an additional sample with a grade greater than or equal to half the nominal cut-off removes the over-estimation. This is particularly important with Kriging as samples at the boundaries of domains receive high Kriging weights.

Re-tagging resulted in an increase in the number of samples in the main ore zone from 5,276 to 6,038. The average grade dropped from 2.37% eCu to 2.22% eCu.

14.4.3 Compositing

The most common sample lengths used for assaying at Osborne are 2 m (pre-mine surface holes) and 1 m in later drilling. The data were composited to 1 m intervals with zonal control (field DOM). Zonal compositing can result in short composites where drillholes leave a domain. It is usual to exclude samples less than half the compositing interval when making estimates. This prevents short samples (especially on boundaries) having equal (or higher in the case of Kriging) weights. Discarding short composites does result in a loss of information.

There are two options available to prevent this happening:

1 The length of all composites in a drillhole intersection through a domain can be adjusted so equal length composites are created. This works best with wide orebodies; and

2 Short composites (less than 0.5 m) can be combined with the neighbouring composites in the zone.

The second method was used as the 1SS ore zone is relatively narrow. Any composites >0.5 m are used for estimation, so there was no need to combine composites >=0.5 m <1.0 m.

14.4.4 Statistical Analysis and Evaluation of Outliers

The unmined portion of the 1SS Zone has been drilled by a mixture of surface holes and holes drilled from underground. In general, all drilling below -60 mRL is by surface holes (with one exception).

Figure 14-5 shows the ore zone intersections, colour-coded by type. The grade of the surface ore zone intersections is lower than those of the underground holes in Figure 14-6, but the restriction of underground drillholes to above -60 mRL makes it difficult to know if this is due to decreasing grade with depth. There is more positional uncertainty in the deep surface intersections as the deepest intersections are more than 1600 m downhole from the surface. This is not critical, as the surface drillhole intersections are wide-spaced.

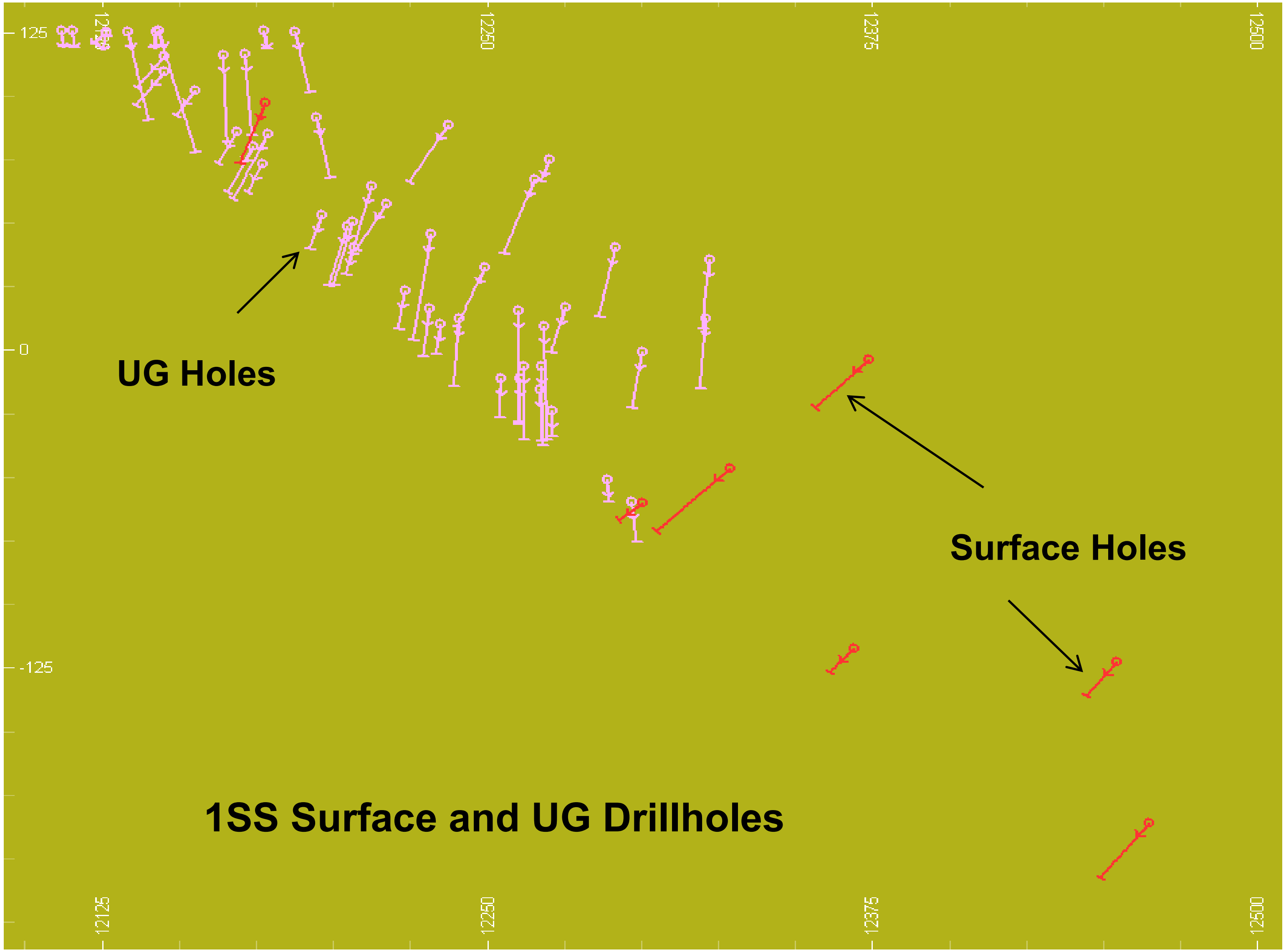


Figure 14-5: Drillhole Intersections below mRL125 (looking north)

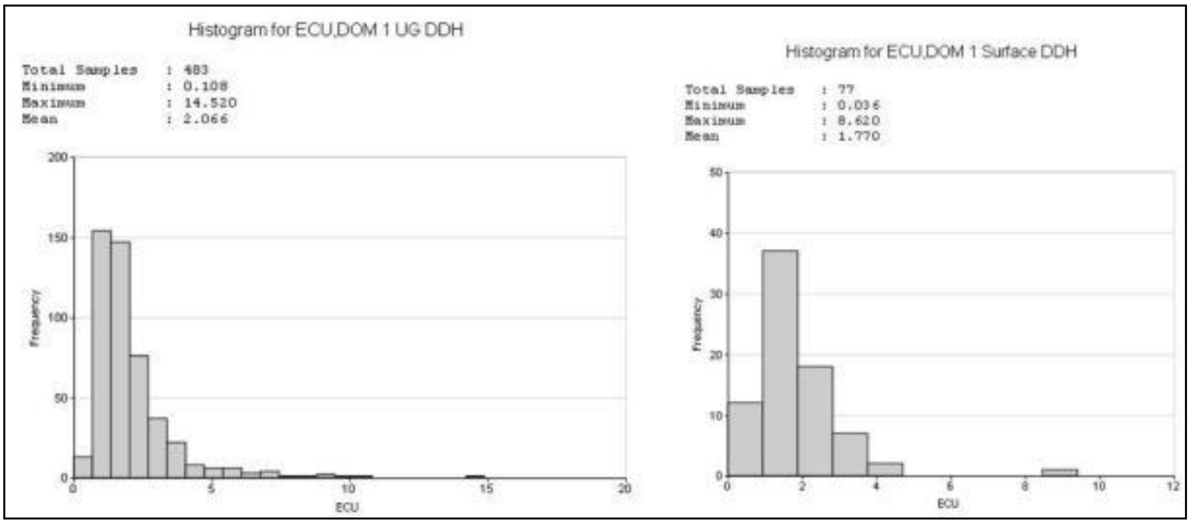


Figure 14-6: Histograms of UG and Surface Ore Zone data

The composites were declustered using the polygonal method. This method utilises a geology block model in the declustering. As polygonal declustering can result in high weights where drillholes enter and leave domains, the weights of such samples were adjusted to be the same as that of the next sample in the same hole inside the domain. Weights were further trimmed to remove outliers.

Figure 14-7 and Figure 14-8 show boxplots of the declustered 1 m composites for copper and gold by domain. The Coefficient of Variation (CV) is relatively low, except for gold in Dom2 (dilution zone). There are high values that will require top-cutting (e.g. the maximum gold grade is 49.6 g/t). Figure 14-9 shows the boxplots for SG. The average SG values by domain were used later to supply default SG values where the lower number of SG data resulted in lack of estimates for blocks with copper and gold grades. The SG data were limited to below 150 mRL as the mean SG values for the full 1SS Zone are lower (3.81, 3.55) than the values in Figure 14-9.

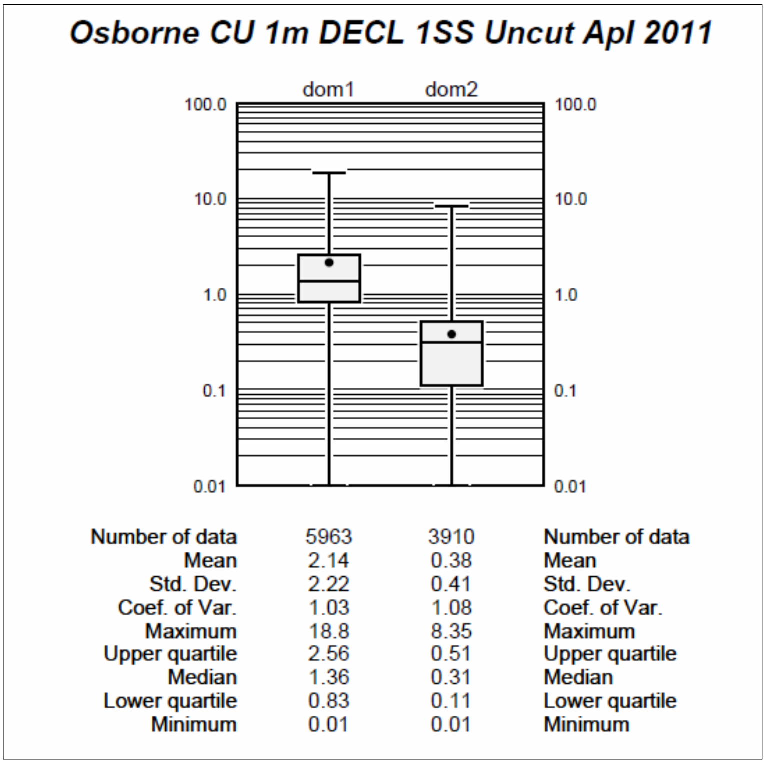


Figure 14-7: Boxplot of copper in declustered 1 m composites

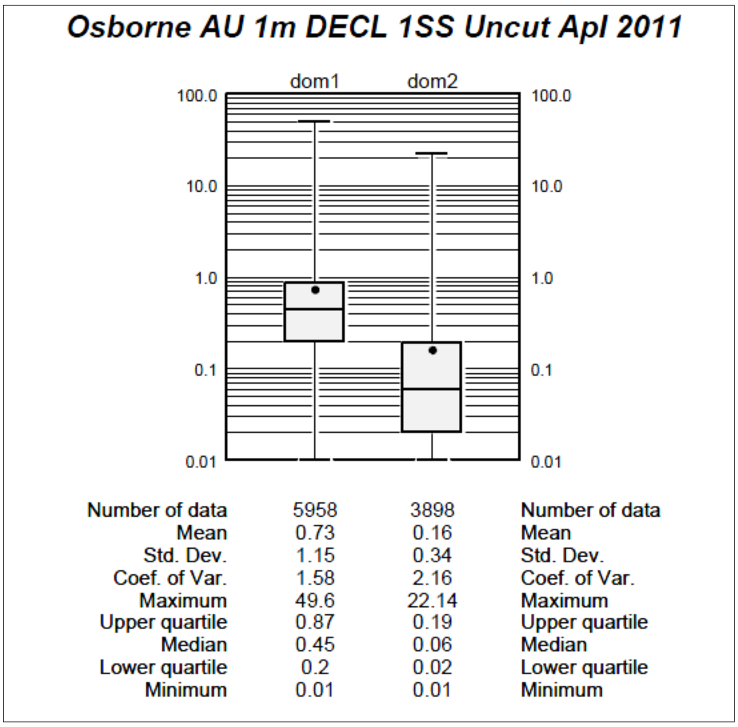


Figure 14-8: Boxplot of gold in declustered 1 m composites

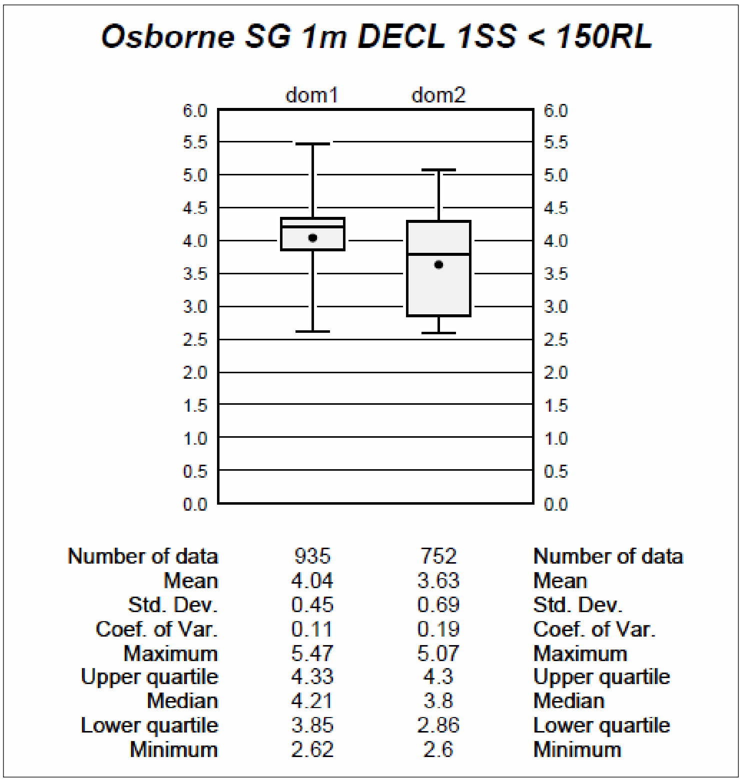
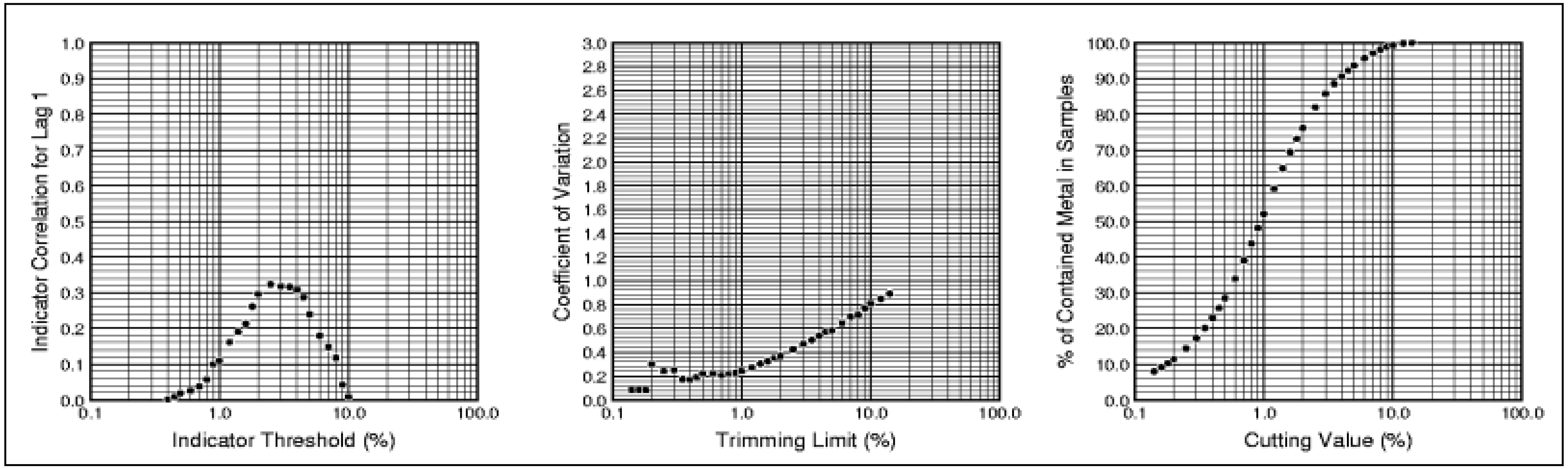
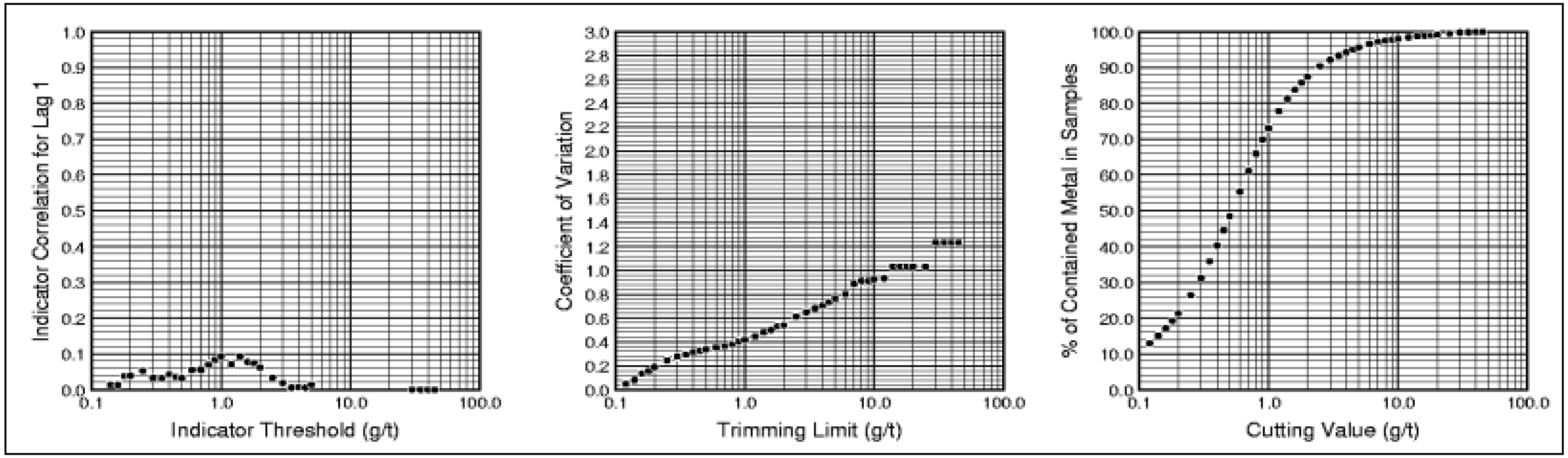


Figure 14-9: Boxplot of SG in declustered 1 m composites

Cutting statistic plots were used along with histogram and probability plots to choose top-cuts. The cutting statistic plots show the relationship between top-cut and CV and between top-cut and total metal. The Indicator Threshold plot provides information about continuity as a function of top-cut. Figure 14-10 shows the plots for copper and gold in the 1SS ore zone, below 150 mRL



Copper



Gold

Figure 14-10: Cutting statistic plots for copper and gold in the ore zone

The histogram and probability plots for copper and gold in the ore zone are shown in Figure 14-11. In this plot, any values in the histogram less than 1.0 have been suppressed so the higher values that might require top-cutting can be seen.

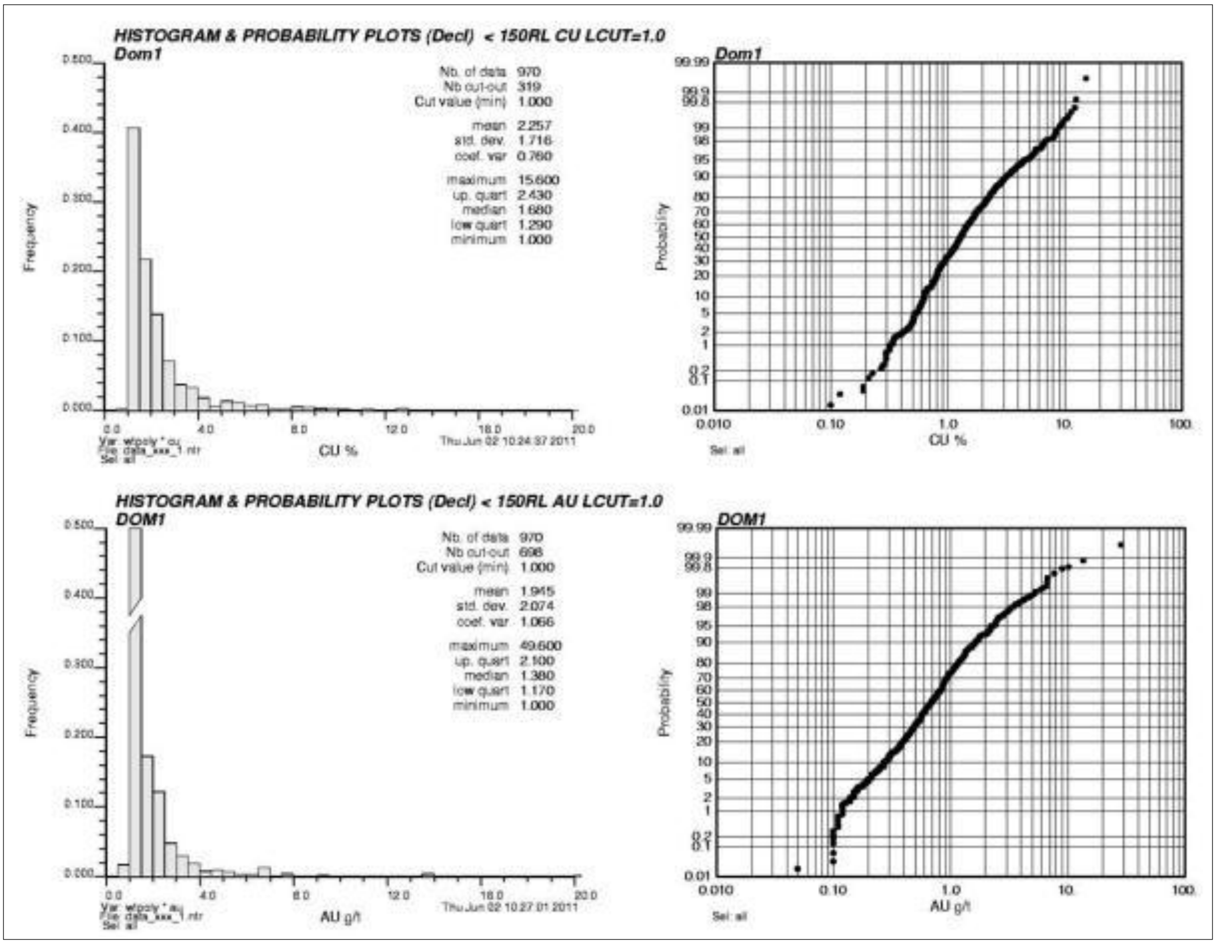


Figure 14-11: Histogram and probability plots for the ore zone

The chosen top-cuts are shown in Table 14-1. The number of 1 m composites top-cut is also shown.

Table 14-1: Top-cuts for copper and gold (1 m composites)

Name	DOM	Top-cut Copper	Nb cut Copper	Top-cut Gold	Nb cut Gold
Ore zone	1	9.0	12	7.0	6
Dil zone	2	3.0	3	2.6	2

Copper and gold have significant correlation as shown in Figure 14-12. This provides justification for using the same search filter dimension for copper and gold. If there are estimates for copper but not for gold, then the mineralisation is being under-valued.

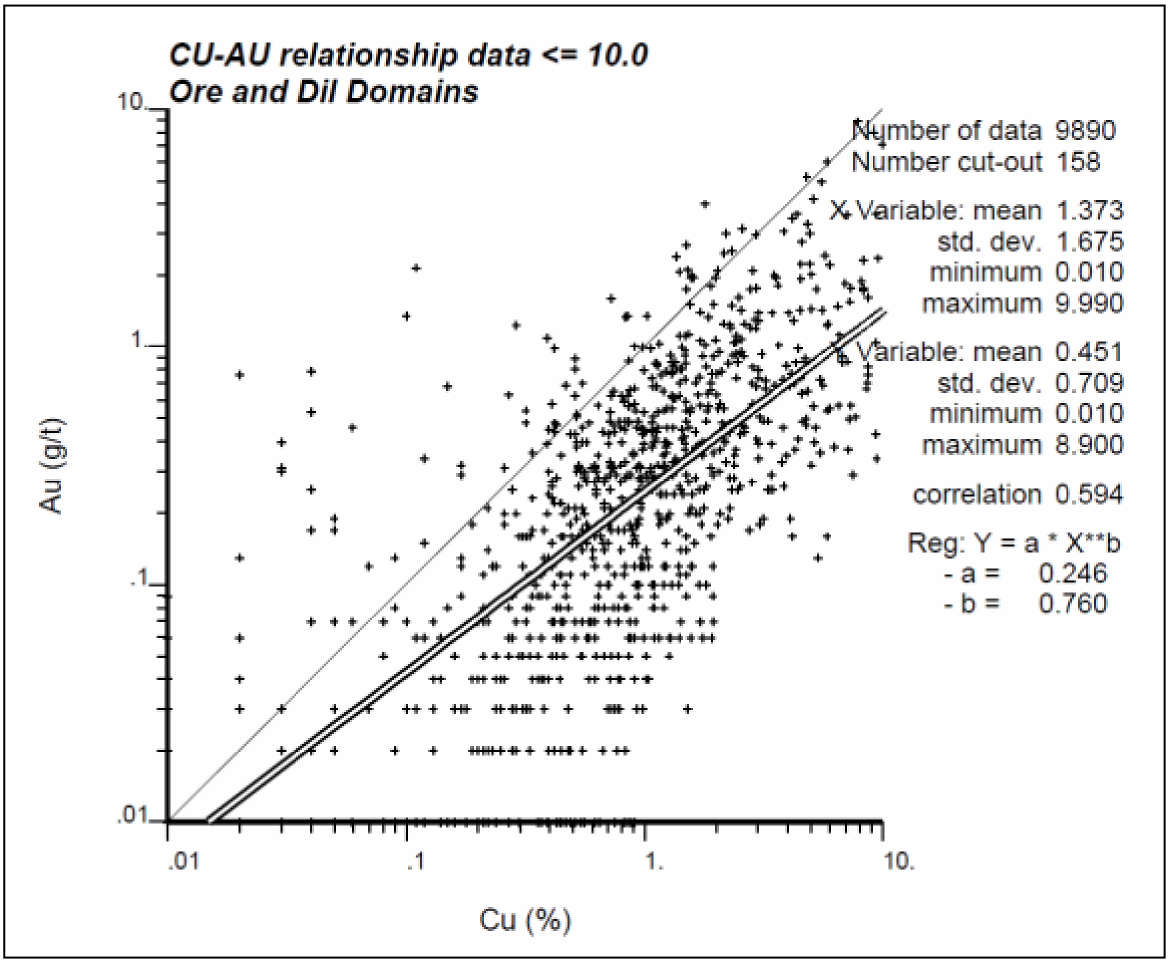


Figure 14-12: Scatterplot between copper and gold in the ore and dilution zones

14.4.5 Variography

Relative by Pair (RLP) semi-variograms were produced for copper, gold and SG for each domain using top-cut data. Both down-the-hole (DTH) and 3D variograms were used. Spherical variogram models were fitted. The variograms for copper and gold for the ore zone are shown in Figure 14-13. The variogram models used for estimation of copper, gold and SG for the ore and dilution zones are shown in Table 14-2.

In general, there were few data available to inform the variograms at short ranges for the down-dip (090/-49) and along-strike (360/0) directions. The variograms were better informed in the cross-body direction as this is the direction of drilling. Ranges were shorter for gold than copper, with more influence of shorter range structures. This is in general agreement with previous observations at Osborne – gold is more erratic and “clustered”. The variogram models for SG were relatively short range, but well informed.

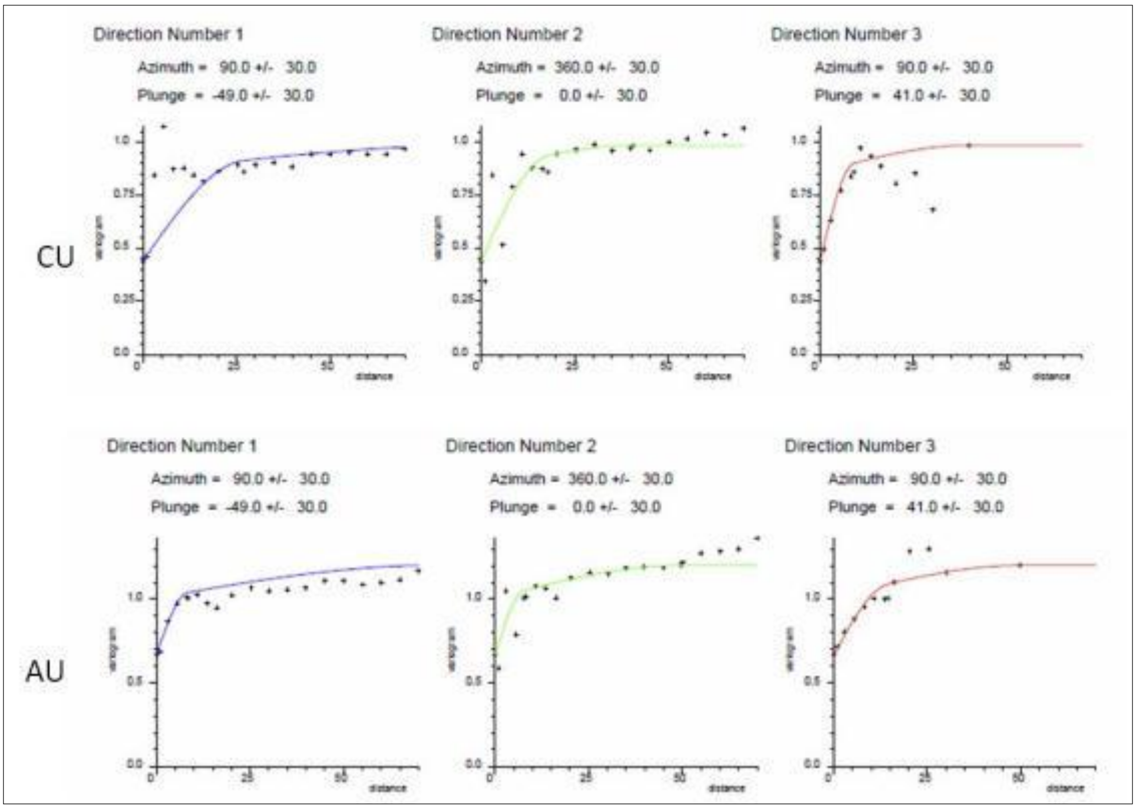


Figure 14-13: RLP variograms for copper and gold in the ore zone

Table 14-2: Variogram parameters

Est1511cvp											
VREFNUM	**DOM**	**METAL**	**VANGLE1**	**VANGLE1**	**VANGLE1**	**VAXIS1**	**VAXIS2**	**VAXIS3**			
1	1	CU	96	47	0	3	1	2			
2	2	CU	96	47	0	3	1	2			
3	1	AU	96	47	0	3	1	2			
4	2	AU	96	47	0	3	1	2			
5	1	SG	96	47	0	3	1	2			
6	2	SG	96	47	0	3	1	2			
VREFNUM	**NUGGET**	**ST1**	**ST1PAR1**	**ST1PAR2**	**ST1PAR3**	**ST1PAR4**	**ST2**	**ST2PAR1**	**ST2PAR2**	**ST2PAR3**	**ST2PAR4**
1	0.4400	1	17.70	27.10	9.12	0.4180	1	40.60	89.80	39.80	0.1260
2	0.2480	1	8.08	32.80	4.60	1.0250	1	43.90	67.50	10.70	0.2970
3	0.6680	1	7.74	8.44	14.50	0.3400	1	49.70	73.70	49.70	0.1950
4	0.7590	1	26.10	10.10	8.32	0.4340	1	58.40	58.40	58.40	0.1800
5	0.0170	1	15.70	16.70	7.78	0.0220	1	-	-	-	-
6	0.0200	1	18.00	14.00	5.99	0.0330	1	-	-	-	-

14.4.6 Block Model and Grade Estimation

The block model extent was increased from the previous estimate:

- Include more mineralisation lateral to the area already mined; and
- Model below the mined out area (below 125 mRL) to the limit of drilling.

Table 14-3: Model Extents 2008 and 2011

2008 Model – Combined Block Model					2011 Model – Combined Block Model			
	Lower	**Upper**	**Size**	**Number**	**Lower**	**Upper**	**Size**	**Number**
East	11700	12530	5	166	11700	12550	5	170
North	20550	20990	5	88	20550	20990	5	88
Elevation	-150	570	5	144	-265	570	5	167
			Number	**2,103,552**			**Number**	**2,498,320**

Sub-blocking allowed blocks to be down to 0.625 m in X, and 1.25 m in Y and Z. The sub-blocking allowed better filling of the narrow ore zone. When the model prototype was assembled by superimposing the ore zone model on the dilution zone model, smaller sub-blocks were created in the dilution zone along its boundary with the ore zone. Prior to superimposition, the minimum block size in the dilution zone was 5 m x 2.5 m x 2.5 m (XYZ), whereas in the ore zone it was 0.625 m x 1.25 m x 1.25 m (XYZ), allowing more resolution in the ore zone.

A regular block model was also estimated as this is required for some validation tests.

14.4.7 Dynamic Anisotropy

Dynamic Anisotropy modelling was used to handle the minor kinks in the 1SS ore zone. In this technique, each block has a unique search orientation. The model was built by digitising strings in plan and in parallel sections approximately normal to strike. The strings were conditioned to short segments (20 m), converted to points and used to estimate unique dip and dip-azimuth orientations for every block. Estimation of the anisotropy directions was done using Inverse Distance Cubed (ID^3) and a maximum of two points to make an estimate. The dips were corrected to true dips using the APTOTRUE process. The anisotropy model has two new fields: TRDIP and TRDIPDIR which allowed the search and variogram directions to be locally correct for every block during grade estimation. This happens automatically if these fields are defined in the search parameter file.

14.4.8 Estimation

Estimation of copper, gold and SG was carried out using a Datamine macro. This macro requires two special parameter files in addition to the three (search, estimation and variogram) parameter files required by the ESTIMA process. The additional files define top-cuts and boundary crossing (if any). The parameter files are kept in named sets with a common prefix and specified suffix (ep, sp, vp, tc, bd). The macro allows estimation with or without Dynamic Anisotropy and parent or sub-block estimation. Discretisation of 2 x 2 x 2 was used when estimating blocks and sub-blocks.

The estimation methods used were ordinary Kriging, Inverse Distance Cubed (ID^3) and Nearest Neighbour (NN). The latter two estimation methods were used to validate the Kriged estimate.

Several estimation runs were completed:

- With and without Dynamic Anisotropy;

- Parent block and sub-block estimation (with parent block estimation, all sub-blocks in a parent block receive the same grade); and
- Sub-block and regular block estimation (a regular block model is useful for model validation, even though it does not completely fill the ore and dilution zones).

The estimation and search parameters are shown in Table 14-4 and Table 14-5.

Table 14-4: Estimation parameters

est1511cep							
DOM	**Method**	**SREFNUM**	**VALUE_IN**	**VALUE_OU**	**IMETHOD**	**POWER**	**NUMSAM_F**
1	ok	1	CU	CU	3	1	NSCU
2	ok	2	CU	CU	3	1	NSCU
1	id3	1	CU	CUID3	2	3	
2	id3	2	CU	CUID3	2	3	
1	cs	1	CU	CUNN	1	1	
2	cs	2	CU	CUNN	1	1	
1	ok	1	AU	AU	3	1	
2	ok	2	AU	AU	3	1	
1	id3	1	AU	AUID3	2	3	
2	id3	2	AU	AUID3	2	3	
1	cs	1	AU	AUNN	1	1	
2	cs	2	AU	AUNN	1	1	
1	ok	1	SG	SG	3	1	
2	ok	2	SG	SG	3	1	
DOM	**VREFNUM**	**ANISO**	**SVOL_F**	**MINDIS_F**	**KRIGNEGW**	**KRIGVARS**	**VAR_F**
1	1	1	SVOLCU	MDISCU	1	1	KVARCU
2	2	1	SVOLCU	MDISCU	1	1	KVARCU
1	-	1			-	-	
2	-	1			-	-	
1	-	1			-	-	
2	-	1			-	-	
1	3	1			1	1	KVARAU
2	4	1			1	1	KVARAU
1	-	1			1	1	
2	-	1			1	1	
1	-	1			1	1	
2	-	1			1	1	
1	5	1	SVOLSG		1	1	
2	6	1	SVOLSG		1	1	

Table 14-5: Search parameters used for estimation

est0511csp								
SREFNUM	DOM	METAL	SMETHOD	SDIST1	SDIST2	SDIST3	OCTMETH	MINOCT
1	1	CU	2	50	50	7	1	2
2	2	CU	2	60	60	7	1	2

SREFNUM	MINPEROC	MAXPEROC	SANGLE1	SANGLE2	SANGLE3	SAXIS1	SAXIS2	SAXIS3
1	1	8	96	47	0	3	1	2
2	1	8	96	47	0	3	1	2

SREFNUM	MINNUM1	MAXNUM1	SVOLFAC2	MINNUM2	MAXNUM2	SVOLFAC3	MINNUM3	MAXNUM3
1	5	25	2.0	5	15	3.0	4	15
2	5	25	2.5	5	15	3.0	4	15

SREFNUM	MAXKEY	SANGL1_F	SANGL2_F	ANSIO
1	0	TRDIPDIR	TRDIP	1
2	0	TRDIPDIR	TRDIP	1

14.4.9 Model Validation and Sensitivity

The following steps were taken to validate the estimated model:

- Walk though of the model and the drillhole data in section and plan;
- Summary simple statistics;
- Comparison of the estimates made with and without Dynamic Anisotropy:
  - Estimates of the sub-block model were made with and without use of Dynamic Anisotropy;
  - The results were very similar;
  - Use of Dynamic Anisotropy resulted in slightly higher tonnes and gold grades in the unmined area below 125 mRL. This shows that Dynamic Anisotropy improves the estimates in Table 14-6. The percentage differences are shown as these are not Mineral Resources; and
  - A cut-off of 1.2% eCu was used.

Table 14-6: Comparison of estimates made with / without Dynamic Anisotropy

Effect of Dynamic Anisotropy eCu>=1.2%			
DOM	**% Diff T**	**% Diff Copper**	**% Diff Gold**
1	2.6	-0.2	2.7
2	7.8	-2.2	7.6
Total	**2.6**	**-0.2**	**2.8**

- Comparison of the estimates with/without parent block estimation:
  - The sub-block model was estimated with and without parent block estimation. In parent block estimation, all the sub-blocks in a primary block receive the same grade. The tonnes were slightly higher for the sub-block estimation Table 14-7, but grades were unchanged. A cut-off of 1.2% eCu was used. Percentage differences are shown as these are not Mineral Resources; and
  - Use of sub-block estimation has not significantly changed the results.

Table 14-7: Comparison of sub-block and parent block estimates

Effect of Sub-Block Estimates eCu>=1.2%			
DOM	**% Diff T**	**% Diff Copper**	**% Diff Gold**
1	0.2	0.4	0.1
2	4.8	1.	-1.7
Total	**0.2**	**0.3**	**0.1**

- Comparison of Kriged, ID^3 and NN estimates:
  - The estimates made by Kriging, ID^3 and NN were compared above a zero cut-off in Table 14-8 for the unmined area below 125 mRL. The results are shown as percentage differences in grade as these are not classified Mineral Resources;
  - At a zero cut-off, the tonnages were the same for all methods;
  - The grades were almost the same by all methods; and
  - The comparison needs to be done above a zero cut-off because the NN estimate has a different variance (higher).

Table 14-8: Comparison of estimation methods

Estimation Differences below 125 mRL (zero cut-offs)				
DOM	**% Difference to Kriged**			
	Copper		**Gold**	
	ID^3	**NN**	**ID^3**	**NN**
1	-1.8	10.5	-1.2	1.6
2	0.0	2.0	2.2	6.7
Total	**-1.4**	**1.7**	**-0.4**	**2.7**

- The Kriged and ID^3 copper estimates in the unmined area are compared as a function of copper cut-off grade in Figure 14-14; and
- Tonnes are shown as percentages of the total tonnage above a zero cut-off grade as these are not classified Mineral Resources:
  - The results are very similar, except at very high cut-offs where the ID^3 grade is higher, as expected.

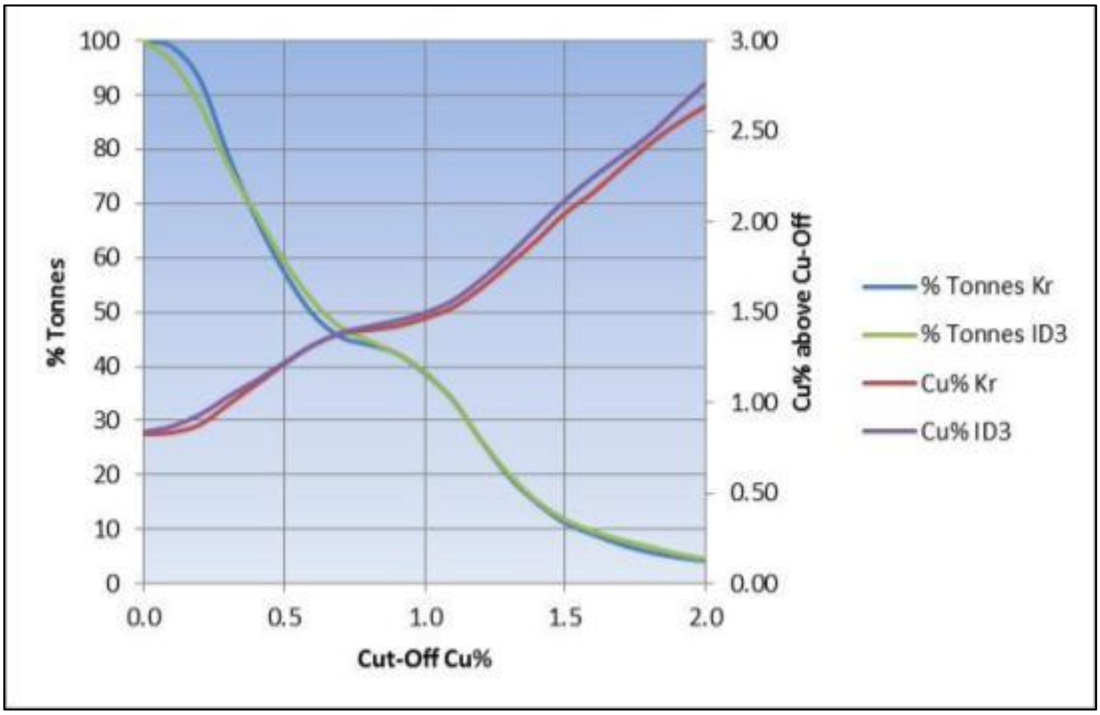


Figure 14-14: Grade-tonne comparison of copper Kriged and ID^3 (tonnes and grade in %)

- Swath Plots:
  - The estimated block grades were compared with the block-average top-cut drillhole composite grades using swath plots in Figure 14-15.

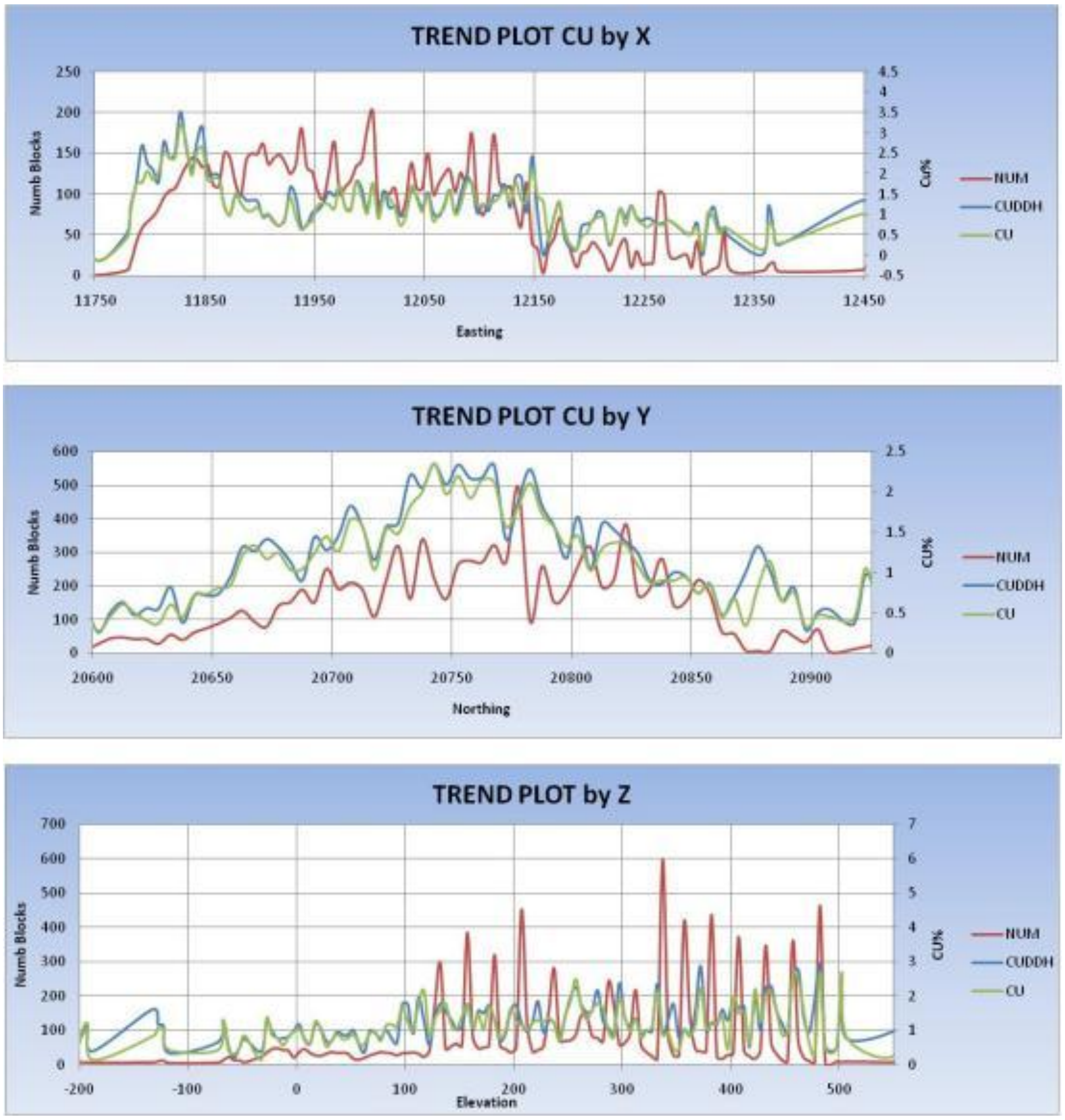


Figure 14-15: Swath plot % copper (copper is model grade, CUDDH is drillhole grade)

- Test of local variability and bias:
  - The drillhole composites were top-cut then averaged into regular blocks and merged with a regularised Mineral Resource model. The grades were compared using a scatterplot in Figure 14-16. Both the grades and the correlation coefficient were satisfactory, indicating a lack of local bias and the model has the correct amount of local variability.

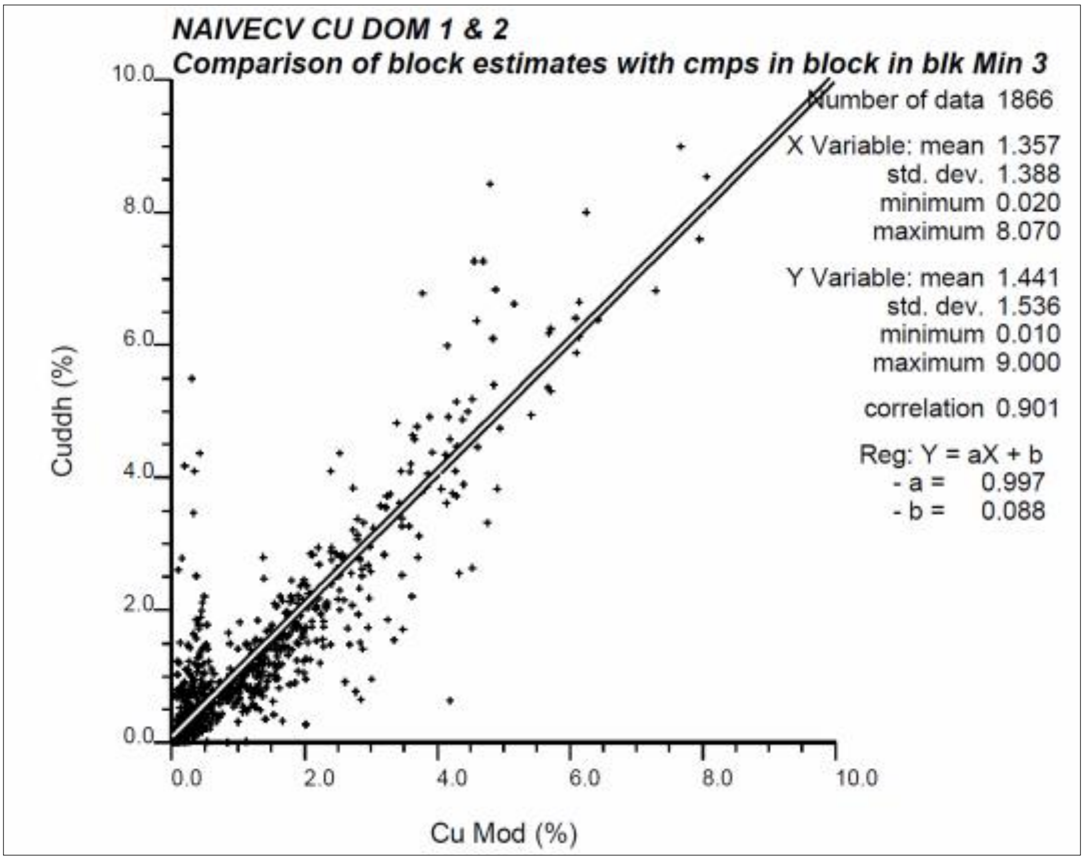


Figure 14-16: Comparison of copper estimates with composites in blocks

- Comparison of the variability of the model with that calculated from the composites:
  - The variability of model for the ore zone was compared with the theoretical variability calculated from the composites and the declustering weights using the Indirect Log-Normal Correction. This uses the variogram model and the block dimensions to adjust grades and declustering weights of the composites, to that of a model. The expected CV for the ore zone model below 125 mRL is 0.52. The estimated model will have about the right amount of smoothing if its CV is 85% of this value or 0.44. In practice, the CV of the model of the ore zone below 125 mRL is 0.37, indicating that the model is slightly over-smoothed. This is a global test and the model contains appreciable Indicated and Inferred Mineral Resource blocks, which would be expected to be more smoothed than Measured Mineral Resource blocks. There are also only 560 ore zone composites below 125 mRL from which to calculate the expected CV of blocks.

14.4.10 Removal of Mined out areas

Much of the 1SS Zone has already been mined. As the stope wireframes imported from Surpac had errors when verified in Datamine, they could not be directly used to deplete the new model. In addition, any mineralisation remaining unmined between stopes is unlikely to ever be mined, and therefore cannot be included in the Mineral Resources.

The technique used to deplete the Mineral Resource model was to digitise a cut-out string in plan view that enclosed the mined areas. Blocks were selected in 2D inside and outside this string. This cookie-cutter approach allows minor between-stope mineralisation to be excluded. Figure 14-17 shows the previous stopes in plan view along with the cut-out string. A MINED field was added to the model. A value of 1 was assigned if inside the cut-out string, and 0 if outside. Mineral Resources can be further limited to be below 125 mRL (completely below any previous mining).

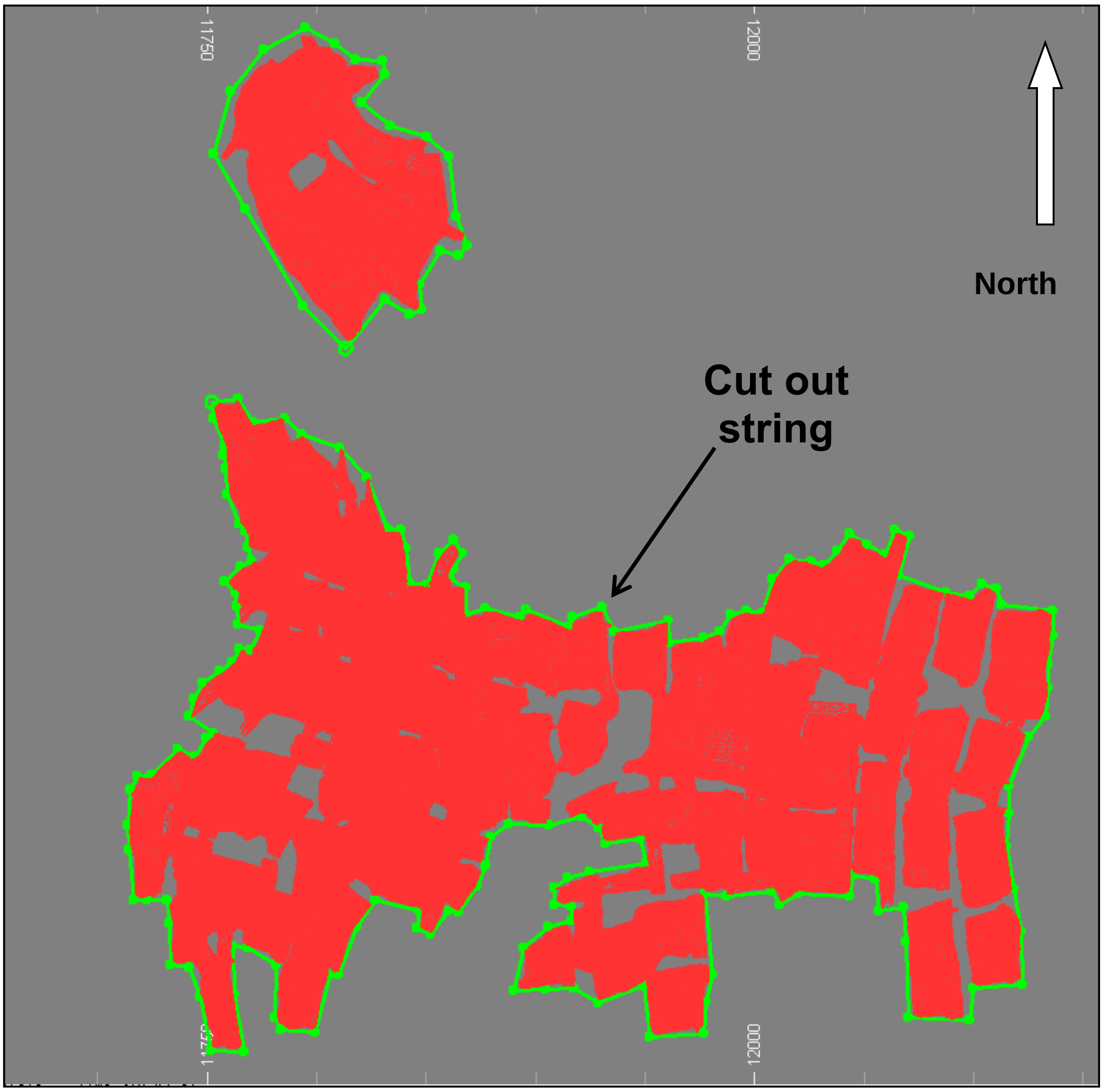


Figure 14-17: Plan view of mined stopes and the cut-out string

Source: LMRC, April 2011

14.4.11 Mineral Resource Classification

Block model quantities and grade estimates for the 1SS Project at Osborne were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by R W Lewis, FAusIMM (No. 100799), an appropriate independent Qualified Person for the purpose of NI 43-101.

Mineral Resource classification is typically a subjective concept, industry best practices suggest that Mineral Resource classification should consider both the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar Mineral Resource classification.

LMRC is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced approximately 20 m apart.

Classification was applied using Kriging variance. A special estimation run was made that did not distinguish the ore and dilution zones.

The search was set to the maximum used in the main estimation when SVOL=3, and a nominal variogram model was used. The Kriging variance break points used to separate the classes of Resources were chosen by reference to the drillhole spacing and the SVOL pass used. Figure 14-18 shows the classification applied to a regular block model for the purpose of illustration. The Mineral Resources are based on the sub-block model. The Measured Mineral Resources have a CLASS value of 1; Indicated Mineral Resources have CLASS of 2 and Inferred Mineral Resources are CLASS 3.

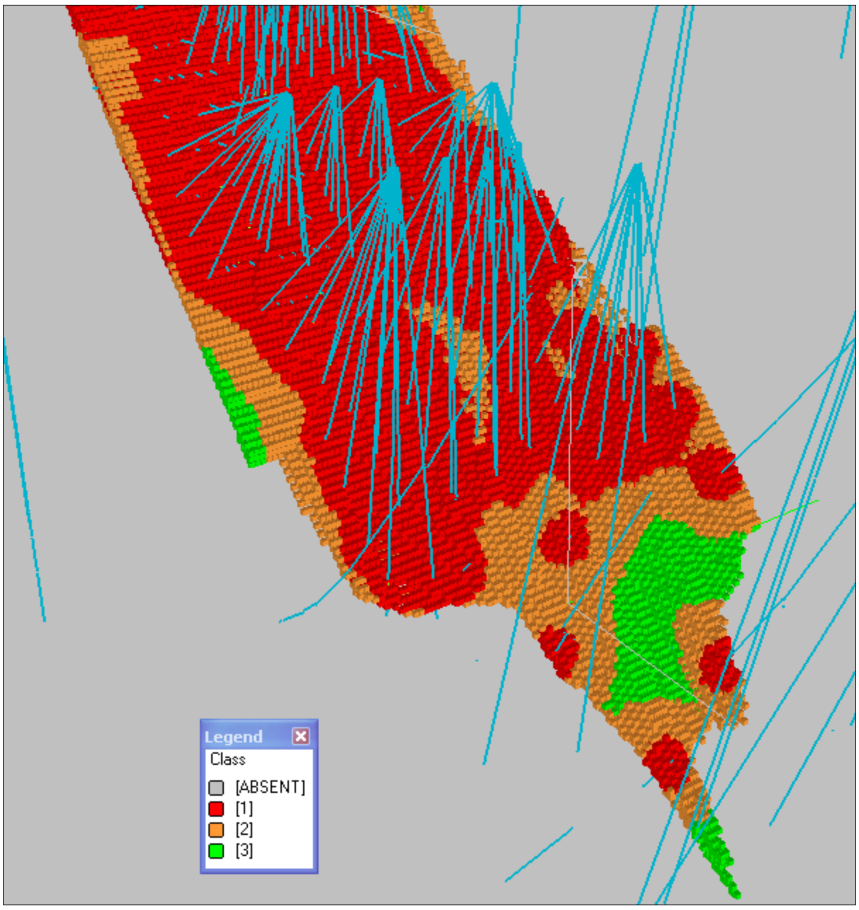


Figure 14-18: Classified Regular Block Model

14.4.12 1SS Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a Mineral Resource as:

> *"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilised organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".*

The classified Mineral Resources for 1SS are shown in Table 14-9. The Mineral Resources below 125 mRL are completely below any previous mining.

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-9: 1SS Classified Mineral Resources below 125 mRL (limit of previous mining)

2011 Model >=1.2% Copper (below 125 mRL)						
Category	**Quantity (Mt)**	**Grade**			**Metal**	
		eCu (%)	**Copper (%)**	**Gold (g/t)**	**Copper (000't)**	**Gold (000' oz)**
Measured	2.1	2.1	1.5	0.9	31.7	57.5
Indicated	0.8	1.7	1.2	0.9	9.7	22.1
Measured + Indicated	**2.9**	**2.0**	**1.4**	**0.9**	**41.5**	**79.6**
Inferred	0.5	1.7	1.2	0.9	5.6	13.4

1 eCu = copper (%) + gold (g/t) x 0.6.

2 The Mineral Resource Estimate is effective as at 27 October 2011.

3 The Mineral Resource Estimates have been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

4 Some totals may not add due to the effects of rounding.

14.4.13 Grade Sensitivity Analysis

The Mineral Resources of the 1SS Project at Osborne are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model unmined quantities and grade of blocks classified as Measured and Indicated are presented in Table 14-10 at different cut-off grades.

The figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. Figure 14-19 presents this sensitivity as grade tonnage curves.

Table 14-10: 1SS Global Block Model quantities and grade estimates

Global Model Quantities and Grades				
Unmined blocks classified as Measured and Indicated				
Cut-off eCu%	**Mt**	**eCu%**	**Copper%**	**Gold g/t**
0.0	6.7	1.1	0.8	0.8
0.1	6.7	1.1	0.8	0.5
0.2	6.4	1.2	0.9	0.5
0.3	5.8	1.3	0.9	0.5
0.4	5.2	1.4	1.0	0.6
0.5	4.7	1.5	1.1	0.6
0.6	4.2	1.6	1.2	0.7
0.7	3.7	1.7	1.3	0.7
0.8	3.3	1.8	1.3	0.8
0.9	3.1	1.9	1.4	0.8
1.0	3.0	1.9	1.4	0.9
1.1	2.9	2.0	1.4	0.9
1.2	2.9	2.0	1.4	0.9
1.3	2.8	2.0	1.5	0.9
1.4	2.7	2.0	1.5	0.9
1.5	2.5	2.1	1.5	0.9
1.6	2.2	2.1	1.6	0.9
1.7	1.8	2.2	1.7	1.0
1.8	1.4	2.4	1.8	1.0
1.9	1.1	2.5	1.9	1.0
2.0	0.9	2.6	2.0	1.0

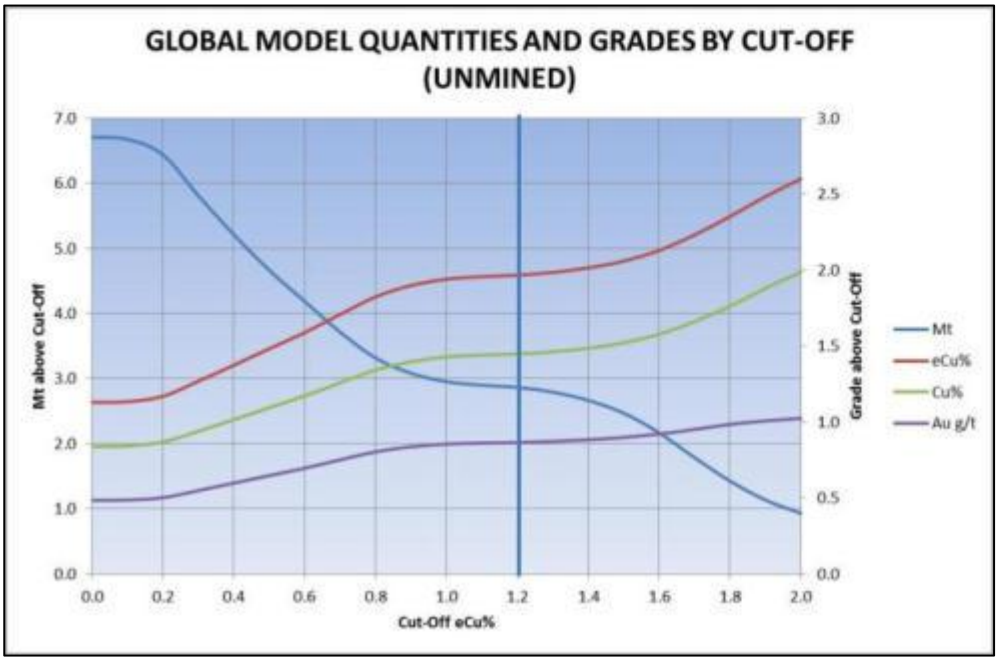


Figure 14-19: Grade-tonnage curves for 1SS

14.4.14 Previous Mineral Resource Estimates

The published Osborne underground Mineral Resources are shown in Table 14-11 (Ivanhoe, 2010). This table includes the 1S and 1M Mineral Resources with the 1SS Mineral Resources. The total is very low, probably because the Mineral Resource is constrained to being above the lower limit of mine development.

The 2009 model 1SS unpublished Mineral Resources below 125 mRL are shown in Table 14-12.

The 2011 Mineral Resources are shown in Table 14-13. Tonnes and grade are quite similar, though the 2009 Mineral Resources were estimated using different ore zone wireframes. The main difference is in the amount of Measured Mineral Resources.

Table 14-11: Published Mineral Resources 1S, 1SS, 1M

Ivanhoe Published Mineral Resources 2010						
		Grade			Metal	
Category	Quantity (Mt)	eCu (%)	Copper (%)	Gold (g/t)	Copper (000't)	Gold (000' oz)
Measured	0.1	3.1	2.5	1.0	2.5	3.2
Indicated	0.1	2.9	2.3	1.0	2.3	3.2
Measured + Indicated	**0.2**	**3.0**	**2.4**	**1.0**	**4.8**	**6.4**
Inferred	0.3	2.7	2.2	0.9	2.7	74.0

(eCu=copper % + gold g/t x 0.6)

Table 14-12: 2009 1SS Mineral Resources above a cut-off of 1.2 % eCu and below 125 mRL

2009 Model eCu >= 1.2% eCu (below mRL 125)						
		Grade			Metal	
Category	Quantity (Mt)	eCu (%)	Copper (%)	Gold (g/t)	Copper (000't)	Gold (000' oz)
Measured	0.7	1.9	1.4	0.8	10.6	19.8
Indicated	0.7	1.9	1.4	0.8	10.6	20.0
Measured + Indicated	**1.5**	**1.9**	**1.4**	**0.8**	**21.3**	**39.8**
Inferred	1.8	1.8	1.3	0.8	23.3	48.0

(eCu=copper % + gold g/t x 0.6)

Table 14-13: 2011 1SS Mineral Resources above a cut-off of 1.2 % eCu and below 125 mRL

2011 Model eCu >= 1.2% eCu (below 125 mRL)						
		Grade			Metal	
Category	Quantity (Mt)	eCu (%)	Copper (%)	Gold (g/t)	Copper (000't)	Gold (000' oz)
Measured	2.1	2.1	1.5	0.9	31.7	57.5
Indicated	0.8	1.7	1.2	0.9	9.7	22.1
Measured + Indicated	**2.9**	**2.0**	**1.4**	**0.9**	**41.5**	**79.6**
Inferred	0.5	1.7	1.2	0.9	5.6	13.4

(eCu=copper % + gold g/t x 0.6)

14.4.15 Recommendations for Conversion of Mineral Resources into Mineral Reserves

There is currently no development below 125 mRL; this will be necessary before any of the Mineral Resources are mineable. The grade of the Mineral Resources decreases with depth, so it will be important to increase the amount of drilling in the lower parts of the 1SS Zone.

14.5 Mineral Resource Estimation Open Pit

14.5.1 Mineral Resource Database Open Pit

The Mineral Resource database for the open pit estimation contains 88,173 samples with informed copper grades. Of these, 59,147 are actually in the ore zones or the surrounding envelope. Most of the data were from diamond core drilling, done from surface and underground. Some RC drilling was done, especially during pre-mine exploration, but also as pre-collars for later core holes. A comparison of the core and RC data is included in a later section. The data to be used for the open pit estimation were selected within a box in Table 14-14.

Table 14-14: Data selection box for Open Pit work, 2011

	Min	Max	Size	Number
X	10900	12000	5	220
Y	20400	22700	5	460
Z	800	1300	5	100

There were some minor problems identified with the database:

- RC holes drilled pre-mine were not in the database. It was possible to add these from previous electronic drill logs (“geologs”); and
- There were 22 holes lacking collar coordinates.

Table 14-15: Drillholes without collar coordinates

UG1S1143A	SU1S1011
260S002	SU1S1041
310S007	TEST0000
335S003b	UG3S0030
380S008	UG3S0031
OSNQ0034	UG3S0032
OSNQ0035	UG3S0034
OSNQ0047	UG3S0035
OSNQ0063	UG3S0036
UG1S1465A	UG3S0037
UG1S1545	UG3S0038

- Some drillholes lacked any assays, including:
  - Holes drilled for metallurgical testing; and
  - Holes without any visible mineralisation.

These holes are shown in Table 14-16.

Table 14-16: Open Pit Area drillholes lacking assays

BHID	XCOLLAR	YCOLLAR	ZCOLLAR
EXP0030	11591.40	21394.80	1010.20
OSB0022	11875.50	20755.10	1273.20
OSVM0001	11308.30	21375.30	1284.90
OSVM0002	11309.90	21378.90	1284.90
OSVM0003	11274.30	21484.80	1285.10
OSVM0004	11275.30	21482.40	1285.10
TTZQ0174	11609.90	21743.70	1274.80
TTZQ0237	11540.10	21176.80	1282.10
TTZQ0249	11526.80	21461.20	1280.60
TTZQ0256	11631.40	21319.70	1283.40
TTZQ0265	11627.40	21389.40	1277.50
TTZQ0276	11189.80	21885.10	1281.20
TTZQ0294	11650.20	21140.80	1280.50
TTZQ0297	11615.00	21000.00	1277.50
TTZQ0308	11620.30	20929.60	1275.80
TTZQ0312	11570.00	20930.00	1275.50
TTZQ0313	11620.20	20859.60	1273.90
TTZQ0319	11819.50	20859.00	1273.60
TTZQ0321	11900.40	20800.20	1274.30
TTZQ0323	11902.80	20800.10	1274.20
TTZQ0325	11809.80	20858.90	1273.60
TTZQ0328	11870.90	20929.80	1275.50
UG1S0109	11717.30	21155.90	931.20
UG2M0177	11414.90	21549.40	1016.80
UG2M0199	11386.40	21234.80	1102.70
UG3E0037	11593.30	21430.20	1017.30

At Osborne, mineralisation is sufficiently visible that not all core needs to be assayed. In general terms, any missing values should therefore be given zero grades.

There are some exceptions to this rule:

- Some holes are completely unassayed as they were drilled for metallurgical samples in Table 14-16;
- Holes drilled as part of a ring pattern (several holes from the same collar position) are usually not all completely assayed, as they are close together; and
- Holes drilled from within one ore zone to intersect another may not be assayed in the sections of the holes that are within the collar ore zone.

These are illustrated in Figure 14-20 and Figure 14-21. Unassayed portions of holes are shown in white.

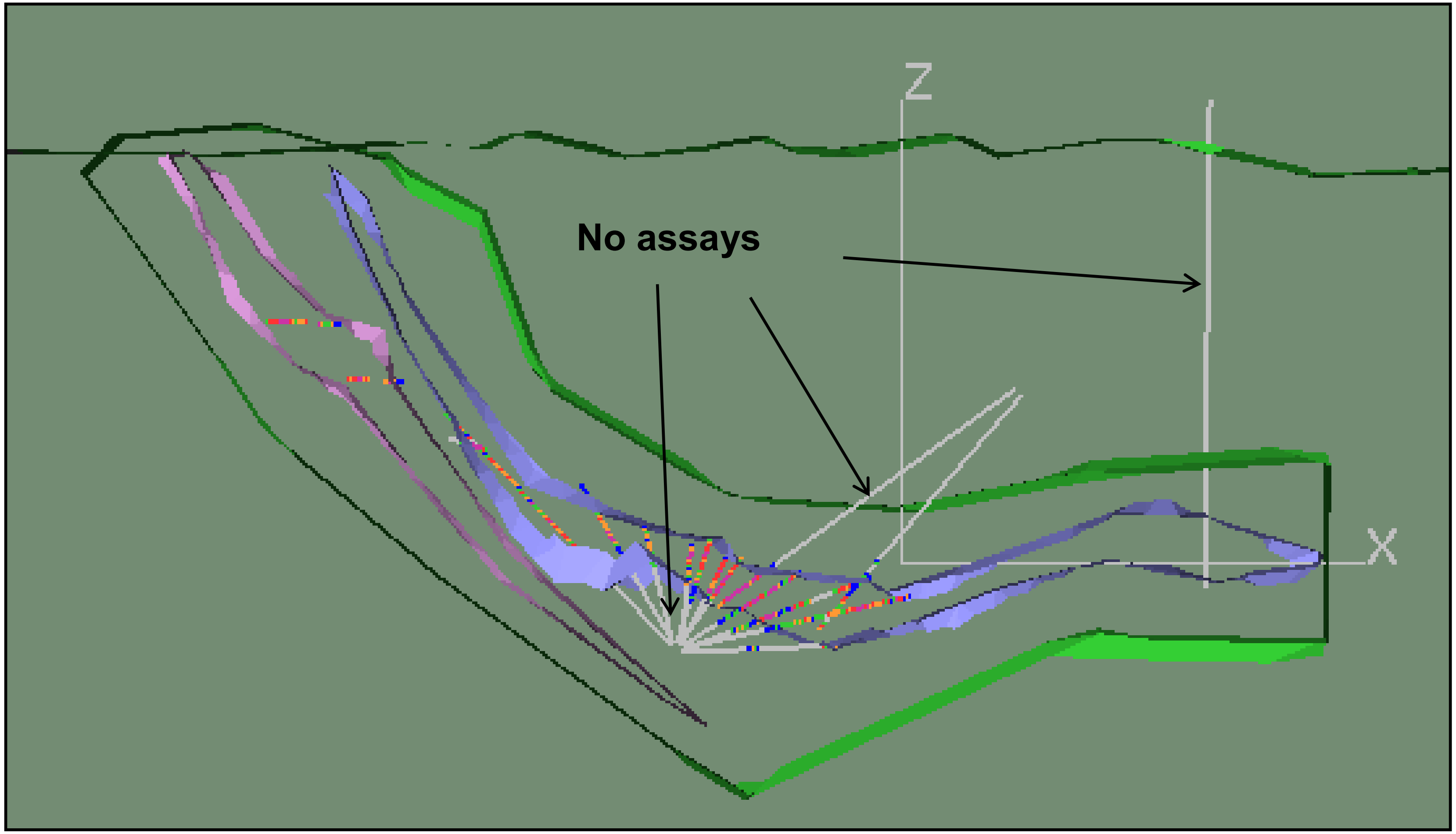


Figure 14-20: Unassayed drillholes and parts of drillholes

Source: LMRC, July 2011

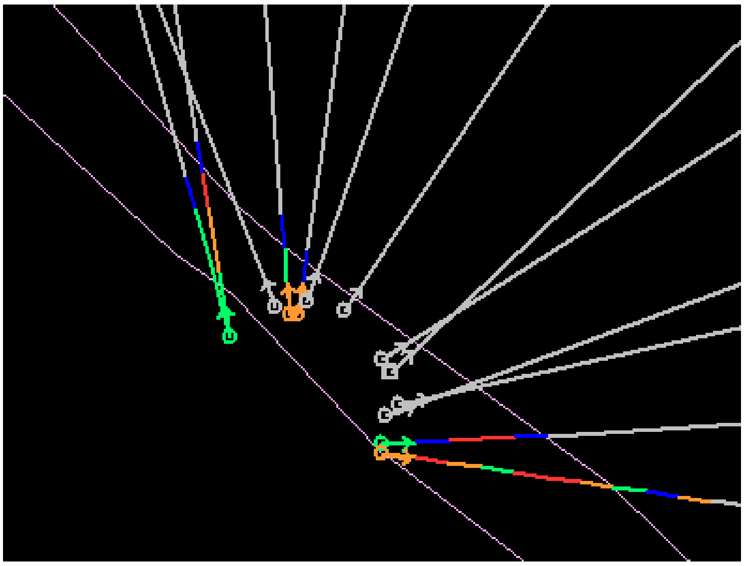


Figure 14-21: Unassayed drillholes in ore zones

Source: LMRC, July 2011

The holes to be excluded from setting "missing" gold and copper (and eCu) assays to zero grade were selected on the basis that they either had no assays at all, or that their hole prefix was one used for ring-drilled holes (many holes radiating from one drill setup). A total of 906 holes were excluded.

The exclusion resulted in slightly higher grades for the main ore zones, but the differences are very small.

There was one problem with the SG values in the database – there is a spike of values at 1.9. This is a problem for older holes in the acQuire database. Any SG values of 1.9 were set to missing.

14.5.2 Surfaces

The open pit area has several significant surfaces that are important for Mineral Resource modelling:

- The current topography includes the partly back-filled open pit mined in the mid-90s;
- The pit surface "as-mined" (before back-filling) is important to distinguish the fill material from in situ rock;
- The Mesozoic surface (approximately 40 m depth) is the upper limit of mineralisation; and
- The base of total oxidation and the base of partial oxidation are important metallurgically.

It was necessary to extend the Mesozoic and oxidation surfaces using WFTREND to provide complete coverage over the model area. The pre-mine drill logs had specific flags for the depths of these surfaces. Information was also obtained from the recent open pit holes. Figure 14-22 shows the base of Mesozoic surface after extension.

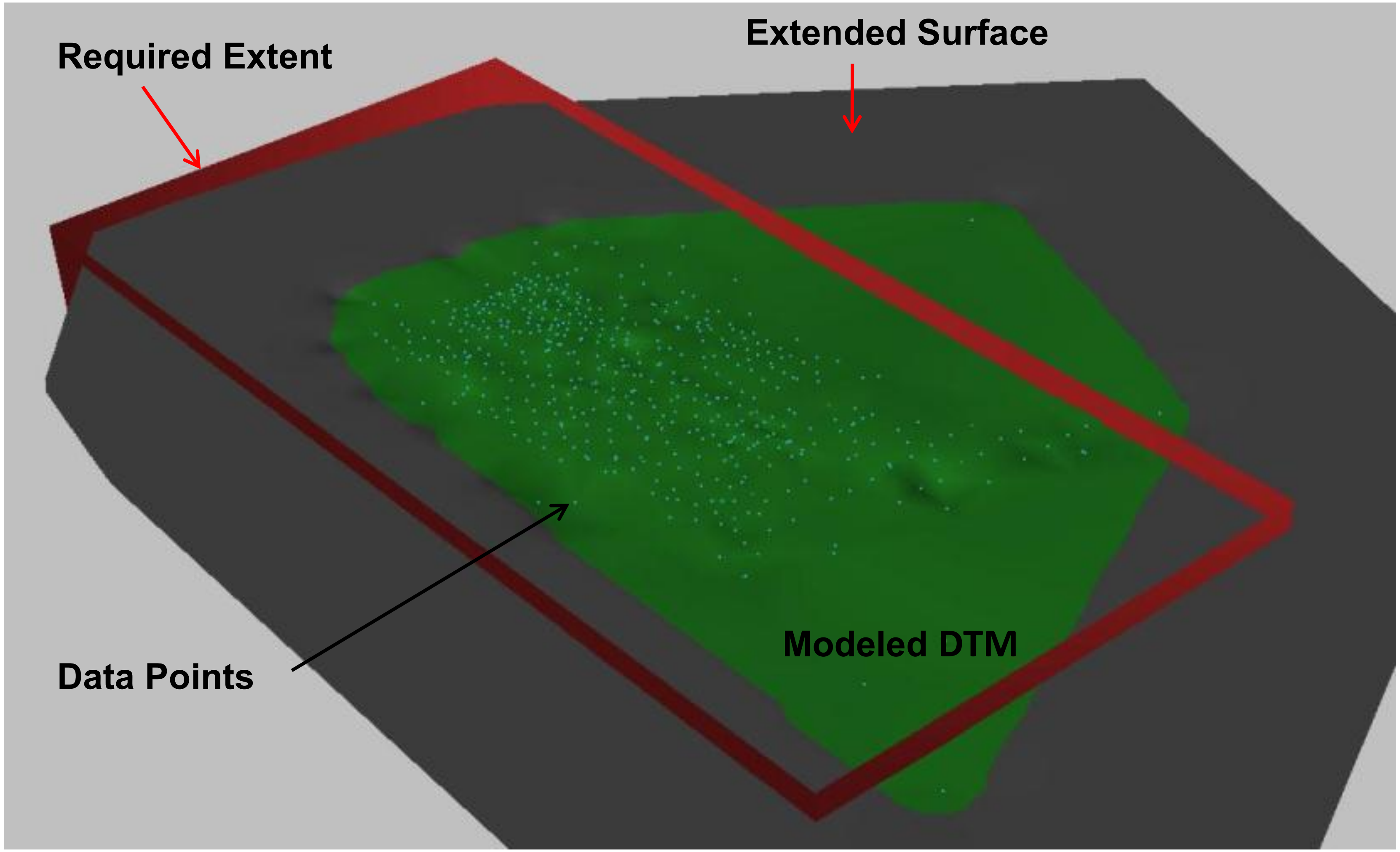


Figure 14-22: Final base of Mesozoic surface

Source: LMRC, July 2011

The surfaces were used to add a SURFACE field to the drillhole data. The wireframe “base” names and the SURFACE codes applied are listed in Table 14-17. The tags were applied in priority order using the table (the table is ordered from highest to lowest priority). This prevented there being any problems with overlapping surfaces. All material below the pox11 surface received a default SURFACE code of 4.

Table 14-17: Tagging by SURFACE

SURFACE	
BASE	**CODE**
ostopo	0
mez11_2	1
topx11	2
pox11	3

A separate PIT field was also added. The back fill in the Osborne pit received a PIT value of 1. All material below the pit received a default code of 2 in Table 14-18.

Table 14-18: Tagging by PIT

PIT	
BASE	**CD**
topo_pit	0
open_pit	1

These priority list files were used to tag both drillhole data and block models.

14.5.3 Solid Body Modelling

The mineralisation was modelled as two large domains (HW and FW) plus two very small subsidiary domains. These were enclosed by a mineralised envelope. The FW domain corresponds to the “1m” wireframe modelled previously. The HW domain corresponds to the “2n” and “2m” wireframes used previously. Different names were used to remove any confusion. A mineralised envelope that enclosed the ore domains was also modelled.

An eCu cut-off was used when modelling the domains (gold factor 0.6). The cut-off grade for ore zones was 0.5% eCu and 0.25% eCu for the mineralised envelope. A broad-brush approach was used with grade displayed as drillhole histograms so major trends were visible.

The domains were modelled as strings on W-E sections 20 m apart. The strings were snapped to the drillholes. The strings were wireframed and verified.

The domain wireframes were used to tag the drillhole file using a priority list file creating a DOM field. Any data outside the domains received a default code of DOM=0. The priority list was necessary as the ore domains are islands in the envelope domain. The SPLIT parameter is used during block model construction to allow use of larger blocks in the mineralised envelope. The domains are shown in plan view in Figure 14-23 and in section in Figure 14-24. The domains are very well informed by drilling.

Table 14-19: Domain priority list file

DOM		
BASE	**CD**	**SPLIT**
fw1	1	2
fw	2	2
hw	3	2
hw1	4	2
env	5	1

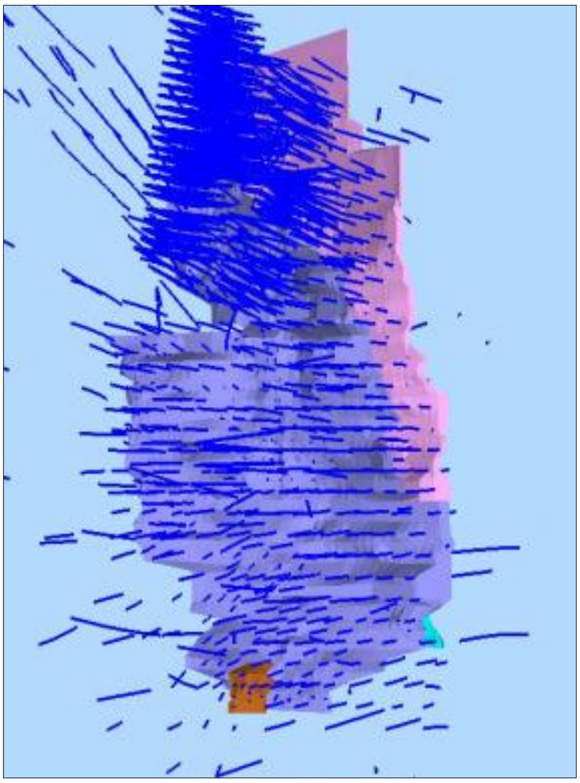


Figure 14-23: Open Pit domains and drillholes

Source: LMRC, July 2011

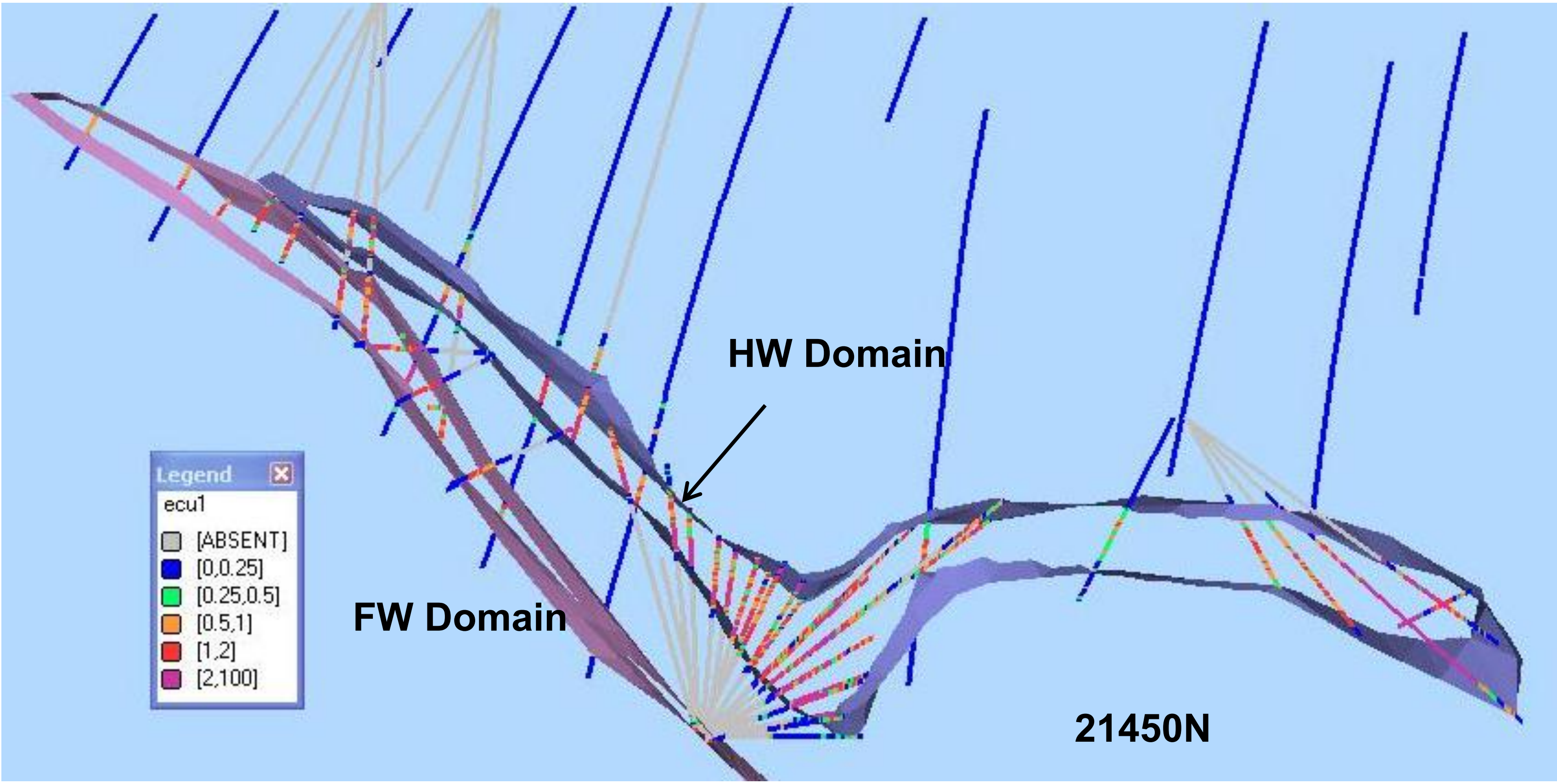


Figure 14-24: FW and HW domains in Section 21450N

Source: LMRC, July 2011

After the drillhole data were tagged by the wireframes (in priority order), they were further processed to adjust the tagging. Any single samples immediately outside the four ore zones that were greater than 0.25% eCu were re-tagged to belong to the main ore zone.

This re-tagging had two purposes:

1 When snapping strings to 1 m samples in drillholes during wireframe construction, it is easy to snap to the midpoint of assays instead of the end of assays. This may result in miss-selection of data by the wireframe; and

2 Although the open pit ore zones have naturally sharp boundaries in most places, there are places where the boundaries are more gradual. In such places, the use of an assay cut-off can result in over-estimation of grade. Adding an additional sample with a grade greater than or equal to half the nominal cut-off removes the over-estimation. This is particularly important with Kriging as samples at the boundaries of domains receive high Kriging weights.

Re-tagging resulted in an increase in samples in the main ore zones:

- The number of new records added to FW1 domain was 1;
- The number of new records added to FW domain was 514;
- The number of new records added to HW domain was 561; and
- The number of new records added to HW1 domain was 13.

The ore domains extend through the base of Mesozoic surface. A new field DOM1 was added based on the DOM and SURFACE fields; all DOM1 values above the base of Mesozoic were set to zero.

14.5.4 Compositing

The most common sample lengths used for assaying at Osborne are 2 m (pre-mine surface holes) and 1 m in later drilling. The data were composited to 1 m intervals with zonal control (field DOM1). Zonal compositing can result in short composites where drillholes leave a domain. It is usual to exclude samples less than half the compositing interval when making estimates. This prevents short samples (especially on boundaries) having equal (or higher in the case of Kriging) weights. Discarding short composites does result in a loss of information.

There are two options available to prevent this happening:

1 The length of all composites in a drillhole intersection through a domain can be adjusted so equal length composites are created. This works best with wide orebodies; and

2 Short composites (less than 0.5 m) can be combined with the neighbouring composites in the zone.

The second method was used as the open pit ore domains are relatively narrow. Any composites >0.5 m are used for estimation anyway, so there was no need to combine composites >=0.5 m <1.0 m. The effect of the adjustment on basic statistics was slight as can be seen in Table 14-20 and Table 14-21.

Table 14-20: Statistics for composite length adjustment (Copper)

Length adjustments of Composites					
Zone	DOM1	Before Adjustment		After Adjustment	
		Nb Copper	Mean Copper %	Nb Copper	Mean Copper %
Bkgnd	0	73568	0.01	73505	0.01
FW1	1	52	0.85	52	0.85
FW	2	11913	1.60	11903	1.60
HW	3	27893	1.46	27867	1.47
HW1	4	189	1.20	187	1.21
Min Env	5	34267	0.11	33991	0.11

Table 14-21: Statistics for composite length adjustment (Gold)

Length adjustments of Composites					
Zone	DOM1	Before Adjustment		After Adjustment	
		Nb Gold	Mean Gold	Nb Gold	Mean Gold
Bkgnd	0	73568	0.01	73505	0.01
FW1	1	52	0.59	52	0.59
FW	2	11913	0.59	11903	0.59
HW	3	27893	0.71	27867	0.71
HW1	4	189	0.90	187	0.91
Min Env	5	34267	0.05	33991	0.05

Much of the open cut area has been mined previously, either by open pit early in the mine life, or later by underground stoping. A PIT and a STOPE field were added to composite file so drillhole data in mined-out areas could be excluded during statistical studies. The PIT field was added by tagging with the surfaces. To add a STOPE field, a horizontal string was used in a cookie-cutter approach in Figure 14-25. This approach excluded drillhole data that are in, above or below stoped areas as well as data in pillars between stopes. It is an approximation used only for statistical analysis of the drillhole data.

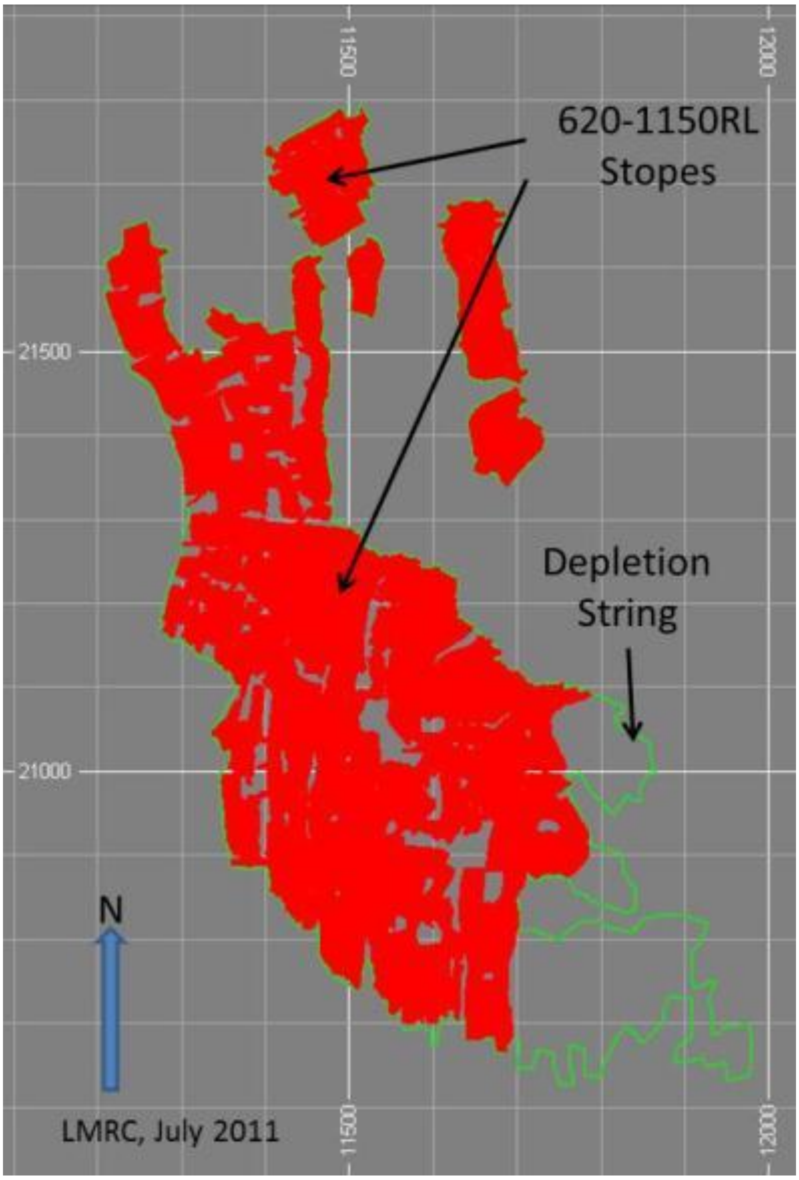


Figure 14-25: Stope depletion string (plan view)

14.5.5 Statistical Analysis and Evaluation of Outliers

The composites were declustered by domain using the polygonal method. This method utilises a geology block model in the declustering. As polygonal declustering can result in high weights where drillholes enter and leave domains, the weights of such samples were adjusted to be the same as that of the next sample in the same hole inside the domain. Weights were further trimmed to remove outliers. As there are fewer SG data than copper and gold data, the SG data were declustered separately.

Figure 14-26 and Figure 14-27 show boxplots of the declustered 1 m composites for copper and gold by domain. The CV is relatively low except for gold in the HW domain. There are high values that will require top-cutting (e.g. the maximum gold grade is 49.55 g/t). The declustering was restricted to data in unstoped areas.

Figure 14-28 shows the boxplot for SG. The average SG values by domain were used later to supply default SG values where the lower number of SG data resulted in lack of estimates for blocks with copper and gold grades. There were no SG data for the FW1 domain.

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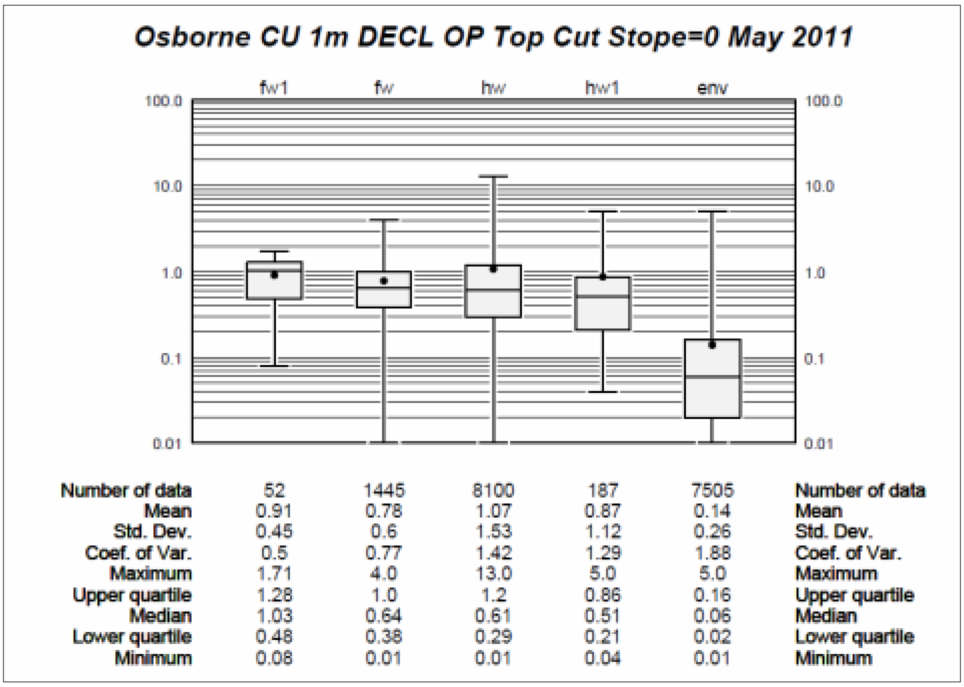


Figure 14-26: Boxplot copper (Declustered)

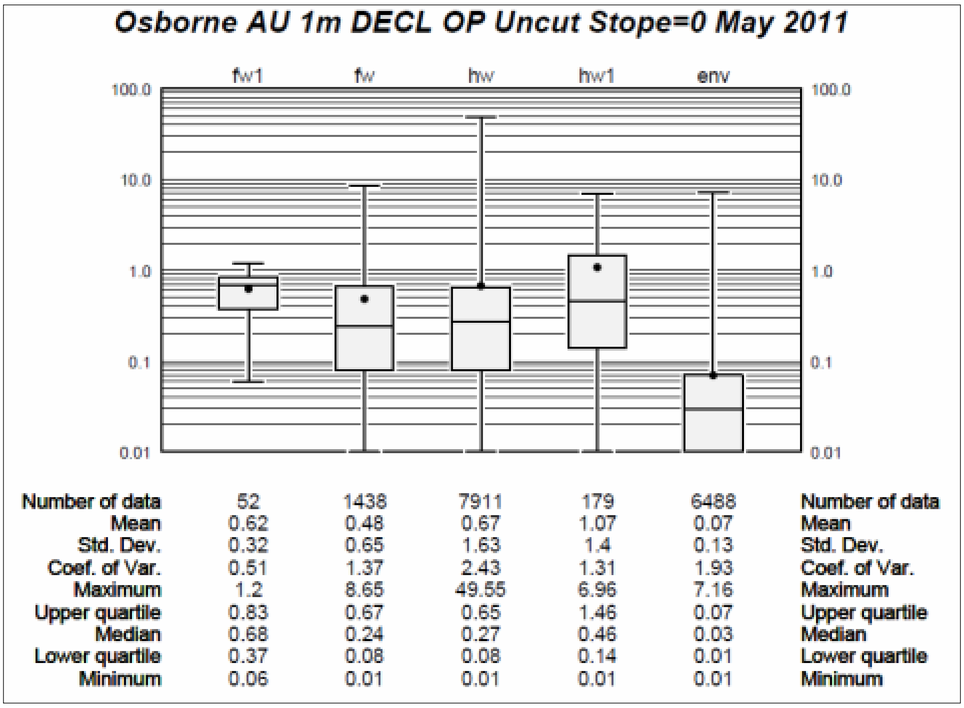
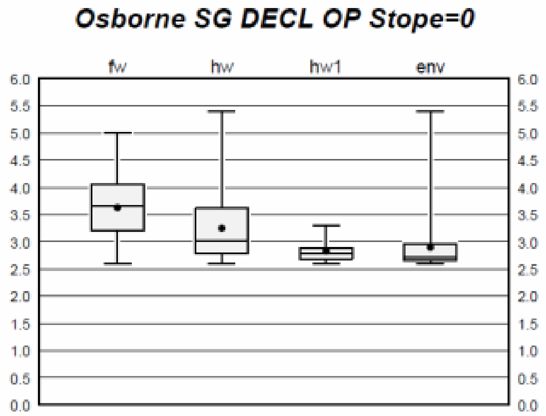


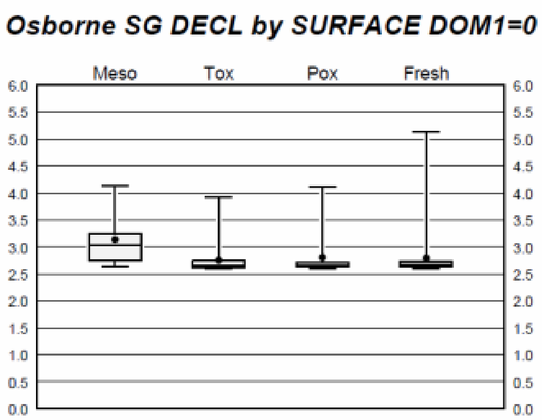
Figure 14-27: Boxplot gold (Declustered)



	fw	hw	hw1	env	
Number of data	871	3943	52	5035	Number of data
Mean	3.63	3.25	2.83	2.9	Mean
Std. Dev.	0.53	0.6	0.19	0.43	Std. Dev.
Coef. of Var.	0.15	0.18	0.07	0.15	Coef. of Var.
Maximum	5.01	5.4	3.3	5.4	Maximum
Upper quartile	4.06	3.62	2.89	2.96	Upper quartile
Median	3.66	3.01	2.79	2.72	Median
Lower quartile	3.2	2.78	2.68	2.66	Lower quartile
Minimum	2.6	2.6	2.6	2.6	Minimum

Figure 14-28: Boxplot SG (Declustered)

SG does not vary much by oxidation type for waste. The mean values in Figure 14-29 were used for default density values for waste in the final model for pit design. There are very few SG values for Mesozoic (10) so the mean value in the Figure 14-29 is probably spurious. A value of 2.76 has been used in the model (for waste).

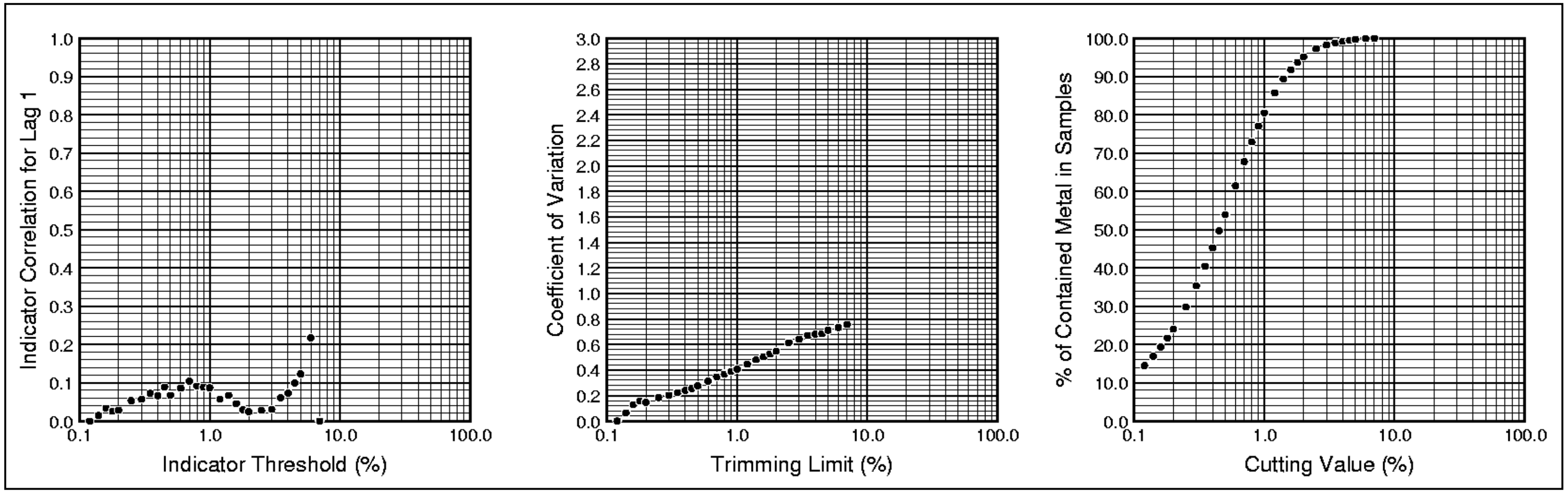


	Meso	Tox	Pox	Fresh	
Number of data	10	55	130	5808	Number of data
Mean	3.14	2.76	2.81	2.8	Mean
Std. Dev.	0.47	0.32	0.39	0.37	Std. Dev.
Coef. of Var.	0.15	0.11	0.14	0.13	Coef. of Var.
Maximum	4.14	3.93	4.12	5.14	Maximum
Upper quartile	3.24	2.75	2.72	2.74	Upper quartile
Median	3.04	2.65	2.65	2.67	Median
Lower quartile	2.75	2.61	2.63	2.64	Lower quartile
Minimum	2.64	2.6	2.6	2.6	Minimum

Figure 14-29: Boxplot SG for Waste

Cutting statistic plots were used along with histogram and probability plots to choose top-cuts. The cutting statistic plots show the relationship between top-cut and CV and top-cut and total metal. The Indicator Threshold plot provides information about continuity as a function of top-cut. Figure 14-30 shows the plots for copper and Figure 14-31 for gold (FW and HW domains). Composites in stopes were not used.

FW



HW

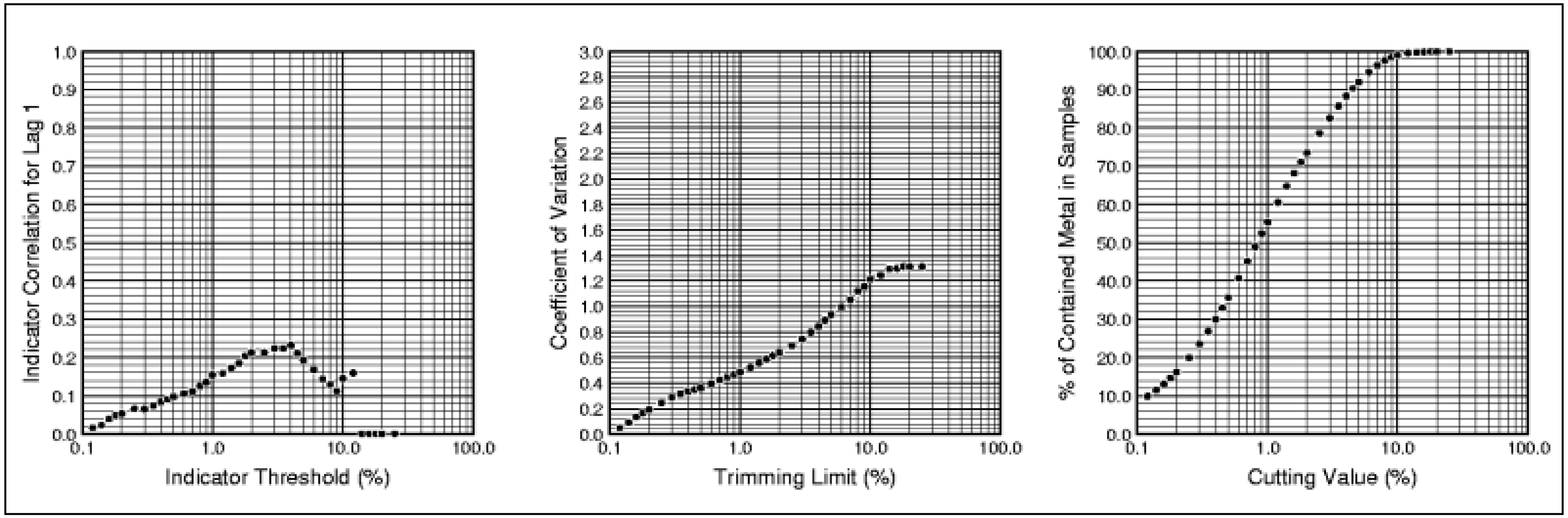
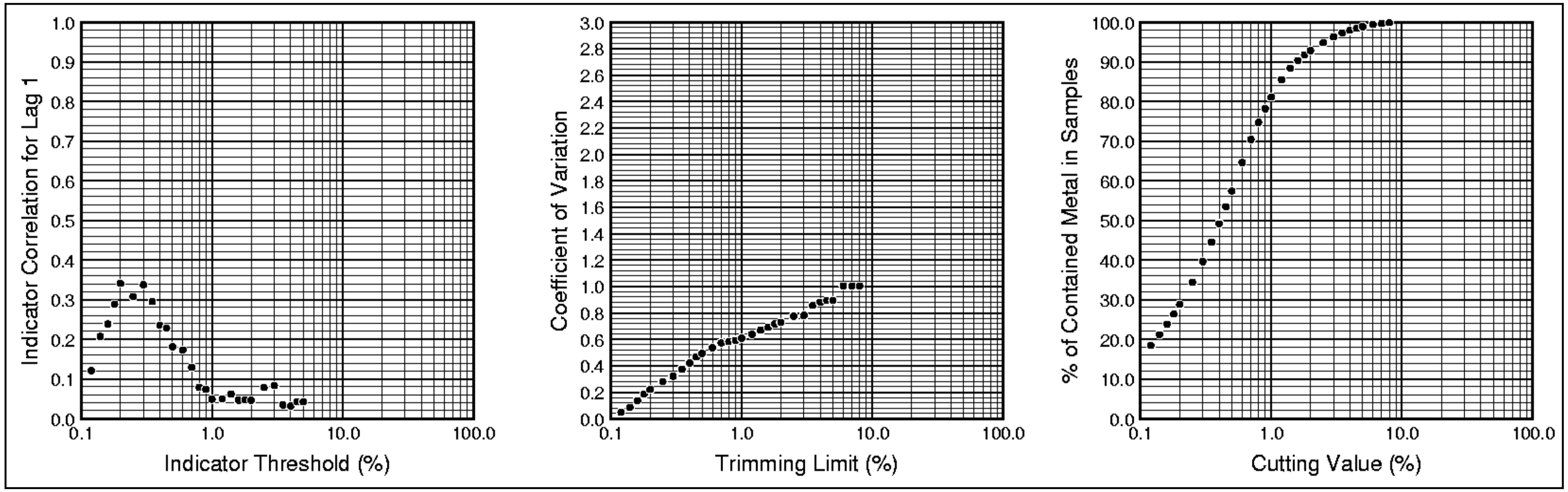


Figure 14-30: Cutting statistic plots copper (FW and HW domains)

FW



HW

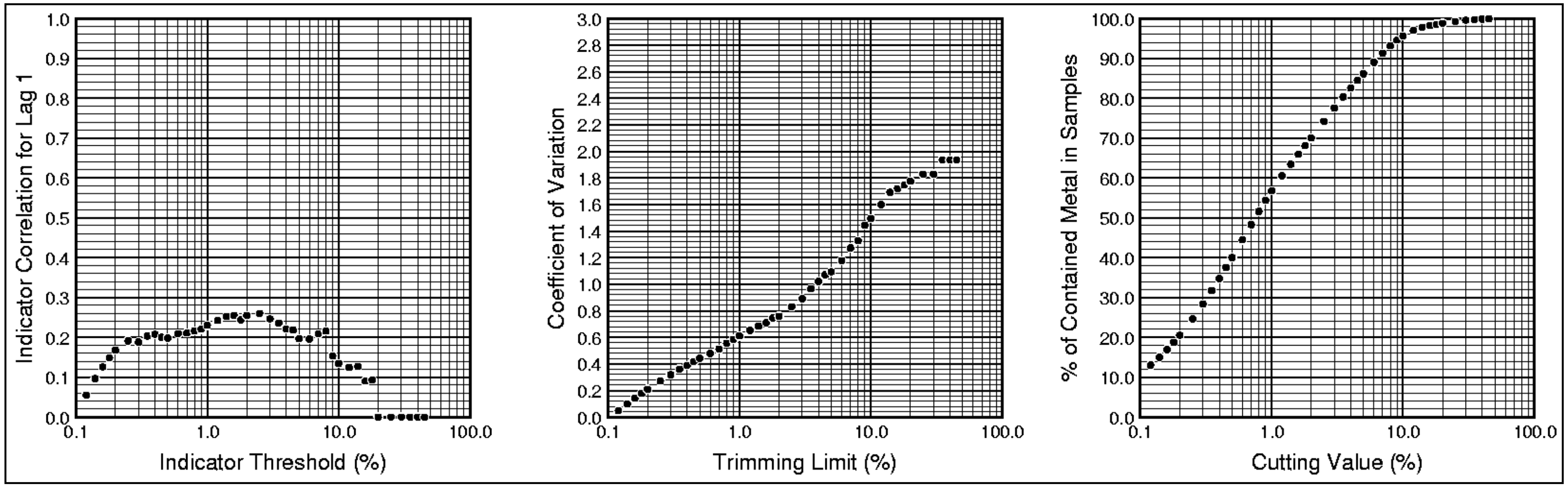


Figure 14-31: Cutting statistic plots gold (FW and HW domains)

Histogram and Probability Plots of the declustered data were done for each domain. The low values (<1.0) were suppressed in the copper and gold histograms to better show the high outliers. The FW and HW domains are shown in Figure 14-32, Figure 14-33 and Figure 14-34.

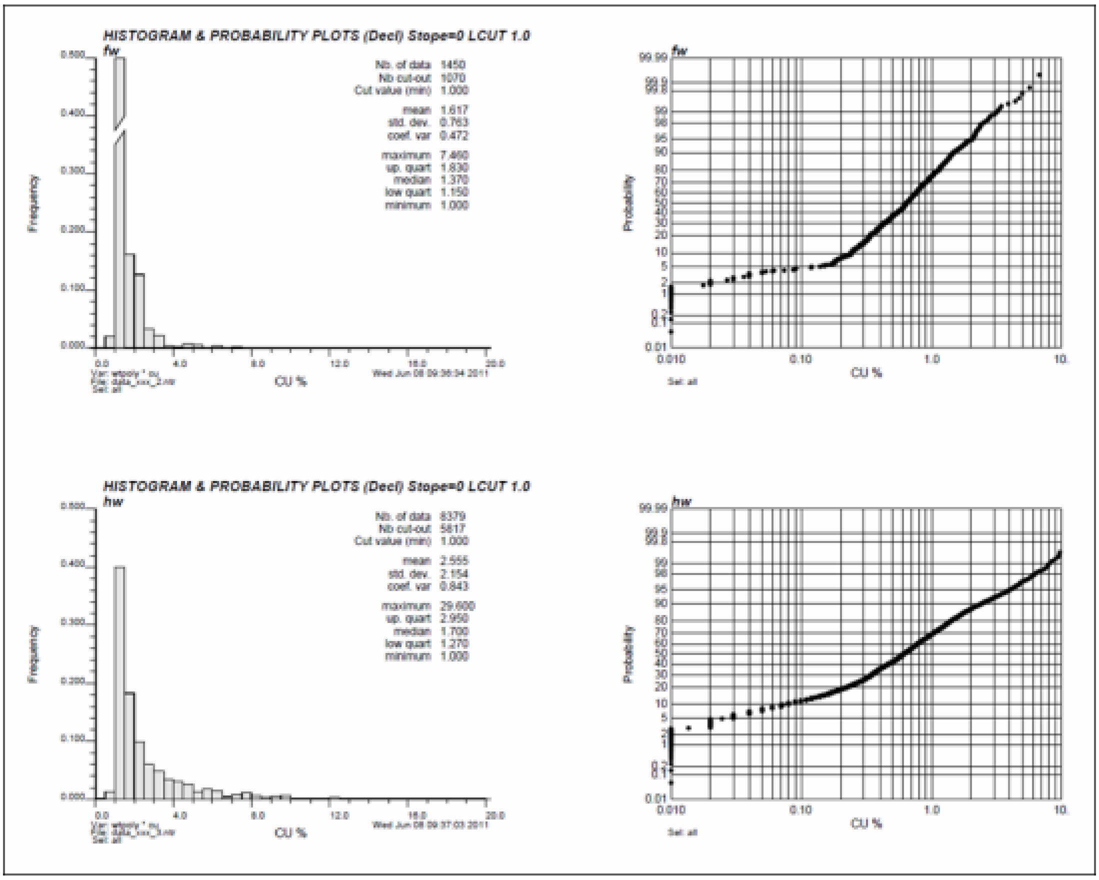


Figure 14-32: Histogram and probability plot copper in the FW and HW domains

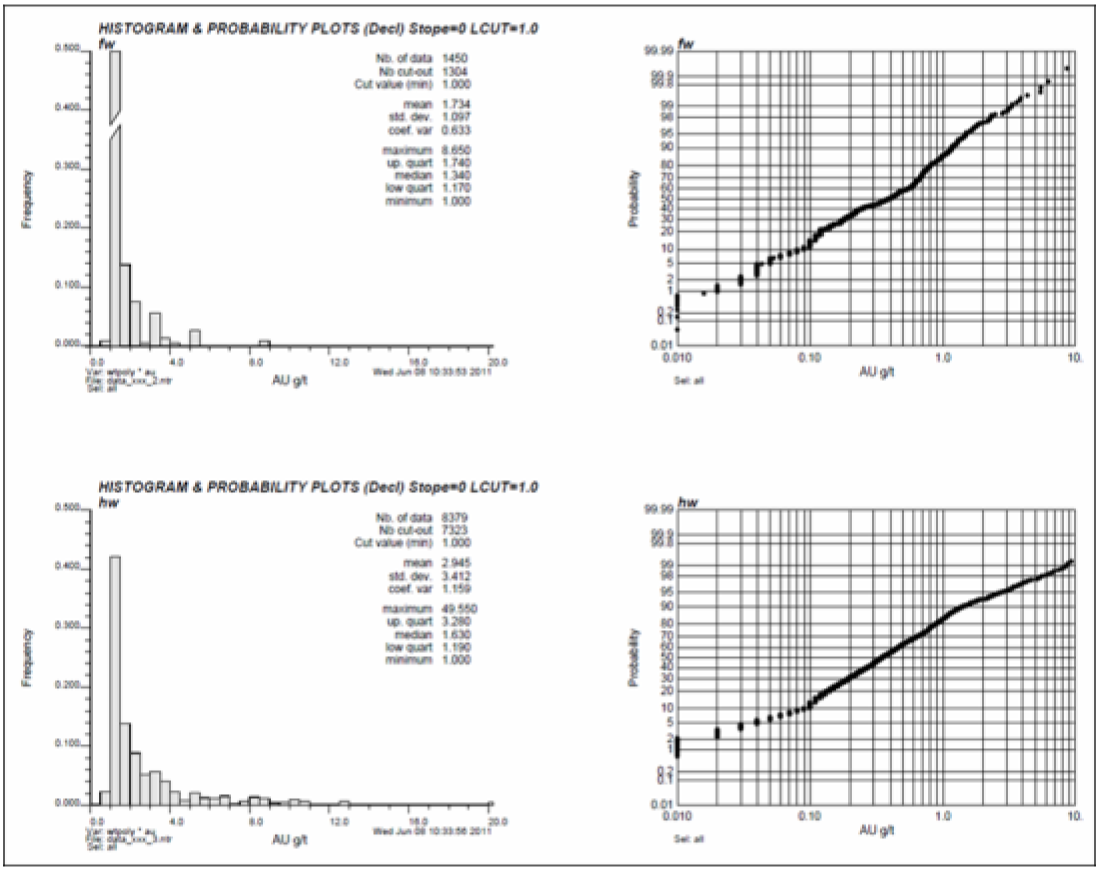


Figure 14-33: Histogram and probability plot gold in the FW and HW domains

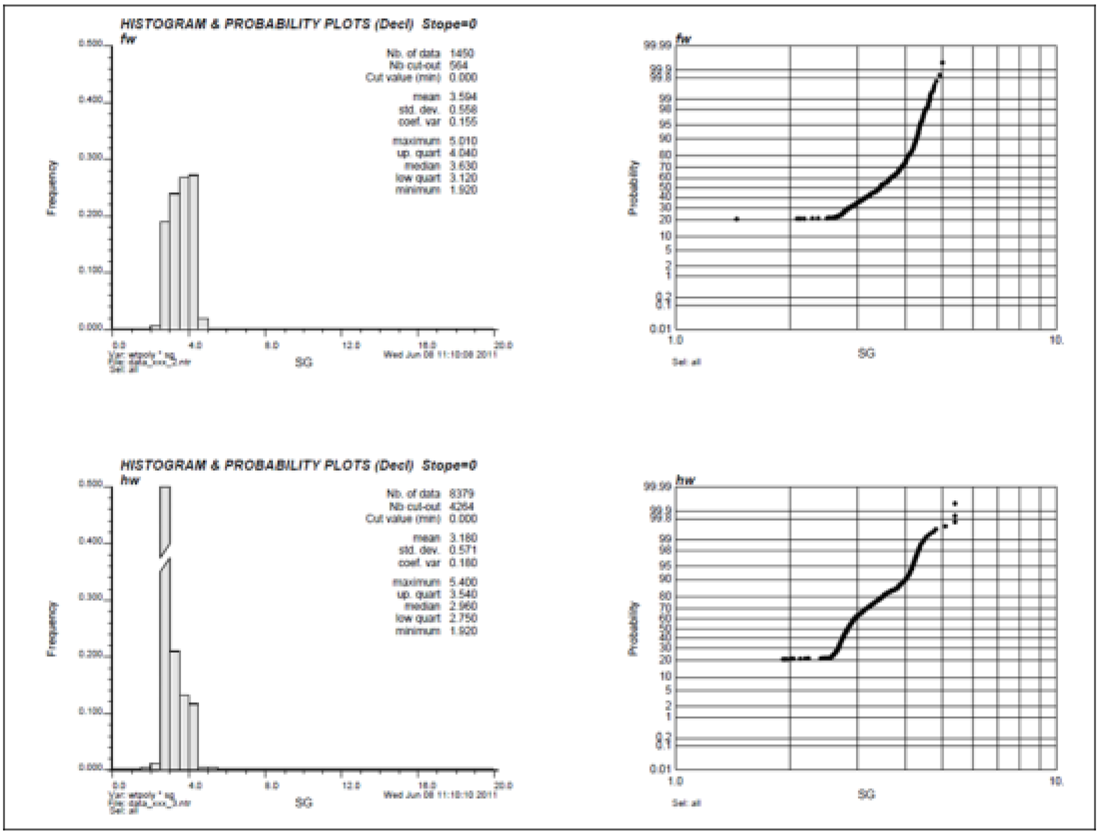


Figure 14-34: Histogram and probability plot SG in the FW and HW domains

The chosen top-cuts are shown in Table 14-22. The number of 1 m composites cut is also shown.

Table 14-22: Top-cuts for copper, gold and SG

Zone	Domain	Copper %	Nb Cut	Gold g/t	Nb Cut	SG g/cm^3	Nb Cut
FW1	1	1.71	0	0.90	6	No data	0
FW	2	4.00	9	5.00	3	5.00	2
HW	3	13.00	14	20.00	9	5.00	25
HW1	4	5.00	4	5.00	5	3.30	0
Env	5	5.00	1	2.00	4	4.40	42

Copper and gold have significant correlation as shown in Figure 14-35. This provides confidence to use the same search filter dimensions for copper and gold during estimation, thereby avoiding having blocks in the model with copper estimates but no gold estimates.

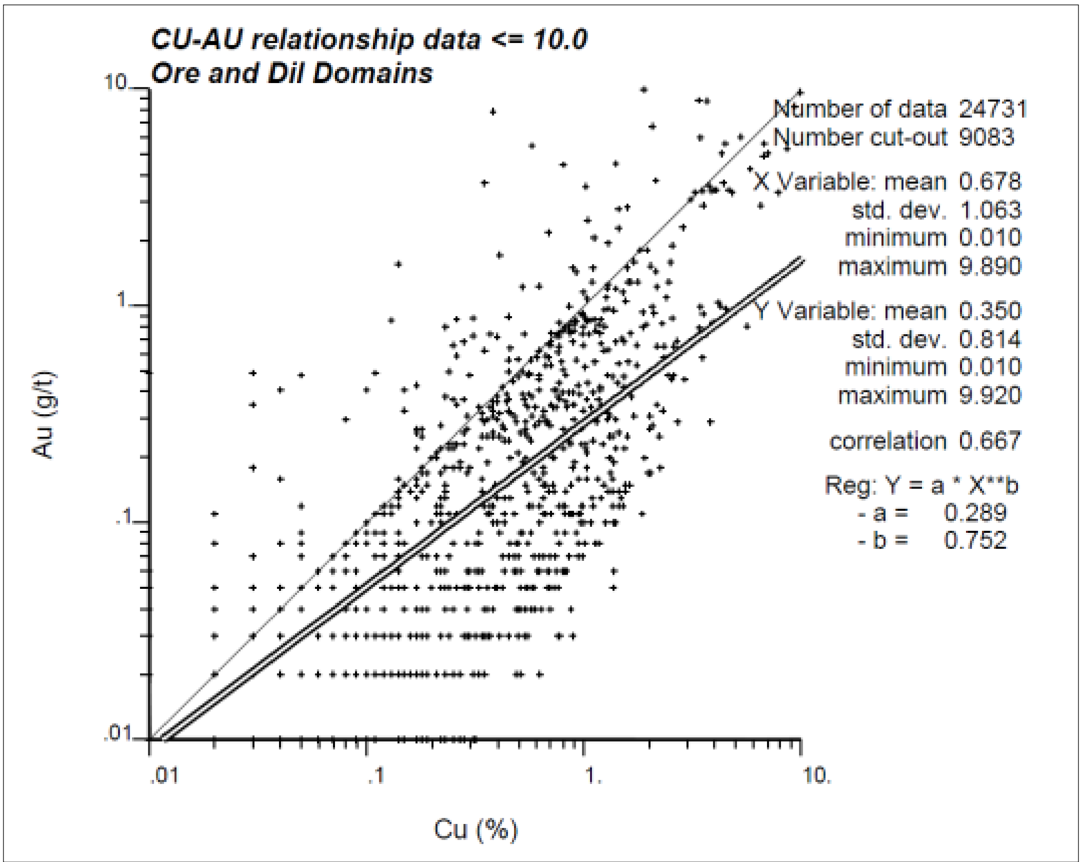


Figure 14-35: Scatterplot between copper and gold in the ore domains and envelope

14.5.6 Comparison of Core and RC Data

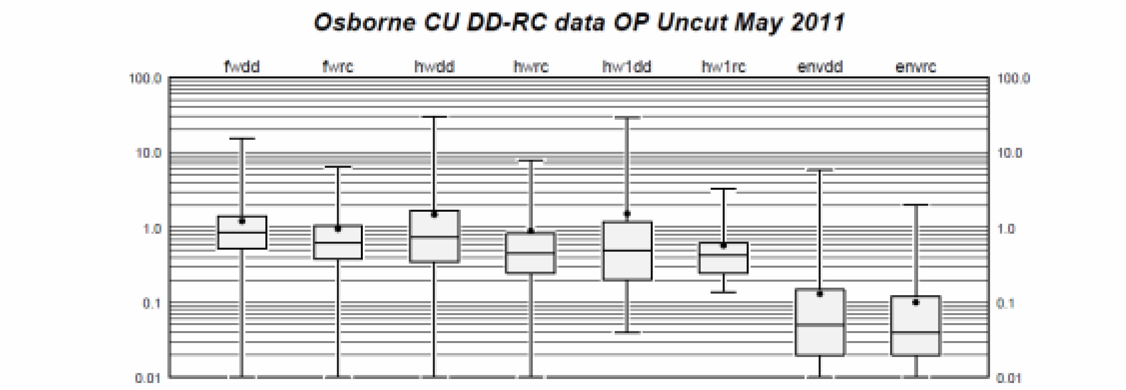
Although Osborne has been drilled mainly using diamond core, there are some RC holes and pre-collars in the Open Pit area. In order to ensure that the RC data was creating bias in the Mineral Resource estimate, a short comparison study was made. The study was limited to holes collared from the surface and further to holes with a TT prefix, as these had pre-collar depth information available in the original Geologs.

There were 374 RC samples within the FW and HW ore zones. There were 6,749 DD samples within these zones. The RC samples make up only 5.5% of the total samples in these zones. The samples are compared by ore zone in Figure 14-36 and Figure 14-37. In general, the DD data, as well as being much more numerous than the RC data, are higher grade.

As a final check, pairs of DD and RC samples in the HW zone no more than 10 m apart were located. These are compared in Figure 14-38. This confirms that the DD data are higher in grade.

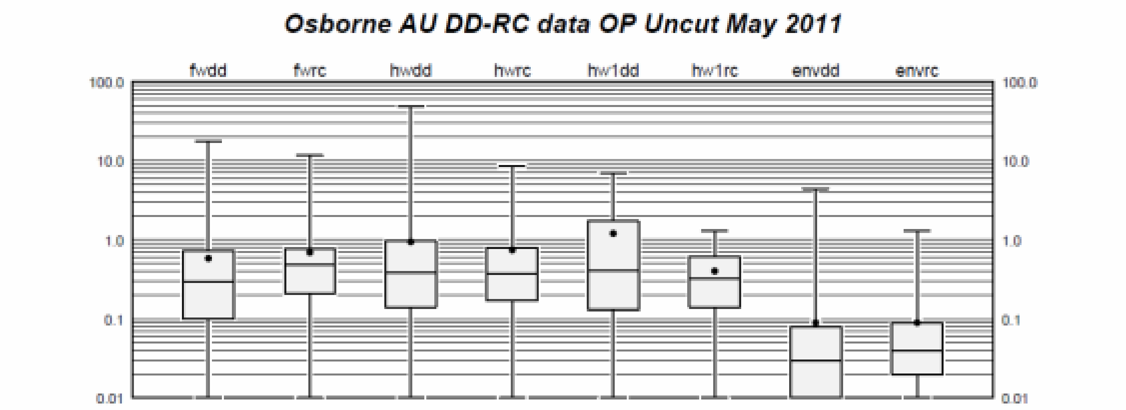
There were insufficient DD-RC pairs in the FW zone to draw meaningful conclusions.

RC data make up approximately 5% of the total data in the HW domain. Despite its lower grade, its effect on Mineral Resource estimates can be ignored.



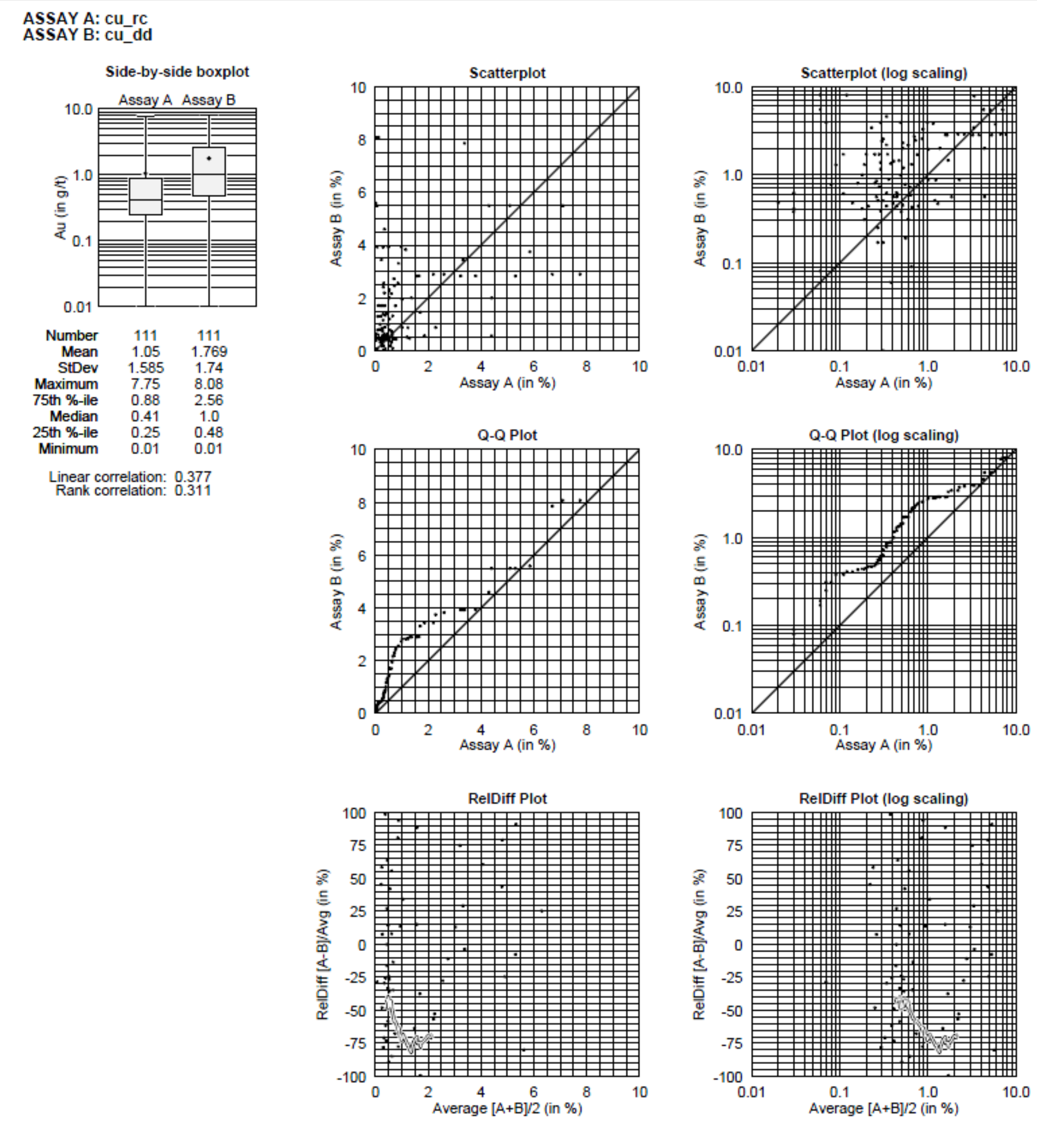
	fwdd	fwrc	hwdd	hwrc	hw1dd	hw1rc	envdd	envrc
Number of data	2105	145	4644	229	62	31	4568	1097
Mean	1.22	0.96	1.49	0.9	1.53	0.58	0.13	0.1
Std. Dev.	1.26	1.09	2.01	1.31	4.13	0.58	0.26	0.17
Coef. of Var.	1.03	1.13	1.35	1.46	2.69	1.01	1.96	1.61
Maximum	15.32	6.4	29.6	7.75	29.0	3.24	5.82	2.0
Upper quartile	1.43	1.06	1.7	0.83	1.19	0.62	0.15	0.12
Median	0.85	0.62	0.75	0.45	0.5	0.43	0.05	0.04
Lower quartile	0.52	0.38	0.35	0.25	0.2	0.25	0.02	0.02
Minimum	0.01	0.01	0.01	0.01	0.04	0.14	0.01	0.01

Figure 14-36: Comparison of DD and RC copper data by domain



	fwdd	fwrc	hwdd	hwrc	hw1dd	hw1rc	envdd	envrc
Number of data	2095	141	4518	224	61	28	4140	859
Mean	0.58	0.69	0.94	0.74	1.2	0.4	0.09	0.09
Std. Dev.	0.98	1.14	1.93	1.16	1.59	0.32	0.22	0.14
Coef. of Var.	1.69	1.65	2.05	1.56	1.33	0.81	2.48	1.65
Maximum	17.66	11.72	49.55	8.6	6.96	1.3	4.44	1.3
Upper quartile	0.73	0.76	0.96	0.8	1.72	0.61	0.08	0.09
Median	0.29	0.48	0.38	0.37	0.41	0.32	0.03	0.04
Lower quartile	0.1	0.21	0.14	0.17	0.13	0.14	0.01	0.02
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01

Figure 14-37: Comparison of DD and RC gold data by domain



Comparison of cu_rc to cu_dd HW Domain, 10m separation

Figure 14-38: Comparison of paired DD and RC copper samples in the HW Domain (10 m max separation)

14.5.7 Variography

The variography for the FW and HW zones used data from outside the stoped area, so they would reflect the undepleted portion of the mineralisation. Correlograms were fitted with spherical models. There were insufficient data in the small fw1 and hw1 domains, so the variograms fitted to the adjacent FW and HW domains were used (respectively). The nugget effect was determined from downhole variograms.

As the HW and FW zones were quite variable in attitude, they were sub-divided on an easting value into two sub-domains with relatively uniform attitudes. The variograms files were averaged irrespective of direction and a combined variogram model fitted. Dynamic anisotropy will re-orient the variogram models to their correct local orientation during estimation.

Where the structure was unclear when using only data outside the mined area, the full data set was used.

The correlograms are shown in Figure 14-39, Figure 14-40 and Figure 14-41. A variogram rather than a correlogram was used for modelling SG in the HW Domain.

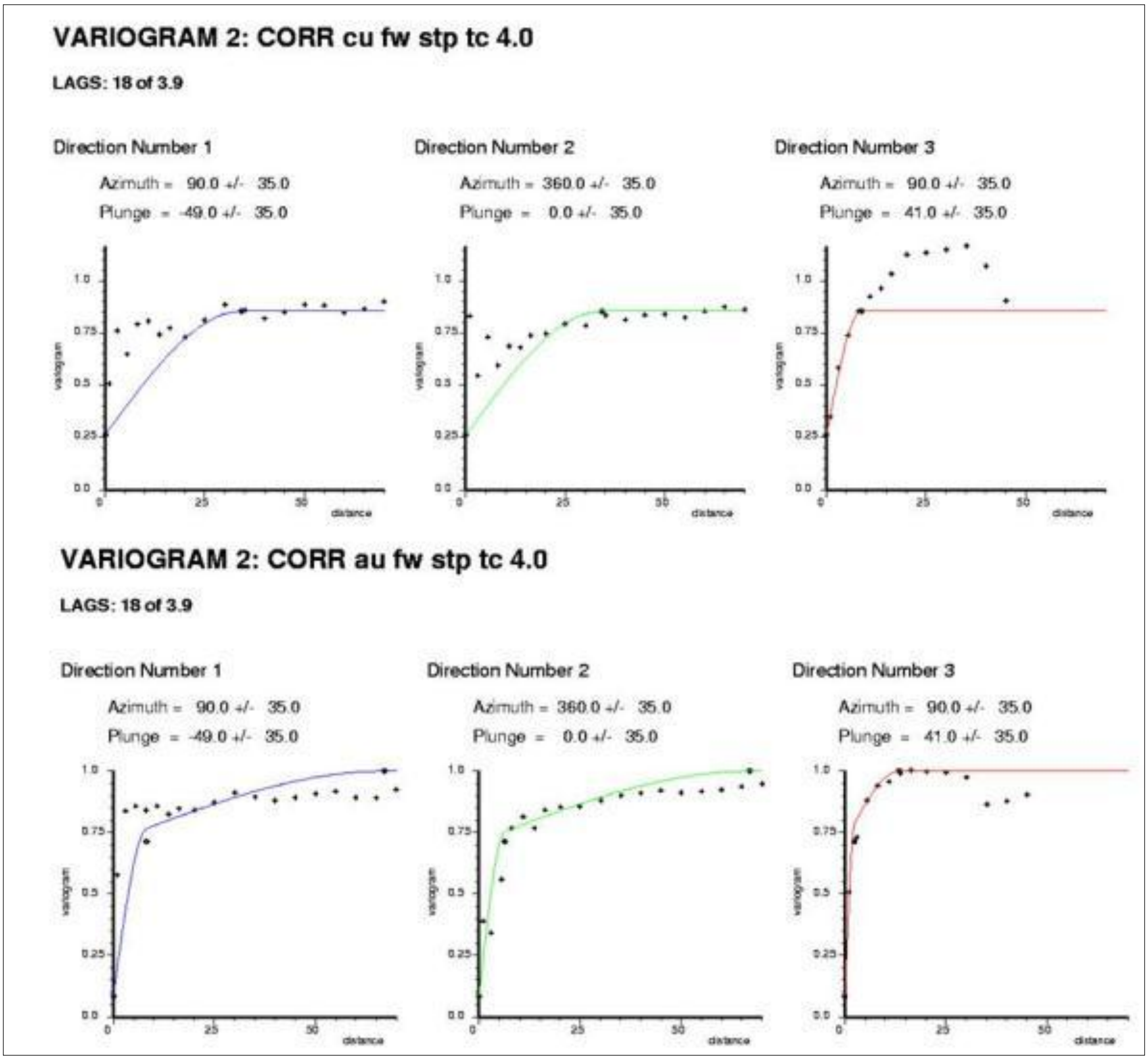


Figure 14-39: Correlograms for copper and gold in the FW Domain

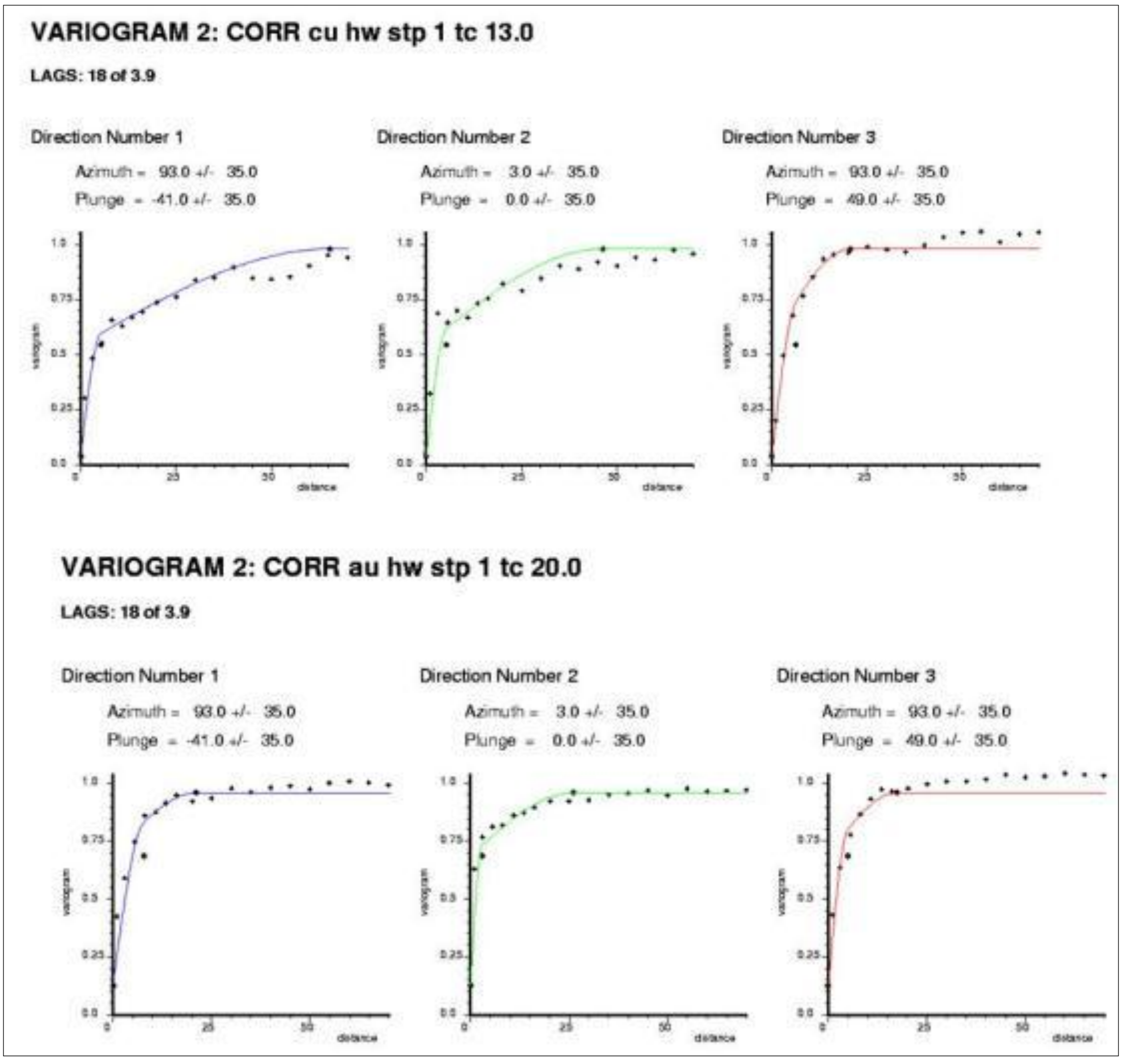


Figure 14-40: Correlograms for copper and gold in the HW Domain

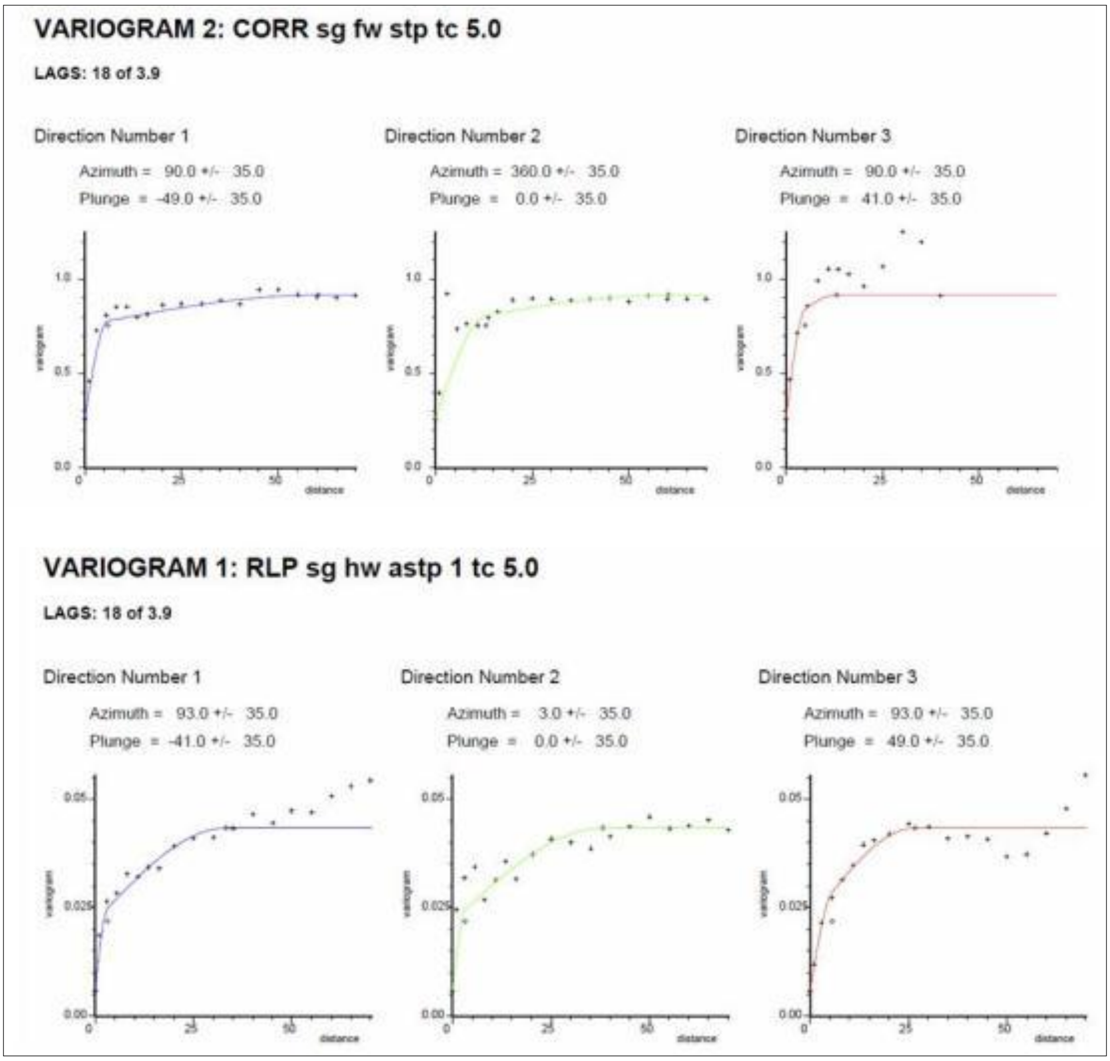


Figure 14-41: Correlograms (FW) and Variograms (HW) for SG

The variogram models are shown in Table 14-23.

Table 14-23: Variogram parameters

est0511a_vp								
DOM1	**METAL**	**VREFNUM**	**VANGLE1**	**VANGLE2**	**VANGLE3**	**VAXIS1**	**VAXIS2**	**VAXIS3**
1	CU	1	90	53	0	3	1	2
2	CU	2	90	49	0	3	1	2
3	CU	3	93	41	0	3	1	2
4	CU	4	90	23	0	3	1	2
5	CU	5	90	49	0	3	1	2
1	AU	6	90	53	0	3	1	2
2	AU	7	90	49	0	3	1	2
3	AU	8	93	41	0	3	1	2
4	AU	9	90	23	0	3	1	2
5	AU	10	90	49	0	3	1	2
1	SG	11	90	53	0	3	1	2
2	SG	12	90	49	0	3	1	2
3	SG	13	93	41	0	3	1	2
4	SG	14	90	23	0	3	1	2
5	SG	15	90	49	0	3	1	2
1	CU	16	90	53	0	3	1	2
2	CU	17	90	49	0	3	1	2
3	CU	18	90	41	0	3	1	2
4	CU	19	90	23	0	3	1	2
5	CU	20	90	49	0	3	1	2

DOM1	NUGGET	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4	ANISO
1	0.262	1	34.20	34.20	6.80	0.592	0	-	-	-	-	1
2	0.262	1	34.20	34.20	6.80	0.592	0	-	-	-	-	1
3	0.035	1	5.29	5.29	6.29	0.509	1	41.60	65.30	10.70	0.437	1
4	0.035	1	5.29	5.29	6.29	0.509	1	41.60	65.30	10.70	0.437	1
5	0.075	1	3.00	13.60	3.00	0.529	1	16.00	46.90	9.40	0.272	1
1	0.083	1	6.43	8.28	2.41	0.630	1	67.00	67.00	10.00	0.287	1
2	0.083	1	6.43	8.28	2.41	0.630	1	67.00	67.00	10.00	0.287	1
3	0.126	1	2.86	7.85	4.92	0.562	1	26.20	21.20	16.40	0.272	1
4	0.126	1	2.86	7.85	4.92	0.562	1	26.20	21.20	16.40	0.272	1
5	0.056	1	15.10	20.10	5.29	0.654	1	25.70	52.70	30.70	0.241	1
1	0.26	1	13.10	5.96	4.97	0.495	1	60.30	60.30	13.10	0.161	1
2	0.26	1	13.10	5.96	4.97	0.495	1	60.30	60.30	13.10	0.161	1
3	0.006	1	3.15	3.15	5.58	0.016	1	38.20	33.20	26.60	0.021	1
4	0.006	1	3.15	3.15	5.58	0.016	1	38.20	33.20	26.60	0.021	1
5	0.002	1	15.90	15.90	9.75	0.011	0	-	-	-	-	1
1	0.2	1	34.00	34.00	7.00	0.800	0	-	-	-	-	1
2	0.2	1	34.00	34.00	7.00	0.800	0	-	-	-	-	1
3	0.2	1	41.60	65.30	10.70	0.800	0	-	-	-	-	1
4	0.2	1	41.60	65.30	10.70	0.800	0	-	-	-	-	1
5	0.2	1	16.00	46.90	9.40	0.800	0	-	-	-	-	1

14.5.8 Block Model and Grade Estimation

The block model extent was increased from the previous estimate in Table 14-24. The model was expanded in case the recent high metal prices might allow mining to a significantly greater depth. The smaller block size was to allow better selectively if mining extends to the vicinity of the previously mined stopes. No sub-blocks were used. Estimation was limited to the defined domains, but background waste blocks were added after estimation to assist pit design.

Table 14-24: Block Model Extents 2008 and 2011

2008 Model					2011 Model				
	Min	Max	Size	Number		Min	Max	Size	Number
X	11000	12000	10	100	X	10900	12000	5	220
Y	20500	22500	10	200	Y	20400	22300	5	380
Z	1000	1310	10	31	Z	690	1300	5	122

14.5.9 Dynamic Anisotropy

Dynamic Anisotropy was used for estimation. This Datamine technique allows search and variogram directions to be defined uniquely for every block to be estimated. The technique is most useful in handling minor changes in strike and dip, and can handle folding, provided sufficient data are available.

Standard digitising of strings in section and plan was adequate for the FW, FW1, HW1 and ENV domains, but the HW Domain was too folded for this technique to work. A mid-plane DTM was modelled for the HW Domain and converted into anisotropy points using the APTOANISO process. The mid-plane points are shown in Figure 14-42, colour-coded by elevation. Some edge-trimming was necessary before the anisotropy could be combined with that of the other domains and the mineralised envelope. The final model was checked by displaying the estimated values as rotated symbols.

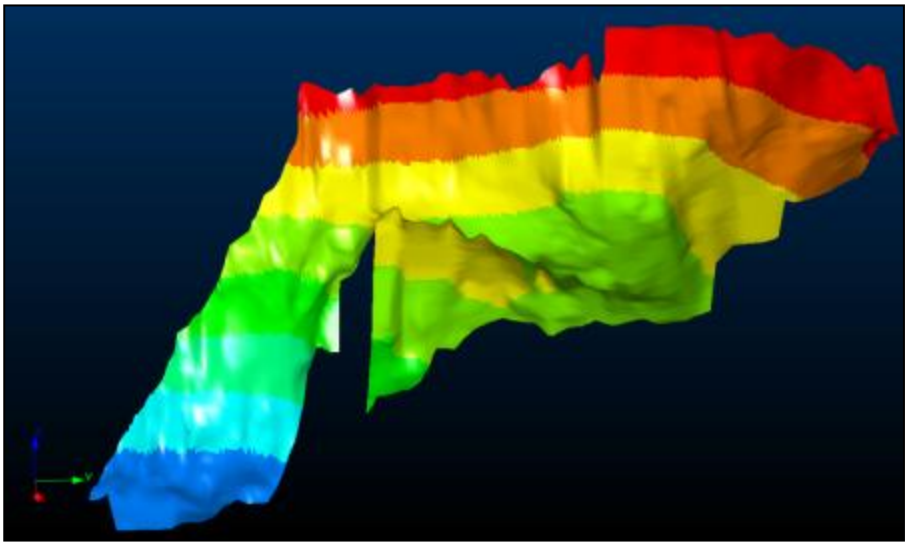


Figure 14-42: Mid-plane used to build the HW Anisotropy Point File

Estimation of the anisotropy directions was done using the NN method. The estimated model was further adjusted for roll-overs. The dips were corrected to true dips using the APTOTRUE process. This was bypassed for the HW Domain, as the APTOANISO process had already created true dips.

The anisotropy model has two new fields – TRDIP and TRDIPDIR – which allowed the search and variogram directions to be locally correct for every block during grade estimation. This happens automatically if these fields are defined in the search parameter file.

14.5.10 Estimation

Estimation of copper, gold and SG was carried out using a Datamine macro. This macro requires two special parameter files in addition to the three (search, estimation and variogram) parameter files required by the ESTIMA process. The additional files define top-cuts and boundary crossing (if any). The parameter files are used as named sets with a common prefix and specified suffix (ep, sp, vp, tc, bd). The name of the set used was “est05111a_”. All hard boundaries were used in the estimation. The top-cuts used are shown in Table 14-22.

Discretisation of 2 m x 2 m x 2m was used when estimating blocks.

The estimation methods used were ordinary Kriging, Inverse Distance Squared (ID^2) and NN. The latter two estimation methods were used to validate the Kriged estimate.

The estimation and search parameters are shown in Table 14-25 and Table 14-26. The variogram parameters are in Table 14-23.

Table 14-25: Estimation parameters

est0511a_ep													
DOM1	METH	SREFNUM	VALUE_IN	VALUE_OU	IMETHOD	POWER	NUMSAM_F	VREFNUM	ANISO	SVOL_F	KRIGNEGW	KRIGVARS	VAR_F
1	krig	1	CU	CU	3	2	NSCU	1	1	SVOLCU	1	1	
2	krig	2	CU	CU	3	2	NSCU	2	1	SVOLCU	1	1	
3	krig	3	CU	CU	3	2	NSCU	3	1	SVOLCU	1	1	
4	krig	4	CU	CU	3	2	NSCU	4	1	SVOLCU	1	1	
5	krig	5	CU	CU	3	2	NSCU	5	1	SVOLCU	1	1	
1	id2	1	CU	CUID2	2	2		-	1		-	-	
2	id2	2	CU	CUID2	2	2		-	1		-	-	
3	id2	3	CU	CUID2	2	2		-	1		-	-	
4	id2	4	CU	CUID2	2	2		-	1		-	-	
5	id2	5	CU	CUID2	2	2		-	1		-	-	
1	nn	1	CU	CUNN	1	2		-	1		-	-	
2	nn	2	CU	CUNN	1	2		-	1		-	-	
3	nn	3	CU	CUNN	1	2		-	1		-	-	
4	nn	4	CU	CUNN	1	2		-	1		-	-	
5	nn	5	CU	CUNN	1	2		-	1		-	-	
1	krig	1	AU	AU	3	2	NSAU	6	1	SVOLAU	1	1	
2	krig	2	AU	AU	3	2	NSAU	7	1	SVOLAU	1	1	
3	krig	3	AU	AU	3	2	NSAU	8	1	SVOLAU	1	1	
4	krig	4	AU	AU	3	2	NSAU	9	1	SVOLAU	1	1	
5	krig	5	AU	AU	3	2	NSAU	10	1	SVOLAU	1	1	
1	id2	1	AU	AUID2	2	2		-	1		-	-	
2	id2	2	AU	AUID2	2	2		-	1		-	-	
3	id2	3	AU	AUID2	2	2		-	1		-	-	
4	id2	4	AU	AUID2	2	2		-	1		-	-	
5	id2	5	AU	AUID2	2	2		-	1		-	-	
1	nn	1	AU	AUNN	1	2		-	1		-	-	
2	nn	2	AU	AUNN	1	2		-	1		-	-	
3	nn	3	AU	AUNN	1	2		-	1		-	-	
4	nn	4	AU	AUNN	1	2		-	1		-	-	
5	nn	5	AU	AUNN	1	2		-	1		-	-	
1	krig	1	SG	SG	3	2		11	1	SVOLSG	1	1	
2	krig	2	SG	SG	3	2		12	1	SVOLSG	1	1	
3	krig	3	SG	SG	3	2		13	1	SVOLSG	1	1	
4	krig	4	SG	SG	3	2		14	1	SVOLSG	1	1	
5	krig	5	SG	SG	3	2		15	1	SVOLSG	1	1	
1	krig	1	CU	CUTMP	3	2		16	1		1	1	KVARCU
2	krig	2	CU	CUTMP	3	2		17	1		1	1	KVARCU
3	krig	3	CU	CUTMP	3	2		18	1		1	1	KVARCU
4	krig	4	CU	CUTMP	3	2		19	1		1	1	KVARCU
5	krig	5	CU	CUTMP	3	2		20	1		1	1	KVARCU

Table 14-26: Search parameters used for estimation

est0511a_sp

DOM1	METAL	SREFNUM	SMETHOD	SDIST1	SDIST2	SDIST3	SANGLE1	SANGLE2	SANGLE3	SAXIS1	SAXIS2	SAXIS3
1	CU	1	2	34.0	34.0	6.8	90	53	0	3	1	2
2	CU	2	2	34.0	34.0	6.8	90	49	0	3	1	2
3	CU	3	2	42.0	65.0	11.0	93	41	0	3	1	2
4	CU	4	2	42.0	65.0	11.0	90	23	0	3	1	2
5	CU	5	2	16.0	46.9	9.4	90	49	0	3	1	2
1	AU	1	2	34.0	34.0	6.8	90	53	0	3	1	2
2	AU	2	2	34.0	34.0	6.8	90	49	0	3	1	2
3	AU	3	2	42.0	65.0	11.0	93	41	0	3	1	2
4	AU	4	2	42.0	65.0	11.0	90	23	0	3	1	2
5	AU	5	2	16.0	46.9	9.4	90	49	0	3	1	2
1	SG	1	2	34.0	34.0	6.8	90	53	0	3	1	2
2	SG	2	2	34.0	34.0	6.8	90	49	0	3	1	2
3	SG	3	2	42.0	65.0	11.0	93	41	0	3	1	2
4	SG	4	2	42.0	65.0	11.0	90	23	0	3	1	2
5	SG	5	2	16.0	46.9	9.4	90	49	0	3	1	2

DOM1	OCTMETH	MINOCT	MINPEROC	MAXPEROC	MINNUM1	MAXNUM1	SVOLFAC2	MINNUM2	MAXNUM2
1	1	2	1	8	5	15	2	5	15
2	1	2	1	8	5	15	2	5	15
3	1	2	1	8	5	15	2	5	15
4	1	2	1	8	5	15	2	5	15
5	1	2	1	8	5	15	2	5	15
1	1	2	1	8	5	15	2	5	15
2	1	2	1	8	5	15	2	5	15
3	1	2	1	8	5	15	2	5	15
4	1	2	1	8	5	15	2	5	15
5	1	2	1	8	5	15	2	5	15
1	1	2	1	8	5	15	2.5	5	15
2	1	2	1	8	5	15	2.5	5	15
3	1	2	1	8	5	15	2.5	5	15
4	1	2	1	8	5	15	2.5	5	15
5	1	2	1	8	5	15	2.5	5	15

DOM1	MAXKEY	SANGL1_F	SANGL2_F	ANISO
1	0	TRDIPDIR	TRDIP	1
2	0	TRDIPDIR	TRDIP	1
3	0	TRDIPDIR	TRDIP	1
4	0	TRDIPDIR	TRDIP	1
5	0	TRDIPDIR	TRDIP	1
1	0	TRDIPDIR	TRDIP	1
2	0	TRDIPDIR	TRDIP	1
3	0	TRDIPDIR	TRDIP	1
4	0	TRDIPDIR	TRDIP	1
5	0	TRDIPDIR	TRDIP	1
1	0	TRDIPDIR	TRDIP	1
2	0	TRDIPDIR	TRDIP	1
3	0	TRDIPDIR	TRDIP	1
4	0	TRDIPDIR	TRDIP	1
5	0	TRDIPDIR	TRDIP	1

Estimation runs were completed with and without Dynamic Anisotropy as a validation step.

14.5.11 Model Validation and Sensitivity

The following steps were taken to validate the estimated model:

- Walk though of the model and the drillhole data in section and plan;
- Summary simple statistics including comparison with the composites:
  - The comparison is made above a zero cut-off to avoid issues with variance differences;
  - Block affected by previous mining are excluded;
  - The estimates are generally very similar; and
  - The lower grade of the declustered composites in then FW zone is probably due to the lower anisotropy possible when doing declustering. By comparison, the NN estimated (made using the same anisotropy as the Kriged estimate) are very similar.

Table 14-27: Comparison of copper estimates (grades in %)

Unmined Copper>=0.01%					
ZONE	**DOM1**	**Copper% KR%**	**Copper%ID2**	**Copper% NN**	**Copper% CMP**
FW1	1	0.9	0.9	0.9	0.9
FW	2	1.1	1.1	1.1	0.8
HW	3	0.9	0.9	0.9	0.9
Env	5	0.1	0.1	0.1	0.1

- Comparison of the estimates made with and without Dynamic Anisotropy:
  - Estimates were made of the model with and without use of Dynamic Anisotropy. The results were very similar (all unmined material >= 0.5 % eCu).

Table 14-28: Comparison of Estimates made with / without Dynamic Anisotropy

Effect of Dynamic Anisotropy eCu>=0.5%			
% Diff T	**% Diff eCu**	**% Diff Copper**	**% Diff Gold**
0.4	0.5	0.6	0.2

- Comparison of Kriged and ID^2 Estimates at different cut-offs:
  - The Kriged and ID^2 estimates in the unmined area are compared as a function of copper cut-off grade in Table 14-29 and Figure 14-43. Tonnes are shown as percentages as these are not Mineral Resources; and
  - The results are very similar, except at very high cut-offs where ID^2 is slightly higher.

Table 14-29: Comparison of copper and copper ID^2 at various Copper cut-offs

Cut-off	% Tonnes Kr	Copper % Kr	% Tonnes ID^2	Copper % ID^2
0.0	100.0	0.3	100.0	0.3
0.1	40.6	0.6	39.4	0.6
0.2	29.2	0.8	29.1	0.8
0.3	23.9	0.9	23.7	0.9
0.4	20.5	1.0	20.4	1.0
0.5	17.6	1.1	17.5	1.1
0.6	14.8	1.2	14.8	1.2
0.7	12.4	1.3	12.4	1.3
0.8	10.2	1.4	10.3	1.5
0.9	8.4	1.6	8.6	1.6
1.0	7.0	1.7	7.2	1.7
1.1	5.8	1.8	6.0	1.9
1.2	4.9	1.9	5.0	2.0
1.3	4.1	2.1	4.3	2.1
1.4	3.5	2.2	3.6	2.2
1.5	3.0	2.3	3.2	2.4

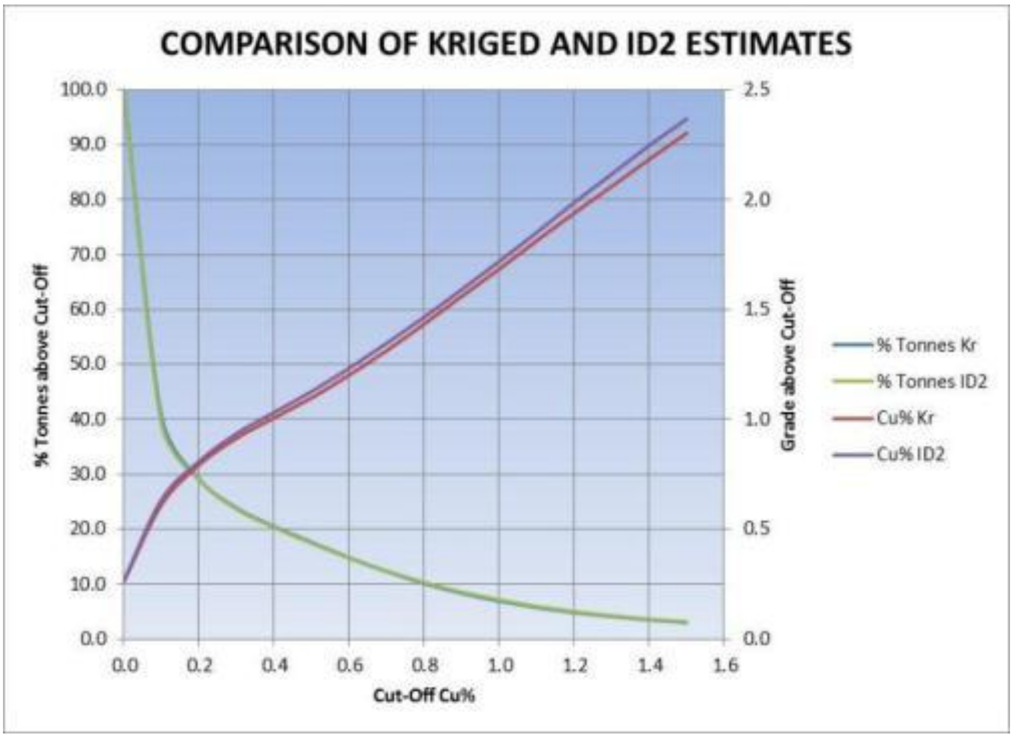


Figure 14-43: Grade-tonne comparison of copper Kriged and ID

- Swath Plots:
  - The estimated block grades were compared with the block-average top-cut drillhole composite grades using swath plots in Figure 14-44 The comparison is between Kriged grade of blocks that contain drillholes, and the block-average grade of the composites in the blocks.

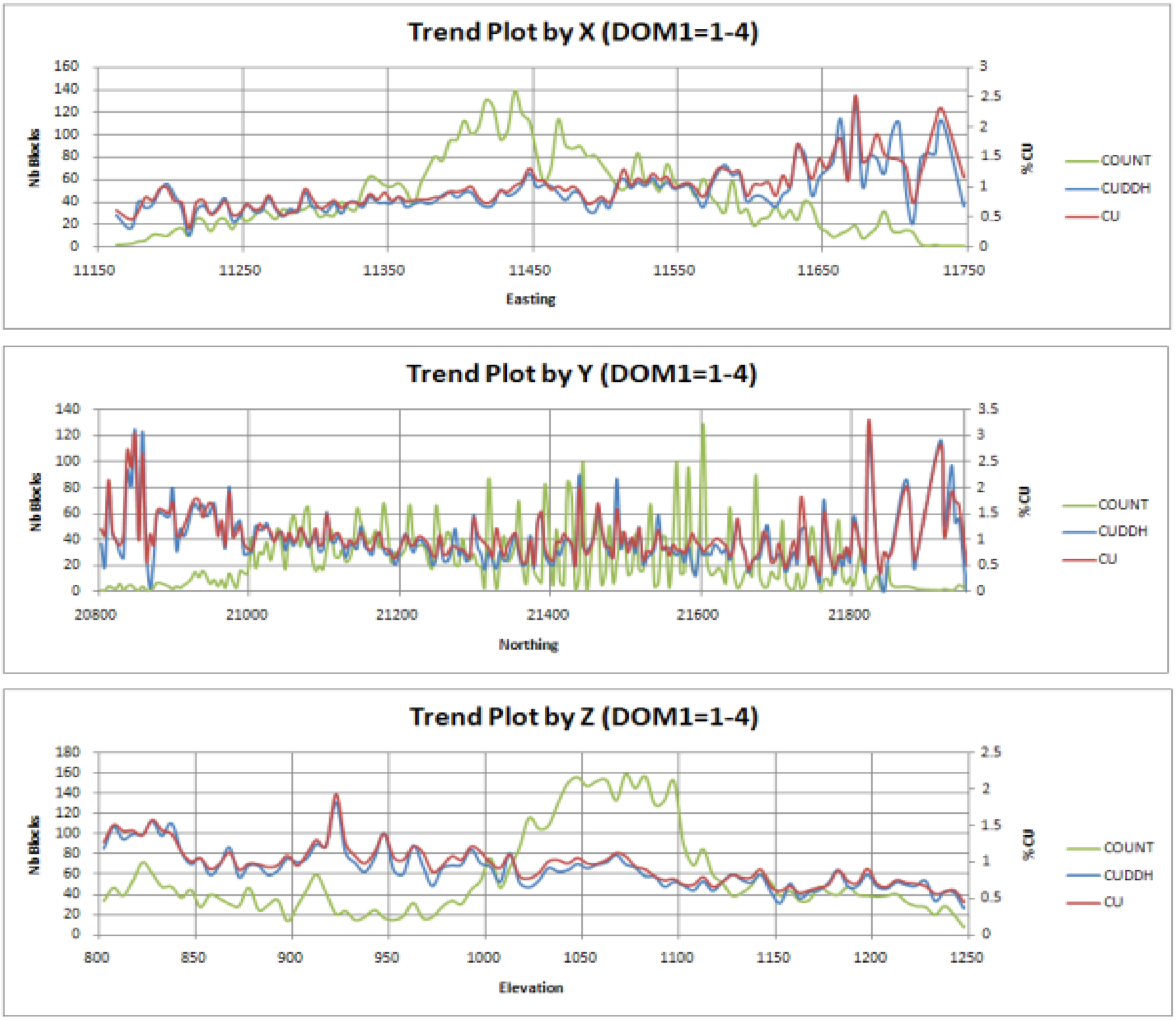


Figure 14-44: Swath plots for copper by direction

- Test of local variability and bias:
  - The drillhole composites were top-cut then averaged into regular blocks and merged with a regularised Mineral Resource model. The grades were compared using a scatterplot. Both the mean grades and the correlation coefficient were satisfactory, indicating a lack of local bias and the model has the correct amount of local variability.

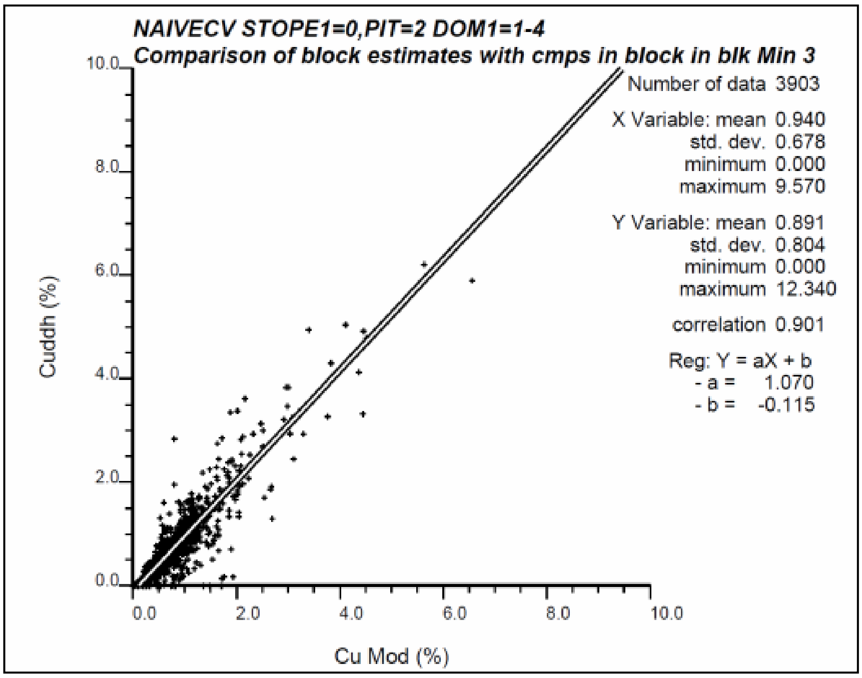


Figure 14-45: Comparison of copper estimates with composites in blocks

- Comparison of the variability of the model with that calculated from the composites:
  - The global variability of model for the Hangingwall (HW) ore zone was compared with the theoretical variability calculated from the composites and the declustering weights using the Indirect Log-Normal Correction. This uses the variogram model and the block dimensions to adjust grades and declustering weights of the composites to that of the model. The expected Coefficient of Variation (CV) for the ore zone model below the current pit is 0.915. The estimated model will have about the right amount of smoothing if its CV is 85% of this value or 0.9=0.78. In practice, the CV of the model of the HW ore zone below the pit is 0.81, indicating that the model has the right amount of smoothing.

14.5.12 Removal of Mined-Out areas and Addition of Waste

Much of the open pit zone has already been mined, either by open cut or by stoping in Figure 14-46. The as-mined pit, the stopes and development wireframes were used to deplete the Mineral Resources using wireframe selection. The cut-out string method used to deplete the 1SS orebody could not be used for model depletion in the open pit area, as there is appreciable mineralisation, possibly mineable by open pit, above the mined-out stopes. The stopes and development were filled with blocks and sub-blocks, so the combined model needed to be regularised after combining.

New fields PIT and STOPE1 were added to the model to flag blocks affected by previous mining. Any blocks that were partly stoped or completely stoped, or whose centres were above the previously mined pit were given zero grades. Density was also changed: blocks in stopes or in the back fill of the pit were given a density of 2.0 g/cm^3. Blocks in development or above the as-mined topography received a density of 0.0.

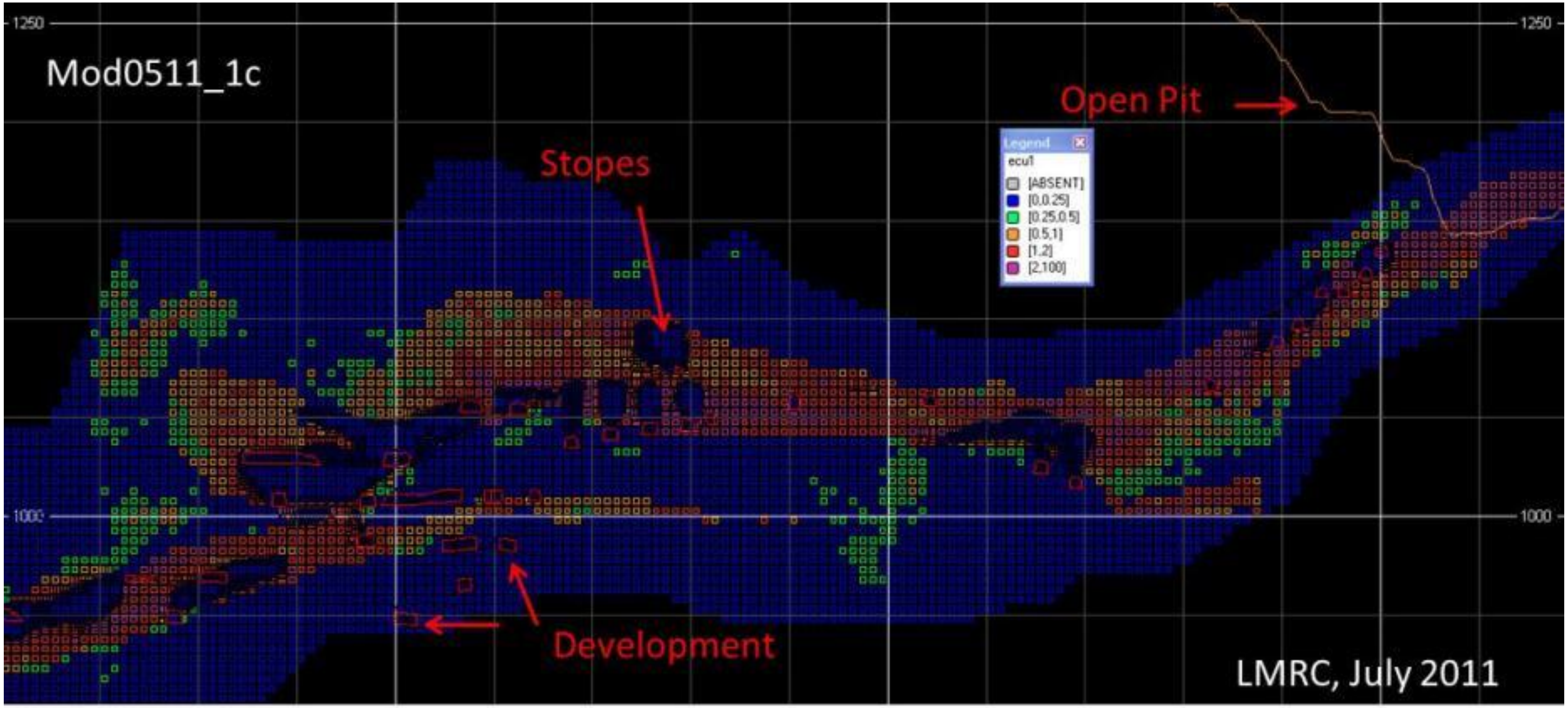


Figure 14-46: Sectional view of mined stopes and current open pit

A background model was added for open pit modelling. This fully-populated, zero grade model was tagged by the surfaces and the open pit and default density values were applied. The estimated model was superimposed on the background model.

The reported Resources are limited by an optimum pit designed in 2010. This pit was designed using the 2007 model and limits the Resources to avoid the previously-mined stopes and development. Though not optimum for the 2011 Resource estimates, the pit does provide a limit to what is currently considered mineable. There is a considerable amount of +0.5% eCu material outside the 2010 pit limits that might be mineable under different economic conditions.

14.5.13 Mineral Resource Classification

Block model quantities and grade estimates for the Open Pit Project at Osborne were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by R W Lewis, FAusIMM (No 100799), an appropriate independent qualified person for the purpose of NI 43-101.

Mineral Resource classification is typically a subjective concept, industry best practices suggest that Mineral Resource classification should consider both the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar Mineral Resource classification.

LMRC is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced approximately 20-30 m apart, closer in places where underground ring-drilling is available. In general, the open pit area is very well informed by close-spaced drillholes.

Classification was applied using Kriging variance. The same variogram model (Nugget Effect 0.2 and C1 of 0.8) was used for all domains. The variogram ranges used were the same as for the principal estimates. Figure 14-48 shows the classification applied (Measured, Indicated and Inferred Mineral Resources have CLASS values of 1, 2, and 3 respectively).

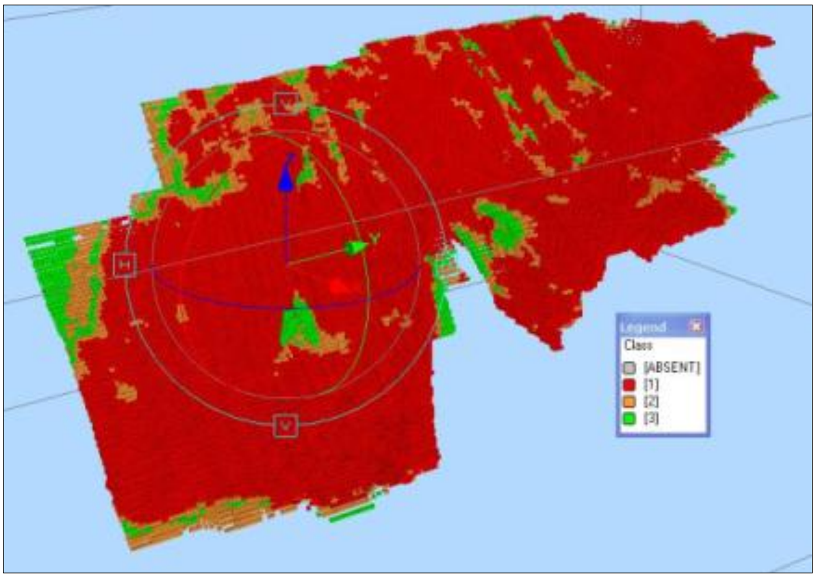


Figure 14-47: Classified regular Block Model

Source: LMRC, July 2011

14.5.14 Osborne Open Pit Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a Mineral Resource as:

"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The classified Mineral Resources above a cut-off of 0.5% eCu are shown in Table 14-30. These Mineral Resources have been depleted for previous open pit and underground mining, and are limited by the 2010 designed open pit. There is considerable additional mineralised material above cut-off below the 2010 pit, but this is currently excluded from Mineral Resources. Mining of this material would impinge on previously mined stopes and development needed to extract other underground Mineral Resources.

Table 14-30: Osborne Open Pit Classified Mineral Resources limited by the 2010 Pit

2011 Model - All unmined material >= 0.5% eCu limited by the 2010 Pit						
Category	**Quantity (Mt)**	**Grade**			**Metal**	
		eCu (%)	**Copper (%)**	**Gold (g/t)**	**Copper (000' t)**	**Gold (000' oz)**
Measured	2.232	1.1	0.7	0.6	16.5	40.8
Indicated	0.209	1.1	0.7	0.6	1.5	4.1
Measured + Indicated	**2.442**	**1.1**	**0.7**	**0.6**	**18.0**	**44.9**
Inferred	0.073	0.9	0.6	0.6	0.4	1.3

1 eCu = copper (%) + gold (g/t) x 0.6.

2 The Mineral Resource Estimate is effective as at 27 October 2011.

3 The Mineral Resource Estimates have been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

4 Some totals may not add due to the effects of rounding.

The breakdown of the total Mineral Resources by material type is shown in Table 14-31 as percentages of the total unmined material limited by the 2010 pit. TOX is totally oxidised, POX is partially oxidised. Most of the Mineral Resources are Fresh.

Table 14-31: Osborne Open Pit Mineral Resources by material type limited by the 2010 Pit

All unmined material >=0.5% eCu (Pit Limited)				
Category	**Quantity (%)**	**Grade**		
		eCu (%)	**Copper (%)**	**Gold (g/t)**
TOX	2.2	1.0	0.6	0.7
POX	22.0	1.0	0.7	0.6
TOX and POX	**24.2**	**1.0**	**0.7**	**0.6**
Fresh	75.8	1.1	0.8	0.6
Total	100.0	1.1	0.7	0.6

1 eCu = copper (%) + gold (g/t) x 0.6.

2 The Mineral Resource Estimate is effective as at 27 October 2011.

3 The Mineral Resource Estimates have been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

4 Some totals may not add due to the effects of rounding.

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.5.15 Grade Sensitivity Analysis

The Mineral Resources of the Open Pit Project at Osborne are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates for blocks classified as Measured and Indicated are presented in Table 14-32 at different cut-off grades. The figures presented in this table should not be misconstrued with a Mineral Resource Statement. *The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.* Figure 14-48 presents this sensitivity as a grade tonnage curve.

Table 14-32: Pit-limited Block Model quantities and grade estimates at various eCu cut-offs

Global Model Quantities and Grades				
Unmined blocks classified as Measured and Indicated				
Cut-off eCu%	**Mt**	**eCu (%)**	**Copper (%)**	**Gold (g/t)**
0.0	5.4	0.6	0.4	0.3
0.1	4.2	0.7	0.5	0.4
0.2	3.4	0.9	0.6	0.5
0.3	3.0	1.0	0.7	0.5
0.4	2.7	1.0	0.7	0.5
0.5	2.4	1.1	0.7	0.6
0.6	2.2	1.1	0.8	0.6
0.7	1.9	1.2	0.8	0.6
0.8	1.6	1.3	0.9	0.7
0.9	1.4	1.4	0.9	0.7
1.0	1.2	1.4	1.0	0.8
1.1	1.0	1.5	1.0	0.8
1.2	0.8	1.6	1.1	0.8
1.3	0.7	1.7	1.1	0.9

Global Model Quantities and Grades				
Unmined blocks classified as Measured and Indicated				
Cut-off eCu%	Mt	eCu (%)	Copper (%)	Gold (g/t)
1.4	0.5	1.7	1.2	0.9
1.5	0.4	1.8	1.3	0.9
1.6	0.3	1.9	1.4	1.0
1.7	0.2	2.0	1.4	1.0
1.8	0.2	2.2	1.5	1.1
1.9	0.1	2.3	1.6	1.1
2.0	0.1	2.4	1.7	1.2

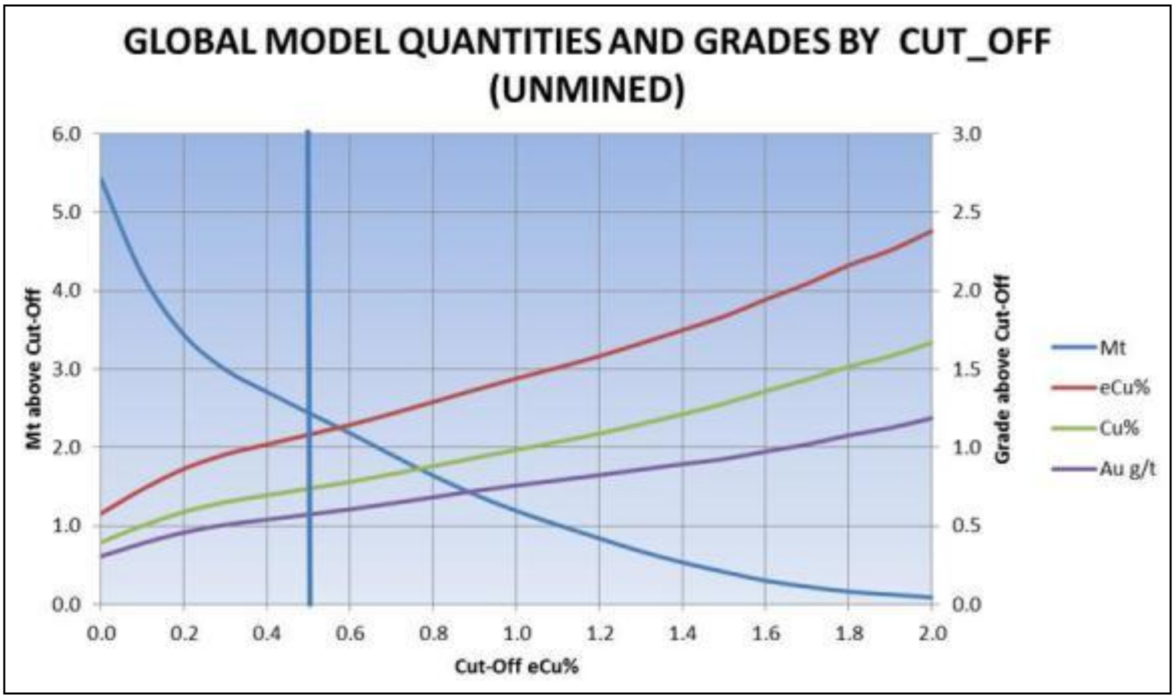


Figure 14-48: Grade-tonnage curves for open pit (pit limited)

14.5.16 Previous Mineral Resource Estimates

The published open pit Mineral Resources (including Mineral Reserves) above a cut-off of 0.6% eCu are shown in Table 14-33 (Ivanhoe, 2010).

Table 14-33: Published 2010 Open Pit Mineral Resources (cut-off 0.6% eCu)

All unmined material >= 0.6% eCu limited by a pit design						
Category	Quantity (Mt)	Grade			Metal	
		eCu (%)	Copper (%)	Gold (g/t)	Copper (000' t)	Gold (000' oz)
Measured	0.10	1.6	1.1	0.8	1.1	2.6
Indicated	0.70	1.3	0.9	0.6	6.3	13.5
Measured + Indicated	**0.80**	**1.3**	**0.9**	**0.6**	**7.4**	**16.1**
Inferred	1.10	1.1	0.8	0.5	8.8	17.7

eCu=copper % + gold g/t x 0.6

This Mineral Resource is above a higher cut-off grade (0.6% eCu) than is currently used. Table 14-34 shows the 2010 Resource above the current eCu cut-off (0.5%). The Mineral Resources in the tables are less than expected despite the lower cut-off used. There is some uncertainty in which block model and pit design were used for the published Mineral Resources.

Table 14-34: 2010 Open Pit Mineral Resources above a cut-off of 0.5% eCu

All unmined material >= 0.5% eCu limited by the 2010 Pit						
Category	**Quantity (Mt)**	**Grade**			**Metal**	
		eCu (%)	**Copper (%)**	**Gold (g/t)**	**Copper (000' t)**	**Gold (000' oz)**
Measured	0.072	1.5	1.1	0.7	0.8	1.7
Indicated	0.595	1.2	0.9	0.6	5.1	11.0
Measured + Indicated	**0.667**	**1.2**	**0.9**	**0.6**	**5.9**	**12.6**
Inferred	1.644	1.0	0.7	0.5	11.3	25.0

eCu=copper % + gold g/t x 0.6

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The new Mineral Resources are shown in Table 14-35. The new Mineral Resource has slightly higher tonnes and grade than the 2010 Mineral Resources above the same cut-off. In addition, the relative proportions of the classes of mineralisation have changed.

Table 14-35: 2011 Open Pit Mineral Resources above a cut-off of 0.5 % eCu

2011 Model - All unmined material >= 0.5% eCu limited by the 2010 Pit						
Category	**Quantity (Mt)**	**Grade**			**Metal**	
		eCu (%)	**Copper (%)**	**Gold (g/t)**	**Copper (000' t)**	**Gold (000' oz)**
Measured	2.232	1.1	0.7	0.6	16.5	40.8
Indicated	0.209	1.1	0.7	0.6	1.5	4.1
Measured + Indicated	**2.442**	**1.1**	**0.7**	**0.6**	**18.0**	**44.9**
Inferred	0.073	0.9	0.6	0.6	0.4	1.3

eCu=copper % + gold g/t x 0.6

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.5.17 Recommendations for Conversion of Mineral Resources into Mineral Reserves

The Mineral Resources are well drilled already. The main uncertainties relate to how close to previously-mined stopes mining should be allowed, and whether underground development may need to be preserved for extraction of other Mineral Resources. There is potential for a considerable increase in Mineral Resources, depending on mining constraints.

LMRC considers that the blocks located within the conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a Mineral Resource.

14.6 Resource Estimation Kulthor

14.6.1 Mineral Resource Database Kulthor

The data were dumped from the Ivanhoe acQuire database as 5 comma-separated value (csv) files. Additional assays for drillhole SUHQ0206 were provided later. The fields were selected and renamed using a series of AWK scripts before importing into Datamine. There were data for 914 surface and underground drillholes.

The drillhole data were divided into two groups: that available as of July 2011 and that provided since (Table 14-36). The newer data are higher in grade, probably because of the higher proportion of short underground drillholes.

Table 14-36: Data distribution by date

drilld_0812at1								
DATE	Nb CU	Mean Cu%	Max Cu%	Min Cu%	Nb Au	Mean Au g/t	Max Au g/t	Min Au g/t
AU12	13581	0.27	16.55	0.00	13580	0.14	16.95	0.00
JU11	31168	0.18	19.10	0.00	31168	0.11	46.00	0.00

The drillholes had coordinates and elevations in the Osborne Mine local grid as the Kulthor mineralisation is accessed by development from the Osborne mine.

14.6.2 Solid Body Modelling

The Kulthor mineralisation was modelled as three zones as shown in Figure 14-49. Two of these zones (“N” and “FN”) do not have additional drilling since last estimated in 2011 and are unchanged.

The mineralised zones were modelled on sections at 15 m to 30 m spacing, approximately normal to the strike of the mineralisation. The boundaries of the mineralised zones were chosen using a combination of geology and grade displayed as down-the-hole histograms. Boundaries were chosen as far as possible where there was an abrupt change in grade rather than by use of a specific grade cut-off. Emphasis was placed on maintaining continuity of shape and orientation from section to section.

The “M” zone has received all the additional drilling completed since the last estimate. Though its general shape and position are relatively unchanged confirming the previous geological interpretation, some changes were made resulting in a slightly narrower and higher-grade zone. Recent surface drilling has located several zones of mineralisation on strike to the west of the “M” zone Figure 14-49). This drilling is too wide-spaced to be able to show that these intersections form one or more coherent bodies. Several other small bodies were modelled south of the “M” zone and are referred to as “Subsidiary M zones”. They are not known to be as continuous as the main portion of the “M” zone.

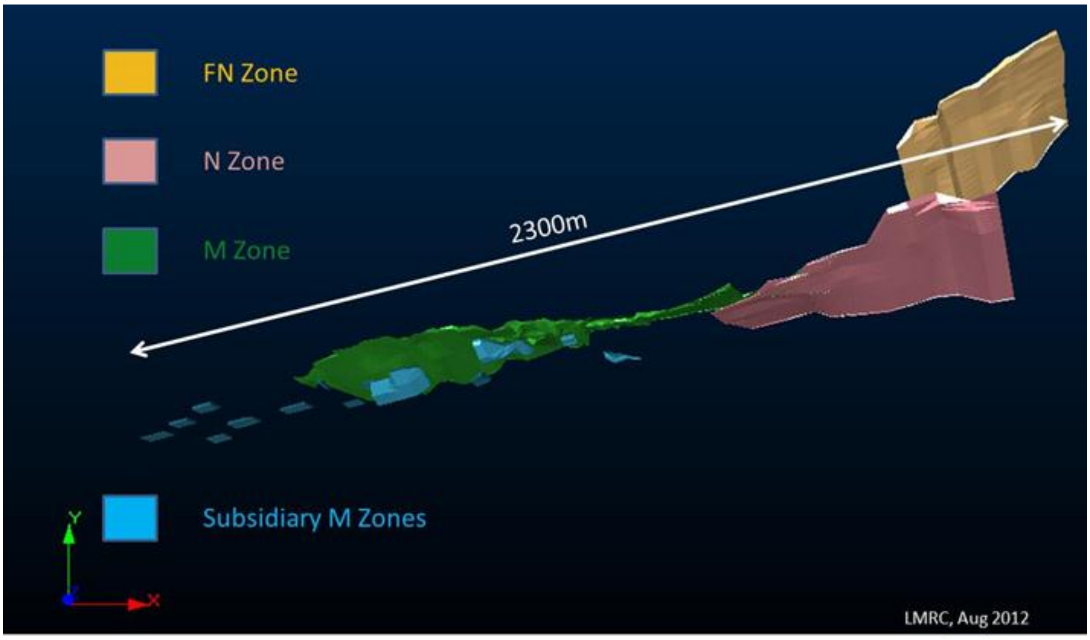


Figure 14-49: Kulthor mineralised zones (looking north)

The Kulthor mineralisation consists of steep easterly-trending mineralised zones that are also shears. These shears have resulted in a deep depression of the base of total oxidation as shown in Figure 14-50. The mineralisation extends in places up to the semi-horizontal base of Mesozoic surface. This surface is approximately 40 m below the current topography in the Kulthor area.

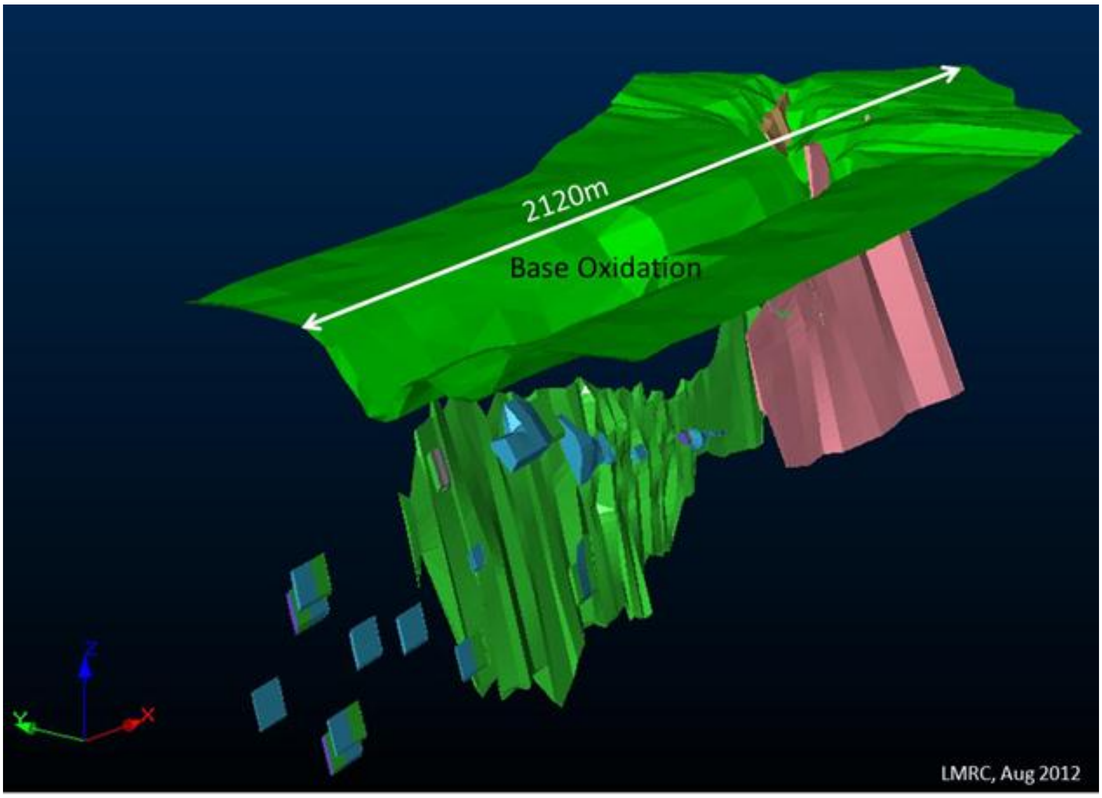


Figure 14-50: Kulthor base of oxidation and mineralised zones

Previous estimates modelled lower-grade dilution zones outside the three mineralised zones. This was done in the new estimate only for the "N" zone. In addition, a new background domain was modelled to encompass all three mineralised zones. This was based on assays and the presence of quartz-dolomite veins and sulphide zones (Figure 14-51).

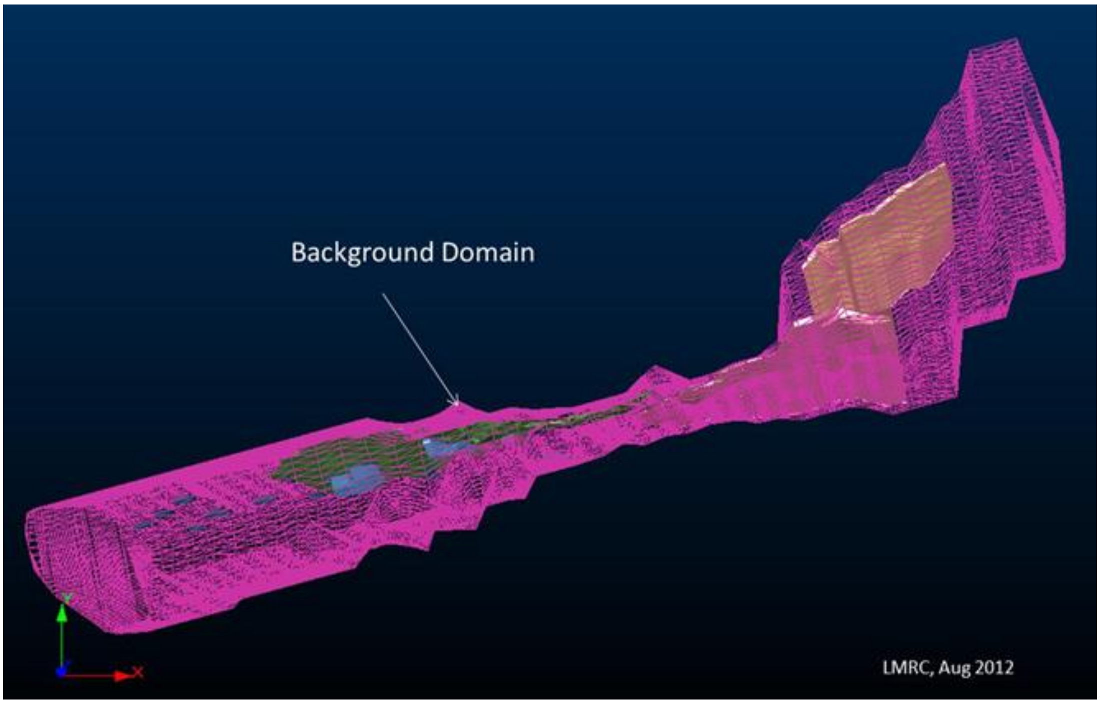


Figure 14-51: Kulthor background domain (magenta)

14.6.3 Data Flagging

The drillhole data were tagged in priority order by the mineralisation wireframes. The domain (field DOM) codes (Table 14-37) were further adjusted outwards by a maximum of 1 sample if the neighbouring sample outside the mineralised domain was >= 0.9% eCU. This slightly softened the boundaries of the mineralisation zones and corrected any miss-selection by the wireframes. Twenty two additional samples were added to domains 1-4 (19 were added to domain 4).

Table 14-37: Wireframe Flagging Codes (CD)

BASE	CD
okn12_11	1
okn05_11	2
okfn05	3
okm08_12_main	4
okm08_12_w1	5
okm08_12_w2	6
okm08_12_w3	7
okm08_12_w4	8
okm08_12_w5	9
okm08_12_w6	10
okm08_12_w7	11
okm08_12_w8	12
okm08_12_w9	13
okm08_12_w10	14
okm08_12_w11	15
okm08_12_w12	16
okm08_12_w13	17
okm08_12_w14	18
okm08_12_w15	19
okm08_12_w16	20
ok08_12_back	21

14.6.4 Compositing

The most common sample length is 1 m, though 2 m samples were used in some drillholes (Figure 14-52).

As the most common sample length was 1 m, the samples were composited to 1 m lengths using zone control (field DOM). Zone compositing can result in short composites where drillholes leave a domain. It is usual to exclude samples less than half the compositing interval when making estimates to prevent short samples (especially on boundaries) having equal (or in the case of Kriging, higher) weights than 1 m composites. Discarding short composites does result in some loss of information.

There are two options available to prevent this happening:

1 The length of all composites in a drillhole intersection through a domain can be adjusted so equal length composites are created. This works best with wide mineralised bodies; and
2 Short composites (less than 0.5 m) can be combined with the neighbouring composites in the zone.

The second method was used as the Kulthor mineralised zones are relatively narrow.

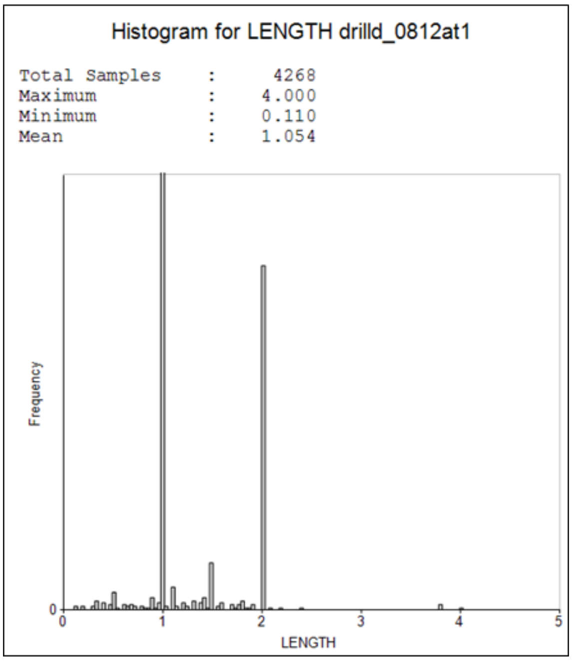


Figure 14-52: Histogram of sample lengths for mineralised samples

The composites were flagged by the topography, base of Mesozoic and base of Total Oxidation surfaces creating a SURFACE code field. The code values (field CD) applied are shown in Table 14-38. All composites below the base of total oxidation received a default SURFACE code of 3.

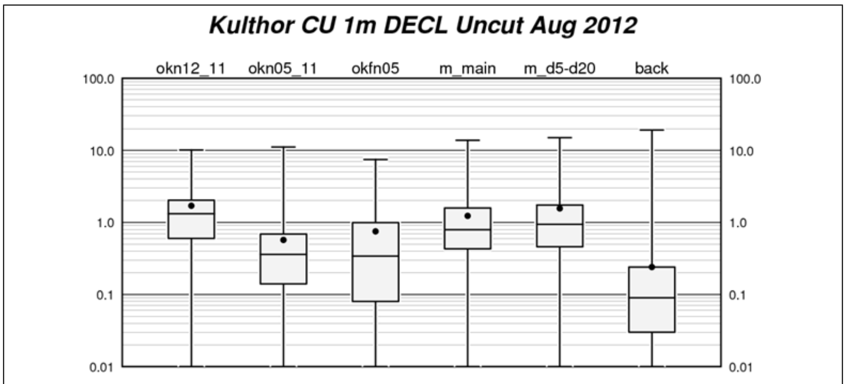
Table 14-38: SURFACE flagging of composites

BASE	CD
kul_topo	0
okmeso	1
kul11box	2
FRESH	3

14.6.5 Statistical Analysis and Evaluation of Outliers

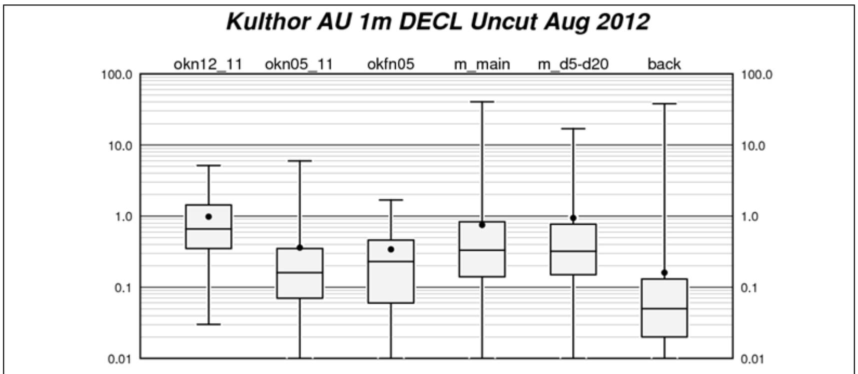
The composites were declustered using the polygonal method. This method utilises a geology block model in the declustering process. As polygonal declustering can result in high declustering weights where drillholes enter and leave domains, the weights of such samples were adjusted to be the same as that of the next sample in the same hole inside the domain. Declustering weights were further trimmed to remove outliers.

Figure 14-53 and Figure 14-54 show boxplots of the declustered 1 m composites for copper and gold by domain. Domains 5-20 (subsidiary M domains) are combined as they do not have many data. The CV is relatively low except for copper and gold in the background domain. There are high values that will require top-cutting (e.g. the maximum gold grade is 40.4 g/t). Figure 14-55 shows the boxplots for density (SG). The average density values by domain were used later to supply default density values where the lower number of density data resulted in lack of estimates for blocks with copper and gold estimates.



	okn12_11	okn05_11	okfn05	m_main	m_d5-d20	back	
Number of data	120	463	117	3020	762	26121	Number of data
Mean	1.7	0.57	0.75	1.23	1.56	0.24	Mean
Std. Dev.	1.72	0.92	1.16	1.33	2.16	0.73	Std. Dev.
Coef. of Var.	1.01	1.62	1.56	1.08	1.39	3.0	Coef. of Var.
Maximum	10.1	11.09	7.45	13.75	14.97	19.1	Maximum
Upper quartile	2.03	0.69	0.99	1.58	1.74	0.24	Upper quartile
Median	1.32	0.36	0.34	0.79	0.94	0.09	Median
Lower quartile	0.6	0.14	0.08	0.43	0.46	0.03	Lower quartile
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	Minimum

Figure 14-53: Boxplots of declustered 1 m copper composites by domain



	okn12_11	okn05_11	okfn05	m_main	m_d5-d20	back	
Number of data	119	445	120	3003	753	22186	Number of data
Mean	0.98	0.36	0.34	0.75	0.94	0.16	Mean
Std. Dev.	0.9	0.65	0.36	1.69	2.05	0.52	Std. Dev.
Coef. of Var.	0.92	1.79	1.06	2.25	2.19	3.34	Coef. of Var.
Maximum	5.12	5.96	1.68	40.4	16.95	37.98	Maximum
Upper quartile	1.43	0.35	0.46	0.83	0.77	0.13	Upper quartile
Median	0.66	0.16	0.23	0.33	0.32	0.05	Median
Lower quartile	0.35	0.07	0.06	0.14	0.15	0.02	Lower quartile
Minimum	0.03	0.01	0.01	0.01	0.01	0.01	Minimum

Figure 14-54: Boxplots of declustered 1 m gold composites by domain

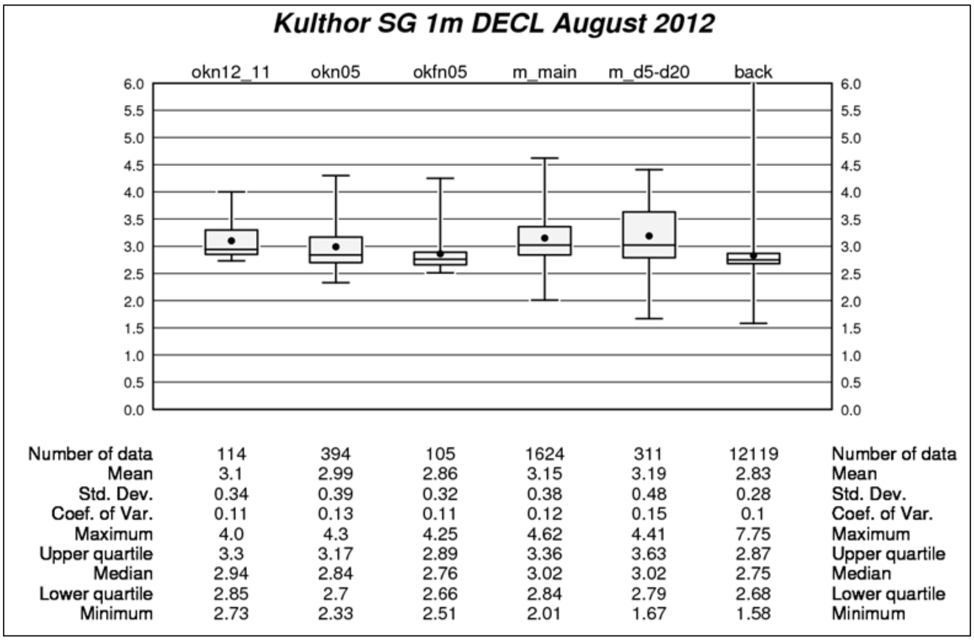


Figure 14-55: Boxplots of declustered 1 m density composites by domain

Most of the Kulthor mineralization is fresh. Figure 14-56 shows the distribution of copper and gold by oxidation type in the mineralised domains (1-20). There are insufficient data to show whether there is a significant grade difference caused by oxidation.

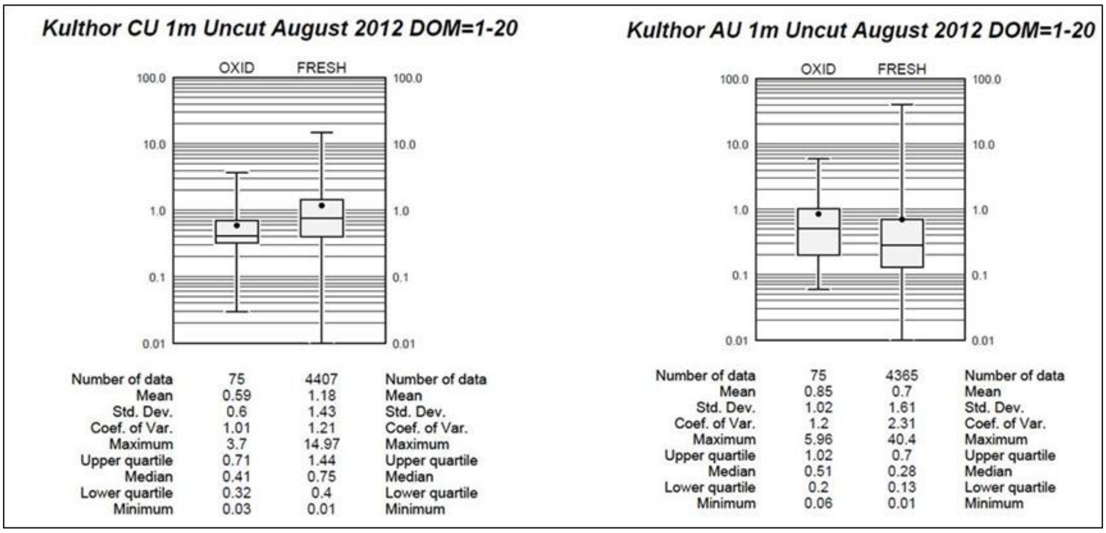


Figure 14-56: Boxplots of declustered 1 m copper and gold composites by oxidation type

The Kulthor mineralisation has been sampled using AC, RC and DDH drilling. Most of the data are Diamond Core. There were no AC data through the mineralisation zones. The distribution of composites by sample type through the mineralised zones is shown in Figure 14-57. No conclusions can be made about the difference between the RC samples and the DDH samples due the small number of RC samples.

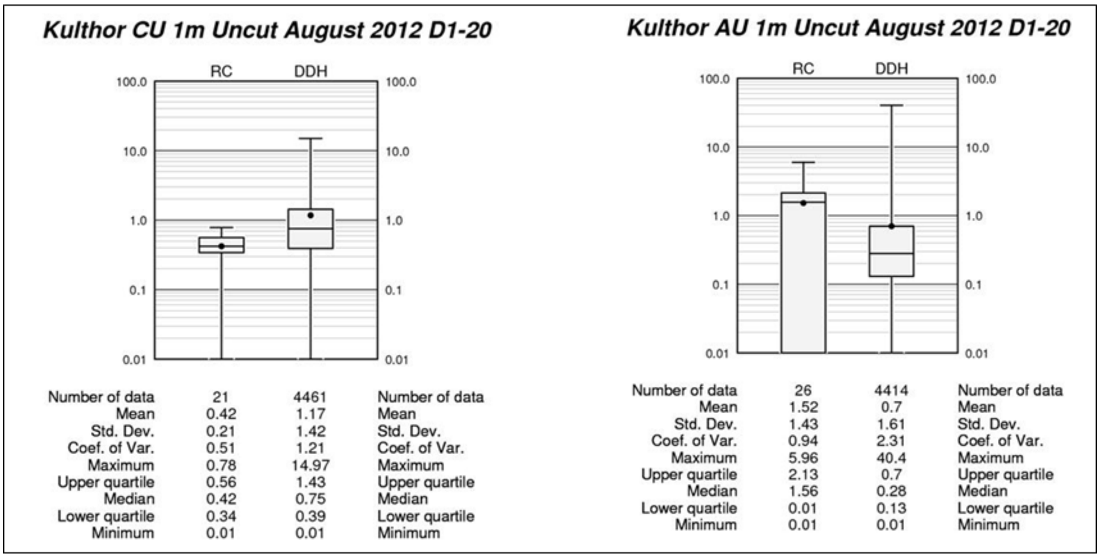


Figure 14-57: Boxplots of declustered 1 m copper and gold composites by sample type

Top-cuts were selected for copper and gold using a combination of cutting statistic plots, histograms and probability plots. The chosen top-cuts are shown in Table 14-39. Domains 5-17 and 20 had insufficient data to determine a top-cut.

Table 14-39: Top-cuts for copper and gold by Domain

Top Cuts							
ZONE	DOM	Nb Data CU	Top Cut CU	Nb Cut CU	Nb Data AU	Top Cut AU	Nb Cut AU
okn12_11	1	120	7.0	2	120	3.0	4
okn05_11	2	477	7.0	2	477	3.5	7
okfn05	3	123	4.0	2	123	1.5	2
okm08_12_main	4	3054	8.0	11	3054	15.0	8
okm08_12_w1	5	4	99.0	0	4	99.0	0
okm08_12_w2	6	12	99.0	0	12	99.0	0
okm08_12_w3	7	7	99.0	0	7	99.0	0
okm08_12_w4	8	17	99.0	0	17	99.0	0
okm08_12_w5	9	14	99.0	0	14	99.0	0
okm08_12_w6	10	9	99.0	0	9	99.0	0
okm08_12_w7	11	10	99.0	0	10	99.0	0
okm08_12_w8	12	7	99.0	0	7	99.0	0
okm08_12_w9	13	28	99.0	0	28	99.0	0
okm08_12_w10	14	5	99.0	0	5	99.0	0
okm08_12_w11	15	5	99.0	0	5	99.0	0
okm08_12_w12	16	109	99.0	0	109	99.0	0
okm08_12_w13	17	62	99.0	0	62	99.0	0
okm08_12_w14	18	254	8.0	11	254	15.0	1
okm08_12_w15	19	27	8.0	3	27	15.0	1
okm08_12_w16	20	207	8.0	2	207	99.0	0
ok08_12_back	21	31251	10.0	13	31520	10.0	4

14.6.6 Correlation of Metals

Figure 14-58 shows the correlation between copper and gold (excluding the background domain). Copper and gold are moderately correlated. This is an important relationship to establish as gold is estimated using the same search parameters as copper.

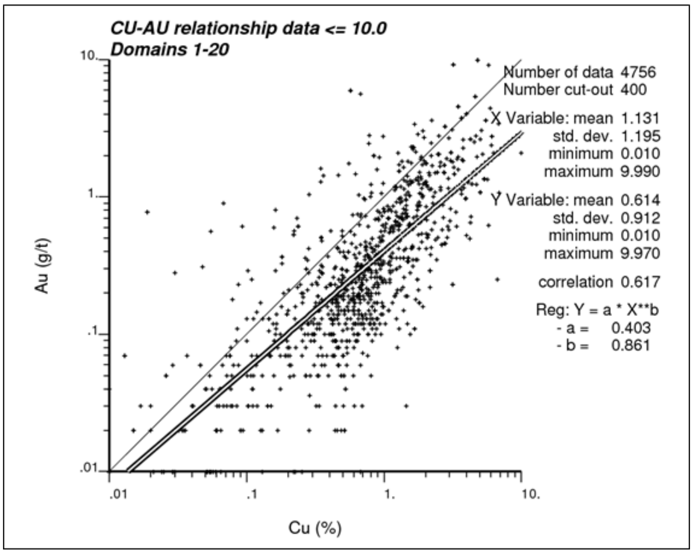


Figure 14-58: Correlation between copper and gold (mineralised domains)

14.6.7 Variography

Variogram modelling was done for domain “M” (okm08_12_main) as this was remodelled in 2012. There were no new data for the other domains. No variograms were modelled for the new background domain (okm08_12_back) as it contains a variety of structures over a very large volume.

The drillhole data are a mixture of surface and underground holes. In general, the surface holes intersect the mineralisation at low angles. This has the effect of introducing a short range structure in the down-dip direction. To avoid this, Relative by Pair (RLP) variograms were produced using only the underground drillhole data. This does bias the variograms to the area of underground drilling, but this is the main area of present economic interest.

Copper

A Spherical model was fitted to relative-by-pair variograms for domain “M” (Figure 14-59). The selected nugget effects were obtained from the down-the-hole variogram. The domain “M” variogram model was used for related domains 5-20 and the background domain.

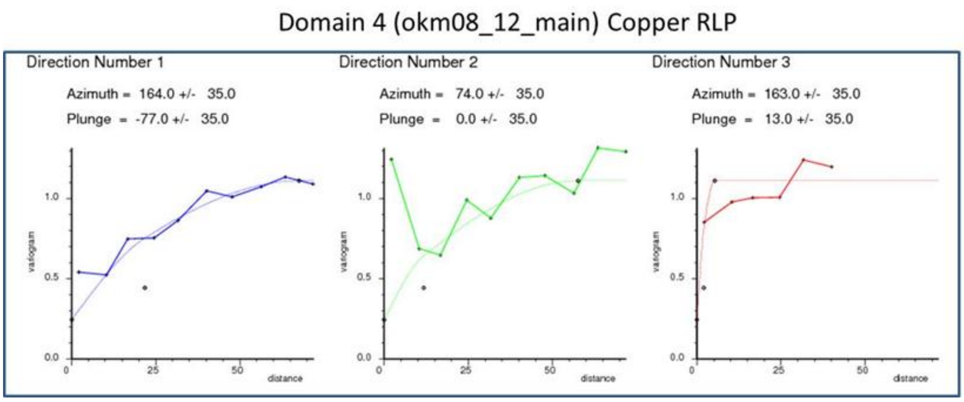


Figure 14-59: RLP variograms domain “M” copper

Gold

Gold was modelled in a similar manner to copper using the underground drillhole data (Figure 14-60). The Nugget effect was obtained from down-the-hole variograms. The domain "M" variogram model was used for related domains 5-20 and the background domain. The variogram ranges for gold were shorter than those modelled for copper. This is in general agreement with previous observations at Osborne and Kulthor: gold is more erratic and occurs in clusters.

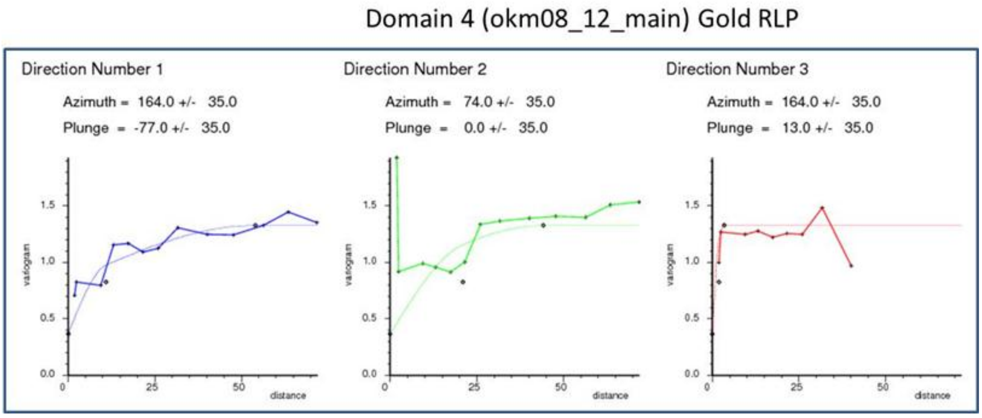


Figure 14-60: RLP variograms domain "M" gold

Density

Spherical models were fitted to correlograms for density data from the underground drillholes. The model for domain "M" was used for domains 5-21 as well as there were insufficient data for most of these domains.

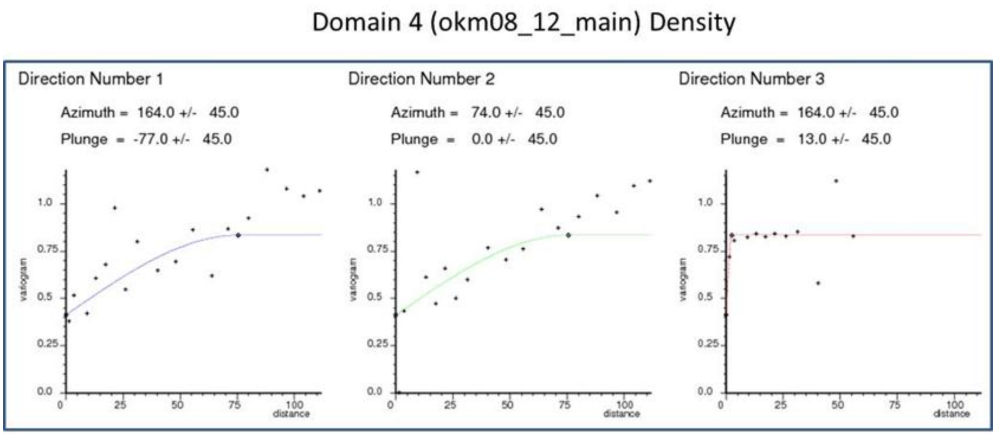


Figure 14-61: RLP variograms domain "M" density

The fitted models for copper, gold and density are shown in Table 14-40. Spherical models were used.

Table 14-40: Variogram models copper, gold density

Copper Variogram Model													
WF	VANGLE1	VANGLE2	VAXIS1	VAXIS2	NUGGET	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
okn12_11	163	80	3	1	0.484	19.7	6.39	2.92	0.669	111.0	41.0	24.0	0.571
okn05_11	163	80	3	1	0.484	19.7	6.39	2.92	0.669	111.0	41.0	24.0	0.571
okfn05	145	62	3	1	0.484	19.7	6.39	2.92	0.669	111.0	41.0	24.0	0.571
okm08_12_main	164	77	3	1	0.243	11.8	21.8	2.03	0.199	57.6	67.6	5.3	0.669
D5-21	164	77	3	1	0.243	11.8	21.8	2.03	0.199	57.6	67.6	5.3	0.669
Gold Variogram Model													
WF	VANGLE1	VANGLE2	VAXIS1	VAXIS2	NUGGET	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
okn12_11	163	80	3	1	0.555	7.76	4.8	2.70	0.726	102.0	26.2	19.50	0.447
okn05_11	163	80	3	1	0.555	7.76	4.8	2.70	0.726	102.0	26.2	19.50	0.447
okfn05	145	62	3	1	0.555	7.76	4.8	2.70	0.726	102.0	26.2	19.50	0.447
okm08_12_main	164	77	3	1	0.367	20.9	10.9	1.82	0.458	44.2	54.0	3.42	0.503
D5-21	164	77	3	1	0.367	20.9	10.9	1.82	0.458	44.2	54.0	3.42	0.503
Density Variogram Model													
WF	VANGLE1	VANGLE2	VAXIS1	VAXIS2	NUGGET	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
okn12_11	163	80	3	1	0.0073	50.1	2.67	3.20	0.0053	120.0	16.2	25.10	0.012
okn05_11	163	80	3	1	0.0073	50.1	2.67	3.20	0.0053	120.0	16.2	25.10	0.012
okfn05	145	62	3	1	0.0073	50.1	2.67	3.20	0.0053	120.0	16.2	25.10	0.012
okm08_12_main	164	77	3	1	0.412	75.5	75.5	2.45	0.422	-	-	-	-
D5-21	164	77	3	1	0.412	75.5	75.5	2.45	0.422	-	-	-	-

14.6.8 Block Model and Grade Estimation

The model parameters (primary blocks) are shown in Table 14-41. The extent of the model was increased to match the new wireframes and in particular, the new mineralisation intersected to the west. The twenty one wireframes were filled with blocks and sub-blocks and combined in priority order. The range of sub-block sizes is shown in Table 14-42. The maximum block sizes in the mineralised domains are smaller than the block sizes shown in Table 14-41, as there are no primary blocks used for the mineralised domains. This prevents there being a few very large blocks and many small blocks which can produce unrealistic grade distributions. The sub-blocking scheme adequately filled narrow domain wireframes.

Table 14-41: Model extents 2011and 2010

	2011 Model				2010 Model			
	Min	**Max**	**Size**	**Nb**	**Min**	**Max**	**Size**	**Nb**
X	8900	10750	5	370	8450	11020	5	514
Y	22750	23650	5	180	22390	23960	5	314
Z	450	1225	5	155	320	1265	5	189

Table 14-42: Sub-blocking

Sub-Blocking Mineralization		
	Min	Max
X	0.125	1.25
Y	1.25	2.5
Z	1.25	2.5

Sub-Blocking Background		
	Min	Max
X	0.125	5
Y	1.25	5
Z	1.25	5

A regular block model prototype (no sub-blocking) was also built as this is required for some validation tests.

14.6.9 Use of Dynamic Anisotropy

Dynamic Anisotropy modelling was used to handle the minor kinks in the Kulthor mineralised zones. In this technique, each block has a unique search orientation. The model was built by digitising strings on levels and in parallel sections approximately normal to strike (Figure 14-62). The strings were conditioned to short segments (20 m) and then processed using the Datamine process ANISOANG into points with dip direction and apparent dip fields. These points were estimated into blocks using the Nearest Neighbour method (NN) under domain control. Soft boundaries were used between domains 1 and 2 and between domain 21 and all other domains. After estimation, the apparent dips were converted to true dips for all blocks that had true dip direction estimates.

The estimated dynamic anisotropy model was used as the model prototype for grade estimation. Both regular and sub-block dynamic anisotropy model prototypes were built.

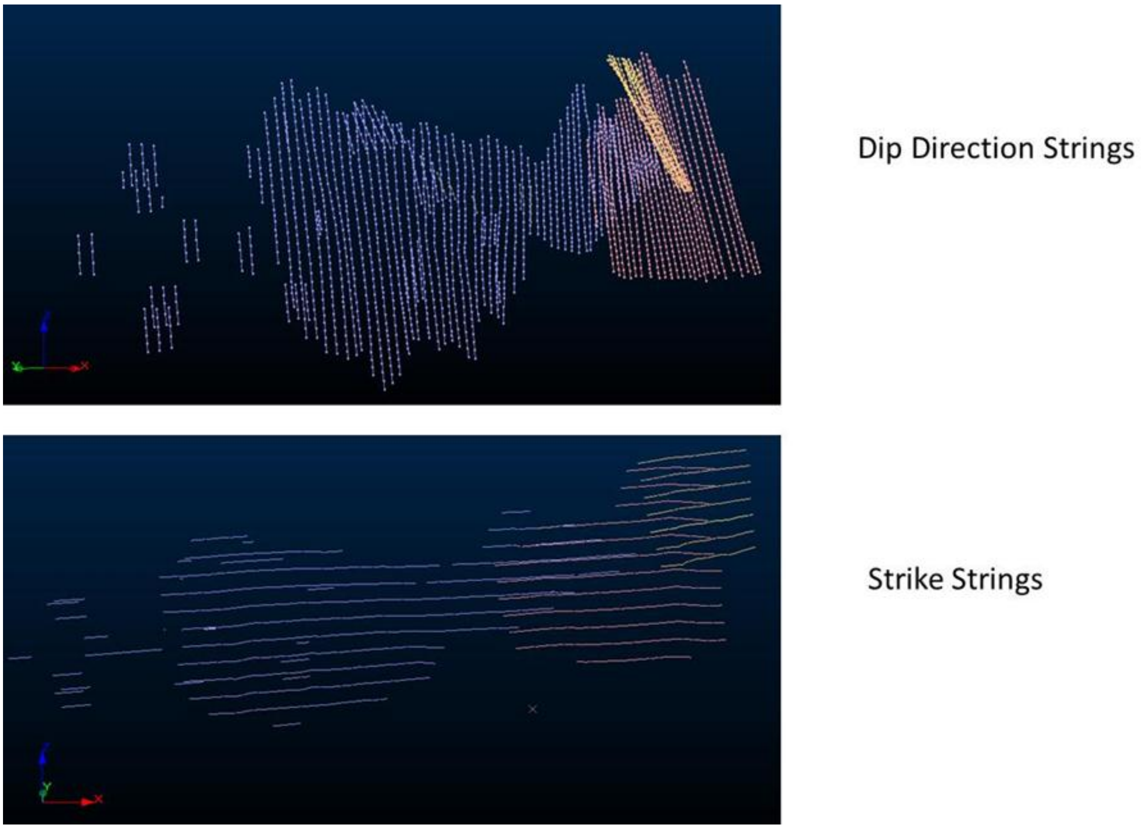


Figure 14-62: Dynamic Anisotropy digitised dip and strike strings

14.6.10 Estimation

Estimation of copper, gold and density was carried out using the Datamine ESTIMA process in a macro. This macro requires two special parameter files in addition to the three (search, estimation and variogram) parameter files required by the ESTIMA process. The additional files define top-cuts and boundary crossing (if any). Discretisation of 2 m x 2 m x 2 m was used when estimating blocks and sub-blocks. All boundaries were hard.

The estimation methods used were ordinary Kriging, ID^3 and NN. The latter two estimation methods were used to validate the Kriged estimate.

Several estimation runs were completed:

- With and without Dynamic Anisotropy;
- Parent block and sub-block estimation (with parent block estimation, all sub-blocks in a parent block receive the same grade); and
- Regular block estimation (a regular block model is useful for model validation, even though it does not completely fill the domain wireframes).

The search parameters are shown in Table 14-43. The same search parameters were used for copper and gold. Estimation of density required a larger search as there were fewer density data. The ID3 and NN estimates used the kriging search parameters. Three search passes were used.

The top-cuts were shown in Table 14-39.

Table 14-43: Search parameters

Search Parameter Cu Au													
WF	SDIST1	SDIST2	SDIST3	SANGLE1	SANGLE2	MINNUM1	MAXNUM1	SVOLFAC2	MINNUM2	MAXNUM2	SVOLFAC3	MINNUM3	MAXNUM3
okn12_11	60	41	17.6	163	80	5	15	2	5	15	3	5	15
okn05_11	60	41	17.6	163	80	5	15	2	5	15	3	5	15
okfn05	60	41	17.6	145	62	5	15	2	5	15	3	5	15
okm08_12_main	60	41	10	169	81	5	15	2	5	15	3	5	15
D5-21	60	41	10	169	81	5	15	2	5	15	3	5	15
Search Parameters Density													
WF	SDIST1	SDIST2	SDIST3	SANGLE1	SANGLE2	MINNUM1	MAXNUM1	SVOLFAC2	MINNUM2	MAXNUM2	SVOLFAC3	MINNUM3	MAXNUM3
okn12_11	75	75	10	163	80	5	15	2.5	5	15	4	5	15
okn05_11	75	75	10	163	80	5	15	2.5	5	15	4	5	15
okfn05	75	75	10	145	62	5	15	2.5	5	15	4	5	15
okm08_12_main	75	75	10	169	81	5	15	2.5	5	15	4	5	15
D5-21	75	75	10	169	81	5	15	2.5	5	15	4	5	15

14.6.11 Density

Density (field SG) has been measured on many of the assayed samples. The domain averages of the declustered data were used as defaults for any blocks that lacked an estimated density. The domain okm08_12_main density default was used for the subsidiary "M" domains and for the background domain as were few density measurements for these domains.

Table 14-44 summarises the default values were used where blocks lacked an estimated "DENSITY" values.

Table 14-44: Default density values

Wireframe	DOM	Default Density
kul11box (base Oxidation)		2.71
okn12_11	1	3.10
okn05_11	2	2.99
okfn05	3	2.86
okm08_12_main	4	3.15
okm08_12_w1 to okm08_12_w16	5-20	3.15
okm08_12_back	21	3.15

14.6.12 Mineral Resource Classification

The same basic classification scheme used in 2011 was used to allow direct comparison with the published Mineral Resources. This used Kriging Variance of the copper estimates to classify the Mineral Resources. The classification scheme was further modified:

- All measured resources had to be estimated using a minimum of 10 samples and be estimated in the first pass (SVOLCU=1);
- All blocks estimated in the third search pass were classified as Inferred (SVOLCU=3);

- All estimates for domains 5-21 were classified as Inferred as the controls of this mineralisation are poorly understood;
- Domain 21 Inferred was further restricted to those blocks with a kriging variance (KVARCU) < 0.6. The remaining material for this domain was left in the model (denoted as CLASS=4). This further restriction was added as this is a large unconfined domain; and
- Further drilling and/or geological interpretation may allow some of the Inferred Mineral Resource material to receive a higher classification.

14.6.13 Model Validation and Sensitivity

The new model was validated using several procedures: (i) Model Walk-Through, (ii) Comparison of estimates made using different estimation methods, (iii) Comparison with the 2011 model, (iv) Comparison of kriged and ID3 estimates, (v) Trend plots, (vi) Variability checks, (vii) Parent block estimation check and (viii) Dynamic Anisotropy check.

Model Walk-Through

The estimated model was viewed in section and plans in relation to the drillhole data to check that the model grades appeared correct. The estimates appeared to be satisfactory.

Comparison of Estimates made with different Estimation Methods

The copper and gold estimates made by Kriging were compared with ID3 and NN estimates above a zero cut-off. The cut-off needs to be zero as the variability of the other estimates (especially the NN estimates) is different. The comparison was limited to the mineralised domains (Table 14-45, Table 14-46). The tonnages are shown as percentages of the total estimated tonnes, as these are not Mineral Resources. The results do not indicate any significant bias. The average grade of the declustered top-cut composites is also shown.

Table 14-45: Comparison of copper estimates

Sub-Block Model CU >= 0.0%					
DOM	% Tonnes	Cu% Kr	Cu% ID3	Cu% NN	Cu% Cmp
1	1.9	1.7	1.7	1.6	1.7
2	35.6	0.6	0.6	0.5	0.6
3	15.2	0.6	0.6	0.5	0.7
4	39.0	1.3	1.3	1.3	1.2
6	0.3	0.6	0.5	0.6	0.5
7	0.2	1.1	1.0	1.1	1.0
8	0.5	1.0	1.0	1.0	1.0
9	0.4	0.8	0.8	0.8	0.8
10	0.2	1.9	1.9	2.0	1.9
11	0.3	2.1	2.0	2.3	2.0
12	0.2	2.1	2.3	1.7	2.3
13	0.2	1.2	1.2	1.2	1.2
14	0.0	3.3	3.5	3.4	3.4
15	0.0	1.9	1.8	1.9	1.9
16	2.7	0.9	1.0	0.9	0.9
17	0.6	0.7	0.7	0.8	0.7
18	2.3	1.2	1.2	1.2	1.4
19	0.1	3.0	3.1	3.0	3.5
20	0.3	0.8	0.9	0.9	0.9

Table 14-46: Comparison of gold estimates

Sub-Block Model AU >= 0.0g/t					
DOM	% Tonnes	Au g/t Kr	Au g/t ID3	Au g/t NN	Au g/t Cmp
1	1.9	1.0	1.0	1.0	1.0
2	35.6	0.3	0.3	0.3	0.4
3	15.2	0.3	0.3	0.2	0.3
4	39.0	0.8	0.8	0.8	0.7
6	0.3	0.4	0.4	0.5	0.4
7	0.2	2.2	2.0	2.4	2.0
8	0.5	0.6	0.6	0.6	0.6
9	0.4	0.8	0.8	0.9	0.8
10	0.2	1.0	1.0	1.0	1.0
11	0.3	0.8	0.8	0.8	0.9
12	0.2	2.1	2.5	1.2	2.6
13	0.2	0.7	0.7	0.7	0.7
14	0.0	3.2	3.6	3.2	3.5
15	0.0	0.5	0.5	0.5	0.5
16	2.7	0.8	0.9	0.9	0.8
17	0.6	0.5	0.5	0.5	0.5
18	2.3	0.7	0.7	0.8	0.9
19	0.1	1.1	1.1	1.3	1.3
20	0.3	0.4	0.4	0.4	0.4

Comparison with 2011 model

The 2011 model used only 4 domains compared to the 21 in the 2012 model. Domain 4 of the 2012 model is similar to domain 3 of the 2011 model. The principal change was an increase in total Measured and Indicated Mineral Resources in 2012 and a small reduction in Inferred Mineral Resources (Table 14-47). The proportion of Measured Mineral Resources increased because of the increased drilling. Copper grades increased slightly but gold was unchanged. The 2012 estimates have not been depleted for mining for this comparison.

Table 14-47: Comparison of 2012 and 2011 Kulthor Mineral Resources (not depleted)

GLOBAL COMPARISON OF RESOURCES (Aug 2012 - 2011)										
					Grade			Metal		
Aug 2012	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m^3	%	%	g/t	000't	000't	000'oz
Cut off >= 1.2% ECU		Measured	3.0	3.1	2.3	1.7	1.0	68.9	50.3	99.7
		Indicated	4.5	3.0	2.1	1.5	1.0	93.9	68.0	138.7
	Sub-Total		**7.5**	**3.0**	**2.2**	**1.6**	**1.0**	**162.8**	**118.3**	**238.3**
		Inferred	5.4	3.0	1.9	1.3	0.9	100.5	72.8	148.2

2011	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m^3	%	%	g/t	000't	000't	000'oz
Cut off >= 1.2% ECU		Measured	0.3	3.1	2.2	1.6	1.0	7.3	5.3	10.8
		Indicated	4.3	3.1	2.1	1.5	1.0	90.0	65.2	132.9
	Sub-Total		**4.7**	**3.1**	**2.1**	**1.5**	**1.0**	**97.3**	**70.5**	**143.6**
		Inferred	5.5	3.1	1.7	1.2	0.8	96.1	68.6	147.7

	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
2012 - 2011 Difference			Mt	t/m^3	%	%	g/t	000't	000't	000'oz
		Measured	2.6	0.0	0.1	0.1	0.0	61.6	45.0	88.9
		Indicated	0.2	0.0	0.0	0.0	0.0	3.9	2.8	5.8
	Sub-Total		2.8	0.0	0.1	0.1	0.0	65.5	47.8	94.7
		Inferred	-0.1	-0.1	0.1	0.1	0.0	4.3	4.2	0.5

	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
2012 - 2011 Percentage Difference		Measured	805.2%	0.4%	4.6%	5.5%	2.4%	846.9%	854.6%	826.6%
		Indicated	3.6%	-1.1%	0.7%	0.7%	0.8%	4.4%	4.3%	4.4%
	Sub-Total		60.1%	-0.4%	4.5%	4.9%	3.7%	67.3%	67.9%	65.9%
		Inferred	-2.4%	-2.9%	7.0%	8.7%	2.8%	4.5%	6.2%	0.4%

(ECU=CU% + AUg/t * 0.6)

Comparison of Kriged and ID^3 Estimates

Table 14-48 and Figure 14-63 compare the Kriged and ID^3 copper estimates for blocks that are Measured or Indicated Mineral Resource as a function of cut-off grade (Inferred Mineral Resource blocks are excluded). As expected the ID^3 estimates are slightly higher grade than the Kriged estimates at most cut-offs (usual relationship).

The figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of grades to the estimation method.

Table 14-48: Comparison of Kriged and ID^3 copper estimates

Cut-Off Cu%	T x 1000 Kr	CU% Kr	T x 1000 ID3	CU% ID3
0.0	13471.7	1.1	13471.7	1.1
0.1	13430.4	1.1	13390.1	1.1
0.2	13263.9	1.1	13108.8	1.2
0.3	12861.9	1.2	12624.7	1.2
0.4	12173.5	1.2	11906.3	1.2
0.5	11226.6	1.3	10957.9	1.3
0.6	10206.8	1.4	9883.0	1.4
0.7	9222.5	1.4	8919.4	1.5
0.8	8331.3	1.5	8056.0	1.6
0.9	7470.7	1.6	7250.7	1.6
1.0	6610.2	1.7	6477.3	1.7
1.1	5839.8	1.8	5659.1	1.8
1.2	5092.1	1.8	4900.9	1.9
1.3	4459.0	1.9	4329.7	2.0
1.4	3873.2	2.0	3794.8	2.1
1.5	3372.0	2.1	3333.0	2.2
1.6	2901.7	2.2	2935.3	2.3
1.7	2495.2	2.3	2577.3	2.4
1.8	2146.9	2.4	2257.3	2.5
1.9	1792.0	2.5	1957.9	2.5
2.0	1472.2	2.6	1694.9	2.6

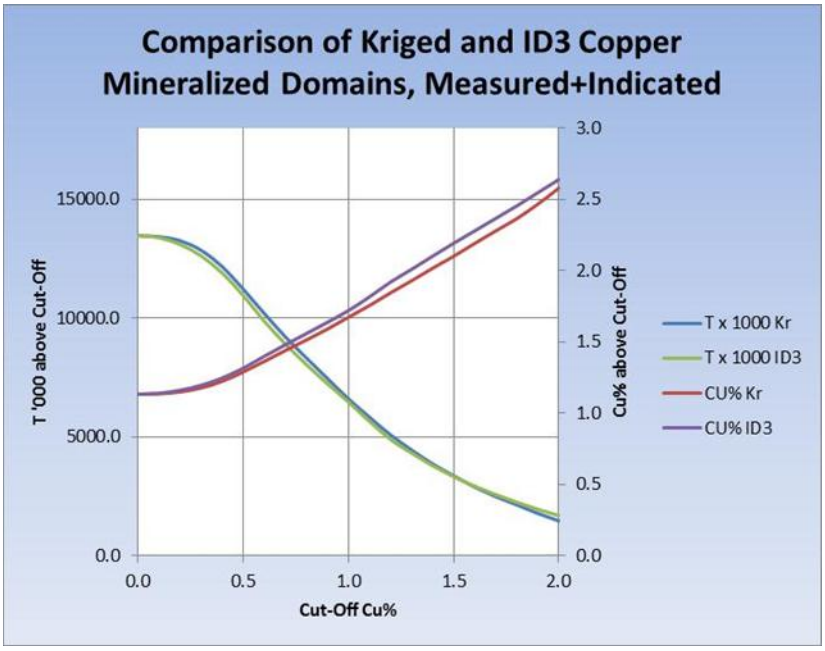


Figure 14-63: Comparison of Kriged and ID^3 copper estimates

Table 14-49 and Figure 14-64 compare the Kriged and ID^3 gold estimates for blocks that are Measured and Indicated Mineral Resource (Inferred Mineral Resource blocks are excluded). At high gold cut-offs, the ID^3 gold estimates are relatively higher the Kriged estimates compared to the copper relationship.

Table 14-49: Comparison of Kriged and ID^3 gold estimates

Cut-Off Au g/t	T x 1000 Kr	AU g/t Kr	T x 1000 ID3	AU g/t ID3
0.0	13471.7	0.7	13471.7	0.7
0.1	13072.8	0.7	12836.0	0.7
0.2	11191.1	0.8	10876.0	0.8
0.3	9729.3	0.9	9346.5	0.9
0.4	8327.3	0.9	8034.3	1.0
0.5	7113.2	1.0	6897.3	1.1
0.6	6131.8	1.1	5902.4	1.2
0.7	5120.6	1.2	4886.0	1.3
0.8	4200.5	1.3	4017.2	1.4
0.9	3394.7	1.4	3343.3	1.5
1.0	2754.9	1.5	2812.7	1.6
1.1	2229.3	1.6	2384.4	1.7
1.2	1867.7	1.7	2031.2	1.8
1.3	1569.4	1.8	1710.5	1.9
1.4	1307.0	1.9	1448.8	2.0
1.5	1091.4	2.0	1230.0	2.1
1.6	904.4	2.1	1034.0	2.2
1.7	742.5	2.2	861.1	2.3
1.8	609.4	2.3	724.2	2.4
1.9	509.8	2.4	605.8	2.5
2.0	422.4	2.5	506.3	2.6

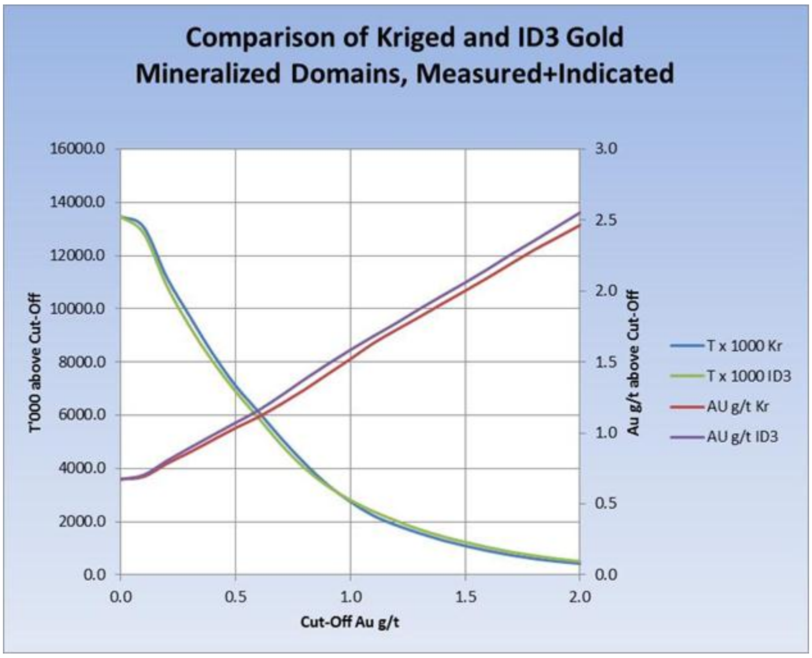


Figure 14-64: Comparison of Kriged and ID^3 gold estimates

Trend Plots

Trend or swath plots compare the average grade of blocks (the estimated grade and the average of the top-cut drillhole composites in the estimated blocks) by coordinate direction. The trend plots for copper by northing and elevation are shown in Figure 14-65. The plot by easting is not shown as this direction is semi-parallel to the strike of the mineralisation and results are erratic. The estimates are very similar to the block-averaged top-cut composites; especially where the number of blocks (field COUNT) is high.

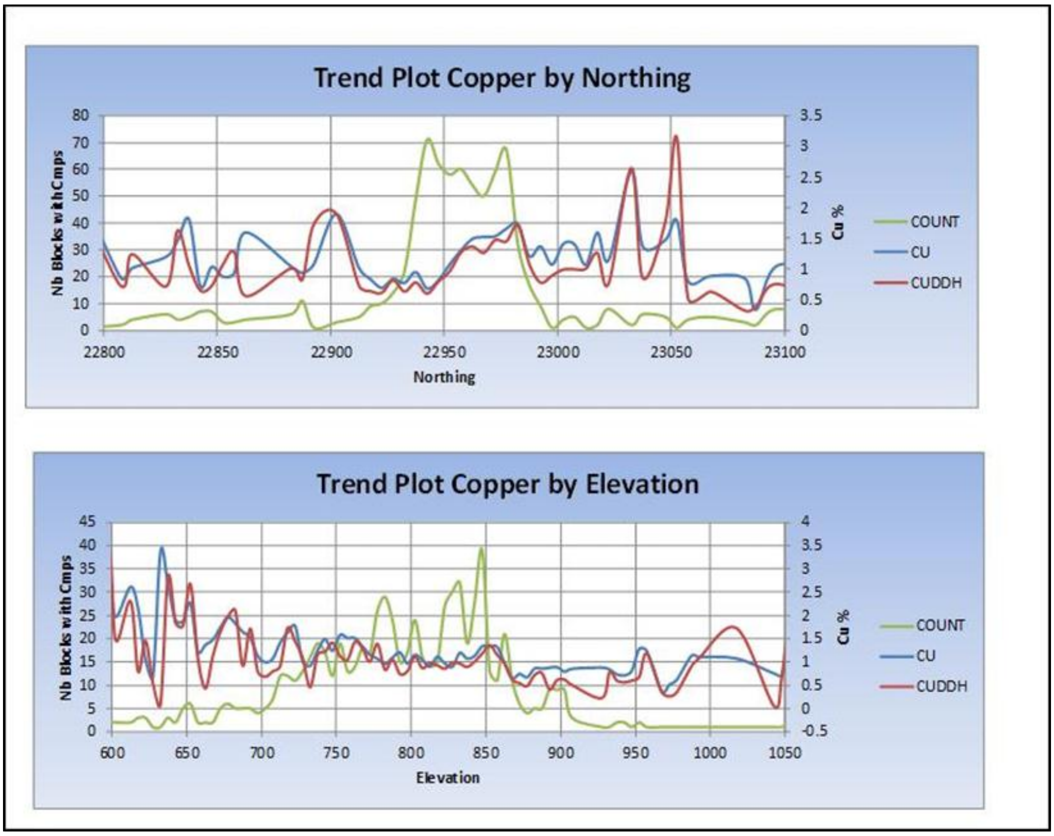


Figure 14-65: Trend Plots for copper

Variability Checks

The variability of the block model needs to be correct, both locally and globally, otherwise the tonnes and grade above practical cut-offs will be incorrect.

The local variability was checked by comparing the estimated grades with the local average drillhole grades. The drillhole composites were top-cut then averaged into regular blocks and merged with a regular block resource model. The grades were compared using a scatterplot (Figure 14-66). Both the grades and the correlation coefficient were satisfactory, indicating a lack of local bias and the model has the correct amount of local variability.

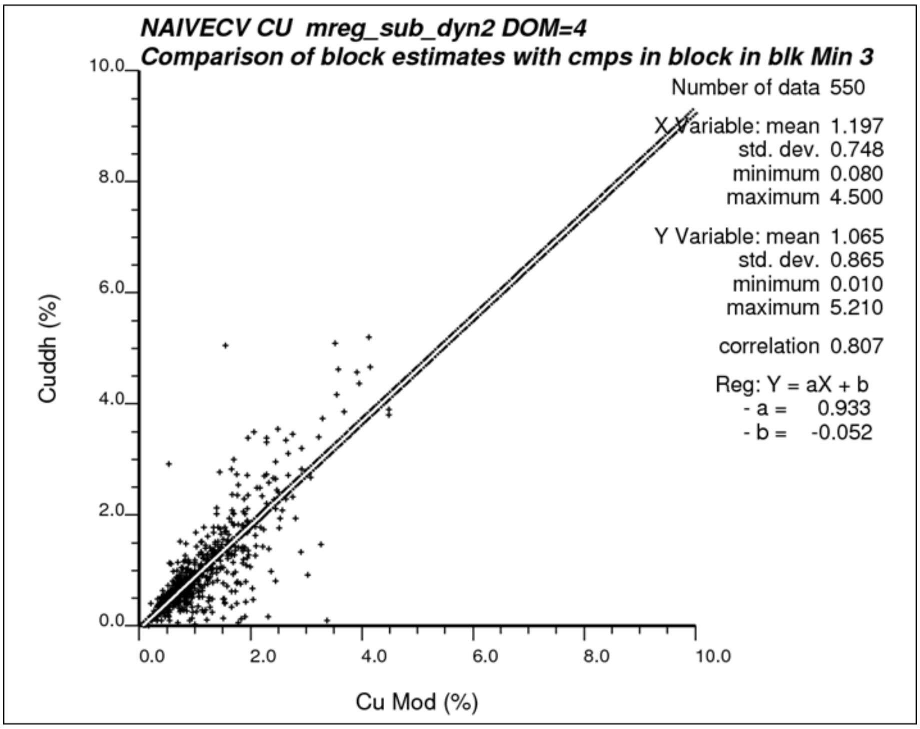


Figure 14-66: Comparison of copper estimates with composites in blocks

The global variability of the model for the most important mineralised zone (okm08_12_main) was compared with the theoretical variability calculated from the composites and the declustering weights using the Indirect Log-Normal Correction. This method uses the variogram model and the block dimensions to adjust grades and declustering weights of the composites to that of a model. The study was restricted to the okm08_12_main domain as this had the most reliable variogram model.

The expected CV for copper in zone okm08_12_main is 0.636. The estimated model will have about the right amount of smoothing if its CV is 85% of this value or 0.538. The CV of the model for zone okm08_12_main is 0.534 (<1% lower), indicating that the model has the correct amount of global variability. This is a satisfactory result. The CV of the ID^3 estimate was 0.586, significantly higher than the calculated value.

Parent Block Estimation Check

The new model was compared with a sub-block model estimated with parent block estimation (all sub-blocks within a parent block receive the same grade). This is a useful test of the sensitivity of the resources to use of small sub-blocks. Parent block estimation is similar to regular block estimation, but the boundaries of the mineralisation zones are better honoured. The average grade of the parent block estimates (ECU >= 1.2%) was 1% lower than the grade of the model with sub-block estimation. The tonnage of the parent block model was 1.2% higher. A model estimated using only regular blocks (no sub-blocks) has tonnes and grade intermediate between those of the sub-block and parent block models.

Dynamic Anisotropy Check

A model was estimated without use of Dynamic Anisotropy. Tonnes and grades were less than 1.5% different for the Measured and Indicated Resources.

14.6.14 Removal of Mined-out areas

Mining of the Kulthor mineralisation commenced during 2012, but as of the Effective Date (5 September, 2012) consisted only of development on mineralisation. The Mineral Resources were depleted for this development using the Datamine TRIVAL process. The calculated production cannot be compared with actual production till the ore is processed.

Table 14-50: Calculated Kulthor production to 5 September 2012

PRODUCTION to 5th September 2012										
					Grade			Metal		
Aug 2012 Mined	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m^3	%	%	g/t	000't	000't	000'oz
Cut off >= 1.2% ECU		Measured	0.1	3.1	2.3	1.7	1.1	1.9	1.4	2.9
		Indicated	0.0	3.0	2.4	1.6	1.3	0.6	0.4	1.0
	Sub-Total		**0.1**	**3.1**	**2.4**	**1.7**	**1.1**	**2.5**	**1.8**	**3.9**
		Inferred	0.001	3.0	1.6	1.3	0.6	0.0	0.0	0.0

14.6.15 Kulthor Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilised organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The classified Mineral Resources for Kulthor are shown in Table 14-51. The 2012 depleted Mineral Resources are compared to 2011 Mineral Resources in Table 14-52. The additional drilling has increased the Measured and Indicated Mineral Resources as well as increasing the proportion of Measured Mineral Resources. The copper grade has increased slightly, while gold is unchanged.

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-51: Kulthor Classified Mineral Resource Estimates

Item	Cut-off Grade eCu (%)	Tonnes (Mt)	Grade			Metal	
			eCu (%)	Cu (%)	Au (g/t)	Cu (000' t)	Au (000' oz)
Measured Resource	1.2	2.9	2.3	1.7	1.0	48.9	96.7
Indicated Resource		4.5	2.1	1.5	1.0	67.6	137.7
Total Measured and Indicated Resource	**1.2**	**7.4**	**2.2**	**1.6**	**1.0**	**116.5**	**234.4**
Inferred Resource	1.2	5.4	1.9	1.3	0.9	72.8	148.2

1 eCu = copper (%) + gold (g/t) x 0.6.

2 The Mineral Resource Estimate is effective as at 5 September 2012.

3 The Mineral Resource Estimate has been prepared by Richard Lewis, FAusIMM, a full-time employee of LMRC Consulting, who is a qualified person as defined by NI 43-101.

4 Some totals may not add due to the effects of rounding.

Table 14-52: Comparison of 2012 depleted Mineral Resources with 2011 Mineral Resources

GLOBAL COMPARISON OF RESOURCES (Aug 2012 - 2011)										
					Grade			Metal		
Aug 2012 Depleted	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m³	%	%	g/t	000't	000't	000'oz
Cut off >= 1.2% ECU		Measured	2.9	3.1	2.3	1.7	1.0	66.9	48.9	96.7
		Indicated	4.5	3.0	2.1	1.5	1.0	93.3	67.6	137.7
	Sub-Total		**7.4**	**3.0**	**2.2**	**1.6**	**1.0**	**160.3**	**116.5**	**234.4**
		Inferred	5.4	3.0	1.9	1.3	0.9	100.5	72.8	148.2

2011	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m³	%	%	g/t	000't	000't	000'oz
Cut off >= 1.2% ECU		Measured	0.3	3.1	2.2	1.6	1.0	7.3	5.3	10.8
		Indicated	4.3	3.1	2.1	1.5	1.0	90.0	65.2	132.9
	Sub-Total		**4.7**	**3.1**	**2.1**	**1.5**	**1.0**	**97.3**	**70.5**	**143.6**
		Inferred	5.5	3.1	1.7	1.2	0.8	96.1	68.6	147.7

2012 - 2011 Difference	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
			Mt	t/m³	%	%	g/t	000't	000't	000'oz
		Measured	2.6	0.0	0.1	0.1	0.0	59.7	43.6	86.0
		Indicated	0.1	0.0	0.0	0.0	0.0	3.3	2.4	4.8
	Sub-Total		**2.7**	**0.0**	**0.1**	**0.1**	**0.0**	**63.0**	**46.0**	**90.8**
		Inferred	-0.1	-0.1	0.1	0.1	0.0	4.3	4.2	0.5

2012 - 2011 Percentage Difference	DOM	CLASS	QUANTITY	DENSITY	ECU	CU	AU	ECU_Metal	Cu_Metal	Au_Metal
		Measured	780.3%	0.4%	4.6%	5.5%	2.2%	820.5%	828.5%	799.3%
		Indicated	3.0%	-1.1%	0.7%	0.7%	0.6%	3.7%	3.7%	3.6%
	Sub-Total		**57.8%**	**-0.4%**	**4.4%**	**4.8%**	**3.4%**	**64.8%**	**65.3%**	**63.2%**
		Inferred	-2.4%	-2.9%	7.0%	8.7%	2.8%	4.5%	6.1%	0.3%

The Measured and Indicated Mineral Resources are 98% Fresh and the Inferred Resources are 91% Fresh.

14.6.16 Grade Sensitivity Analysis

The Mineral Resources of Kulthor are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model unmined quantities and grade of blocks classified as Measured and Indicated are presented in Figure 14-67 at different cut-off grades.

The figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Cut-Off	000t	ECU%
0.0	13471.7	1.5
0.1	13459.1	1.5
0.2	13350.5	1.6
0.3	13196.5	1.6
0.4	12869.9	1.6
0.5	12369.1	1.6
0.6	11667.7	1.7
0.7	10778.5	1.8
0.8	9956.7	1.9
0.9	9274.4	2.0
1.0	8665.2	2.0
1.1	8053.5	2.1
1.2	7474.0	2.2
1.3	6958.2	2.2
1.4	6440.1	2.3
1.5	5900.0	2.4
1.6	5381.5	2.5
1.7	4891.5	2.6
1.8	4400.2	2.7
1.9	3937.8	2.7
2.0	3527.5	2.8

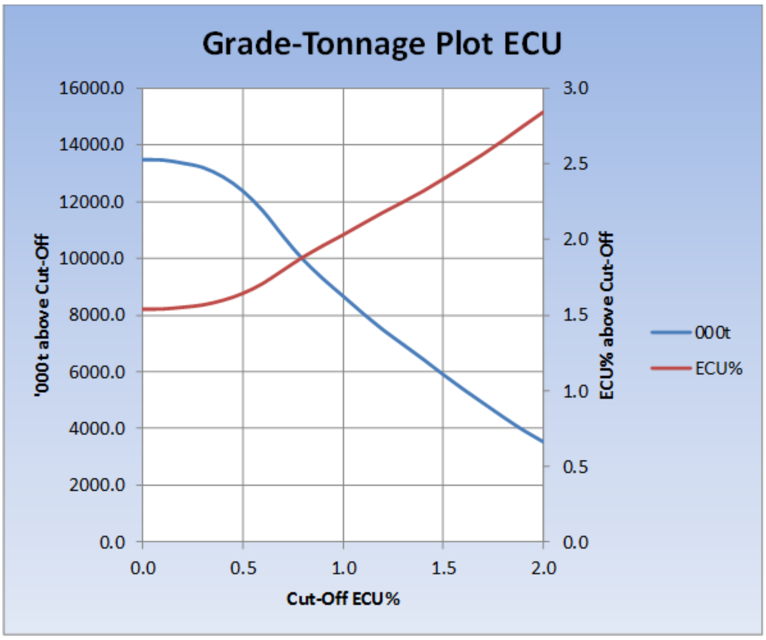


Figure 14-67: Kulthor Grade sensitivity mineralised domains (Measured and Indicated Mineral Resource)

14.6.17 Sensitivity of the Kulthor Mineral Resource to Other Factors

The Kulthor Deposit is currently in production. There are no known environmental, permitting, legal, titles, socio-economic, marketing, political or other factors that could materially affect the Mineral Resource. Changes to metal prices, taxation, royalties and transport costs would affect the cut-off grade used for the Mineral Resources. The chosen cut-off grade for Kulthor is also dependent on achieving the planned metallurgical recoveries.

15 Mineral Reserve Estimates

The Mineral Reserve Estimates are a subset of the Mineral Resource, effective in the case of Osborne deposits herein and in the case of Kulthor, dated 27 October 2011, which can be found in Section 14.6.15. The assumptions and design basis for the Osborne Open Pit, Osborne Underground and Kulthor Underground are presented in Section 16. Table 15-1 shows the combined Mineral Reserve Estimate for the Osborne copper-gold project.

Table 15-1: Combined Mineral Reserve Estimate[(1),(2)]

Classification	Tonnes (Mt)	Copper Grade (%)	Gold Grade (g/t)	eCu[(3)] (%)	Contained Copper (t)	Contained Gold (ozs)
Proven						
Osborne Open Pit	2.4	0.83	0.57	1.17	19,920	43,982
Osborne Underground	0.5	1.93	0.90	2.47	9,742	14,602
Kulthor Underground	0	0	0	0	0	0
Total Proven	**2.9**	**1.02**	**0.63**	**1.39**	**29,662**	**58,584**
Osborne Open Pit	0.1	0.72	0.54	1.04	720	1,736
Osborne Underground	0	0	0	0	0	0
Kulthor Underground[(4)]	2.58	1.47	0.94	2.04	37,787	77,706
Total Probable	**2.68**	**1.44**	**0.93**	**2.00**	**38,507**	**79,442**
Total Mineral Reserve	**5.58**	**1.22**	**0.77**	**1.69**	**68,169**	**138,026**

(1) The Mineral Reserve is as at 1 June 2012.

(2) The Mineral Reserve has been prepared by Ms Anne-Marie Ebbels, MAusIMM (CP), an employee of SRK Consulting (Australasia) Pty Ltd, who is a qualified person as defined by NI43-101.

(3) eCu = copper (%) + gold (g/t) x 0.6.

(4) Based on 2011 Mineral Resource Estimate

15.1 Osborne Open Pit

The Mineral Reserve Estimate for the Osborne underground mine is shown in Table 15-2. The figures are inclusive of the modifying factors for mining recovery and dilution.

Table 15-2: Mineral Reserve Estimate for Osborne Open Pit[(1),(2)]

Classification	Tonnes (Mt)	Copper Grade (%)	Gold Grade (g/t)	eCu[(3)] (%)	Contained Copper (t)	Contained Gold (ozs)
Proven	2.4	0.83	0.57	1.17	19,920	43,982
Probable	0.1	0.72	0.54	1.04	720	1,736
Total Mineral Reserve	**2.5**	**0.82**	**0.57**	**1.16**	**20,640**	**45,718**

(1) The Mineral Reserve is as at 1 June 2012.

(2) The Mineral Reserve has been prepared by Ms Anne-Marie Ebbels, MAusIMM (CP), an employee of SRK Consulting (Australasia) Pty Ltd, who is a qualified person as defined by NI43-101.

(3) eCu = copper (%) + gold (g/t) x 0.6.

The pit has been designed to be mined in two stages, Cutback 1 and Cutback 2. Table 15-3 shows the split for the two stages of the open pit for the Mineral Reserve Estimate.

Table 15-3: Mineral Reserve Estimate for Osborne Open Pit by Cutback

Classification	Tonnes (Mt)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (t)	Contained Gold (ozs)
Cutback 1					
Proven	1.38	0.79	0.57	10,898	25,288
Probable	0.05	0.86	0.64	458	1,100
Cutback 1 Total	**1.44**	**0.79**	**0.57**	**11,356**	**26,388**
Cutback 2					
Proven	1.03	0.97	0.64	10,009	20,998
Probable	0.04	0.63	0.48	257	626
Cutback 2 Total	**1.07**	**0.96**	**0.63**	**10,267**	**21,624**
Total Mineral Reserve	**2.51**	**0.86**	**0.60**	**21,622**	**45,718**

Figure 15-1 shows the difference between the 2011 Preliminary Economic Assessment (PEA) mining inventory and the 2012 Mineral Reserve Estimate. There are an additional 0.5 Mt of ore, 7.6 kt of copper metal and 13,834 ozs of gold included in the Mineral Reserve Estimate from the re-optimisation of the open pit design.

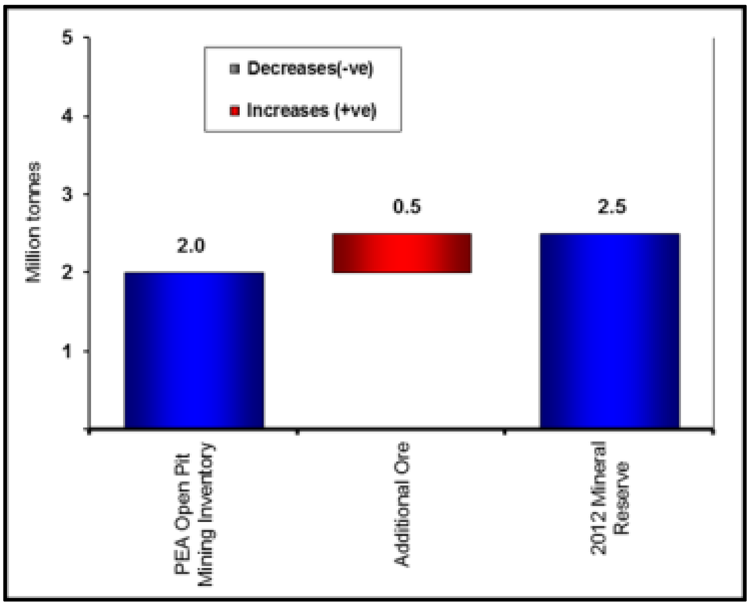


Figure 15-1: Difference between the 2011 PEA and 2012 Mineral Reserve Estimate for the Osborne Open Pit Mine

15.2 Osborne Underground

Mining of the Osborne Underground has commenced and the extracted ore tonnes have been removed from the Mineral Reserve. Table 15-4 shows the mined tonnes for the Osborne underground mine since August 2011.

Table 15-4: Osborne Underground Mine Tonnes

Classification	Tonnes (kt)	Copper (%)	Gold (g/t)	eCu (%)	Contained Copper (t)	Contained Gold (ozs)
Mined	120.7	1.80	0.86	2.31	2,172	3,337

The Mineral Reserve Estimate for the Osborne underground mine is shown in Table 15-5. The figures are inclusive of the modifying factors for mining recovery and dilution and exclude the mined ore in Table 15-4.

Table 15-5: Osborne Underground Mineral Reserve(1),(2)

Classification	Tonnes (kt)	Copper (%)	Gold (g/t)	eCu(3) (%)	Contained Copper (t)	Contained Gold (ozs)
Proven	504	1.93	0.90	2.47	9,742	14,602
Probable	-	-	-	-	-	-
Total Mineral Reserve	**504**	**1.93**	**0.90**	**2.47**	**9,742**	**14,602**

(1) The Mineral Reserve is as at 1 June 2012.

(2) The Mineral Reserve has been prepared by Ms Anne-Marie Ebbels, MAusIMM (CP), an employee of SRK Consulting (Australasia) Pty Ltd, who is a qualified person as defined by NI43-101.

(3) eCu = copper (%) + gold (g/t) x 0.6.

Figure 15-2 shows the difference between the 2011 PEA mining inventory and the 2012 Mineral Reserve Estimate. There are an additional 185 kt of ore, 986 t of copper metal and 2,153 ozs of gold included in the Mineral Reserve Estimate from the addition of one stope of each of the three levels. The development tonnes are calculated from actual designs using a 0.74 eCu cut-off whereas the 2011 PEA development ore only includes the development inside the stopes.

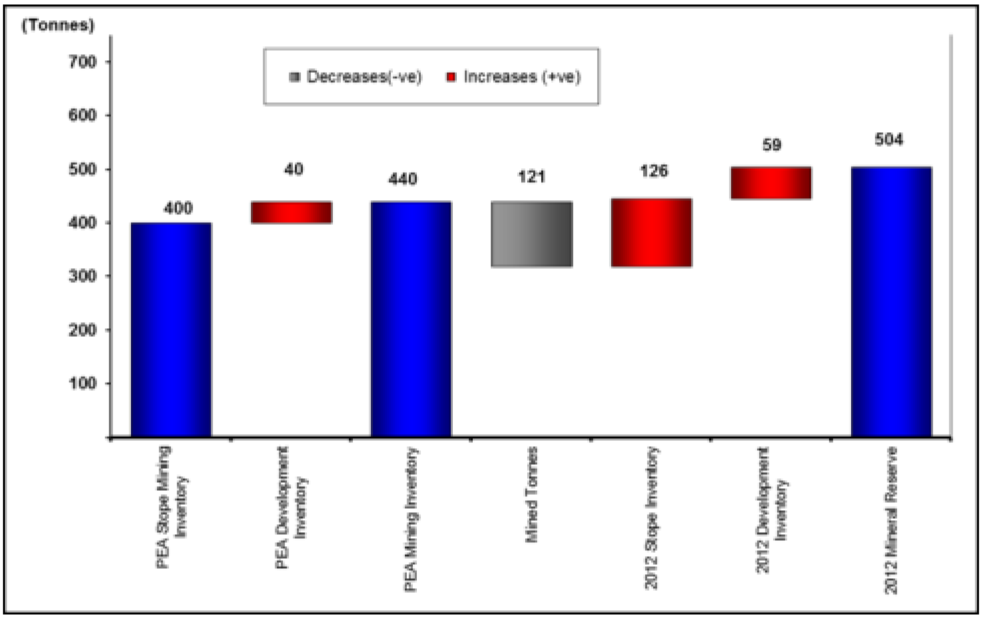


Figure 15-2: Difference between the 2011 PEA and 2012 Mineral Reserve Estimate for the Osborne Underground Mine

15.3 Kulthor Underground

No Mineral Reserves based on the Mineral Resource described in this Technical Report have been prepared at this time.

The current Mineral Reserves (effective date of 1 June 2012) for Kulthor are based on the previous Mineral Resource (effective date 27 October 2011, SRK, 2012). Information on the previous Kulthor Mineral Resource Estimate can be found in Section 14.6.15 and the previous Technical Report by SRK (SRK, 2012).

Mining of the development at Kulthor has commenced and the extracted ore tonnes have been removed from the Mineral Reserve. Table 15-6 shows the mined tonnes for the Kulthor underground mine.

Table 15-6: Kulthor Mine Tonnes as at 1 June 2012

Classification	Tonnes (kt)	Copper (%)	Gold (g/t)	eCu (%)	Contained Copper (t)	Contained Gold (ozs)
Mined	97	1.20	0.76	1.66	1,165	2,375

The Mineral Reserve Estimate for the Kulthor underground mine, at 1.4% eCu cut-off, is shown in Table 15-7. The figures are inclusive of the modifying factors for mining recovery and dilution and exclusive of the mined ore in Table 15-6.

Table 15-7: Kulthor Underground Mineral Reserve(1),(2),(3),(4)

Classification	Tonnes (Mt)	Copper (%)	Gold (g/t)	eCu (%)	Contained Copper (t)	Contained Gold (ozs)
Proven	-	-	-	-	-	-
Probable	2.58	1.47	0.94	2.04	37,787	77,706
Total Mineral Reserve	**2.58**	**1.47**	**0.94**	**2.04**	**37,787**	**77,706**

(1) The Mineral Reserve is as at 1 June 2012.

(2) The Mineral Reserve has been prepared by Ms Anne-Marie Ebbels, MAusIMM (CP), an employee of SRK Consulting (Australasia) Pty Ltd, who is a qualified person as defined by NI43-101.

(3) eCu = copper (%) + gold (g/t) x 0.6.

(4) Based on 2011 Mineral Resource Estimate

The Mineral Reserve has included rib and crown pillars that were not in the PEA design, the tonnes in these pillars shown in Table 15-8. The tonnes in the pillars are not included in the Mineral Reserve Estimate. Figure 15-3 shows the difference between the 2011 PEA mining inventory and the 2012 Mineral Reserve Estimate. There are an additional 0.18 Mt of ore, 1,907 t of copper metal and 4,145 ozs of gold included in the Mineral Reserve Estimate from the additional stopes to the north and additional level below 570 mRL.

Table 15-8: Ore included in Kulthor Pillars

Pillar Type	Tonnes (Mt)	Copper (%)	Gold (g/t)	eCu (%)	Contained Copper (t)	Contained Gold (ozs)
Crown	0.24	1.61	0.98	2.20	4.035	7,925
Rib	0.89	1.67	1.07	2.31	14,720	30,359
Total Pillar Inventory	**1.13**	**1.65**	**1.05**	**2.28**	**18,755**	**38,284**

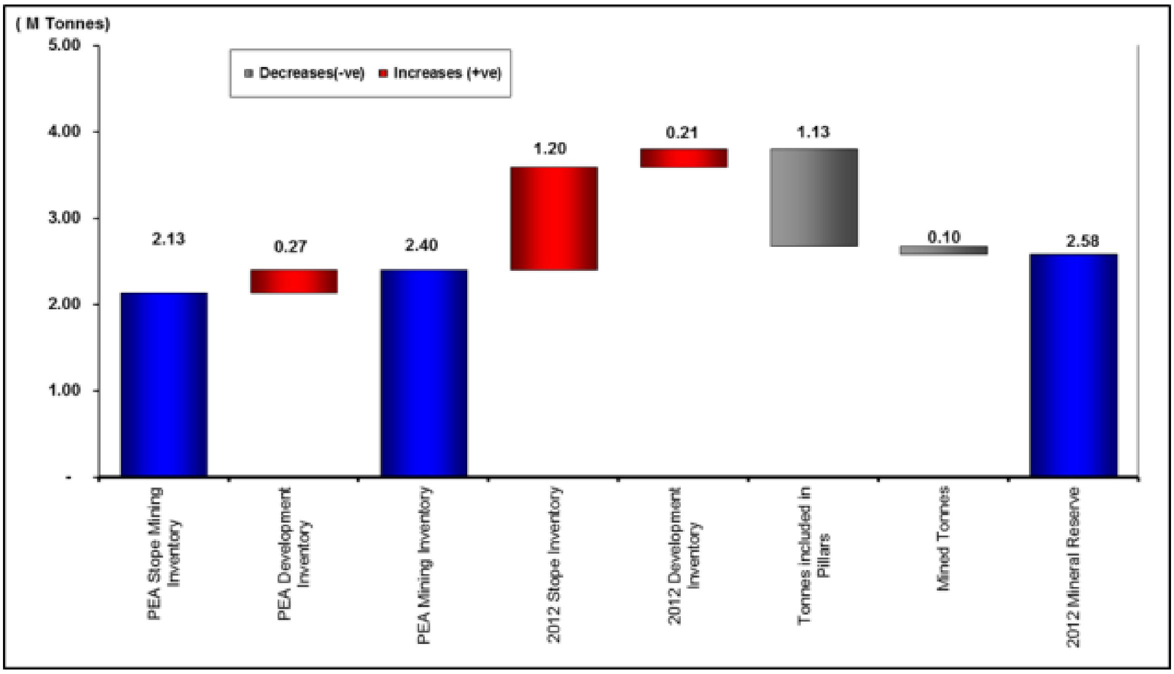


Figure 15-3: Difference between the 2011 PEA and 2012 Mineral Reserve Estimate for the Kulthor Underground Mine

16 Mining Methods

16.1 Osborne Open Pit

16.1.1 Introduction

Ivanhoe are proposing to extend the current Osborne open pit via a cutback to south-west of the current workings. The open pit deposit is a copper-gold mineralisation of approximately 2.0 Mt ore grading 0.7% copper and 0.5 g/t gold.

Open pit mining at Osborne is to be completed using standard open pit mining methods - drill and blast followed by load and haul. This was previously undertaken at Osborne with the use of contractors from 1995 to 1996 when operations focused on underground mining only.

Current open pit operations lie to the northeast of the underground operations, with a portal accessing the underground operation currently at the 1200 mRL within the open pit operation. As part of the open pit expansion, underground access is required to be maintained while the underground operations continue. For this reason, the open pit will be scheduled in two stages. This will allow an initial cutback, maintaining the underground access with a second final cutback to complete the pit once underground access is no longer required. The underground operation can also be accessed for personnel and smaller materials via the main shaft.

All open pit earthmoving activities are to be performed by third party contractors.

Ore from the pit is to be hauled approximately 800 m to a ROM pad located adjacent to the crushing facility. Waste is to be hauled to nearby waste rock dumps and also used to undertake construction of rehabilitation bunding. The waste rock haul distance will be variable, but on average the haul is approximately 1,200 m.

From the ROM pad, ore material will be blended and hauled to the crusher via wheel loader. The production schedule indicates the material movement will vary over the life of the project (approximately 2 years) and have a maximum rate of 310,000 bcm per month. The mine production rate tapers down as haul lengths increase as the depth of the pit increases and the ratio of Ore to Waste increases.

16.1.2 Open Pit Optimisation

The pit optimisation process was completed using Whittle™ software. As part of this process, SRK analysed both discounted and undiscounted scenarios with a range of sensitivities. As part of the optimisation work, in conjunction with ongoing understanding of the underground operations access requirements, it was identified that the current access to the underground workings was not to be interrupted as part of the initial planned open pit operation.

Parameters for the optimisation were provided by Ivanhoe.

Osborne Block Model

The Osborne Mineral Resource block model was imported into Whittle and verified against the original Mineral Resource block model (block model), created in Surpac. The Surpac block model subsequently was coded in preparation for optimisation. The verification process indicated no material changes to the block model tonnes and grade during the process of importing into Whittle.

The block model was constructed with the following parameters in Table 16-1.

Table 16-1: Block Model Block Sizes

Item	(m)
X	5.0
Y	5.0
Z	5.0

Optimisation Constraints

The optimisation process is restricted to the Mineral Resource classifications of Indicated and Measured in accordance with the NI 43-101 guidelines. For the purpose of the undiscounted Osborne optimisation, there were no production, or processing limits used within Whittle and all material not classified as Measured or Indicated has been treated as waste.

Optimisation Parameters

The Osborne pit optimisation has been carried out using Whittle optimisation software (Whittle Version 4.4). Revenue, mining costs, processing values and other factors as described below are considered within Whittle. These parameters are used to determine the optimum pit shell to be used as a guide for the preparation of open pit designs. The parameters used for the optimisation discussed below and summarised in Table 16-2 have been supplied by the client.

Mining Dilution and Ore Losses

The block model as imported into Whittle is undiluted. The optimisation process has included factors of 5% mining dilution and 95% ore recovery.

When Whittle applies dilution, a grade of 0% copper and 0 g/t gold is used by default and is therefore conservative considering the likely diluting material may be marginally below the economic cut-off.

Geotechnical Parameters

Table 16-2 summarises the geotechnical parameters used for the optimisation process based on the suggested pit design parameters from a previous geotechnical study (Rosengren, 1994). SRK has assumed the number of access ramps and ramp width to calculate the overall wall angle (required for the optimisation process).

Table 16-2: Optimisation Geotechnical Parameters

Category	Mesozoic Cover	Oxide	Fresh
Bench Height	10 m	20 m	30 m
Batter Angle	60^o	70^o	70^o
Berm Width	6 m	8 m to 10 m	10 m
Inter-ramp angle	40^o	53^o to 49^o	55^o
Access Ramps	1	1	1
Ramp Width	25 m	25 m	25 m
Overall Wall Angle	22.3^o	37.4^o	50.5^o

The walls are assumed to be completely dewatered.

Discount Rate

The pit optimisation for Osborne utilises a discounting factor of 10%. Inflation is not factored into the costs.

A maximum processing rate of 2 Mtpa was used as an upper limit for the optimisation schedule. In reality, the realistic processing rate was restricted by the possible physical mining rate was restricted by the possible physical mining rate for the resulting pit size. For the case selected the possible processing rate was substantially less than 2 Mtpa.

Royalties

Royalties have been defined by the client. This was applied in the Whittle model as a selling cost. No other private royalties are included.

Mining Costs

The client supplied SRK with several quotes from contractors to provide the earthmoving and drill and blast activities. In addition, the client provided SRK with grade control and mine planning cost estimates. SRK has averaged the contractor quotes and included any other associated mining costs to generate a base mining cost.

The contractor quotes were provided in several formats including low cost per tonne with high management costs, and higher cost per tonne with lower management costs, however the overall estimated cost was comparable. Therefore, to generate the base mining cost for inclusion in the optimisation process, SRK has averaged the quotes supplied and divided by the total tonnes. An incremental depth factor has not been included specifically as this has been incorporated into the quotes supplied by the contractors.

All material has been classified as Fresh material, and a constant mining cost has been applied.

SRK understands the operating mining costs to include the following:

- Loading;
- Hauling;
- All auxiliary mining activities;
- Drilling and blasting;
- Grade control costs;
- Mine planning;
- Dewatering; and
- Any required rehandling activities.

Processing Costs and Recoveries

The estimated processing cost for the Osborne deposit is constant, based on the deposit only containing Fresh material.

The processing, concentrate transport, refining costs and metallurgical recovery presented in Table 16-3 with other optimisation parameters.

Table 16-3: Optimisation Parameters

Parameter	Unit	Value
Mining Dilution	%	5
Mining Dilution Grade		0.00
Mining Recovery	%	95
Overall Slope Angle	°	45
Mining Cost	$ / t	5.17
Mining Rate	Mtpa	unlimited
Processing Rate	Mtpa	2.0
Process Recovery Copper-Gold Ore (Copper)	%	92
Process Recovery Copper-Gold Ore (Gold)	%	65
Processing Costs (Copper-Gold Ore)	$ / t_{ore}	10.60
Copper Price	USD / lb	3.75
Gold Price	USD / oz	1,400
Copper Royalty	% Revenue	5.0
Gold Royalty	% Revenue	5.0
Smelter Recovery (Copper)	%	95.80
Smelter Recovery (Gold)	%	94.0
Concentrate Transport Costs	USD / t_{conc}	119.00
Smelter Costs	USD / t_{conc}	55.00
Refining Costs	USD / lb Copper	0.055
Exchange Rate	USD / AUD	1.00
Copper Concentrate Grade	%	24.0

Optimisation Process

To optimise the Osborne deposit, a series of nested pit shells have been calculated over a range of Revenue Factors (RFs). Each of the nested pit shells are generated based on the maximum undiscounted cash flow calculated for the applicable RF. The nested pit shells will increase in size as the RF increases.

To determine the optimum pit shell and for reporting purposes within Whittle, the maximum discounted reported cash flow has been used.

As part of the optimisation process, Whittle uses the pit tonnages from nested pits and calculates the cashflow based on RF=1. Nested pit shells generated for a RF less than 1 will have cash flows greater than those used to determine the physical nested pit shell. Nested pit shells generated at a RF greater than 1 will have cash flows less (even negative) than those used to determine the physical nested pit shell. This is because material is mined (in the larger pits) that is economic when the original RF is applied, however when Revenue Factors greater than 1 are used, some material within the pit becomes uneconomic, thus reducing the cashflow of that pit shell.

Optimisation Results

Table 16-4 shows the results of the optimisation for a range of RF values in tabular form. Figure 16-1 shows pit shell 18 contains the highest discounted cashflow.

Table 16-4: Osborne Discounted Optimisation Results

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Copper Grade	Gold Grade
	(AUD M)	(Mt)	(Mt)	(%)	(g/t)
1	0.246	0.004	0.003	1.023	1.432
2	0.530	0.008	0.009	1.035	1.471
3	0.572	0.009	0.009	0.996	1.389
4	0.585	0.009	0.009	0.979	1.375
5	0.707	0.012	0.012	0.944	1.186
6	0.824	0.016	0.015	0.886	1.031
7	0.863	0.017	0.017	0.854	1.034
8	1.201	0.028	0.050	0.874	0.923
9	1.230	0.028	0.055	0.887	0.918
10	1.290	0.032	0.059	0.849	0.869
11	3.620	0.069	0.689	1.736	0.986
12	4.683	0.086	1.022	1.932	1.022
13	5.214	0.102	1.191	1.862	0.971
14	29.305	2.775	15.055	0.82	0.554
15	30.255	2.854	16.092	0.838	0.562
16	30.442	2.909	16.254	0.832	0.559
17	30.476	2.957	16.343	0.826	0.556
18	30.519	3.040	16.627	0.819	0.551
19	30.371	3.289	17.474	0.798	0.539
20	-5.005	10.322	57.011	0.708	0.506
21	-7.887	10.645	59.478	0.711	0.507
22	-7.890	10.646	59.483	0.711	0.507
23	-8.652	10.740	60.066	0.711	0.506
24	-8.767	10.753	60.135	0.711	0.506
25	-9.959	10.847	60.962	0.712	0.506
26	-13.108	11.062	62.787	0.713	0.506
27	-13.126	11.066	62.796	0.713	0.506
28	-13.383	11.097	62.973	0.713	0.506

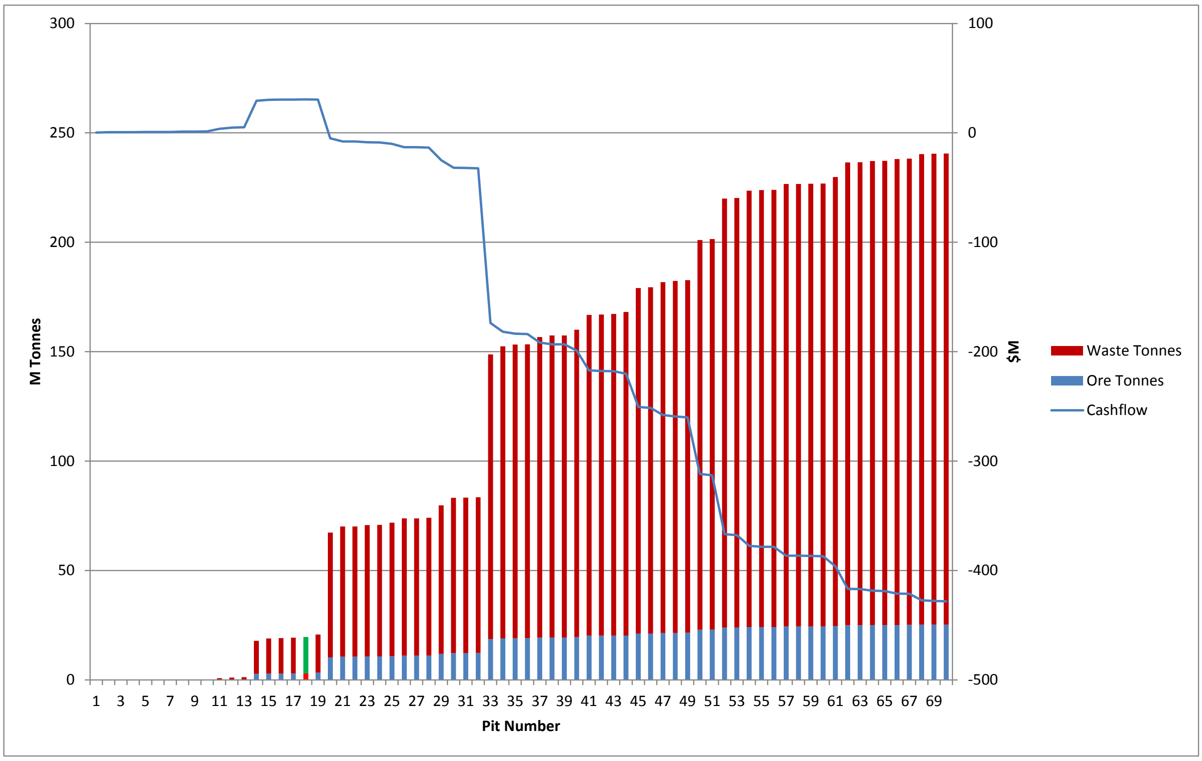


Figure 16-1: Osborne Discounted Optimisation Results (Cashflow)

Figure 16-2 shows the pit inventory for each pit shell that there are several major "steps" in pit value and subsequent tonnage, which infers there are geological or grade features which affect the pit economics. These features are representative of ore zones with grade values which change the pit economics as the pits extend at depth.

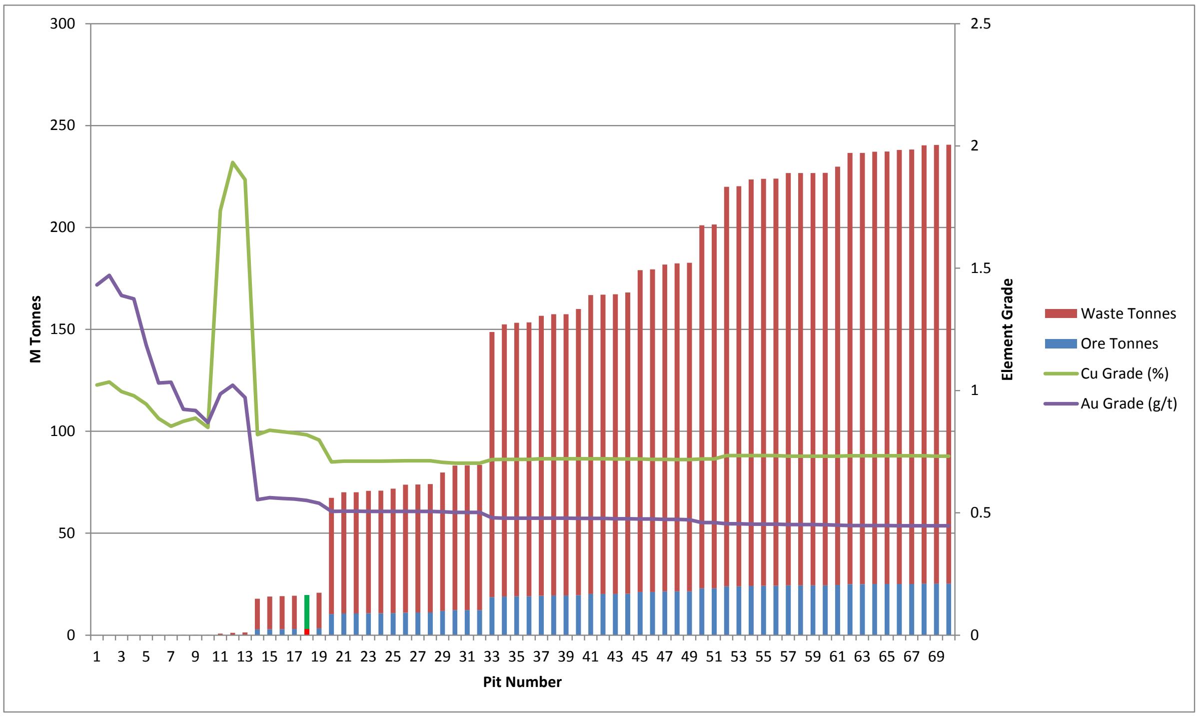


Figure 16-2: Osborne Discounted Optimisation Results (Pit Inventory)

16.1.3 Mine Design

The mine design is based on previously achieved geotechnical parameters and a review of the block model.

The mine has been designed in two phases, namely Cutback 1 and Cutback 2. A third phase, Final Design has been included for reporting purposes only and contains the same design strings as Cutback 2. The Final Design includes Cutback 1 and Cutback 2.

Design Parameters

Table 16-5 defines the geotechnical design parameters utilised to design the proposed open pit cutbacks. The wall angles and berm parameters match the existing Osborne open pit which has remained stable since completion in 1996.

Table 16-5: Open pit design parameters

Category	Mesozoic Cover	Oxide	Fresh
Bench Height	10 m	20 m	30 m
Batter Angle	60^{o}	70^{o}	70^{o}
Berm Width	6 m	8 m to 10 m	10 m
Inter-ramp angle	40^{o}	53^{o} to 49^{o}	55^{o}

Pit Design

Ivanhoe informed SRK that access to the underground portal was not to be interrupted in the short term as part of the initial pit design. This constrained the design conformance to the optimised shell, particularly above the portal and to the southwest corner of the design. This resulted in the pit being mined as two cutbacks.

Cutback 1 was designed to maintain access to the current underground workings.

Cutback 2 was designed to match the optimised pit shell (and includes extracting the current underground access portal).

Table 16-6 summarises the mining inventory for the two cutbacks.

Table 16-6: Open pit mining inventory

Category	Ore Tonnes (Mt)	Copper Grade (%)	Waste Tonnes (Mt)
Cutback 1	1.44	0.79	9.57
Cutback 2	1.06	0.96	6.52
Total (Final Design)	**2.50**	**0.86**	**16.09**

Cutback 1

As part of the design work, Ivanhoe indicated that access to the current underground portal was to be maintained during the first phase of mining. This phase has been termed Cutback 1. Figure 16-3 displays the plan view for Cutback 1 design.

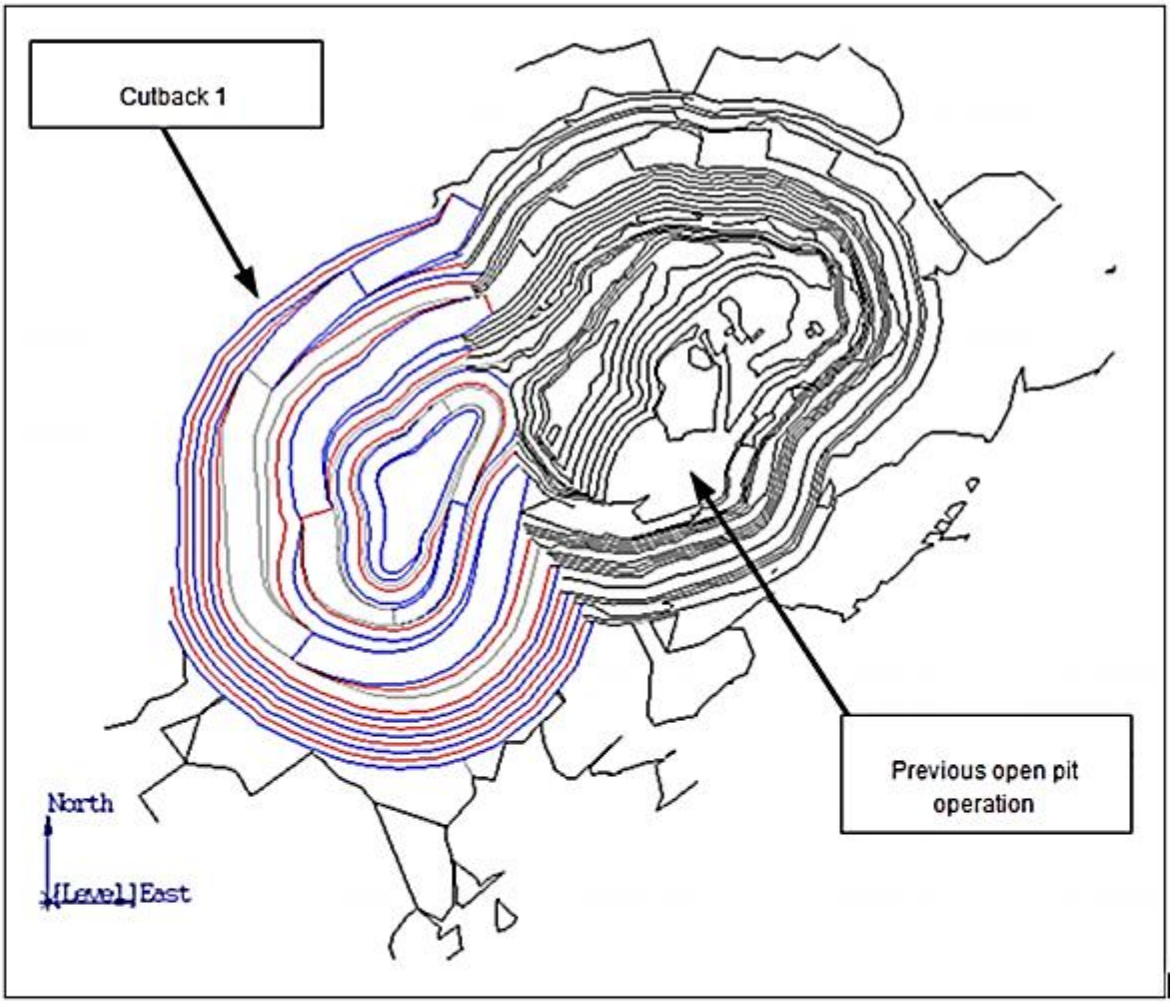


Figure 16-3: Plan view of proposed Cutback 1 design

The maximum depth of Cutback 1 is 170 m deep from surface (pit exit).

As can be seen in Figure 16-4, a twin ramp system has been designed. This allows for production to occur on two production faces, ensuring continuous production. The upper ramp is to be built so that it links in to the current underground operations portal, with the lower ramp being used to access the material deeper in the open pit operation.

Figure 16-5 highlights the cutback extension to the west of the current open pit.

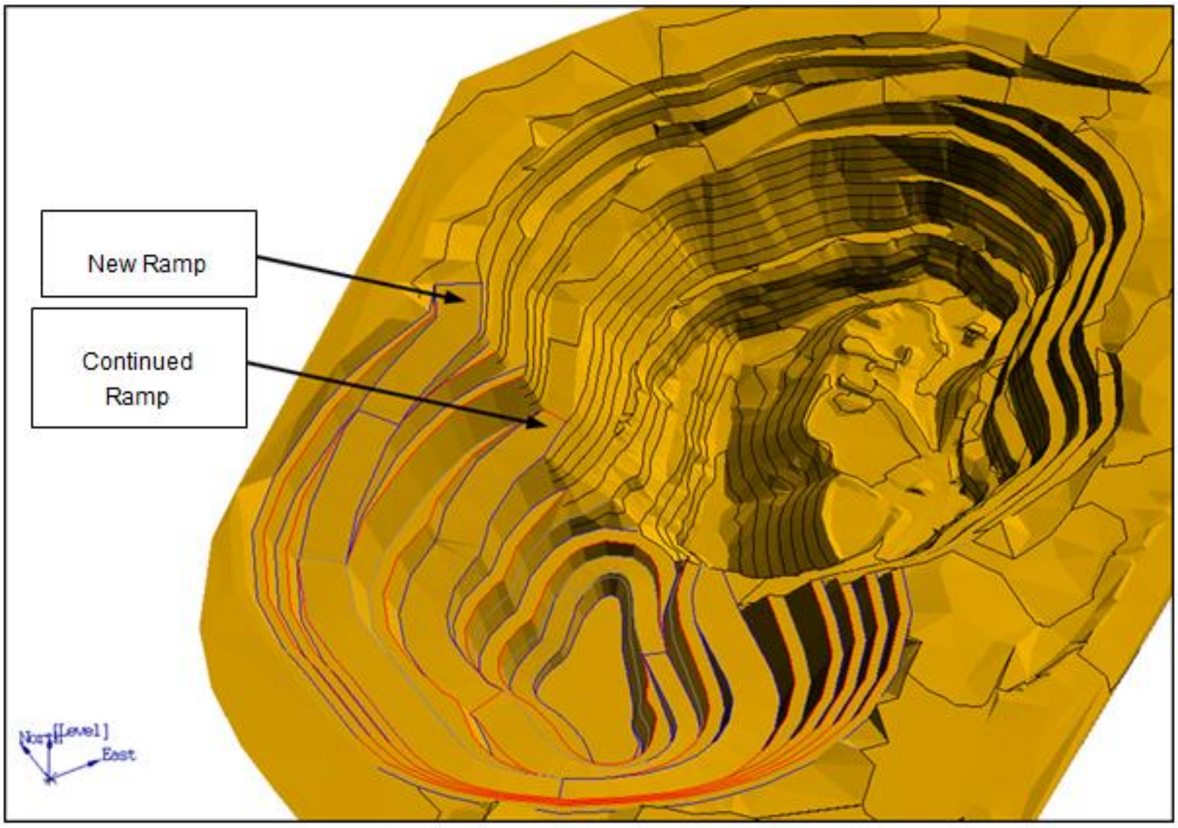


Figure 16-4: Oblique view of proposed pit design (looking north)

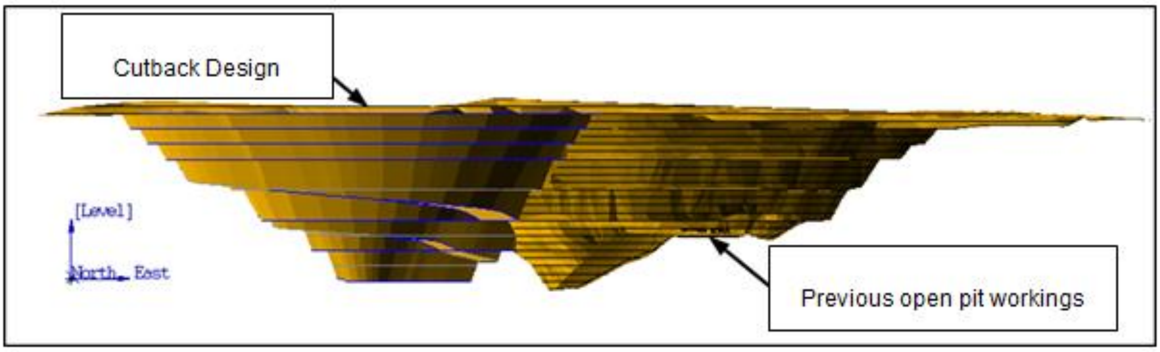


Figure 16-5: Sectional view of proposed pit design (looking north)

Table 16-7 summarises the pit inventory for Cutback 1, defined by JORC classification. The inventory has not had mining dilution or mining recovery applied.

Table 16-7: Cutback 1 Mining Inventory

Classification	Tonnes (Mt)	Copper Grade (%)
Proven	1.38	0.79
Probable	0.05	0.86
Total	**1.44**	**0.79**
Waste	9.57	

Cutback 2

Cutback 2 is planned to be mined following Cutback 1 and involves stripping the remaining material to achieve the optimised pit shell. An interim ramp will be required on the southern wall to access the waste material on the upper benches; however this ramp will be extracted as part of Cutback 2.

Design Features

To extract the full pit design, an interim ramp is required on the southern wall to provide access to the waste material on the upper benches above the current underground access portal. It is expected once the access ramp is established; the ramp will progressively be mined and hauled out of the pit via the ramp which exits the pit on the north.

Figure 16-6 shows the interim southern ramp access which will be used to access the waste material above the current underground access portal. This ramp will be progressively mined off and hauled via the northern ramp.

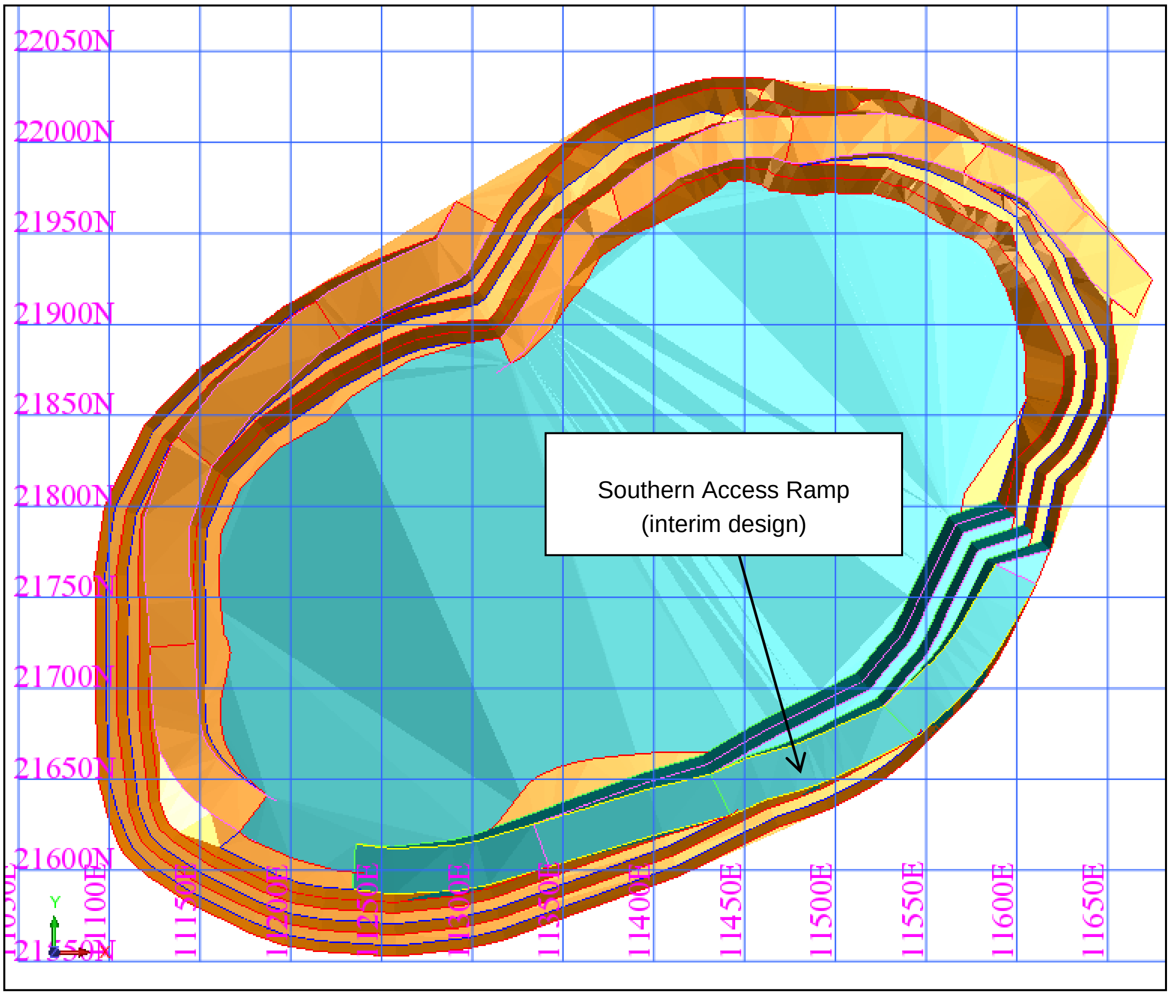


Figure 16-6: Osborne Pit Design (interim ramp access) – Plan View

Following the establishment of the ramp on the southern wall, the final pit design can be implemented and extracted (Cutback 2 design strings). Backfilling at the base of Cutback 1 will be required for the ramp to the bottom of the Cutback 2 pit. Figure 16-7 shows the Osborne designed pit in plan view.

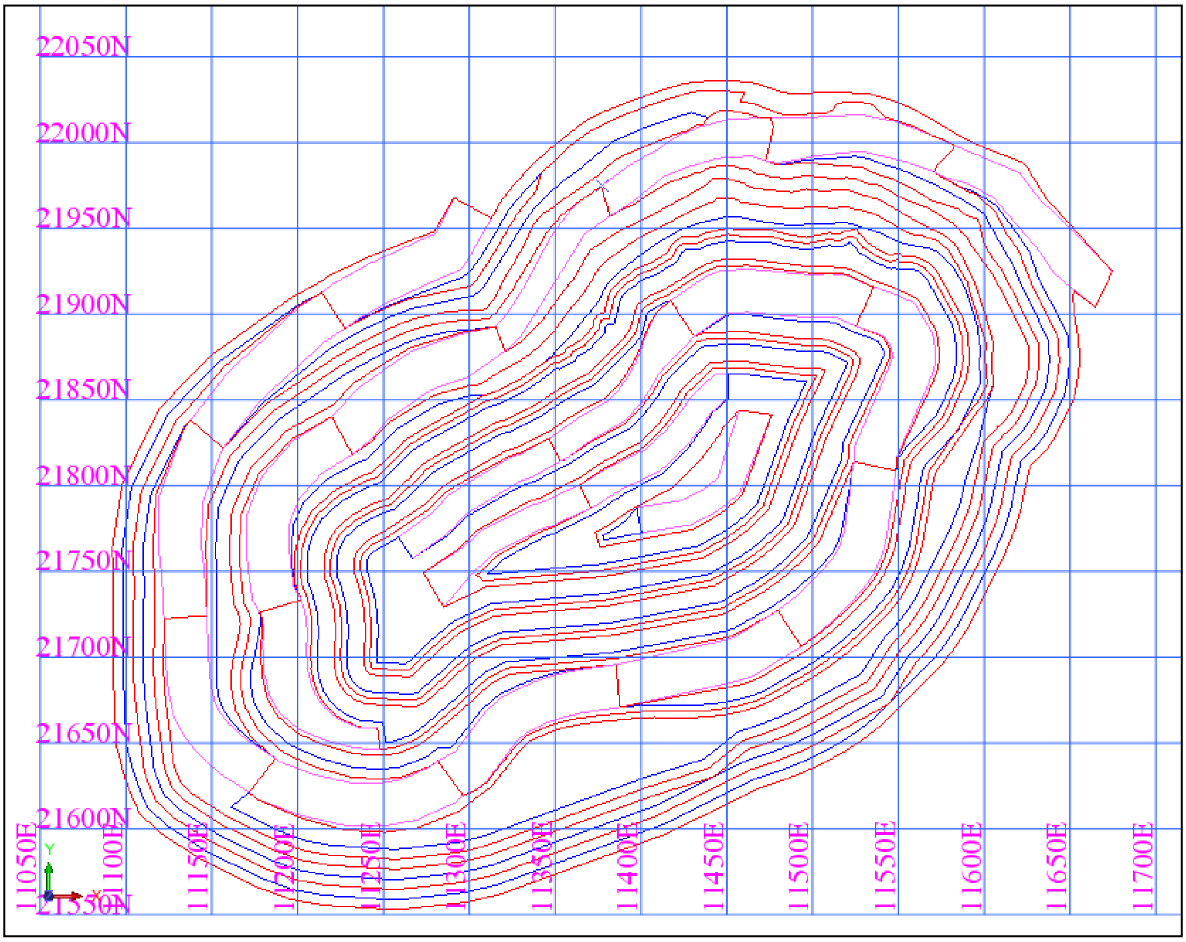


Figure 16-7: Cutback 2 Design – Plan View

Table 16-8 summarises the mining inventory for Cutback 2. The inventory has not had mining dilution or mining recovery applied.

Table 16-8: Cutback 2 Mining Inventory

Classification	Tonnes	Copper Grade (%)
Proven	1.03	0.87
Probable	0.04	0.76
Total	**1.07**	**0.86**
Waste	6.52	

Underground Workings Interaction

The Final Design has minimal interaction with the current and proposed underground workings. It is expected any interaction will be managed locally, with potentially some localised backfill required and a reduced production rate whilst mining within the vicinity of the underground workings.

Figure 16-8 and Figure 16-9 shows the current pit design interaction with the current underground workings. The proposed pit design has minimal interaction with any underground workings and is not expected to materially impact the project.

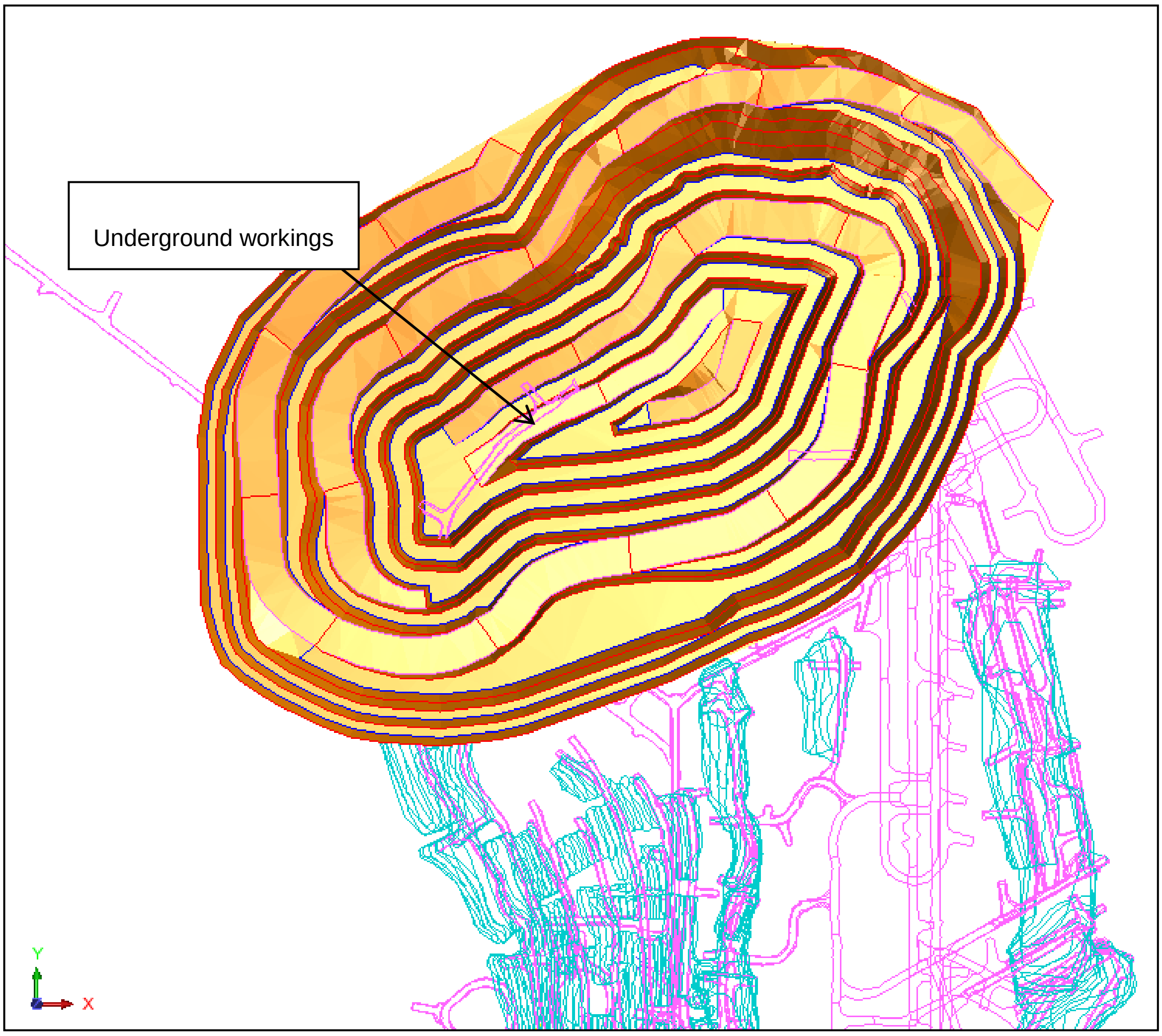


Figure 16-8: Final Design with current Underground workings (purple)

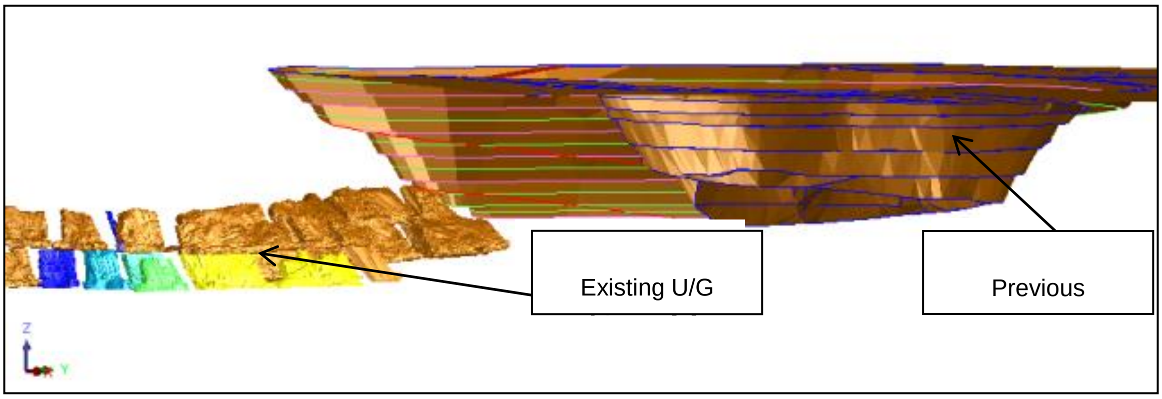


Figure 16-9: Sectional view of proposed pit design looking north (including proposed underground mine)

Pit Design Conformance with Optimised Shell

Figure 16-10 graphically shows the designed pit (brown) with the optimised shell (purple).

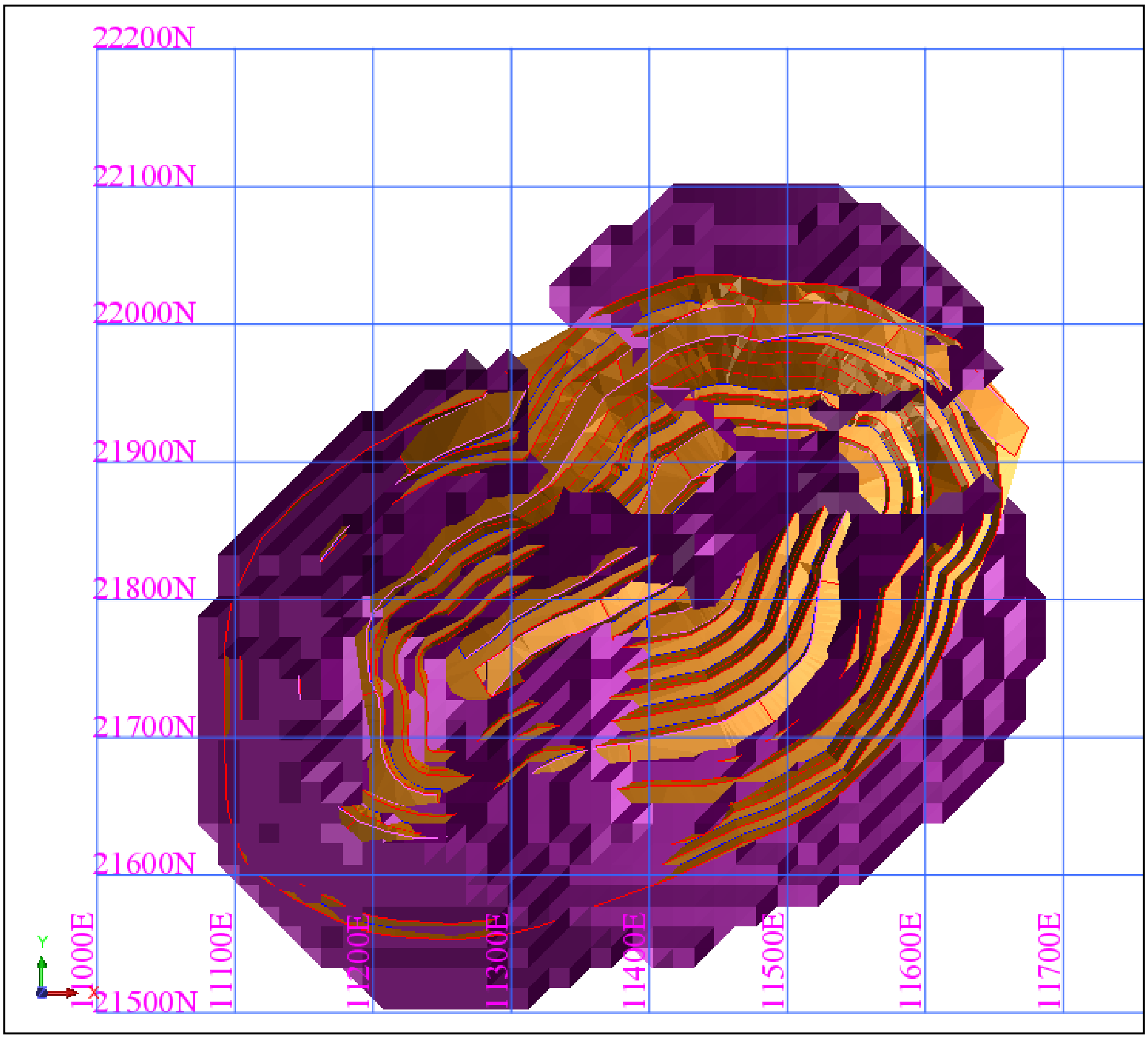


Figure 16-10: Osborne Pit design compared with Optimised Shelf Plan View

For the purposes of conformance, 'Ore' has been defined as Fresh material with a JORC classification of Measured or Indicated, and has a minimum grade of 0.35% copper. All other material is classified as waste.

Table 16-9 summarises the conformance between the Osborne designed pit and the optimised shell.

Table 16-9: Conformance of Final Design versus optimised shelf

Item	Optimised Shell (tonnes)	Final Design (tonnes)	Conformance
Ore	2,697,694	2,505,653	93%
Waste	16,879,874	16,089,831	95%

16.1.4 Mine Operations

Introduction

The following mining parameters have been applied:

- All productivities and activities are based on contractor mining;
- A 10 m drill and blast bench for all material types;
- The pit ramps are dual lane, the ramp gradient is designed at 10% (1 in 10);
- Hydraulic excavators (~200 t), in backhoe configuration will be the primary loading unit(s), with a smaller excavator (~100 t) in the fleet to use for batter trimming, topsoil removal and backup for the primary excavator as required to meet the production targets;
- Drilling, loading and hauling operations will be carried out on two 12 hour shifts (night and day), whilst explosive charging and blasting will be carried out on day shift only;
- All direct mining personnel will be employed on a roster amenable with current industry guidelines (nominally 8 days on, 6 days off); and
- All grade control sampling will be undertaken utilising either blast hole or RC drilling.

Mining Method

Excavate and Load

The standard bench height will be 10 m. Each bench will be mined by in a standard method using a hydraulic backhoe. Each bench will be mined in three 4 m flitches and loaded into dump trucks. In the harder material where extensive heaving of blasts is likely to occur, the top flitch may be greater than 4 m.

Ore and waste boundaries will be delineated with colour coded flagging tape to differentiate the grade of the material. This delineation will be completed using data from exploration, development and grade control drilling and set out on the mining floor by the survey department. This may be changed by visual controls, based on immediate geological information. The mining operation will excavate and load the ore and waste in accordance with the relevant boundaries. A geologist or trained field technician will be used as a spotter during ore mining to minimise dilution.

Mining along the ore block strike will be the standard practice and all grading or dozer clean-ups will be restricted to along strike in the ore zones to minimise dilution even further.

Ore Haulage

The Osborne ROM pad is situated adjacent to the Osborne Mill. The ore extracted will be stockpiled on ore pads adjacent to the processing facility. The current mining and processing schedule has a stockpile reaching a maximum capacity of just under 1 Mt before being drawn down. A wheel loader will maintain the stockpiles and feed the crusher as required.

Waste Haulage

Waste material will be hauled directly to waste dumps when not required or suitable for road construction. All fresh waste rock material will be encapsulated within the dump.

Ivanhoe has developed multiple waste options. The main waste option is a waste dump located within 1,000 m to the northeast of the open pit operation. The other seven placement locations involve tailings facility assistance (capping, bunding, strengthening tailings facility containment walls).

Outer slopes of the waste dumps will be constructed at angles as required by environmental approvals. Berm widths and maximum lift heights will be refined to best achieve the final profile as stated within the environmental requirements.

Drill and Blast

Drilling will be undertaken utilising Atlas Copco L8 size drill rigs (or equivalent). A bench height of 10 m will be used for blasting as these are the ideal height for the proposed 200 t excavator. Drillholes will be 89 mm up to 152 mm. The drill patterns used will be designed dependant on type of blast required (production, trim, presplit), material type and the powder factor required. Some material may be free dig, but the majority will be blasted. Ground water may be present, and adjustments will be made as required; however, the likelihood of groundwater issues is low, due to the interactions with the existing pit and underground workings. Since all proposed mining involves cutting back on an existing pit, the use of ammonium nitrate fuel oil (Heavy ANFO) has been assumed down to 30 m above the existing pit floor levels and then all emulsion for the remainder of the pit. This will be reviewed based on localised ground conditions.

Final-wall control will be maintained via the use of pre-splitting and where required, buffer and trim blasting.

Grade control sampling may be taken on all vertical blast holes in and adjacent to the ore zone, or by use of an RC drill rig drilling one bench ahead. All assay samples will be processed at the Osborne Mine assay laboratory.

Ancillary Services

A fleet of mining equipment will be used to support the primary load and haul fleet.

This is likely to include the following:

- Small excavator (80-100 t);
- Tracked dozer(s);
- Water truck; and
- Motor grader.

The smaller excavator will be used for batter faces, topsoil removal, drainage and dewatering; in addition to being a backup to the primary production excavator.

The dozer will be used for the maintenance of pit benches, dump management, haul road and ramp construction, clearing and topsoil removal.

Graders and water trucks will be used for haul road maintenance and dust suppression.

Selected Mining Fleet

Table 16-10 summarises the contractor proposed mining fleet. This fleet is indicative only and will be reviewed upon finalisation of scheduling requirements and equipment availability.

Table 16-10: Proposed Mining fleet

Type	Make	Model	Number of units
Excavator		200 t op. weight	3
Drills	Atlas Copco	L8	2
Dump trucks	Caterpillar	777 / 785	6
Track dozer	Caterpillar	D9	2
Grader	Caterpillar	16G	1
Water truck	Caterpillar	769C	1

Mine Schedule

Introduction

The schedule for the Osborne Open Pit has been compiled based on the historic performance of the likely available mining fleet. The schedule has been based on 10,000 bcm / day total material movement and has been generated based on quarterly time increments.

Within the scheduling process, vertical and horizontal lags have been included to ensure all development activities will be completed prior to mining commencing on a new bench. A small ROM stockpile is planned to be utilised prior to material being fed to the processing facility. As the planned ROM stockpile will be quite small (<3 days of ore feed) for the purposes of the schedule, it is assumed all material mined will be processed immediately.

All values quoted have been modified by mining recovery of 95% and mining dilution of 5%. Dilution has been applied with a 0% grade.

Production Rates

For scheduling purposes, only Measured or Indicated materials have been classified as ore. All other material has been scheduled as waste.

Table 16-11 summarises the proposed production schedule.

Table 16-11: Total material movements

Year			Yr 1 H1	Yr 1 H1	Yr 1H2	Yr 1 H2	Yr 2 H1	Yr 2 H1	Yr 2 H2	Yr 2 H2
Period	Units	Totals	1	2	3	4	5	6	7	8
Diluted Ore Tonnes	(kt)	2,499	0	0	181	629	625	15	812	237
Waste Tonnes	(kt)	16,730	2,613	2,650	2,786	1,642	2,034	2,232	2,397	377
Total Material Movement	(kt)	18,391	2,481	2,516	2,827	2,190	2,557	2,134	3,090	596
Copper Grade	(%)		0.00	0.00	0.91	0.76	0.70	1.36	0.66	1.75
Gold Grade	(g/t)				0.74	0.53	0.50	0.78	0.46	1.08

Figure 16-11 summarises the scheduled ore production from Osborne open pit with the respective grade production.

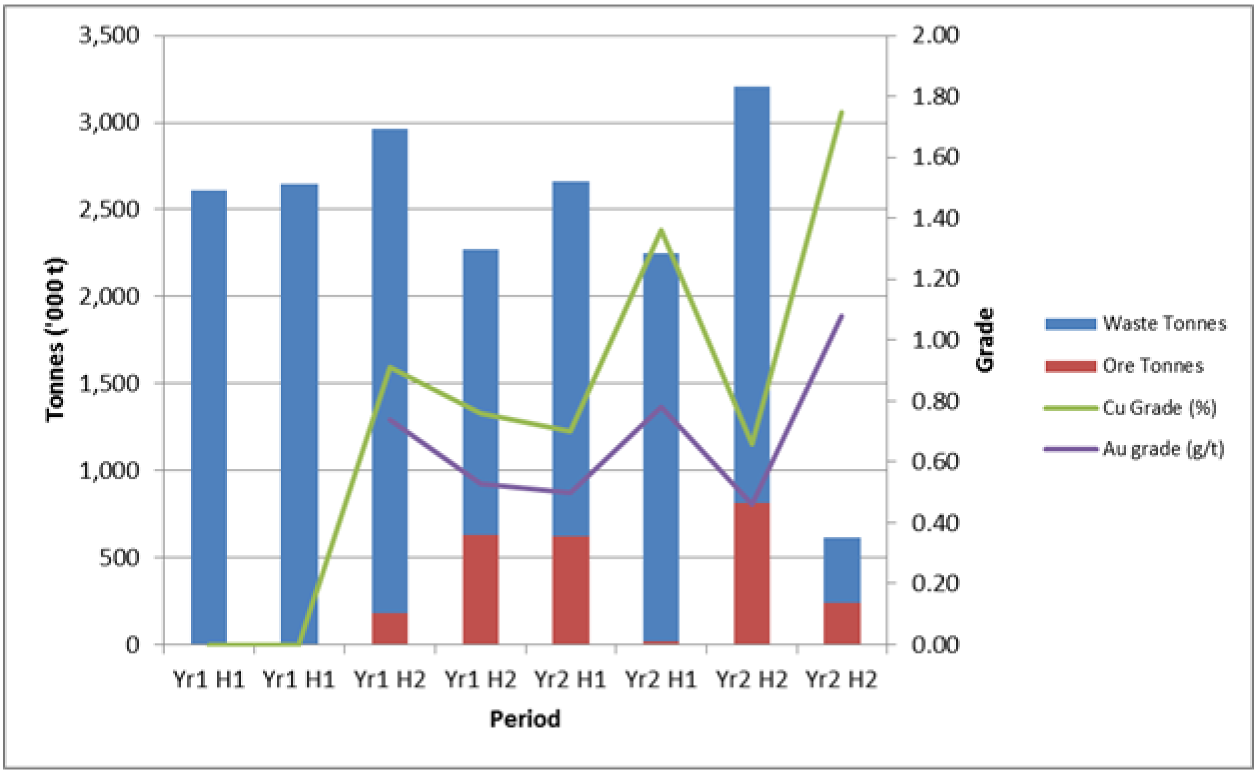


Figure 16-11: Ore production with Element grades

Human Resources

Shift Schedule

The mining costs have been estimated using a continuous mining operation, 24 hours a day, 365 days per year. All employees will FIFO from Townsville and utilise the Osborne village while on site.

Operators and maintenance personnel will work a FIFO roster amenable to current industry standards. This will constitute a rotation of 8 days on followed by 6 days off, 12 hour shifts alternating between dayshift and nightshift.

Ivanhoe support staff will work 8 days on, then 6 days off, with a 12-hour dayshift only.

All on-costs for annual / sick leave and training have been estimated in the operating costs.

Personnel Levels

All earthmoving and ancillary activities will be performed by the nominated earthmoving contractor. It will be the responsibility of the contractor to ensure sufficient manning and training levels are maintained throughout the contract to meet the production schedule.

SRK understands the contractor activities to include (but not limited to) the following:

- Loading;
- Hauling;
- Drilling and blasting;
- Dewatering; and
- Any required rehandling activities.

Maintenance personnel have been estimated based on machine hours and site location. The maximum open pit workforce will be approximately 42 Contractor personnel and 15 Supervisory Staff.

Personnel numbers for each position and total work force levels are shown in Table 16-12.

Table 16-12: Owner surface mining personnel requirements

Position	Total
Production Superintendent	1
Senior Mining Engineer	1
Senior Geologist	1
Safety and Training Officer	1
Mining Engineer	1
Mine Surveyor	2
Geologist	2
Technicians	4
Labourer	2

Table 16-13 lists the indicative mining contractor personnel requirements for surface mining at Osborne as advised by contractors.

Table 16-13: Indicative contractor surface mining personnel requirements

Position	Total
Project Manager	1
Project Superintendent	1
Operations Supervisor	3
Administration Clerk	1
Workshop Manager	1
Workshop Leading Hand	2
Fitters	8
Servicemen	3
Operators	22

Labour Costs

Labour costs have been estimated and include superannuation, salary continuance, health insurance, fringe benefit tax, workers compensation and payroll tax.

All operator labour costs are included in the mining tenders supplied by the relevant contractors.

16.1.5 Metallurgy and Processing

The section provides a brief overview of the Metallurgical and Processing aspects related to Osborne Open Pit. Additional detail is presented in Section 13 and Section 17.

All ore material will be treated in the current Osborne Mill. The processing facility is to be utilised by other feed sources in addition to the Osborne open pit.

As the material is to be fed directly from a ROM stockpile, rather than directly from the mine, the processing facility capacity is not likely to impact the mine production.

Metallurgical recoveries are calculated from previous data, client received information and previous experience.

The metallurgical recovery at Osborne is dependent upon the lithology and head grade of the material being processed.

Recoveries have been based on the grade recoveries supplied by the Processing department for the 2008 LOM. Metallurgy for open pit material may vary depending on the deposit and material type. The average recovery for each deposit has been used in the mill cut-off grade estimate. The recoveries applied for each processing route are listed in Table 16-14.

Table 16-14: Metallurgical recoveries applied

Type	High Grade	Low Grade
Copper Concentrate Recovery	85%	60%
Gold Concentrate Recovery	75%	45%

16.1.6 Environmental Management

This section provides an overview of environmental aspects relating to the Open Pit operations. Detail on the overall project environmental considerations are presented in Section 20.

Introduction

Topsoil will be removed to a minimum depth of 200 mm where possible by tracked dozer, excavator and truck methods. Topsoil will be either transported directly to areas requiring final soil coverage for rehabilitation or stockpiled in areas in close proximity to the waste dumps and no more than 1.5 m high to prevent a decline in beneficial aerobic micro-organisms. The topsoil will also be stockpiled in bunds at the toe of the final dumps.

The waste dump sides will be progressively battered down to the final design slope at the completion of each segment. Topsoil will be placed using the ancillary equipment on the crest of these walls in readiness for spreading over the slopes.

At the completion of mining, stockpiled topsoil will be re-spread over all other remaining disturbed areas. These areas will then be contoured, ripped and seeded. Only seed that is native to the area or appropriate to the prevailing conditions will be used to revegetate the site.

Environmental Permitting and Regulatory Approvals

The operation of the processing plant and the associated tailings storage facility (TSF) is currently covered under the DEHP. Osborne must operate its facilities in accordance with the licence conditions.

Environmental Management System (EMS)

All mining activities will be conducted in accordance with the existing EMS for Osborne Mines. All site personnel and contractors will be obligated to conform to this standard.

All environment associated information, including survey reports and mining proposals are maintained within the Environment Department library at Osborne Mine. Electronic copies of these reports are available as required.

Copies of all associated permits, contracts and legal undertakings are maintained on site. Electronic copies are available on request.

Rehabilitation and Mine Closure

The objective of the Osborne Mine rehabilitation programme is to return sites affected by mining to a stable, non-eroding, and safe condition. In addition, these areas will be restored to biologically sustainable ecosystems, requiring minimum long-term management. Rehabilitation of disturbed areas will be conducted in accordance with current Queensland government guidelines.

Rehabilitation will commence as soon as is feasible and will proceed so that maximum benefit from stored topsoil will be achieved. Rehabilitation will include spreading of topsoil, ripping of all disturbed areas, and seeding with regionally local native species.

Rehabilitation contractors will be selected to conduct the rehabilitation activities and aid in the determination of the seed mix. The supply of seed will be from a reliable local supplier. The progress of the rehabilitation will be monitored annually through Ecosystem Function Analysis techniques to determine revegetation success.

Dependent upon cover trials currently in progress additional material of a suitable coarseness, and geochemical characteristics, may be sourced from the Osborne Pit Expansion.

Revegetation

Topsoil will contain the initial seed source for the revegetation programme. The topsoil will be spread over the disturbed surfaces along with remnant organic matter to encourage the return of native flora, reduce erosion and enhance surface stability.

The progress of revegetation will be monitored throughout the mine life and reported to the Queensland Department of Natural Resources and Mines (DNRM) via the Annual Environmental Reports.

Closure of Roads

Upon project completion, disused roads will be ripped, spread with topsoil where practicable and revegetated. Some project roads may be left to provide access to pastoral lease activities. Roads that obstruct natural drainage will be removed.

Closure of Waste Dumps

Closure of the waste dump will involve:

- Shaping the outer slopes to 15°;
- Placing a windrow around the outer edge of the dump to prevent runoff occurring over the waste dump sides;
- Spreading 0.30 m of topsoil over disturbed surfaces; and
- Ripping and seeding of disturbed surfaces.

Rehabilitation Costs

Rehabilitation costs have been included in the mining costs at AUD0.04 /t of material, based on historical costs at other operations.

16.1.7 Infrastructure

This section provides an overview of infrastructure requirements relating to the Open Pit operations. Detail on the overall project infrastructure is presented in Section 18.

The contractor supplying all earthmoving services to Ivanhoe are expected to provide all necessary buildings for the operation. Ivanhoe will supply all utilities including:

- Water;

- Power;
- Telecommunications; and
- Fuel.

However, all structures are to be supplied and erected as part of the contractors quote to perform services on site. Typical infrastructure that will be required includes (but not limited to) the following:

- Covered work shop for maintenance activities;
- Fuel distribution system;
- Any administration buildings; and
- Tyre changing facility.

16.1.8 Waste Rock Dump

Cutback 1 Waste Storage Requirements

SLR Consulting as part of a previous scope of work analysed the waste rock dump locations and volumes. The study resulted in eight locations identified for waste rock storage. Table 16-15 summarises and gives a brief description of each of the waste rock dump options identified.

Table 16-15: Waste rock dump options (identified by SLR Consulting)

	Area	Volume (m^3)	Comments
1	North Dump Extension	3,991,345	Maximum volume if the runoff is to be directed to existing environmental dam 2/3. A larger footprint will require a new silt and runoff control dam.
2	TSF 1 oxide cell cover	53,685	1 m thick cover. This will drain towards the new TSF1 area
3	TSF 1 sulphide cell cover	233,108	1 m thick cover, this will be shaped to drain towards the TSF1 wedge dump and into a new silt and runoff dam.
4	TSF1 wedge dump	1,140,943	Maximum volume as a waste dump rather than a wedge, designed within the road alignment.
5	TFS2 cover	508,336	1 m thick cover, with surface runoff draining towards the East over TSF2 wedge and into a new silt and run off dam.
6	TSF1 wedge dump	19,256	This volume is for a wedge construction at a 1:45 (Vertical : Horizontal). A new silt and runoff dam will be required.
7	New TSF 2 Bunds	40,843	Volume will vary with the new proposed design for TSF2 extension.
8	New TSF 1 Bunds	60,335	Volume will vary with the new proposed design for TSF1 extension.
	Total	**6,047,851**	

Figure 16-12 displays the potential waste rock dumps graphically around the Osborne mine complex.

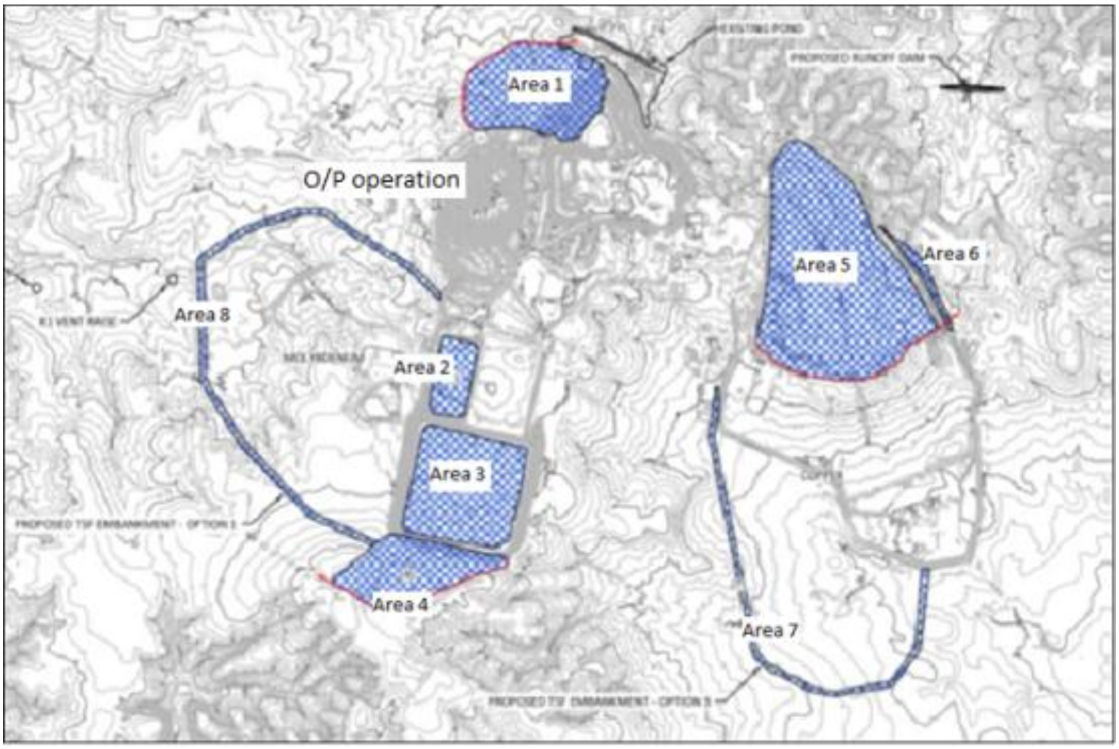


Figure 16-12: Waste rock dump locations

As part of the review process, SLR Consulting were informed that approximately 4.2 Mm^3 in situ material would be required. A swell factor of 1.4 has been used to calculate the required volume for waste rock storage. This calculation estimates that 5.9 Mm^3 capacity is required. Table 16-15 indicates that just over 6 Mm^3 capacity is available in the options reviewed.

Locations shown have been selected to complete rehabilitation earth works as a priority. The intent is to reduce the rehabilitation liability through selective placement of material from the open pit excavation.

16.1.9 Cutback 2 Waste Storage Requirements

The current scope of work defined a slightly larger pit. This resulted in additional waste rock storage requirements. For Cutback 2, a combined total of approximately 8 Mm^3 waste storage capacity is required. SRK reviewed the previous work by SLR Consulting, and after discussion with the client, agreed to increase the capacity of Area 1 (as defined by SLR Consulting).

Figure 16-13 shows the revised waste dump design.

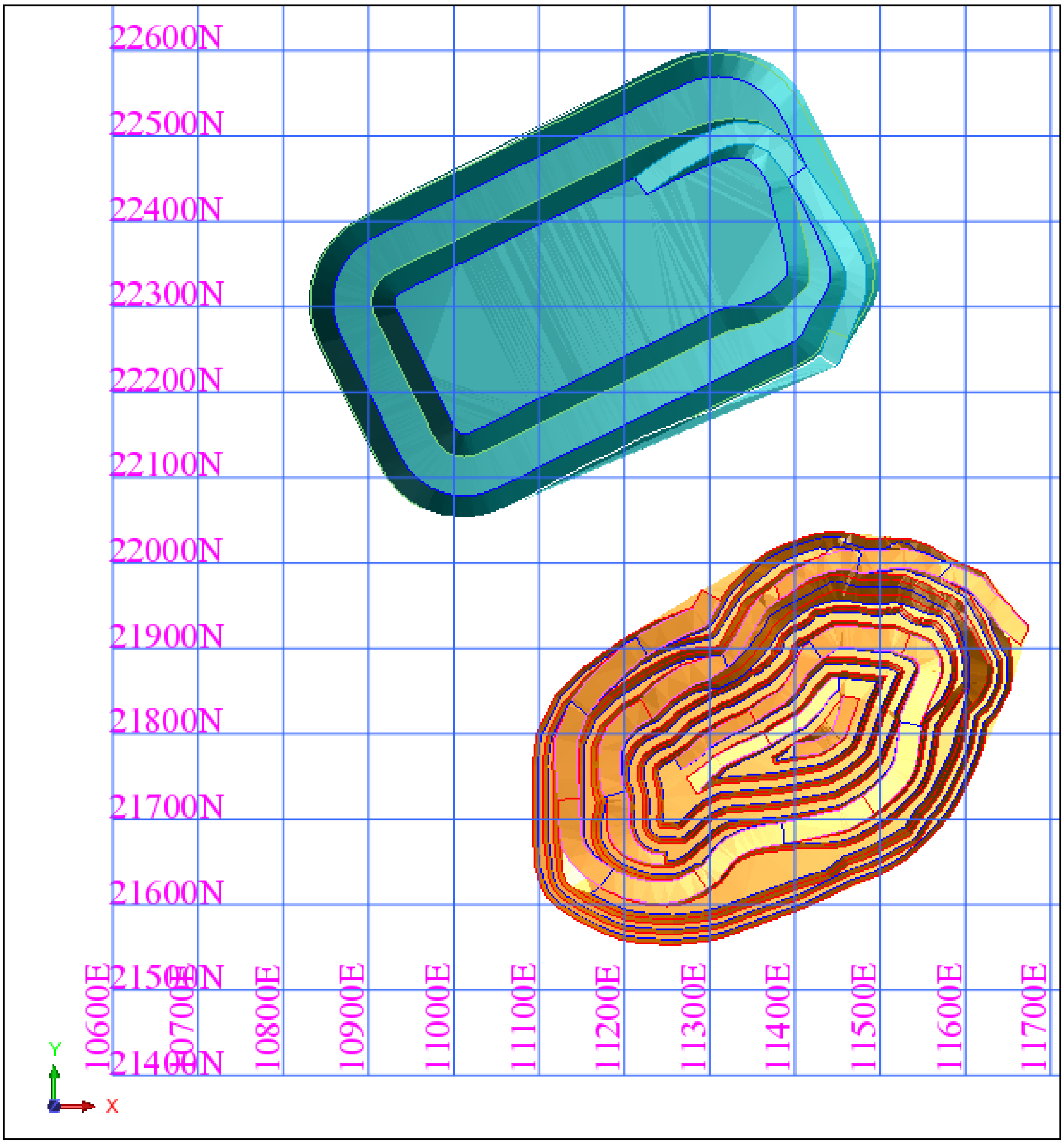


Figure 16-13: Osborne Conceptual Waste Dump

SRK notes this waste dump design is conceptual, with the final location to be determined after further consultation with site based personnel. SRK is not aware of any factors which would preclude this design from implementation.

16.2 Osborne Underground

16.2.1 Introduction

Ivanhoe proposes to mine the extension to the Osborne Underground Mineral Resource between 135 mRL and 60 mRL. A summary of the Mineral Resource as described in Section 4 to Section 14 is shown in Table 16-16.

Table 16-16: Osborne Mineral Resource

Category	Quantity (Mt)	Grade			Metal	
		eCu (%)	Copper (%)	Gold (g/t)	Copper ('000 t)	Gold ('000 t)
Measured	2.1	2.1	1.5	0.9	31.7	57.5
Indicated	0.8	1.7	1.2	0.9	9.7	22.1
Inferred	0.5	1.7	1.2	0.9	5.6	13.4

The Osborne mine has been mined to a depth of over 1,100 m below surface. Osborne mine is serviced by both a decline system and haulage shaft.

The mine will employ long-hole open stoping (LHOS) methods with longitudinal uphole retreat working a top down sequence. This is consistent with previous mining activities during 2009 and 2010 at Osborne. No backfill is used during the mining cycle; however, development mullock is placed in many of the open voids after mining is complete in the block.

During previous mining throughout the 400 mRL Block and other selected high grade areas, a bottom up extraction sequence was employed with paste fill to maximise extraction. All stopes between 355 mRL and 560 mRL have been filled with either paste or rockfill.

16.2.2 Geotechnical

16.2.3 Overview

The geotechnical studies undertaken for the Osborne underground deposit are:

- AMC Consultants, 2012. *Osborne Deeps Numerical Modelling (draft),* unpublished report by AMC Consultants for Ivanhoe Australia (author B Coombes), 23 p;
- Barrick, 2009. *Lower Mine Stress Analysis,* unpublished internal memorandum (author S Muir), 8 p;
- Barrick, 2009. *Geotechnical Review of Map3D stress model*, unpublished internal memorandum (author P Andrews), 10 p; and
- Western Australian School of Mines, 2009. *Stress measurements from oriented core using the Acoustic Emission method,* unpublished report by Western Australian School of Mines (authors: E Villaescusa, L Machuca), 20 p.

The lower levels of the Osborne underground operation are to be extracted in a top-down process, retreating towards the access. To ensure long term stability of the hangingwall, it is proposed that a combination of rib pillars and sill pillars are left to support the hangingwall.

It is expected that the rockmass conditions in the Osborne underground mine are similar to the rockmass conditions previously experienced. The intact rock strength of all sedimentary units is classified as very strong (UCS 95 – 200 MPa).

Stope Design Criteria

The geotechnical analysis completed, by S Muir for Barrick Ltd (Barrick), in 2009 for the stopes between 335 mRL and 135 mRL, recommended stope lengths of 29 - 35 m, with pillar design based on the following conditions:

- Sill pillars should have a minimum height of 5 m or a ratio of 0.5:1 based on the ore thickness;
- Rib pillars should have a minimum thickness of 5 m or a ratio of 0.5:1 based on ore thickness; and
- The factor of safety on rib pillars is 1.1 using degraded rockmass strength of 150 MPa. These recommendations were used for the preliminary design.

Further numerical modelling work completed, by AMC Consultants (AMC), in 2012 for the stopes between 135 mRL and 60 mRL recommended that the rib pillars are modified as follows:

- The 110D/E and 60D/E rib pillars were increased by 3.0 m in width. This resulted in optimised pillar width increasing from 6.0 m to 9.0 m;
- The 85D/E rib pillar was increased by 4.0 m in width. This resulted in optimised pillar width increasing from 6.0 m to 10.0 m width; and
- The 110E/F, 85E/F and 60E/F rib pillars were increased by 3.5 m, 6.0 m and 1.0 m in width respectively. This resulted in optimised pillar width being 11.0 m, 8.5 m and 7.0 m respectively.

AMC considered that the sill pillars recommended by Barrick geotechnical staff were adequate.

The AMC recommendations for the rib pillars and the Barrick recommendations for the sill pillars have been used for the mine design.

Ground Support Requirements

The existing Osborne ground support practices are planned to be continued for the lower levels. The Osborne ground support standards are:

- Ore drive development - split set support, with meshed backs, as per current Osborne standard practice is recommended to provide adequate support;
- Main Osborne decline and access cross cuts - resin grouted bolts and mesh as per current Osborne practice; and
- Large span intersections will require cable bolt support.

AMC also recommend that all drawpoints are cable-bolted.

Ore Drive Development

Ore drives are considered temporary excavations. A discrete wedge analysis conducted on the joint sets identified within the ore zone domain, indicates potential for wedges in the order of 5-10 t being formed. Split set support, with meshed backs, as per current Osborne standard practice, is recommended to provide adequate support. Large span intersections will require cable bolt support.

Decline Location

To complete the extraction of the Osborne underground operation, the current decline is to be extended at the 5.8 mH x 5.2 mW, 1:7 gradient as per the existing decline.

The design stand-off of 30 m is considered adequate, given the performance of the Osborne decline. The proximity of the decline to the stoping blocks in the historical Osborne underground workings has been 10-15 m, and the decline has remained relatively stable and largely unaffected by stoping.

Rock bolts and mesh are recommended as ground support in the access cross cuts, as per current Osborne standard practice.

16.2.4 Mine Design

Mining Method Description

LHOS was selected as the preferred mining method for the remaining Osborne Mineral Resource. LHOS was the primary method of extraction utilised at the Osborne operation and is a generally well understood and accepted extraction technique.

The orebody has widths varying from 4-20 m with an average of about 10 m and is dipping at 50° which suits LHOS. It has a generally competent hangingwall and footwall.

The sub-level spacing of 25 m floor to floor has been designed and includes consideration of stable spans and drilling accuracies.

The production cycle for LHOS includes the following:

- Develop access to the orebody;
- Develop bottom sill drive;
- Drill long holes to approximately 5 m from level above (max 34 m holes);
- Blast rings and extract ore; and
- Leave rib pillar and commence next stope.

Cut-Off Grade

Table 16-17 shows the cut-off grade assessment that has been undertaken for the Osborne underground. The inputs are based on the site costs for the previous year and the development contractor schedule of rates. The cut-off grade for Osborne underground has been determined to be 1.28% eCu. The cut-off grade for the Osborne Mine is 1.5% eCu. This has not been re-designed for the assessed cut-off grade because the development of the stopes is almost completed and is unable to be modified for a change in cut-off grade.

Table 16-17: Cut-off Grade Assessment

Item	Unit	Cost
Grade Control Costs	AUD/t	1.27
Development Costs	AUD/t	19.14
Production Drill and Blast	AUD/t	6.48
Production Loading and Backfill	AUD/t	3.18
Trucking to surface ROM	AUD/t	7.82
Other Costs – Technical Services, Maintenance, UG Services	AUD/t	11.84

Mining cost	Direct Mining Cost	AUD/t	49.73
	Sustaining Capital Allowance	AUD/t	0.00
Mill cost	Direct Milling cost	AUD/t	10.60
	Processing Sustaining Capital	AUD/t	0.50
G & A Cost	Direct G & A cost	AUD/t	7.30
Total Ore Cost		**AUD/t**	**68.13**
Processing Recovery	Copper Recovery	%	90
	Gold Recovery	%	80
Operations Freight Cost		AUD/t.conc	30
Export Shipping Cost		AUD/t.conc	119
Concentrate	Copper Grade	% Copper	24
	Contained Metal	t/t .conc	0.24
	Gold Grade	g/t	0.5
Payable Scale	Smelter Recovery deduction Copper	%	1
	Payable metal	t/t.conc	0.23
	Smelter Payable Factor Copper	%	95.8
	Smelter Payable Factor Gold	%	94.0
	Paid Gold	g/t	0.47
Treatment Costs	Concentrate Treatment Cost	AUD/t.conc	55
	Refining Charge	USD/lb	0.055
	Conversion lb to tonne	lb/tonne	2204
	Royalty	%	5

Item		Unit	Cost
Selling Cost Summary Copper	Concentrate treatment cost	AUD/t payable metal	234.04
	Refining cost	AUD/t payable metal	26.18
	Freight	AUD/t payable metal	635.74
	Total	**AUD/t payable metal**	**895.97**
	Concentrate treatment cost	AUD/t.conc	55.00
	Refining cost	AUD/t.conc	26.18
	Freight	AUD/t.conc	149.40
Total Concentrate Costs	**Total**	**AUD/t.conc**	**230.58**
Metal Price Assumptions	Copper metal	USD/lb	3.25
	Gold metal	USD/oz	1,400.00
	Exchange rate	AUD:USD	1.00
	Copper metal	AUD/lb	3.25
	Gold metal	AUD/oz	1,400.00
	Copper metal	AUD/t	7,163.00
	Gold metal	AUD/g	45.01
Revenue	Copper metal	AUD/t.conc	1,683.31
	Gold metal	AUD/t.conc	12.69
	Total		**1,696.00**
Net Smelter Return		AUD/t.conc	1,465.42
Royalty Charge		AUD/t.conc	71.80
Net Smelter Return after Royalty		AUD/t.conc	1,393.62
Calculated Cut-off	Revenue at Concentrate	AUD/t contained copper	5,930.29
	Mill Recovery	%	90
	Revenue at Ore	AUD/t copper in ore	5,337.26
	Operating Costs		68.13
Breakeven Grade		eCu	**1.28**

Material Handling

All underground production ore will be trucked to the 676 mRL where the ore will be dumped into the crushing system. The ore is fed through a system of chain controls and plate feeder to an underground crusher. Crushed ore is dropped into a holding bin with 5,000 t capacity, directly over a reciprocating plate feeder at the loading station. The ore is dropped directly from the plate feeder into a loading flask then loaded into 14 t skip to be hoist to surface. The haulage system has no underground conveyors and is rated at 1.5 Mtpa.

The haulage shaft is 700 m deep, 3.6 m diameter blind bored shaft equipped with a double drum winder with a skip / man riding cage combination with a counterweight. The 14-man capacity, man riding cage is fixed above the skip and is used to efficiently move personnel in and out of the mine at shift start and finish. The required hoisting rate to achieve the production schedule proposed is approximately 0.7 Mtpa and well within the capability of this equipment.

All other waste will be backfilled into stope voids the previous workings of Osborne underground.

16.2.5 Mine Design Guidelines

Design Parameters

The mine design parameters that have been used in the design for the Osborne Mine are summarised in Table 16-18.

Table 16-18: Mine Design parameters

Item	Size	Gradient
Decline	5.8 mH x 5.2 mW	1:7 down
Level Access	5.5 mH x 5.0 mW	1:50
Ore Drives	5.5 mH x 5.0 mW	1:50
Slot Drive	5.5 mH x 5.0 mW	1:50
Escapeway Access	5.5 mH x 5.0 mW	1:50
Vent Access	5.0 mH x 5.0 mW	1:50
Sumps	5.0 mH x 5.0 mW	1:7 down
Ventilation Raise (Longhole)	4.0 mH x 4.0 mW	
Escapeway Raise (Raisebore)	1.5 m diameter	< 60 degrees
Sub level Spacing	25 m	
Sill Pillar	> 5 m	
Rib Pillar	>7 m	

Mining Sequence

The mining sequence will follow a top-down sequence retreating both from the southern and northern extent back towards a central access. The sequence to the 135 level was previously assessed by Barrick for potential stress problems using 3D elastic stress modelling. Stress results in the lower central pillar were less than 30% of the intact rock strength, and the shear stress induced along structures parallel to the orebody was in the range of 3-5 MPa. The Barrick stope design parameters were used in conjunction with numerical modelling undertaken by AMC, (AMC 2012). AMC's recommendations for the sill and rib pillar dimensions have been applied for the stopes between 135 mRL and 60 mRL. The modelling results are considered first pass only because there are no recent performance observations available upon which to base a calibration. Further analysis will be required during stoping.

Figure 16-14 indicates the proposed mining method for Osborne underground.

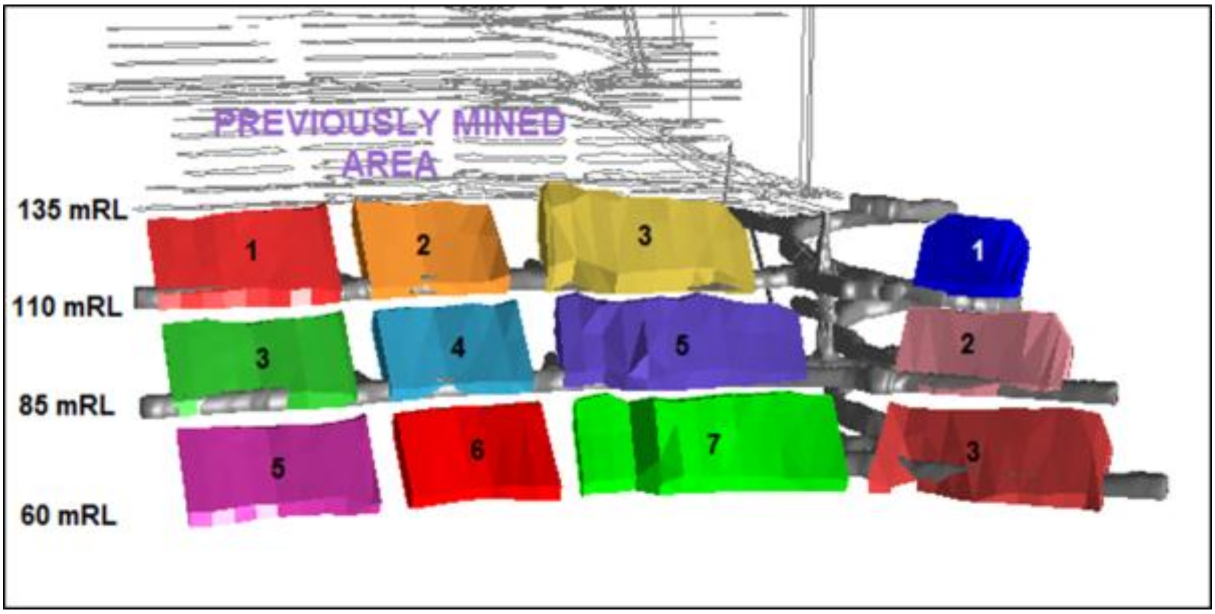


Figure 16-14: Long section of stope mining sequence between 135 mRL and 60 mRL

Decline Development

Access to the remaining Mineral Resource is via the existing decline down to the 60 mRL access. The decline was designed to continue in the footwall with a stand-off distance of a minimum 30 m from the proposed stopes. The decline development is nominally 5.8 mH x 5.2 mW.

Level Development

On each main level access there is an internal vent access and a sump. The orebody development is designed to follow the footwall in most cases to minimise the hangingwall cross-sectional area of pillars in Figure 16-15 and provide simple draw of the stope material from the lower level. The very strong and competent hangingwall should not be affected greatly by the drilling of blast holes toward the hangingwall. This methodology is a continuation of the process previously used at Osborne that achieved good reliable results.

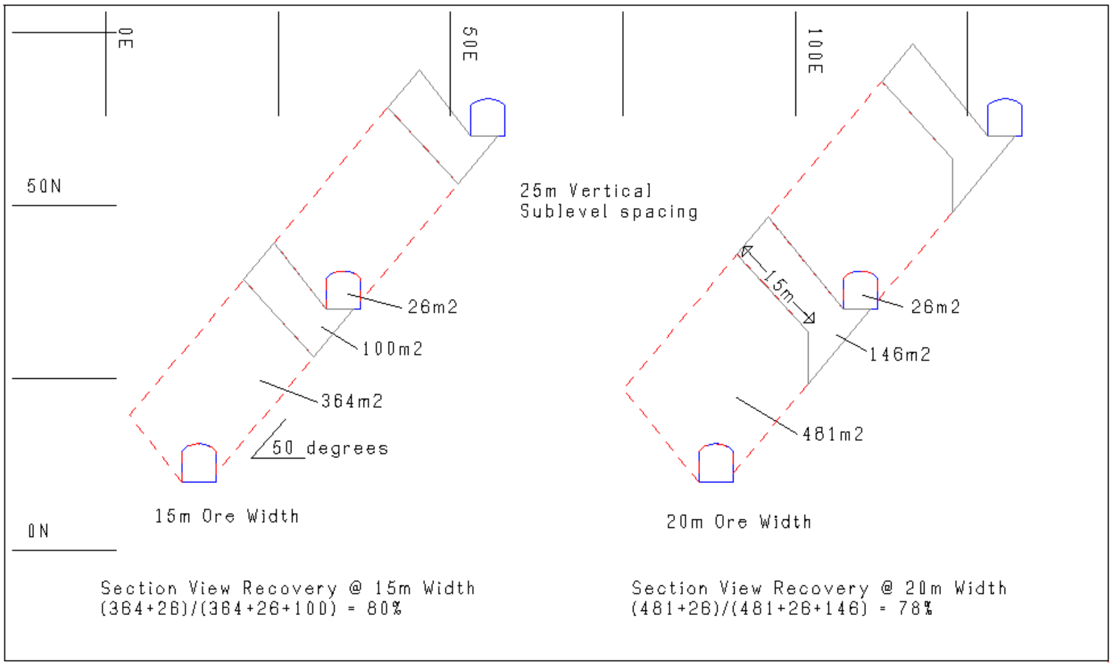


Figure 16-15: Typical sections through blind uphole retreat stopes in lower sections of Osborne

Vertical Development

The return airway ventilation circuit connects to the existing return airway raise on the 135 mRL. The vent rises connect to the existing circuit and subsequently to the other levels are 4.0 mH x 4.0 mW. The ladder way system connects into the existing emergency egress system with a 1.5 m diameter and at less than 60 degrees from horizontal.

Stope Inventory

Table 16-19 summarises the underground inventory for Osborne and Figure 16-16 shows the layout of the stopes.

Table 16-19: Osborne stope inventory

Level	Stope	Diluted Stope Tonnes			
		Tonnes	Copper %	Gold g/t	eCu
110	D	34,586	2.29	1.15	2.98
110	E	38,374	1.94	0.91	2.48
110	F	96,513	2.67	1.04	3.30
110	G	18,428	1.27	0.76	1.73
85	D	26,483	1.88	1.02	2.49
85	E	34,179	1.96	0.86	2.47
85	F	79,848	2.42	0.98	3.01
85	G	30,388	1.28	0.69	1.69
60	D	22,318	1.37	0.98	1.96
60	E	30,647	1.54	0.84	2.05
60	F	79,858	1.63	0.80	2.11
60	G	34,777	1.17	0.71	1.60
Stope Total		**526,399**	**1.96**	**0.91**	**2.51**
Development Tonnes		99,181	1.61	0.79	2.09
Total		**625,580**	**1.91**	**0.89**	**2.44**

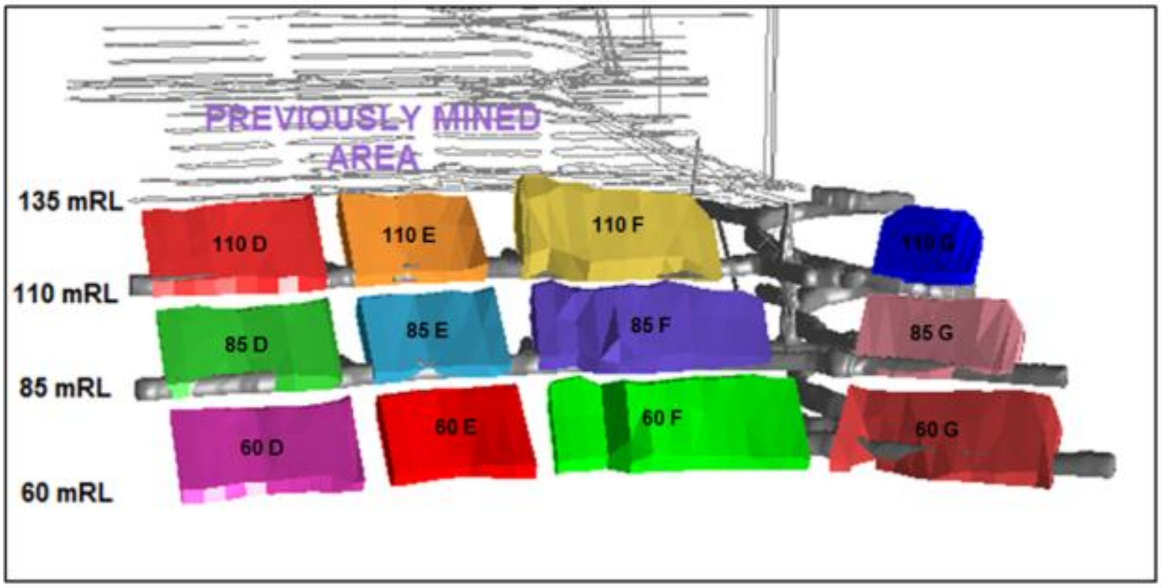


Figure 16-16: Long section of designed stopes between 135 mRL and 60 mRL

16.2.6 Ventilation

Existing Circuit

Figure 16-17 shows the ventilation circuit at Osborne. Fresh air flows down the decline from the main portal and the 1125 mRL portal and the fresh air shaft which delivers chilled air to the decline system at 365 mRL, over 900 m below surface. The haulage shaft also provides fresh air into the mine at 1000 mRL and 650 mRL crib room facilities and around the crushing and shaft loading infrastructure.

Fresh air from the decline is force ventilated into the active levels. Each level has a return air raise (RAR) which is connected to the exhaust ventilation system powered by twin primary fans located in the Osborne open pit. The fans draw a total of 270 m^3/s of air from the mine.

The RAR system consists of long raise-bored shafts in the hangingwall between the 925 mRL, 755 mRL, 660 mRL, 290 mRL and 135 mRL. These rises are supplemented with shorter blasted rises (15 to 20 m long, 4 m x 4 m) between each level. These are excavated to establish return ventilation as soon as each new level is developed off the decline. There are currently 16 such raises between 520 mRL and 135 mRL.

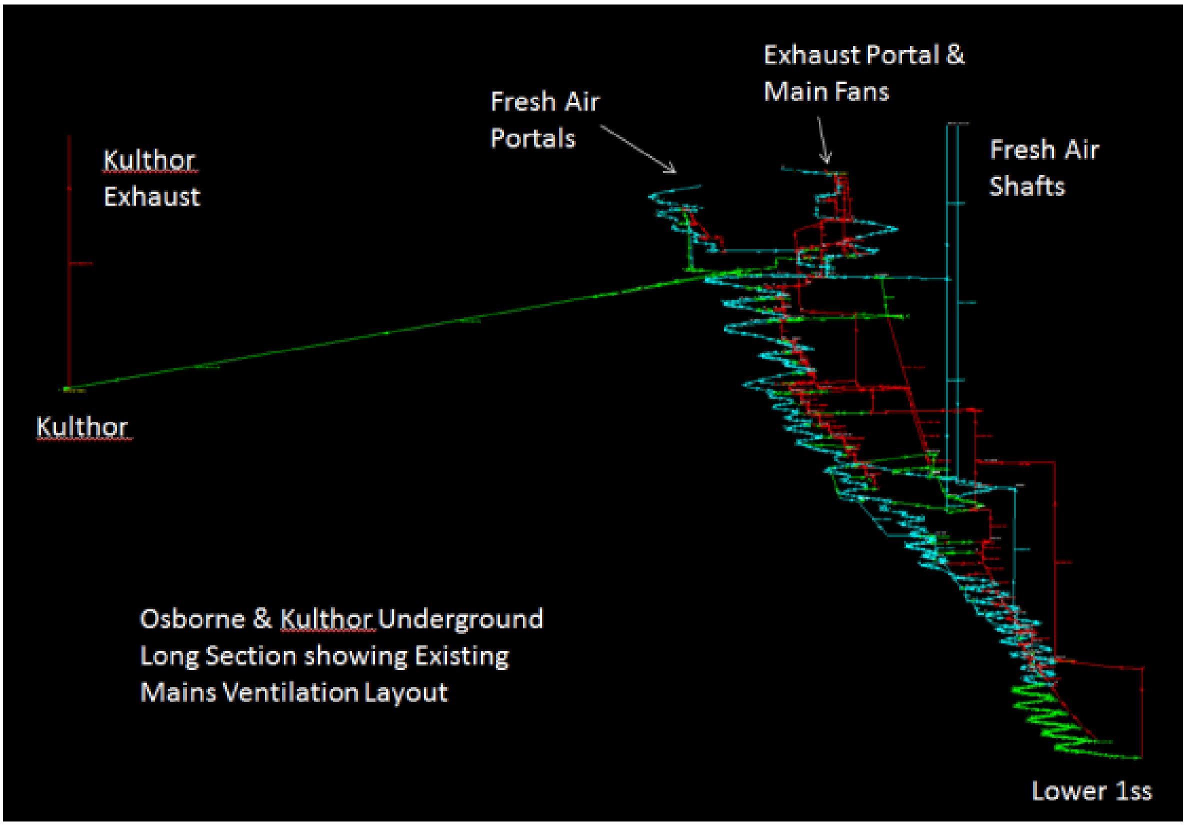


Figure 16-17: Osborne ventilation circuit

Ventilation Circuit

The ventilation network for Osborne underground utilises the existing ventilation infrastructure.

Fresh air from the decline is force ventilated into the active levels. Each level has a RAR which is connected to the exhaust ventilation system powered by twin primary fans located in the Osborne open pit.

A second means of egress is made up of ladder ways and an emergency hoist facility within the fresh intake shaft.

Airflow Requirements

The minimum airflow required in Queensland is stated below as per the Mining and Quarrying Safety and Health Regulation 2001.

"A person who has an obligation under the Act to manage risk in relation to ventilation at a mine must ensure appropriate measures are taken to ensure the ventilating air in a place where a person may be present at the mine is of a sufficient volume, velocity and quality to achieve a healthy atmosphere."

Osborne has historically used 0.04 cubic metres per second per kilo watt (m^3/s/kW) for all diesel equipment as a minimum standard. In light of the hot conditions experienced in summer at Osborne, 0.07 m^3/s/kW will be used as a minimum. From past experience at Osborne, 0.04 m^3/s/kW was insufficient during the warmer, more humid months of December, January and February.

Decline development has been stopped on several occasions in the past for up to 8 weeks due to excessively hot conditions arising from inadequate airflow. A 2.0 MW air chilling plant was commissioned in 2008 to improve the conditions at depth. The system chills all air entering the lower region of the mine via the fresh air shaft 365 mRL.

The scheduled machine usage and diesel kW unit calculation estimates the total kW for the equipment to be 3,275 kW during peak mining periods. When applying 0.07 m^3/s/kW, the required total air flow is 239 m^3/s. The current infrastructure – which will be utilised –is capable of supplying approximately 440 m^3/s, including exhaust from Kulthor, which has sufficient capabilities to supply the required airflow.

Raise sizes

The sizes of the existing raises are shown in Table 16-20.

Table 16-20: Osborne Mine Ventilation Raises

Raise	Diameter (m)
Fresh Air Rise (surface to 365 mRL)	3.6
Hoisting Shaft (Fresh Intake)	3.5
Kulthor Return Air Rise	4.0
Kulthor Fresh Air Rise	2.4

If additional ore is discovered down dip or production rates are increased, the main vertical airways have potential to accommodate additional air flow. The main exhaust airway has a velocity of 11 metres per second (m/s) and the intake airway 12.5 m/s. As the recommended upper velocity is 20 m/s for such shafts, additional capacity is apparent.

Conventional practice recommends vertical exhaust airway velocities between 7 and 12 m/s be avoided to eliminate the possibility of suspended water. Suspended water, created by condensation, has the potential to place a fan in stall or introduce vibration problems within an airway. Historically Osborne's exhaust airway velocities range between 7 and 10 m/s. Suspended water problems have not been detected or experienced in the past at Osborne, therefore the main exhaust airway velocity of 11 m/s is not anticipated to cause problems.

Intake ground water also has the potential to cause the problems discussed above. Osborne historically has been a relatively dry mine, hence ingress water is not expected to be problematic.

Egress ladder ways have a chosen diameter of 1.8 m. This diameter allows ladder way installation to be carried out easily and service lines incorporated at a later date (i.e. paste fill and dewatering lines).

16.2.7 Backfill

Stopes will be backfilled with development waste from the Osborne Mine. Kulthor waste will also be tipped into Osborne stopes until a suitable backfill location is available at Kulthor. All tip heads in these areas will be stripped out, rehabilitated for ground support and stop blocks placed to ensure safe truck tipping.

16.2.8 Mine Services and Infrastructure

Electrical and Communications

Power for Osborne underground is provided by Osborne's onsite power station.

An allowance has been made for a 20 m service hole to be drilled between sub-levels on each access. This will allow optimum placement of jumbo boxes and distribution boards and reduce cable runs in the decline throughout the project.

Osborne underground uses a leaky feeder system for communications with a dedicated emergency channel.

Compressed Air

The underground operations at Osborne are supplied with compressed air via 4-inch poly lines in the main decline and 2-inch lines on each sub-level. The lines are fed by the existing surface compressor plumbed into the underground workings.

Raw and Potable water

Raw and potable water are provided to Osborne via the existing system in place from the original operations.

Explosives

All storage, transport and handling of explosives have been assumed to be in accordance with the Australian Standards and the current Queensland Mine's Regulations and Act.

A third party contractor is utilised for the duration of the project. It has also been assumed that Osborne will use the contractor-supplied production charging unit.

The designated contract supplier batches all ANFO and Emulsion on site. It is assumed that primarily ANFO would be used for development and production, with emulsion used in wet conditions.

Magazine

The current Osborne magazine is located off the main decline (approximately 1108 mRL) which will be operated by the Osborne designated explosives contract supplier. The magazine is used to store explosives for the recommenced underground Osborne project. The magazine at the 1108 mRL also serves as the lockup for the development charging unit outside charging operations.

Firing System

At firing the existing Mains Electrical Firing system will be used for blasting.

Emergency Egress and Refuge Chambers

A combination of ladder ways and refuge chambers has been adopted for emergency management within the mine.

Emergency Egress

Ladder way design is the same as that adopted previously at Osborne (i.e. enclosed ladder way with drop bars for personnel resting) which will link in with the existing secondary egress.

There is also potential to use the Kulthor Vent Raise 1 emergency egress, as an alternative is necessary during an evacuation.

Refuge Chambers

The final refuge chamber locations, during the production phase, will enable all personnel to be within 1,000 m of a refuge chamber or fresh air source.

It is not intended for refuge chambers to substitute as a second means of egress, but to provide refuge during fire or containment when ladder ways may be inoperative or inaccessible.

16.2.9 Hydrogeology / Dewatering

Hydrogeology

The hydrogeological regime at the Osborne site is considered to consist of three aquifer systems, namely the Mesozoic Cover, the Kulthor Shear Zone and the generally massive Proterozoic rock hosting the orebody. Whilst minor groundwater inflows may be derived from the Mesozoic and the Proterozoic host rock, the major risk to mining is related to the water held in storage within the Kulthor Shear Zone. Test pumping of a bore constructed in the Shear Zone has been undertaken at a constant rate of over 10 L/s by Australasian Groundwater and Environmental Consultants Pty Ltd (AGE). This test work demonstrated the highly permeable nature of this shear. If mining or development intersects fractures that are hydraulically connected with the shear zone, sustained inflows of groundwater can be expected.

In order to reduce the risk of this occurrence, it is recommended that pumping for dewatering purposes continues from the existing bore (KWB001). Monitoring of the progress of dewatering should be continued. Supplementary dewatering bores may be required if monitoring indicates that effective dewatering cannot be achieved with a single bore within the time frame required.

Hydraulic compartmentalisation of the Shear Zone may also necessitate the need for additional dewatering bores. Dewatering bores will not fully dewater the Shear Zone and as such provision for a pumping station located at the upper levels of the mine should be made.

Water quality from the Shear Zone is of neutral pH with a salinity of about 5600 milligrams per litre (mg/L). This water pumped from KWB001 is currently used in the existing mine mill, reducing the need for pumping from the mine’s water supply bores.

Additional groundwater issues related to depletion of the aquifer and subsequent impact on other nearby groundwater users are considered low risk and can generally be managed through review of existing data and negotiations with Environmental Performance Evaluation (EPA) and Natural Resource Management (NRM).

Dewatering

Osborne underground is dewatered by the existing rising main and pumping infrastructure. The internal pumping system will need to be extended as the mine progresses vertically, using standard decline dewatering and pumping systems to deliver water to the main pumping system to deliver water to the surface rising main.

16.2.10 Mining Schedule

Scheduling Strategy

The scheduling strategy for the mine is:

- Stopes to commence production as soon as possible;
- Production drilling to be completed on a level before production commences;
- Level development a priority over the decline development; and

- All level development, vent rises, sump and escapeways, completed before stoping commences on a level.

Because the ore is to be hoisted to a ROM pad prior to being fed into the processing facility, there is a lag between ore being extracted from underground and being processed. It is expected that, at an operational planning phase, greater detail will be provided to understand specific material movements.

All tonnes and grade quoted have been modified by mining recovery of 91% and mining dilution of 10%. These are based on historical performance at Osborne.

Development Schedule

There is approximately 200 m of development to be completed in the Osborne Mine. This is anticipated to be completed by the end of July 2012.

Production profile

Figure 16-18 summarises the scheduled tonnes of ore production from Osborne underground operation. The production reaches approximately 60,000 tonnes per month for five months before ramping down to 40,000 tonnes in the final four months. The mine life will be ten months.

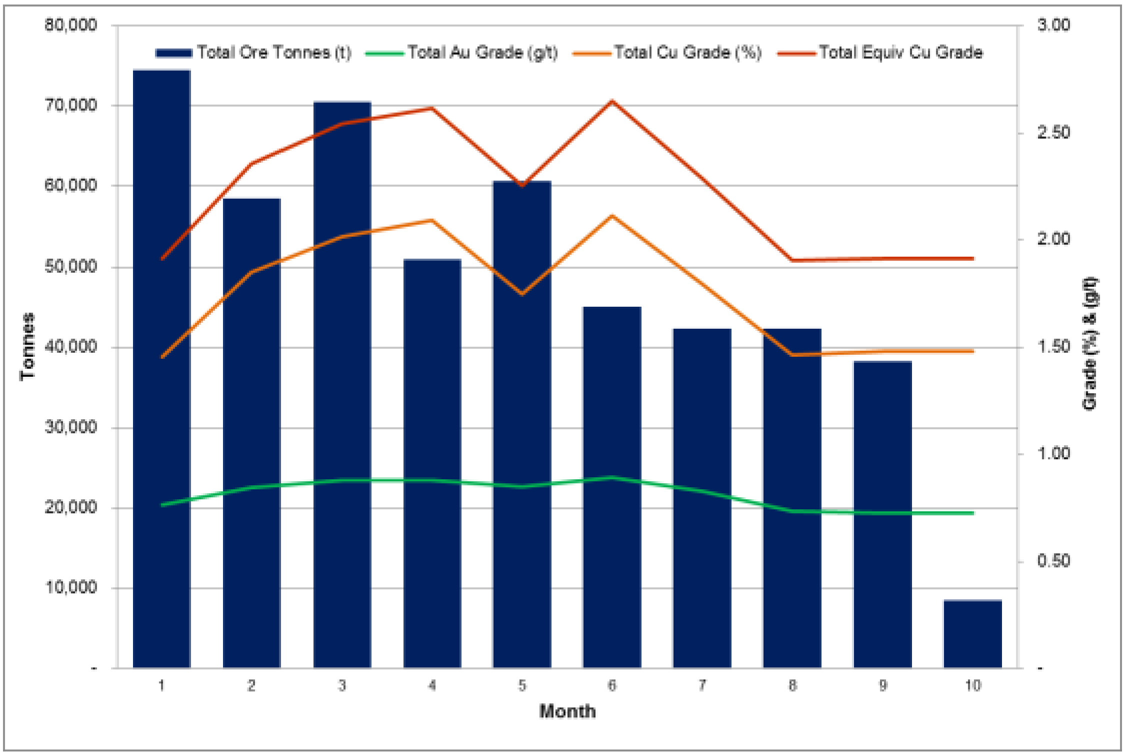


Figure 16-18: Production schedule with gold and copper grades

Production Drilling Schedule

Figure 16-19 shows the production drilling profile for Osborne underground operation. Production drilling has only two remaining month before the stopes are all drilled out for production as at the beginning of June 2012.

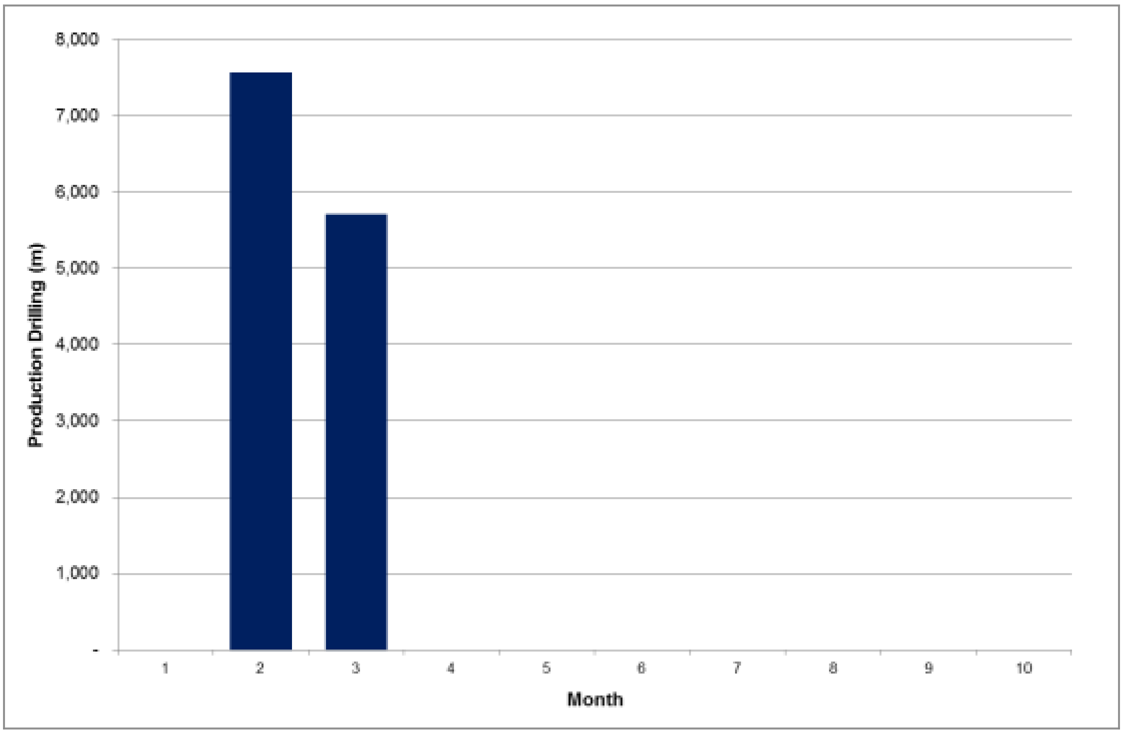


Figure 16-19: Production drilling requirements

Manpower and Supervision

Shift Schedule

Mining costs have been estimated using a continuous mining operation, 24 hours a day, 365 days per year. All employees will commute from Townsville and utilise the Osborne village while on site.

Operators and maintenance personnel work 14 days on, then 7 days off, 12-hour shift alternating between dayshift and nightshift. Blast crew work is 14 days on, then 7 days off, and is a 12-hour dayshift only.

Ivanhoe support staff will work 8 days on, then 6 days off, with a 12-hour dayshift only.

All on-costs for annual / sick leave and training have been estimated in the operating costs.

Personnel Levels

All equipment has been assigned with one operator per crew per machine. It is assumed that cross training will occur for all operators, ensuring that each shift panel is adequately multi-skilled to relieve for sickness, annual leave and general absenteeism.

Personnel numbers for each position and total work force levels are shown in Table 16-21. The underground staff mining personnel are shared with the Kulthor deposit.

Table 16-21: Underground staff mining personnel requirements

Staff	Number
Superintendent	2
Underground Supervisors	4
Underground Technical Services Personnel	8
Administration & Pitram Operators	4

Table 16-22 lists the indicative mining personnel requirements for underground mining at Osborne.

Table 16-22: Underground shift mining personnel requirements

Personnel	Number
Jumbo Operators	8
Loader Operators	8
Truck Drivers	16-20
Longhole drillers	4
Blasting Personnel	4

Labour Costs

Labour costs have been estimated and include superannuation, workers compensation, payroll tax and partial allowances for leave accrual. The labour costs are incorporated in the unit costs for the mining and processing activities.

16.3 Kulthor Underground

16.3.1 Introduction

Ivanhoe proposes to mine Kulthor mine which is located 2.5 km west of the Osborne Mine. Access to the Kulthor mine will be through the existing Osborne Mine. The Kulthor Mine is scheduled to produce at a maximum rate of 720 kt per annum using open stoping. The Kulthor mine ore will be truck hauled to 1000 mRL ore pass for crushing and hoisting to surface via the existing Osborne shaft.

16.3.2 Geotechnical

Northwind Enterprises Pty Ltd (Northwind) undertook a review of a Pells Sullivan Meynink Pty Ltd (PSM) geotechnical study of the Kulthor deposit for the 2007 Kulthor Deposit Feasibility Study. The following section has been summarised from the Northwind report.

The geotechnical database is extensive and no additional geotechnical data has been collected since the initial Pre-Feasibility Study (PFS) was completed in 2004.

PSM geotechnically assessed LHOS and benching mining methods for the deposit. For the assessment the orebody was broken into three main geological areas:

- Main (Western) Shoot;
- Northern (North end of the Western) Shoot; and
- Central (Central) Shoot.

Table 16-23 summarises the geotechnical stoping parameters reviewed by Northwind for the LHOS and benching mining methods.

Table 16-23: Summary of geotechnical stoping parameters (Northwind 2007)

Ore Shoot	Surface Dip	Q'			N'	Design HR	Pillar Dimensions (m)
		Minimum	Average	Maximum			
Western Shoot	Wall 70-80	10	46	100	25	8.1	-
Central Shoot	Wall 70-80 Wall 30-60	5 10	61 61	150 200	44 27*	7.8 9.5*	-
Northern Shoot	Wall 70-80	10	16	50	11	6	6 x 5

*section where flatter hangingwall occurs in central/west lode

No direct stress measurements have been undertaken for Kulthor; however, the Osborne Mine stress regime has been adopted for Kulthor, given the close proximity and the similar nature of the geometry and orebody geology. This is a reasonable assumption, given the consistency between the Osborne Mine stress measurement, the nearby Cannington Mine stress regime, and the general stress regime for this region of Queensland.

Additional geotechnical work has been completed by AMC Consultants in 2012. This work has determined that the stope dimensions in Table 16-24 are used for the Kulthor Mine.

Table 16-24: Kulthor Stope Design Parameters (AMC, 2012)

Stope Category	Shear Zone Distance (m)	Stope Dimensions			
		Strike Length	Transverse Width	Stope Height	Cablebolt Support
1	0.0 – 3.0	25 m	15 m	30 m	No
2	3.0 – 6.0	20 m	15 m	30 m	Footwall
3	6.0 – 10.0	30 m	15 m	30 m	No
4	10.0+	40 m	12 m	30 m	No

AMC recommend that that the rib pillar design considers the minimum strike length equal to the true width of the stope (1:1). Rib pillars at this size have reasonable performance to provide permanent support to the hangingwall and footwall. The rib pillar spacing was determined based upon the initial empirical stable stope span assessment discussed prior. Once stoping commences, the empirical designs are to be calibrated based upon continual observation and recording of rib pillar performance.

16.3.3 Mine Design

Method selection

As part of Kulthor Feasibility Study (2007), a series of trade-off studies were conducted to determine the optimum mining method. The mining methods selected for the Kulthor deposit are:

- Longhole open stoping (LHOS); and
- Longhole bench and fill (LHBF).

Longitudinal sub-level caving has been considered for the deposit, but further geotechnical studies to be undertaken have determined that this mining method is not suitable for the deposit.

Mining Method Description

Uphole Open Stoping

The orebody has widths between 5 - 15 m with an average of about 10 m and is steeply dipping, which suits uphole stoping. It has generally a competent hangingwall and footwall. Uphole open stoping was the primary method of extraction utilised at the Osborne operation and is a generally well understood and accepted extraction technique.

The sub-level spacing of 30 m floor to floor has been designed taking consideration of stable spans and drilling accuracies.

The production cycle illustrated in Figure 16-20 for uphole open stoping includes the following:

- Develop access to the orebody;
- Develop bottom sill drives;
- Drill upholes to the level above;

- Blast between levels and extract ore; and
- Leave rib pillar and commence next stope.

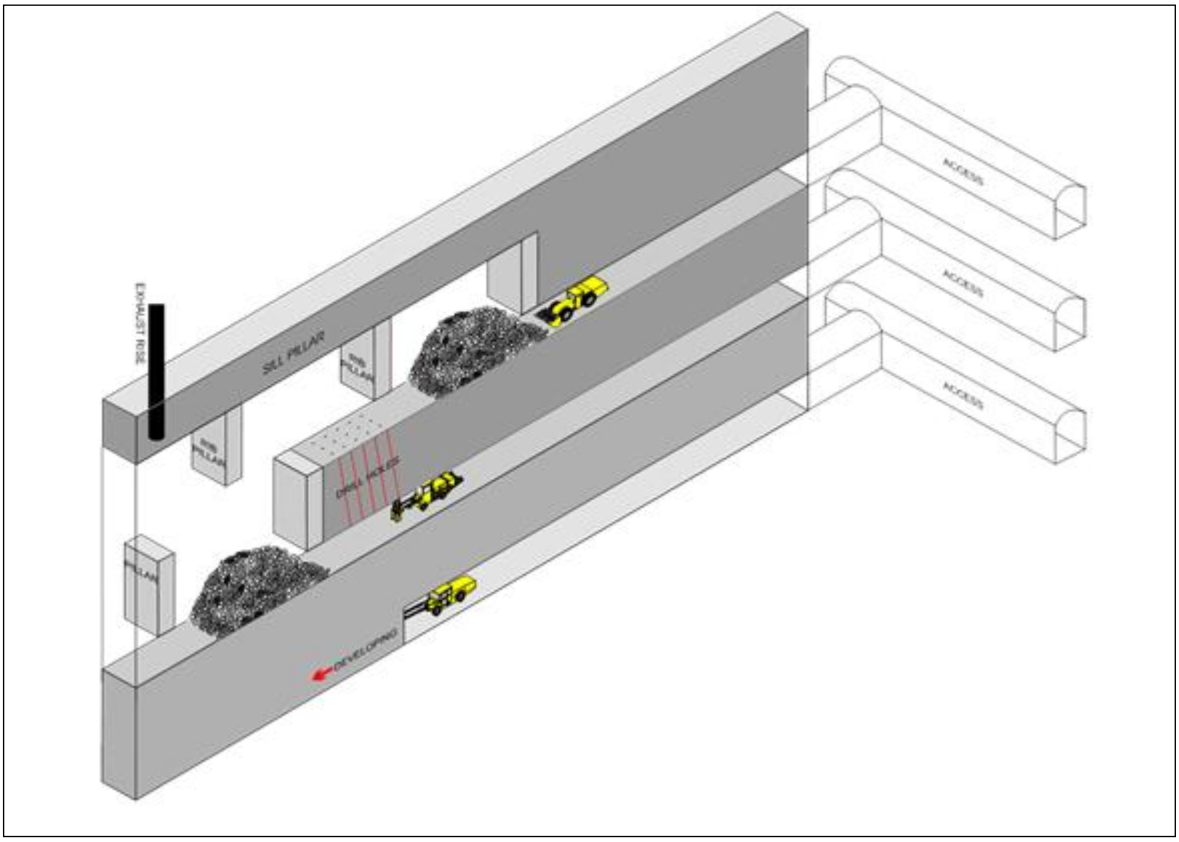


Figure 16-20: Uphole Open Stoping with Rib Pillars

Cut-Off Grade

Table 16-25 shows the cut-off grade assessment that has been undertaken for the Kulthor Mine. The inputs are based on the site costs for the previous year and the development contractor schedule of rates. The cut-off grade for Kulthor has been determined to be 1.40 eCu.

Table 16-25: Cut-off Grade Assessment

Item	Unit	Cost
Grade Control Costs	AUD/t	1.27
Development Costs	AUD/t	26.96
Production Drill and Blast	AUD/t	6.48
Production Loading and Backfill	AUD/t	3.18
Trucking to surface ROM	AUD/t	4.84
Other Costs – Technical Services, Maintenance, UG Services	AUD/t	11.84

Mining cost	Direct Mining Cost	AUD/t	53.32
	Sustaining Capital Allowance	AUD/t	0.50
Mill cost	Direct Milling cost	AUD/t	10.60
	Processing Sustaining Capital	AUD/t	0.00
G & A Cost	Direct G & A cost	AUD/t	7.30
Total Ore Cost		**AUD/t**	**71.72**
Processing Recovery	Copper Recovery	%	85

Item		Unit	Cost
	Gold Recovery	%	75
Operations Freight Cost		AUD/t.conc	30
Export Shipping Cost		AUD/t.conc	110
Concentrate	Copper Grade	% Copper	24
	Contained Metal	t/t .conc	0.24
	Gold Grade	g/t	0.5
Payable Scale	Smelter Recovery deduction Copper	%	0
	Payable metal	t/t.conc	0.24
	Smelter Payable Factor Copper	%	95.8
	Smelter Payable Factor Gold	%	94.0
	Paid Gold	g/t	0.47
Treatment Costs	Concentrate Treatment Cost	AUD/t.conc	55
	Refining Charge	USD/lb	0.055
	Conversion lb to tonne	lb/tonne	2204
	Royalty	%	5
Selling Cost Summary Copper	Concentrate treatment cost	AUD/t payable metal	229.17
	Refining cost	AUD/t payable metal	26.74
	Freight	AUD/t payable metal	585.00
	Total	AUD/t payable metal	840.91
	Concentrate treatment cost	AUD/t.conc	55.00
	Refining cost	AUD/t.conc	26.74
	Freight	AUD/t.conc	140.40
Total Concentrate Costs	**Total**	**AUD/t.conc**	**222.14**
Metal Price Assumptions	Copper metal	USD/lb	3.25
	Gold metal	USD/oz	1,400.00
	Exchange rate	AUD:USD	1.00
	Copper metal	AUD/lb	3.25
	Gold metal	AUD/oz	1,400.00
	Copper metal	AUD/t	7,163.00
	Gold metal	AUD/g	45.01
Revenue	Copper metal	AUD/t.conc	1,719.12
	Gold metal	AUD/t.conc	12.69
	Total		**1,731.81**
Net Smelter Return		AUD/t.conc	1,509.67
Royalty Charge		AUD/t.conc	73.92
Net Smelter Return after Royalty		AUD/t.conc	1,435.75
Calculated Cut-off	Revenue at Concentrate	AUD/t contained copper	5,982.29
	Mill Recovery	%	85
	Revenue at Ore	AUD/t copper in ore	5,084.95
	Operating Costs		71.72
Breakeven Grade		eCu	**1.41**

Mine Design Parameters

The mine design parameters used for the Kulthor Mine are shown in Table 16-26. These parameters are based on the geotechnical parameters discussed in Section 16.3.2. The maximum stope dimension used has assumed that the shear zone is within 3 – 6 m of the stope wall for the stope layout.

Table 16-26: Kulthor Mine Design Parameters

Item	Units	Dimension
LHOS / LHBF		
Minimum mining width	m	4
Maximum stope length	m	20
Minimum pillar thickness	m	Width of stope
Cut-off grade (design)	%	1.40 eCu
Production drilling - 89mm	m drilled / tonne	10
Sublevel Spacing	m	30
Lateral Development		
Decline	m	5.8 H x 5.2 W 1:7 down
Ore Development	m	5.5 H x 5.0 W
Stope Level Development	m	5.5 H x 5.0 W
Ventilation Access	m	5.5 H x 5.0 W
Sumps, Stockpiles, Escape-way Access	m	5.5 H x 5.0 W
Vertical Development		
Ventilation Raise	m diameter	4
Ventilation LH Raise	m	4.0 H x 4.0 W
Escape-way	m diameter	1.8

Surface Infrastructure

The Cannington Lateral on the Carpentaria High Pressure Gas Pipeline is located on the surface above the Kulthor deposit. The operators of the gas pipeline, Alinta Asset Management Pty Ltd, have provided guidance that blasting within 500 m of the pipeline must be strictly controlled and monitored (Alinta, 2007). Figure 16-21 shows the stopes that are affected by the location of the gas pipeline. Figure 16-22 shows the location of the gas pipeline in plan view and where the stopes are located below the pipeline.

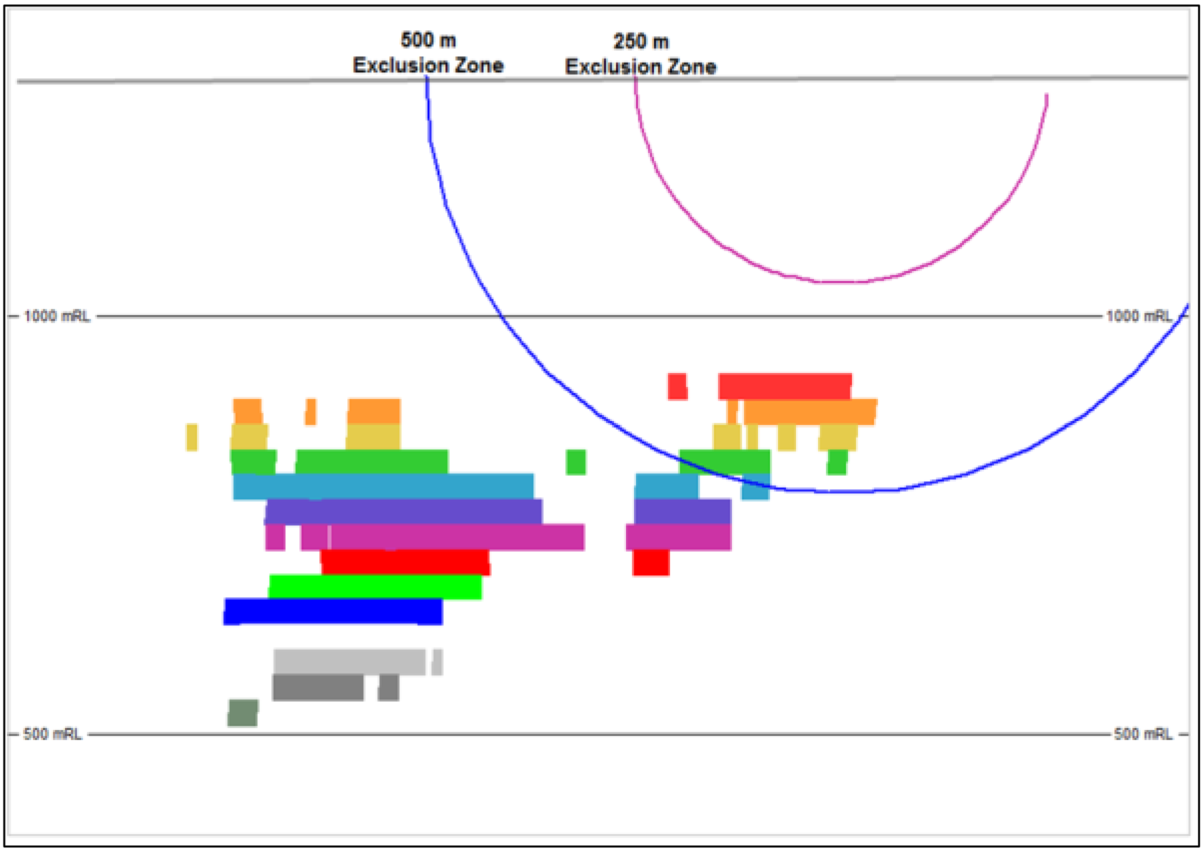


Figure 16-21: Gas Pipeline Monitoring Area

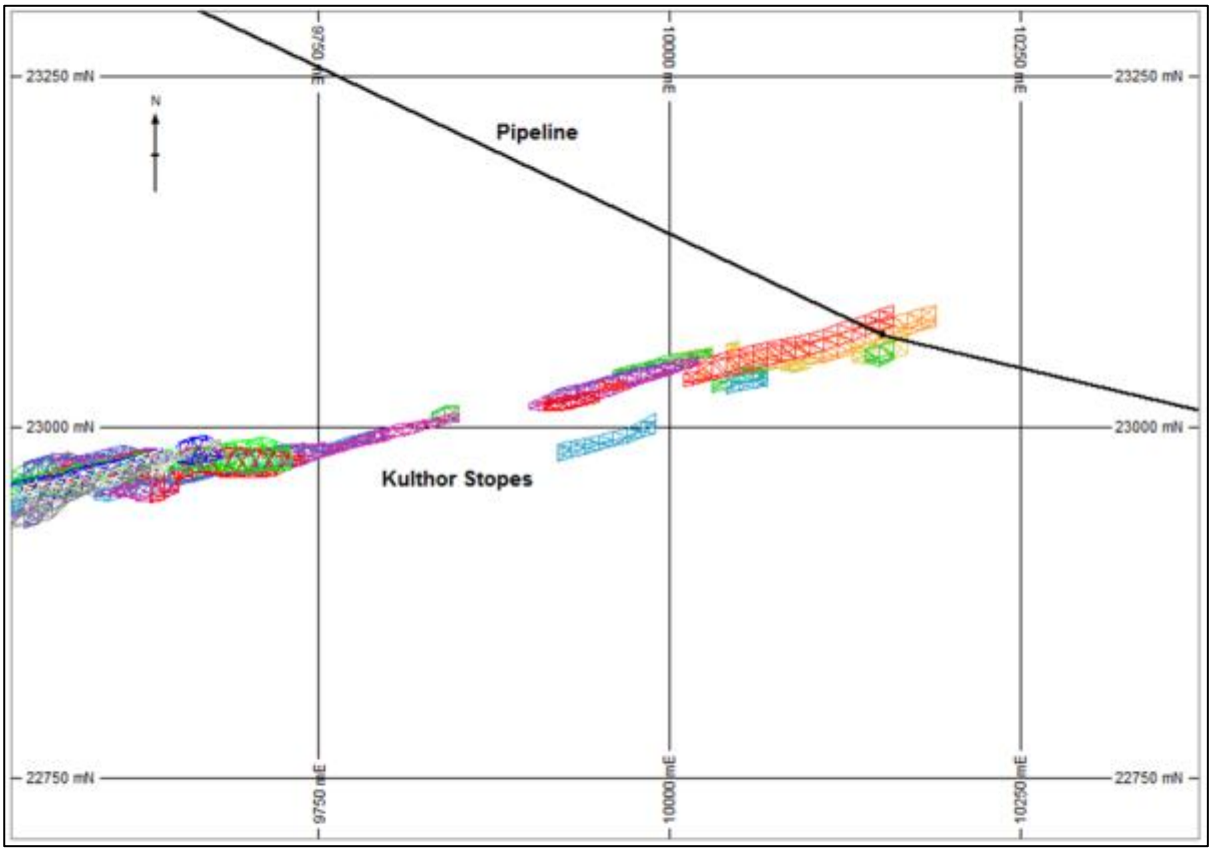


Figure 16-22: Surface Plan of Gas Pipeline location

Existing Development

The access drive from the Osborne Mine is completed and development on 720 mRL, 750 mRL, 780 mRL, 810 mRL and 840 mRL has commenced to access the stoping areas. The K1 and the surface-to-840 ventilation shafts have been completed and the 830 internal ventilation shaft between 830 mRL and 715 mRL has also been completed. Ladderways have also been completed between 830 mRL and 715 mRL. Figure 16-23 shows the existing development at the Kulthor Mine.

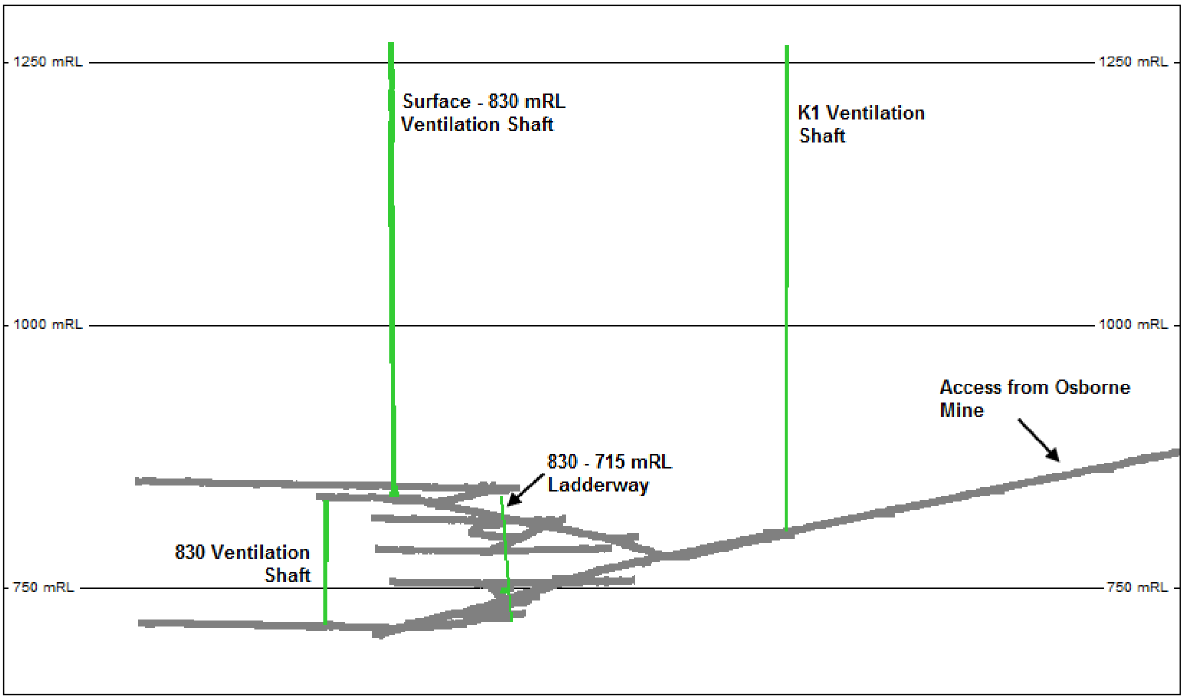


Figure 16-23: Kulthor Mine Existing Development

Access Development

The access to the levels has been designed from the existing access from Osborne and from the 710 mRL; a decline is mined to access the levels. The decline stand-off distance is 30 m from the proposed stopes. The decline development is nominally 5.8 mH x 5.2 mW. Each level has an access from the decline to the stoping areas.

Level Development

The level development is designed to be in the ore and at the base of each stope. The development arrangement restricts the stoping sequence to a retreat sequence along the level back to the level access. Access to the ventilation system has been designed for each level.

Vertical Development

The designed vertical development includes internal ventilation raises which connect each level to the surface ventilation shafts. Ladderways have also been designed to link the levels back to the existing ladderways.

Mine Layout and Inventory

Figure 16-24 shows the layout of the Kulthor Mine. The mining sequence is top-down and retreating along the level back to the level access.

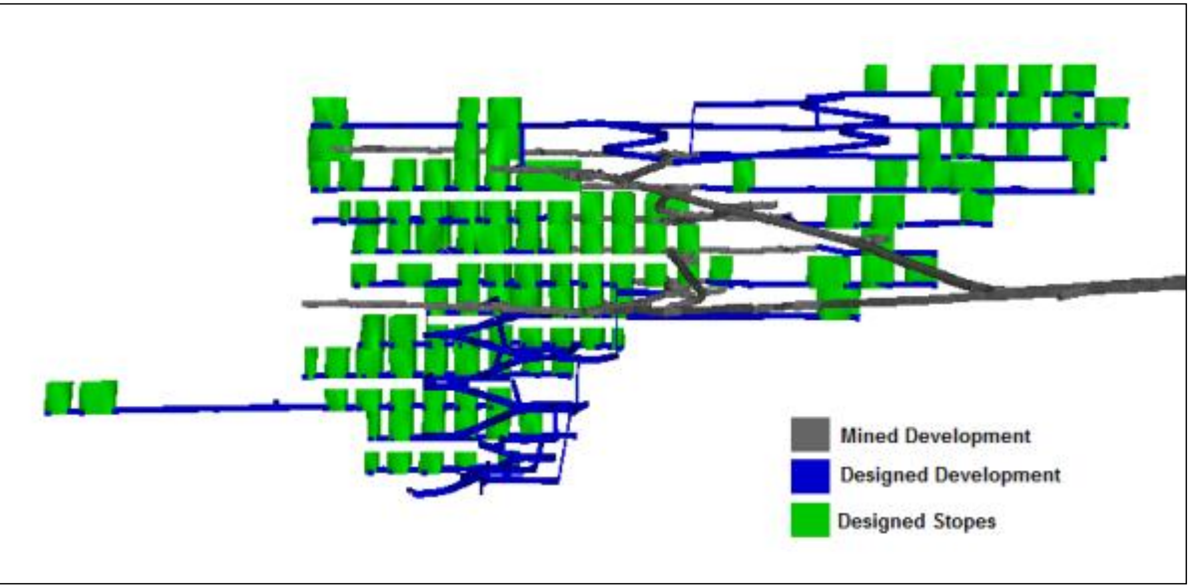


Figure 16-24: Kulthor Mine Design

The modifying factors for the Kulthor Mine are detailed in Table 16-27. Table 16-28 summarises the underground inventory after the modifying factors have been applied for the Kulthor.

Table 16-27: Kulthor Modifying Factors

Mining Method	Mining Recovery (%)	Mining Dilution		
		Tonnes (%)	Copper Grade (%)	Gold Grade (g/t)
Uphole Open Stope	95	10	0	0
Uphole Open stope with crown pillar	70	10	0	0

Table 16-28: Kulthor Mine Inventory

Level	Stope	Diluted Stope Inventory			
		Tonnes	Copper %	Gold g/t	eCu
900	S_00003001	17,911	1.41	1.93	0.88
900	S_00200001	28,242	1.27	1.66	0.65
900	S_00201001	31,771	1.32	1.75	0.71
900	S_00202001	22,412	1.19	1.59	0.67
900	S_00225001	5,632	1.15	1.48	0.55
870	S_00007001	15,775	1.96	2.58	1.03
870	S_00008001	22,435	1.57	2.20	1.05
870	S_00208001	15,535	1.67	2.15	0.81
870	S_00209001	17,056	1.76	2.19	0.72
870	S_00226001	9,832	1.37	1.76	0.65
840	S_00015001	6,487	1.18	1.52	0.57
840	S_00016001	5,570	1.03	1.32	0.49
840	S_00017001	23,899	1.31	1.73	0.68
840	S_00210001	7,995	1.28	1.62	0.57
810	S_00030001	11,771	1.17	1.49	0.53
810	S_00031001	12,798	1.40	2.06	1.11
810	S_00032001	12,696	1.23	1.64	0.68
810	S_00214001	8,344	1.20	1.55	0.59
780	S_00056001	17,332	1.38	1.91	0.89
780	S_00057001	13,167	1.71	2.35	1.06
780	S_00216001	18,625	1.21	1.64	0.73
750	S_00220001	20,136	1.81	2.68	1.46
720	S_00107001	11,705	1.43	2.04	1.01
720	S_00223001	19,643	1.32	1.89	0.95
720	S_00224001	16,575	1.64	2.34	1.16
690	S_00121001	19,144	1.18	1.59	0.70
870	S_00004001	18,870	1.28	1.98	1.18
870	S_00006001	11,928	1.17	1.54	0.63
870	S_00203001	35,010	1.15	1.51	0.59
840	S_00010001	18,954	1.53	2.41	1.46
840	S_00011001	23,824	1.31	1.76	0.76
840	S_00012001	45,149	1.22	1.63	0.69
840	S_00009001	4,848	1.22	1.68	0.77
810	S_00018001	8,911	1.93	2.89	1.61
810	S_00019001	9,162	1.85	2.84	1.64
810	S_00020001	38,558	1.32	1.79	0.78
810	S_00021001	34,996	1.17	1.62	0.74
810	S_00022001	32,909	1.18	1.63	0.76
810	S_00023001	51,111	1.14	1.64	0.82
810	S_00024001	14,484	0.98	1.31	0.55
810	S_00211001	18,164	1.28	1.69	0.70
780	S_00195001	7,872	1.21	1.53	0.54

Level	Stope	Diluted Stope Inventory			
		Tonnes	Copper %	Gold g/t	eCu
780	S_00034001	10,563	1.00	1.61	1.01
780	S_00035001	11,294	1.13	1.85	1.20
780	S_00036001	11,405	1.22	1.64	0.69
780	S_00038001	22,129	1.24	1.66	0.71
780	S_00039001	33,500	1.18	1.61	0.72
780	S_00040001	37,360	1.33	1.83	0.84
780	S_00041001	25,213	1.43	1.96	0.89
780	S_00042001	19,798	1.36	1.90	0.90
780	S_00043001	24,079	1.24	1.72	0.80
780	S_00044001	22,122	1.06	1.48	0.70
780	S_00045001	15,412	1.00	1.37	0.60
750	S_00058001	10,780	0.95	1.46	0.85
750	S_00059001	12,205	0.98	1.52	0.91
750	S_00060001	20,370	0.98	1.34	0.60
750	S_00061001	26,456	1.17	1.60	0.72
750	S_00062001	24,472	1.25	1.78	0.88
750	S_00063001	27,818	1.42	2.00	0.97
750	S_00064001	15,757	1.06	1.75	1.15
750	S_00065001	37,333	1.35	1.82	0.78
750	S_00066001	23,994	1.49	1.96	0.79
750	S_00067001	25,286	1.75	2.45	1.18
750	S_00068001	24,894	1.98	2.77	1.32
750	S_00069001	9,612	1.80	2.46	1.10
750	S_00070001	12,311	1.54	2.17	1.04
720	S_00082001	11,635	1.21	1.47	0.42
720	S_00083001	10,363	1.32	1.60	0.46
720	S_00085001	14,320	1.20	1.66	0.77
720	S_00086001	17,059	1.33	2.00	1.11
720	S_00087001	8,217	1.30	1.62	0.53
720	S_00088001	17,069	1.41	2.05	1.05
720	S_00089001	13,155	1.36	1.61	0.42
720	S_00090001	16,503	1.47	2.07	0.99
720	S_00091001	20,943	1.53	2.16	1.04
720	S_00092001	17,014	1.93	2.71	1.30
720	S_00093001	7,423	1.72	2.53	1.34
720	S_00094001	8,430	1.39	2.04	1.09
720	S_00095001	9,291	1.23	1.81	0.98
690	S_00108001	14,407	1.21	1.53	0.53
690	S_00109001	17,420	1.31	1.67	0.60
690	S_00110001	19,643	1.28	1.75	0.78
690	S_00111001	56,327	1.29	1.77	0.79
690	S_00112001	30,334	1.49	2.13	1.05
690	S_00113001	32,114	1.44	2.00	0.95

Level	Stope	Diluted Stope Inventory			
		Tonnes	Copper %	Gold g/t	eCu
690	S_00114001	13,600	1.45	2.07	1.04
660	S_00122001	27,076	1.49	2.05	0.92
660	S_00123001	28,282	2.29	3.17	1.47
660	S_00124001	16,192	2.11	2.90	1.32
660	S_00125001	19,184	1.75	2.35	0.99
660	S_00126001	13,322	1.16	1.49	0.55
660	S_00127001	14,566	1.47	1.85	0.63
660	S_00128001	15,930	1.78	2.35	0.95
660	S_00129001	25,554	1.56	2.05	0.82
660	S_00130001	9,217	1.28	1.75	0.78
630	S_00139001	5,237	1.36	1.80	0.72
630	S_00140001	16,365	1.33	1.75	0.69
630	S_00141001	21,166	2.01	2.74	1.21
630	S_00142001	20,417	2.76	3.76	1.68
630	S_00143001	13,646	2.65	3.63	1.62
630	S_00144001	16,887	1.97	2.58	1.02
630	S_00145001	9,609	1.75	2.22	0.78
630	S_00146001	8,634	1.48	1.89	0.69
630	S_00147001	16,855	1.26	1.57	0.52
600	S_00156001	13,528	0.98	1.94	1.60
600	S_00157001	20,115	0.98	1.94	1.60
600	S_00159001	10,921	1.66	2.20	0.90
600	S_00160001	19,057	1.78	2.39	1.00
600	S_00161001	21,057	1.89	2.59	1.18
600	S_00162001	17,374	2.16	3.00	1.40
600	S_00163001	19,191	2.10	2.85	1.25
600	S_00164001	17,824	1.99	2.65	1.10
570	S_00173001	6,151	1.70	2.57	1.44
570	S_00174001	13,868	1.87	2.87	1.68
570	S_00175001	12,434	2.31	3.46	1.92
570	S_00176001	9,235	2.35	3.39	1.74
570	S_00177001	10,799	2.05	2.80	1.26
570	S_00178001	6,234	1.52	2.17	1.08
540	S_00187001	18,721	1.68	2.52	1.40
540	S_00188001	18,306	2.59	3.78	1.97
540	S_00189001	4,475	3.10	4.33	2.06
540	S_00190001	6,701	1.64	2.38	1.23
Stope Total		**2,190,797**	**1.46**	**0.94**	**2.03**
Development Tonnes		384,261	1.49	0.95	2.06
Total		**2,575,058**	**1.47**	**0.94**	**2.03**

16.3.4 Material Handling

Ore Haulage

All underground production ore will be hauled to the 1000 mRL orepass, near the 1000 mRL Cribroom, via the 1006 mRL drive and will report to the 676 mRL drawpoint at the base of the orepass. The ore will then be rehandled by loader to the feeder pass above the underground crushing station. The ore will be crushed underground and hoisted to surface using the existing Osborne hoisting infrastructure.

Once the Osborne Mine has ceased production, it may be more cost effective to truck haul the ore directly to the surface to the surface crusher. A trade-off study will need to be undertaken to determine which ore haulage option is the most effective for the Kulthor Mine, once Osborne has ceased production.

Waste Haulage

All development waste from Kulthor will be hauled to designated tip heads in the 1000 and 800 Blocks in the Osborne Mine. As stope voids become available at Kulthor, the development waste is tipped preferentially in these stopes.

16.3.5 Ventilation

Ventilation Circuit

The proposed mine development and production can be readily ventilated with a conventional shaft exhaust system supplemented by auxiliary ventilation for underground development. The total required flow capacity will be 175 m^3/s during mine production.

The primary ventilation circuit consists of fresh air entering the Kulthor working via the access from Osborne and the K1 ventilation shaft. The mine will be exhaust via interconnect ventilation raises at each sub level to the Surface-830 mRL ventilation shaft.

The Kulthor ventilation network has minimal influence on the existing mine's ventilation system. Figure 16-25 shows the Kulthor ventilation system and emergency egresses.

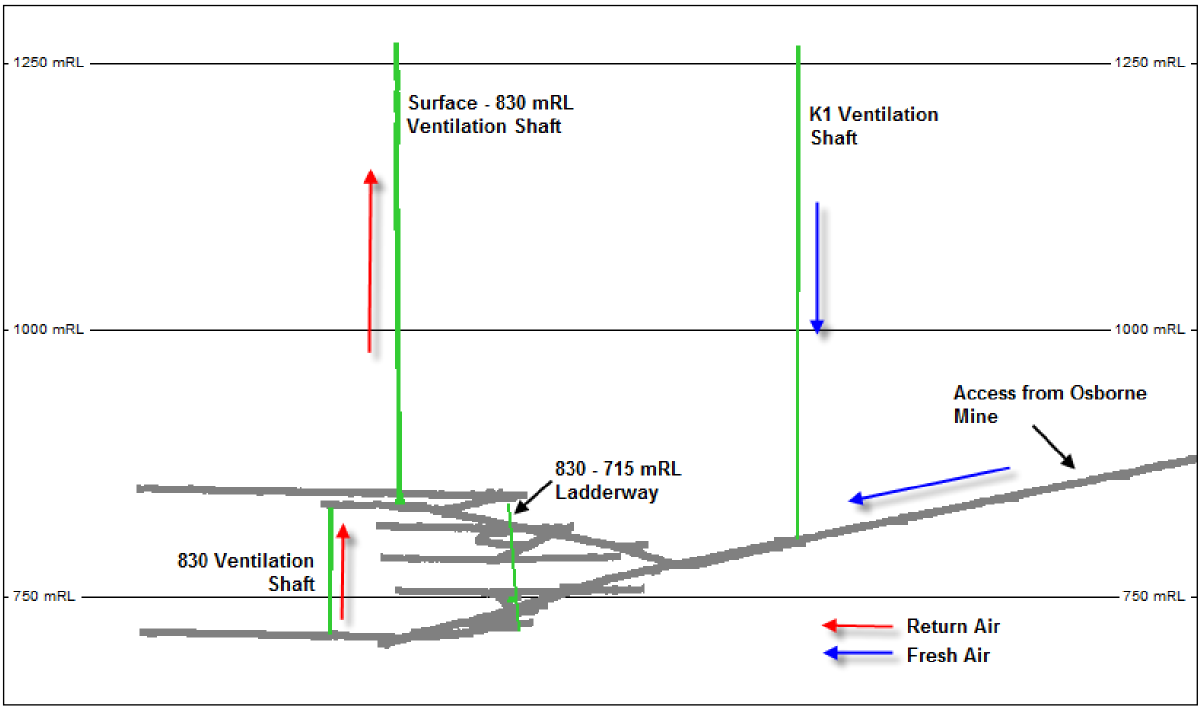


Figure 16-25: Kulthor Ventilation Circuit

Airflow Requirements

The minimum airflow required in Queensland is stated below as per the Mining and Quarrying Safety and Health Regulation 2001.

"A person who has an obligation under the Act to manage risk in relation to ventilation at a mine must ensure appropriate measures are taken to ensure the ventilating air in a place where a person may be present at the mine is of a sufficient volume, velocity and quality to achieve a healthy atmosphere."

Osborne has historically used 0.04 m^3/s/kW for all diesel equipment as a minimum standard.

In light of the hot conditions experienced in summer at Osborne, 0.07 m^3/s/kW has been used as a minimum for design purposes. Table 16-29 shows the scheduled machine usage and diesel kW unit calculation and the air flow required for the Kulthor mine.

Table 16-29: Estimated Kulthor Ventilation Requirements

Equipment	kW/unit	Units	Total kW	Total Airflow (m^3/s)
LHD Elphinstone 2900	350	2	700	49
Truck AD55	450	3	1350	95
Service Vehicles	125	2	150	18
Light Vehicles	75	2	150	10
Total Fleet			2450	172

16.3.6 Backfill

There are no plans to backfill the Kulthor stopes, but waste from development mining will be placed in voids when available.

16.3.7 Mine Services and Infrastructure

Electrical and Communications

Power is reticulated by 11 kV line from the Osborne Mine via the 1006 mRL. The primary fan is located on surface and surface reticulation via power poles and overhead 11 kV lines has been allowed for.

Communications underground at Kulthor is similar to the system at the Osborne Mine using a leaky feeder system. The system is integrated with Osborne Mine and includes four channels – general calling, general discussion, Pitram / Mine Control and a dedicated Emergency Response Channel.

Compressed Air

The underground operation at Kulthor is supplied with compressed air via 4-inch poly lines in the main declines and 2-inch lines on each sub-level. The lines are fed by Osborne Mine existing surface compressor, which is plumbed into the underground workings via the 1006 mRL drive. Upon completion of mining at the lower levels of the Osborne mine, the underground compressor currently located at 355 mRL can be relocated to Kulthor 830 mRL. This will add additional redundancy to compressed air supply thereby minimising breakdown delays.

Raw and Potable Water

Raw water is supplied from Osborne to Kulthor via 110 mm PN16 polypipes installed in the connection decline. No potable water service is available at Kulthor; the nearest potable water is located at the 1000 mRL cribroom adjacent to the Osborne main shaft.

Explosives

All storage, transport and handling of explosives have been assumed to be in accordance with the Australian Standards and the current Queensland Mine's Regulations and Act.

The designated contract supplier will batch all ANFO and Emulsion at the Osborne Mine surface plant. It is assumed that ANFO would be used for development mining and stope production, and that Emulsion would be used where wet conditions are encountered.

The current Osborne Mine magazine located off the main decline at 1108 mRL will be used to store explosives for the Kulthor project. If the distance from the Kulthor operations to the Osborne Mine magazine becomes too great, a new magazine location will need to be designed – however, no allowance for this has been made in the cost estimates. There is a lockup for the development charge car at Kulthor XC-2 (300 m along the connection decline) for parking the charge vehicle when not undertaking charging operations.

It is assumed that the existing Mains Electrical Firing system used at Osborne will be utilised for mining Kulthor. Independent firing regions will exist during development activities and the firing point for such systems will utilise lockable tag board firing arrangements as per the existing Osborne standard.

Emergency Egress and Refuge Chambers

The 830 mRL – 715 mRL ladderway has been mined and the ladderway has been installed. Future mining has provision for ladderways to be installed from 710 mRL to 540 mRL.

Coupled with the incline and decline, personnel will have a second means of egress from all locations within the mine.

The egress winder for Kulthor is a 37 kW unit supplied by Australian Winch and Haul. The unit incorporates a small 2-man cage that will normally be parked on surface. The egress winch is currently in stock at Osborne, and still packaged as new because it was purchased by Barrick prior to the sale of the site to Ivanhoe.

During initial development it is recommended a refuge chamber be advanced 300 m to 400 m from the development face. The final refuge locations will enable all personnel to be within 1,000 m of a refuge chamber or fresh air source.

16.3.8 Hydrogeology and Dewatering

Hydrogeology

The hydrogeological regime at the Kulthor site is considered to consist of three aquifer systems, namely the Mesozoic Cover, the Kulthor Shear Zone and the generally massive Proterozoic rock hosting the orebody. Whilst minor groundwater inflows may be derived from the Mesozoic and the Proterozoic host rock, the major risk to mining is related to the water held in storage within the Kulthor Shear Zone. Test pumping of a bore constructed in the Shear Zone has been undertaken at a constant rate of over 10 L/s by AGE, demonstrating the highly permeable nature of this shear. If mining or development intersects fractures that are hydraulically connected with the shear zone, sustained inflows of groundwater can be expected.

In order to reduce the risk of this occurrence, it pumping for dewatering purposes has been undertaken on an ongoing basis from the existing bore (KWB001). Monitoring of the progress of dewatering should be maintained. Supplementary dewatering bores may be required if monitoring indicates that effective dewatering cannot be achieved with a single bore within the time frame required. The Kulthor Feasibility Study (2007) made allowances for three dewatering bores at Kulthor with depths up to 250 m to be drilled and commissioned before mining approaches the Shear

Zone. Two of these bores have been developed, however only one is fitted with a dewatering pump. Monitoring of dewatering activity has shown reasonable response and drawdowns to date. Additional dewatering will be undertaken utilising horizontally drilled diamond drillholes from 830 Level, that are connected to the main dewatering pump stations underground. It is anticipated that this will assist in decreasing the risk associated with high water inflows during development.

Water quality from the Shear Zone is of neutral pH with a salinity of about 5600 mg/L. Currently, the pumped water from KWB001 is returned to the existing mill raw water pond, reducing the need for pumping from the water supply bores located in the Great Artesian Basin.

Additional groundwater issues related to depletion of the aquifer and subsequent impact on other groundwater users are considered low risk and can generally be managed through review of existing data and negotiations with Environmental Performance Evaluation (EPA) and Natural Resource Management (NRM).

Dewatering

Two parallel PE100 110 mm diameter poly pipes will run from Kulthor to 1000 mRL mono pumps at Osborne for dewatering purposes. Once at 1000 mRL, Kulthor mine water will be pumped to surface via the Osborne rising main. The main mono pumps used in Osborne and Kulthor will have a capacity to pump at in excess of 25 L/s. The anticipated inflow of water from mining activity is expected to be 10 to 20 L/s.

16.3.9 Mining Schedule

Scheduling Strategy

The scheduling strategy for the mine is:

- Stopes to commence production as soon as possible;
- Level development a priority over the decline development; and
- All level development, vent rises, sump and escapeways, completed before stoping commences on a level.

Because the ore is to be hauled to a ROM pad prior to being fed into the processing facility, there is a lag between ore being extracted from the pit and being processed. It is expected at an operational planning phase, greater detail will be provided to understand specific material movements.

Development Schedule

Figure 16-26 shows the development profile for the Kulthor Mine. The lateral development continues at a rate of 400 – 450 m per month for the first 12 months and decreases to 200 – 250 m per month for 30 months.

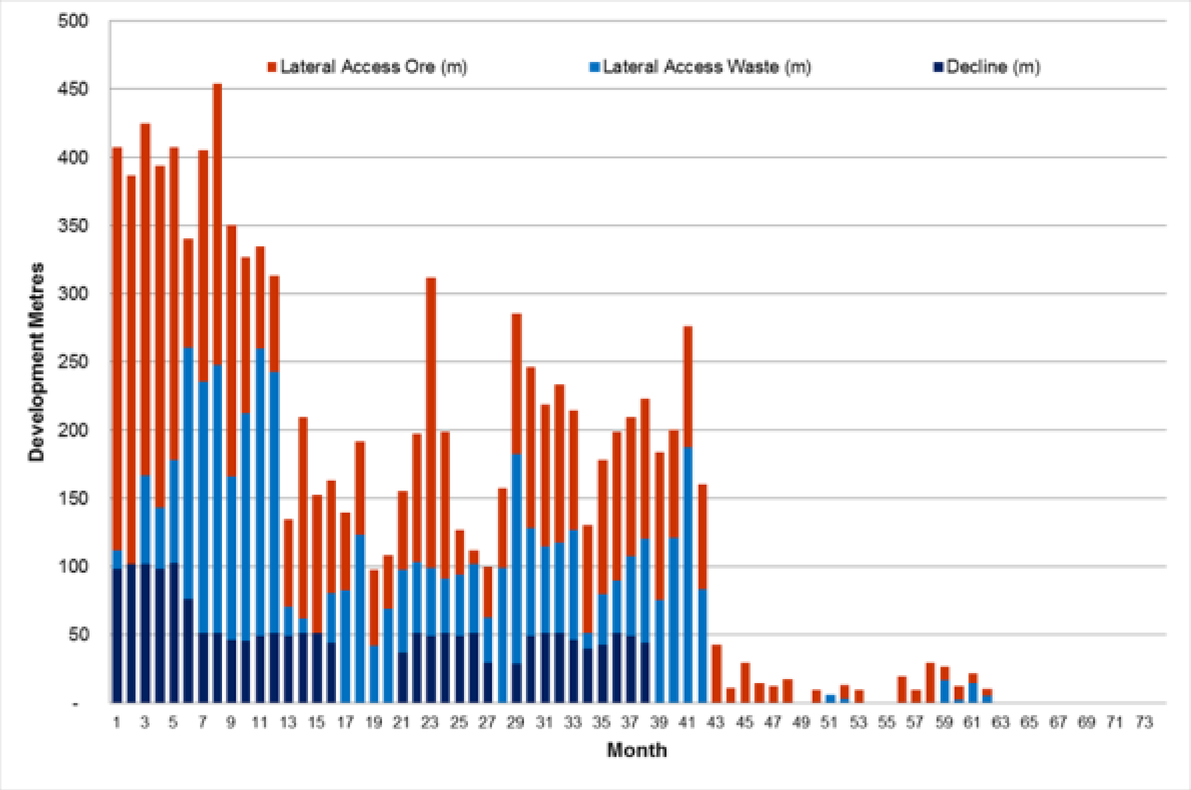


Figure 16-26: Kulthor Development Profile

Production Schedule

Figure 16-27 summarises the scheduled ore from the Kulthor Mine. The ramp-up is over 11 months and the steady rate production rate achieved is for 32 months, before decreasing to 25,000 t/mth for a further 18 months. The mine life is 5 years.

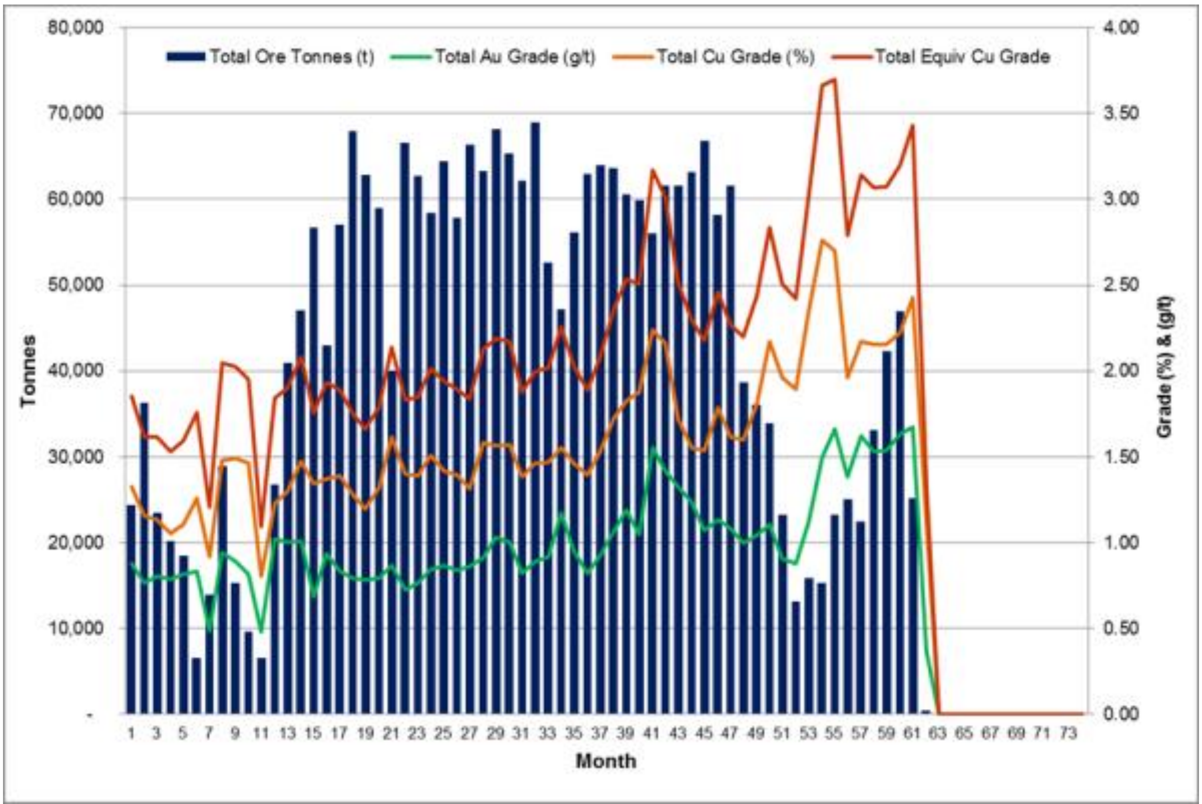


Figure 16-27: Kulthor Production Profile

Production Drilling Schedule

Figure 16-28 shows the production drilling profile for the Kulthor underground mine.

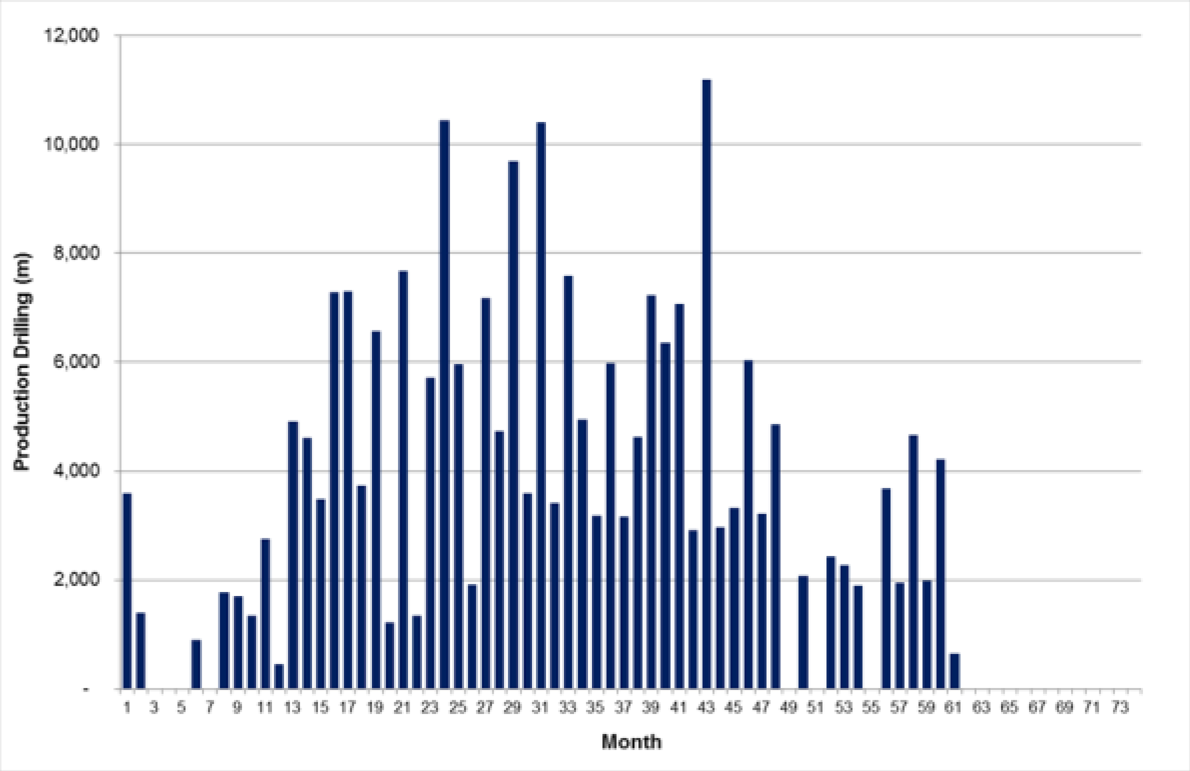


Figure 16-28: Kulthor Production Drilling Profile

Equipment Requirements

Table 16-30 summarises the proposed mining fleet for the Kulthor Mine. This fleet is indicative only and will be reviewed upon finalisation of scheduling requirements.

Table 16-30: Kulthor Mining Fleet

Machine	Number	Engine Power (kW)	Total kW
Loader	2	350	700
Truck	3	500	1500
Service Vehicles	1	100	100
Drill Rig	1	75	75
Light Vehicles	2	75	150

Personnel Requirements

The mining costs have been estimated using a continuous mining operation, 24 hours a day, 365 days per year. All employees will commute from either Townsville or Brisbane and utilise the Osborne village while on site.

Contract operators and maintenance personnel work on a 14 days on, then 14 days off, 12-hour shift alternating between dayshift and nightshift. Ivanhoe support staff will work 8 days on, then 6 days off, with a 12-hour dayshift only.

The labour costs are incorporated in the unit costs for the mining and processing activities. All on-costs for annual / sick leave and training have been estimated in the operating costs.

All equipment has been assigned with one operator per crew per machine.

It is assumed that contractor cross-training will occur for all operators, ensuring that each shift panel is adequately multi-skilled to relieve for sickness, annual leave and general absenteeism. Maintenance personnel have been estimated based on machine hours and site location.

Personnel numbers for each position and total work force levels are shown in Table 16-31 and Table 16-32.

Table 16-31: Underground Mining Staff Requirements

Staff	Number
Superintendent	2
Underground Supervisors	4
Underground Technical Services Personnel	8
Administration & Pitram Operators	4

Table 16-32: Underground Mining Contractor Personnel Requirements

Production Roles	Number
Jumbo Operators	12
Loader Operators	8
Truck Drivers	12
Longhole drillers	4
Blasting Personnel	4
Services Crew	12
Grader Operator	2

Maintenance	Number
Mechanical Planner	2
Electrical Planner	1
Leading Hand Fitter	4
Fitter	11
Leading Hand Electrician	1
Electrician	4

17 Recovery Methods

Copper in form of sulphide minerals, principally chalcopyrite and gold will be recovered by the conventional industry method of:

- Comminution by crushing and grinding;
- Flotation; and
- Dewatering of the copper-gold concentrate by thickening and filtration.

Flotation recovery of the gold may be supplemented by centrifugal gravity concentration to either make doré bullion or a gold-rich product to be added to the copper-gold concentrate.

There is the possibility that oxide copper oxide minerals in deposits such as Starra 276 could also be recovered by sulphidisation flotation but this has not been examined in this Technical Report.

In 2011, Xstrata Copper began producing magnetite concentrate from its Ernest Henry operation with the product being railed to Townsville for export. The Osborne tailing storage facility contains over 15 x 106 t of material at 35-50% magnetite (Coe and Evans, 2008), from which the recovery of a magnetite concentrate might be examined as part of the feasibility study.

Figure 17-1 pictorially describes the overall process flow.

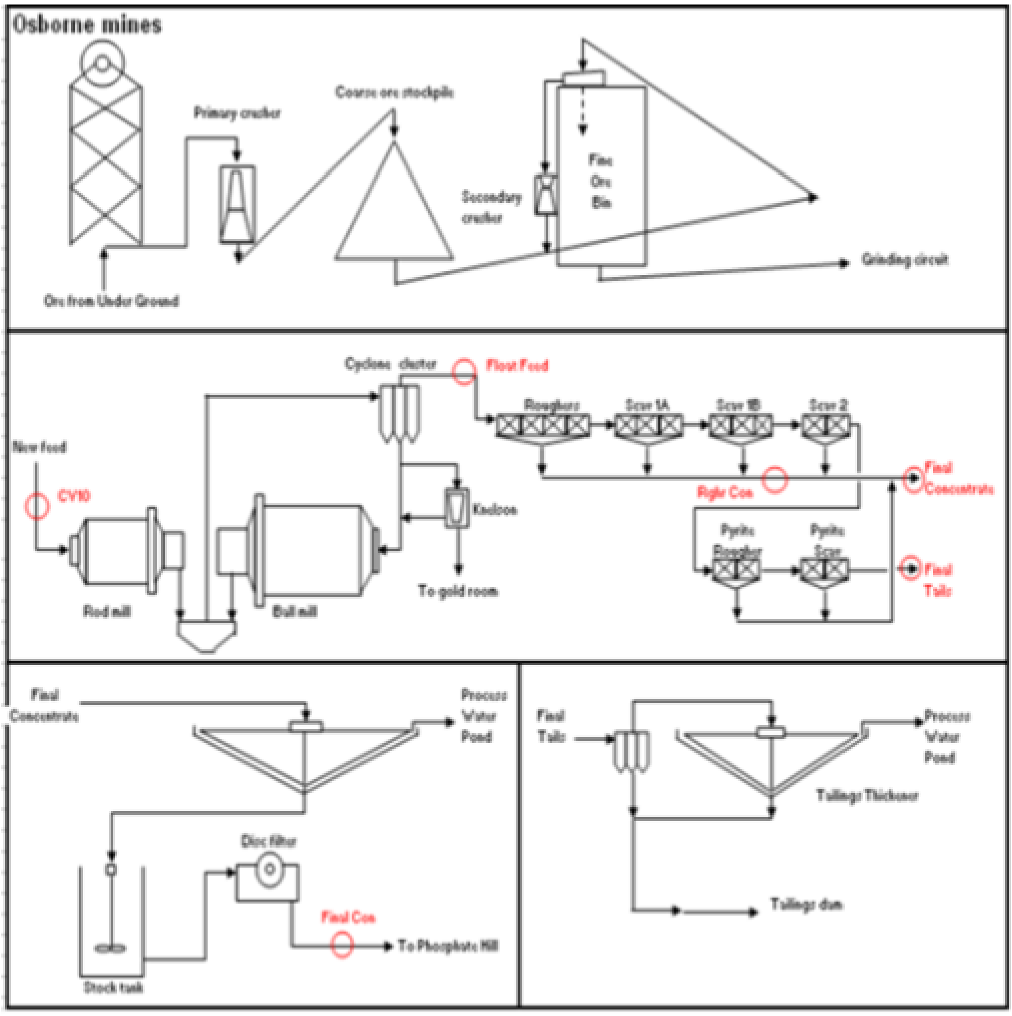


Figure 17-1: Osborne concentrator crushing section flowsheet

Material will be processed in the existing concentrator at the Osborne Mine. This is a conventional sulphide flotation concentrator plant commissioned in 1995 and operated continuously until 2010 initially treating ore from the Osborne open pit, then from the Osborne underground mine and finally material from the Osborne underground and the Trekelano deposit 95 km to the northwest. Design capacity of the plant was 119 tph at a flotation feed density of 35% weight/weight (w/w) solids but by 2008 actual throughput had reached ~265 tph at a flotation feed density of 50% w/w solids after a series of upgrades. The original flowsheet included a carbon-in-pulp gold recovery circuit, intended to recover around 50% of the gold as doré from a pyrite flotation concentrate. However, this system was abandoned after it was discovered that 60-70% of the contained gold reports to the copper concentrate, and a new gravity circuit, designed around a Knelson concentrator, was installed in its place.

The Osborne concentrator is briefly described (Dance, et al., 2009, Crosbie et al., 2009, Crimeen et al., 2009). The crushing section flow sheet is shown in Figure 17-2.

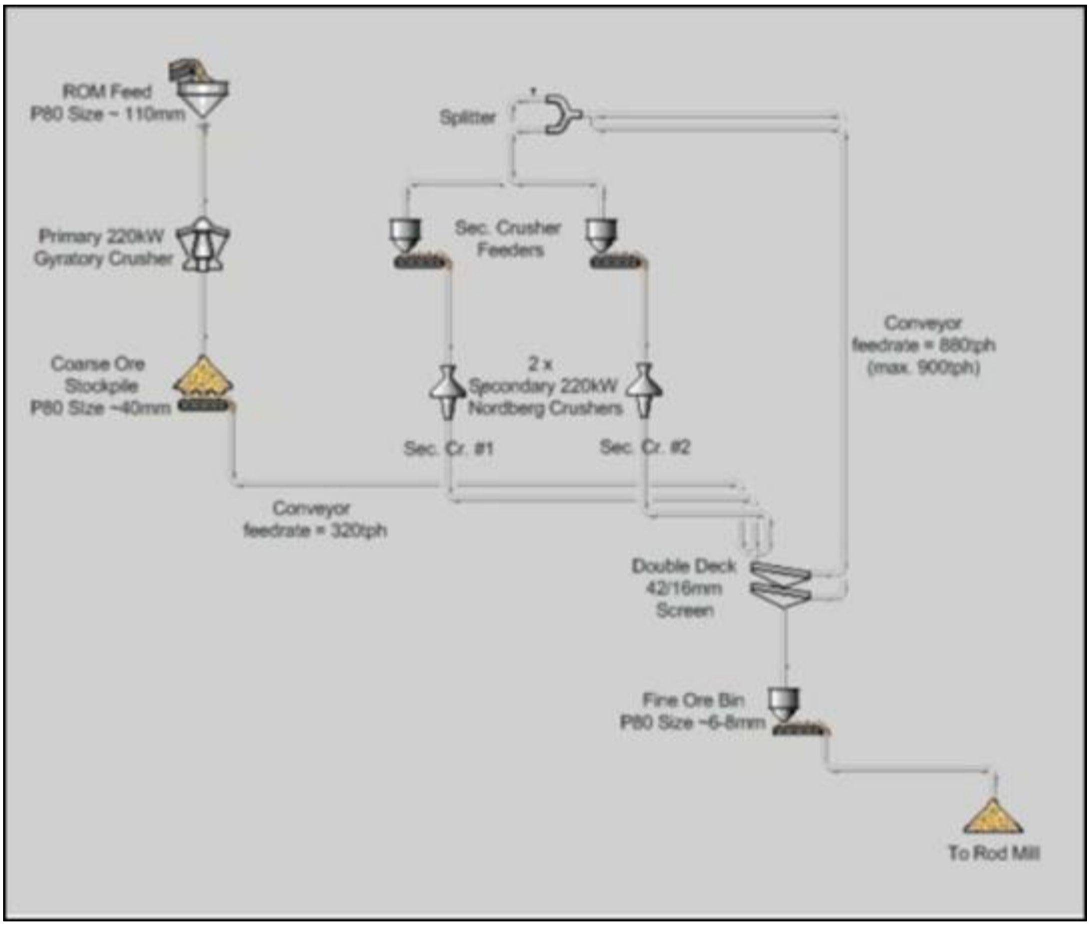


Figure 17-2: Osborne concentrator crushing section flowsheet

ROM ore was crushed in a primary gyratory unit with a 220 kW motor and conveyed to a coarse ore stockpile. Ore at nominally -40 mm was withdrawn from the stockpile via five vibrating feeders and sent to a double-deck vibrating screen with a 40 mm aperture cloth on the top deck and 16 mm aperture on the bottom deck. Screen oversize reported to two Nordberg HP300 cone crushers in closed-circuit with the double-deck screen while the undersize at an 80% size of 8 mm fell into the fine ore bin feeding the grinding circuit.

The Osborne grinding section had a 520 kW (2.9 m × 5.2 m) rod mill in open circuit followed by a 2.2 MW (4.3 m × 7.3 m) ball mill in closed-circuit with 5 x Cavex hydrocyclones making a flotation feed sizing of 80% passing 185 microns. The grinding section flowsheet is shown in Figure 17-3 and the grinding performance of throughput versus product sizing from 2004 to 2010 is in Figure 17-4. The Operating Work Index for Osborne + Trekelano ore was 11.4 kWh/t (Dance et al., 2009).

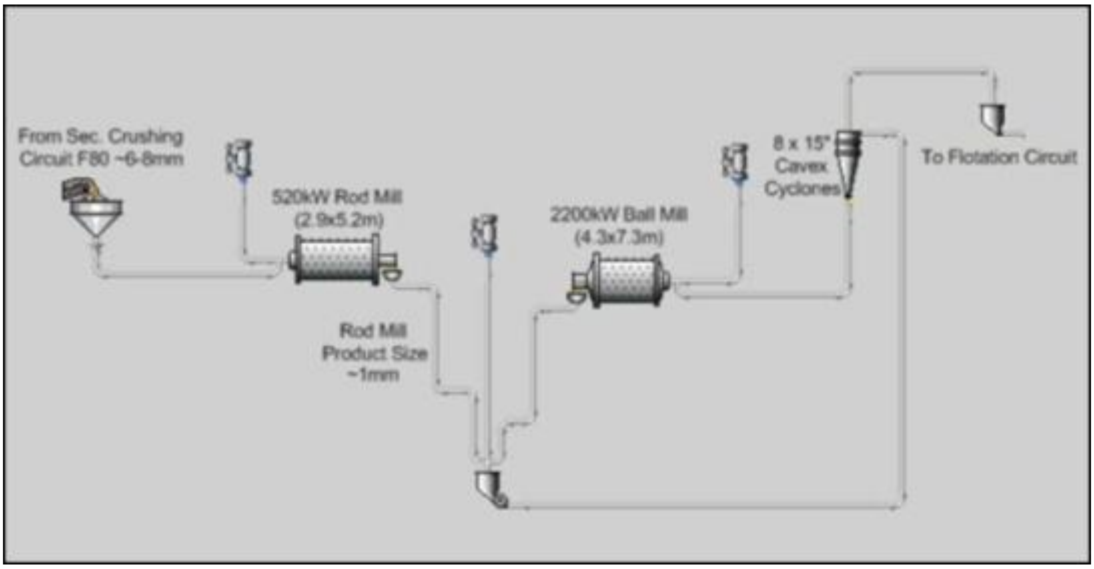


Figure 17-3: Osborne concentrator grinding section flowsheet

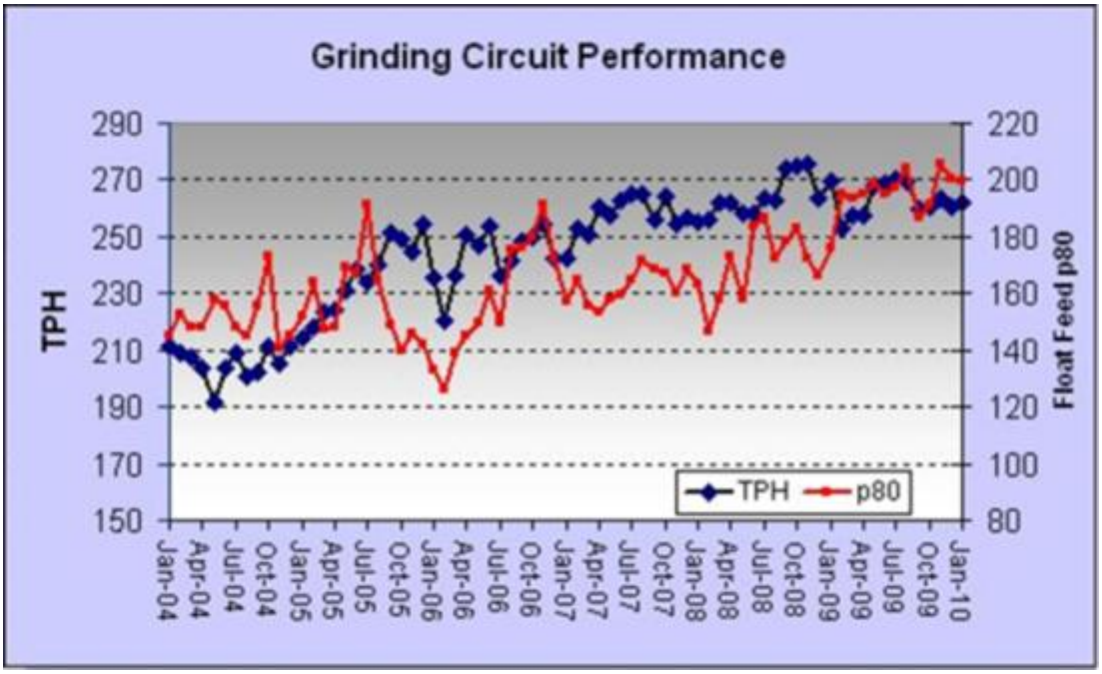


Figure 17-4: Osborne grinding section performance

A portion of the hydrocyclone underflow was sent to a 762 mm (30 inch) Knelson centrifugal concentrator for gravity recovery of gold, the Knelson concentrate was tabled and smelted to produce gold bullion.

The flotation circuit has a number of flotation banks operating in series, when treating Osborne and Trekelano ore the concentrate of each was combined into a final concentrate as shown in Figure 17-5.

All the flotation cells are rectangular and have volumes of either 8 m^3 (the rougher, scavenger 1A and scavenger 1B cells) or 16 m^3 (the scavenger 2, pyrite rougher and pyrite scavenger cells). The rougher cells have four peripheral launders and all other cells have single peripheral launders.

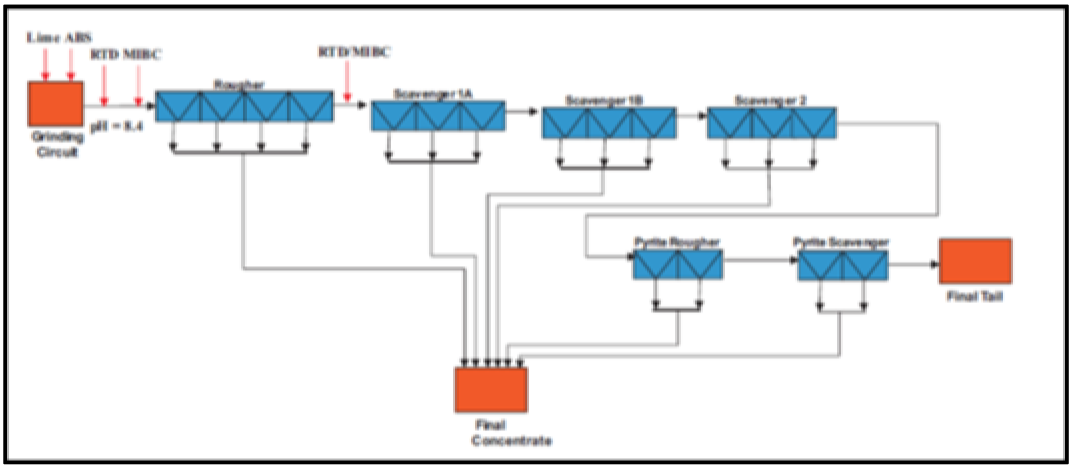


Figure 17-5: Osborne flotation circuit

This is a most unusual flowsheet as there is no regrinding of a rougher concentrator or cleaner flotation as is normal in conventional copper sulphide flotation practice i.e. a single stage of flotation sufficed to make a saleable copper-gold concentrate at ~22-23% copper and ~7 g/t gold at +95% copper recovery and 85% gold recovery thus demonstrating the high amenability of the ores treated to beneficiation. However, it should be noted that for the final years of the Osborne plant operation was at a time of high copper prices and relatively low treatment charge + refining charge with no disincentive for making a low grade copper concentrate.

A cleaner circuit was included in the original Osborne flowsheet and the cells were installed and operated in the early years of operation until it was discovered that a saleable concentrate could be made without them. This circuit should be reactivated to give greater assurance that the predicted flotation performance can be achieved with all ore types.

Copper-gold concentrate was thickened to 65% w/w solids and further dewatered in a ceramic disc filter to 9% w/w moisture and stored on a concrete pad on site prior to trucking to the rail siding at Phosphate Hill for movement to the port at Townsville.

18 Project Infrastructure

18.1 Introduction

SRK reviewed the project infrastructure in the compilation of the Technical Report.

As part of the review, the following considerations have been made:

- Most of the infrastructure is already in place, and has been successfully operated previously for a number of years and has been recommissioned having been on care and maintenance; and
- A simple spread sheet of costs is to be nominally sourced from Ivanhoe.

The key documents sited during the review are listed in Table 18-1.

Table 18-1: Key Infrastructure documents

Storage Location	File	Date
Base Directory	AECOM Ore Handling.pdf	13/05/2011
\2011_05_01\Basis for Geology History	KULTHOR_Feasibility.pdf	27/04/2011
\2011_05_01\Cu-Au Study\Library\Presentations	110212_FINAL_Osborne copper-gold evaluation TJF.pptx	15/04/2011
\2011_05_01\Cu-Au Study\Library\Production Cases\Option 3 - Kul Osb UG & St276 & Osb Pit	DRAFT_Copper Gold Study financial model_110212.xlsm	15/04/2011
\2011_05_01\Cu-Au Study\Library\Production Cases\Option 3 - Kul Osb UG & St276 & Osb Pit	Ore Treatment and Stockpiles option3.xlsx	15/04/2011
\2011_05_01\Received on Site Visit May	ILA Maint restart budget.pdf	24/05/2011
\2011_06_13	11402-00-M0703_Rev0 Osborne Mechanical equipment review report[1].pdf	16/06/2011
\2011_06_13\Infrastructure	os3256 Osborne Fine Ore Bin Inspection & Analysis.doc	16/06/2011
\2011_06_17	Kulthor & Osborne Underground Project Information.pptx	17/06/2011
\2011_06_17	Kulthor Osborne Underground Development Schedule.pptx	17/06/2011
\2011_06_17	AFE Kulthor Osborne Underground.docx	17/06/2011
\2011_06_17	Kulthor & Osborne Underground Board Paper.docx	17/06/2011
\2011_06_21 (Ivanhoe)(TFisher)	Osborne Cost History.xlsx	17/06/2011
\2011_06_21 (Ivanhoe)(TFisher)	Kulthor & Osborne Underground Project Information.pptx	17/06/2011
\2011_06_21 (Ivanhoe)(TFisher)	Kulthor Osborne Underground Development Schedule.pptx	17/06/2011
\2011_06_21 (Ivanhoe)(TFisher)	Kulthor & Osborne Underground Board Paper.docx	17/06/2011
\2011_06_21 (Ivanhoe)(TFisher)	Aerodrome and Facility Information.doc	21/06/2011
\2011_07_08(Haul Road)(Ivanhoe FTP)	IVA1060-G-001_Gen Arrangement_Rev0 Layout1 (1).pdf	18/07/2011
\2011_07_08(Haul Road)(Ivanhoe FTP)	5.0 Scope of Work.docx	18/07/2011
\2011_07_08(Haul Road)(Ivanhoe FTP)\Osborne Power	Proposal.pdf	19/07/2011
\2011_07_20(Ivanhoe)(FTP download)	20110216 PoO Amendment Final.pdf	20/07/2011

18.1.1 Methodology

The review methodology was:

- Two site visits by consultant; as follows:
  - 15-16 May 2011 (Hugh Thompson);
  - 22-23 June 2011 (Frank Soa);
- Three short meetings in Townsville, to clarify site visit items; and
- Various follow-up telephone conversations, to clarify site visit items.

Key Ivanhoe personnel who provided data and clarification during this review have been:

- Tim Fisher – Manager Mining Studies;
- Nicholas Wright – Superintendent Electrical Engineering; and
- Scott Powell – Superintendent Maintenance.

18.2 Osborne Site

The Osborne site is the location for the following:

- Osborne Underground mine;
- Kulthor Underground mine;
- Osborne Open Pit mine;
- Osborne Processing facility;
- Osborne accommodation; and
- Ivanhoe regional management, business infrastructure and technical services.

18.2.1 Osborne Communication Infrastructure

The Osborne site has an optic fibre network and a microwave Internet link which should not need extension or modification. Additionally, the current telephone and radio nets will not need to be altered.

18.2.2 Air Strip – Osborne

At the Osborne site, there is a Civil Aviation Safety Authority (CASA) rated 3C airstrip with a current licence. The landing strip is asphalt sealed, and approximately 2 km long x 23 m wide. The airstrip is “all weather”; however, its operation is restricted to daylight hours only. The recommendations noted on a recent (October 2010) facility inspection and relicensing amounted to typical housekeeping items. The airstrip is used by commercial aviation companies contracted to Ivanhoe, operating SAAB 340s (Twin Turbo prop plane, seating 34 passengers).

In previous operations, this facility was sufficient for the required traffic, which was up to 10 flights per week.

Therefore, it is anticipated that only maintenance (as part of normal operations) will be required to support the five-year business plan. It is recommended that the suitability of this airstrip and associated infrastructure is reviewed once proposed manning levels and rosters are known.

18.2.3 First Aid / Emergency response

Mine rescue equipment, supplies, training materials and control infrastructure has been viewed at the following locations:

- Osborne and Kulthor underground;
- Osborne processing; and
- Osborne accommodation.

These have generally been found to be adequate; however additional resources will be required to restore this equipment to fully operational status and re-certification.

18.2.4 Osborne Materials Handling – Underground

The Kulthor ROM ore is to be tipped into the 1000 mRL ore pass at Osborne by mine trucks, via the underground connection drive. The Osborne underground ore would be trucked up the decline from levels between 60 mRL and 110 mRL and then directly tipped into the 676 mRL grizzly. It is recommended that this grizzly will need the previous rock breaker to be re-installed (or equivalent).

It is anticipated the haulage system will follow the material flow as shown in Figure 18-1 sourced from the AECOM materials handling audit report. That is, ore will then be hoisted up the shaft, from the 578 mRL level to surface.

Generally, this infrastructure appears to be in reasonable condition. However, most of the significant components are 16+ years of age. Therefore it would be prudent to allow for increased operating costs in the future.

In the proposed business case, the peak period for hoisting will be in years 2 to 4, when approximately 780,000 t will require hoisting. This is equivalent of 2,300 tonnes per day (tpd) and within the designed system capacity.

The peak hoisting year was 2005 when approximately 1.8 Mt was hoisted, and short term (i.e. 24-hour) totals exceeding 5,500 tpd were regularly hoisted. This peak loading revealed a practical limitation and design constraints in the winder system design and control operations. This level of production (1.8 Mtpa) has not been scheduled since 2005 due to limitations of available underground feed.

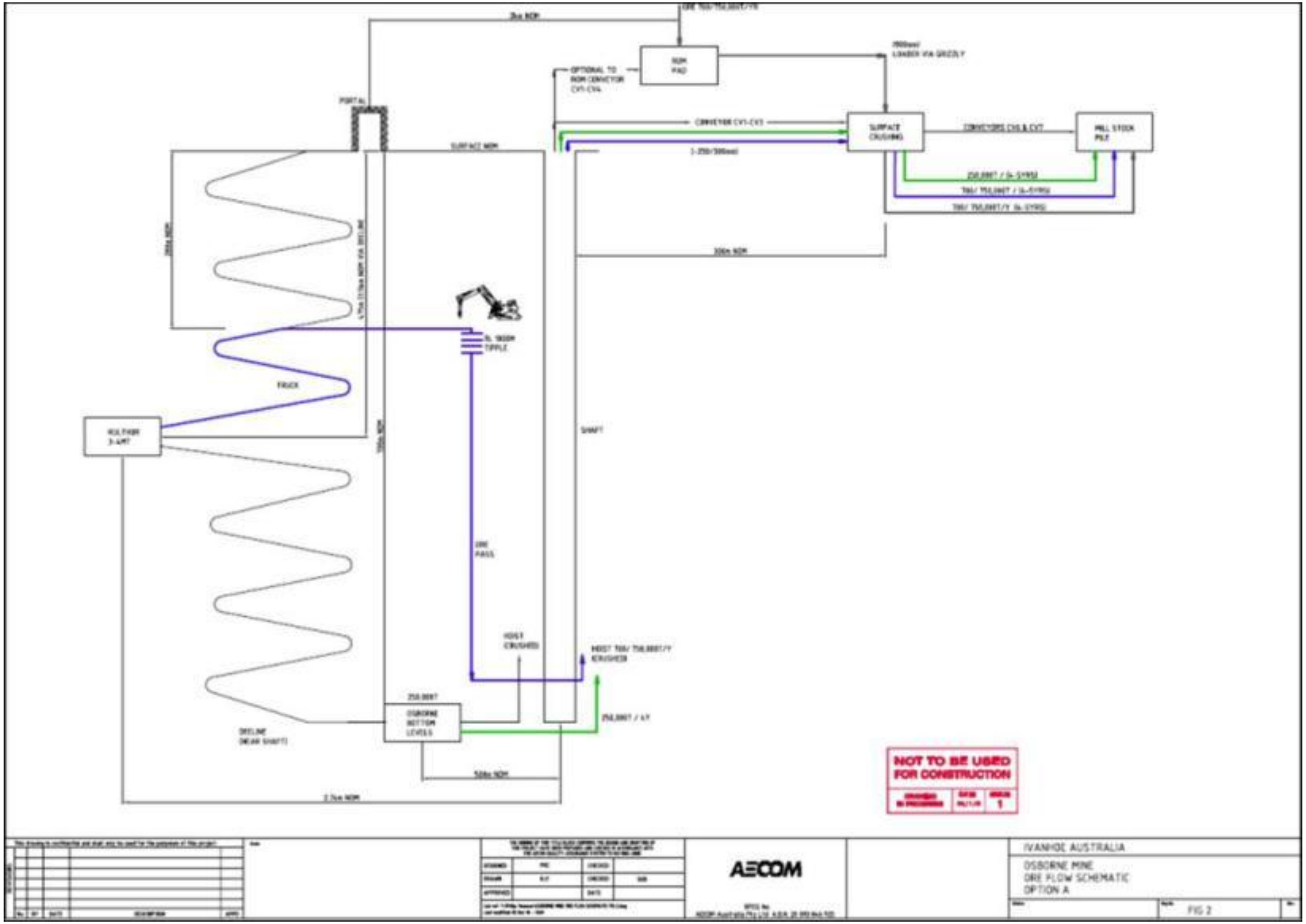


Figure 18-1: Schematic of material flow

18.2.5 Osborne Materials Handling – Surface

General Observations

The general works planned on the surface components of the materials handling system appear to be known and progressing. Ivanhoe's current plan is shown in Table 18-2. It is worth noting that there is a significant inventory of fabricated spares, liners, tanks etc. already on site. Therefore, some of the repairs and maintenance items discussed here already have the materials on hand, and the costs involved are essentially labour and direct fabrication consumables.

A summary of these discussions with Ivanhoe, and observations recorded include:

- Discussion of the gyratory crusher needs further analysis in time, particularly as it is nominated as the weakest link of this system. The main issue relates to the difference in feed that will be reporting to this crusher. According to the AECOM report, effectively the gyratory crusher is not well suited to a dual purpose role of a secondary crusher (i.e. when receiving previously crushed underground ore) and as a primary crusher (when receiving other ROM ores as direct feed; from Osborne Open Pit, and Starra 276) – when it would be operating at a minimum setting. It is recommended at this stage of study to account for this reduced operating efficiency by assuming greater downtime / higher unit cost, as the percentage of direct feed material changes over time;
- The issue of the rotation of the mantle of the surface cone crusher through 90° is highlighted in the AECOM report. However, given that it will still receive ore flows from both front-on, and side on; it is likely beneficial to have the "inefficient" axis lined up with the flow of underground dirt that is entering the surface crushing chamber from the side. This is due to the underground material being more likely to be more finely crushed than the ROM feed coming in from the front (sourced from Starra and Osborne Open Pit), and hence is likely to suffer less in terms of inefficiencies, and producing higher maintenance. Further to this, the schedule requires hoisting less than 50% of the previous best single year, and the underground ore is 43% of the total feed;
- The works associated with removal of tramp material needs to be given a high priority to ensure that the gyratory crusher can operate effectively. Standard weekly periodic maintenance plans should be developed to ensure this item receives appropriate attention during operation;
- The retaining walls on the ROM pad crusher station have had a history of movement and failure. Substantial remedial works have been completed on the eastern side. Visual indications reveal similar works may be required on the western side;
- AECOM highlight this as a significant risk, and in need of remedial works. There appears to be no firm plans from Ivanhoe to mitigate this risk, however verbally it was acknowledged as requiring work;
- The ROM pad surface requires civil earth works to recondition it to an operational condition. This is relatively minor; however it may need specialised equipment and materials, hence needs planning. At the Osborne site the ROM pad is sized for roadtrains – where wider turning circles and single level dumping is required; and
- If further distant ore tonnages are economically viable, then an option may exist to install a fixed material handling solution on the Osborne ROM more suited to efficient material handling.

Table 18-2: Ivanhoe fixed plant refurbishment plan

IVANHOE AUSTRALIA

OSBORNE MINE - PROCESSING REFURBISHMENT

Item #	Area	Action Required	Status
1	Grinding	Refurbish primary cyclone cluster underflow launder	Needs to be remove and sent off site for lining. $50,000
2		Replace primary cyclone feed hopper	On site and relined. Remove old hopper and install new hopper, replace all floor mesh
3		Rod and Ball mill discharge launders badly corroded	Requires fabrication of Trommel splash guards and flooring.
4		Replace hand rails and flooring	$10,000
5		Refurbish Ball mill feed box	Minor fabrication required
6		Ball kibble area	Concrete pad to increase the size of the apron.
7		Sump pump # 2.	To be replaced with associated supports fabricated.
8		Ball mill pinion seal leaking	Hoffman's to do in situ - $50,000
9		Ball mill rod charger	Redesign / rebuild of existing charger - $20,000
10		Rod mill trunnion	Replace liner - $10,000
12	Crushing	Fine Ore bin	Cost options from Col Hooper - rebuild / reline or patch. $200,000
13		CV 7	Replace swivel bearing - on site.
14		CV 6, 7 & 11	Belts to replace
		Reline Secondary Crusher	Inspect all components during inspections
		Replace Main frame liners	inspect and repair main frame liners
15		Primary Crusher	Retaining wall - Anaylise survey history for movement, Cost options [Col Hooper] $200,000
16		Underground	Fabrication repairs to flask and loading station. Reline crusher station feeder. $100,000. Liners on site

17	Water Systems & Sewerage	Process water pumps	Refurbish stands and foot valves
18		Water pipes	Replace corroded pipe work where required.
19		Knelson	Change over to raw water feed [currently on potable]
20		RO Plant - site	Purchase new plant and relocate away from mills
		Chillier plant	Relocate dedicated Chillier RO plant to chiller plant area.
21		STP - Village	Finish commissioning previously purchased plant
22		STP - Site	Relocate unused plant to site to replace existing site plant
23		Potable water - Village	Purchase new potable water tank. $40,000
		Site Fire Pumps	inspect and rebuild fire pumps
		Underground	Purchase rotable spare for Mono's $80,000
24		Underground	Replace and line 1000L Mono tanks x 2. $50,000
25	Floatation	ABS	Refurbish & reline ABS tank - $50,000
26		Sandfill & Scav 2 area	Refurbish pump hoppers [sandblast & reline]
27		Final Cons hopper	Refurbish
		Blower	Replace floatation blower. $80,000
28		Reagents	Refurbish all pipe work on reagent feed and return lines
29		Compressor	Replace burnt out #1 compressor with unused comp from Paste plant. $10,000
30	Filtering	CV 11 discharge pocket	Refurbish discharge pocket on filter - $20,000
31		Cons cyclone	Remove redundant cons cyclone cluster
		Cons thickener	Replace cons thickener. Thickener on site. Installation $100,000
32		Tails area	Cost options from Col Hooper to replace steel structure at tails area bund. $200,000

33	Control Room	Roof	Replace corroded and badly leaking roof
34		Control room	Replace door & floor covering. Redesign control desk.
36	ROM	ROM pad.	Reshape to allow for better drainage. Set up perimeter bunding for traffic control
	Pumps	Rebuild All Pumps In Floatation	Sandblast And Paint Frames
	Winder	Supervisory link.	Purchase 2nd rotable spare. $100,000
38		Shaft	Replace corroded dewatering line. Refurbish shaft. $150,000
39	All plant areas	Stairs, handrails and grid mesh flooring	Replace as required - $50,000
		Compressor	Install The Paste Plant Compressor Into Plant position #1

Conveyors

Table 18-3 describes the surface conveyors from the mine hoist (winder) to the ROM pad and / or process plant.

Table 18-3: Ore handling conveyors

CV1-CV2	Surface conveyors from winder to elevated ROM / Crusher area. Both have undergone various maintenance refurbishments including bucket and chute re-lining and belt replacement. These should not require further upgrades.
CV3	Surface conveyor or bi-directional, allowing ore to be stockpiled or fed into primary grizzly and / or primary crusher. Bucket and chute refurbishments completed, minor belt works near completion.
CV4-CV5	Surface conveyor, at the time of inspection, works ongoing to replace belt and magnetic separator. These should not require further upgrades.
CV-7	Surface conveyor to the coarse ore stockpile, at the time of inspection, completed refurbishments to chute with ongoing roller sections being worked on. These should not require further upgrades.
CV-8	Below ground conveyor from the coarse ore stockpile to CV9. This is now operational and surface feeder chutes have been re-lined.
CV-9	Surface conveyor feeding ore into secondary crusher, various upgrades completed including the automatic chute cleaner section are planned.
CV-10	This covered conveyor has structural weathering issues with the corrugated covers at the surface entry. The required maintenance to belt and rollers is ongoing.

Ore crusher

The primary Gyratory Crusher appears to be in working order with regular maintenance planned prior to start up including:

- Complete re-lining of the rock box;
- Checks and change out to the existing mandible; and
- Checks to the incoming dumping chute.

Major ongoing mechanical works on the process plant are to be completed prior to the start-up of the mill these include the following:

- Replace the concentrator thickening tanks and rakes;
- Fine ore bin refurbishment;
- Replace the tails thickening rakes and tanks;
- Works on charger to roller mills;
- Upgrade to 1210 Warman pump and rationalisation of the feed hoses;
- Cleaning of slag from scavenging cells including upgrades to agitators and various elbow joins need replacing. It appears the use of ceramic liners, particularly on high wear elbows, has been quite advantageous;
- Regular maintenance and cleaning required for the Knelson separator;
- Disc filter rebuild and associated pipework were being refurbished; and
- Steel walk ways around roller mill, and other areas are to be replaced.

According to Ivanhoe, all of the above tasks have been planned as part of the refurbishment works, however there is no overall works plan that can be reviewed, other than the task list shown in Table 18-2.

Currently, efforts are being made to reinstall the previous maintenance planning systems. It is assumed that when that system is functional, a detailed maintenance plan will be available.

The main risk identified to the fixed plant refurbishment plan would appear to be the availability of skilled tradesman to complete the works.

Assay Laboratory

This facility was noted on site but not inspected. It is assumed this facility will require only minor modification to return the laboratory to its previous operational condition. The main function of the onsite laboratory is for quick and efficient feedback relating to processor operations and mining grade control. Ivanhoe does not intend to use the onsite laboratory for assay work required for Resource reporting. Should this intent change then the facility would require certification to support an NI 43-101 estimate.

Concentrate Handling

There is sufficient space for concentrate load out and handling at the back of the process plant. Consideration is being given to re-routing the haul truck service road via which roadtrains travel to Phosphate Hill. The current schedule contains some 90,000 tonnes per annum of concentrate to move, which should be easily accomplished.

Additional detail on concentrate handling is provided in Section 19.

18.2.6 Power Distribution

Osborne Supply

Osborne has six captive power stations, capable of supplying approximately 22 MW.

These units include:

- Five 3.96 MW Diesel / Gas Wartsila units;
- One 2 MW full gas unit; and
- Two 250 kilo volt ampere (kVA) emergency power sets.

The five original ~3.96 MW Wartsila stations have been partially upgraded from original straight diesel to a diesel / gas (nominally 30:70) mix. A refurbishment and automation upgrade is required before these units can be fully brought back online. An allowance for this work has been included in the cost estimate. The units are currently capable of supplying the care and maintenance power requirement, due to the low power load. The significant load will come when the plant needs to be brought fully on-line, which is planned for late 2012. It is recommended to complete the gas conversion on these Wartsila stations at the time of this refurbishment (i.e. enable 100% gas fired power).

The single full gas fired power station (capable of supplying an estimated 2 MW per annum) is also on site. This unit was installed during the partial gas conversions on the Wartsila units in 2008.

There are two 250 kVA power generators (gen sets) available for emergency power.

A current / forecast site power balance needs to be completed by Ivanhoe as part of future studies. The diesel to gas conversion has apparently down rated the peak power available from the generator sets, so current capacity need to be re-established. However, it can be assumed at the moment that power requirements can be met with the existing capacity.

Power draw statistics for the month of November 2007 indicate a steady state load of some 8 MW, or a daily draw of 190 kWh/day.

It is assumed that the current contract negotiations with gas suppliers will be successfully concluded well in advance of requirement.

Osborne Power Reticulation

The current high voltage (HV) reticulation into and around the underground operations at Osborne has been deemed adequate. It should be noted however, that the proposed production is coming from significantly deeper locations; hence cable runs inside the mine will become more costly. These costs are reflected within the mine operating costs.

The current underground has HV power delivered through the main shaft with a take-off at 633 Level and through the portal and down the decline. The decline feed currently supplies the development works at Kulthor. A surface 11 kVA line is being installed on surface out to the Kulthor second egress/vent shaft. This is required for powering the surface ventilation fan, and the secondary egress winch.

When the final configuration of the open pit mining is known, it is recommended an assessment be undertaken to determine if any works are required to protect the underground electrical feed via the portal, from deleterious open pit blasting practices. The current portal HV feed services Kulthor work areas via underground. Kulthor will soon also be serviced from the cable down the vent shaft. The lower levels at Osborne (i.e. the crushing / hoisting system and remnant lower level mining) are powered via the main shaft feed.

18.2.7 Osborne Diesel Storage

The diesel storage farm at Osborne Mine main facility has a capacity of 2.4 ML. These holding volumes for diesel fuel vary; with 1.0 ML typically being stored during the wet season (November to May). Annual average usage at Osborne has been as much as 21.0 ML prior to conversion of the main power station to gas. It is anticipated the Kulthor underground operation will also utilise the Osborne surface fuel storage.

Other heavy hydrocarbons are stored at the Osborne facility, including oil, grease, and processing reagents. It is assumed that all future (and current where appropriate) storage for flammable and combustible materials will be handled in accordance with current Australian standards (AS1940:2004, AS 3780:1994).

18.2.8 Osborne Tails Storage and Disposal

As indicated in Section 20, a new tails storage facility is to be constructed.

18.2.9 Osborne Waste Water Disposal

Refurbishment to the existing aeration type sewer treatment plant (STP) is being undertaken. Table 18-4 details the observations related to the STPs.

Table 18-4: Sewer treatment plant

Unit Location	Description	Observations
Osborne Accommodation	≤400 EP Aeration type Sewer Treatment Plant	Approx 15 years old, in fair condition
Osborne Accommodation	≤600 EP Aeration type Sewer Treatment Plant	New, not operational
Osborne Process	≤400 EP Aeration type Sewer Treatment Plant	Approx 15 years old, in poor condition

The effluent from the STP is pumped to an evaporation pond on the Osborne processing site. It was noted from observations and discussions with David Hebert (Heal Group), that the existing process plant STP are in poor condition. It is planned to replace this unit with the Osborne Village 400 EP STP.

18.2.10 Trade Waste Disposal

The disposal of all used heavy oils and grease is dispatched from the store department and collected by a licensed contractor based in Mt Isa. The system of disposal of these trade waste items is generally instigated by Ivanhoe supply manager as operations require.

Other large generators of waste are the vehicle workshop and mine operations. Regulated wastes on site include packaging, drums of lubricant, used tires and batteries (i.e. recyclables and regulated waste would be taken from site for off-site disposal in accordance with the current Osborne Mine waste management practices).

18.2.11 Osborne Water Supply

Process water is supplied via a 25 km pipeline from a borefield south of the Osborne processing facility. The pipeline and borefield is covered by maintained mining leases (ML). Whilst not inspected, this borefield is understood to be similar to that of the Mt Dore borefield which was inspected. It is assumed that this system can provide adequate process water quantity, and of sufficient quality for the business case.

As part of future studies, it is recommended a site wide water balance be undertaken to demonstrate that capacity exists to deliver the water requirements. It is understood that work on a site-wide water balance has commenced.

The Osborne Mine is licensed to extract up to 947 ML of groundwater from the southern borefield per year. The bore field extracts water from the sub-artesian Longsight Sandstone Aquifer, which is a part of the Great Artesian Basin. Anecdotally this water is of reasonably good quality as delivered. This water is processed through Reverse Osmosis (RO) equipment which is located near the water ponds and fire tanks. Water from the RO plant is then distributed around the site as potable water.

18.2.12 Osborne Maintenance Facilities

The following workshop facilities will be needed to deliver this business plan:

- Underground temporary / daily servicing type workshop – Osborne: One exists in the Osborne underground workings, near the 1000 mRL although this is distant from proposed workings. One of the incomplete projects when the mine was put on care-and-maintenance was the relocation of the underground mobile workshop closer to the bottom crusher entrance at 636 mRL. It is recommended that this location be re-assessed as Osborne underground mining will be down around the 60 mRL;
- Underground temporary / daily servicing type workshop – Kulthor: A temporary facility exists near the 1000 mRL truck tipple for the Kulthor workings. A more permanent underground maintenance facility may be required closer to the workings once the final underground design has been established;
- Underground mobile equipment – Osborne surface workshop: This currently exists and is in good condition, and seems adequate for the current requirement. This current location is someway from the portal – an inefficiency which may be exacerbated with mining at Kulthor. When final equipment numbers, working modes (mix of owner-operator and contractor) and tramming distances are understood, then the adequacy and location of this facility can be re-assessed. However at present it should be considered as adequate;
- Surface Fixed Plant workshop – Osborne: This facility will need to support:
  - Materials handling systems;
  - Concentrator plant operations;

- Power generation and electrical distribution; and
- Process water delivery.

This does currently exist, and is effectively run out of the same (or adjacent) facility as the surface mobile plant workshop.

- Open Pit Mining Workshop – Osborne: When open pit mining recommences, a dedicated surface facility for maintenance of the open pit fleet will be required. It is recommended this be sited as close to the pit as feasible. This is likely to be a contractor operated service; hence the fit-out of the workshop will be covered in the contractor's site mobilisation fee.

18.2.13 Osborne Site Accommodation

Osborne site has a fully functional accommodation facility 4 km south of the mine. There is permanent housing for 300 people. There is also temporary demountable units for a further 130 contract work force. When manning estimates are available for the full workforce, the capacity of this camp should be reassessed.

The camp catering contract should be reviewed to see if there are expansion notice periods or payments which may be triggered by resuming full utilisation of the camp.

Current maintenance budget allows for the replacement of a potable water tank at the camp.

19 Market Studies and Contracts

19.1 Concentrate Transport

It is planned for all concentrate products to be moved through the port of Townsville. Ideally, the concentrate would be railed from an inland rail hub (e.g. Phosphate Hill rail loadout facility) as has been done in the past, however, this is an unlikely scenario. At the Townsville port, there are two rail tippling facilities, both of which are privately held, and are unlikely to grant access to a third party, therefore for the purposes of this report, it has been assumed Ivanhoe does not have access to a rail tippling facility, undercover storage or a shiploading facility. Therefore, it is not possible to follow the previously used method of bulk rail wagons and bulk loading on ships.

An alternative method is to load into a ships hold by use of a tippling half-container. This method is used at a number of ports around Australia for volumes larger than planned by Ivanhoe.

Contract pricing has been received from a third party to collect concentrate from the Osborne site, transport, store and load to the hold of the ship.

This method of ship loading provides a completely different concentrate movement logistic. The use of the half-container for concentrate provides a number of synergies in the movement of material.

A stock of half-containers will be provided by the third party contractor for the movement, storage and loading of ships. The half-containers are stored in a secondary storage area at Townsville port. They are loaded onto triple roadtrains, with the use of a large port forklift. The roadtrains travel to the Osborne concentrate pad (a road distance of approximately 980 km). The half-container is loaded with 30 t of copper concentrate by a front end loader (FEL). It can be sampled at this point. A lid is secured on the load. Depending on roadtrain configuration (and lifting capacities of port forklifts and ships cranes), the payload to port would be about 80-90 t per roadtrain. The material would stay secured in the half-container. It will be unloaded in the port area, stacked, until required for ship loading. The roadtrain would return to site with more empty half-containers, for the next load. In 2012, this would need to occur at 1,200 t per week. Loading of ships with half-containers is currently being done at Townsville port.

Contract pricing from a third party has been used to estimate the costs associated with concentrate handling between the Osborne site and the ship's hold.

A contract is in place Northern Stevedoring Services Pty Ltd (NSS) of AUD110 per dry metric tonne (dmt) for transport from the Osborne pad loaded onto the ship at Townsville. All transportations requirements are handled by NSS.

19.2 Marketing

Osborne is predominantly a copper mine and therefore the business is highly sensitive to the price of copper. The global copper market reached unprecedented price levels in 2006 and these levels continued through 2007 and Q1 to Q3 2008. The impact of the US financial crisis at the end of September 2008 was immediately reflected in the copper price which dramatically decreased along with other base metals. Investors and traders are concerned about the rate and extent of a global economic slowdown and its effects on copper consumption.

However, the impact of China's stimulus package, along with a realisation that the recession was not as deep as anticipated and improvements in the US economy, has had a significant effect on the copper price with the price returning to >AUD2.50 /lb by August and >AUD3.00 in November 2009. The price of copper at the end of July 2012 was approximately AUD4.40 /lb.

The key export market for Osborne copper concentrate in 2012 will be either China or Japan depending on deliveries by traders. The majority of sales in 2012 will be on the spot market, taking advantage of the low spot quotations for smelting and refining. This is predicted to continue with a shortfall in concentrate availability, as well as investment and commissioning of new smelter capacity in China continues.

For the purposes of this study, it is expected that all product will be sold on the spot market, with the potential to negotiate fixed term contracts available at a later stage of project development.

19.3 Contracts

The sale of copper concentrates will be undertaken within the global copper market. Traditionally concentrate producers will market their product through a combination of both long term frame contracts, which are based around the annual Japanese benchmark pricing for TC/RC, as well as commit a proportion of production to the spot market. Over the past three years, the annual contracts have been in the range of AUD40-60 /dmt and 4.0-6.0 c/lb while spot contracts showed peaks in January 2009 and January 2011 of higher than AUD60 /dmt and 6.0 c/lb, but as low as AUD10 /dmt and 1.0 c/lb in the intervening period. Going forward, annual terms are around AUD55 /dmt and 5.5 c/lb, while spot terms are in the same range of AUD50-65 /dmt and 5.0- 6.5 c/lb. The spot terms will have pressure applied when other factors, such as adverse weather, labour relations and major production disruptions occur with concentrate production.

The volume of concentrate produced by Ivanhoe for 2012 is estimated at around 50,000 dmt, or four parcels. With the production schedule being developed, it is likely that these parcels will be offered to traders on the spot market. Monitoring of the market, as well as company cashflow requirements will be considered in the timing and volume to be offered. This will mean that a number of parcels will be offered with a specification range. Traders will reply with their offers, which will be considered, with the successful tenderer contracted to take delivery. Concentrates can be delivered to smelters in Korea, China, Japan, Philippines and India.

Shipping of concentrate will be conducted through a shipping agent, who will provide market information, organise contracts with ship owners, freight administration, verification for demurrage, other port costs and payments for freight volume. A tender for the shipping contract will conducted.

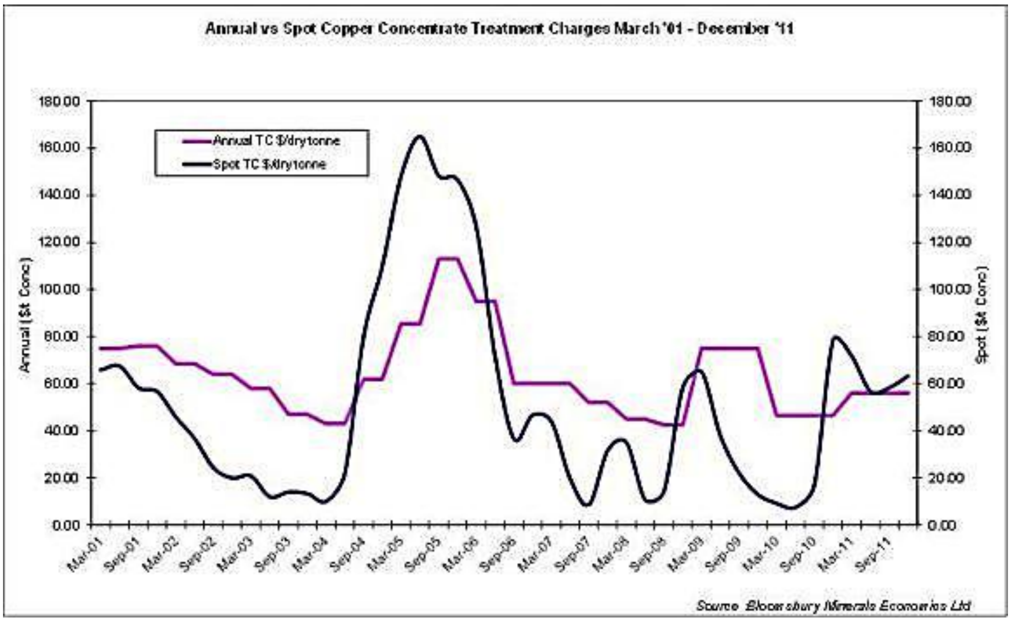


Figure 19-1: Annual versus spot copper concentrate treatment charges

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Relevant Environmental Legislation

The environmental management of mining operations in Queensland is covered by a number of state and federal Acts, including:

- *Aboriginal Lands Act 1991;*
- *Aboriginal Cultural Heritage Act 2003;*
- *Environmental Protection Act 1994* (EP Act) and the related *Environmental Protection Regulations 2008* (EP Reg);
- *Environment Protection and Biodiversity Conservation Act 1999* (EPBC) (Commonwealth)
- *Land Act 1994;*
- *Mineral Resources Act 1989* (MR Act);
- *National Greenhouse and Energy Reporting Act 2007* (NGER) (Commonwealth);
- *Native Title Act 1993 (Commonwealth);*
- *Native Title (Queensland) Act 1993;*
- *Vegetation Management Act 1999; and*
- *Water Act 2000.*

In Queensland, the *National Environmental Protection ("National Pollutant Inventory") Measure* (NPI) reporting requirements are implemented under the EP Act.

20.2 Environmental Permitting

Generally, approvals and related documents required under Queensland legislation are in place and there is no proven history of regulatory non-compliance. The Osborne project has environmental aspects that are administered by Department of Environment and Heritage Protection (DEHP) under the EP Act. An Environmental Management Plan (EMP), Environmental Authority (EA) and Plan of Operations (PoO) are in place for the Osborne and Kulthor operations.

Any proposed change or expansion that will result in the activity no longer being in accordance with the EA will require reassessment by DEHP. Depending on the scope of the proposed change, the assessment and approval process may vary from a minor modification of the EA (and associated update of the EMP and PoO to reassessment under the Environmental Impact Study (EIS) process. This is determined through an Assessment Level Decision under the EP Act which is supported by criteria found in the Amendment of Environmental Authority application form as follows:

Will the project, the subject of the application, be likely to:

- Have a significant impact on a category A or category B environmentally sensitive area;
- Involve mining in a marine area;
- Involve any mining less than 500 m landward from the highest astronomical tide;
- Require the construction of more than 150 new dwelling units;
- Include an environmentally relevant activity with an aggregate score of more than 165;
- Involve the mining of more than 2 Mt of mineral or ROM ore per year;
- Involve the abstraction of more than 2 Mm^3 of water per year from natural surface or groundwater Resources;

- Result in more than 25 ha non-beneficial land remaining post mining where an acceptable alternative may be feasible;
- Involve any Level 1 mining activity less than 2 km from a town;
- Contain a dam which requires a dam failure assessment under the Water Act 2000; and
- Include mining for uranium or asbestos.

An EIS will be required if an application for EA Level 1 Mining Project meets any of the above triggers. An EIS may also be required if the administering authority (DEHP) considers that there could be a significant environmental impact, if there is a high level of uncertainty about the possible impacts or there is a high level of public interest in the proposal.

In regards to the amendments related to the copper-gold business it is unlikely that an EIS would be required, as the mining and processing activities have been previously authorised under the relevant site EAs. However, should proposed southern expansion of TSF2 occur, it may be deemed to be the construction of a dam requiring failure assessment, in which case an EIS may be required.

Assessment at a Commonwealth level is not likely to be required as it is unlikely to trigger any actions under the EPBC Act.

Due to the purchase of Osborne Mine, Ivanhoe is now required to submit reports to meet National Greenhouse and Energy Reporting System and NPI requirements. It is understood that a consultant has been engaged to assist Ivanhoe with the preparation of these reports. Determination of potential future emissions should be considered in the context of the recently proposed implementation of a 'carbon tax'.

20.3 Summary of Environmental Liabilities: Osborne and Kulthor Operations

The key environmental liabilities associated with the Osborne and Kulthor operations are:

- Potentially acid-generating material stored in waste rock dumps, tailings storage facilities and minor materials stored in the Osborne Pit;
- Management of ground and surface water which cannot be discharged from site due to high concentrations of high sulphate, Total Dissolved Solids (TDS) concentrations and elevated metal levels;
- Soil contamination due to dust generated from the concentrate plant;
- The current level of monitoring is considered not to be fully proportionate with the level of risk associated with acid drainage or dust management; and
- Further work is required to implement control for these issues as the operation is expanded.

These issues could potentially result in a financial liability being incurred, through imposed fines or through additional closure costs or could impact on the operations licence to operate. These liabilities are discussed in the following sections.

20.3.1 Potential Acid Generation

Waste Rock Dumps

Development of the Osborne Pit from 1995 generated approximately 15 Mt of waste rock. 10 Mt of waste rock classified as non-acid forming (NAF), silcrete, mudstones and siltstones was used to build the ROM stockpile, hard stand areas and a single waste rock dump (waste dump) located immediately to the north-west of the plan (Williams, 1996).

The report 'Capping Potentially Acid Forming Waste Rock at Osborne Mines" (Williams, 1996) states that 950,000 t of potentially acid forming (PAF) waste rock, containing 0.5% to 3% Sulphur in Figure 20-2 was excavated and placed within NAF rock which forms the upper two lifts of the waste dump (Williams, 1996). The PAF containment cell was completed by 1996 and was designed to encapsulate a layer of PAF material which is 14 m thick and has a diameter of approximately 120 m.

The PAF material is underlain by 20 m of NAF material and overlain by between 6 m and 20 m of NAF material, which in turn is covered by a metre of 'rocky soil' mulch, designed to act a 'store and release' cover (Sustainability Plan, 2010).

The following comments summarise the main issues associated with the Osborne Waste Rock Dump (waste dump):

- The material held in the waste dump is potentially acid forming and contains appreciable levels of copper and cobalt;
- Although the single piezometer installed through the waste dump has not recorded standing water within the waste mass, seepage has been observed at the eastern side of the waste dump. The Osborne Sustainability Plan 2009 identifies the likely source of the acid seepage as the ROM pad. The seepage water is acidic, saline and high in some metals. It is not of a quality that could be legally discharged under the current EA. Seepage water from the eastern side of the waste dump reports to Environment Dam 3;
- Environment Dam 3 has discharged in the past (2000 & 2004); however recent work has increased the dam's storage capacity, and the addition of a Site Water Management Plan in 2009 has ensured no further discharges have occurred despite several recent above average wet seasons; and
- The Osborne Sustainability Plan 2009 includes a proposal to remove acid producing waste rock from the waste Rock Dump to the open pit on closure as a management option.

Figure 20-1: PAF cell on northern edge of the waste dump and RoM stockpile

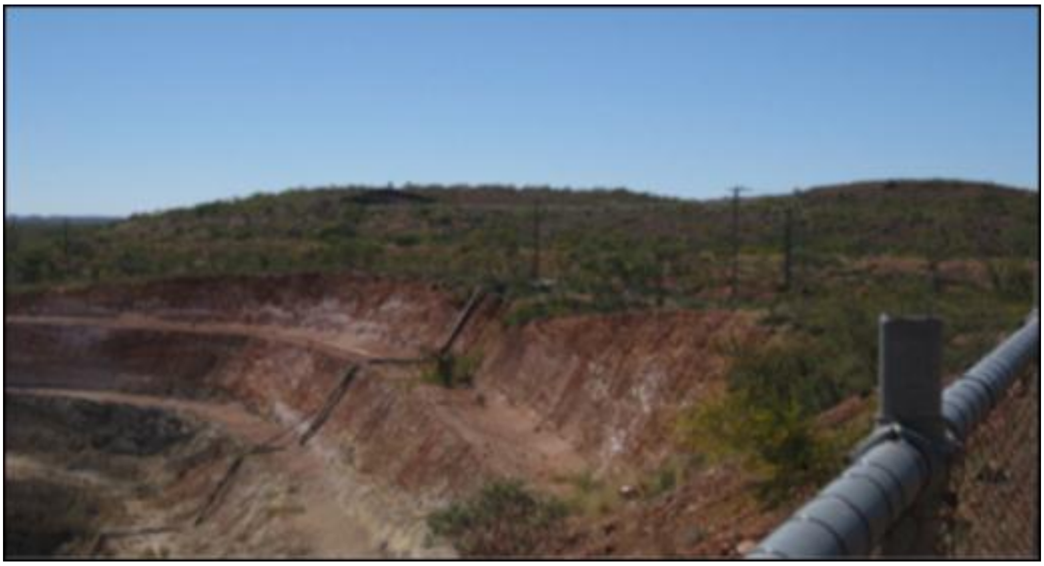


Figure 20-2: Photo of waste dump from the pit edge

Based on the available information, it is considered that the existing waste dump currently represents an environmental liability, due to the seepage issues noted above. The acid mine seepage and the presence of acid generating materials will require effective rehabilitation and management of this structure in the longer term to achieve surrender of the mining lease.

Tailings Storage Facilities

Two TSFs are present on the Osborne site (both on ML90040). The older of the two is TSF1 which comprises three cells. TSF1 is located to the south-west of the processing plant and was not being used at the time of closure. Prior to closure, tailings were being deposited into TSF2, a single cell facility to the southeast of the plant.

Tailings Storage Facility 1

The three cells forming TSF1 are known as, the ‘oxide dam’, the ‘pyrite dam’ and the ‘sulphide dam’. As suggested by the names, the cells were constructed to take the two separate waste streams initially produced from the plant. The ‘pyrite dam’ has been rehabilitated with a water shedding cover, which is has been vegetated. It has been also instrumented as a trial to determine cover effectiveness; however these results have not been reviewed. A stockpile of granular material was noted in the area between the ‘oxide dam and ‘pyrite dam’ which appeared to be leaching copper, as shown in Figure 20-3, which suggests copper could be migrating via runoff or dust.

Figure 20-3: Surface of Oxide Dam and Copper leaching material

The ‘sulphide dam’ is the largest of the three cells with an area of 25 ha and a height of 23 m (the ‘pyrite cell’ is 12 m high and the ‘oxide dam’ is 6.5 m high). The ‘sulphide dam’ contains the sulphide (flotation) tailings which were produced from the treatment of the primary sulphide orebody. This dam has also been in care and maintenance since 2002, and has been subject to dust suppression spraying and clearing of internal drains. The surface of the cell is shown in Figure 20-4.

Figure 20-4: Sulphide Dam

During the wet season, discharge to the collection ponds occurs at a rate of approximately 2 L/s from the ‘sulphide dam’ and 1 L/s from the ‘pyrite dam’, reducing to 0.2 L/s and 0.1 L/s respectively in the dry season. Water collected in these ponds is automatically pumped back to the process plant. Overflow from the collection ponds due to wet season events is contained by Environmental Dam 1 (ED1) which forms to the north-west of the seepage ponds. The report by Metago, detailing the 2010 Annual Inspection Audit, indicates that an effluent discharge of approximately 2,000 m^3 occurred to ED1 from TSF1 in January 2010. ED1 has a design storage allowance of 140,000 m^3 and a current storage capacity of 192,147 m^3. ED1 is designed to collect run off or discharges from TSF1 that cannot be managed by the two collection dams. SRK expects that discharge events from the collection ponds to ED1 would be expected to occur during above average wet season events. The current PoO (March 2011) states that ED1 has not discharged during the history of the project. ED1 is unlined and the discharge events could potentially cause contamination of the sediment/soils, which would be a liability to the relinquishment of the lease.

The 2007 TSF inspection report stated that piezometers installed in the ‘sulphide cell’ in 2003, indicated that the phreatic level was 20 m below the surface of the cell and that it was dropping by 0.3 m per year. The 2010 TSF inspection report indicated that these piezometers were no longer being monitored.

The tailings contained within TFS1 are known to be net acid producing (approximately 50.2 kg/t H_2SO_4 and a sulphur-sulphide content of 2.8%) and also saline (4,700 µS/cm to 59,000 µS/cm), while elevated levels of copper, zinc, cobalt, nickel and selenium have been noted in the discharge water (PoO, 2011). The quality of the water discharged to the collection ponds varies depending on the amount of rainfall. Initial higher flows associated with heavy rain have lower pH values than flows during the dry season.

The pH and conductivity of water discharged from the ‘sulphide cell’ (as reported in the 2010 TSF inspection report) generally ranged from pH 5 to 6.5 (with occasional readings of pH 2) and conductivity ranged from 6.5 millisiemens per centimetre (mS/cm) to 12 mS/cm. The results reported as part of the 2010 inspection are similar to those reported in 2004 and 2007 (pH: 5 to 6.5, EC 9.8 mS/cm to 12.3 mS/cm), no results were included in the 1997 report.

The pH and conductivity of water discharged from the pyrite cell (as reported in the 2010 TSF inspection report) generally ranged from pH 3.8 to 4.2 (with occasional readings of pH 3) and conductivity ranged from 4 mS/cm to 6 mS/cm. The results reported as part of the 2010 inspection are more acidic but less saline than those reported in 2004 and 2007 (pH: 4.5 to 6, EC 13 mS/cm to 16 mS/cm). No results were included in the 1997 report. It appears that the salinity of the water discharge water is decreasing; however there is also a decrease in the pH.

No impact on the groundwater quality has been noted, however, limited sampling has been conducted due to the majority of bores being dry on most sampling occasions. This is not a categorical indication that groundwater contamination is not occurring. Consideration could be given to the installation of new monitoring bores and recommencement of monitoring of bores within the ‘sulphide cell’.

TSF1 represents a high environmental liability to the operation as it is known to be acid producing and could potentially cause contamination of the surrounding surface waters, groundwater and soil. Effective rehabilitation and management of this structure in the longer term to achieve surrender of the mining lease will be difficult.

Failure to continue the proper management of the facility could result in the release of water/sediment with low pH and elevated salinity and metals which could lead to contamination of the surrounding environment.

Tailings Storage Facility 2

TSF2 also represents a potential liability as the tailings contained are also acid generating and saline (42.5 kg/tonne H_2SO_4 and 2.4% sulphur sulphide). TSF2 is a single cell facility which covers an area of 103 ha. Supernatant water and seepage (collected by toe drains, an under drain and Sump A) is directed to and retained in the TSF2 reclaim pond and typically has a pH range of 4 to 7, with an EC range of 9 mS/cm to 20 mS/cm. Some water or tailings maybe flowing to the east of the TSF, in the vicinity of MB6, as discussed below. The pH of the dam water varies, depending upon rainfall. When the plant is in operation, water is pumped from the reclaim pond into the process pond for re-use.

The reclaim dam has a capacity of 422,734 m^3 and is also used to store water from the underground operations, and overflow from the process pond in the event of a high rainfall storm event.

The 2004 TSF Inspection report indicated that during that reporting period (2003/04) there were five kangaroo deaths due to animals becoming trapped in soft tailings. No fauna incidents were recorded in the 2007 or 2010 inspection reports. Ivanhoe staff indicated that the site is now fully fenced (though the date this was completed is unknown) to prevent stock access and there have been no further fauna deaths.

Figure 20-5: View of TSF2 looking north towards deposition point

The northern half of TSF2 has been covered by a 'dust cover' comprising a 0.5 m thick layer of crushed Mesozoic rock in Figure 20-6. This cover is designed to be a temporary cover during the life of the recommenced operation to limit dust generation from the area of the facility which is not being used for active deposition. Currently, the remainder of the TSF is being irrigated to limit dust. The 'dust cover' has been seeded and vegetation generally comprises tumble weed, acacias and succulents. It is understood that the dust cover will be progressively expanded over the TSF as the deposition point moves southwards. During the site visit, areas of salt were observed on the uncovered portion of the TSF beach in Figure 20-7.

Figure 20-6: Dust Cover on TSF2

Figure 20-7: Salt Crust on TSF2

The majority of monitoring bores installed around TFS2 have not detected groundwater; these bores (with the exception of MB6 and MB8) are typically 35 m to 50 m deep and as such are above the natural groundwater level (which is typically greater than 50 m deep). MB6, which is on the eastern side of the facility, was drilled to 50.86 and contained groundwater prior to the placement of tailings. A slight change in water quality has been recorded, with an increase in conductivity and sulphates; however copper levels and pH have remained relatively unchanged and do not exceed the EA levels.

No rise in water level in response to deposition or rainfall has been seen within this bore and as such it has been concluded by other consultants (Martin Bosch Sell (MBS) Environmental and others) that the bore and the TSF are not hydraulically connected. The Rob Lait Pty Ltd report Hydrological Assessments for Close Down suggests that water from the TSF may be flowing across the ground surface and down the annulus and reaching the slotted casing due to a failure of the grout / bentonite seal, however this would indicate surface waters are potentially contaminated by tailings leaving the TSF and that the seepage containment system is not fully effective.

MB8 was installed in 2009 after the expansion of the dam and is stated as being drilled to a depth of 78 m (as stated in the Osborne Sustainability Plan and the MBS report on Water Quality Assessment, 2010) which is below the Mesozoic/Palaeozoic boundary; however, other reports indicate that the bore was only drilled to 53 m depth. Ivanhoe staff indicated that the bore was drilled to 78 m but was only completed as a standpipe to 53 m depth. Water with similar chemistry to that of the TSF was detected in this bore soon after installation.

A number of studies, including detailed analysis of the mineral composition, determination of mineral Saturation Indices and an electromagnetic induction (EMI) survey were undertaken to determine if this water was seepage or natural groundwater. The studies undertaken by Rob Lait & Associates Pty Ltd (2010) and MBS Environmental Pty Ltd (2010) concluded that while there were similarities in water quality (pH levels, elevated salinity, and metals), the water in MB8 was closer in composition to natural groundwater associated with Palaeozoic basement rocks and that the EMI survey indicated that there was no connection between the decant pond and the aquifer intersected by MB8. MB10 was drilled to 51.72 m, between the TSF and MB6, and did not encounter any water.

This is taken to confirm that water encountered in MB8 is from an aquifer associated with the Palaeozoic unit.

As noted above, TSF2 represents a source of environmental liability. TSF2 has the potential to produce acid and could potentially cause contamination of the surrounding surface waters, groundwater and soil. Effective rehabilitation and management of this structure in the longer term to achieve surrender of the mining lease will be difficult.

Failure to continue the proper management of the facility could result in the release of water/sediment with low pH and elevated salinity and metals which could lead to contamination of the surrounding environment.

20.3.2 Osborne Open Pit

Open cut mining at Osborne commenced in 1996 and is described in Section 16-1. The pit walls have exposed faces of all rock types including the meta-sedimentary ironstone which forms the host rock of the orebody. This material is potentially acid generating, no visible evidence of this has been observed during the site inspection, however this inspection was made from a viewing platform and close-up inspection was not undertaken.

At the time of inspection, no water was present within the pit. Groundwater levels are reported to be below the depth of the pit and no inflow into the pit occurs, as per PoO and the Osborne Sustainability Plan. Ivanhoe staff indicated that during the wet season, rainfall and run-off from the pit walls collects in the pit. This water is pumped from a sump at the base of the pit into the process pond and incorporated as process water.

Figure 20-8: Osborne Pit

Figure 20-9: Low grade stockpiles

At the time of the site visit portions of the pit floor were being used to store stockpiles of low grade ore. It is understood that this material was collected from the ROM and other areas of the plant. As shown in Figure 20-8, this material shows evidence of leaching of copper. This material is stored at the northern and southern ends of the pit, above the level of the sump and it is likely that run-off from the stockpiles will report to the sump and from there be pumped into the process pond, which potentially adds to the contaminant load present in the water management system.

20.3.3 Underground Mine Void Water Quality

During operations, the Osborne shaft was dewatered at a rate of 4.69 L/s into the process pond and then to the reclaim pond associated with TSF2, and then pumped back for use in the plant. Currently, mine void water is being pumped into ED1 and ED3 and also into the oxide dam to maximise evaporation. Mine void water is generally very saline, with high sulphate TDS concentrations and elevated metal levels.

Information provided by Ivanhoe indicated that it is proposed to dewater the mine to allow operations to recommence and that this water will be discharged to the TSF2 reclaim pond. SRK is not aware of any water balance or studies which confirm that there is sufficient storage to contain this water along with water from other sources, particularly if dewatering coincides with a significant wet season.

It is further noted that, based on results provided in the MBS report (2010), the quality of the water to be discharged has the potential to exceed the environmental dam limits set in the EA for chloride, sulphate, cobalt and copper. It is recommended that future studies for the proposed recommencement include the review of the water management system.

Should operations not recommence, it is understood that the strategy to deal with water collected in the reclaim pond is to pump it into the mine void. A report, “Hydrological Assessments for Close Down” by Rob Lait Pty Ltd (2010) was prepared in support of this strategy. The report concluded that only minor changes to the groundwater chemistry should occur and that these would be limited in extent, only affecting water within 200 m of the void. It is not known if this strategy has been reviewed by the regulatory authorities.

20.3.4 Surface Water Management

The following issues are noted in relation to surface water management at the site:

- Four unlined environmental dams are present on the site and are used to store and distribute potentially contaminated water generated by run off from the TSF, waste dump and process plant. The environmental dams are typically only required in the wet season and due to high evaporation rates, do not hold water for extended periods of time;
- The Osborne Water Management Plan (October 2010) indicates that there have been two instances of overflow from the dams on site that resulted to a release of water to the receiving environment. These occurred in December 2000 and January 2004, both from Environmental Dam 3 which captures run off from the plant and waste dump;
- No monitoring bores are present downstream from Environmental Dams 2, 3 and 4. Ivanhoe staff have indicated that there are no visible signs of any shallow seepage occurring; and
- A drain and sump immediately to the north of the process pond (as shown in Figure 20-10 and Figure 20-11) was observed to have signs of salt precipitate and/or oxidation and copper leaching. This appears to run off from the process plant.

Figure 20-10: Drain/sump showing possible metal leaching

Figure 20-11: Close up of drain/sump

The main risk associated with surface water dams on the site is the risk of contamination of natural drainage systems with metals and salts through over-spilling or seepage. Planning and scoping work will need to include determination of the storage requirements for the expanded operation.

Another risk is the contamination of soil within the dams by heavy metals. This risk has been accounted for in the Osborne Sustainability Plan (2009) which includes the requirement for investigation and remediation of sediment within the dams.

The capacity and location of drainage features and environmental dams will need to be reviewed during the preliminary planning stage to ensure that they are able to contain potentially contaminated run off from the expanded operation.

20.3.5 Process Plant

At the time of the site visit, the process plant was in care and maintenance. It is understood that the plant will be re-commissioned to take ore from the Osborne and Kulthor underground deposits, as well as from the Starra 276 deposit and that it is proposed to construct a roaster and concentrator for molybdenum-rhenium extraction.

Inspection of the plant was confined to a drive through of the site and discussion with Ivanhoe staff. It appeared that all chemicals on site were either in dangerous goods containers or within bunded areas. Oil-water separators were present at the re-fuelling point and in the vehicle service bay.

There are two 1.1 ML aboveground storage tanks on site. These are used to store diesel to feed the power station on site. The tanks are within concrete bunds. Ivanhoe staff indicated that in 2003, approximately 500 kL was spilt due a pipe/valve malfunction (outside the bunded area). It was further indicated that the soil contaminated as a result of the spill was most likely placed in one of the TSFs and that is the current practise for dealing with spills of fuel or other hydrocarbons.

A contamination investigation (Detailed Site Investigation – Land Contamination Associated with the Osborne Ore Processing and Concentrate Handling Areas, 2010) was undertaken by Barrick Osborne as part of mine closure work. The investigation determined that elevated levels of copper, cobalt and sulphur (exceeding the environmental investigation levels (EIL) and the health investigation levels for open space (HIL E)) were present in the areas around the process plant.

Concentration of up to 16,000 mg/kg copper, 33,900 mg/kg sulphur and 377 mg/kg cobalt were stated in the above report. These levels were found to decrease with distance from the ROM and process plant.

Other metals such as chromium and nickel were also detected in levels exceeding the EIL, but only in a limited number of locations. The Osborne ore itself has relatively low levels of other potentially contaminating metals, with none exceeding the EIL.

Testing of water at downstream monitoring sites during the wet season by Ivanhoe indicated that total copper levels (0.082 mg/L to 0.373 mg/L), exceeding the EA trigger value of the 80th percentile of background levels (0.036 mg/L) but were less than the EA limit of 1 mg/L were noted in the first flush samples.

The report indicates that the migration of metals, either in soil or water is not occurring, however some localised contamination in present in soil close to the process plant. While monitoring has not detected any 'exceedences' of EA limit, the presence of metals above the background levels suggests to SRK that some migration of metals is occurring via dust and surface water, which over time may lead to contamination of down-stream sediments.

Ivanhoe staff indicated that there had been a programme to clean up surface copper contamination on areas to the north of the plant and TSF2 after vegetation stress was noted. It was thought that this contamination was due to dust from the process plant. The clean-up involved the scraping up of discoloured soil (typically grey) over an area of 4 ha. This is understood to have cost approximately AUD45,000. Ivanhoe staff indicated that changes to the plant and concentrate storage shed were being undertaken to reduce the potential for dust generation. Dust monitoring is being undertaken every two months and will increase to monthly when operations recommence. Under EA conditions, dust monitoring is required to ensure that dust released from the site does not cause an 'environmental nuisance, at any sensitive or commercial place'.

Soil and water contamination due to processing represents a potentially major environmental liability, particularly if contamination continues to migrate off the site (i.e. downstream from the environmental dams) and could potentially affect the sites licence to operate. It is considered that there are some controls and management strategies in place to control this issue of soil and water contamination; however the proposed operation will provide additional pressures on these systems. Ivanhoe will need to consider these issues during the approvals and design phase.

20.3.6 Kulthor Underground Operation

The Kulthor deposit lies approximately 2.5 km to the west of Osborne Mine and will be developed as a satellite deposit. Extraction of ore has not commenced from the deposit; development of the linking drive was halted approximately 140 m away from the deposit. Ore will be transported via the drive back to Osborne for processing in the existing plant, with tailings reporting to TSF2. The PoO indicates that waste rock generated at Kulthor will stay underground as back filling, this limits the potential for impacts due to acid mine drainage from surface waste dumps. During operations, Kulthor will require dewatering at a rate up to 400 kL per day, prior to mining the drive. The orebody which is within the Kulthor Shear zone will require dewatering, and this is expected to require the extraction of between 100 ML to 200 ML of water (PoO, 2011). Water generated from the Kulthor deposit will be similar in composition to that of the Osborne mine (refer to Section 20.3.3) and the same issues with storage capacity and water quality potentially exist.

Surface development at Kulthor (following completion of development) will be limited two ventilation shafts and dewatering bores. Licences are in place for the dewatering bores (refer to Section 4.2). The ventilation shafts could potentially generate a saline aerosol, which could impact on vegetation near the shaft rise. Monitoring should be undertaken to ensure this does not occur.

Saline aerosols can generally be managed by use of a shrouding system to collect and condensate the vapour prior to discharge into the process water system or reinjection.

20.3.7 Closure and Financial Security

A closure plan for the site exists in the form of the Osborne Sustainability Plan. SRK were provided with Version 3.1 of the plan (updated in March 2010). The plan covers the previous operation and will need to be updated to reflect the proposed recommencement and expansion.

As given in the PoO, the estimated rehabilitation cost is AUD23.8 M, with an applicable Category 2 discount of 20% applying, making the amount required to be lodged with DEHP AUD19.1 M. A Category 2 discount is applicable when the holder of the EA has demonstrated good environmental performance over the preceding two years and has shown that progressive rehabilitation has been undertaken. It is understood that the amount of security currently held by DEHP/DNRM is AU18.3 M. As such, an additional AUD0.8 M needs to be lodged. This amount will increase with the expansion of the project.

20.4 Social and Community Impacts

The Osborne copper-gold project is within the current mining area which is under existing arrangements with the local communities. IAL have indicated that there are not expected to be any concerns with the community that may interrupt the mining of the Osborne copper-gold project.

21 Capital and Operating Costs

Costs for the following section have been derived from a variety of sources, including:

- Historic Barrick production from Osborne;
- Manufacturer suppliers; and
- First principle calculations (based on historic production values).

21.1 Capital Costs

An overview of the capital costs for the Osborne copper-gold project is presented in Table 21-1.

Table 21-1: Capital Costs overview

Item	Total (AUD M)
Osborne Open Pit	29.02
Osborne Underground	0
Kulthor Underground	43.57
Total	**72.59**

21.2 Osborne Open Pit

The capital expenditure requirements for the Osborne Open Pit are shown in Table 21-2. Pre-stripping of the open pit has not been included in the capital expenditure for the open pit.

Table 21-2: Osborne Open Pit Capital Expenditure

Item	Total (AUD M)
Pre-strip	16.25
Mobilisation	3.02
Demobilisation	0.67
Grade control drilling	1.08
Sustaining capital	8.00
Total	**29.02**

21.3 Osborne Underground Mine

There are no further capital works to be completed for the extractions of the Osborne underground mine Mineral Reserve.

21.4 Kulthor Underground Mine

The capital expenditure requirements for the Kulthor underground mine are shown in Table 21-3. The mine development capital includes the decline, level access, ventilation and escapeway development.

Table 21-3: Kulthor Underground Capital Expenditure

Item	Total (AUD M)
Infrastructure	2.08
Mine Development	36.99
Sustaining Capital	4.50
Total	**43.57**

21.4.1 Lateral Development

Kulthor development is currently being undertaken by PYBAR Mining Services, a recognised Australian underground mining contractor. Approximately 9,900 m of development is required at a quoted price of AUD6,780 /m which includes the diesel and fly-in-fly-out (FIFO) costs for the contractor.

21.4.2 Vertical Development

The vertical development for the main shaft has been completed at Kulthor. Table 21-4 summarises the capital vertical development at the Kulthor Mine.

Table 21-4: Capital Vertical Development

Item	Cost (AUD M)	Status
K1 Ventilation Shaft		Complete
Internal Ventilation Shaft 830 mRL – 710 mRL		Complete
Egress Ladderway 830 mRL – 715 mRL		Complete
Egress Ladderway 715 mRL – 540 mRL	0.88	To be completed
Internal Vent Raises (drill and blast) (6 raises)	0.21	To be completed

21.4.3 Underground Infrastructure

Allowance for underground infrastructure is detailed in the Table 21-5.

Table 21-5: Kulthor Underground Capital Infrastructure Estimates

Description	Unit Rate	Quantity	Total (AUD '000)
Underground Pumping Stations	132,000 each	3	395
Underground Electrical Substations	180,000 each	3	540
11 kV Cabling and Reticulation		1	200
1000 V Cabling and Reticulation		1	300
Refuge Chambers	120,000 each	2	240
Ladder way Installations	2,000 /m	200	400
Total Capital Infrastructure			**2,075**

21.5 Operating Costs

21.5.1 Osborne Open Pit

The total operational costs anticipated for the Osborne open pit mine are summarised in Table 21-6.

Table 21-6: Osborne Open Pit Total Operating Costs

Item	Unit Cost (AUD / ore t)	Total Cost (AUD M)
Mining	14.97	37.41
Processing	10.60	26.49
General and Site Administration	4.24	10.59
Off Site Costs	5.99	14.96
Total	**35.79**	**89.45**

The mine operating costs for the Osborne open pit mine are summarised in Table 21-7.

Table 21-7: Osborne Open Pit Mine Operating Costs

Item	Unit Cost (AUD / ore t)	Total Cost (AUD M)
Project Management	1.43	3.58
Plant Ownership	1.73	4.32
Load and Haul	4.95	12.37
Drill and Blast	4.00	9.99
Fuel	2.39	5.96
Other Costs – Technical Services, Maintenance, UG Services	0.48	1.19
Total	**14.97**	**37.41**

21.5.2 Osborne Underground Mine

The total operational costs anticipated for the Osborne underground mine are summarised in Table 21-8.

Table 21-8: Osborne Underground Total Operating Costs

Item	Unit Cost (AUD /t)	Total Cost (AUD M)
Mining	32.11	15.78
Processing	10.60	5.21
General and Site Administration	7.30	3.59
Off Site Costs	15.34	7.54
Total	**65.35**	**32.10**

The mine operating costs for the Osborne underground mine are summarised in Table 21-9.

Table 21-9: Osborne Underground Mine Operating Costs

Item	Unit Cost (AUD /stope t)	Unit Cost (AUD /t)	Total Cost (AUD M)
Grade Control	1.33	1.27	0.62
Lateral Development	2.91	2.78	1.37
Drill and Blast	6.48	6.20	3.05
Loading and backfill	3.18	3.04	1.49
Trucking	7.82	7.48	3.68
Other Costs – Technical Services, Maintenance, Underground Services	11.84	11.33	5.57
Total	**33.56**	**32.11**	**15.78**

21.5.3 Kulthor Underground Mine

The total operational costs anticipated for the Kulthor underground mine are summarised in Table 21-10.

Table 21-10: Kulthor Total Operating Costs

Item	Unit Cost (AUD /t)	Total Cost (AUD M)
Mining	37.89	97.58
Processing	10.60	27.30
General and Site Administration	7.30	18.80
Off Site Costs	11.30	29.09
Total	**67.09**	**172.76**

The mine operating costs for the Kulthor Mine are summarised in Table 21-11.

Table 21-11: Kulthor Underground Mine Operating Costs

Item	Unit Cost (AUD /stope t)	Unit Cost (AUD /t)	Total Cost (AUD M)
Grade Control	1.49	1.27	3.27
Lateral Development	15.71	13.36	34.41
Drill and Blast	6.48	5.51	14.20
Loading and backfill	4.16	3.54	9.11
Trucking	4.84	4.12	10.60
Other Costs – Technical Services, Maintenance, UG Services	11.86	10.09	25.98
Total	**44.54**	**37.89**	**97.57**

21.5.4 Processing Costs

Processing unit cost per tonne of AUD10.60 /t has been estimated from Osborne operating budget.

21.5.5 General and Site Administration Costs

The general and site administration costs include costs that have not been included elsewhere in the operating costs. These costs include maintenance, site management, occupational health and safety, accounting and finance, warehousing and logistics and information services.

21.6 Offsite costs

21.6.1 Concentrate Transport

Site to ship, contract pricing received for transport from Osborne site to be loaded in ship hold for AUD110 /t, allowing for 9% moisture equates to AUD119.90 per dmt of concentrate.

21.6.2 Concentrate Shipping

The average price concentrate shipping to overseas smelters of AUD43.60 /dmt is based on current contractual arrangements. This equates to AUD39.68 /t allowing for 9% moisture.

21.6.3 Smelting & Refining

Forecast treatment and refining charges in line with information received from market reports (Refer Section 19) of USD55 per concentrate tonne and USD 0.055 per pound of copper and USD5 per ounce gold have been used.

21.6.4 Marketing & Assays

Costs associated with marketing and assays have been estimated using historical data from Osborne operation prior to suspending shipping and sales in February 2011. An allowance of

AUD5.5 per dmt of concentrate has been used. This equates to AUD5.01 /t allowing for 9% moisture.

21.6.5 Royalties

Royalties are payable to the Queensland Government based on the sale of copper and gold as shown in Table 21-12. The amount of royalty will change based on metal prices and exchange rates.

Table 21-12: Queensland Government Royalty

Item	Rate (% of sales)
Copper Royalty	4.8
Gold Royalty	5.0

22 Economic Analysis

22.1 Introduction

The Osborne Open Pit, Osborne Underground and Kulthor cashflow models were developed by SRK for each deposit. All costs and revenues are constant in 2012 AUD with no provision for inflation or escalation.

The annual cash flow projections were estimated over the Project's initial production period based on capital expenditures, production costs, corporate costs and sales revenue. The financial indicators examined included net present value (NPV).

22.2 Financial models

The economic models for the project were prepared with the aim of evaluating each deposit separately. It has been assumed that the sufficient mill feed will be available to maintain the milling rate when each deposit is mined.

Table 22-1 provides the key economic assumptions used in the financial model. The commodity price assumptions are based on a short to medium term view following the review of:

- Historical pricing;
- Spot prices;
- Analyst forecasts; and
- Forward curves, hedging arrangements as observed in the marketplace.

Table 22-1: Key economic assumptions used in the Financial Model

Assumption	Unit	Rate
Commodity prices		
Copper price	USD/lb	3.25
Gold price	USD/oz	1,400
Exchange rate		
AUD/USD	AUD: USD	1.00
Other		
Discount rate (real)	%	8.6

22.3 Model Inputs

Table 22-2 summarises the common assumptions utilised as part of the economic modelling.

Table 22-2: Common assumptions

Item	Units	Value	Source
Royalty payments			
Queensland State Government			
QSG royalty—Copper	%	4.8	MRR 2003, Schedule 4, Part 1, 2
QSG royalty—Gold	%	5.0	MRR 2003, Schedule 4, Part 1, 2
Common cost assumptions			
Marketing costs			
Concentrate marketing	AUD/dmt (conc)	5.5	Ivanhoe estimate
Gold sales unit cost	AUD/oz	5.0	Ivanhoe estimate
Shipping costs			
Transport to Townsville port/storage	AUD/dmt (conc)	119.9	Ivanhoe estimate
Shipping (sea freight)	AUD/dmt (conc)	43.6	Ivanhoe estimate
Other			
Refining charge	AUD/lb	0.055	Ivanhoe estimate
Treatment charge	AUD/t (conc)	55.0	Ivanhoe estimate
Item	**Units**	**Value**	**Source**
Other assumptions			
Concentrate grade			
Concentrate grade—Copper	% copper	24.0	Ivanhoe estimate
Gold reporting to Copper concentrate	% copper	100.0	Ivanhoe assumption (simplifying assumption)
Processing Recoveries			
Osborne Open Pit - Copper	%	85	Ivanhoe estimate
Osborne Open Pit - Copper	%	75	Ivanhoe estimate
Osborne Underground - Copper	%	90	Ivanhoe estimate
Osborne Underground - Gold	%	80	Ivanhoe estimate
Kulthor Underground - Copper	%	85	Ivanhoe estimate
Kulthor Underground - Gold	%	75	Ivanhoe estimate
Payable metal in Copper concentrate			
Payable metal—Copper	%	95.8	Ivanhoe estimate
Payable metal—Gold	%	94.0	Ivanhoe estimate

22.4 Osborne Open Pit

The key metrics are summarised in Table 22-3. The output from the financial model (Osborne Open Pit Cost Model_Rev1.xls) is shown in Table 22-4. No NPV or payback period has been calculated for the Osborne open pit given the mine life is less than two years.

Table 22-3: Summary of Key Financial Parameters

Parameter	Units	Value
Tonnes Processed	t	2,499,389
Total OPEX	AUD M	73.20
Total CAPEX	AUD M	29.02
Royalty	AUD M	7.33
Total Cost	AUD M	109.55
Copper Produced	Mlb	33.14
Gold Produced	ozs	28,950
Total Revenue	AUD M	148.2
Cashflow	AUD M	21.23
IRR	%	17.6

Table 22-4: Osborne Open Pit Cost Model

	Units	Yr 1 Qtr 1	Yr 1 Qtr 2	Yr 1 Qtr 3	Yr 1 Qtr 4	Yr 2 Qtr 1	Yr 2 Qtr 2	Yr 2 Qtr 3	Yr 2 Qtr 4	Total
Diluted Ore Tonnes	t			180,676	629,191	624,568	15,742	777,787	271,424	**2,499,389**
Waste Tonnes	t	2,482,862	2,518,542	2,647,730	1,560,612	2,098,681	2,146,298	2,298,144	495,525	**16,248,395**
Total Material Movement	t	**2,482,862**	**2,518,542**	**2,828,406**	**2,189,803**	**2,723,250**	**2,162,040**	**3,075,931**	**766,950**	**18,747,784**
Copper Grade	%	0.00	0.00	0.91	0.76	0.70	1.34	0.65	1.64	**0.82**
Gold Grade	g/t	0.00	0.00	0.74	0.53	0.50	0.77	0.46	1.00	**0.57**
Contained Copper	t			1,644	4,782	4,372	211	5,056	4,451	**20,516**
Contained Gold	ozs			4,274	10,750	10,044	389	11,445	8,725	**45,627**
Copper Concentrate	t			5,021	14,602	13,350	644	15,438	13,593	**62,648**
Copper Produced	lb			2,655,711	7,723,870	7,061,819	340,726	8,166,058	7,190,038	**33,138,223**
Gold Produced	oz			2,712	6,821	6,373	247	7,262	5,536	**28,950**
Project Management	AUD M			(0.46)	(0.53)	(0.65)	(0.52)	(0.74)	(0.18)	(2.16)
Plant Ownership	AUD M			(0.56)	(0.64)	(0.79)	(0.63)	(0.89)	(0.22)	(2.61)
Miscellaneous Fixed Costs	AUD M			(0.15)	(0.18)	(0.22)	(0.17)	(0.25)	(0.06)	(0.72)
Load and Haul	AUD M			(1.60)	(1.82)	(2.26)	(1.79)	(2.55)	(0.64)	(7.47)
Drill and Blast	AUD M			(1.29)	(1.47)	(1.82)	(1.45)	(2.06)	(0.51)	(6.03)
Fuel	AUD M			(0.77)	(0.88)	(1.09)	(0.86)	(1.23)	(0.31)	(3.60)
Processing Costs	AUD M			(1.30)	(6.67)	(6.62)	(0.17)	(8.24)	(2.88)	(25.88)
Transport Costs	AUD M			(0.51)	(2.19)	(2.00)	(0.10)	(2.31)	(2.03)	(9.14)
Treatment & Refining Costs	AUD M			(0.30)	(1.26)	(1.15)	(0.06)	(1.33)	(1.17)	(5.27)
G & A Costs	AUD M			(0.90)	(4.59)	(4.56)	(0.11)	(5.68)	(1.98)	(10.16)
Marketing and Assays Costs	AUD M			(0.02)	(0.07)	(0.07)	(0.00)	(0.08)	(0.07)	(0.16)
Total Operating Costs	AUD M			(7.85)	(20.28)	(21.23)	(5.86)	(25.37)	(10.06)	(73.20)
Pre-Strip Capital	AUD M	(6.23)	(6.32)	(3.70)						**(16.25)**

	Units	Yr 1 Qtr 1	Yr 1 Qtr 2	Yr 1 Qtr 3	Yr 1 Qtr 4	Yr 2 Qtr 1	Yr 2 Qtr 2	Yr 2 Qtr 3	Yr 2 Qtr 4	Total
Sustaining Capital	AUD M	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	**(8.00)**
Mobilisation	AUD M	(3.02)								**(3.02)**
Demobilisation	AUD M								(0.67)	**(0.67)**
Grade Control Drilling	AUD M	(0.36)	(0.36)	(0.36)						**(1.08)**
Total Capital Costs	AUD M	**(10.61)**	**(7.68)**	**(5.06)**	**(1.00)**	**(1.00)**	**(1.00)**	**(1.00)**	**(1.67)**	**(29.02)**
Royalties	AUD M			(0.61)	(1.71)	(1.58)	(0.07)	(1.81)	(1.54)	**(7.33)**
Revenue - Copper	AUD M			8.63	25.10	22.95	1.11	26.54	23.37	**107.7**
Revenue - Gold	AUD M			3.80	9.55	8.92	0.35	10.17	7.75	**40.5**
Total Revenue	AUD M			**12.43**	**34.65**	**31.87**	**1.45**	**36.71**	**31.12**	**148.2**
Undiscounted Cashflow (EBIT)	AUD M	**(10.61)**	**(7.68)**	**(1.09)**	**11.66**	**8.06**	**(5.48)**	**8.53**	**17.85**	**21.23**

The project shows a positive cashflow of AUD21.2 M and produces 62,648 t of copper concentrate and 28,950 ozs of gold. Figure 22-1 shows the capital and operating cost expenditure profile.

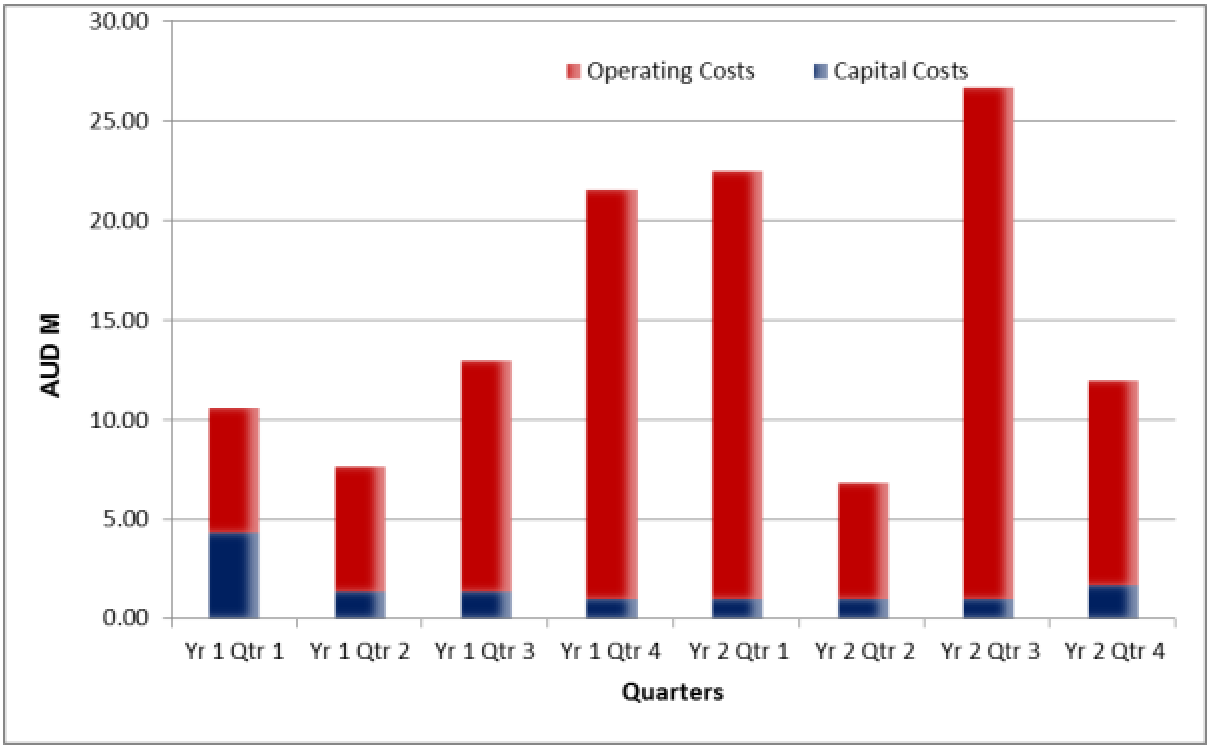


Figure 22-1: Capex and Opex Expenditure by Quarter for Osborne Open Pit Mine

22.5 Osborne Underground

The key metrics are summarised in Table 22-5. The output from the financial model (Osborne Underground Cost Model_Rev1.xls) is shown in Table 22-6. No NPV has been calculated for the Osborne Underground Mine given the remaining mine life is less than one year. The IRR has not been calculated because there is no negative cashflow and the payback period is not relevant to the remaining project life.

Table 22-5: Summary of Key Financial Parameters

Parameter	Units	Value
Tonnes Processed	t	491,304
Total OPEX	AUD M	32.10
Total CAPEX	AUD	0
Royalty	AUD M	3.35
Total Cost	AUD M	35.45
Copper Produced	Mlb	16.58
Gold Produced	ozs	9,779
Total Revenue	AUD M	67.59
Cashflow	AUD M	32.13

Table 22-6: Osborne Underground Cost Model

	Units	Jun-12	Jul-12	Aug-12	Sep-12	Oct-12	Nov-12	Dec-12	Jan-13	Feb-13	Mar-13	Total
Physicals												
Development Metres												
Decline	m											
Waste Development	m											
Ore Development	m	197	5									**202**
Total Lateral Development	m	197	5									**202**
4.0 m Diameter Raise	m											
1.8 m Diameter Raise	m											
Total Vertical Development	m											
Tonnes and Grade												
Development Ore Tonnes	t	20,778	448									**21,225**
Development Copper Grade	%	1.40	0.91		0.00	0.00	0.00	0.00	0.00	0.00	0.00	**2.30**
Development Gold Grade	g/t	0.83	0.75	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**1.58**
LHOS Ore Tonnes	t	53,710	58,012	70,525	50,937	60,618	44,987	42,315	42,315	38,220	8,440	**470,079**
LHOS Copper Grade	%	1.48	1.86	2.01	2.09	1.75	2.12	1.79	1.46	1.48	1.48	**17.51**
LHOS Gold Grade	g/t	0.74	0.84	0.88	0.88	0.85	0.89	0.83	0.73	0.73	0.73	**8.09**
Total Ore Tonnes	t	74,487	58,459	70,525	50,937	60,618	44,987	42,315	42,315	38,220	8,440	**491,304**
Total Copper Grade	%	1.45	1.85	2.01	2.09	1.75	2.12	1.79	1.46	1.48	1.48	**1.78**
Total Gold Grade	g/t	0.76	0.84	0.88	0.88	0.85	0.89	0.83	0.73	0.73	0.73	**0.82**
Contained Copper	t	1,083	1,081	1,420	1,065	1,058	952	759	619	566	125	**8,728**
Contained Gold	ozs	1,828	1,586	1,997	1,437	1,652	1,291	1,125	999	892	197	**13,004**
Copper Concentrate	t	3,976	3,971	5,216	3,911	3,886	3,494	2,786	2,274	2,077	459	**32,050**
Copper Produced	lb	2,103,390	2,100,326	2,759,221	2,068,759	2,055,323	1,848,323	1,473,766	1,202,937	1,098,666	242,622	**16,953,333**
Gold Produced	oz	1,358	1,177	1,483	1,067	1,227	958	836	742	662	146	**9,657**

	Units	Jun-12	Jul-12	Aug-12	Sep-12	Oct-12	Nov-12	Dec-12	Jan-13	Feb-13	Mar-13	Total
Costs												
Operating												
Ore Development Costs	AUD M	(1.34)	(0.03)	-	-	-	-	-	-	-	-	**(1.37)**
Drill and Blast Costs	AUD M	(0.35)	(0.38)	(0.46)	(0.33)	(0.39)	(0.29)	(0.27)	(0.27)	(0.25)	(0.05)	**(3.05)**
Load and Back fill costs	AUD M	(0.17)	(0.18)	(0.22)	(0.16)	(0.19)	(0.14)	(0.13)	(0.13)	(0.12)	(0.03)	**(1.49)**
Haulage Costs	AUD M	(0.42)	(0.45)	(0.55)	(0.40)	(0.47)	(0.35)	(0.33)	(0.33)	(0.30)	(0.07)	**(3.68)**
Other Production Costs	AUD M	(0.64)	(0.69)	(0.84)	(0.60)	(0.72)	(0.53)	(0.50)	(0.50)	(0.45)	(0.10)	**(5.57)**
Grade Control	AUD M	(0.09)	(0.07)	(0.09)	(0.06)	(0.08)	(0.06)	(0.05)	(0.05)	(0.05)	(0.01)	**(0.62)**
Processing Costs	AUD M	(0.79)	(0.62)	(0.75)	(0.54)	(0.64)	(0.48)	(0.45)	(0.45)	(0.41)	(0.09)	**(5.21)**
Transport Costs	AUD M	(0.58)	(0.58)	(0.76)	(0.57)	(0.57)	(0.51)	(0.41)	(0.33)	(0.30)	(0.07)	**(4.69)**
Treatment & Refining Costs	AUD M	(0.33)	(0.33)	(0.44)	(0.33)	(0.33)	(0.29)	(0.23)	(0.19)	(0.17)	(0.04)	**(2.69)**
G & A Costs	AUD M	(0.54)	(0.43)	(0.51)	(0.37)	(0.44)	(0.33)	(0.31)	(0.31)	(0.28)	(0.06)	**(3.59)**
Marketing and Assays Costs	AUD M	(0.02)	(0.02)	(0.03)	(0.02)	(0.02)	(0.02)	(0.01)	(0.01)	(0.01)	(0.00)	**(0.16)**
Total Operating Costs	AUD M	**(5.27)**	**(3.79)**	**(4.65)**	**(3.39)**	**(3.85)**	**(3.00)**	**(2.71)**	**(2.59)**	**(2.34)**	**(0.52)**	**(32.10)**
Royalties	AUD M	(0.43)	(0.41)	(0.54)	(0.40)	(0.41)	(0.36)	(0.29)	(0.24)	(0.22)	(0.05)	**(3.35)**
Revenue												
Revenue - Copper	AUD M	6.69	6.68	8.77	6.58	6.53	5.88	4.69	3.82	3.49	0.77	**53.90**
Revenue - Gold	AUD M	1.92	1.67	2.10	1.51	1.74	1.36	1.18	1.05	0.94	0.21	**13.69**
Total Revenue	AUD M	**8.61**	**8.35**	**10.87**	**8.09**	**8.27**	**7.24**	**5.87**	**4.88**	**4.43**	**0.98**	**67.59**
Undiscounted Cashflow (EBIT)	AUD M	2.91	4.15	5.69	4.30	4.01	3.87	2.87	2.05	1.87	0.41	**32.13**
Cumulative Undiscounted Cashflow (EBIT)	AUD M	**2.91**	**7.06**	**12.74**	**17.04**	**21.05**	**24.93**	**27.80**	**29.84**	**31.71**	**32.13**	

The project shows a positive cashflow of AUD32.13 M and produces 31,354 t of copper concentrate and 9,779 ozs of gold. Figure 22-2 shows the capital and operating cost expenditure profile.

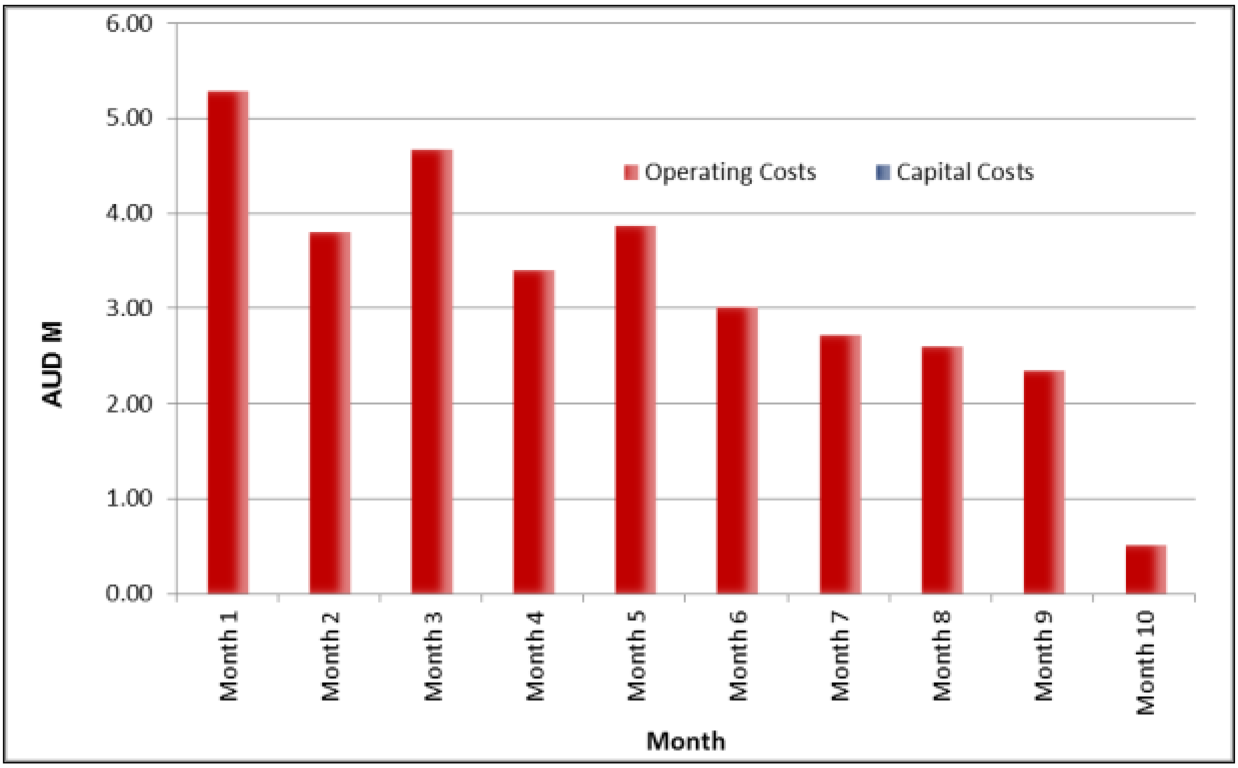


Figure 22-2: Capex and Opex Expenditure by month for Osborne Underground Mine

22.6 Kulthor Underground

The key metrics are summarised in Table 22-7. The output from the financial model (Kulthor Cost Model_Rev1.xls) is shown in Table 22-8.

Table 22-7: Summary of Key Financial Parameters – Kulthor Underground

Parameter	Units	Value
Tonnes Milled	t	2,575,058
Total OPEX	AUD M	172.76
Total CAPEX	AUD M	43.57
Royalty	AUD M	13.82
Total Cost	AUD M	230.14
Copper Produced	Mlb	63.83
Gold Produced	ozs	55,116
Total Revenue	AUD M	284.60
Cashflow	AUD M	54.46
Discounted Cashflow (EBIT) (8.6%)	AUD M	38.43
IRR	%	71
Payback Period	year	1.8

Table 22-8: Kulthor Underground Cost Model

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Physicals								
Development Metres								
Decline	m	631	438	447	324			**1,840**
Waste Development	m	568	1,202	798	839	10	40	**3,456**
Ore Development	m	1,569	1,230	976	1,093	116	93	**5,075**
Total Lateral Development	m	2,768	2,870	2,220	2,255	126	133	**10,371**
Vertical 1.8m dia	m		108	49	63			**221**
Vertical 4 x 4m raise	m	14	67	63	65			**209**
Total Vertical Development	m	14	175	112	128			**429**
Development Ore Tonnes	t	109,325	99,608	73,123	92,268	9,937		**384,261**
Development Copper Grade	%	1.45	1.48	1.53	1.55	1.16		**1.49**
Development Gold Grade	g/t	0.95	0.92	0.85	1.06	0.68		**0.94**
Tonnes and Grade								
LHOS Ore Tonnes	t	14,723	291,311	320,503	569,694	438,988	164,931	**1,800,150**
LHOS Copper Grade	%	1.01	1.31	1.4	1.46	1.49	1.85	**1.46**
LHOS Gold Grade	g/t	0.57	0.89	0.87	1.02	0.99	1.03	**0.96**
LHBF Ore Tonnes	t		67,282	290,578	32,787			**390,647**
LHBF Copper Grade	%		1.42	1.49	1.27			**1.46**
LHBF Gold Grade	g/t		0.90	0.86	0.82			**0.86**
Total Ore Tonnes	t	124,048	458,201	684,204	694,749	448,925	164,931	**2,575,058**
Total Copper Grade	%	1.40	1.36	1.45	1.46	1.48	1.85	**1.47**
Total Gold Grade	g/t	0.90	0.90	0.86	1.02	0.98	1.03	**0.94**
Contained Copper	t	1,734	6,246	9,935	10,164	6,656	3,051	**37,787**

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Contained Gold	ozs	3,609	13,229	18,998	22,691	14,190	5,462	**78,178**
Copper Concentrate	t	5,537	19,945	31,727	32,457	21,255	9,744	**120,665**
Copper Produced	lb	2,928,835	10,550,027	16,782,398	17,168,613	11,243,275	5,153,961	**63,827,108**
Gold Produced	oz	2,544	9,326	13,393	15,997	10,004	3,851	**55,116**
Costs								
Operating								
Ore Development Costs	AUD M	(10.64)	(8.34)	(6.62)	(7.41)	(0.79)	(0.63)	**(34.41)**
Drill and Blast Costs	AUD M	(0.10)	(2.32)	(3.96)	(3.90)	(2.84)	(1.07)	**(14.20)**
Load and Back fill costs	AUD M	(0.06)	(1.49)	(2.54)	(2.51)	(1.83)	(0.69)	**(9.11)**
Haulage Costs	AUD M	(0.07)	(1.74)	(2.96)	(2.92)	(2.12)	(0.80)	**(10.60)**
Other Production Costs	AUD M	(0.17)	(4.25)	(7.25)	(7.15)	(5.21)	(1.96)	**(25.98)**
Grade Control Costs	AUD M	(0.16)	(0.58)	(0.87)	(0.88)	(0.57)	(0.21)	**(3.27)**
Processing Costs	AUD M	(1.31)	(4.86)	(7.25)	(7.36)	(4.76)	(1.75)	**(27.30)**
Transport Costs	AUD M	(0.83)	(2.99)	(4.75)	(4.86)	(3.18)	(1.46)	**(18.06)**
Treatment & Refining Costs	AUD M	(0.48)	(1.72)	(2.73)	(2.81)	(1.84)	(0.84)	**(10.42)**
G & A Costs	AUD M	(0.91)	(3.34)	(4.99)	(5.07)	(3.28)	(1.20)	**(18.80)**
Marketing and Assays Costs	AUD M	(0.03)	(0.10)	(0.16)	(0.16)	(0.11)	(0.05)	**(0.60)**
Total Operating Costs	AUD M	**(14.75)**	**(31.73)**	**(44.08)**	**(45.03)**	**(26.52)**	**(10.64)**	**(172.76)**
Capital								
Infrastructure	AUD M	(2.08)						**(2.08)**
Mine Development	AUD M	(8.14)	(11.62)	(8.70)	(8.20)	(0.07)	(0.27)	**(36.99)**
Processing	AUD M							
Sustaining Capital	AUD M	(0.75)	(0.75)	(0.75)	(0.75)	(0.75)	(0.75)	**(4.50)**
Total Capital Costs	AUD M	(10.96)	(12.37)	(9.45)	(8.95)	(0.82)	(1.02)	**(43.57)**

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Royalties	AUD M	(0.63)	(2.30)	(3.56)	(3.80)	(2.45)	(1.07)	**(13.82)**
Revenue - Copper	AUD M	9.52	34.29	54.54	55.80	36.54	16.75	**207.44**
Revenue - Gold	AUD M	3.56	13.06	18.75	22.40	14.01	5.39	**77.16**
Total Revenue	AUD M	**13.08**	**47.34**	**73.29**	**78.19**	**50.55**	**22.14**	**284.60**
Undiscounted Cashflow (EBIT)	AUD M	(13.27)	0.94	16.21	20.42	20.76	9.40	**54.46**
Discounted Cashflow (EBIT)	AUD M	(13.27)	0.87	13.74	15.94	14.92	6.22	**38.43**
Cumulative Discounted Cashflow (EBIT)	AUD M	(13.27)	(12.40)	1.34	17.28	32.20	38.43	**38.43**

The project shows a positive cashflow of AUD38.4 M and produces 120,665 t of copper concentrate and 55,116 ozs of gold. Figure 22-3 shows the capital and operating cost expenditure profile.

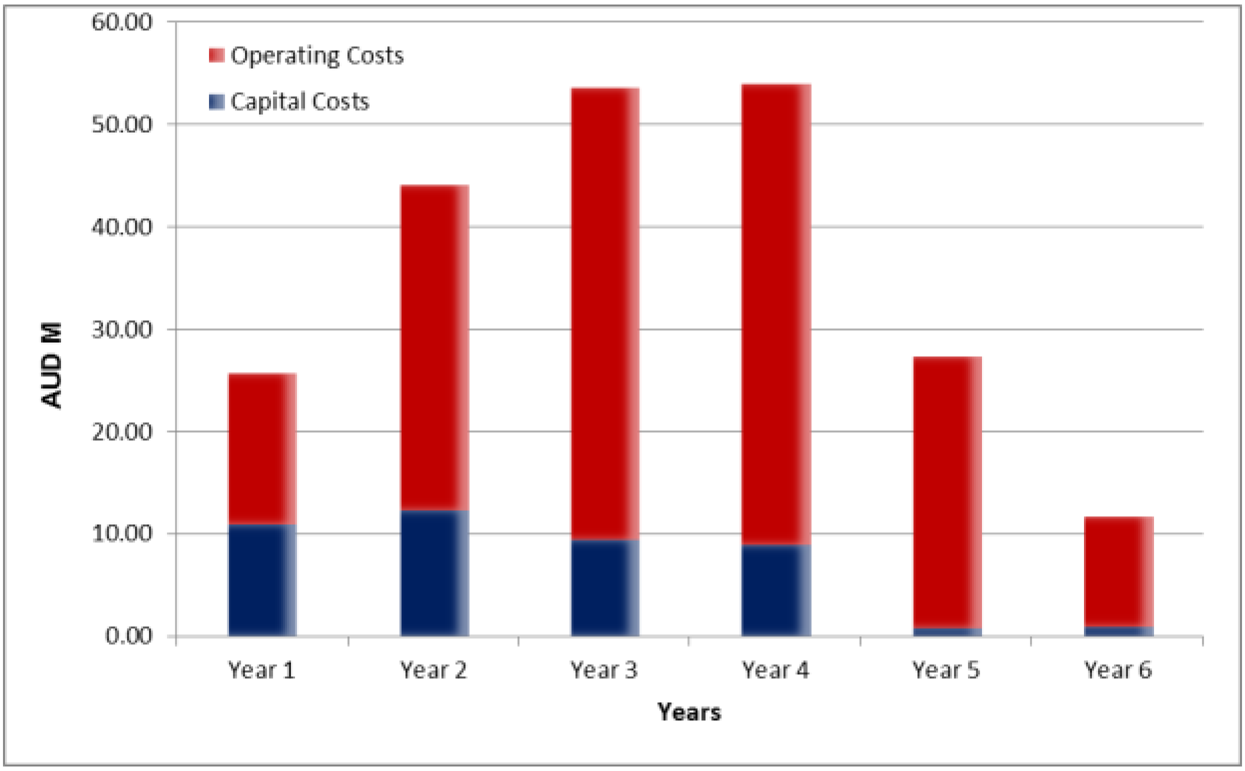


Figure 22-3: Capex and Opex Expenditure by month for Kulthor Underground Mine

Several sensitivities were analysed for the Kulthor underground cashflow model. The sensitivities were applied at ± 10% to determine which changes have the highest impact on the project. Table 22-9 and Figure 22-4 shows the results from the sensitivity analysis. Commodity prices and orebody grade have largest impact on the project financial results.

Table 22-9: Sensitivity Results

Item	NPV			ΔNPV	
	-10%	Base	+10%	-ve	+ve
Commodity prices	16.46	38.43	60.39	-21.97	21.96
Grade	24.75	38.43	52.11	-13.68	13.68
Operating costs	52.69	38.43	24.17	14.26	-14.26
Capital costs	42.29	38.43	34.57	3.86	-3.86
Metal Recovery	18.82	38.43	58.03	-19.61	19.60

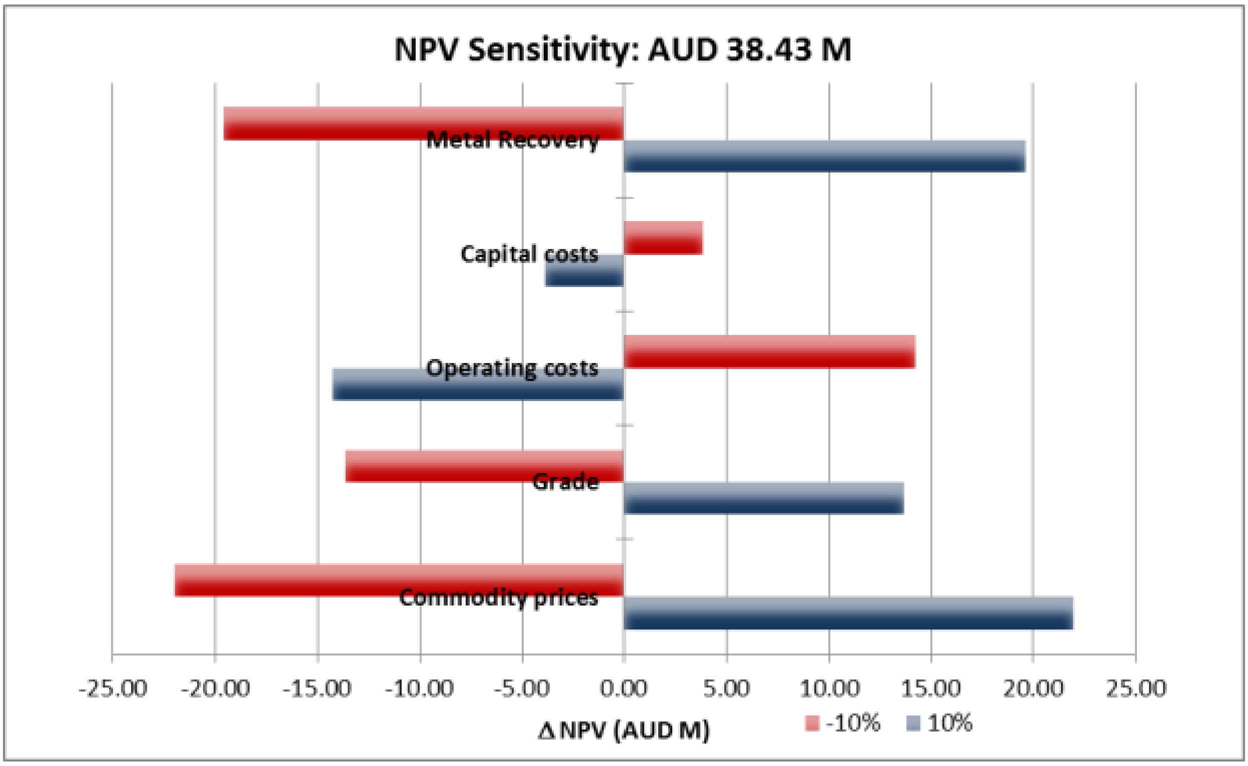


Figure 22-4: Impact of Sensitivities on Project

23 Adjacent Properties

The Osborne and Kulthor deposits are part of larger system as described in Section 7.

There are a number of projects and Resources, owned by Ivanhoe within the Cloncurry district in proximity to those discussed in this report. The Merlin, Starra Line, Mt Dore and Lady Ella and Mount Elliott deposits are under evaluation by Ivanhoe. While these deposits may not be directly related to or influenced by each other in terms of the mineralisation, the proximity of the deposits presents an opportunity to share infrastructure. The Osborne Mill and Treatment Plant (currently in operation) may add value to smaller deposits that would otherwise be uneconomic. Inclusion of such deposits could significantly extend the project life beyond the period considered within this report.

24 Other Relevant Data and Information

SRK and LMRC consider that all data and information relevant to the Osborne and Kulthor deposits has been disclosed by Ivanhoe and discussed appropriately in this Technical Report.

25 Interpretation and Conclusions

25.1 Interpretations

From the sensitivity analysis, the Kulthor Underground deposit has been shown to be sensitive to commodity price and metallurgical recovery. Any variance to these items would have a significant impact, positively or negatively, to the overall financial performance of the project as shown in Figure 22-4.

The Osborne Open Pit, Osborne Underground and Kulthor deposits are part of an overall mining strategy for the Osborne copper-gold project to provide mill feed to the Osborne processing plant. Mining of the deposits contribute to the overall mill feed and ensures that the processing plant is utilised to capacity. If the processing plant is not fully utilised, this has an impact on the operating costs of the project and potentially makes the remaining deposits uneconomical to mine.

The existing surface infrastructure, on care and maintenance, will operate at below its historical capacity. This reduces the project risk to inefficiencies and provides potential for an increase in throughput without significant injections of capital costs.

Cost estimates for the concentrate handling and power generation are based on current base cases. Ivanhoe are currently engaging in discussions that have the potential to reduce both operating and capital costs.

25.2 Conclusions

The technical and financial aspects for each of the deposits in the Osborne copper-gold project have been shown to be robust at this level of study. The Mineral Reserve based on a NI 43-101 compliant Mineral Resource estimate is at a pre-feasibility study level of detail and supports the reporting of Mineral Reserves.

26 Comments and Recommendations

26.1 Comments

26.1.1 Osborne Open Pit

The Open Pit Mineral Resources are well drilled already. The current Open Pit design is considered conservative as it avoids interaction with previously-mined stopes and underground development. There is potential for a considerable increase in Mineral Resources, depending on mining and economic constraints.

LMRC considers that the blocks located within the conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a Mineral Resource.

26.1.2 Osborne Underground Mine

The Mineral Resources included within the Mineral Reserve are well defined. Establishment of a Mineral Resource definition drilling programme has the potential to convert Inferred Mineral Resources at depth thereby extending the known Mineral Resource.

The grade of the Mineral Resources decreases with depth, so it will be important to increase the amount of drilling in the lower parts of the 1SS Zone.

26.1.3 Kulthor Underground Mine

The Mineral Resource definition drilling programme should be continued to increase the proportion of Measured Mineral Resources and convert the Inferred Mineral Resources to Measured and Indicated Mineral Resources.

Reconciliation and assessment of mined stopes should be undertaken to understand the impact of the shear zone on the stope performance and for revision of the modifying factors.

The current Osborne practice of using short stab drillholes to further define the limit of mineralisation for stoping should continue. The boundaries of the Kulthor mineralisation are not always sharp.

Surface drilling has shown the presence of new mineralisation along strike to the west of the “M” zone, and also to the south. Additional drilling is required to move this mineralisation into Measured and Indicated Mineral Resources. This drilling has not been costed.

26.2 Recommendations

There has not been a work programme recommended because the deposits have been incorporated into the Osborne Mine Development Plan.

An important component of the next phase is to develop an improved understanding of the geo-metallurgical properties and variability between and within each of the deposits through specific testwork. Understanding each of the deposits and how they perform metallurgically in a blend is important in the process of optimising the production profile.

The Kulthor Mineral Reserves in this Technical Report are based on the 2011 Kulthor Mineral Resource Model. The Kulthor Mineral Reserves should be re-estimated with the Kulthor Mineral Resource reported in the Technical Report.

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Appendices

Appendix A: Mineral Tenure Information

PCF Capital Group

AFS Licence No 291385

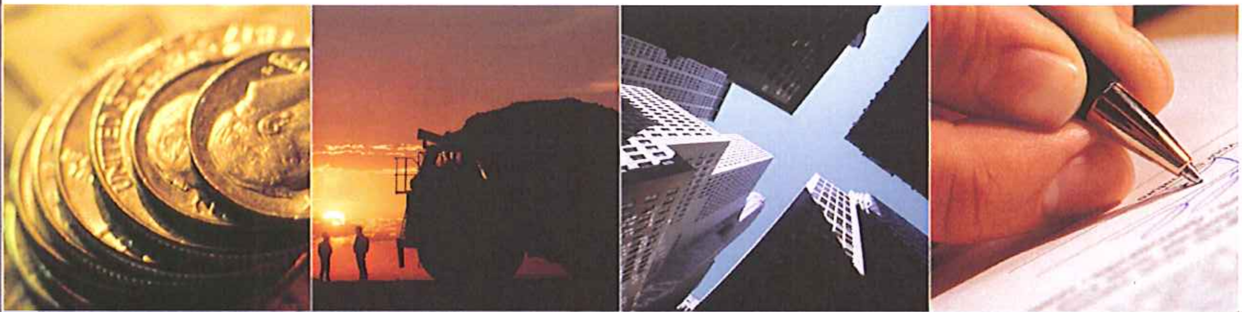
PRIVATE AND CONFIDENTIAL

BARRICK
AUSTRALIA PACIFIC

Barrick (Australia Pacific) Limited

Osborne Copper Gold Project
Information Memorandum

16 June 2009



INVESTMENT BANKERS

AND CORPORATE ADVISERS

4 TENEMENTS

The Osborne, Kulthor and Trekelano Mining Leases are 100% owned by Barrick (Osborne) Pty Ltd. Exploration leases are owned entirely by Osborne, apart from the Duchess JV lease in which the company holds a 70% share. The project covers an approximate area of $586km^2$ and comprises 4 Exploration Permits ("EPM") and 8 Mining Leases ("ML", one under application) as summarised in Table 4 and shown in Figure 8.

Project	Lease	Lease Name	Comment
Osborne Mine	EPM15281	Felece Tank	
Mayfield Option	EPM14111	Mayfield	Option to Purchase from Newcrest Operations Ltd
Duchess JV	EPM9083	Duchess	Have met expenditure commitment to earn 70%
Osborne Mine	EPM9624	Tough Tank	
Osborne Mine	ML90040	Osborne	
Osborne Mine	ML90057	Osborne Borefield & Services	
Osborne Mine	ML90068	Osborne Concentrate Loading	
Osborne Mine	ML90125	Trekelano	
Osborne Mine	ML90128	Trikilana	
Osborne Mine	ML90158	Kulthor	
Osborne Mine	ML90183	Trekelano Extended	
Osborne Mine	ML90187	Lucky Luke	Application

Table 4 Osborne Copper and Gold Project Tenement Listing

The Osborne operation is located on ML 90040, with the underlying tenure being a long term pastoral lease (Chatsworth Station). The access road to the Osborne operation, beyond the boundary of ML90040, is located on the Chatsworth Station and is subject to a specific landowner agreement. The Osborne bore field and services corridor are located on ML90057 and the rail load-out facility is located on ML90068.

The Trekelano operation is located on ML90125 and ML90128. The underlying land tenure for the Trekelano operation is covered by long term pastoral leases (Mayfield Station) and unallocated State Land (formerly the Trekelano Town Reserve). Part of a small (i.e. approximately 2 hectare) historical State School Reserve is located within ML90125.

The tenement package has current annual rents of $237,917, and expenditure commitments in the order of $1,000,000. For further details, refer to the tenement schedule in Appendix 2.

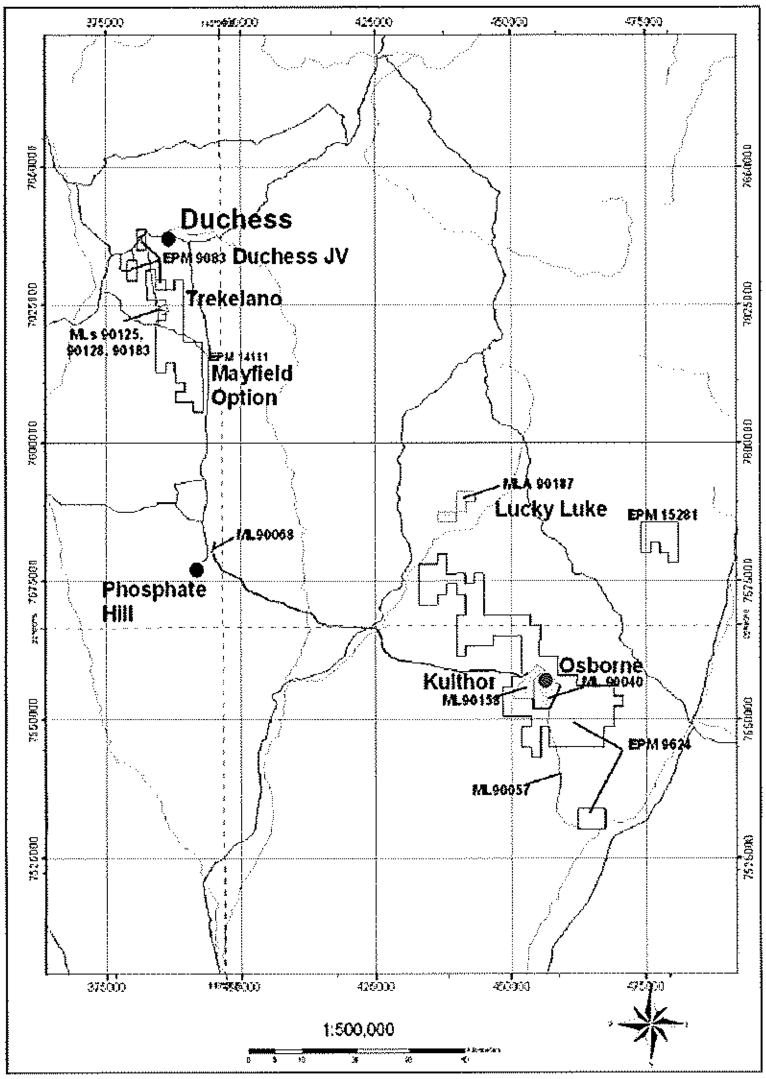


Figure 8 Tenement Location

Duchess JV

Goldminco Corporation, through its wholly owned subsidiary, Goldminco Resources Limited, entered into a farm-in agreement with Barrick (Osborne) Limited (formerly Placer Pacific (Osborne) Pty Limited) on the 10 November 2005 whereby Barrick (Osborne) Limited could spend $1.5M over three years to earn 70% of a EPM 9083. In October 2008, Barrick (Osborne) Limited completed the expenditure requirements and acquired a 70% interest in the Duchess JV. A formal JV agreement is currently being prepared by Barrick Corporate and Legal.

Mayfield Option

In July 2008, Barrick (Osborne) Pty Limited was granted an option by Newcrest Operations Limited to acquire 100% of the Mayfield tenement, EPM 14111. Consideration of payment for the tenement, and hence exercise of the option is:

a) $10,000 payable within 10 business days after the commencement date;

b) $90,000 at any time on or before the first anniversary of the commencement date;

c) $100,000 at any time on or before the second anniversary of the commencement date: and

d) $1,300,000 at any time on or before the third anniversary of the commencement date.

Upon commencement of mining within EPM 14111, Barrick (Osborne) Pty Limited must pay a 2% Net Smelter Return on minerals mined to Newcrest Operations Limited.

4.1 ROYALTIES

The Project is subject to Queensland State Government royalties on the payable metal quantity after a royalty free threshold of $100,000. Companies can elect for each mining project, for a five year period, between a fixed (2.7%) or variable (1.5% - 4.5%) ad valorem rate royalty, with the latter dependent on London market metal prices.

The Osborne Copper Gold Project is currently on the fixed rate, expiring at end 2010. The fixed rate will be eliminated at end 2010, and the variable rate revised from 1.5%-4.5% to 2.5% -5%, with effect from 1 January 2011.

All royalty payable metal quantities are determined by the formula:

RPMQ = D x A

Where – RPMQ = royalty payable metal quantity;

D = dry metric tonnes of concentrate invoiced by the producer; and

A = the payable metal content for royalty purposes. The determination of "A" for more common concentrate combinations is detailed in Table 5.

Concentrate	Prescribed Mineral	Deduction from Assay	% applied after deduction	% of Assay	Whichever is lower
Copper	Copper	1%		96.5%	Yes
	Silver	30 grams	90%		
	Gold	1 gram	90%		

Table 5 Determination of payable metal content for royalty purposes

The Trekelano operation and the Kulthor deposit are both subject to payments to the Yulluna native title claimants relating to production. The payment terms are contained within comprehensive Ancillary Agreements attached to the Deeds relating to the grant of Mining Leases. Copies of these Deeds and the pertinent Ancillary Agreements detailing the payments and other benefits attributable to the Yulluna People require consent to be disclosed and will be made available at the appropriate time prior to submission of final binding bids.

In relation to the Lucky Luke Project in January 2008 Ivanhoe assigned their rights to the application of a mining lease over all or part of EPM10783 (owned by Ivanhoe) to Barrick for a 5% NSR royalty on production after 10 million tonnes of ore from the mining lease application area has been processed at the Osborne mill or at any equivalent processing plant.

APPENDIX 2 - TENEMENT SCHEDULE

Project	Lease	Lease Name	Lease Status	Current Area	Area Type	Application Date	Grant Date	Expiry Date	Rent	Commitment	Managing Company	Operator
Osborne Mine	EPM15281	Felece Tank	Granted	12.0	Blocks	25/10/2005	31/10/2006	30/10/2011	$3,115.00	$150,000.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Mayfield Option	EPM14111	Mayfield	Granted	41.0	Blocks		9/08/2005	8/08/2008	$5,108.60	$200,000.00	Newcrest Operations Ltd	Barrick (Osborne) Pty Ltd
Duchess JV	EPM9083	Duchess	Granted	5.0	Blocks				$0.00	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	EPM9624	Trough Tank	Granted	130.0	Blocks	30/07/1993	10/11/1993	9/11/2010	$19,063.80	$500,000.00	Barrick (PD) Australia Limited	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90040	Osborne	Granted	2152.0	Hectares	11/11/1993	9/06/1994	30/06/2014	$103,511.20	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90057	Osborne Borefield & Services	Granted	64.4	Hectares	9/06/1994	15/12/1994	31/12/2014	$3,126.50	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90068	Osborne Concentrate Loading	Granted	18.0	Hectares	3/02/1995	29/06/1995	30/06/2014	$865.80	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90125	Trekelano	Granted	180.1	Hectares	26/02/1997	20/10/2005	31/10/2025	$8,706.10	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90128	Trikilana	Granted	181.7	Hectares	10/06/1997	20/10/2005	31/10/2025	$8,754.20	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90158	Kulthor	Granted	1770.9	Hectares	14/05/2004	28/06/2007	30/06/2027	$85,185.10	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90183	Trekelano Extended	Granted	9.6	Hectares	6/10/2006	1/05/2008	30/04/2018	$481.00	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Osborne Mine	ML90187	Lucky Luke	Application	1572.6	Hectares	29/01/2008			$0.00	$0.00	Barrick (Osborne) Pty Ltd	Barrick (Osborne) Pty Ltd
Total									**$237,917.30**	**$850,000**		