MMG Mineral Resources and Ore Reserves Statement

as at 30 June 2013

Technical Appendix

13 December 2013

TABLE OF CONTENTS

APPROVALS PAGE

This signature was scanned for the exclusive use in the MMG Mineral Resource and Ore Reserve Statement as at 30 June 2013 with the author's approval. Any other use is not authorised.	Gustavo Gomes	GM Technical Services	27/11/13
Signature	**Name**	**Position**	**Date**
This signature was scanned for the exclusive use in the MMG Mineral Resource and Ore Reserve Statement as at 30 June 2013 with the author's approval. Any other use is not authorised.	Jared Broome	Group Manager Geology	27/11/13
Signature	**Name**	**Position**	**Date**
This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 13* with the author's approval. Any other use is not authorised.	Julian Poniewierski	Group Manager - Technical Governance	27/11/13
Signature	**Name**	**Position**	**Date**

The above signed endorse and approve this Mineral Resource & Ore Reserves Statement Executive Summary.

1. INTRODUCTION

On 20th December 2012 an updated JORC[1] Code was released – the previous release being the 2004 Edition. The JORC Code 2012 Edition defines the requirements for public reporting of Exploration Results, Mineral Resources and Ore Reserves by mining companies. Reporting according to the JORC Code is a requirement of the MMG listing on The Stock Exchange of Hong Kong[2] as per amendments to Chapter 18 of the Listing Rules that were announced on 3rd June 2010.

The core of the changes to JORC Code is enhanced disclosure of the material information prepared by the Competent Person with the requirement for the addition of a publicly released detailed Appendix to the Mineral Resource and Ore Reserves release document, which outlines the supporting details to the Mineral Resource and Ore Reserves numbers.

This Technical Appendix provides these supporting details.

Under the JORC Code, reporting in compliance with the guidelines of JORC Code 2012 Edition becomes compulsory from 1 Dec 2013.

The principles governing the operation and application of the JORC Code are Transparency, Materiality and Competence:

- Transparency requires that the reader of a Public Report is provided with sufficient information, the presentation of which is clear and unambiguous, to understand the report and not be misled by this information or by omission of material information that is known to the Competent Person.
- Materiality requires that a Public Report contains all the relevant information that investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgement regarding the Exploration Results, Mineral Resources or Ore Reserves being reported. Where relevant information is not supplied an explanation must be provided to justify its exclusion.
- Competence requires that the Public Report be based on work that is the responsibility of suitably qualified and experienced persons who are subject to an enforceable professional code of ethics (the Competent Person).

---

[1] JORC = Joint Ore Reserves Committee.

[2] Specifically, the Updated Rules of Chapter 18 of the Hong Kong Stock Exchange Listing Rules require a Competent Person's report to comply with standards acceptable to the HKSE including JORC Code (the Australian code), NI 43-101 (the Canadian code) and SAMREC Code (the South African code) for Mineral Resources and Ore Reserves. MMG Limited has chosen to report using the JORC Code.

2. COMMON TO ALL SITES

2.1 Revenue Factors (Price Assumptions)

The price environment assumptions used for 2013 Mineral Resource and Ore Reserves estimation at the date at which work commenced on the Mineral Resources and Ore Reserves are as shown in Table 1.

Table 1 **Price (real) and foreign exchange assumptions**

	CY14	CY15	CY16	Long Term
Zn $/lb	0.89	0.97	1.08	1.18
Cu $/lb	3.50	3.16	3.05	2.80
Pb $/lb	1.06	1.06	1.09	1.12
Au $/oz	1,525	1,318	1,258	1,200
Ag $/oz	27.34	23.79	18.79	20.94
A$:US$	0.99	0.95	0.92	0.84
CAD:US$	0.98	0.95	0.93	0.90
US$:LAK	8,000	8,000	8,000	8,000

Mineral Resource work used long-term pricing only, with cut-off grades or cut-off values generally applied at no less than 70% of the grades or values used in determination of the Ore Reserves.

For the Ore Reserves work, Prices and Exchange Rates were used as follows:

(i) For Long-Term (Life-of-Asset) Ore Reserves planning (> 3 years), the "Long-Term" price and exchange rate values were used.

(ii) For medium-term (< 3 years) the average of the price and exchange rate combination of the CY14-CY16 three years where price forecast is declining (Cu/Au/Ag), and first year price and exchange rate where price forecast is increasing (Zn/Pb).

(iii) For ultra-short term planning, where it is definitely known that the Ore Reserves will be mined out and completed in CY14, the sites used CY14 price/exchange assumptions.

For long-term mines (or planning horizons of > 3 years) those mining Zinc and/or Lead where the price is rising in the long-term, had to consider the medium term case and use the CY14 price and exchange rate for material scheduled to be mined in the next three years. If in this process, Mineral Resource was sterilised for long-term mining, it was not included in the Ore Reserves even if economic at the long-term price assumptions.

2.2 Metal Market Analysis – Basis for Pricing Assumptions

The pricing assumptions used for evaluation of Mineral Resources and Ore Reserves were based on an evaluation of broker consensus at the time.

2.2.1 Market Assessment – The Global Demand for Metals

The outlook for growth in the metals and mining industry on a global scale remains positive. While demand for metals has been affected by worsening economic conditions in the United States and Europe, this has been offset by the strong demand that flows from the expansion of developing economies which are driven by domestic demand. Growth in domestic demand in most emerging economies is projected to continue. However, this growth will be more moderate than previous projections, with world annual GDP growth expected to be 2.5% to 3%, of which advanced economies will account for 1.5% to 2% and emerging economies for 5.5% to 6%.

The growth in demand from emerging economies is expected to drive demand for all basic commodities.

High barriers to entry exist in the mining industry due to the high capital costs of establishing or acquiring operations, heavy market regulation of this sector in many countries and long lead times to production. In recent times, there have been only a small number of discoveries of significant deposits of high grade copper and zinc.

2.2.2 Zinc Demand and Supply

The Company takes a long term view of zinc market fundamentals and while current prices are below long term averages, we believe over the long term there will be increasing tightness in the market driven by:

Zinc Supply

Expected mine closures will remove 1.8Mt pa of zinc from the existing market - refer Table 2.

There are few committed greenfield or brownfield developments expected to commence operations in the short term. Few options to acquire assets or advanced exploration targets which are meaningful in scale and quality.

Supply will continue to be tight with historically low investment in exploration resulting in a thin development pipeline with declining quality. The market is forecast to enter a supply deficit from 2014.

Table 2 Upcoming zinc mine closures

Mine Closures (TOP 5)	Operator	Location	2012 Production	Expected closure
Century	MMG	Australia	515kt	2015
Brunswick	Xstrata	Canada	219kt	Closed 2013
Lisheen	Vedanta	Ireland	180kt	2015
Skorpion	Vedanta	Namibia	159kt	2017
Perseverance	Xstrata	Canada	128kt	Closed 2013
Total			1,201kt	

Zinc Demand

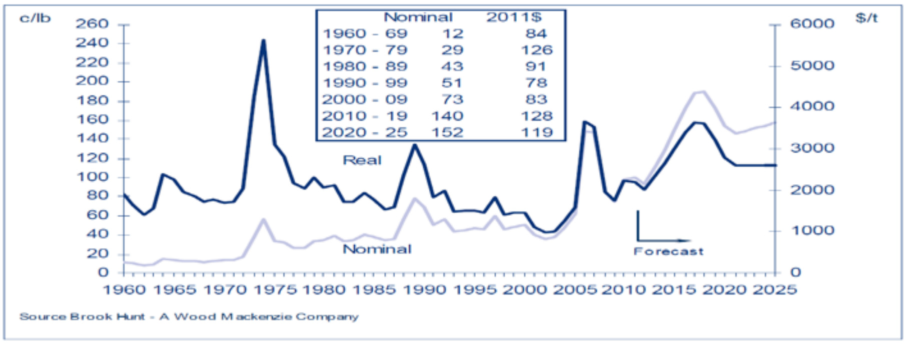
The main use of zinc is in galvanising steel which is then used mainly in building & construction, transport (including automotive) and consumer goods and appliances.

The end use of zinc is essential for the continuing industrialisation of the developing world. Galvanised steel is required in the construction of new buildings and the upgrade of existing urban infrastructure in developing economies – especially in China. Many consumer items associated with the industrialisation of developing economies use galvanised steel – for example cars, white goods and electrical goods such as air conditioners and microwaves.

The zinc component of these end-use products is small and price increases for the zinc component will not have a significant impact on the total cost of galvanised steel, for example.

Market expectations of zinc demand growth are in excess of 5% per annum for the next 5 years, underpinned by continued growth in the Chinese steel sector and trend towards value added steels (i.e. galvanised steel for corrosion protection).

Figure 1 Long-term zinc price



Source Brook Hunt - A Wood Mackenzie Company

2.2.3 Copper Demand and Supply

MMG takes a long term positive view of copper market fundamentals and we believe that demand will continue to grow at a faster rate than new supply coming online.

Copper Supply

The copper price has been supported by supply side constraints which have assisted coppers outperformance compared to other commodities.

Cost inflation is a major issue amongst current producers. Equipment and labour costs are impacting the cost structure in major producing regions. The focal point for copper in recent years is the ability of mine supply to deliver nameplate production consistently.

Declining grades are a significant issue amongst existing producers. In 2005, the top 10 mines produced 5Mt; in 2012 the same 10 mines produced 4Mt. This degradation has played a major part in supply underperformance.

Project finance – Higher interest rates may reduce supply significantly as this will further add to the general difficulty in obtaining project funding.

Many copper deposits are located in regions with a high degree of country risk, including political and social volatility, which pose challenges and increase the costs associated with developing these deposits. While global copper output is forecast to expand, MMG's view is that it will tighten over time due to a reduction in discoveries, higher production costs and declining grades.

Copper Demand

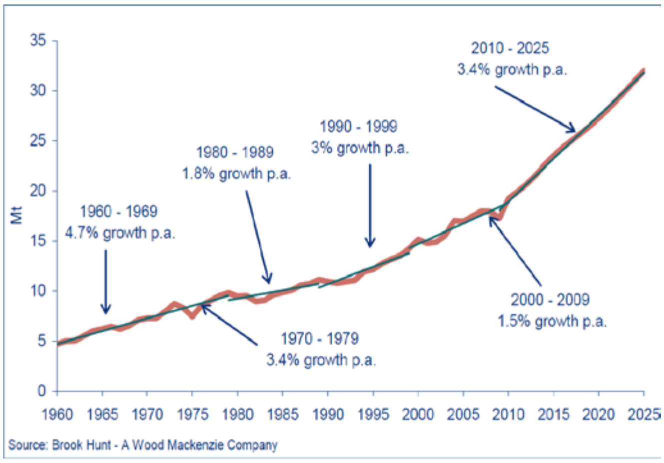
The main uses of copper are:

- As a conductor of electricity as it encounters much less resistance compared to other metals.
- Worldwide information and communications technologies through data transmission through copper infrastructure of ordinary telephone wire.
- Building construction - for example plumbing, taps, valves and fittings.
- Transportation – for example motors, wiring, radiators, brakes and bearings of vehicles, airplanes, trains.
- Industrial machinery and equipment due to durability, machinability and ability to be cast with high precision.
- Approximately 54% of copper is used in equipment, 32% in building construction and 14% in infrastructure.

Construction of social housing in China is expected to be a key driver of copper demand in the short term. China is expected to commence building approximately six million affordable housing units in 2013 as part of China's twelfth five-year plan targeting the construction of 36 million new homes by 2015.

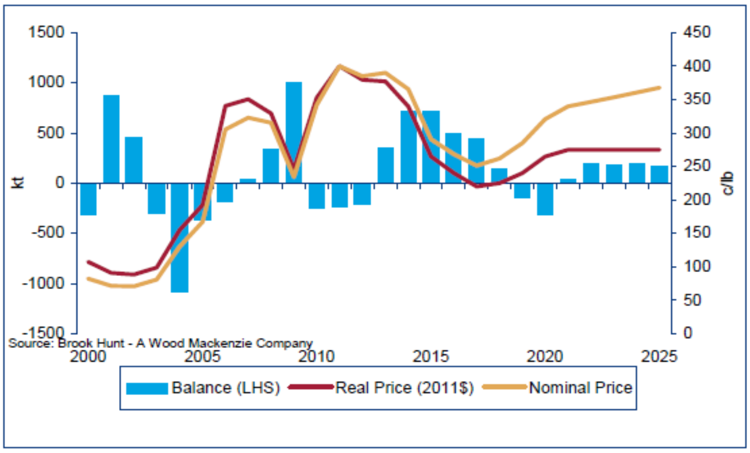
Affordable housing and the move toward a consumption-based economy is anticipated to support long-term copper demand growth.

Figure 2 Long-term copper consumption



Source: Brook Hunt - A Wood Mackenzie Company

Figure 3 Long-term copper price



Source: Brook Hunt - A Wood Mackenzie Company

3. SEPON – COPPER AND GOLD OPERATIONS

3.1 Introduction and Setting

The Sepon gold and copper operations are located in south-central Laos. The property is located in the Vilabouly district of Savannakhet province, 235km east of the town of Savannakhet, 40km north of the town of Sepon (Figure 4).

Figure 4 Sepon Mine location

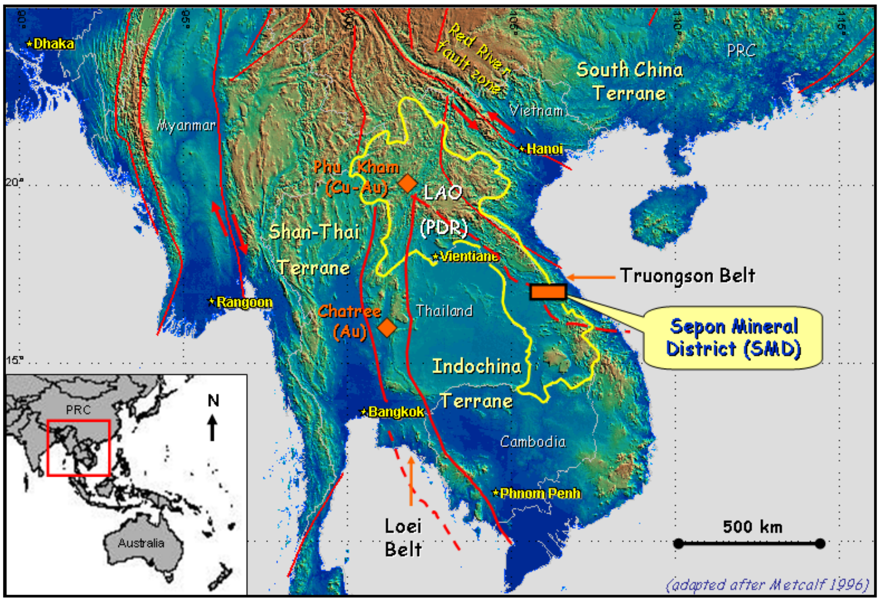
The main road from Vientiane to Savannakhet is National Route 13, a paved single-carriageway highway. The route is located within the Mekong River basin and crosses many tributaries few of which compromise travel during exceptionally heavy wet season events. The Sepon Operation is located east of Savannakhet via National Route 9 then northward from Ban Nabo along National Route 28A.

MMG Lane Xang Minerals Limited Sepon (LXML) operates the Sepon gold and copper operations and is a subsidiary of MMG Limited. MMG owns 90% of LXML, while the Government of Lao owns the remaining 10% of LXML.

3.2 Geological Setting

The Sepon project area is situated near the eastern margin of the intra-continental Khorat Basin and on the western flank of the Anamite Range fold belt, as shown in Figure 5. It lies within the Truongson geological region covering a broad spectrum of rocks ranging in age from Upper Proterozoic to Jurassic. Further to the southeast in Vietnam lies the Archean basement of the Khontoum Massif.

Figure 5 Sepon Regional geology



The regional geology of the project area is dominated by an Upper Palaeozoic sedimentary belt of arkosic and feldspathic sandstone, variably calcareous and carbonaceous siltstone, shale and limestone which is variably dolomitized and locally marble. There are lesser volcanic rocks, typically comprised of agglomerate, conglomerate, tuffaceous sandstone, and rare coherent volcanics. The belt is cut by plutonic to sub-volcanic bodies of granite, monzodiorite, granodiorite, quartz porphyry, rhyodacite porphyry (RDP) and andesite porphyry. The intrusive rocks are preferentially emplaced along either east or north-west trending well-developed structures.

The bulk of the mineralisation across the Sepon district is spatially associated with RDP intrusive centres. All known copper deposits are immediately adjacent to intrusive centres and the main sediment-hosted gold zone is located between two of the largest intrusive centres and clearly located in a peripheral position with respect to the copper zones.

The structural architecture around the margins of the intrusions is a key control on the distribution of mineralisation. North-west to west-north-west and east to east-north-east faults localise mineralisation from district to outcrop scale and likely acted as the main conduits for mineralizing fluid. These faults provided traps and focused fluids into other depositional sites such as lithological contacts and fold axes. Much of the mineralisation occurs to the east and west of the intrusive centres, rather than the north or south. This also likely relates to the overall architecture of the basin and more specifically the orientation of favourable feeder structures like north-west and east- striking faults (Figure 6 and Figure 7).

Figure 6 Summary geological map of the Sepon Mineral District (Smith, S.G. et.al, 2005)

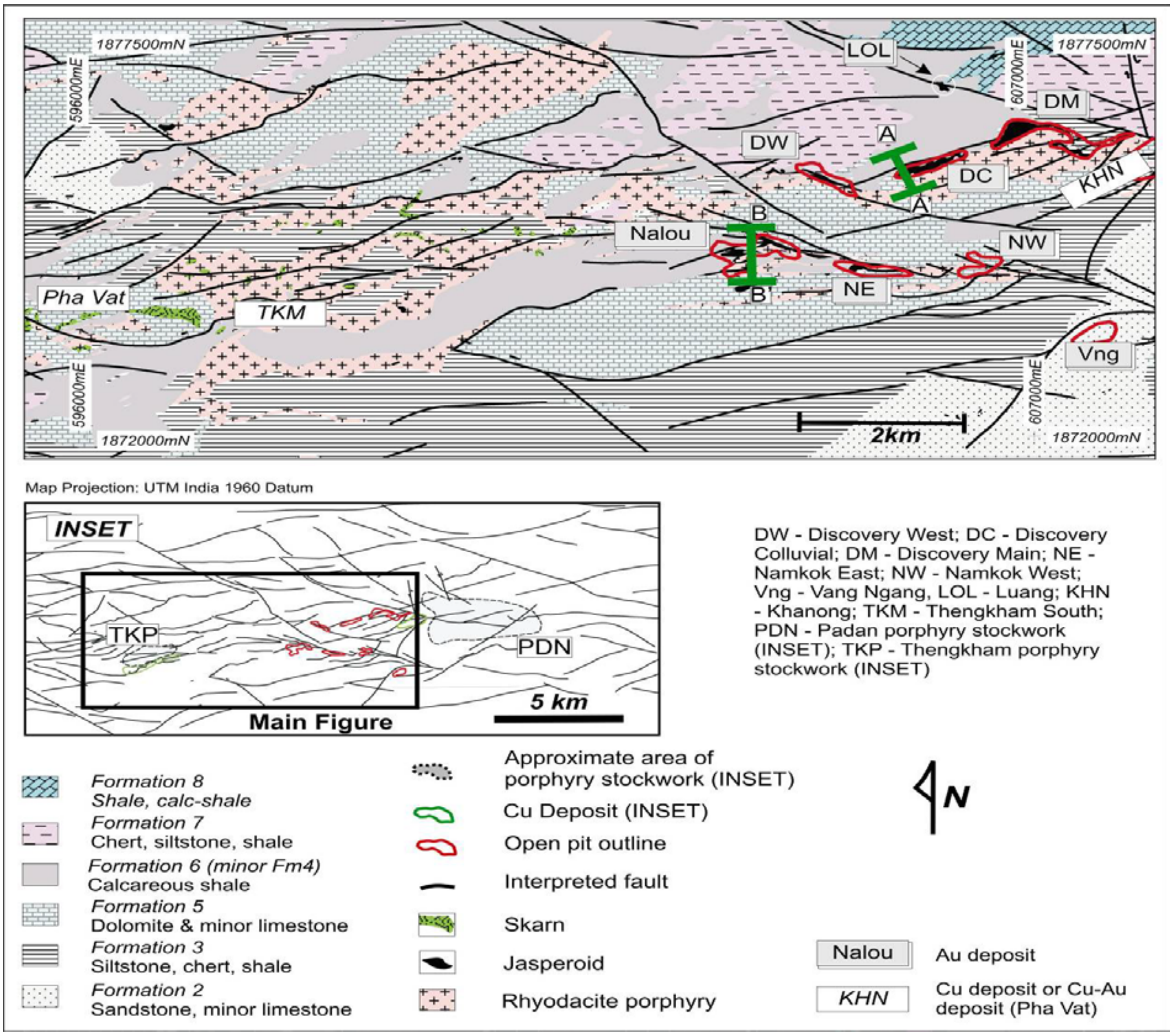
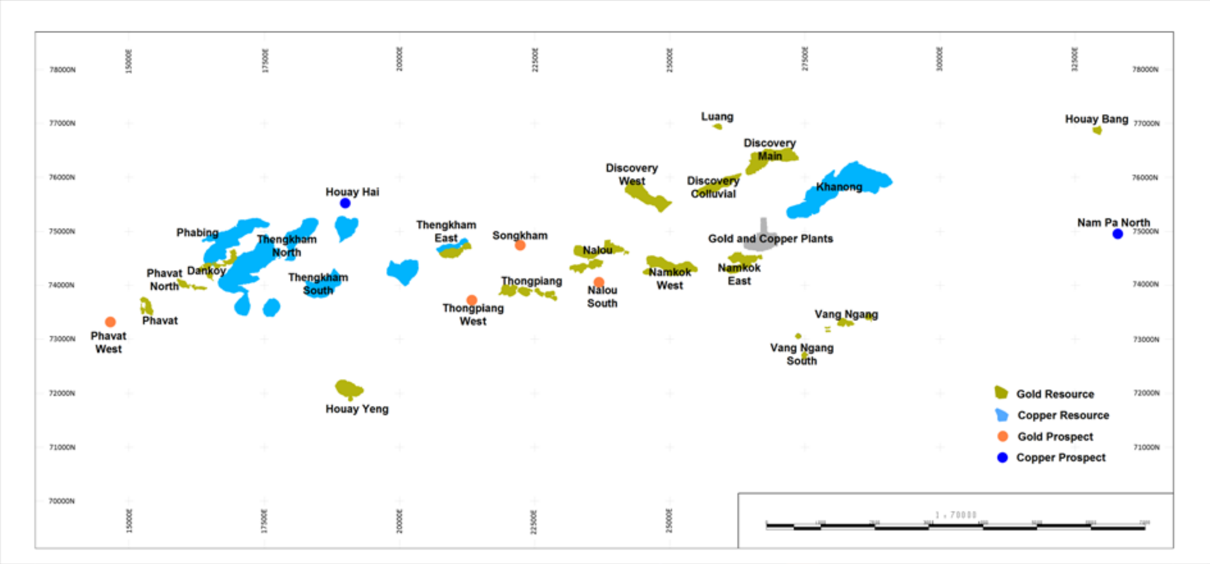


Figure 7 Location of Sepon gold and copper deposits



3.3 Mineral Resources - Sepon

3.3.1 Results

MMG updated the Sepon Mineral Resource in June 2013 in accordance with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (JORC Code) 2012 edition. The Mineral Resource was estimated and compiled for all Sepon deposits however some estimates remain unchanged from those reported in June 2012 while others were subjected to minor changes due to additional drilling incorporated in the estimates up to 30th June 2013 (Table 3).

All Mineral Resources quoted in this report were estimated from three dimensional block models created with Mintec MineSight and Maptek Vulcan software. Mineral Resources are modelled using solid wireframes of geological boundaries and/or a minimum 0.3% Cu or 0.3g/t Au-0.5g/t Au cut-off boundary which approximates the natural break between copper and gold mineralisation and background grades.

The Mineral Resource includes Measured, Indicated and Inferred categories and is inclusive of the Mineral Resource used to derive the Ore Reserves. The 2013 Mineral Resource estimate results are shown in Table 3.

Table 3 Sepon Mineral Resource as at June 30 2013

Sepon Mineral Resources

					Contained Metal		
Copper	Tonnes	Copper	Gold	Silver	**Copper**	**Gold**	**Silver**
0.5% Cu cut-off grade	(Mt)	(% Cu)	(g/t Au)	(g/t Ag)	**('000 t)**	**(Moz)**	**(Moz)**
Supergene Copper							
Measured	12	2.3			280		
Indicated	19	2.6			490		
Inferred	11	1.5			170		
Total	**42**	**2.2**			**940**		
Primary Copper							
Measured	-	-	-	-	-	-	-
Indicated	3.1	1.2	0.2	8	40	0.02	0.7
Inferred	11	0.8	0.3	5	90	0.1	1.9
Total	**14**	**0.9**	**0.2**	**6**	**130**	**0.1**	**2.6**
Oxide Gold[A]							
Measured	2.0	-	2.2	6	-	0.1	0.4
Indicated	4.5	-	1.4	7	-	0.2	1.0
Inferred	2.4	-	1.2	4	-	0.1	0.3
Total	**8.9**	**-**	**1.5**	**6**	**-**	**0.4**	**1.7**
Partial Oxide Gold[B]							
Measured	1.1	-	3.1	12	-	0.1	0.4
Indicated	2.3	-	2.0	8	-	0.1	0.6
Inferred	1.8	-	1.4	5	-	0.1	0.3
Total	**5.2**	**-**	**2.0**	**8**	**-**	**0.3**	**1.3**
Primary Gold[C]							
Measured	-	-	-	-	-	-	-
Indicated	14	-	3.0	10	-	1.4	4.5
Inferred	8.7	-	2.7	7	-	0.8	2.0
Total	**23**	**-**	**2.9**	**9**	**-**	**2.2**	**6.5**
Total Contained Metal					**1,070**	**3.0**	**12**

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.

A. Cut-off: 3.8Mt reported above 0.5g/t Au, 4.7Mt reported above 0.6g/t Au

B. Cut-off: 2.6Mt reported above 0.5g/t Au, 2.6Mt reported above 0.6g/t Au

C. Cut-off: 19Mt reported above 1g/t Au, 3.2Mt reported above 3g/t Au

Competent Person:

1. Reginald Boryor (Member of AIPG, employee of MMG)

Copper and gold Mineral Resources have decreased since 2012.

The reduction in copper Mineral Resources from 2012 are due to:

- Reporting within a long-term pit shell and above a cut-off grade for 2013 (19Mt).
- 1.3Mt depleted from mining at the active Khanong, Thengkham South and Phabing copper open pits.

The changes between the 2012 and 2013 copper Mineral Resource are shown in waterfall charts in Figure 8 and Figure 9.

The reduction in gold Mineral Resources from 2012 are due to:

- 1.9 Moz Au depleted from the previously reported Mineral Resource, from mining at the Thongpiang, Phabing, Khanong and Thengkham South oxide gold pits.
- 32.4Mt decrease in tonnes for all material due to an increase in the reporting cut-off and reporting of Mineral Resources within pit-shells.
  - Four separate cut-off grades have been used in 2013 for the reporting of the gold Mineral Resource:
    - 0.5g/t Au for oxide and partial oxide gold Mineral Resources that were not remodelled in 2013.
    - 0.6g/t Au for oxide and partial oxide gold Mineral Resources that were remodelled in 2013.
    - 1.0g/t Au for primary gold Mineral Resource material.
    - 3.0g/t Au for gold material in the Dau Leuk deposit. Internal work by MMG has determined that the Dau Leuk deposit is amendable to underground mining methods; as such the reporting cut-off grade has been increased.

  - Reporting within pit-shells was introduced in 2013 to ensure compliance with JORC (2012) of material having reasonable prospects for eventual economic extraction. Reporting within pit-shells is within the MMG criteria for reporting of Mineral Resources at no less than 70% of the Ore Reserves financial parameters. The gold pit-shells used represent the up-side case for the current Pre-Feasibility Study on primary gold.
- Thengkham East, Thengkham South and Thengkham North oxide gold Mineral Resources (0.08Mt) was entirely written off due to increase in cut-off grade for 2013.
- 3.9Mt (0.5 Moz Au) material was added as a result of infill drilling and model update from the Phavat West and Vang Nyang South deposits.

The changes between the 2012 and 2013 gold Mineral Resource are shown in waterfall charts in Figure 10 and Figure 11.

Figure 8 Sepon copper Mineral Resource waterfall chart (tonnes) 2012 – 2013 (Measured, Indicated and Inferred > 0.5% Cu)

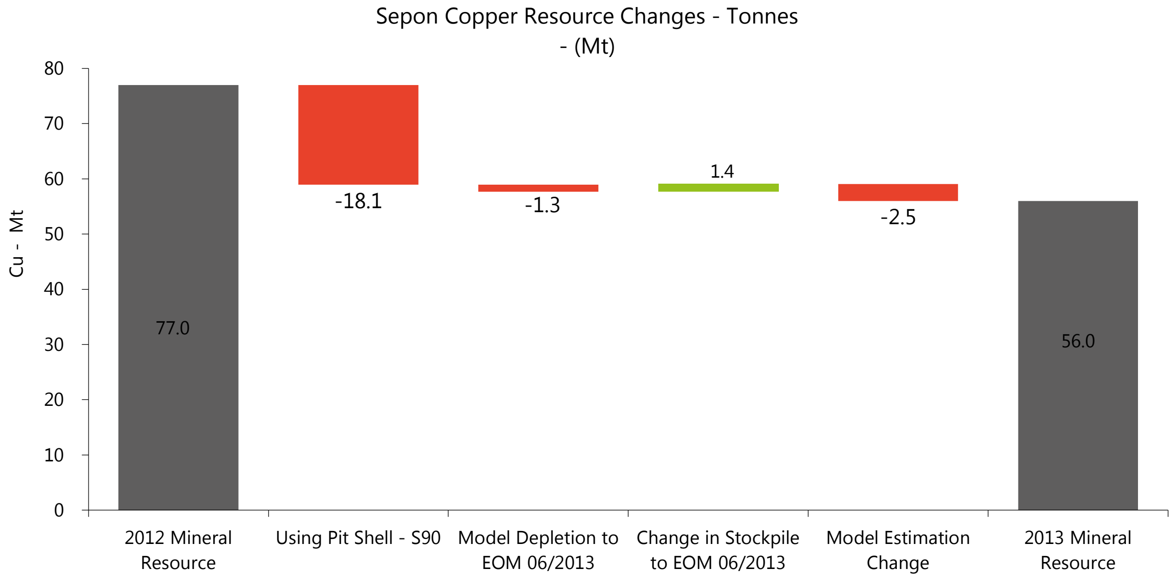


Figure 9 Sepon copper Mineral Resource waterfall chart (contained metal tonnes) 2012 – 2013 (Measured, Indicated and Inferred > 0.5% Cu)

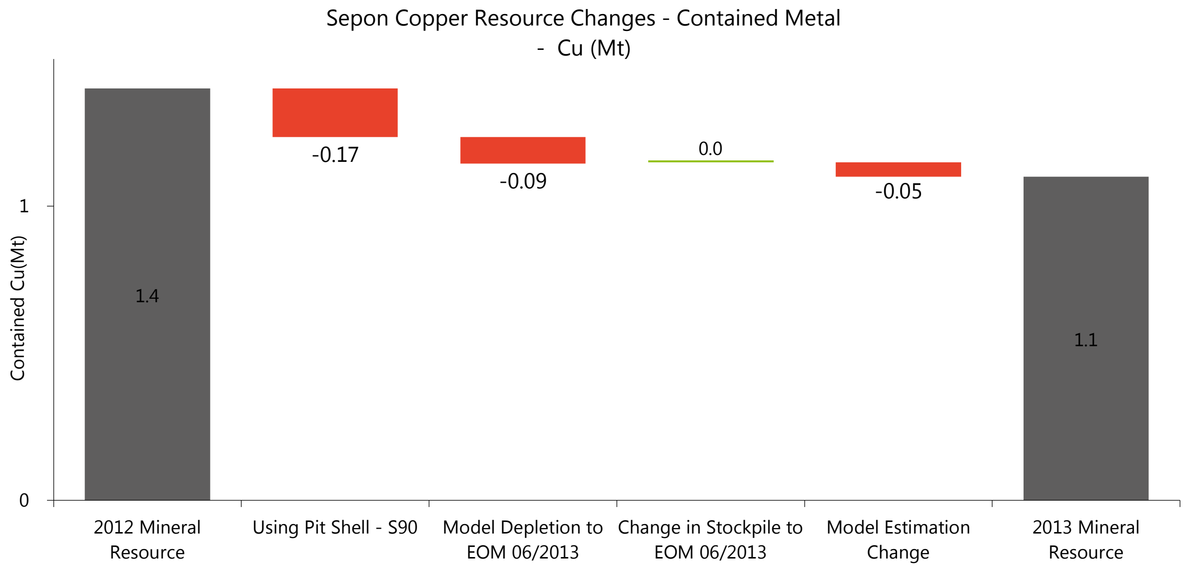


Figure 10 Sepon gold Mineral Resource waterfall chart (tonnes) 2012 – 2013 (Measured, Indicated and Inferred > 0.5g/t Au)

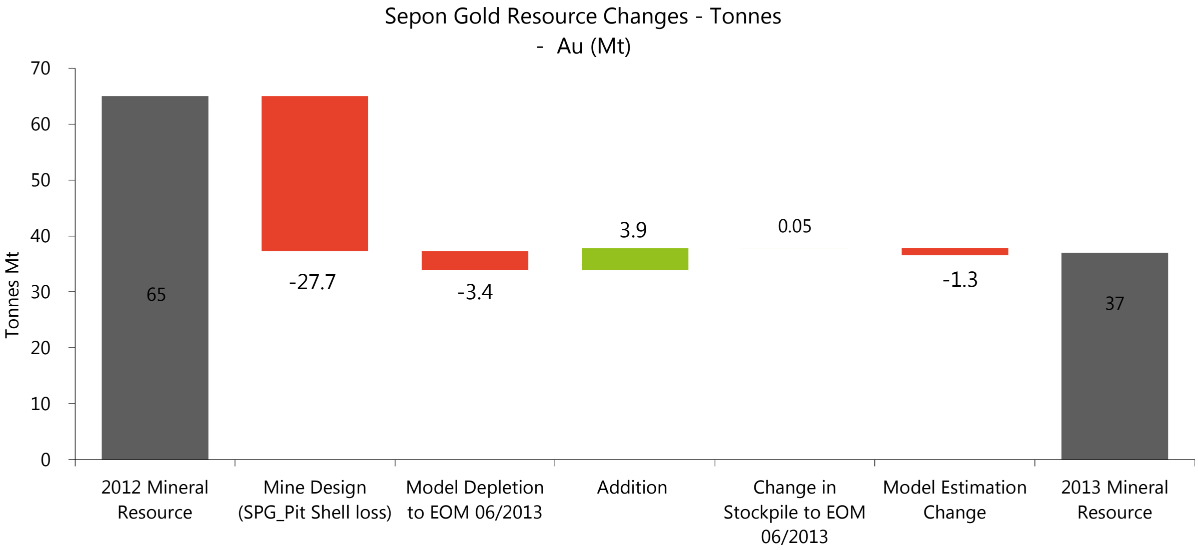
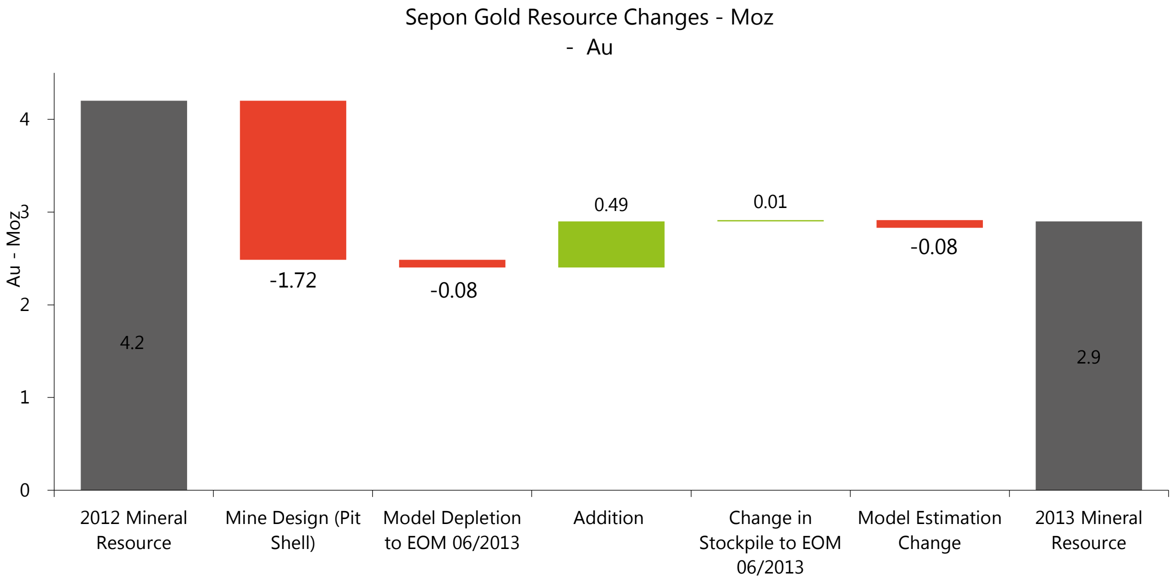


Figure 11 Sepon gold Mineral Resource waterfall chart (contained metal tonnes) 2012 – 2013 (Measured, Indicated and Inferred > 0.5g/t Au)



3.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Reginald Boryor, confirm that I am the Competent Person for the Sepon Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The American Institute of Professional Geologists - a 'Recognised Professional Organisation' (RPO) suitable for JORC Code reporting.
- I have reviewed the relevant Sepon Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Sepon Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Sepon Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Reginald Boryor – 26/11/13

Michael Stott (Witness)

3.4 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Sepon Mineral Resources.

Table 4 Checklist of assessment and reporting criteria for the Sepon Mineral Resources

Assessment Criteria	Commentary
	Section 1 Sampling Techniques and Data
Sampling techniques	▪ Diamond drilling (DD) (HQ triple tube) was sampled on nominal 1m length (+/-0.5m) samples or at geologically selected intervals. Core was sawn in half to provide half core samples that were submitted for analysis. ▪ Reverse circulation (RC) drill samples were collected at 1m intervals from a cyclone at the side of the drilling rig.
Drilling techniques	▪ DD HQ triple tube and occasionally PQ drilling and RC was used for the geological interpretation. ▪ A summary of drillholes by deposit is listed in Table 5.

Table 5 Drillholes by deposit and total metres

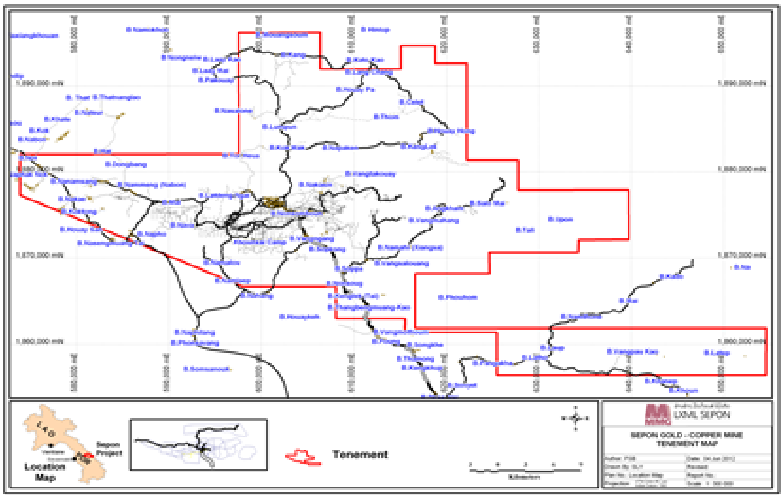
		Pre- June 2012		July 2012- June 2013	
Deposit	**Drillhole type**	**Number drillholes**	**Metres**	**Number drillholes**	**Metres**
VNE	RC	275	19,336	-	-
VAT	RC	152	7,544	-	-
	GC	1,116	19,029	-	-
PVN-DKY	RC	535	32,035	-	-
NKW	RC	550	32,216	14	1,460
TPG	RC	417	16,094	-	-
KHN2008	RC	4,714	27,698	-	-
KHN2002	RC	241	1,307	-	-
TKS	RC	1,618	46,653	-	-
TKN	RC	926	99,784	-	-
TKE	RC	-	-	42	2,177
DSW	RC	-	-	28	2,640
Total		**10,544**	**301,697**	**84**	**6,277**
VNE	DD	69	4,022	-	-
VAT	DD	35	4,208	9	192
DKY		-	-	19	930
PVN	DD	327	27,995	57	3,093
NKW	DD	38	2,045	9	357
TPG	DD	417	8,589	50	2,064
VNS	DD	-	-	53	3,336
DSM(DSC, DSE, LOL)	DD	1,302	92,009	37	1,586
DSW	DD	775	69,484	10	592
NLU	DD	1,235	88,966	97	7,216
PVW	-	-	-	56	3,924
KHN 2002	DD	65	6,060	-	-
KHN 2008	DD	237	18,301	-	-
TKS	DD	1,618	74,248	-	-
TKN	DD	1,912	113,840	-	-
PHB	DD	-	-	-	-
TKE	DD	-	-	-	-
Total		**8,030**	**509,769**	**397**	**23,290**

Deposit	**Pre-Jun 2000**	**July 2000 – June 2002**	**July 2002 – May 2006**	**June 2006 – June 2007**	**June 2007 – June 2013**
DSM	5,412	8,619	32,265	8,069	71,804
DSW	2,414	7,736	38,680	23,358	81,468
NLU	6,968	5,814	54,147	18,515	98,411
NKW	1,227	2,785	31,004	193	39,706
NKE	1,333	1,221	8,341	-	20,379
VNG	519	30	6,518	803	20,748
LOL	3,984	585	9,319	10,330	26,471
PVN	306		19,347	4,160	52,005
DKY	1,067		9,590	12,652	28,497
YNG	420		23	9,719	29,188
KHN	4,258	14,690	18,643	6,382	69,980
TKN	1,336	84	29,237	23,963	70,967
TKM	1,285	1,032	15,117	13,096	120,832
PHB	174	-	3,041	10,187	54,522

TKE	516	-	1,008	3,531	33,179
VAT	1,879	-	7,748	377	22,265
TPG	535	-	2,168	968	22,640
PVW	-	-	-	-	3,924
VNS	-		-	-	3,336
Stockpiles	-	-	-	-	1,360
Total	**33,633**	**42,596**	**286,196**	**146,303**	**871,682**

VNE - Vang Nyang East, VAT- Phavat, PVN - Phavat North, PVW - Phavat West, DKY – Donkay, NKW - Namkok West, TPG – Thongpiang, KHN - Khanong, TKS - Thengkham South , TKN - Thengkham North, TKE - Thengkham East, YNG - Houay Yeng, DSM – Discovery Main, DSW - Discovery West, VNS - Vang Nyang South, LOL - Muang Luang, NLU - Nalou, PHB - Phabing

Drill sample recovery	▪ Sample recoveries tend to be better in DD (90%) than RC (70% - calculated) with minor differences between mineralised zones and waste. ▪ Sample recovery is better in primary rock than transitional and oxide rock. ▪ Recoveries tend to be marginally lower in mineralised zones.
Logging	▪ Detailed logging is undertaken on all RC sample chips in chip tray and DD core on paper log sheets and entered manually into the Micromine GBis© database. All RC chip trays are kept for reference purposes. ▪ Logging uses pre-determined Sepon codes for; lithology, structure, mineralisation, geotechnical, oxidation, alteration and a site developed metcode (geometallurgical characteristics). ▪ DD core is photographed and stored digitally on the MMG server. ▪ All drill core is stored at the Sepon core shed.
Sub-sampling techniques and sample preparation	▪ DD core is orientated along the bottom of hole using a down-hole spear (usually offset 1cm from structural orientation mark when available), then half-core samples are taken using a diamond core saw for competent core or sampling by hand using a spatula or blade for clay-rich material. ▪ RC samples are collected at 1m intervals in a cyclone at the side of the drilling rig and a sub-sample collected via a riffle splitter if dry. If RC samples are considered moist or wet, then sampling is completed by quartering. The split portion weighing 3kg-5kg is collected in numbered sample bags for analysis. ▪ Field duplicates are taken every 15m for RC, and every 20m for DD half core samples. ▪ Upon receipt of samples at the laboratory samples are: – sorted, barcode tagged for tracking and weighed, – oven dried at 110°C (core samples: minimum of 12 hours drying. RC samples: 24 hours or longer - until the sample is completely dry to pass through a crusher without pelleting), – reduced in size through a jaw crusher (70% passing 2mm), – rotary split to 3kg if required, then pulverised using an LM5 to 85% passing 85µm, – a 110g pulp aliquot for gold fire assay and 20g pulp aliquot for ICP multi element is taken. ▪ Sample preparation technique, quality and size of sample are considered appropriate for the nature and grainsize of materials being sampled for both DD and RC samples.
Quality of assay data and laboratory tests	▪ Sample analysis typically takes place at ALS laboratory Vientiane for resource definition drilling, and at the on-site laboratory for grade control drilling. ▪ The analytical procedure at ALS Vientiane is as follows: – Samples are analysed for gold by fire assay method at a detection limit of 0.01g/t – If Au grade > 10g/t, re-analysed by fire assay gravimetric method. – If Au grade > 0.4g/t Au, re-analysed using CN Leachwell technique. – Ag, As, Bi, Ca, Cd, Co, Cu, Fe, Mg, Mn, Mo, Ni, P, Pb, S, Sb, Sr and Zn are analysed by ICP-OES. – If Cu > 0.5%, the sample is re-assayed using an ore grade technique. ▪ Assay data quality was determined through submission of matrix matched certified standards, coarse and pulp blanks, field duplicates and pulp repeats which were inserted at a rate of at least 1 in 15 samples (earlier deposits range from 1 in 25 to clusters of 3 in 25). ▪ Rigorous checks of the laboratory results and data import procedures are undertaken regularly to identify any spurious results for verification and re-assay. ▪ Any suspect data is excluded from Mineral Resource estimation.
Verification of sampling and assaying	▪ Significant assay results were verified against logging and core photos. ▪ For deposits containing large proportions of RC drilling, twinned holes were periodically drilled as part of drill quality analysis and are discussed in the 'Drill Sample Recovery' Section of this table. ▪ Generally, wet and moist samples (RC) have demonstrated smearing and a positive grade bias. These wet samples are given a lower level of confidence in the database and are excluded during Mineral Resource estimation.

	▪ Current practice is to use DD rather than RC when wet conditions are experienced. ▪ Assay results are loaded via automation into the database, no manual input occurs.
Location of data points	▪ New drillhole collars have been surveyed by hand held GPS Instruments and confirmed after drilling by mine site surveyors. ▪ Historical drillhole collars have been validated through a process of database and spatial checking, which was enabled by a LIDAR (Light Detection and Ranging) survey completed in 2008, which facilitated the checking of drillhole collar locations where GPS pick-up was not possible due to the heavily vegetated terrain. A number of drillholes were identified as having suspected locations. These issues were resolved prior to modelling of the data. ▪ Down-hole surveys have been carried out using Eastman single-shot cameras or Reflex EZ tools. Surveys are taken at depths of 12m, 30m, 60m (then every 30m to the bottom of hole). ▪ Sepon uses multiple grid systems, for Mineral Resource estimation work, all drillhole collars were converted from UTM/Indian60 projection to SPG06 local grid coordinate systems.
Data spacing and distribution	▪ Drillhole spacing generally ranges from 100m to 25m. On section spacing is generally 25m to 50m. ▪ The data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource estimation and classification methods used at Sepon.
Orientation of data in relation to geological structure	▪ Geological mapping and interpretation show that mineralisation is generally striking east-west; hence drilling is conducted on north-south directions, to intersect the mineralised zones at orthogonal angles and vary to suit individual deposits. ▪ Most drillholes were drilled with dips of -60 degrees from horizontal to intersect steeper structures, or vertically for flatter dipping stratigraphically controlled mineralisation. Drillhole orientation and depths were checked against site generated cross-sections. ▪ Drilling orientation is not considered to have introduced any sampling bias.
Sample security	▪ Measures to provide sample security include: – All samples are collected by adequately trained and supervised MMG sampling personnel. – Cut core are sampled and stored in calico bags, tied and clearly numbered in sequence. The core yard facility is enclosed with a security fence and sampling sheds are well maintained. – Calico sample bags are transported to the assay laboratory by commercial transport companies. – The assay laboratory both onsite and offsite, check sample dispatch numbers against submission documents.
Audit and reviews	▪ Drilling systems, data collection, sampling, dispatch and data input are managed by MMG geologists and technicians. ▪ Two laboratory audits were conducted at ALS Vientiane between July 2012 and June 2013 with no material issues identified during these visits.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ LXML Sepon Mineral Resources fall under an agreement with the Lao government entitled a Mineral Exploration and Production Agreement (MEPA) which is 90% owned by MMG, and 10% owned by the Laos Government. The MEPA provides for exploration, development and extraction of any Mineral Resources discovered. ▪ The current MEPA boundary encloses an area of 1,247km^2 (refer Figure 12). **Figure 12 Sepon Copper and Gold Mine Tenement Map** 

Exploration done by other parties	Exploration summary by other parties: ▪ CRA Exploration (CRAE, later RTZ) first identified the Sepon Mineral District as an area of interest in 1990. ▪ Between 1995 and 1999 RTZ (RTZ was formed from the merger of CRA and Rio Tinto in 1997) discovered and defined gold Mineral Resources at the Discovery Main, Discovery East, Discovery West, Discovery Colluvium, Nalou, Namkok East, and Namkok West prospects and copper and gold Mineral Resources at the Khanong prospect. ▪ Oxiana became manager of the Sepon Project by buying 80% of LXML in 2000. ▪ Oxiana later bought the remaining 20% interest from RTZ before the Laos government exercised its option to acquire a 10% interest in LXML in 2006. ▪ Exploration drilling at Thengkham East commenced in 2008, effectively seeking strike extensions to the Thengkham South deposits.
Geology	▪ Mineralisation appears to be both structurally and stratigraphically controlled, and is related to the hydrothermal systems associated with porphyry intrusives. ▪ Six hydrothermal alteration systems, centered on porphyry intrusives have been identified; Padan, Thengkham, Nakachan, Ban Mai, Katia, and Kaban. The Sepon deposits are largely associated with the Padan and Thengkham porphyry centres. ▪ Four broad primary mineralisation styles are recognised: – sediment-hosted gold (e.g. Discovery, Nalou, Namkok), – copper-gold carbonate replacement (e.g. Khanong copper), – copper-gold skarn (e.g. Thengkham), – quartz stockwork porphyry. **Copper Mineralisation:** ▪ Ranges from central porphyry style Mo-Cu-Ag mineralisation through to retrograde skarn Cu-Mo-Au-Ag-Bi mineralisation associated with carbonate rocks close to the intrusive centres. The molybdenum is not of economic significance. ▪ Mineralisation is often focused along flat-lying brecciated shear which has facilitated the introduction of the copper-rich fluids. Where the shear zone intersects carbonate-rich rocks, carbonate replacement has taken place. ▪ Original massive and semi-massive pyrite and chalcopyrite mineralisation is interpreted to have been upgraded by weathering and supergene enrichment processes. Detailed mineralogical examination has identified digenite, covellite and bornite in addition to the dominant chalcocite. ▪ Copper oxide and carbonate mineralisation is best developed further down slope and the copper in this mineralisation has been remobilised during the weathering process. Copper oxide and copper carbonate minerals (principally malachite and azurite with some cuprite and native copper) occur within 5m to 10m thick fault-bounded blocks below the chalcocite clay zone and overlying fresh dolomitic footwall lithology's. ▪ Primary pyrite/chalcopyrite mineralisation has been intersected at depth. The bulk of supergene copper and associated Au-Mo mineralisation exists as moderately dipping tabular to flat zones within the weathering profile. ▪ The continuity of mineralisation has varying strike lengths along slopes ridge. ▪ The copper mineralisation is often overlain by 10m to 20m of gossaniferous ironstone and limonitic clay containing low grade gold ranging from 0.5g/t Au-1.5g/t Au. **Gold Mineralisation:** ▪ Gold mineralisation occurs in association with decalcification and partial silica replacement of calcareous mudstones, and typically is best developed within the Discovery formation but can also occur as a karst-controlled residual or collapse breccia deposit within the underlying Nalou formation, with mineralised jasperoid boulders occurring within a matrix of decomposed rock and clays. ▪ Massive jasperoid is widely mineralised and typically contains the highest-grade gold mineralisation. ▪ Gold deposits at Sepon are of three main styles: – Sediment hosted, associated with de-calcification and silicification of calc-siltstones with preferential mineralisation along faults, contacts and in massive jasperoid e.g. Discovery, Namkok, Nalou. – Karst-controlled residual mineralisation in carbonate rocks and infill collapse breccias e.g. Houay Yeng. – Near surface iron and manganese-rich gossanous zones overlying supergene copper mineralisation e.g. Khanong and Thengkham project areas. ▪ Gold mineralisation generally has a gradational boundary, although in places the boundary can be sharp, especially in fault controlled primary mineralisation and at the base of karst fill gold mineralisation overlying unweathered carbonates.
Drillhole information	▪ Over 30,727 drillholes are used to estimate the Sepon Mineral Resources, none of which on their own are considered material, hence no additional information is provided for this section.

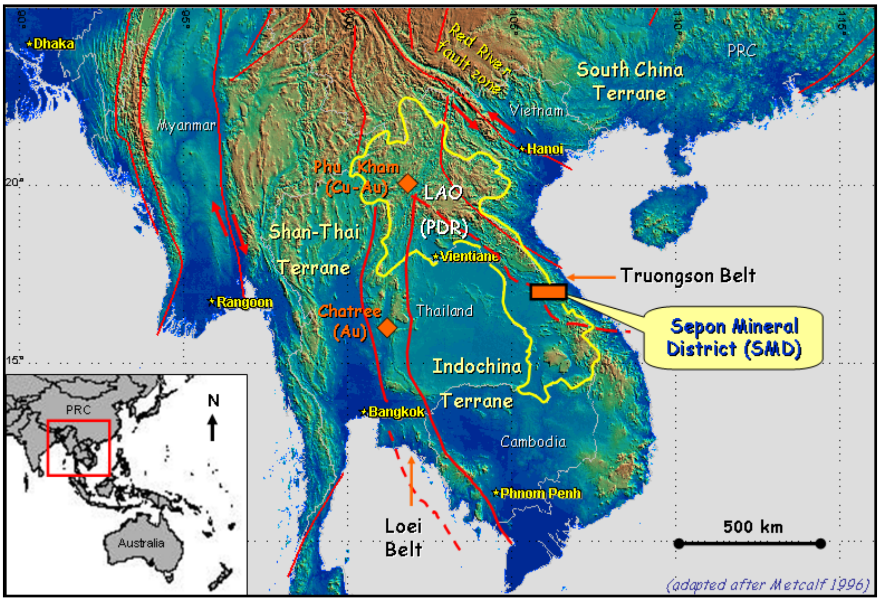
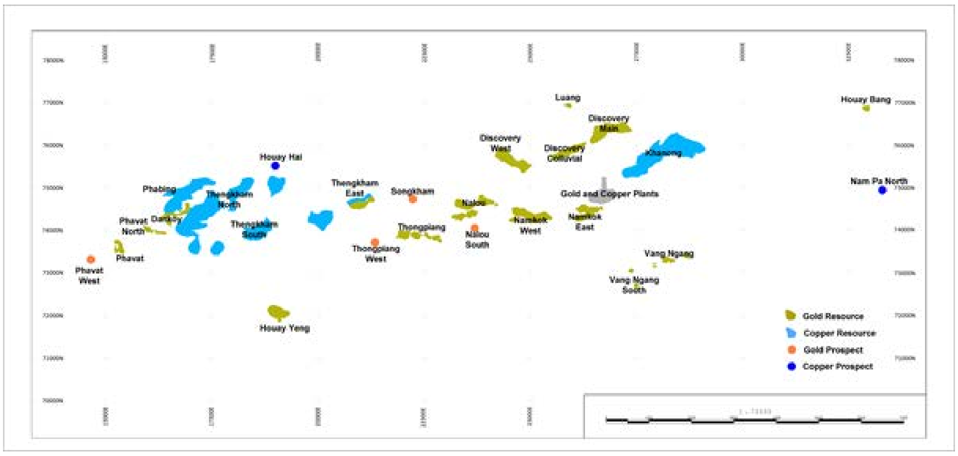
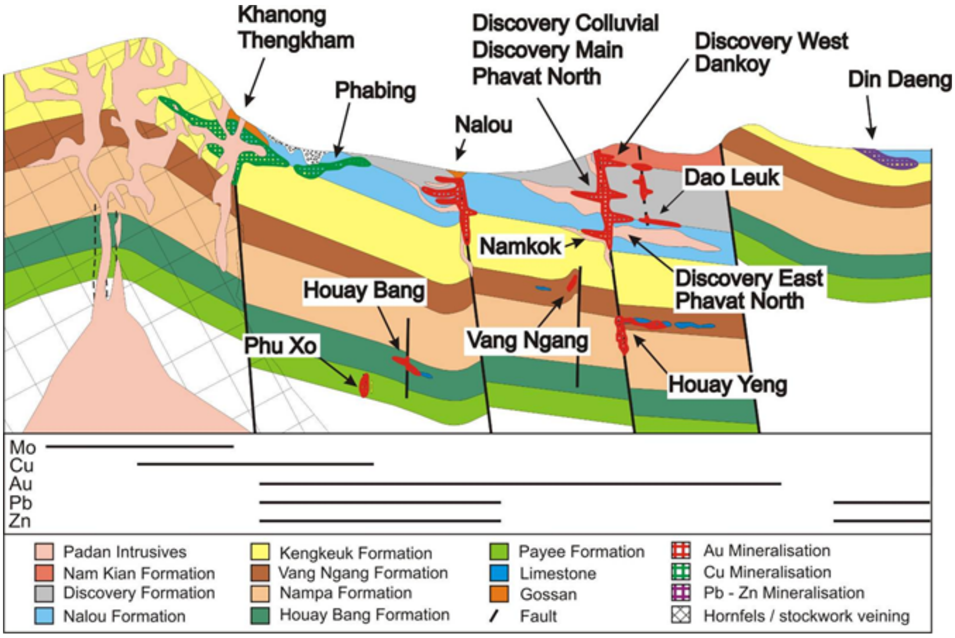
Data aggregation methods	- Over 30,727 drillholes are used to estimate the Sepon Mineral Resources, none of which on their own are considered material, hence no additional information is provided for this section.
Relationship between mineralisation width and intercepts lengths	- Over 30,727 drillholes are used to estimate the Sepon Mineral Resources, none of which on their own are considered material, hence no additional information is provided for this section.
Diagrams	**Figure 13 Location of Sepon Mine with Regional Geology, and Select Deposits in the Region**  (adapted after Metcalf 1996) **Figure 14 Location and distribution of known copper and gold deposits at Sepon** 

	Figure 15 Generalised east-west cross-section of the Sepon mineral deposits 
Balanced reporting	▪ Over 30,727 drillholes are used to estimate the Sepon Mineral Resources, none of which on their own are considered material, hence no additional information is provided for this section.
Other substantive exploration data	▪ Over 30,727 drillholes are used to estimate the Sepon Mineral Resources, none of which on their own are considered material, hence no additional information is provided for this section.
Further work	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	▪ The Sepon geological database system consists of three components; – Manual Field Logging System – Data Entry Database (DEDB), and – Master Database (LaoDB). ▪ Each digital component is configured to run in SQL Server with specific user access and permissions. ▪ The DEDB works as a quarantine and compilation system. The supervising database geologist reviews all new data against original paper logs, with corrections made prior to loading into LaoDB which is done via SQL Server stored procedures to detect and hold any errors on import. ▪ The GBis database and logging system was introduced in 2006 and populated from the pre-existing aQuire™ database. Ongoing analytical data is uploaded directly from laboratory SIF files.
Site visits	▪ The Competent Person is a full time employee of LXML Sepon Mine Site, and is permanently based onsite.
Geological interpretation	▪ All wireframe interpretations were based on sectional interpretations, with sectional interpretation strings snapped to drillholes. ▪ The extrapolation distances for triangulation interpretation are generally half the drillhole spacing typically 12.5m to 25m but occur up to 50m in less well drilled areas. Extrapolation distances are taken into account during Mineral Resource classification. **Copper mineralisation:** ▪ All copper domains were modelled at a nominal 0.3%-0.5% cut-off however the major consideration for copper domain modelling is on the metallurgical material type (METCODE) to take into account processing considerations. **Gold mineralisation:** ▪ Visual assessment of the relationship between copper and gold grade distribution and underlying geology supports the use of grade-based domains to constrain the Resource estimation. This is confirmed by statistical analysis which shows a clear step in grade across mineralisation boundaries. ▪ Gold domains were modelled at a nominal 0.3g/t-0.5g/t cut-off.

Dimensions	Sepon hosts a number of deposits each having variable dimensions. Details of mineralisation dimensions in two major deposits (Khanong and Thengkham) are described, and a high-level summary of the remaining deposits dimensions are listed below. - The Khanong copper mineralisation forms a flat-lying, oxidised, supergene-enriched blanket of primarily kaolinitic chalcocite clay. The lens dips shallowly to the northwest with a dip extent of 400m and a north-easterly strike of approximately 1km. Maximum thickness of the lode is around 70m, thinning to the south and west. The copper mineralisation is overlain by 10m-20m of gossaniferous ironstone and limonitic clay containing low grade gold. - The Thengkham copper mineralisation trends east-northeast and is associated with the Thengkham felsic intrusive complex, occurring to the north and south of the Thengkham ridge. Primary skarn mineralisation characterised by pyrite, chalcopyrite and molybdenite occurs at depth. Supergene copper carbonates, primarily malachite, and chalcocite-clays, occur where acidic copper-bearing ground waters have reacted with oxidised ground waters and carbonate rocks. The mineralisation extends over a strike length of more than 2,500m in a series of generally sub-horizontal pods, with a maximum thickness of up to 40m. High-level summary of deposit dimensions at Sepon: - DSW: 23875mE - 25555mE, 75250mN - 75970mN, 0mRL - 300mRL - DSM: 25500mE-28200mE, 75250mN-77110mN, 150mRL - 450mRL - NLU: 22700mE - 24500mE, 73730mN - 75350mN, 150mRL - 325mRL - NKW; 24500mE - 26060mE, 74000mN - 75320mN, 0mRL - 300mRL - PVN_DKY: E15300 - E17460, N73700 - N75200, RL-0 - RL650 - PVW: E14600 - E15020, N73000 - N73600, RL0 - RL275 - TPG: E21600 - E23100, N73500 - N74160, RL100 - RL275 - VNS: E 18700 - E27790, N72400 - N73300, RL-0 - RL500 - VNG: E27350 - E28850, N72900 - N73740, RL100 - RL400 - HYN: E27100 - E 19975, N 71700 - N 72420, RL-50 – RL400 - LOL: E25500 - E26550, N76600 - N7720, RL-400 - RL350 - PON: E17200 - E17875, N71500 - N72040, RL150 - RL425 - PVT: E15000 - E15768, N73100 - N73895, RL75 - RL300 - TKE: E20240 - E21620, N74000 - N75140, RL0 - RL425 - TKN: E16600 - E19650, N73750 - N75502, RL150 - RL600 - TKS: E16430 - E20850, N73160 - N74720, RL40 - RL600 - PHB: E15950 - E17750, N74250 - N75470, RL0 - RL500 - KHN: E26750 - E29150, N74748 - N76512, RL150 - RL650 Where: VNE - Vang Nyang East, VAT- Phavat, PVN - Phavat North, PVW - Phavat West, DKY – Donkay, NKW - Namkok West, TPG – Thongpiang, KHN - Khanong, TKS - Thengkham South , TKN - Thengkham North, TKE - Thengkham East, YNG - Houay Yeng, DSM – Discovery Main, DSW - Discovery West, VNS - Vang Nyang South, LOL - Muang Luang, NLU - Nalou, PHB - Phabing
Estimation and modelling techniques	Summary of estimation and modelling techniques: - Grades were interpolated into blocks using the Ordinary Kriging (OK) algorithm using MineSight or Vulcan software. - The optimal estimation block size is typically 15mE x 6mN x 2.5mRL (gold deposits) and 10mE x 6mN x 2.5mRL (copper deposits). This block size adequately delineates the ore zones within the block model, without compromising the localised calculated block variances. This block size can be re-blocked to 5mE x 3mN x 2.5mRL for grade control modelling. - Samples were composited to 2m down-hole lengths. The compositing process was checked and validated. - Variography and search neighbourhood optimisation for each domain was performed using Snowden Supervisor, MineSight and Geovariances Isatis geostatistical software packages. - The overall coefficient of variation for most deposits are relatively low (i.e. CV ~ 0.8 – 1.5). - For some deposits with high coefficient of variations, grade capping was used. This was typically in the 99th percentile, however varied depending on the results of statistical analysis including log probability plots. - The estimates of copper and gold were undertaken using hard domain boundaries and a series of elliptical search passes orientated in the plane of mineralisation. These search orientations and sizes were supported by variography analysis.

<table>
<tr><td></td><td>- The estimation search distance for most deposit ranges between 50mE x 20mN x 10mRL to 100mE x 60mN x 40mRL, additional larger passes were used to estimate less well informed blocks. However this varies from deposit to deposit.
- Statistical analysis and estimation of other ancillary elements (where appropriate); silver, arsenic, sulphide sulphur, total sulphur, gold (leachwell assay technique), carbon as carbonate, magnesium, calcium, organic carbon, iron and manganese were undertaken.
- In most cases sample selection during the interpolation process was constrained using a maximum of 3 composites from any given drillhole to interpolate a block. The minimum and maximum number of composites required to interpolate a block was typically set at 3 and 30 respectively.
- Acid Rock Drainage (ARD) characteristics were assigned to the block model using the lithology domain and sulphur grade (refer Table 6).

Table 6 Acid rock drainage characteristics assigned to the block models
<table><tr><th>ARD</th><th>Sulphur (%)</th><th>Lithology</th></tr><tr><td>PAF</td><td>>=0.3</td><td>Not dolomite or limestone</td></tr><tr><td>NAF</td><td>>=0.3</td><td>Dolomite or limestone</td></tr><tr><td>NAF</td><td><0.3</td><td>Other</td></tr></table>
- Block models were validated using the following techniques:
  – A visual check of block model interpolated grades and drillhole data in plan and section view,
  – Global statistical comparisons between interpolated grades and raw drillhole grades,
  – Swath plots between interpolated grades and raw drillhole grades.</td></tr>
<tr><td>Moisture</td><td>- Tonnes have been estimated on a dry basis.</td></tr>
<tr><td>Cut-off parameters</td><td>- Mineral Resources have been reported within long-term pit-shells (where they exist), at a cut-off of 0.5% Cu, and at variable cut-off's for gold ranging from 0.5g/t Au to 3g/t Au. Where:
  – 0.5g/t Au and 0.6g/t Au applied to open pit oxide and partial oxide gold material
  – 1g/t Au applied to primary gold material
  – 3g/t Au applied to the Dau Leuk deposit (primary material) which has potential for underground mining.
- Copper and gold cut-off grades used for reporting are comparable to current mining practices, or within the future expectation of cut-off grade for the underground Mineral Resources.</td></tr>
<tr><td>Mining Factors or assumptions</td><td>- No mining factors or assumptions have been applied to the Mineral Resource.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>- Metallurgical test work was completed during the Prefeasibility Study for both copper and gold. Test work included standard variability, comminution, grinding and float tests and the treatment of bulk samples from selected mineralisation types.
- Currently, only oxide material and limited transition material is treated through the gold and copper plants.
- **Copper deposits:** To account for the orebody complexity with reference to oxidation state, a series of "metcode" domains have been developed for copper deposits that include: Chalcocite clay; Copper oxide; Limonitic clay; Limonite ironstone; Limonitic soil; Manganese oxide; Oxide; Partial oxide; Primary; Pyrite-chalcopyrite.
- **Gold deposits:** To account for the orebody complexity with reference to oxidation state, the gold Mineral Resources have been defined to separate the three main oxide zones with respect to metallurgical characterisation. The surfaces are defined as the base of complete oxidation, the base of the transitional (or mixed oxidation) zone, which also forms the top of the third - primary zone.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>- No environmental factors or assumptions have been applied to the Mineral Resource.</td></tr>
<tr><td>Bulk Density</td><td>- Samples for density determination were taken from diamond drill core every 10m using a weight in air / weight in water wax immersion method.
- The density determinations were the basis for assigning densities to the Mineral Resource estimates and were predominantly based on mineralised/waste zones, consideration of the host lithology and oxidation state.</td></tr>
<tr><td>Classification</td><td>Classification is determined by examination of the following criteria:
- Geological: geological and mineralisation continuity.
- Statistical: commonly kriging variance, occasionally slope of regression is reviewed.
- Data: the relative data density, distance of nearest composite and number of composites used.</td></tr>
</table>

<table>
<tr><td></td><td>▪ Depending on the deposit, Mineral Resource classification is applied on a block-by-block basis, or Classification solids are constructed around aggregate areas.</td></tr>
<tr><td>Audits or reviews</td><td>▪ Historical models have been subject to a series of internal and external reviews during their history of development, with any material issues corrected.
▪ Sepon employs a rigorous internal peer review at the completion of every model update.
▪ Independent technical reviews:
– 2008: an independent technical review of the Sepon copper gold Mineral Resources was undertaken by Behre Dolbear Australia.
– Independent technical review on copper and gold Mineral Resources was completed in 2010 by AMC Consultants with no material issues identified.</td></tr>
<tr><td>Discussion of relative accuracy / confidence</td><td>▪ Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support.
▪ Monthly reconciliation between the Mineral Resource block model and the grade control block model show some variability in the estimated tonnes and grade of the Mineral Resource block model. There is no clearly defined trend for the Mineral Resource model under/over-estimating tonnage and grade.
▪ Reconciliation factors are commonly used onsite. Table 7 shows the F1annual factor comparing the grade control block model with the Resource block model. Where F1annual factor = grade control block model/Resource block model (tonnes, grade, contained metal).
Table 7 $F1_{Ann}$ Annual reconciliation figures by pit (June 2012 to May 2013) Mineral Resource Model/Grade Control Model
<table>
<tr><th></th><th>$F1_{Ann\_Tonnes}$</th><th>$F1_{Ann\_Grade}$</th><th>$F1_{Ann\_Metal}$</th></tr>
<tr><td>Khanong Copper</td><td>1.15</td><td>0.96</td><td>1.10</td></tr>
<tr><td>Phabing Copper</td><td>1.10</td><td>0.87</td><td>0.95</td></tr>
<tr><td>Thengkham South D Copper</td><td>1.38</td><td>0.87</td><td>1.20</td></tr>
<tr><td>Phabing Gold</td><td>2.09</td><td>0.68</td><td>1.45</td></tr>
<tr><td>Khanong Gold</td><td>0.73</td><td>1.27</td><td>0.88</td></tr>
<tr><td>Thengkham South D Gold</td><td>0.84</td><td>1.17</td><td>0.94</td></tr>
<tr><td>Thongpiang</td><td>0.90</td><td>1.02</td><td>0.92</td></tr>
<tr><td>Muang Luang</td><td>1.12</td><td>1.01</td><td>1.13</td></tr>
</table>
*June 2012 - May 2013</td></tr>
</table>

3.5 Ore Reserves - Sepon

3.5.1 Results

The Sepon June 2013 Ore Reserves statement is derived from the 2013 gold and copper Mineral Resources at the Sepon mining operation.

This Ore Reserves statement does not include any partially oxidised (POX) or primary gold or copper Mineral Resources.

Ore Reserves for Sepon as at 30 June 2013 are given in Table 8 and Table 9.

Table 8 2013 Sepon Gold Ore Reserves tonnage and grade (as at 30 June 2013)

Classification	Deposit	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained Metal†	
					Au ('000 oz)	Ag ('000 oz)
Proved	Discovery East	0.05	2.2	7.0	3.6	11
	Discovery Main	0.05	2.6	9.7	3.1	12
	Phavat North 1	-	-		-	
	Phavat North 2	-	-		-	
	Sub-Total	**0.09**	**2.4**	**8.2**	**6.7**	**23**
Probable	Discovery East	0.01	2.1	6.2	0.72	2.1
	Discovery Main	0.005	2.7	7.4	0.40	1.1
	Phavat North 1	0.44	1.8	3.7	25	53
	Phavat North 2	0.02	1.3	4.8	1.0	3.4
	Sub-Total	**0.48**	**1.7**	**3.9**	**27**	**59**
Proved & Probable	Discovery East	0.06	2.2	6.8	4.3	13
	Discovery Main	0.04	2.6	9.5	3.6	13
	Phavat North 1	0.44	1.8	3.7	25	53
	Phavat North 2	0.02	1.3	4.8	0.95	3
	Total	**0.57**	**1.8**	**4.5**	**34**	**82**

**Totals may differ due to rounding; †Contained metal does not imply recoverable metal*

Table 9 Sepon Copper Ore Reserves tonnage and grade (as at 30 June 2013)

Classification	Deposit	Tonnes (Mt)	Grade (%Cu)	Contained Metal† Copper('000t)
Proved	Khanong	0.002	2.6	0.04
	Phabing	-	-	-
	Thengkham South	-	-	-
	Thengkham South D	-	-	-
	Thengkham North	-	-	-
	Thengkham East	-	-	-
	Stockpiles	5.4	2.6	138
	Sub-Total	**5.4**	**2.6**	**138**
Probable	Khanong	1.7	7.1	118
	Phabing	1.4	3.7	51
	Thengkham South	1.6	3.9	60
	Thengkham South D	0.7	4.8	34
	Thengkham North	2.7	4.5	121
	Thengkham East	0.6	4.1	24
	Stockpiles	-	-	-
	Sub-Total	**8.6**	**4.8**	**408**
Proved & Probable	Khanong	1.7	7.1	118
	Phabing	1.4	3.7	51
	Thengkham South	1.6	3.9	60
	Thengkham South D	0.7	4.8	34
	Thengkham North	2.7	4.5	121
	Thengkham East	0.6	4.1	24
	Stockpiles	5.4	2.6	138

3.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (“2012 JORC Code”).

Competent Person Statement

I, Julian Poniewierski, confirm that I am the Competent Person for the Sepon Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Sepon Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited since August 2012.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 767,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 October 2013 was $HKD 1.72).

I verify that the Sepon Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in the supporting documentation relating to Ore Reserves as compiled by MMG Sepon and MMG Melbourne staff, and audited by Julian Poniewierski.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Sepon Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Julian Poniewierski – 26/11/13

Mauro Bassotti (Witness)

3.5.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed in Table 10.

Table 10 Contributing experts – Sepon Mine gold and silver Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Reginald Boryor, Superintendent - Resource Geology MMG Ltd (Sepon)	Geological Resources
Cameron Legg, Senior Mining Engineer MMG Ltd (Melbourne)	Mining Engineering
Won Hong, Principal Mining Engineer MMG Ltd (Sepon)	Mining Engineering
James McTiernan, Superintendent - Metallurgy MMG Ltd (Sepon)	Metallurgy
Gavin Marre, Senior Business Analyst MMG Ltd (Melbourne)	Economic Assumptions

3.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

3.6.1 Pit Design

A feature of Sepon site is multiple small to medium scale pits of short duration (from 6 months to four years). As such the pits are generally not re-optimised every year. Previously optimised pits have not been re-optimised in 2013, but have been checked for deviation from the original design parameters to determine if a re-optimisation might be warranted. From past experience, once a pit has been designed and started, changes in prices do not generally create a change in a pit design that is practical to adopt (specifically with respect to maintaining a minimum mining width).

The pit design is based on shells selected using Whittle Four-X software to run pit optimisations.

Pits are designed with 10m batter heights (generally mined at 2.5m height). Batter wall angles and berm widths were as per geotechnical consultant recommendations and vary according to expected rock mass conditions.

The various prices used in optimisation and design work are summarised in Table 11 and Table 12. Additionally, Table 11 and Table 12 also show the year when resource models were created, and date of Pit Optimisation and Design work completed.

Table 11 Sepon Gold Ore Reserves pit design status

Pit Name		Resource Model	Pit Optimisation Price	Date of Pit Design Optimisation
Discovery	DS	2012	$1,600 /oz	Nov 2012
Phavat North	PVN	2012	$1,600 /oz	Nov 2012

Table 12 Sepon Copper Ore Reserves pit design status

Pit Name		Resource Model	Pit Optimisation Price	Date of Pit Design Optimisation
Khanong	KHN	2009	3.00 $/lb	Jul 2011
Phabing	PHB	2011	3.00 $/lb	Aug 2011
Thengkham South (A,B, C)	TKS	2010	3.00 $/lb	May 2011
Thengkham South D	TKS	2010	3.00 $/lb	May 2011
Thengkham North	TKN	2010	3.00 $/lb	Jul 2011
Thengkham East	TKE	2013	2.80 $/lb	Jun 2013

3.6.2 Geotechnical Parameters

In terms of engineering geology, the Sepon deposits can generally sub-divided into “Soft Rock” and “Hard Rock” domains at any given location. This distinction has implications for the potential mode of failure that might occur. In general terms, the soft rock areas have the potential to fail through the fabric of the intact material or at an interface between soft and hard materials on approximately circular failure surfaces. Hard rock areas tend to fail along structures within the rock mass and not through the fabric of the rock. The latter may not necessarily always be fresh rock but could have some degree of weathering with relict rock structure.

Soft Rock Analysis

Section analyses using 2D limit equilibrium software (GALENA; SLIDE) are undertaken to assess all the soft rock stability issues. The soft rock analysis required the following steps of evaluation:

- The preparation of a detailed geological section for each location of interest, including material, structure and groundwater boundaries involved within each section.
- The evaluation of the appropriate material properties for each of the materials involved.
- Evaluation of the mining stages and estimated groundwater profiles.
- Assess the impact of varying the slope angle, material properties and slope heights where necessary.

Hard Rock Analysis

The appraisal of slope stability in the more competent rock materials involves assessment of the data as follows:

- The identification of the structural orientations within the rock mass derived from the geotechnical drill holes and mapping using stereonet software (DIPS). This is usually conducted within each lithology.
- The assessment of suitable shear strength parameters to the structures on a basis of defect type and lithology was conducted where possible.
- The completion of a kinematic analysis on the data (DIPS) to assess which orientations and combinations of structures have the potential to cause instability. The kinematic assessment attempts to identify the mode of potential instability as well as the potential sensitivity of such structures to outside influences such as groundwater pressure.
- Numerical modelling of the potential failure modes is conducted for the stability of wedge geometries (SWEDGE), potential for planar failure on apparent dip angles (ROCPLANE, Golder code), and toppling modes of failure (TOPPLE). All analyses are deterministic evaluations, the aim being to identify the most important or controlling structures rather than assess the situation purely on a population density basis which may have the potential to mask the most important structures in the rock mass.

Analysis Assumptions

The following assumptions are made for the geotechnical analysis:

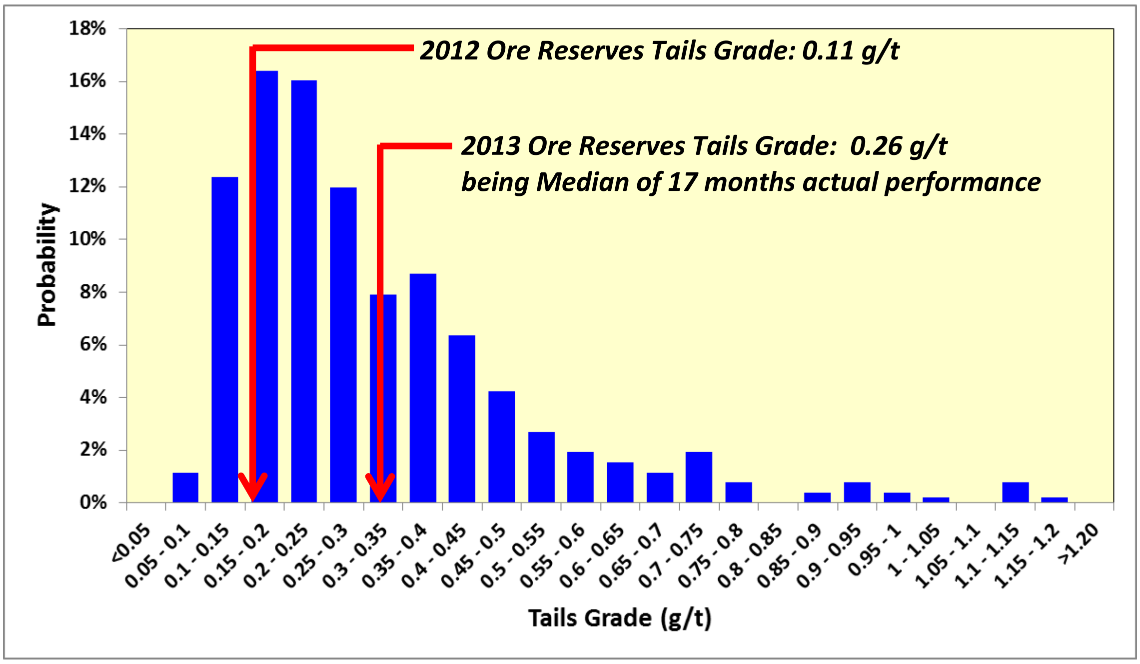
- Most of the rock mass and defect shear strength parameters used are derived from back-analysis of existing slopes and failures.
- All defect populations represent persistent structures.
- Blasting practices will be undertaken during mining such that there will be minimal damage to the rock mass beyond the pit design and hence defect shear strengths will remain at peak values.
- All analyses conducted using a density of 2.5 to 2.7 tonnes/m^3.
- The slopes are assessed using a range of slope heights in each case. (NB: stable slope angles will generally decrease with increasing height).
- Groundwater levels are below the potential failure planes. Where this is unlikely to be the case, then dewatering will be undertaken to drawdown the groundwater levels ahead of the mining front. Modelling of the slopes has therefore been completed assuming dry slope conditions with water drained and depressurised to at least 50 m below the existing topographic surface. This value has been derived based on the hydrogeological data obtained during pumping tests.

3.6.3 Processing (Metallurgical) Recovery Factors

Gold Metallurgical factors and assumptions

The metallurgical recovery was based on the median fixed tail grade of 0.26 g/t. This median fixed tail grade was derived based on historical daily tail grades from Jan 2012 to May 2013. The distribution of this daily tails data is shown in Figure 16. The historical data was provided by site metallurgists and the calculation of median fixed tail grade was completed by MMG Corporate. This logic is applied to all gold deposits in the Ore Reserves.

Figure 16 Gold tails distribution (January 2012 - May 2013)



Copper Metallurgical Factors and Assumptions

The overall copper recovery is based upon a copper leach tails grade and a soluble loss from the CCD washing circuit. The fixed tail is expressed as the copper grade (%) of the tails and the soluble loss is expressed as the percentage of loss from the total copper in the feed.

Metal recovery has been estimated using the formula below, as supplied by James McTiernan (Sepon Superintendent Technical Advisor - Process Engineering). The recovery increases with head grade using the following formula:

Cu recovery (%) = (Cu Grade – Tails Grade (0.38%) / Cu Feed Grade) – Soluble Loss (2.6%)

A median fixed tails grade of 0.38% was derived from analysis of historical daily tails grades from January 2010 to October 2013 as well as routine laboratory test work. Figure 17 shows the distribution and median of the daily tails grade from January 2010 to October 2013.

Figure 17 Copper tails grade distribution (January 2010 - October 2013)

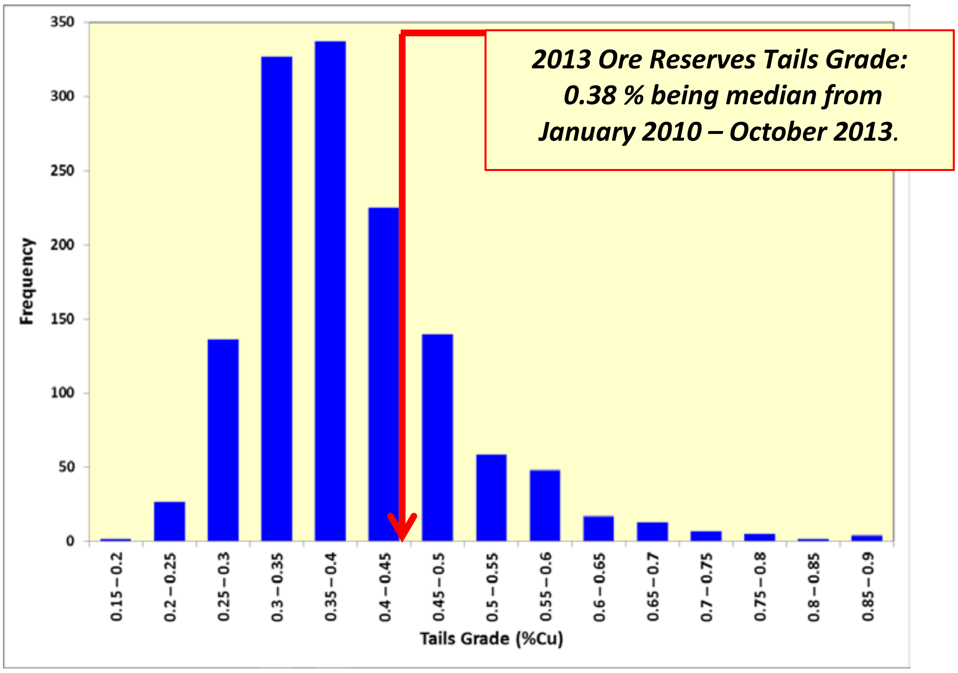
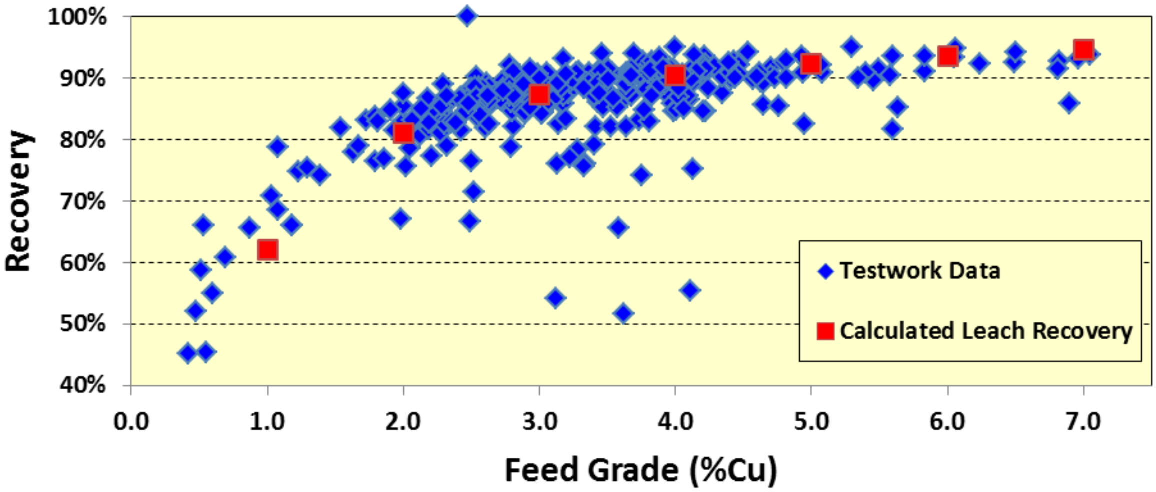


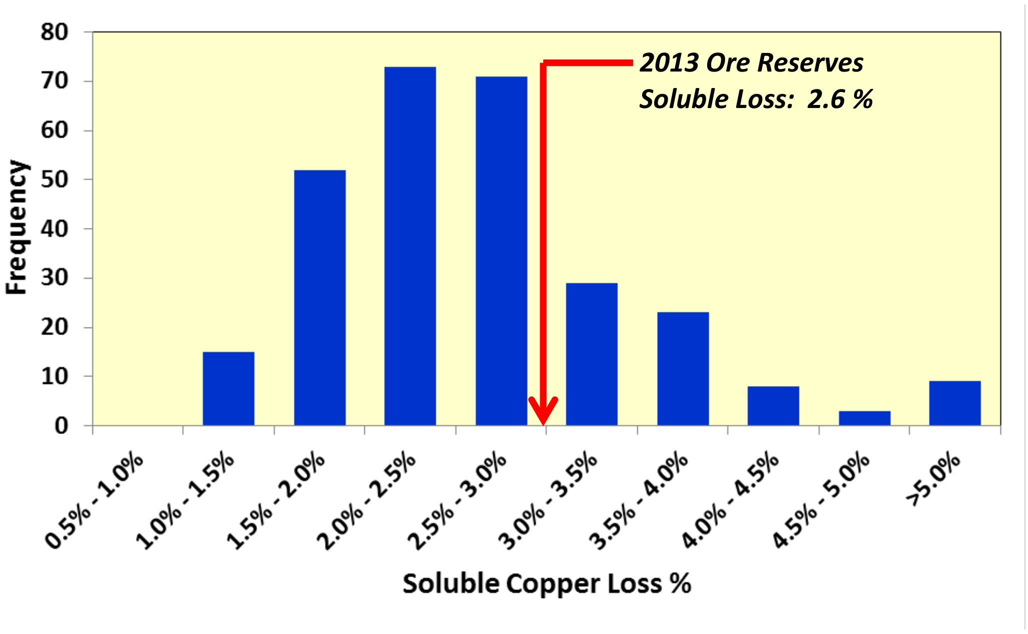
Figure 18 shows the correlation between the calculated leach recovery using a fixed tails grade of 0.38% and laboratory leach recovery test work.

Figure 18 Calculated leach recovery vs. laboratory results



A soluble loss of 2.6% was used in the recovery formula and was based on historical data (January 2013 – October 2013), shown in Figure 19. A copper cementation plant was commissioned in early 2013 which reduced the median soluble loss from 2.9% to 2.6%.

Figure 19 Copper soluble loss distribution (January 2013 - October 2013)



The 2013 copper Ore Reserves estimation used a new method to derive the net acid consumption (NAC) of the ore. Previously an averaged value for each mine area was determined from metallurgical test work. The NAC formula was supplied by James McTiernan (Sepon Superintendent Technical Advisor - Process Engineering) and included a gangue acid consumption (GAC) formula developed by Leonardo Paliza and Michael Hollitt (both of Group Technical Services, MMG).

The GAC formula is used to determine the gangue acid consumption of carbonate ores. It does not apply to sulphide ore which has an assumed GAC of 15kg/t (based on historical data). The formula uses calcium and manganese values combined with constants to estimate the percentage of gangue acid consuming minerals, such as dolomite, manganese minerals, iron and aluminium in the ore. The formula used was:

$$GAC\ (kg/t\ Ore) = (28 + 43.2 \times Ca\% + 15 \times Mn\%)$$

The GAC of carbonate ores is highly variable and greatly influences the cost of processing the ore. Figure 20 shows the historical change and variability in NAC after the introduction of carbonate ores into the feed. Figure 21 shows the distribution of GAC values in the total Sepon Mineral Resource.

Figure 20 Historical net acid consumption (January 2010 - October 2013)

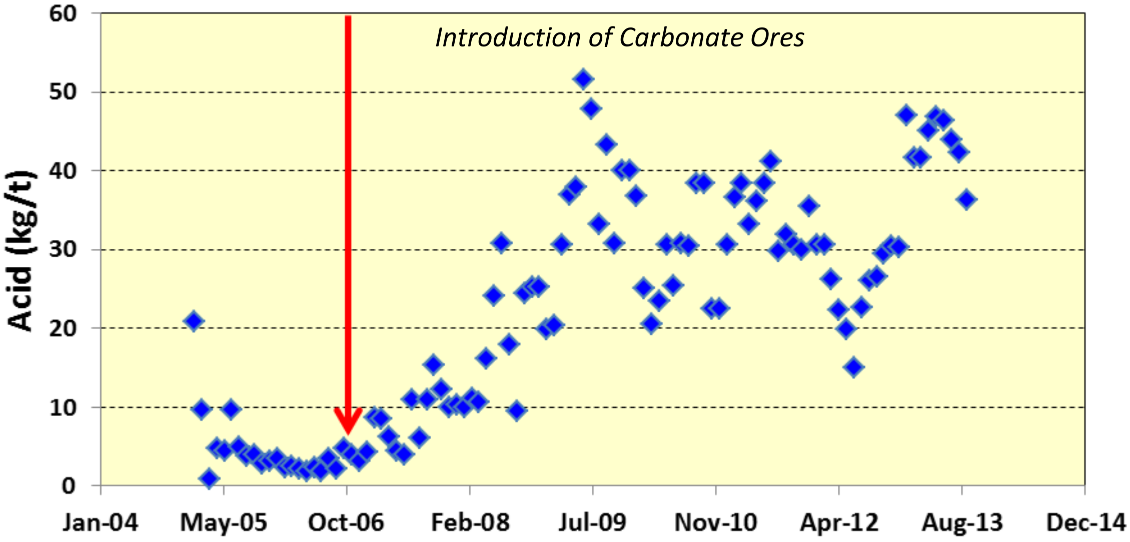
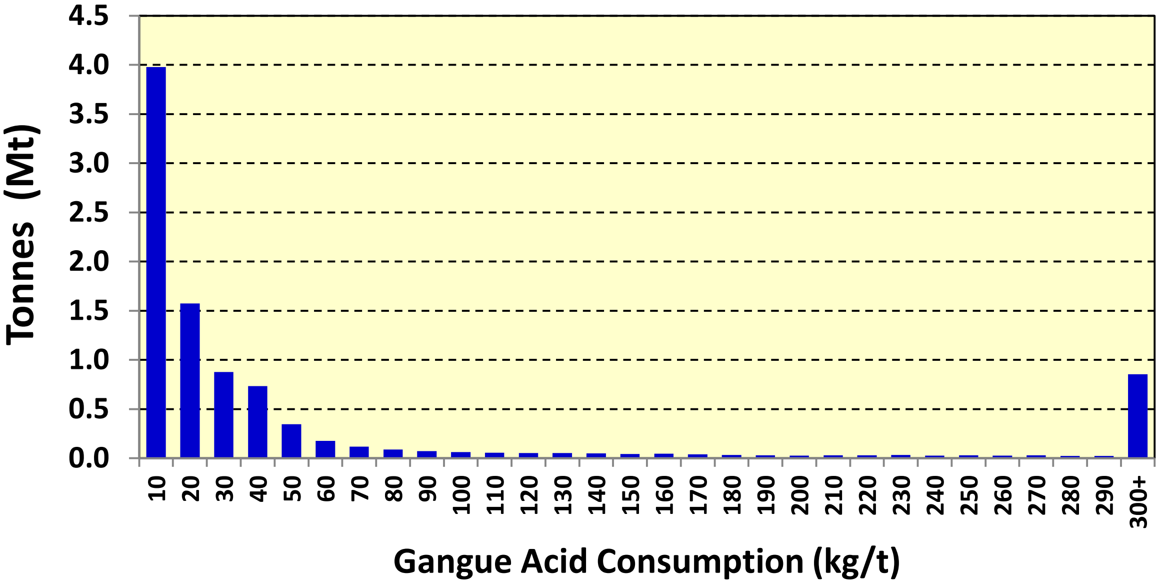


Figure 21 Distribution of GAC of the Mineral Resource



The NAC for sulphide ore was calculated using the following formula:

NAC (kg/t) = GAC (kg/t) + Acid Lost To Tails (kg/t) – Acid Generated From POX (kg/t)

This formula includes consideration for the acid that is lost to tails in addition to the acid that is generated from an autoclave through oxidation of pyrite.

A separate NAC formula was used for carbonate ore which does not include the acid generated from pyrite. The formula used was:

NAC (kg/t) = GAC (kg/t) + Acid Lost To Tails (kg/t)

Further work has been completed on the GAC formula, however these updates were not finalised at the time of the Ore Reserves estimation work. The new formulas are not expected to produce any material changes from the GAC formula used in in the derivation of the 2013 Ore Reserves.

3.6.4 Realised Revenue Factors

The Ore Reserves used information supplied by MMG Corporate in regards to metal prices and economic assumptions. See Section 2.1 for discussion of corporate prices.

Gold Ore Reserves were evaluated using the short-term (CY14) prices. Gold doré is produced on site. Selling costs are included in the total costs used to calculate the break-even cut-off grade.

Copper Ore Reserves were evaluated using medium term prices for the pits with life of less than three years (specifically Khanong, Phabing and Thengkham East) and long term prices for all other pits.

Copper cathode is produced on site limiting the selling costs to US$57/tonne copper metal. Transportation and marketing costs are included in the selling costs.

An LME premium of US$80/tonne copper metal is received on Grade A cathode that is produced. It is assumed that 90% of produced copper will receive this premium.

No exchange rate is used in the Ore Reserves estimate as all expenditure and revenue is reported in US dollars.

A royalty of 4.5 % to be paid to the Government of the Lao P.D.R has been used for both copper and gold.

3.6.5 Costs

The site operating costs used in determination of Ore Reserves were provided by LXML commercial department. Information was sourced from the historical (January 2011 to May 2013) actual operating costs, the 2013 Budget and The Sepon Value Driver Tree model. All costs referenced are in US denomination.

General Site Costs

The site general and administration costs that are used in the Ore Reserves calculations are portioned between gold and copper using a 34%/66% split respectively. Table 13 shows the breakdown of site G&A between gold and copper.

Table 13 2013 Sepon total site G&A costs

Site G&A	Gold (US$M)	Copper (US$M)	Total (US$M)
Total	**19.2**	**37.3**	**56.6**

Mining Costs

Mining costs were included in the Whittle pit optimisation. The pit optimisations used the mining cost at the time of the optimisation, and therefore included costs when the operation used contractors to undertake the mining. In late 2012, Sepon transitioned to an owner operator operation and as such the mining costs have changed slightly, Table 14, however the slight increase in costs does not produce any material changes to the pit shell geometry.

Table 14 Mining Costs – Historical costs used in pit optimisations and current costs

	Contract Mining Costs Used in Pit Optimisations (US$/t)	Current Mining Costs (US$/t)
G&A	2.91	3.65
UXO flitch check		0.26
Maintenance		0.87
Load & Haul	1.57	1.84
Fuel	1.75	
Drill & Blast	0.34	1.95
GC Gold Ore	1.49	1.47
GC Copper Ore	1.31	
Total Waste	**6.57**	**7.10**
Total Gold Ore	**8.06**	
Total Copper Ore	**7.88**	**8.57**

Processing Costs - Copper

The cost and physical actuals from August 2012 to July 2013 were used in the calculation of the production costs for the Ore Reserves cut-off grade calculations. The various cost codes and elements were categorised as follows:

- Ore Variable – Reagents used in the treatment of ore and variable power (e.g. milling);
- Ore Fixed – Consumables used in the treatment of ore and fixed power (e.g. agitators);
- Copper Variable – Power used in electrowinning for copper plating;
- Copper Fixed – All other Solvent Extraction and Electrowinning costs.
- Common Fixed – Management, general contractors and site services (e.g. air, water).

The "Common Fixed" costs were then split between the ore tonnage related fixed costs ("Ore Fixed") and copper metal related fixed costs ("Copper Fixed") based on the ratio between them. The "Ore Variable" costs were then divided by the tonnage treated and the "Copper Variable" costs divided by the copper plated to give a variable unit cost ($/t).

Maintenance costs from the July 2013 forecast were used to calculate the maintenance cost component. Costs were first split between copper, gold and common costs. The common costs were then split between copper and gold based on the ratio between them. The copper maintenance costs were then split between common (services), ore treated, POX and copper metal. It was assumed that 50% of the maintenance costs were fixed and 50% variable. The common costs were again split between the ore treated, POX and copper metal fixed cost based on the ratio between them. The variable component was then divided by the tonnage treated and the copper variable costs divided by the copper plated to give a variable unit cost ($/t).

The water treatment costs in the current operating costs are not representative of the actual cost as the current polishing plant (PP2) is unable to treat the required volume of water from the WTSF (Western Tailings Storage Facility) to maintain the site water balance. Work is currently underway to upgrade PP2 to enable the treatment of the required volume of water; estimated at 2.4Mm^{3}pa. The treatment cost is expected to be 1.5$/m^3. The total annual treatment cost has then been calculated and divided by the annual ore tonnage treated to give a variable unit cost ($/t).

To calculate the cost associated with gangue acid consumption (GAC) it was calculated that at 230tph leach feed Sepon site is typically producing 4.96tph of acid in the autoclave and losing 3.9tph of acid to tailings.

Converting the tph acid production loss into kg/day and then dividing by 230tph leach feed will give the acid production loss in kg/t of ore, the same units as the GAC. Cost data (including freight) and acid consumption data was then used to calculate a unit price for the acid of 0.227$/kg. These elements were then combined with the GAC formula to produce a calculation for the net plant acid consumption (NAC) and the cost in terms of $/t of ore treated.

$$\text{Net Acid Cost (\$/t)} \quad = ((GAC) + 15.2 - 21.7) \times 0.227$$
$$= ((28 + 43.2 \times Ca\% + 15 \times Mn\%) + 15.2 - 21.7) \times 0.227$$

The current cost calculations for sulphide ore (excluding the net acid costs discussed above) are shown in Table 15. These costs assume that when treating sulphide ore that there is adequate pyrite in the feed to meet the autoclave feed requirement.

Table 15 Sulphide ore processing costs

Component	Cost	Unit
Ore Variable	12.3	$/t Ore
Ore Fixed	37,720,000	$/pa
Copper Variable	198	$/t Copper
Copper Fixed	20,230,000	$/pa
Water Treatment	1.9	$/t Ore

The carbonate ore costs, Table 16, are similar to those of the sulphide ore except that costs associated with POX, flotation and the oxygen plant have been removed. This decreased the fixed ore treatment costs which also impact the percentage of common fixed costs attributed to the ore and the copper fixed costs.

Table 16 Carbonate Ore Processing Costs

Component	Cost	Unit
Ore Variable	9.2	$/t Ore
Ore Fixed	20,530,000	$/pa
Copper Variable	198	$/t Copper
Copper Fixed	24,960,000	$/pa
Water Treatment	1.9	$/t Ore

The NAC calculation also has the acid generated in the pyrite removed from the calculation.

$$\text{Net Acid Cost (\$/t)} \quad = ((GAC) + 15.2) \times 0.227$$
$$= ((28 + 43.2 \times Ca\% + 15 \times Mn\%) + 15.2) \times 0.227$$

Processing Costs - Gold

An analysis of the historical actual costs (from January 2012 to May 2013) associated with processing of gold at Sepon are summarised in Table 17.

Table 17 Historical gold processing costs (January 2012 to May 2013)

Component	$/t (ore)
Milling	8.64
Mill Maintenance (Gold)	3.31
Sub-Total	11.95

3.6.6 Cut-Off Grade

The general basis for cut-off grade calculations is a break-even grade. In addition, for gold a check is made for profitability as it was noted that the nature of the tonnage-grade curve had changed from previous years, and a standard break-even grade no longer guaranteed profitability of the operations.

Gold Cut-Off Grade

The cut-off grade (COG) used in the gold Ore Reserves estimate was 0.58g/t site wide.

A fixed tails grade of 0.26g/t was used as discussed in Section 3.6.3.

Costs used for processing and haulage to the processing plant were based on historical actual data. The general and administration costs for site are attributed to gold operations at a proportion of 34%.

At the time of the start of the Ore Reserves work a site policy of allowing the gold plant to run at a total break-even head grade was in place - a policy that was in place to avoid gold plant closure whilst further studies into primary gold project options were being undertaken. This required the operation to run at no lower than 1.2g/t to achieve break-even operations, although during 2012/13 financial year the plant ran below this break-even grade for significant periods of time. Based on interrogation of the whole of site current tonnage-grade curve for Mineral Resources within current pit designs, a site-wide COG of 0.58g/t was determined able to deliver the plant a head-grade of 1.2g/t, noting that higher grade ore was effectively subsidising the lower grade ore.

Towards the end of the Ore Reserves work, the site policy was amended to one that would no longer allow subsidising of unprofitable gold ounces. Hence all individual ore sources that did not achieve the break-even operating grade of 1.2g/t were eliminated from the Ore Reserves.

For reporting gold Ore Reserves from deposits which are predominantly copper deposits, a maximum copper grade of 0.9% Cu is used as a limit to the acceptable ore grade in addition to the gold cut-off grade. This is based on historical performance of copper bearing gold ores at the gold processing plant.

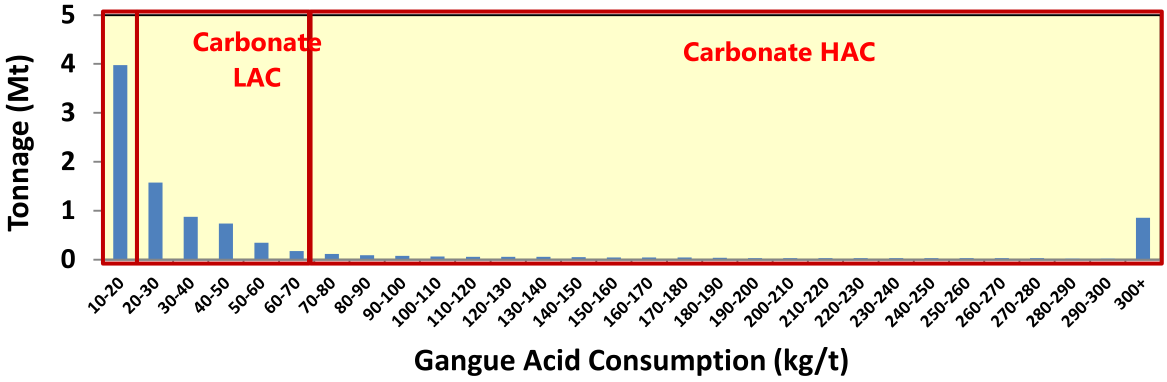
Copper Cut-Off Grade

The 2013 copper cut-off grade (COG) calculation used a new method to derive the net acid consumption (NAC) of the ore, as discussed in Section 3.6.3. Previously an averaged value for each mine area was determined from metallurgical test work and this was used in the cut-off grade calculation.

To calculate the 2013 COG's the carbonate ore was separated into low acid consumption (LAC) and high acid consumption (HAC) material types using a GAC cut-off of 70kg/t and then an average GAC for the material type was determined and used in the COG calculation.

Figure 22 shows the distribution of GAC across the entire Mineral Resource and the separation of the carbonate material into the low acid consumption (LAC) and high acid consumption (HAC) material types. The sulphide ore is assumed to have a GAC of 15kg/t based on historical data and remained as one material type.

Figure 22 GAC distribution and ore type definition split



Three break-even COG's were therefore calculated: sulphide material, carbonate LAC and carbonate HAC material; based on the costs incurred after mining and as "at the pit edge". As all deposits are located at different haulage distances from the copper processing plant, a COG for the three material types was calculated for each deposit. Phabing is a carbonate material only deposit and therefore a sulphide material COG was not calculated.

Table 18 shows the 2013 copper COGs calculated for each deposit and the rehandle costs used in the calculations. Ore from Khanong is hauled directly from the pit to the ROM and does not incur a rehandle cost.

In addition to applying a copper based cut-off grade, a net value script was created to calculate the net value of each block in the Resource block model. This script was used to remove carbonate HAC material where the revenue from the recovered copper is too low to cover the costs incurred due to GAC, even though it is above the calculated COG.

This process was not used on the low acid consuming carbonate material as it assumed that if there is any sub-economic material in this type it can be used to blend with high acid material and be economically processed.

Table 18 Sepon copper Ore Reserves - 2013 cut-off grades

		Cut-Off Grade (%Cu)			Rehandle Cost ($/t)
		Sulphide Ore	**Carbonate Ore LAC**	**Carbonate Ore HAC**	
Khanong	KHN	1.21%	1.26%	2.41%	-
Phabing	PHB	-	1.34%	2.36%	$3.81
Thengkham East	TKE	1.24%	1.32%	1.95%	$1.46
Thengkham North	TKN	1.41%	1.54%	2.65%	$3.00
Thengkham South D	TKS	1.25%	1.32%	2.19%	$1.84
Thengkham South	TKS	1.40%	1.48%	2.51%	$2.31

3.6.7 Mining Factors and Assumptions

Minimum Mining Width

The Ore Reserves estimate is based on current open pit mining practices, which comprise drill and blasting, and small to medium sized excavators in backhoe configuration (90 tonne class) loading articulated off-highway trucks (40 tonne). The minimum mining width used for optimisations and design consideration was 20m, based on the size of current LXML mining fleet.

The fleet has been found to be well suited to the material movement requirements, mined concurrently from multiple locations. This activity is in an environment characterised by periods of high rainfall, steep terrain and much of the material being of high clay content.

Ore from pits to the processing plant are hauled (up to 13.2km in distance) in a separate operation by a local contractor using smaller 10-wheel trucks.

Mining Dilution & Recovery

For the gold Ore Reserves the mining dilution used was 5% and the mining recovery was recently updated from 97.5% to 95%. This was undertaken to qualitatively reflect observations made during ongoing reconciliations with mill production.

For the copper Ore Reserves mining dilution used was 5% and mining recovery used was 95%.

Reconciliation

In preparation for the 2013 Mineral Resource and Ore Reserves estimation a draft MMG Reconciliation Group Standard for Reconciliation was created and a first pass of a reconciliation report for Sepon was produced. The reconciliation information is sourced from this report which used all life-of-pit data for all pits mined within the previous twelve months. Sepon has complete production data back to January 2007 for both grade control and resource block models however due to the current changing of software from MineSight to Vulcan, not all data was accessible. It is envisaged that a more in-depth report including additional historical information will be compiled for 2014 Ore Reserves.

No manipulation of the Ore Reserves were undertaken in order to account for any reconciliation issues.

Grade Control to Resource Model Reconciliation

Table 19 shows the annual reconciliation of the grade control model to the Mineral Resource model in all active mining areas from July 2012 until May 2013. The table shows that the reconciliation is highly variable across deposits and commodity (NB: "F1" is defined as a ratio of the value for the grade control model divided by the value of the Mineral Resource model for the respective parameter: tonnage, grade or contained metal. A value greater than "1" indicates more tonnage grade or metal in the grade control model that in the Mineral Resource model).

Table 19 Sepon annual grade control model to Mineral Resource model reconciliation July 2012 to May 2013

Pit and Metal	Tonnage – F1	Grade – F1	Metal – F1
Khanong Copper	1.15	0.96	1.10
Phabing Copper	1.10	0.87	0.95
Thengkham South D Copper	1.38	0.87	1.20
Phabing Gold	2.09	0.68	1.45
Khanong Gold	0.73	1.27	0.88
Thengkham South D Gold	0.84	1.17	0.94
Thongpiang (gold)	0.90	1.02	0.92
Muang Luang (gold)	1.12	1.01	1.13

Mill Production to Grade Control Model Reconciliation

Figure 23 shows the reconciliation ratio factor ("F2") of gold mill head grade to the grade control model predicted grade for the period from July 2012 until May 2013.

Figure 23 Sepon gold reconciliation: Ratio factor of mill head grade to grade control model grade

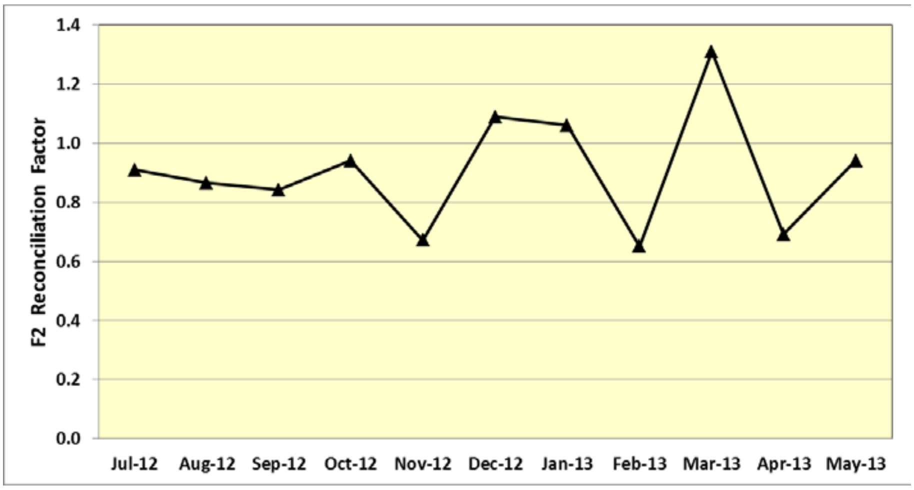
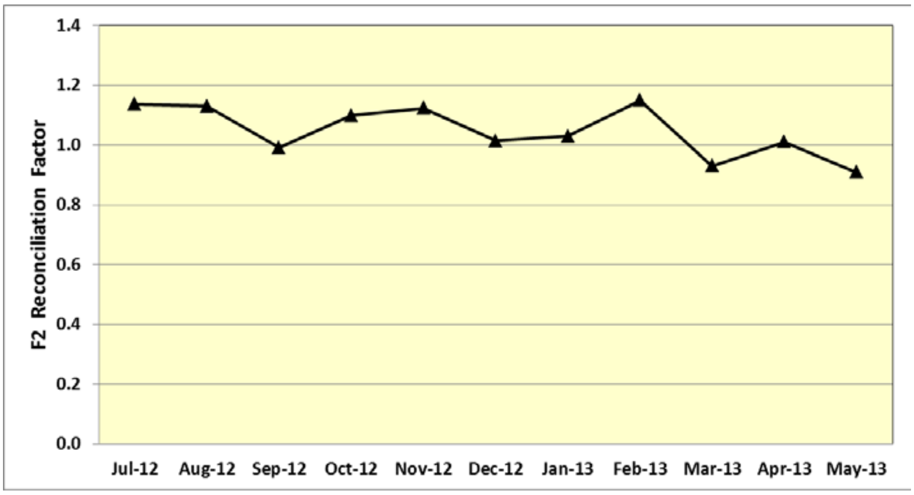


Figure 24 shows the reconciliation ratio factor ("F2") of gold mill head grade to the grade control model predicted grade for the period from July 2012 until May 2013. It should be noted that the presence of a large tonnage (> 5Mt) of copper ore stockpiles present a significant issue in the reconciliation for copper.

In late 2010 a block model of high grade stockpiles was created in order to track the ore grade variability within stockpiles, as some stockpiles have very wide grade bins. Some of the high grade stockpiles contain ore blocks grading from 4.4% copper to over 20% copper. The current method of predicting the daily grade of the plant is to use a combination of block model grades, grab sample grades and visual estimates as an experienced geologist is able to identify high, medium and low grade zones within each stockpile.

Figure 24 **Sepon copper reconciliation: Ratio factor of mill head grade to grade control model grade**



Grade Control Model to Blocked-Out Grade Control Model Reconciliation

This reconciliation evaluation addresses the dilution and loss caused by the blocking out (dig block mark-up) process, comparing the ratio of the value for the blocked out dig blocks to the grade control model data (referred to here as an "F2A" factor).

Currently data is only available for Khanong Pit. With the introduction of Vulcan mining software during 2013, systems have been implemented to ensure this data is captured for all pits to allow future reconciliation and historical data will be migrated across in the future allowing better analysis.

The data was reported by dig bench and broken into the various ore categories. From this a total bench dig block to grade control reconciliation factor was calculated, shown in Table 20. The results show that the dig block design process is efficient in designing ore blocks with minimal dilution. This is due to several factors including the fact that high grade blocks are predominantly diluted with surrounding low grade blocks rather than being diluted with zero grade material.

Dilution for high grade ore shows that ore blocks are being designed with marginally increased tonnes to ensure that all high grade ore is correctly assigned. The copper grade is correspondingly slightly decreased as the ore blocks are diluted however the contained metal is on or above the design.

Dilution for the low grade blocks shows conservatism with reduced tonnes to not include additional dilution of non-current milling ore. The copper grade is slightly increased due to the inclusion of minor amounts of high grade included in the low grade block, usually in order to make the dig block more practical.

The CUDO (copper deferred ore) shows some degree of designed dilution with marginally increased tonnes and metal.

The process of ore blocking is relatively straight forward, well documented and has strict validation of ore block design involving all blocks being peer reviewed before release. Combined with favourable geological geometry, this has ensured designed dilution is minimised.

Table 20 Sepon copper F2A reconciliation: Grade control model to blocked-out model for Khanong (500mRL - 150mRL)

Ore Classification	Copper Tonnes F2A	Copper Grade F2A	Copper Metal F2A
CuHG	1.06	0.99	1.03
CBHG	1.03	0.97	1.00
CuLG	0.99	1.00	0.99
CBLG	0.93	1.04	0.98
CUDO	1.03	0.98	1.01

3.6.8 Environmental

Operations at Sepon are permitted by the Government of Lao (GoL) under the Decree for Environmental Impact Assessment (2010) and the National Agreement on Environment Standards (2010). Individual permits for effluent or air emissions are not required. The environmental and social impact assessments, which are reviewed and approved by the GoL, outline the general framework for controlling impacts to the environment. Detailed mitigation and environmental performance criteria are outlined in the Sepon Environmental Management and Monitoring Plan, which is updated every two years and submitted to the GoL.

An Environmental and Social Impact Assessment (ESIA) is required for the GoL approval of new mine pits in accordance with Environmental Impact Assessment (EIA) Decree No 112-PM. Recent oxide gold pits have been permitted under the Initial Environmental Evaluation (IEE) process.

Sepon mine has Potential Acid Forming (PAF) wastes from copper pits and sometimes from gold pits, especially when a gold pit is located within a copper pit. Stand-alone gold pits do not usually have PAF wastes. PAF wastes from pits are dumped in designated Acid Waste Dumps (AWD) and, both copper pits and AWD have two levels of drainage system and sediment control dams to mitigate any Acid Rock Drainage (ARD). Any PAF waste dumps and/or exposed PAF wastes are systematically encapsulated by Non Acid Forming (NAF) waste during the mining operations.

Sepon produces tailings from gold and copper processing plants. These tailings are managed by two Tailings Storage Facilities (TSF); one in eastern area called TSF1 and the other in western area called Western TSF (WTSF). The design and sustaining management of these two TSF are managed by site civil engineers and consultants.

Current management of all mineral wastes at Sepon is governed by the MMG Waste Rock Management Sustainability Standard and Waste Rock Management Code of Practice.

The governmental environmental approvals are usually in the form of ESIA and/or IEE approval. The following Table 21 shows current licences and certificates with their status.

Table 21 Summary of Licences and Certificates related to Environmental Approvals

Description	Issue Date	Expiry Date
ESIA Certificate – Gold No. 1956	08/10/2002	NONE
ESIA Certificate – Copper No. 02002	30/09/2004	NONE
ESIA Certificate – Gold expansion No. 2083	12/10/2004	NONE
WTSF (TSF2) Certificate No. 743	29/03/2007	13/05/2015
ESIA Certificate – Gold Oxide Expansion No. 573	24/04/2008	13/05/2015
ESIA Certificate – Copper Expansion No. 1387	25/05/2010	NONE
Certificate of settlement plan for 115 Kv Grid Expansion Project	30/01/2009	NONE
IEE Certificate for new Gold Oxide Pits	17/11/2011	NONE

Currently Sepon has a very good record of regulatory compliance. There has been no Level 3 or higher environmental incidents in more than three years. Under the terms of the Environmental Management and Monitoring Plan (EMMP), Level 3 or higher incidents have to be reported to the GoL.

3.6.9 Social

Sepon operations have developed within an unusually complex stakeholder environment where host community settlements occur both within and around the operational footprint. Between 10,000 and 20,000 people reside within the wider operations tenement.

The use of land in current and completed mining areas within ESIA and/or IEE requires compensation agreements and payments according to GoL guidelines. Legal agreement documents between land owners and LXML are stored on site.

The Social Management and Monitoring Plan (SMMP) is the social equivalent of the Environmental Management Plan. It is a guiding document that describes the strategies used by LXML in cooperation with key stakeholders to manage the social impacts and opportunities for local communities affected by mining operations at Sepon. It describes LXML's commitments and obligations and outlines the context for each mitigation objective and strategies for meeting those objectives.

The SMMP has been developed in cooperation with external consultations and dialogue with all levels of Government of Lao PDR. The SMMP is designed according to a government approved template. It was submitted to the government for the first time in tandem with the Environment Management Plan in 2012.

3.6.10 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 22. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 22 JORC Code Ore Reserves assessment and reporting criteria for Sepon 2013 Ore Reserves

Assessment Criteria	**Risk Assessment**	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	Low - Medium	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. MMG updated the Sepon Mineral Resource in June 2013 in accordance with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code) 2012 edition. The Mineral Resources were estimated and compiled for 18 deposits, however, some estimates were unchanged since June 2012 and others updated due to additional drilling. Mineral Resources are modelled using solid wireframes of geological boundaries and/or a minimum 0.3% Cu and 0.3g/t Au-0.5 g/t Au cut-off boundary which approximates the natural break between copper and gold mineralisation and background grades. The Ore Reserves includes ore on stockpiles. The confidence in the calcium and manganese grade is unclear and as this is the basis for the GAC calculation and forms a large proportion of the processing costs it poses a risk to the economics used in the COG calculation. Some risk has been removed though the averaging of the GAC. Further details are discussed in the Mineral Resources Section of this report
Classification	Low	The Ore Reserves estimate is based on the Mineral Resource estimates classified as "Measured" and "Indicated" after consideration of all mining, metallurgical, social, environmental and financial aspects of the operations. All Proved Ore Reserves has been derived from the Measured Mineral Resource where grade control drilling has been carried out on a 5m x 3m pattern and material mined and stockpiled. All Probable Ore Reserves has been derived from either the Measured or Indicated Mineral Resource based on the supporting data. Indicated Resources exist where grade control has not been conducted with drilling generally based on a 25m to 50m spacing. The Ore Reserves do not include any Inferred Mineral Resource in any of Ore Reserves classifications. Gold Ore Reserves are based on the Ore Resource classifications and gold processing plant decommissioning at end of 2014. Gold ore mined up to end of 2014 were only considered as Gold Ore Reserves.

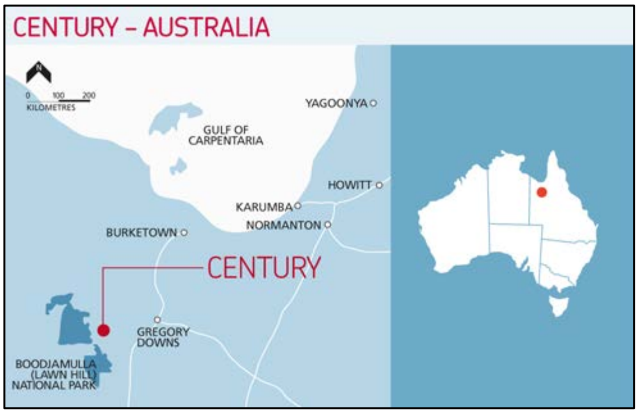
Assessment Criteria	**Risk Assessment**	**Commentary**
Site visits	Low	The Competent Person, Julian Poniewierski, visited the Sepon site on 17-22 June 2013 to inspect the site surface facilities and mine and to liaise with site staff.
Study status	Low	The mine is an operating entity. The Ore Reserves are based on actual operating data.
Cut-off parameters	Low	See Section 3.6.6 for details. Improvement has been made in the method used to calculate copper and gold cut-off grades. Further work is required on site costs.
Mining factors or assumptions	Low	See Section 3.6.7 for details. Further work is required to better quantify and define mining dilution and recovery.
Metallurgical factors or assumptions	Low	See Section 3.6.3 for details. Improvement has been made to better quantify and define the metallurgical inputs used to calculate the Ore Reserves, there is now a sufficient level of data to justify the inputs used.
Environmental	Low	See Section 3.6.8 for details.
Infrastructure	Low	No additional site infrastructure is required to realise the open pit Ore Reserves.
Costs	Low	See Section 3.6.5 for details. More work is required to better understand the general and administration site costs and the mining costs. Although it is felt that all costs have been included in the Ore Reserves estimation, a greater understanding and separation of fixed and variable costs in needed.
Revenue factors	Low	See Section 3.6.4 for details.
Market assessment	Low	See Section 2.2 for details.
Economics	Low	At the cut-off grades used for the Ore Reserves the Sepon operations have robust economics.
Social	Low	See Section 3.6.9 for details. Where community access agreements to land for mining have not been settled, no Ore Reserves have been declared.
Audit or Reviews	–	An internal audit was undertaken in May 2012 by Protivit, however this audit was not competed at a sufficient level of detail or completed by personnel with an in-depth knowledge of the compilation of Mineral Resource and Ore Reserves estimation.
Discussion of relative accuracy/ confidence		A qualitative risk assessment of each discussed item is included with each individual item in the second column of this table. Whilst there are a number of parameters for which there is low confidence, the impact of this uncertainty on the remaining Ore Reserves is such that the likelihood of destroying the robust economics of the remaining Ore Reserves is extremely low.
Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Low	Mountainous with sections of low lying flat terrain.
Climate	Low	Tropical monsoonal climate that consists of a distinct dry season, November – March and a wet season, June – October.
Government Agreements	Low	MEPA Agreement between LXML and GoL.
Hydrogeological Parameters	Low	Active Dewatering is undertaken through the use of production bores and surface pumps. Additionally surface water is controlled though concrete drains and diversions.
Waste Storage (Including Tails Storage)	Medium	The tailings dam is currently at the 295mRL, to provide capacity for the Ore Reserves it needs to be increased to the 306mTL at an approximate cost of $15M US$. Additionally, excess water is currently being stored in the TSF, but approvals for an expanded water treatment plant are in progress. This will allow water levels to be brought back under control.

4. CENTURY OPERATION

4.1 Introduction and Setting

The Century Zinc-Lead-Silver mine is located in the remote lower Gulf region of north-west Queensland, approximately 250km north-west of Mount Isa. The mine is 100% owned by MMG Limited and has been in operation since 1999. Century operations comprises of two sites: the mine at Lawn Hill, and associated concentrate dewatering and ship-loading facilities at Karumba on the Gulf of Carpentaria connected by a 300km slurry pipeline. Century's regional location is shown in Figure 25.

Figure 25 Century Mine location



Century is a conventional open pit mining operation using drilling and blasting with large excavators loading off-highway trucks and produces separate zinc and lead concentrates that are delivered by a dedicated slurry pipeline to the port of Karumba, where they are loaded onto ships and exported.

4.2 Geological Setting

The deposit is hosted within the Lawn Hill Formation, a Middle Proterozoic sequence of shale, siltstone and sandstone overlain by younger Cambrian limestone. Structurally, the deposit is located within the Page Creek syncline and is terminated to the east by Cambrian limestone and faults associated with the Termite Range Fault. Magazine Hill Fault and Nikki's Fault define the southern and northern boundaries respectively. The western boundary is truncated by Cambrian limestone and by present day surface at the Discovery Hill gossan. The mineralisation is divided into northern and southern blocks by the north dipping normal Pandora's Fault (Kelso, I., et.al, 2001).

High grade mineralisation at Century occurs mostly in black shales, dominantly as fine grained sphalerite and galena lamellae with siderite and minor pyrite. The black shale units are separated by less mineralised, siderite rich, siltstone horizons. The deposit is unmetamorphosed, only weakly deformed, and displays excellent lateral stratigraphic and grade continuity apart from small-scale fault dislocations (Broadbent, G.C., et.al., 2002).

4.3 Mineral Resources - Century

4.3.1 Results

The Century Mineral Resource estimate for June 30 2013 was carried out by Quantitative Group (QG) utilising geological interpretations and data provided by MMG geologists. The approach varied from previous years following an independent review of the Century Resource by QG in 2012.

The MMG Century Mineral Resource is based on the June 2013 Geological block model (2013_Geological_Model_v6.bmf). This model was built from stratigraphic and structural surfaces generated by MMG Geologists from diamond drillhole data.

The Mineral Resource estimate is based on drilling conducted between 1990 and 2013. The three dimensional block model was generated in Vulcan software, with grade estimation carried out in Isatis using the Ordinary Kriging estimation method.

The Eastern Fault Block (EFB) is a mega-clast of Century style mineralisation within the Thorntonia Limestone and located underneath the current Run of Mine (ROM) stockpiles. The Eastern Fault Block Mineral Resource is based on the 2013 Eastern Fault block model (efb2013.bmf). This model was completed by MMG Century geologists and is based on the parameters set out in the 2008 Mineral Resource generated by Snowden Mining Industry Consultants (Snowden) for the June 2008 Mineral Resource Statement. There has been no mining in this area. The updates to the model are based upon drill data from the 2012 campaign which further tested the extent of the Eastern Fault Block mineralisation.

Silver King, a small lead deposit previously reported, has been removed from the Century area Mineral Resource as it was not compliant with JORC (2012) reporting requirements.

The Mineral Resource at the MMG Century Mine as of the 30 of June 2013 is summarised in Table 23. The Century Mine Open Pit Resource is reported within the current Final Pit Shell design.

Table 23 Century Mineral Resource as of the 30 June 2013

Century Mineral Resources

					Contained Metal		
Century and East Block **3.5% Zn cut-off grade**	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
Century							
Measured	0.1	8.4	1.3	27	10	2	0.1
Indicated	17	10.0	1.5	37	1,700	255	21
Inferred	-	-	-	-	-	-	-
Total	**17**	**10.0**	**1.5**	**37**	**1,710**	**257**	**21**
Century East Block							
Measured	-	-	-	-	-	-	-
Indicated	0.5	12.4	1.0	49	59	5	0.8
Inferred	-	-	-	-	-	-	-
Total	**0.5**	**12.4**	**1.0**	**49**	**59**	**5**	**0.8**
Total Contained Metal					**1,770**	**260**	**22**

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.

Competent Person:

Michael Smith (Member of AusIMM(CP), employee of MMG)

The Century Open Pit Mineral Resource has decreased by 700kt of zinc, 120kt of lead and 7 Moz of silver (Table 24).

Table 24 Century Mineral Resource change 2012 to 2013 at 3.5% Zn

	Tonnes (Mt)	**Zn (%)**	**Pb (%)**	**Ag g/t**	**Zn metal (000't)**	**Pb metal (000't)**	**Ag metal (Moz)**
2012 Mineral Resource	21	11.6	1.8	43	2,500	400	29
2013 Mineral Resource	18	10.0	1.5	37	1,770	278	22
Variance	**-3**	**-1.6**	**-0.3**	**-6**	**-731**	**-122**	**-7**
% Variance	**-17%**	**-13%**	**-17%**	**-14%**	**-29%**	**-30%**	**-24%**

Resource depletion was completed by mining out blocks from the 2013 Geological model using pit shells generated from survey pick-ups and Vulcan software. The depleted Mineral Resource reported was contained within the end of June 2013 pit shell after the areas mined up to the end of June 2013 were depleted from the 2013 Geological model.

The Mineral Resource that remained underneath the pit shell in areas that were completely mined out during the 12 month period was also depleted. This is a factor of discrepancies between the model and orebody geometry. The 2012 pit shell was adjusted to encompass the updated footwall surface defined by the 2013 drilling campaign. The pit footprint and final walls remain unchanged, but intra-stage final depths were subject to local adjustments to ensure full extraction of the Mineral Resource in Ore Reserves is possible.

The change in stockpiles between the June 2012 Mineral Resource and the June 2013 Mineral Resource was accounted for based on: survey volumes, a standard stockpile density of 1.95 tonnes per cubic metre, and estimated mined grades including mining dilution adjustments.

The changes between the 2012 and 2013 Mineral Resources are shown in waterfall graphs in Figure 26 to Figure 30.

Figure 26 Century Mineral Resource waterfall chart (total tonnes) 2012 - 2013 Mineral Resource Estimates, excluding Eastern Fault block

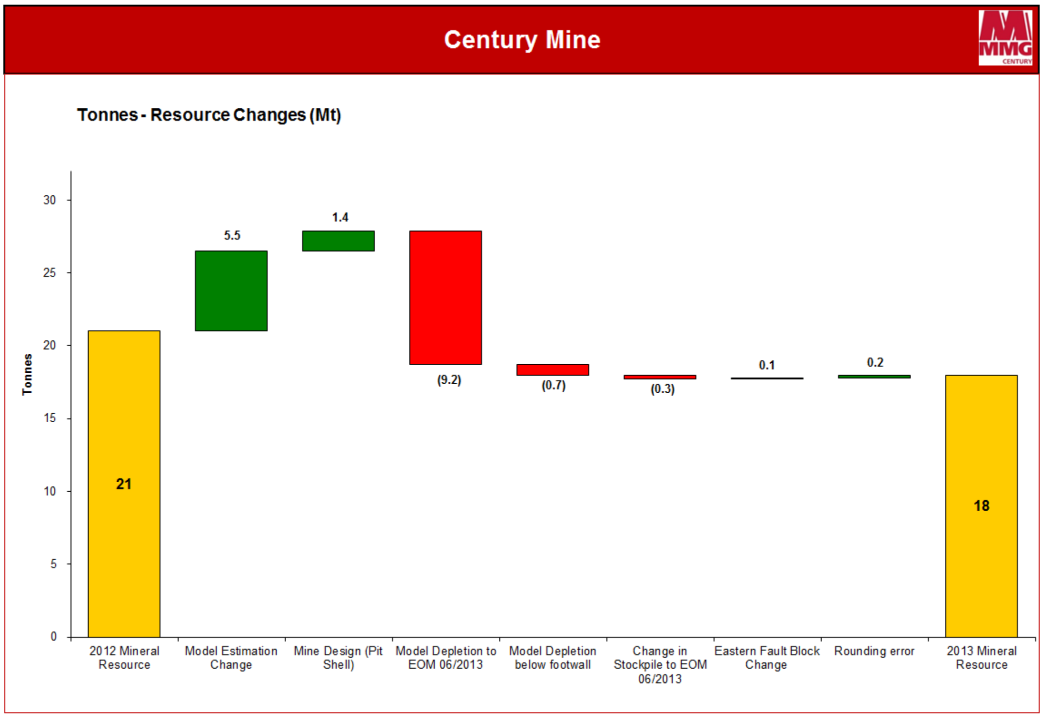


Figure 27 Century Mineral Resource waterfall chart (contained zinc metal) 2012 - 2013 Mineral Resource Estimates, excluding Eastern Fault block

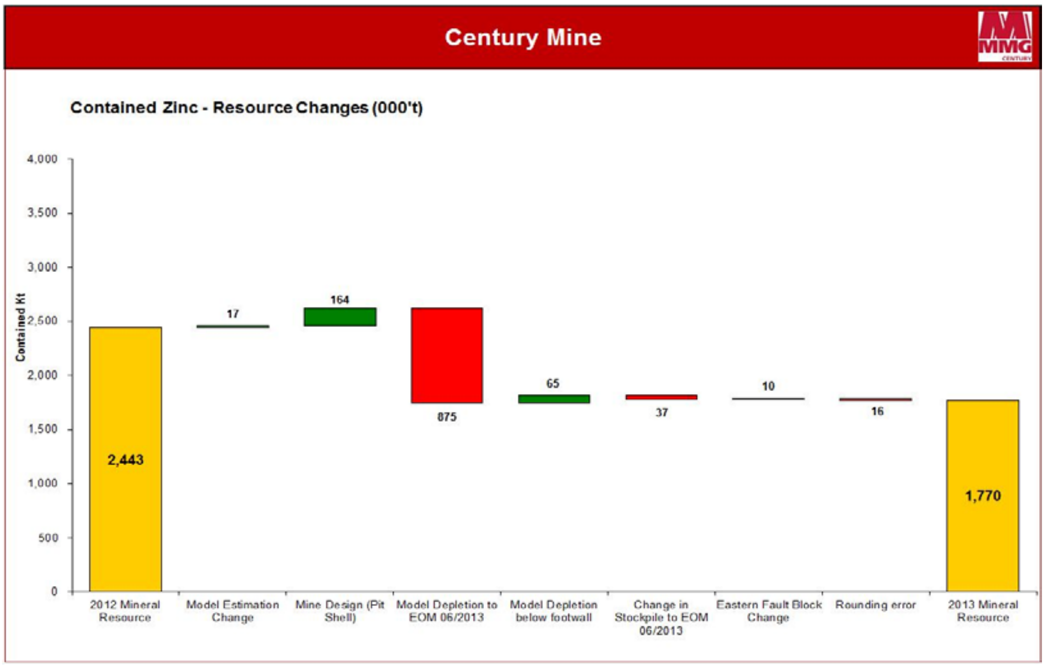


Figure 28 Century Mineral Resource waterfall chart (zinc grade (%)) 2012 - 2013 Mineral Resource Estimates, excluding Eastern Fault block

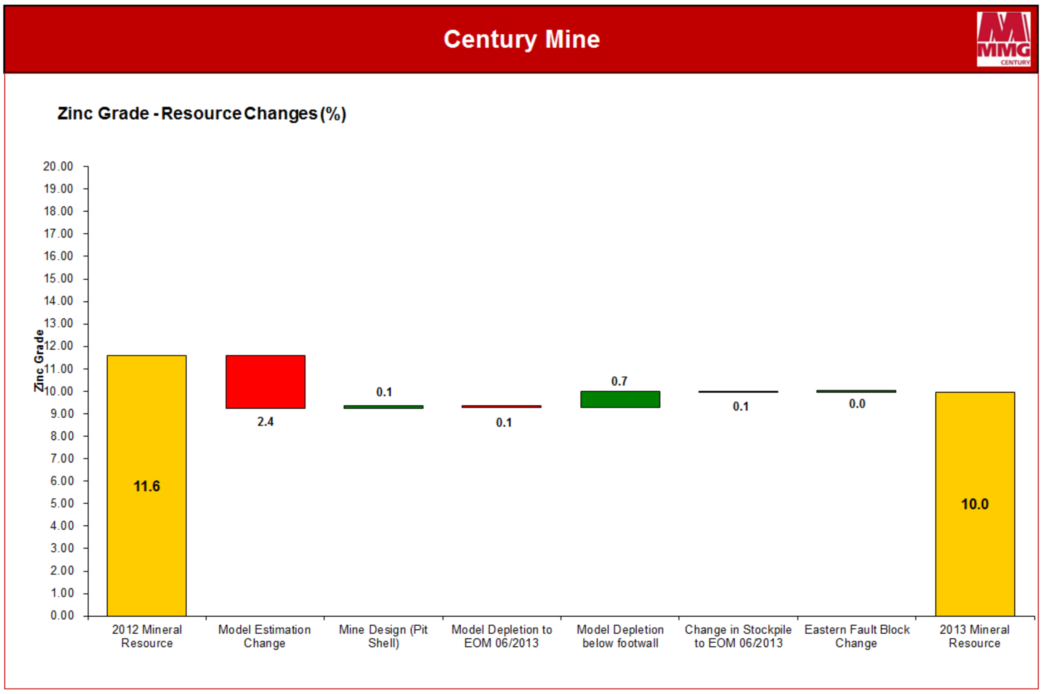


Figure 29 Century Mineral Resource waterfall chart (contained lead metal) 2012 - 2013 Mineral Resource Estimates, excluding Eastern Fault block

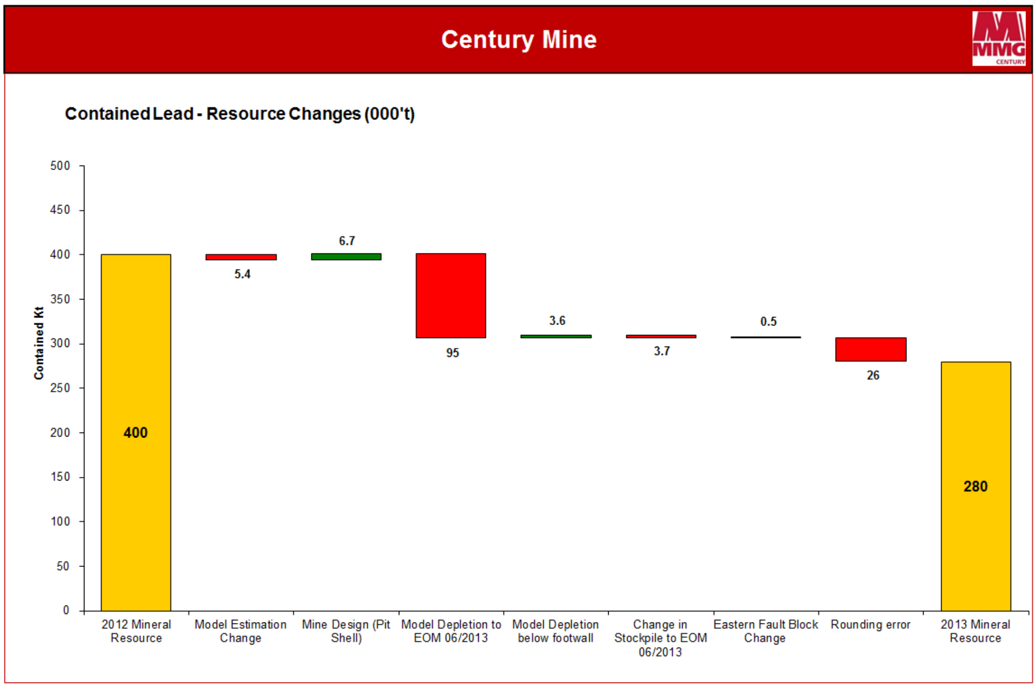
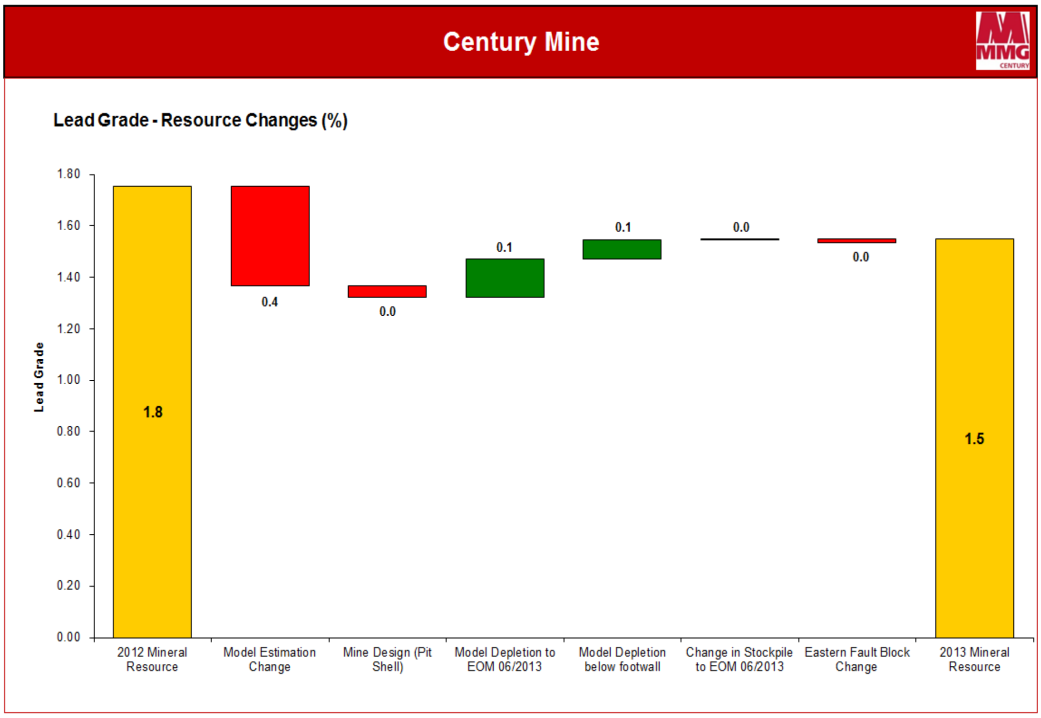


Figure 30 Century Mineral Resource waterfall chart (lead grade (%)) 2012 - 2013 Mineral Resource Estimates, excluding Eastern Fault block



4.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Michael Smith, confirm that I am the Competent Person for the Century Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Century Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Century Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in the supporting documentation relating to Mineral Resources as undertaken by Mike Stewart, employed by Quantitative Group (QG), and compiled and reviewed by Michael Smith, Mine Technical Services Manager, Century.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Century Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Michael Smith – 26/11/13

Claudio Coimbra (Witness)

4.4 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Century Mineral Resources.

Table 25 Checklist of assessment and reporting criteria for Century Mineral Resource

Criteria	Status
Section 1 Sampling Techniques and Data	
Drilling techniques	▪ The geological interpretation was based on 534 drillholes, of which 436 contained valid intersections and were used in grade estimation. ▪ 8 diamond drillholes intersect the Eastern Fault Block mineralisation. ▪ The drilling types used in the resource estimate include NQ and HQ size diamond drill core.
Drill Sample Recovery	▪ Drill core recovery within the mineralised sequence is approximately 100%. ▪ All diamond drillholes have had core recovery recorded, and other basic geotechnical information.
Logging	▪ Detailed geological logging was completed on all drillholes using the refined stratigraphy developed by Solid Geology (2002). Logs were then uploaded into the GBis database. ▪ A geophysical logging system is used to log production blast holes and has also been used to log deeper RC drillholes. This system measures natural gamma radiation and magnetic susceptibility to differentiate units within the mineralised zone.
Sub-sampling techniques and sample preparation	▪ Half-core samples were taken using a diamond core saw. ▪ Sample intervals were confined to geological boundaries and have depth and unit information recorded. ▪ In the 1990 to 1995 drilling campaigns; – Sample preparation of all drill samples were completed by Amdel at their Mt. Isa facility. – Samples of approximately 4kg to 5kg were dried in aluminium trays. – Samples were jaw crushed to 6mm. – Samples were pulverised to 70% passing 75µm in a mixer-mill with three concentric rings. – A split of 200g was bagged and sent for assay. – The sample residue was re-bagged and stored at the CRAE facility in Canberra. ▪ In the 1999 to2007 drilling campaigns; – Sample preparation of all drill samples were completed at the Century mine laboratory. – Samples were dried and the entire sample crushed to 5mm through a jaw crusher. – Samples were riffle split to produce 300g for pulverising. – Sample pulverised to minus 200µm in a ring mill. ▪ In the 2013 drilling campaign; – Sample preparation of all drill samples was completed at ALS laboratory in Townsville – Samples up to 7kg were weighed and dried at 105°C as received in their calico bags. – Samples were crushed to nominal 6mm in Jacques jaw crusher. – Samples greater than 3.3kg were split down to 2kg to 3kg using a riffle splitter. – Up to 3kg of sample was then pulverised in LM5 pulveriser to 85% passing 75µm. – 300g to 400g of pulverised sample was collected for analysis.
Quality of assay data and laboratory tests	▪ Samples from 1990 to 1995 were assayed through Amdel Laboratories in Mount Isa and Adelaide, Analabs in Townsville and Genalysis Laboratory Services, Perth. Analytical methods used were Atomic Absorption Spectrometry (AAS), Induced Coupled Plasma Optical Emission Spectroscopy (ICP-OES) and Leco furnace methods. ▪ After 1999 samples were analysed in the Century mine laboratory using a combination of x-ray fluorescence (XRF – Zn, Pb, Fe, Mn, SiO_2, S), atomic absorption spectrometry (AAS – Ag) and Leco furnace (total carbon) methods, and each month a sub-set of samples are despatched to ALS in Brisbane for check assay. Each internal and external batch of samples is accompanied by selected Standard reference sample material. ▪ Samples from 2013 were prepared at the Townsville laboratory of Australian Laboratory Services (ALS), with the pulps being transferred to the Brisbane laboratory for analysis using a combination of x-ray fluorescence (Zn, Pb, Fe, Mn, SiO_2, S), AAS – Ag and Leco furnace (total carbon and organic carbon) methods.

Verification of sampling and assaying	▪ The QAQC controls for all sets of drilling campaigns included: – The insertion of a variety of laboratory certified standard samples based on Century mineralisation, – Duplicate samples of quarter core, with the exception of the 2013 campaign, – Duplicate samples of 5mm splits (Century laboratory only), – Submission of pulps to off-site "umpire" laboratory, – Repeats of assayed pulps.
Location of data points	▪ Collar co-ordinates of all drillholes were determined to an accuracy of 0.1m in all directions by a licensed surveyor. ▪ Down-hole surveys were taken at 30m intervals for all inclined drillholes and 30% to 40% of vertical holes using single-shot Eastman camera equipment.
Data spacing and distribution	▪ Drillhole collars are located on an approximate grid pattern with a spacing of between 50m and 70m on north-south sections across the deposit. ▪ An in-fill drilling campaign was carried out in 2013 to reduce the drillhole spacing in the remaining Mineral Resource to 30m to 40m. ▪ Eastern Fault Block drillhole spacing varies from 25m to 50m.
Orientation of data in relation to geological structure	▪ The Century mineralised sequence dips at between 5 and 25 degrees over most of the deposit area, with dips up to 70 degrees around the margins. ▪ The Eastern Fault Block mineralisation dips approximately 65 degrees toward the north-north-west. ▪ The majority of drillholes are therefore vertical with inclined drillholes targeted at the more steeply dipping zones.
Audits or reviews	▪ In 1996 Mining and Resource Technologies (MRT) completed data validation and review of the initial drilling completed by CZL from 1990 to 1995. ▪ In 2002 and 2003 Snowden completed reviews on the data quality and QAQC procedures for geology sample data from 1999 to 2003.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The Century Mine Lease is ML 90045/90058. ▪ Tenure is held by MMG Century for 40 years from 19th September, 1997. ▪ Lease expiry date is 19th September, 2037.
Exploration done by other parties	▪ Significant exploration has been completed by various companies and individuals in the known Burketown mineral field over 100 years since the initial discoveries of lead and silver mineralisation. ▪ No significant exploration drilling results in the 2013 reporting period.
Geology	▪ The deposit is hosted within the Lawn Hill Formation, a Middle Proterozoic sequence of shale, siltstone and sandstone overlain by younger Cambrian limestone. ▪ Structurally, the deposit is located within the Page Creek syncline and is terminated to the east by Cambrian limestone and faults associated with the Termite Range Fault. ▪ Magazine Hill Fault and Nikki's Fault define the southern and northern boundaries respectively. ▪ The western boundary is truncated by Cambrian limestone and by present day surface at the Discovery Hill gossan. ▪ The mineralisation is divided into northern and southern blocks by the north dipping normal Pandora's Fault.
Drillhole information	▪ No exploration results to report for the 2013 reporting period.
Data aggregation methods	▪ No exploration results to report for the 2013 reporting period.
Relationship between mineralisation widths and intercept lengths	▪ No exploration results to report for the 2013 reporting period.
Diagrams	▪ No exploration results to report for the 2013 reporting period.
Balanced reporting	▪ No exploration results to report for the 2013 reporting period.
Other substantive exploration data	▪ No exploration results to report for the 2013 reporting period.
Further work	▪ Exploration is ongoing on the Century Mine Lease, with significant results reported as required.

Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	▪ All drilling, sampling, assay, density and geological data previously stored in Microsoft Access databases was migrated to a central GBis database in 2011, this was the source for all drilling data used in the 2013 Mineral Resource. ▪ The geology database was validated and audited by independent parties; Snowden and MRT prior to the migration. ▪ All data was entered manually into Excel spread-sheets with look up tables, and then uploaded into GBis.
Geological interpretation	▪ Mineralisation in the deposit is strata bound and confined by stratigraphy, which is well understood and well defined throughout the deposit. ▪ The interpretation of the deposit geology was based on all available drilling information at the time of model generation. ▪ In addition to the information gained from geological logging, down-hole dip-metre information was used to verify structural interpretations and geophysical probing of blast holes in the mineralised sequence are used in the geological interpretation along with mapping data collected during mining. ▪ There are a number of bounding structures which limit the model extents: – Magazine Hill Fault forms the southern boundary of the deposit. It has an east-west strike, dips to the north and has a North Block down offset. – Nikkis Fault forms the northern boundary of the deposit and is the northern wall of the graben in which the deposit lies. The fault strikes east-west and dips steeply to the south. – Pandoras Fault is the major boundary between the North and South Blocks. Vertical offsets range from 5m at the eastern margin to greater than 200m at the western edge of deposit. – The overlying Cambrian Limestone forms the boundary to the deposit along the eastern margin of the South and North Blocks of the deposit. It also forms the boundary to the western margin of the North Block. ▪ There are also several modelled internal structures that displace the ore by various amounts including: – Geckos Fault – Rayners Fault – Homers Fault ▪ The Eastern Fault Block (EFB) mineralisation is similar in style to the Century main mineralisation. EFB is fault bounded on the lower, southern margin and limited by haematite mineralisation near-surface, toward the north.
Dimensions	▪ The Century mineralisation extends from 26,850N to 28,350N, 46,400E to 47900E and 1125RL to 814RL. ▪ Eastern Fault Block extends from 48043N, where it outcrops in the east wall of the pit to 48223N, 27130E to 27223E and 1129RL to 1055RL.
Estimation and modelling techniques	▪ Interpretation and construction of stratigraphic surfaces bounding the pre-defined Upper Ore Zone, Lower Ore Zone, the Interburden Waste unit and the 'Marginal' 165, 155 and 145 units was used for the estimation. ▪ The steps involved in the estimation are listed below: – A volume block model was created in Vulcan software from the surfaces defined above. Block dimensions are fixed in easting and northing, but block height can vary in Z. Each unit is represented by a single block in the Z direction. – Importation of 3D block centroids to Isatis. – Creation of a 20m x 20m 2D grid file for each unit. – Migration of unit coding from 3D block centroids to 2D grid. – Manual checking and editing of coding, including definition of a 'unit' code corresponding to the units defined above, and flagging of the 'ore' and 'waste' stratigraphic members inside these units. – Creation of 'vein' composites across the full width of the unit. Ore and waste members are composited separately, resulting in co-located 'ore' and 'waste' composites for each unit in each drillhole. A manual step is required to ensure that a unit code and zero length are given to units where either a waste or ore member is missing. – Import composites to Isatis, migration of co-located ore and waste composites to a single file. – Calculation of ore and waste proportion (e.g. ore proportion = ore length/(ore length + waste length)). – Convert 3D composites to 2D by dropping Z coordinate. – Variography – definition of variogram models for ore proportion, waste proportion, ore grade variables and waste grade variables; – Quantitative Kriging Neighbourhood Analysis (QKNA) to determine estimation search parameters. – For each unit, for ore and waste proportion and the grades of Zn/Pb/Ag/Fe/Mn/S for 'ore' and 'waste' components were estimated separately.

<table>
<tr><td></td><td>– For each unit, estimate C (total) and C (organic).
– Validate estimates.
– Migrate proportion and grade estimates from 2D grid, back to 3D block centroids. Export to ASCII and reload to Vulcan block model.
– Run Vulcan script to set missing grade values and calculate stoichiometric sulphide mineralogy and bulk density based on Pb, Zn, S and Fe estimates.
▪ Century block model origin and extents are presented in Table 26.
Table 26 Century block model origin and extents
<table>
<tr><th>Dimension</th><th>Origin</th><th>Extent</th><th>Parent cell size (m)</th><th>Sub-cell size (m)</th></tr>
<tr><td>Easting</td><td>45900</td><td>2400</td><td>5</td><td>5</td></tr>
<tr><td>Northing</td><td>26300</td><td>2400</td><td>5</td><td>5</td></tr>
<tr><td>Relative Level</td><td>800</td><td>400</td><td>400</td><td>0.05</td></tr>
</table>
▪ The Eastern Fault Block (EFB) mineralisation was estimated by inverse distance squared interpolation within the defined units of the EFB. The EFB block model origin and extents are presented in Table 27.
Table 27 Eastern Fault Block, block model origin and extents
<table>
<tr><th>Dimension</th><th>Origin</th><th>Extent</th><th>Parent cell size (m)</th><th>Sub-cell size (m)</th></tr>
<tr><td>Easting</td><td>48020</td><td>800</td><td>10</td><td>5</td></tr>
<tr><td>Northing</td><td>26650</td><td>800</td><td>10</td><td>5</td></tr>
<tr><td>Relative Level</td><td>1200</td><td>600</td><td>100</td><td>0.05</td></tr>
</table></td></tr>
<tr><td>Moisture</td><td>▪ Tonnes have been calculated on a dry basis.</td></tr>
<tr><td>Cut-off parameters</td><td>▪ The Mineral Resource is reported above a 3.5% Zn cut-off.
▪ No assumptions were made regarding cut-off grade for the Mineral Resource due to the deposit being restricted to certain strata, which are themselves constrained by structural surfaces and topography.</td></tr>
<tr><td>Mining factors and assumptions</td><td>▪ No mining factors or assumptions have been applied to the Mineral Resource.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>▪ No metallurgical factors or assumptions have been applied to the Mineral Resource.</td></tr>
<tr><td>Bulk density</td><td>▪ The process of estimating bulk density involved calculating the stoichiometric density of composites, applying a correction factor for porosity based on grab sample results and then estimating the corrected stoichiometric density using Ordinary Kriging, which is summarised below:
– Select samples that have assay results for all elements required in the stoichiometric equation.
– Composite these samples for intervals that have been coded as having valid samples.
– Calculate the stoichiometric density for these samples.
– Apply correction factor derived from the grab sample bulk density to convert the stoichiometric density into bulk density.
– Derive variograms for bulk density for each Unit.
– Estimate bulk density into block model.</td></tr>
<tr><td>Classification</td><td>▪ The Mineral Resource has been classified according to the guidelines of the JORC code (2012) and takes into account the drillhole spacing, estimation results and the internal and bounding structures of the deposit. The model variable class has been coded as either Measured (class = 1), Indicated (class = 2) or Inferred (class = 3).</td></tr>
<tr><td>Audits or reviews</td><td>▪ Quantitative Group (QG) carried out an independent review of the Century Mineral Resource model in 2012. Based on the recommendations of this review the modelling approach was altered in 2013. The new QG Mineral Resource model is the basis of this report.</td></tr>
<tr><td>Discussion of relative accuracy/confidence'</td><td>▪ Estimation confidence is largely a function of data density and the variogram model applied. Within the area of remaining resource, the distribution of drill holes analysed for zinc is generally fairly even across the Mineral Resource. The estimation quality as quantified by metrics such as slope of regression throughout the deposit is high. Along the south margin of the Gecko block, and the southern and western margin of Pandora, the estimation confidence is somewhat lower (slope of regression in range 0.90-0.95) because grades are extrapolated beyond drillholes. Overall however, the quality of zinc estimates in the area of remaining resource is high, and will be similar for all domains, and for the other value variables (lead and silver), because the informing data is the same, and variograms very similar.
▪ In QG's opinion, the estimation quality in the remaining Mineral Resource generally supports a classification of Indicated. This level of Mineral Resource classification is easily supported by estimates of the value variables (zinc, lead and silver), which have high confidence. However, this statement presumes that the geometry of the principal ore units will be reliably identified during the grade control phase prior to mining.</td></tr>
</table>

4.5 Ore Reserves - Century

4.5.1 Results

This June 2013 Ore Reserves statement is based on the June 2013 Mineral Resource block model (2013_Geological_Model.bmf). This model was completed by Quantitative Group Pty Ltd (QG) and is built from stratigraphic and structural surfaces generated by MMG geologists from diamond drillhole data and re-evaluating the dilution criteria calculations.

The 2013 Century Ore Reserves is summarised in Table 28.

Table 28 2013 Century Ore Reserves tonnage and grade (as at 30 June 2013)

Century Ore Reserves

					Contained Metal		
	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
Proved	0.1	8.4	1.1	27	10	1	0.1
Probable	14	9.8	1.5	36	1,380	200	16
Total Ore Reserves	**14**	**9.8**	**1.5**	**36**	**1,390**	**200**	**16**

Ore Reserves are generally rounded and reported to 2 significant figures to reflect confidence in estimates. Totals may differ due to rounding. Contained metal does not imply recoverable metal.

Details of relevant modifying factors used in estimating Ore Reserves are given in the Technical Appendix published on the MMG website.

Competent Person:

Moses Bosompem (Member of AusIMM, employee of MMG)

*

The three major differences from the 2012 Ore Reserves are:

(i) Use of the 2013 Mineral Resource model rather than the April 2011 Mineral Resource block model.

(ii) A downgrade of the classification of all in-pit Ore Reserves to all Probable classification – no Proved Ore Reserves in the pit is stated. This is primarily due to the number of high risk modifying factors (in particular the uncertainty in the ore dilution parameters, the reconciliation issues and the estimation quality in the remaining resource areas of Gecko and Pandora) as discussed in Section 4.6.7 of the JORC Assessment and Reporting Criteria discussion.

(iii) Updated footwall surface as a result of the in-fill drilling program conducted in the first quarter of 2013.

The Proved Ore Reserves stated are all associated with stockpiled material as detailed in Table 29 below.

Table 29 Stockpiled 2013 Century Proved Ore Reserves tonnage and grade (as at 30 June 2013)

					Contained Metal		
	Tonnes (Mt)	**Zinc %Zn**	**Lead %Pb**	**Silver Ag (g/t)**	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
ROM	0.05	7.7	0.6	18	3.7	0.3	0.03
Crushed Ore	0.07	8.9	1.4	33	6.5	1.0	0.08
2013 Total Ore Reserves	**0.12**	**8.4**	**1.1**	**27**	**10.2**	**1.3**	**0.11**

4.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (“2012 JORC Code”).

Competent Person Statement

I, Moses Bosompem, confirm that I am the Competent Person for the Century Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Century Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Century Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Ore Reserves.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Century Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Moses Bosompem – 26/11/13

Damian O’Donohue (Witness)

4.5.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 30.

Table 30 Contributing Experts – Century Mine Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
David Purdey, Principal Mining Engineer QG Consulting Pty Ltd	Cut-off Grade Optimisation; Pit Optimisation; Reserves Reporting; Auditing; Reconciliation Systems
Mike Stewart, Senior Principal Consultant QG Consulting Pty Ltd	Geological Block Model
Damian Sullivan, Senior Business Analyst MMG Ltd (Century)	Site Operating Costs
Roger Wynn, Senior Plant Metallurgist, MMG Ltd (Century)	Metallurgy
Erin Sweeney, Senior Geotechnical Engineer MMG Ltd (Century)	Geotechnical Parameters
Greg Ballarino, Engineering Superintendent MMG Ltd (Century)	Engineering Information
Matthew Lord, Mine Closure Superintendent MMG Ltd (Century)	Environmental
John Maconachie, Senior Surveyor MMG Ltd (Century)	Surveying
Gavin Marre, Senior Business Analyst MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

4.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

4.6.1 Pit Design

The 2013 Ore Reserves is based on pits designed in 2011 and revised in 2013 with an updated footwall surface to a 5.3% zinc equivalent cut-off. After assignment of dilution, any blocks not exceeding a 2013 Net Smelter Return After Royalty (NSRAR) of A$48/t-material in-situ (equating to 5.3% ZnEq) were excluded from the Ore Reserves.

A planning block model was created using the new (2013) regularised Mineral Resource model by performing reconciliation and SMU adjustments (discussed in Section 4.6.7). This model was then used in a Whittle software analysis to generate new optimal Whittle shells for current parameters (costs, prices, geotechnical and surface topography).

Comparison of the new optimal Whittle shells to the current design showed no material difference in both ore and waste, hence redesign of stages was not practical and economic. Comparison of the selected optimisation shell (0.95 Revenue Factor) to the current pit designs showed a difference in potential Ore Reserves of less than one per cent.

The Century open pit was designed into stages; the sub-stages and the final stage through geotechnical guidelines. Design specifications are shown in Table 31.

Table 31 Century Mine open pit design specifications

Parameter	Pit value	Stage ramps value
Slopes	Geotechnical Guidelines	
Bench Height	16/12m	
Flitch Height	2m-4m	
Minimum Mining Width	100m (where feasible)	
Road Design		
Total Road Width	40m	40m
Running Surface	28m	28m
Drain on Inside	2m	2m
Windrow on Outside	5m	5m
Maximum Gradient	10%	10%
Switch Back Inside Radius	25m	25m
Whopper Stopper	5m	5m
Berms	Berm diminishes to zero at ramp access	10m access on Footwall side

4.6.2 Geotechnical Parameters

Geotechnical parameters are well understood from mining over 13 years and managed through the Geotechnical Management Plan, collection of monitoring data and external auditing. Monitoring of the Century open pit has improved with the implementation of slope movement monitoring radar. Further improvements in monitoring are expected to come from the installation of tri-axial geophones in 2013.

Geotechnical Influences

Century pit is a geotechnically complex system with small to large scale discontinuities that define geotechnical domains and dictate overall wall designs. Design and review of pit walls and pit wall stability is a geotechnically intensive exercise resourced on site by geotechnical engineers. Century's geotechnical engineers operate in accordance with the Century Geotechnical Management Plan (GMP). The GMP is an integrated system of geotechnical documentation and processes that are used to manage the ground stability and safety of pit walls and dumps at the Century mine. It provides an overview of the geotechnical program for management and external auditing. Definition of major faults is suitable on a large scale with constant updating of minor to major structures as mining progresses.

A principal risk to all open pit mining operations is that of wall failure. While the pit walls are not particularly high at Century at a maximum of 340m, there are some very challenging geotechnical conditions associated with the south-west wall that could affect the amount of ore extracted in the final years at Century.

The maintenance of slope stability is a challenge at Century due mainly to the pronounced jointing and bedding planes and faulting that has led to severe displacements of the strata that includes the orebody. The most significant of these is the Pandora's Fault that bisects the orebody with a vertical displacement of the ore of some 100m. The ore in the shallower side was the target of initial mining and most of the ore mined is now from the deeper ore on the north side of Pandora's Fault in the northern block.

Typically faults with large throws have either parasitic or splays along strike and often contain fault gouge. Figure 31 illustrates the major faults in the Century Main pit.

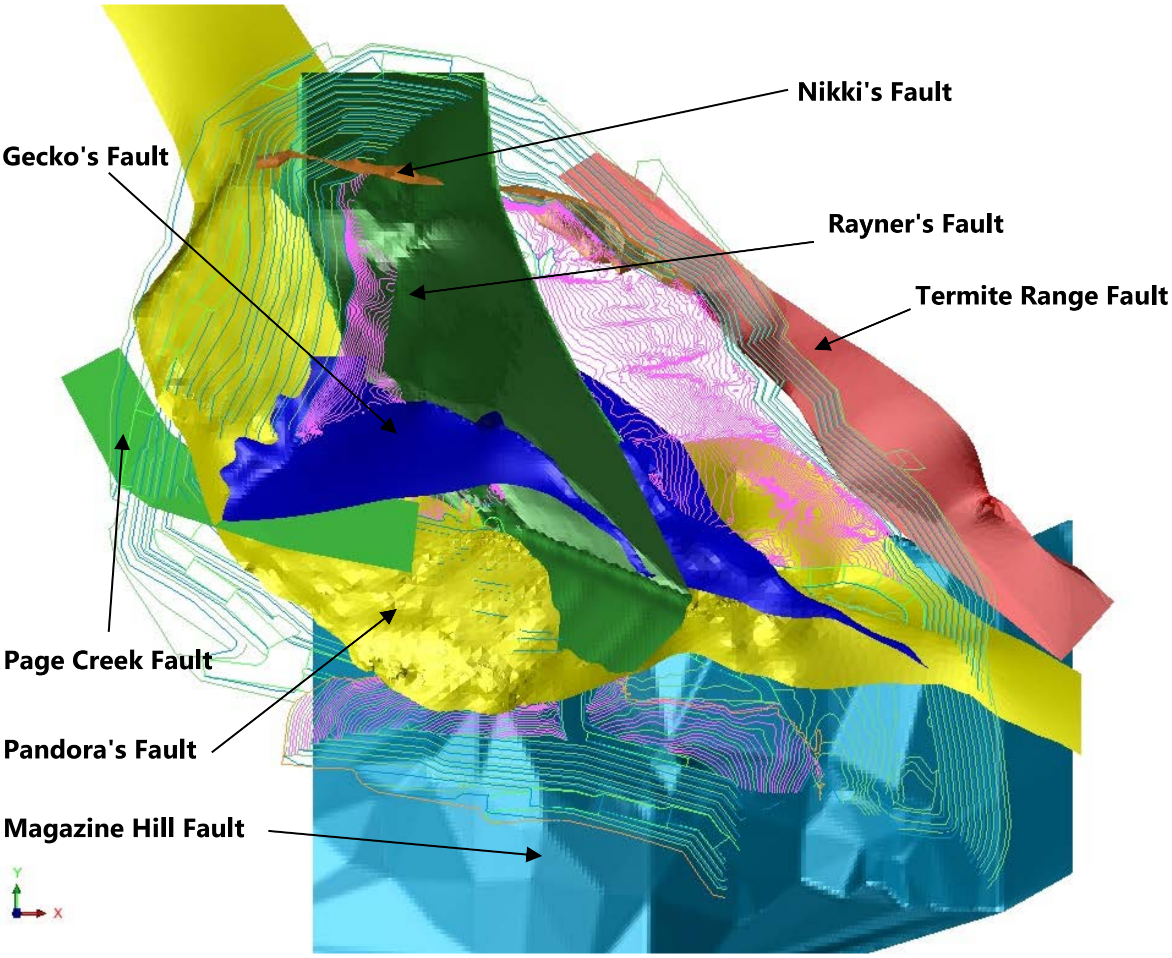
The waste:ore ratio has to date been high at Century and thus it has been essential to maximise the pit slope angles throughout the life of the mine. This has been done effectively using a combination of design, monitoring and mining techniques that aim to maximise the slope angles within acceptable risk parameters, with the primary risk consideration being the safety of the mining personnel.

Geotechnical Auditing

The consulting firm MiningOne Pty Ltd has been involved since the inception of the Century project and the continuity of the involvement of one of the senior partners has resulted in a high level of external expertise for the slope management. Regular audits of practices and data collected are undertaken on a quarterly basis.

Most of the slopes at the Century pit are in a stable configuration as a result of a combination of careful design, monitoring and sound mining techniques that have been adapted to suit the mining conditions.

Figure 31 Major faults in Century Main Pit



Slope Design

Two main strategies have been developed for pit slope design depending on the criticality of the slope.

- For the less critical slopes average values are used for the rock properties and the Factor of Safety of the slope design has to be in excess of 1.3.
- When the more critical slopes are being designed a probabilistic approach is used in which a range of values are used for each of the rock properties and the final slope has to have a 99% probability of success, i.e. a 99% probability of having a Factor of Safety ≥ 1.

Slope Monitoring

Monitoring of the slopes is done using the following methods:

- Prism monitoring. There are over 250 prisms on the slopes around the Century pit with the position of them being logged 1-2 times per day using two Leica Automated Total Stations. If any of the prisms have moved more than a preset limit then the geotechnical engineering team is immediately notified using an automatic e-mail system. The positions of the two measuring stations are such that if one fails then it is possible to measure the position of most of the prisms from the other station. A spare Leica unit is held onsite.

- Ground movement monitoring radar. Site has two Ruetech MSR radar systems. These are currently monitoring the slopes around Stage 8 and Stage 10. The trailer mounted systems remotely scans walls up to 1800 metres away to sub-millimetre accuracy using interferometry techniques. The system continuously monitor the slope face for deformations, and a remote computer produces an image showing spatial deformation relative to a fully geo-referenced image for the entire slope scanned. A series of measurements over time is used to track slope movement.
- Photogrammetric survey of the slopes to map the geological structures evident in the faces

Pit Wall Depressurisation

Groundwater is known to exist within the west wall rock units. The main source of groundwater is believed to be a perched aquifer at the base of the Cambrian Limestone (CLS). Infiltration of this water through the underlying shale is believed to occur at a very slow rate, mainly through structures such as joints and bedding. Locally perched aquifers also occur in the shales where bedding dips into the wall and prevents natural drainage from structures.

Wall depressurisation is typically achieved by drilling 42m long drainage holes inclined at +5°. Drain holes are typically installed mid-batter at a spacing of between 20m to 40m depending on area requirements.

In early 2013 Rock Australia were engaged to drill 150m-deep depressurisation holes in the South-West and South Walls, for deeper depressurisation and to test whether any significant flows would be produced by drilling beyond the standard 42m depth. Six holes were drilled at 928 RL and 940 RL in the South-West Wall area. Two holes were drilled in the Stage 10 south wall at 1104 RL. One hole in the South-West Wall produced minor flows and the others produced none at all. Significant flows were produced from the Stage 10 hole, which is understood to have penetrated the Magazine Hill Fault to the south.

Current Slope Concerns

The slopes of current principal concern are in the south-west corner of the pit above Stage 8. The concerns are raised as a consequence of very complex structures that include steep dipping bedding of the western wall intersecting the faulting running parallel to the Pandora Fault and a former wedge failure. In addition to the complex geology, there are fluctuating levels of pore pressure in the rock as a result of seepage that increases as a result of rainfall and flow in Pages Creek.

The section of South-West wall of the pit between Pages Creek fault and Pandora's fault has displayed significant movement in response to past wet seasons, however the movement in 2012-2013 wet season was greater that the preceding two years despite a relatively dry wet season. These wall movements extended for a longer period and did not stop after the wet season rainfall finished – as was the case in previous years. As such there is a probability of failure occurring during the 2013-2014 wet season, which will involve potentially between 2 and 6 weeks of production delay whist the expected failure is cleared to re-access Stage 8 ore.

Trigger Action Response Plan is in place to act immediately on adverse radar readings.

A large buttress of waste rock has been left in the toe area of this wall below the 936 mRL in order to reduce aggravation of the movement.

4.6.3 Processing (Metallurgical) Recovery Factors

Century uses a series of equations to determine the expected metallurgical performance of ore. These equations have been derived using historical plant operating data and cover circuit recoveries for the three payable metals in Century ore: zinc, lead and silver.

Century concentrator contains four distinct circuits, each of which has its own set of performance equations: the pre-flotation (or carbon) circuit, the lead circuit, the primary zinc circuit (zinc roughers and scavengers) and the ultra-fine cleaner circuit. Hence, zinc recovery is not calculated as a single number, but must first be calculated separately per circuit. Only by combining all the constituent recoveries can total plant performance be ascertained.

Apart from feed zinc, lead and silver grades, the main other input to determine circuit performance is the feed carbon grade, specifically total organic and elemental carbon (TOEC). The proportion of TOEC material in plant feed has a major impact to pre-flotation and zinc roughing performance, and is typically viewed in a ratio with zinc grade as inputs into the metallurgical performance equations.

The quantity of lead concentrate produced influences zinc recovery (as lead concentrate contains some zinc), while the feed flow rate to the ultra-fine cleaners influences that circuit's recovery.

Metallurgical recoveries selected are based on 2014 forecasted recoveries predicted based on the expected improvement from the increase mill throughput (IMT) project in July 2013. These predicted metallurgical recoveries were compared to May 2013 year-to-date performance.

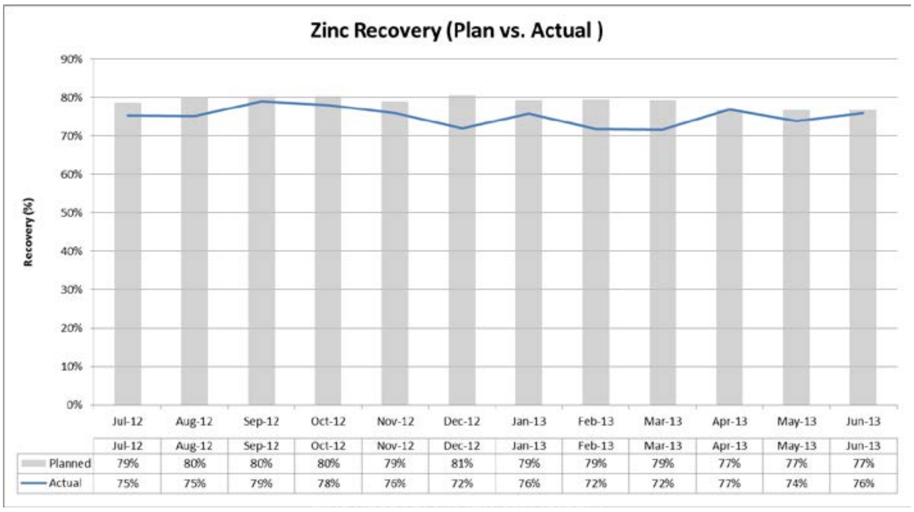
For the 2013 Ore Reserves the metallurgical recoveries used were:

- 75.7% for zinc producing a 57.3% Zn-concentrate; and
- 54.2% for lead producing a 62% Pb-concentrate.
- Recovery of silver is 57.2% to the Zn-concentrate and 8.5% to Pb-concentrate.

Recovery is heavily influenced by the quantity of TOEC in feed, as well as feed zinc/lead grades.

Typically, zinc recovery is 75 to 80%, while lead recovery is 50 to 60%. Reconciliation for FY2013 is shown in Figure 32 below.

Figure 32 FY2013 Historical metallurgical recovery



	Jul-12	Aug-12	Sep-12	Oct-12	Nov-12	Dec-12	Jan-13	Feb-13	Mar-13	Apr-13	May-13	Jun-13
▨ Planned	79%	80%	80%	80%	79%	81%	79%	79%	79%	77%	77%	77%
— Actual	75%	75%	79%	78%	76%	72%	76%	72%	72%	77%	74%	76%

The main gangue in plant feed is silica, and final zinc concentrate has tight restrictions around the quantity of silica permitted to make the saleable specification of 5.2% contained silica. Three-hourly spot samples are taken of final concentrate and analysed by XRF to determine the silica level, such that circuit operation can be modified to maximise recovery while producing concentrate that is within sales specifications.

The Century ore is extremely fine grained, and requires liberation to 6μm in order to remove sufficient silica to make saleable specification.

Apart from being a dilutent in final concentrate, TOEC material absorbs reagents readily. To counteract this, sufficient TOEC material must be rejected via the pre-flotation stage such that downstream performance is maximised without suffering excessive losses of zinc and lead to the pre-flotation concentrate.

4.6.4 Realised Revenue Factors (Net Smelter Return)

The realised revenue from the ore is expressed using a calculated Net Smelter Return After Royalty (NSRAR).

The metal prices and exchange rates used for the 2013 Ore Reserves estimate are shown in Table 32. These are based on the MMG Limited medium term price environment (< 3 years) as discussed in Section 2.1. These prices are in real terms and based on the corporate economic assumptions as at 1 February 2013.

The realisation costs for zinc concentrates are shown in Table 34, and the realisation costs for lead concentrates are shown in Table 35.

Based on these realisation costs, the calculated realised revenue (including royalty deduction effects) for the main payable metals were:

Zinc: US$1,195/tonne-metal in concentrate.

Lead: US$1,723/tonne-metal in concentrate.

Table 32 2013 Century Ore Reserves Metal Prices and Exchange Rate

Metal	Unit	Value	Imperial Equivalent
Zinc price	US$/t	1,961	US$0.89/lb
Lead price	US$/t	2,327	US$1.06/lb
Silver price	US$/oz	27.30	
Exchange rate	A$/US$	0.99	

Concentrate moisture estimates assumptions are given in Table 33.

Table 33 Concentrate Moisture Assumptions

Concentrate	Moisture
Zinc	11.0%
Lead	10.0%

Table 34 NSR Inputs for Zinc Concentrate Realisation Costs

Zinc		
Metal Paid - Zn (total)	85%	%
Minimum Deduction - Zn	8%	% dry
Base Treatment Charge - Zn	200	US$ / dmt con
TC Basis Price - Zn	2,000	US$ / t Zn
TC Escalator - Zn	0.030	US$ / (US$ / t)
TC Deflator - Zn	0.020	US$ / (US$ / t)
Silver		
Deduct - Ag	93.3	g / dmt con
Metal Paid - Ag (remainder)	65.0%	%
Penalties (Zn-Con.)		
Penalties - Zn Con. - Silica	1.66	US$/dmt/ %SiO2 > Penalty Trigger
Penalties - Zn Con. - Silica Trigger Level	3.75%	
Freight, Sampling and Insurance		
Concentrate Pipeline & Port Logistics – Export via Karumba	12.6	A$ / wmt con
Sea Freight	33.1	US$ / wmt con

Table 35 NSR Inputs for Lead Concentrate Realisation Costs

Lead		
Metal Paid - Pb (total)	95%	%
Minimum Deduction - Pb	3%	% dry
Base Treatment Charge - Pb	175	US$ / dmt con
Silver		
Minimum Deduction - Ag	50	g / dmt con
Metal Paid - Ag (remainder)	95%	%
Refining Charge - Ag	10	US$/kg payable
Penalties (Pb-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Concentrate Pipeline & Port Logistics – Export via Karumba	13.1	A$ / wmt con
Sea Freight	14.8	US$ / wmt con

4.6.5 Royalties

Queensland State Government royalties payable are prescribed by the Minerals Resources Regulation 2013 and are based on a variable ad valorem rate between 2.5% to 5.0% depending on metal prices, advised quarterly and calculated on payable metal. They are published by the Queensland Government Department of Mines and Energy and can be found at the web-site of the "Office of State Revenue":

https://www.osr.qld.gov.au/royalties/rates.shtml.

For the CY14 prices used in the Ore Reserves estimation at the time of evaluation, the relevant rates were 2.60% for zinc, 4.74% for lead, and 5.00% for silver.

A royalty discount applies for base minerals processed within Queensland to a particular metal content, as prescribed by Section 51 of the Mineral Resources Regulation 2013. This discount is 35% for zinc and 25% for lead.

4.6.6 Mining Costs and Cut-Off Value

The cut-off grade used for the Ore Reserves estimate is the non-mining break-even cut-off grade taking into account mining and metallurgical recovery, concentrate transport costs, concentrate treatment and refinery charges and royalties. The zinc equivalent used to apply a cut-off grade to cut-off grade model includes zinc with lead factors; the Zn equivalent is the equivalent grade of zinc only which is required to generate the same revenue as the combined zinc and lead grades present. The zinc equivalent is expressed as **"ZnEq = Zn + (1.03*Pb)"** and is based on a net smelter return analysis**.**

The cut-off grade used for the Ore Reserves estimate is 5.3% ZnEq, based on a marginal break-even cut-off grade calculation. This is an increase from the 4.6% ZnEq value used in the 2012 Ore Reserves estimate. The Tonnage-Grade curve for the remaining Mineral Resource indicates a 10% decrease in tonnes between 4.6% ZnEq and 5.3% ZnEq - this value is exclusive of mining dilution but reflects sensitivity to cut-off grade within a reasonable range of adjustment.

The marginal cut-off grade was determined by the following calculation:

$$\text{Cut-off Grade} = \frac{\text{Treatment Plant Costs}}{\text{Realised Metal Price (inc. Royalty effect)} * \text{Metallurgical Recovery}}$$

Where:

Treatment Plant Costs = Process related cost + General and Administration cost (AU $/t)

Realised Metal Price = A $/t (Zinc, Lead) and A$/Oz (Silver)

Recovery = Metallurgical recovery (%)

The impacts of the two saleable metals (zinc and lead) were incorporated into the calculation of the appropriate cut-off grade results which resulted in zinc equivalent (ZnEq) cut-off grade. Allowance for lead was made by using zinc equivalent calculation based on relative NSRAR of each metal. Basically, the calculation allowed for the relative value contribution and presented the resulted cut-off in zinc grade terms. Greater than 90% of Century's value derives from zinc sales (at the current saleable quantities and the relative price differences), so there was little change compared to a straight zinc cut-off.

The costs estimates that were used are based on the June 2013 budget reforecast for 2013. Those costs, which do not change when Century project ends were excluded. Significant among them was the corporate overhead.

With respect to costs:

- The best prediction of operating costs was used in the calculation of the cut-off grade. These predicted costs were compared to actual site performance in 2012
- Only costs typified as the total cost to process ore were included (metallurgical process cost, general and administration overhead cost, the extra cost of mining ore compared to waste, royalties)

Table 36 Costs used in cut-off grade calculation

Costs	**Value A$/t**
Concentrator & Port Processing Operating costs	26.3
Concentrator & Port Maintenance costs	10.3
Finance & Administration (Support Services) - excluding Group Recharge	11.5
Finance & Administration (Support Services) - including Group Recharge	13.9
Total Costs (exc. Group Recharge)	**48.1**

$$\text{Cut-off Grade} \qquad \frac{\text{Treatment Plant Costs}}{\text{Realised Metal Price (inc. Royalty effect) * Metallurgical Recovery}}$$

$$\text{Cut-off Grade (Zn)} \quad = \quad \frac{48.1}{(1{,}195/0.99) * 75.7\%}$$

$$= \quad \mathbf{5.3\ \%Zn}$$

4.6.7 Mining Factors and Assumptions

Historical Mine Call Factors

Historically the Century Ore Reserves has been calculated by applying global mine call factors to the in situ Mineral Resources. In summary the process consisted of three steps:

(i) Compare reported ore treatment plant production to predicted performance from the resource model. Calculate the Mine Call Factor (MCF) required to balance predicted and actual production;

(ii) Identify and consolidate vertical stacks of mineralisation above a nominal cut-off (the 'accumulation' process); and

(iii) Apply MCF to the vertical stacks. Different factors were applied for upper and lower zone and steeply dipping/shallow dipping regions in the pit.

The former approach assumes that 'ore' and 'waste' stratigraphy could be selectively mined within combined stratigraphic units. Essentially an undiluted, fully selective estimate, with a vertical resolution that is probably significantly smaller than achievable in mining.

Reconciliation

Century does not currently carry out a full Pit to Port Reconciliation process. However, the historic conversion of Mineral Resources to Milled grades and tonnes over the past five years, form the basis for the Ore Reserves generation process. The reconciliation of the 2012 Ore Reserves model to the Milled tonnes and grades for the period of July 2012 to June 2013 is summarised in Table 37.

Table 37 Summary of Century Mine 2012 Ore Reserves model to mill production reconciliation July 2012 to June 2013

	Tonnes	**%Zn**	**%Pb**	**Ag (g/t)**	**Contained Metal**		
					Zn (t.)	**Pb (t.)**	**Ag (kg)**
2012 Ore Reserves model depletion	7,481,297	10.3	1.1	27	770,401	80,754	200,178
Reconciled mine production	5,786,941	10.2	1.0	27	589,376	59,432	156,874
Differences	***-22.6%***	***-1.1%***	***-4.9%***	***1.3%***	***-23.5%***	***-26.4%***	***-21.6%***

The unsatisfactory nature of the reconciliation results has been the catalyst for much of the investigations of and changes to the Ore Reserves process for 2013.

2013 Ore Reserves Process (Dilution and Regularisation)

The new approach used for the 2013 Ore Reserves is more closely aligned to operating practices while at the same time incorporating the site's reconciliation performance as a validating principal. The latter approach assumes that the whole of a combined stratigraphic unit can be mined, with no ore-loss or dilution on the margins of the units. Essentially a partially diluted estimate (dilutes individual stratigraphic units into combined stratigraphic units).

A diluted and regularised model suitable as an input to mine planning was then created by simply regularising the proportion based estimates of ore and waste to a fixed SMU of 10x10x3. This SMU was based on investigation into site's mining selectivity achieved in practice.

The revised Ore Reserves estimation approach after regularisation of the block model is divided into nine steps:

(i) Assign material types to a material ('mat') field in the block model based on unit
  - Mat = "BW" (Bulk Waste), if unit = 2 or unit = 100
  - Mat = "MZ" (Marginal Zone), if unit = 145 or unit = 155 or unit = 165
  - Mat = "FWW" (Foot Wall Waste), if unit = 9
  - Mat = "LZ" (Lower Zone – ore), if unit = 450
  - Mat = "UZ" (Upper Zone – ore), if unit = 200
  - Mat = "IBW" (Inter Burden Waste), if unit = 320

(ii) Transfer some from MZ to SM (Sub Marginal) by ZnEq ranges – 1 'low grade' range (cut-off grade)
  - Mat = "SM", if Mat = MZ and ZnEq < 5.30

(iii) Transfer some LZ to LW (Lower Zone Waste) by assessing partial percentages (Waste within LZ)
  - Mat = "LW", if Mat = LZ and P1 < 55%

(iv) Transfer some UZ to UW (Upper Zone Waste) by assessing partial percentages (Waste within UZ)
  - Mat = "UW", if Mat = UZ and P1 < 55% (P1 being the partial percentage of the material)

(v) Transfer some of IBW to UZ to simulate 311 & 312 units (by partial percentage)
  - Mat = "UZ", if Mat = IBW and P1 < 55%

(vi) Transfer some from MZ to SM by partial percentage (diluted marginal reclassified as sub-marginal)
  - Mat = "SM", if Mat = MZ and P1 < 60%

(vii) Assign "FILL" to already mined out area
  - Mat = "FILL", if above June 2013 eop.dtm and below the digplan.dtm

(viii) Assign Density to "FILL" to already mined out area
  - Density = 1.84 (loose SG) if Mat = "FILL"

(ix) Assign "AIR" coded to material above digplan
  - Mat = "AIR" if above digplan.dtm

The final Ore Reserves are reported as the flagged Upper and Lower Zones plus Marginal Units above cut-off, inclusive of dilution and internal waste but exclusive of mining loss.

Material above cut-off was calculated after consolidating all horizons between the 170 and 311 (the Upper Zone) into a single mining unit and all horizons between the 410 and 450 into a second mining unit (the Lower Zone). These mining zones correspond to the operations current mining practices. The Upper and Lower Zones plus individual horizons 140, 150, 160 and 460 were defined as ore if they were above cut-off grade.

Global dilution and ore loss factors were applied to the material defined as ore to account for historical mining performance. These factors were determined iteratively from Century reconciliation data. The results of the iterative process were reviewed for reasonableness given the current mining practices and equipment fleet. Dilution was calculated using a 0.5m minimum mining width and 0.5m limit on internal waste. Dilution was set to 12% for all material classified as above cut-off.

4.6.8 Infrastructure

Mining Infrastructure

Mining is by a single large scale open cut mine.

Primary Crusher

The primary crusher is a large MMD sizer that uses slowly rotating breakers to break up the oversize lumps. Prior to milling the ore is conveyed across to a large conical stockpile so that there is a buffer between the crushing and milling operations. Under the coarse ore stockpile there are two apron feeders that feed the ore onto the SAG mill feed conveyor. The breaker is arranged so that any oversize rides over the top of the rotating breakers and is discharged off the end where it is periodically picked up by a loader and taken away for breaking elsewhere.

Concentrator

Site has a concentrator with a throughput capacity in excess of 7Mtpa of ore, to produce lead and zinc concentrates.

Milling is carried out by:

- 1 x SAG mill, 12MW gearless motor drive (wrap around motor).
- 1 x ball mill (#1), conventional single pinion drive of 6.7MW.
- 1 x ball mill (#2), about 20 years old, purchased second hand and refurbished with an 8MW GMD.

After grinding the ore down to 50µm-80 µm for flotation the slurry is pumped across to a differential flotation circuit which is extract carbon, lead and zinc, in that order.

79 large tank cells (most of which have a capacity of 100m^3) are used to provide the flotation. Grinding of the intermediate concentrates down to 20 microns and finally to 6.5 microns is required to maximise the recovery of zinc. Sand mills are used for the fine grinding.

The concentrates are thickened and then stored in surge tanks, three for the zinc concentrates and one for the lead concentrates.

The tailings are pumped to their own thickener to recover much of the water in the tailings and then pumped to the tailings dam.

Pipeline and Port

The lead and zinc concentrates from the plant are pumped down a pipeline 300km to the coastal township of Karumba on the Gulf of Carpentaria, for shipping to various customers, including Nyrstar zinc refineries in:

- Budel, the Netherlands;
- Hobart, Australia; and
- various customers in China and Japan.

The lead and zinc concentrates are pumped by three Wirth piston diaphragm pumps separately to Karumba in campaigns and hence the need for surge tanks. Typically both the zinc and lead concentrates are pumped to Karumba each day with slugs of water used to separate the batches of zinc and lead concentrates in the pipeline.

At the port of Karumba on the mouth of the Norman River in the Gulf of Carpentaria the company owns and operates:

- a filtration plant to dewater the piped concentrate slurries;
- a storage shed for the concentrates (designed to withstand a Category 4 cyclone);
- a barge-loading facility; and
- a self-propelled and self-unloading barge (the ‘MV Wumna’) to transfer the concentrates from a dedicated wharf to the ships anchored in the Gulf in 5,000t shipments.

Power

Power supply to Century Lawn Hill is via 220kV line from Mt. Isa, supplying a contract nominal amount of up to 50MW. A secondary back up for emergency power only is provided by five site generators. Commercially, the mains power supply is made up of three separate contracts, gas (to the power station), power supply and transmission.

- The existing natural gas supply contract is with Santos until December 2015 and a new contract with Origin is in place until December 2019.
- The existing electricity supply contract is with Stanwell Corporation Limited. The original contract was due to expire in July 2014, but an extension clause has been activated through to 1st January 2020.

Water

Century Lawn Hill is situated in an arid region of NW Queensland and has 100% of its raw water supplied by site borefields. These borefield provide raw water for mill processing, fire systems and treatment plants for potable water.

Two separate fields operate: the Western and the Eastern borefields. The Western borefield is located near the mining operation to assist with dewatering of the pit from ground water movement. This water is combined with the supply from the Eastern fields at the raw water dam at the Concentrator for processing. The Concentrator is the greatest user of water on site, requiring approximately 100m^3/hr per 100t/hr of crushing which equates to 222 L/s if operating at 800t/hr. The anticipated total water usage once the Increased Mill Throughput project is completed is in the order of 400 L/s and the Borefield Upgrade project aims to develop a total system capacity of 575 L/s.

Buildings and Accommodation

The site building facilities on site are in satisfactory condition and suit the requirements of organisation. The A&R Site Services department manage the maintenance and upgrades of these facilities.

All site accommodation is at the Darimah Village which has a capacity of 770 persons per night which adequately handles the standard site needs, which averages 480 per night, and allows for further peak demands such as concentrator shutdowns which require a further 150 to 200 persons to be on site.

Accommodation at Karumba is provided at Pelicanns Inn and Savannah. Building facilities at both sites are in satisfactory condition and suit the requirements of organisation.

Communications

There are a number of diverse communication systems in place to meet the varying requirements of the mining operations and processes. These include terrestrial, satellite, radio and wireless systems at the Century mine site, village and airport as well as along the slurry pipeline, at Karumba and on the transfer vessel MV Wunma. Most of the infrastructure is owned by MMG and operated / maintained by MMG personnel or designated contractors, exceptions being the Telstra and Optus land line and mobile phone systems.

Maintenance Workshops

Workshops exist for all mobile and fixed plant maintenance.

Since July 2011, all mobile and fixed plant maintenance at Century Lawn Hill fall under a singles department: the Asset & Reliability (A&R) Department

The A & R Department is divided into teams that cover:

- Mobile Maintenance.
- Fixed Plant Maintenance.
- Engineering & Reliability.
- Planning & Scheduling
- Site Services (Building Services, Light Vehicle Maintenance, Cleaning & Road Maintenance).

Airport

The mine is serviced by Lawn Hill airport in close vicinity to the mine site. The airport has a sealed runway and is equipped for night landings.

Medium sized jet aircraft are used to transport the staff to and from Townsville and Cairns.

Smaller light aircraft bring in employees from some of the closer towns such as Mt. Isa, Doomadgee, Normanton and Karumba

Road Access

Century Mine is in an isolated location and relies primarily on the road transport link with Cloncurry and to a lesser extent on the rail transport link to Townsville for bringing in the bulk materials.

A gravel road connects Century Mine to the Barkly Highway, which is part of Australia's Highway No.1 that circles the continent. It is an 800km journey by road and rail from Townsville to Cloncurry.

4.6.9 Environmental Factors

Century operations act within the following environmental permits;

Lawn Hill Environmental Authority

Environmentally Relevant Activities 1, 6, 7 20, 24, 25, 29 and 37 are conducted at the Lawn Hill Mine pursuant to MIN100737008. This environmental authority (EA) was granted to support mine development and has been in effect since commissioning on 30 January 2009. Minor amendments have occurred during the life of the permit. The EA is administered by the Queensland Department of Environment & Resource Management (DERM).

The current EA came into effect on 30 January 2009 and has no termination date.

Karumba Dewatering and Load-out Facility Environmental Authority

Environmentally Relevant Activities 18, 31, 50, 58 & 63 are conducted at the Karumba Dewatering & Load-out Facility pursuant to IPCE01710409. This development approval (DA) was granted following an administrative error by the Queensland Government in effect since January 2011. Minor amendments have occurred during the life of the permit. The EA is administered by the Queensland Department of Environment & Resource Management (DERM).

The current EA came into effect on 14 January 2011 and has no termination date.

The three most material environmental liabilities for Century mine site are considered to be:

- Mineralised waste rock final landforms (northern, southern and western waste rock dumps).
- Mineralised tailings landforms.
- Proximity of the pit void to Pages Creek.

Waste Dumps

Waste rock from the mining operation has been arranged in three (3) ex-pit waste rock dumps. In addition, a large volume of mineralised waste has been stored in-pit. A waste rock management plan has been developed that describes the processes for siting and developing the final landforms. Potentially Acid Forming (PAF) and Non Acid Forming (NAF) waste has been preferentially handled to resist the ingress of water into the dumps and the development of acid mine drainage. Store & release cover systems have been trialled and selected as the means of final encapsulation.

The southern waste rock dump was capped in 2009 and approximately 90% of the encapsulation effort has been completed. Establishment of a vegetation cover is ongoing. The western and northern waste dumps are still in active use and are considered adequate for life of asset requirements.

Neutral mine drainage is currently being released from both the south and western waste rock dumps.

Table 38 shows the volume capacity of the existing waste dumps and potential sites with status of approval.

Table 38 Waste Rock Balance

Dump Location (waste destination)	**Design capacity (Mlcm)**	**Utilised capacity (Mlcm)**	**Remaining capacity (Mlcm)**
*North Waste dump	5.0	0.5	4.5
West Waste dump	64.4	62.7	1.7
Main in-pit dump	2.1	1.2	0.9
Stage 7 in-pit dump	1.4	0.2	1.2
**South Access ramp (SAR) in-pit dump	0.6		0.6
Total	**73.5**	**64.6**	**8.9**

* Partial (intermediate) design of dump, full design currently awaiting approval

**Currently under undergoing review and risk analysis for approval

TSF

Process residues are stored in the on-site Tailings Storage Facility (TSF). The TSF has been developed in three (3) lifts. The final lift was constructed in 2011 and is considered adequate for life of asset processing requirements. Following the completion of processing, an encapsulation system must be established to retard the migration of oxidation products from inside the mass to the wider environment. The current capping system design is at concept level.

Pages Creek

Storage of mineralised waste rock in the open pit has and will result in poor water quality in the pit following the cessation of processing and following lease relinquishment. It is anticipated that at some point in the future, the western wall of the main pit will fail and Page Creek will drain into the pit void. This may not occur for some time, perhaps one or two centuries but on an infinite time scale the probability of this event is expected to be certain. Draining Page creek to the pit will result in the pit water balance moving into substantial surplus and eventually spilling to the environment.

4.6.10 Social Factors

The Gulf Communities Agreement (GCA) was negotiated between Pasminco Century Mine Limited, the Queensland Government and four Native Title groups - the Waanyi, Mingginda, Gkuthaarn and Kukatj - under the right to negotiate provisions of the Native Title Act 1993 (Cth). It came into effect in February 1997. The GCA covers a wide range of issues and commitments, including social impact assessments, health facilities, and the development of local businesses, compensation at the mine site and along the pipeline corridor, strategic plan funding, employment and training.

The long-term legacy goals and aspirations of the GCA are:

(i) To remove the Native Title Groups and the other members of the Communities from welfare dependency and, to the greatest extent possible, promote economic self-sufficiency;
(ii) To participate as fully as possible in the Project and mine related ventures;
(iii) To be able to live on their traditional lands;
(iv) To protect fully their natural environment and its resources;
(v) To identify and protect sites of significance to the Native Title Groups;
(vi) To ensure that the material benefits do not corrupt Indigenous cultures but enable people to re-affirm the cultures and enhance the lifestyles of the members of the Native Title Groups and other members of the Communities through community and cultural development initiatives; and
(vii) To ensure that the standard of health, employment rates, education opportunities and other social indices of Native Title Groups and other members of the Communities is comparable to ordinary Australian standards.

The Century Liaison Advisory Committee (CLAC) has been re-established to oversee the end of the Project, especially the completion of the GCA. All Parties to the Agreement are represented on the CLAC and MMG Century provides administrative and executive support to the committee.

Table 39 below shows the current status of all clauses in the Gulf Communities Agreement (GCA).

Table 39 Current Status of Gulf Communities Agreement

Summary of All Schedules

	S1	S2	S3	S4	S5	S6	S7	S8	S9	S10	S11		Status
Actioned & Ongoing	9	58	39	35	15	9	10	6	2	12	1	196	**37%**
Completed	51	24	18	16	15	5	4	17	3	2	0	155	**28%**
Requires actioning	0	1	1	0	0	0	0	0	0	0	0	2	**1%**
At or during mine closure	0	0	0	1	3	0	0	0	1	0	4	9	**1%**
No actions required	16	7	56	18	22	4	3	3	2	3	44	178	**33%**
Totals	**76**	**90**	**114**	**70**	**55**	**18**	**17**	**26**	**8**	**17**	**49**	**540**	**100%**

4.6.11 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with “Table 1 Section 4” of the code – are given in Table 40. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 40 JORC Code Ore Reserves assessment and reporting criteria for Century 2013 Ore Reserves

Assessment Criteria	**Risk Assessment**	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	Low -Medium	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The Ore Reserves are based on the 2013 Mineral Resource model built by Quantitative Group (QG) utilising geological interpretations and data provided by MMG geologists. The block model was generated in Vulcan software, with grade estimation being carried out in Isatis using the ordinary kriging estimation method. The Mineral Resource estimate is based on significant drilling conducted between 1990 and 2013. The Competent Person for the Mineral Resource estimate was Michael Smith of MMG Century. The approach varied from previous years following an independent review of the Century Resource by QG in 2012. The assumptions and approach are detailed in the Century Resource Report (MMG15106) compiled by Quantitative Group in September 2013. Risk exists with respect to structural complexity potentially causing off-sets of the ore zones. Drilling has however not shown any blank zones in the mineral package. Further details are discussed in the Mineral Resources Section of this report
Classification	Low	The Ore Reserves estimate is based on the Measured and Indicated Mineral Resource estimate after consideration of all mining, metallurgical, social, environmental and financial aspects of the project. Due to uncertainties with a number of the modifying factors, no Proved Ore Reserves has been claimed for in-pit Mineral Resources. These modifying factors with uncertainty include ore dilution parameters, reconciliation issues, and geotechnical structural impacts (Stage 8 west wall "buttress"). The reconciliation issues, plus interim status of models prior to grade control drilling update makes it difficult to support the classification of Proved Ore Reserves. The estimation quality in the remaining resource areas of Gecko and Pandora generally supports a classification of Indicated. This is obviously most confident in close proximity (say within 40m) to grade control drilling. Further away from grade control drilling, the confidence in carbon estimates is significantly lower. All Probable Ore Reserves are derived from the Measured or Indicated Mineral Resource. No Inferred Mineral Resource is included in the Ore Reserves. Stockpiled ore has been classified as Proved.
Site visits	—	The Competent Person is based on site.
Study status	Low	The mine is operating. Factors and costs used are based on current and recent historical values.
Cut-off parameters	Medium	See Section 4.6.6 for details.
Mining factors or assumptions	Low-Medium	Century has established mining operations with well understood and managed mining risks and mining methods. Pit design parameters are discussed in Section 4.6.1. Geotechnical parameters are discussed in Section 4.6.2. Other Mining Factors including dilution, loss, and reconciliation are discussed in Section 4.6.7.

Assessment Criteria	**Risk Assessment**	**Commentary**
Metallurgical factors or assumptions	Low	Century has established milling operations with well understood and managed operating risks. The Century concentrator follows a standard lead/zinc flow sheet, with a pre-flotation circuit to remove Total Organic and Elemental Carbon material prior to the lead circuit, and a fine grind stirred mill circuit to remove silica from the zinc concentrate. Plant production is approximately 500kt of zinc metal contained in concentrate, and 40kt of lead contained in concentrate. Metallurgical recovery factors are discussed in Section 4.6.3.
Environmental	Medium	In pit and ex-pit water management is an ongoing issue across wet seasons. Increased dam and pumping capacities over the past two years have worked toward mitigating this risk based on hydrological forecasts.
Infrastructure	Low - Medium	The concentrate pipeline and concentrate storage shed was refurbished in 2012 and will be in operational state past Century's current LOA of 2016
Costs	Low	Processing and production costs have been derived on an activity basis and built from the bottom up based around production, contracts and corporate assumptions.
Revenue factors	Medium	See Section 4.6.4 for details.
Market assessment	Low	For discussion of market conditions and expectations refer to Section 2.2.LOM sales contracts were established at commencement of production and are still in place. In addition, a ready market exists for zinc and lead concentrates.
Economics	Low	As costs are based on current and recent historical values, revenues are based on near term forecasts, and the mine stripping ratio is now in process of decline the economics of the remaining Ore Reserves are robust. Impending Closure Costs are expected to be significantly greater remaining incomes.
Social	Low	The project enjoys strong local support with significant continuing contributions to the local communities and economy. The main regional community of Lawn Hill supports mining developments. See Section 4.6.10 for details.
Audit or Reviews	—	In late 2012, Quantitative Group (QG) carried out an independent review of the 2012 Century Mineral Resource model. Based on the recommendations of this review the modelling approach was altered in 2013. The new QG Resource model is the basis of this report. Quantitative Group Pty Ltd (QG) was commissioned to rebuild and re-estimate the Mineral Resource model for MMG Limited's (MMG) Century zinc mine in Northwest Queensland. This project was intended to improve on the operation's existing resource estimate approach and to provide a model more suited for long term mine planning use. In 2012, Quantitative Group Pty Ltd (QG) was engaged to develop a new approach to the Ore Reserves estimate at MMG Limited's (MMG) Century mining operation. MMG's request for a new approach followed an earlier QG review of historical practices which identified concerns with the application of global mine call factors (MCF). In addition to developing the new approach, MMG requested QG to investigate and comment on the operation's cut-off calculations and the life-of-mine planning approach.
Discussion of relative accuracy/ confidence	—	Potential risks affecting each of the relevant criteria are qualitatively indicated in the "Risk Assessment" column of this table.
Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Low	The survey integrity of the topography is sufficient for purposes and is not an issue for the operations at Century. There is potential to better integrate this into the hydrological modelling, with changes in the mine footprint. An aerial survey is undertaken every two years and is used as a base line for site volume adjustments. The 2012 aerial adjustment was 74,198bcm, or 0.15%.
Climate	Low	Century Mine is in the semi-tropical zone of inland northern Australia. There are two distinct seasons, the dry season between March and November and the wet season between December and February. The rainfall in the wet season is very variable; some seasons having almost none in drought years, and others being severely affected by rain bearing depressions that are the remnants of cyclones moving inland from the Gulf of Carpentaria. The average rainfall is 527mm, but is very variable. The highest annual total recorded in more than 100 years is 1243mm, and contrasts sharply with the lowest recorded annual total of 183mm. The average

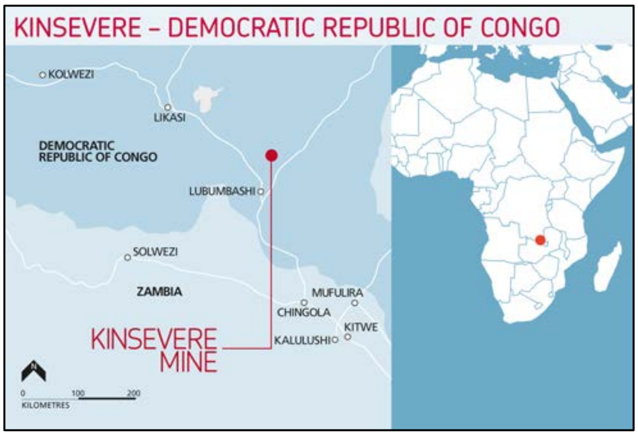
Assessment Criteria	**Risk Assessment**	**Commentary**
		rainfall at Karumba is 922mm and the highest recorded annual total is 1913mm. The highest daily rainfall recorded at Burketown, which is also in this portion of the Gulf of Carpentaria is 430mm. Humidity and High Rainfall in summer may cause problems in mining rates, but not with the Ore Reserves itself. Century Mine has pumping, and storage infrastructure to dewater following large rain events, however with an increasing catchment this system may become stretched. Heavy rainfall events also increase the risk of pit wall failure and potential sterilization of Ore Reserves.
Government Agreements	Low	No problems are expected in maintaining the necessary Federal, State and Local Government permits and the project has strong local support. The required mining tenements and rights have all been granted by the government.
Hydrogeological Parameters	Low	The hydrogeological parameters were obtained during the feasibility study stage and further data have been collected through hydrogeological drilling programs over the years. No significant risk related to hydrogeology is expected.
Waste Storage (Including Tails Storage)	Low	Tailings dam lift works were completed less than 2 years ago to provide sufficient capacity for current LOA. In addition, the TSF was properly designed and constructed under the supervision of a recognised engineering firm and is not expected to cause any issues. See Section 4.6.9 for discussion of Waste Dump capacity.

5. KINSEVERE OPERATION

5.1 Introduction and setting

Kinsevere is located in the Katanga Province, in the southeast of the Democratic Republic of Congo (DRC). It is situated approximately 27 kilometres north of the provincial capital, Lubumbashi (Figure 33), at latitude S 11° 21′ 30″ and longitude E 27° 34′ 00″.

Figure 33 Kinsevere Mine location



Kinsevere is conventional truck and excavator operation with atmospheric leaching of the oxide ore using an SX-EW plant. The mine was started in 2006 using heavy media separation (HMS) and an electric arc furnace operation. The electric arc furnace was put on care and maintenance in 2008 with HMS then producing a direct shipping ore product grading 25% copper. The HMS was decommissioned in June 2011 when the Stage II SXEW plant was commissioned. The Stage II plant is able to comfortably process up to 1.6Mtpa of ore and produce approximately 65,000 tonnes of copper cathodes.

5.2 Geological Setting

The Kinsevere Project area is located in the north-eastern section of the Central African Copperbelt (the CACB). Together with the Zambian Copperbelt to the south, this celebrated metallogenic province contains some of the world's richest copper and cobalt deposits.

Both Congolese and Zambian portions of the belt are located within a continuous fold zone known as the Lufilian Arc, one of several major Pan-African structures bordering the Congo and Kalahari cratons. Each portion exhibits early Neo-Proterozoic intra-cratonic rift development, coincident with the break-up of a Meso-Proterozoic supercontinent (approximately 800 Ma to 600 Ma). Late Neo-Proterozoic collisional deformation and metamorphism is also documented regionally, linked to the formation of central Gondwana (approximately 600 Ma to 500 Ma).

The deposits occur within an internally folded but originally continuous fragment of R1 (Red RAT) and R2 (Mines Group) rocks surrounded largely by Kundelungu argillaceous sediments, which separated into several segments:

- The main 1.3km domain comprising the Tshifufia and Tshifufiamashi deposits;
- A smaller westerly block in the southern portion of Tshifufia which is interpreted as the gently folded arch at the head of a thrust ramp abutting the western boundary of the principal massif along a major, vertical north-south tectonic melange or thrust fault, (the central cataclastic break); and
- A south-easterly-trending segment culminating in Kinsevere Hill.

Mineralisation at Kinsevere is hosted within three stratigraphic horizons:

- The Lower Ore Body (LOB) which is hosted in the DStrat and RSF units of the Kamoto Dolomite Formation.
- The Upper Ore Body (UOB) which is hosted by the SD Dolomitic Shale Formation.
- The Third Ore Body (TOB) which is hosted in the CMN Kambove Dolomite Formation.

Oxide mineralisation lies beneath a thin but irregular leached zone with fracture hosted/disseminated manganese oxides plus heterogenite and minor iron oxides.

Oxide ore mineralogy at Tshifufia, Tshifufiamashi and Kinsevere Hill is composed predominantly of malachite ($Cu_2CO_3(OH)_2$) with minor chrysocolla ($(Cu,Al)_2H_2Si_2O_5(OH)$) and azurite ($Cu_3(OH)_2(CO_3)_2$), accessory pseudomalachite ($Cu_5(PO_4)_2(OH)_4$) and libethenite ($Cu_2(PO_4)(OH)$), and rare intergrown heterogenite ($CoO(OH)$). These occur as disseminations and/or in veins and veinlets, which sometimes coalesce into prominent "clots"; while heterogenite is probably limited to vuggy infills in well-developed malachite veins along with manganese oxides. Whilst the principal copper mineral occurs as malachite veins, mineralisation also occurs as fracture-infill and bedding coatings, plus erratic chrysocolla with subordinate azurite, which are mainly in close proximity to carbonaceous shales.

Supergene mineralisation exhibits a profile that mirrors the weathering boundaries, but is not always developed, so that a classic supergene blanket is largely absent. Normally, the only signs are sporadic minor chalcocite and/or cuprite within malachite veins. However, supergene mineralisation is particularly well formed along the margins of carbonaceous sediments where a contrasting redox front exists

Where fresh, hypogene copper sulphide mineralisation is dominated by mostly chalcopyrite with local zones of bornite, and sometimes associated with infusions of quartz – dolomite veinlets.

5.3 Mineral Resources - Kinsevere

5.3.1 Results

The June 2013 Mineral Resource estimate for the Kinsevere deposit is shown in Table 41.

The reporting cut-off grade applied to the model is 0.75% acid soluble copper (ASCu%) for the oxide Mineral Resource and 0.75 total copper (TCu%) for the primary sulphide Mineral Resource. This grade defines mineralisation which is prospective for future economic extraction. The Mineral Resource has been depleted to account for mining of ore.

Table 41 June 2013 Kinsevere Mineral Resource at 0.75% acid soluble copper (for oxide Mineral Resource) and 0.75% total copper (for primary sulphide Mineral Resource)

Kinsevere Mineral Resources

				Contained Metal	
0.75% Acid soluble Cu cut-off grade (oxide) **0.75% Total Cu cut-off grade (primary)**	Tonnes (Mt)	Copper (% TCu *)	Copper (% ASCu*)	**Copper TCu* ('000 t)**	**Copper ASCu * ('000 t)**
Oxide Copper					
Measured	12	4.0	3.2	-	380
Indicated	16	2.8	2.4	-	380
Inferred	0.8	2.5	2.0	-	20
Total	**29**	**3.3**	**2.7**	**-**	**780**
Primary Copper					
Measured	1.5	2.7	1.0	41	-
Indicated	10	2.8	0.6	280	-
Inferred	11	2.1	0.3	230	-
Total	**23**	**2.5**	**0.5**	**550**	-
Total Contained Metal				**550**	**780**

* TCu stands for Total Copper, ASCu stands for Acid Soluble Copper.

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.

Competent Person:

Mauro Bassotti (Member of AusIMM(CP), employee of MMG)

Mineral Resource numbers are inclusive of 3.5Mt of stockpile material with a grade of 2.4% TCu and 1.9% ASCu (Table 42). Stockpiles have been classified as Indicated due to uncertainty in copper grade variance for short term planning and absence of calcium estimates. Calcium is used for long term planning of acid consumption cost required to economically extract copper. More detail on the calcium values in the in situ Mineral Resource is provided in Table 1 Section 3 "*Metallurgical factors or assumptions*".

Table 42 Kinsevere Stockpiles

Stockpiles - June 2013				
Resource Category	Tonnes (MT)	Density	ASCu (%)	TCu (%)
Indicated	3.5	1.9	1.9	2.4

Stockpiles have been classified as Indicated

Excludes non processable stockpile of 0.3Mt at 1.6 ASCu % and 2.5 TCu (%)

The breakdown of changes between the 2013 and 2012 Mineral Resource are illustrated in Figure 34 and Figure 35) for total tonnes and copper metal tonnes. Numbers in the waterfall charts are exclusive of stockpiles and only refer to the in situ Mineral Resource.

Figure 34 Waterfall comparison of tonnes for in situ Mineral Resource

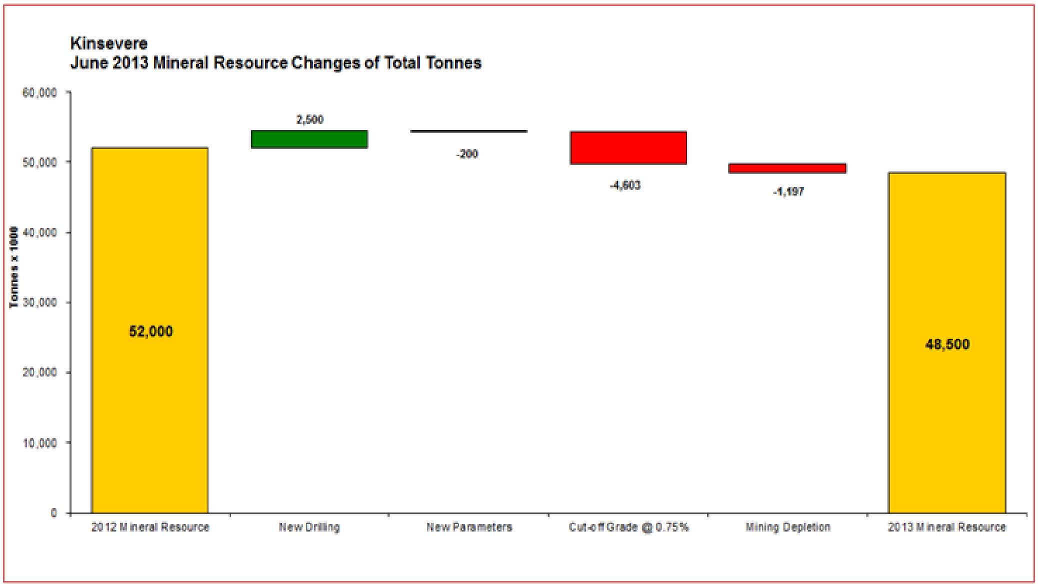
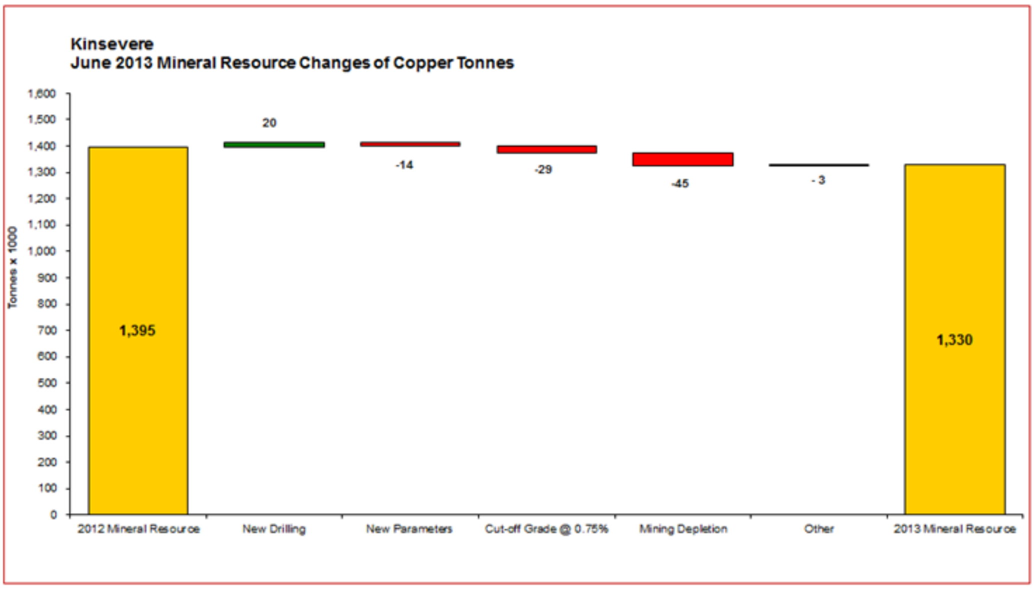


Figure 35 Waterfall comparison of copper tonnes metal for in situ Mineral Resource



All Mineral Resources quoted in this report were estimated from 3 dimensional block models created with CAE Datamine™ software. Wireframe volumes and surfaces were created for the domains and zones of similar weathering, stratigraphy and style of mineralisation.

TCu (total copper), ASCu (acid soluble copper), cobalt (Co) and calcium (Ca) grades were interpolated using an ordinary kriging algorithm. Variogram and estimation parameters were defined using Supervisor Software. Fe, S, Mn, Mg, Mo and U were interpolated using Inverse Distance Squared.

Density values were assigned to the block model per lithology and weathering or oxide domain. Assigned values were determined from 1,696 diamond core density measurements, four in-pit bulk sample measurements and a series of twelve in-pit measurements from specific lithologies having different degrees of weathering.

5.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Mauro Bassotti, confirm that I am the Competent Person for the Kinsevere Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Kinsevere Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Kinsevere Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Kinsevere Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Mauro Bassotti – 26/11/13

Anna Lewin (Witness)

5.4 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Kinsevere Mineral Resources.

Table 43 Checklist of assessment and reporting criteria for Kinsevere Mineral Resource

<table>
<tr><th>Criteria</th><th>Status</th></tr>
<tr><td colspan="2">Section 1 Sampling Techniques and Data</td></tr>
<tr><td>Sampling techniques</td><td>

The Mineral Resource uses both grade control RC drilling and exploration – resource delineation diamond drilling.
Grade control samples are obtained by reverse circulation drilling and composited into 2m samples.
Grade control samples are passed through a cyclone and four-tier splitter and bagged in calicos as 1kg to 2kg samples.
Resource delineation and exploration drilling done as diamond drilling.
Core was sampled every 1m length from quarter core for PQ and half core for HQ.
In unmineralised zones sampling is done at 4m lengths. Sampling is done by cutting half core, with half retained on site for future reference.
</td></tr>
<tr><td>Drilling techniques</td><td>

RC drilling is used to obtain 2m composited RC chip samples.
Diamond drilling was used to recover PQ and HQ size core.

Table 44 summarises the metres drilled by year and drilling type.
Table 44 Drilling type and metres by year
<table>
<tr><th>Year</th><th>Drillhole type</th><th>Metres</th><th>Metres %</th></tr>
<tr><td>2005</td><td>DD</td><td>2,334</td><td>2%</td></tr>
<tr><td>2005</td><td>RC</td><td>6,042</td><td>6%</td></tr>
<tr><td>2006</td><td>DD</td><td>4,803</td><td>5%</td></tr>
<tr><td>2006</td><td>RC</td><td>7,729</td><td>8%</td></tr>
<tr><td>2007</td><td>DD</td><td>3,950</td><td>4%</td></tr>
<tr><td>2007</td><td>RC</td><td>26,025</td><td>26%</td></tr>
<tr><td>2008</td><td>DD</td><td>16,895</td><td>17%</td></tr>
<tr><td>2008</td><td>RC</td><td>9,852</td><td>10%</td></tr>
<tr><td>2011</td><td>DD</td><td>11,868</td><td>12%</td></tr>
<tr><td>2012</td><td>DD</td><td>8,747</td><td>9%</td></tr>
<tr><td>2012</td><td>RC</td><td>100</td><td>0.1%</td></tr>
<tr><td>TOTAL m</td><td></td><td>98,344</td><td>100%</td></tr>
</table>
<i>DD = Diamond drilling</i>
<i>RC = RC collars and grade control drilling</i></td></tr>
<tr><td>Drill sample recovery</td><td>

Recovery recorded during RC drilling is generally high, with minor losses in broken ground.
Core recovery recorded was generally 100%, with minor losses in broken ground. There is no relationship between core loss and mineralisation or grade.
</td></tr>
<tr><td>Logging</td><td>

For the grade control samples: 1m samples are piled in depth sequence, sieved and logged.
For grade control logging: Geologists log directly into an Excel logging template with general geological information logged – lithology, stratigraphy, weathering, oxidation, colour, texture, grain size, mineralogy and alteration.
For grade control logging: Excel files are imported into DataShed database.
Core logging recorded geological and geotechnical information including lithology, stratigraphy, mineralisation, weathering, alteration and geotechnical parameters, strength, RQD, structure measurement, roughness and infill material.
Core and chips trays are stored in a core shed in the Kinsevere mine area.
Core photographs were taken prior to splitting and are available for all drillholes
</td></tr>
</table>

Sub-sampling techniques and sample preparation	▪ Grade control (RC) samples are sorted, weighed and oven dried before crushing with 70% passing 2mm sieve test. ▪ After crushing the samples are split and pulverised with 85% passing 75µm sieve test. ▪ Core was split in half or quartered using diamond saw. Sample lengths were cut as close to 1m as possible while respecting geological contacts. ▪ Samples were generally 2kg to 3kg in weight. ▪ Laboratory process followed drying, crushing, milling and homogenising entire sample to 80% passing 75µm.
Quality of assay data and laboratory tests	The following assaying and analytical procedure is for grade control samples: ▪ Samples are prepared and assayed at the site SGS Laboratory. ▪ Following preparation, 50g pulp samples are analysed for total and acid soluble copper, cobalt and manganese. ▪ A 3-acid digest with AAS finish is used to analyse for total values ▪ A sulphuric acid digest with AAS finish is used to analyse for acid soluble copper The following steps broadly outline the analytical process for the SGS laboratory: Digestion: ▪ Weigh 0.400g +/- 0.004g of sample pulp into a clean 100 ml beaker. ▪ Add 4ml aqua regia to the beaker and allow the sample to digest cold. ▪ Add 2ml of perchloric acid to the beaker. ▪ Digest at 200°C until incipient dryness. ▪ Cool the beakers and add 10 ml hydrochloric acid. ▪ Heat to near boiling to dissolve soluble salts then cool to room temperature. ▪ Transfer contents of the beaker to a clean 100ml volumetric flask through a clean plastic funnel. ▪ Wash beaker into the funnel several times with deionised water. ▪ Make up to volume with de-ionised water and mix well. ▪ Transfer the solution to a test tube. ▪ The solution is analysed by AAS. AAS: ▪ Switch on the AAS instrument to warm up for an hour. ▪ Optimise the hollow cathode lamp and select the appropriate standards. ▪ prepare the calibration graph, read the clear solution and record the results ▪ Any samples above the calibration range are diluted 10 times and re-read. ▪ Results are entered into an MS Excel file and also into Centric. ▪ Detection limits are 0.01% for all methods and elements. Diamond drilling: ▪ All diamond core samples prior to 2011 were assayed at: – ALS Chemex Laboratory, Johannesburg – McPhar Laboratory, Philippines – ACTLabs Laboratory, Perth ▪ Samples were analysed for total copper and acid soluble copper with some having a full suite of elements analysed with a four acid digest and ICP-OES analysis. ▪ From 2011, prepared samples were submitted to the SGS Laboratory in Johannesburg (ISO 17025 accredited). The assay scheme is complex and it involves: – ICP40B – Mixed elements, 32 elements suite including Cu from 0.5ppm to 1% o Between 0.5g and 2g is decomposed by strong acid digestion using HF, HClO and HNO3 acids. o It is then evaporated to dryness. o The precipitate is leached in concentrated HCl acid, transferred to a flask and diluted with distilled water, with HCL representing 10% of the final volume. o It is then analysed by ICP-OES. – ICP90A – Alkali fusion for over range Cu and Co grades. o Pulverised sample amounting to 0.5g to 2g is weighed into a crucible and NaO2 added and the sample is fused. o The sample is acid leached as above and made up to volume. o It is then analysed by ICP-OES. o A minimum of one reagent blank and certified in-house reference material and one replicate is used for every 50 samples. – XRF75G – XRF for uranium at 10ppm detection limit o Weigh 20g with 3g of binder; mix well in 50cc carbon steel grinding vessel for 5 minutes. o Press 40mm pellet.

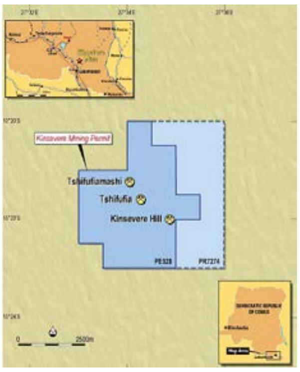
<table>
<tr><td></td><td>
o Conduct XRF analyses.
o Calibration is done using CRM's with background corrections using the Feather & Willis method. Matrix corrections are obtained by calculating the Mass Absorption Coefficients with drift corrections accomplished by inserting drift monitors.
o A consolidated ppm reading is obtained by comparison with calibration plots of certified calibration standards.
– AAS72C – Cold acid (sulphuric) for Cu and Co
o A pulverised sample amounting to 0.5g is weighed and sulphuric acid is added and the sample agitated for one hour.
o The solution is left for 30 minutes then diluted with 100ml of distilled water.
o The sample is analysed by AAS and read with standards according to the copper content.
o QAQC employs in house and/or standard reference materials and blanks for every batch of 50 samples analysed.
</td></tr>
<tr><td>Verification of sampling and assaying</td><td>
▪ Assay results are verified in section with the Mineral Resource model, previous drilling, logging and mapping data.
▪ RC logging carried out and checked by team of experienced geologists.
▪ RC data is loaded into industry-standard DataShed database with built-in validation and rigorous QAQC reporting.
▪ Core logging data recorded in Excel spread sheets by experienced geologists. Then transferred into a GIBIS database on the MMG Server.
▪ Core assay results quality is checked using GBis by assessing standards, duplicates and standards correlations on a monthly basis. This check is done by the site geologists. If quality control issues are identified the entire batch is reanalysed.
</td></tr>
<tr><td>Location of data points</td><td>
▪ All grade control RC drillhole collar surveys are undertaken by qualified surveyors.
▪ Coordinates are in Kinsevere Mine Grid (a close approximation of WGS84). Transformation to Mine Grid involves subtracting 8000000 to the northing and subtracting 22.3m to the elevation (Table 45).
Table 45 Transformation to mine grid
<table>
<tr><th></th><th>Easting</th><th>Northing</th><th>Elevation</th></tr>
<tr><td>WGS84</td><td>563801</td><td>8743404</td><td>1234</td></tr>
<tr><td>Mine Grid</td><td>563801</td><td>743404</td><td>1211.7</td></tr>
</table>
▪ Down-hole dip recorded with Reflex single shot camera up to 2011.
</td></tr>
<tr><td>Data spacing and distribution</td><td>
▪ Grade control (RC) drill pattern spacing is 5m x 15m. Key criteria used to define spacing includes:
– Sufficiently close drill grid spacing to adequately define areas close to contacts or transition zones with adjacent/surrounding waste.
– Cover the horizontal extents of mineralisation.
– Optimise sample recovery for different lithologies.
– The overall diamond drilling space at Kinsevere is from 30m to 100m.
▪ The 2012 diamond drilling was done to infill the primary sulphide resource at approximately 25m to 50m spacing.
</td></tr>
<tr><td>Orientation of data in relation to geological structure</td><td>
▪ Drillholes are oriented such that holes have a high angle of intersection with dominant strike and dip of bedding and structures, the local scale of mineralisation is also considered.
▪ RC grade control holes are oriented either east or west with dip of- 60°.
▪ Diamond holes are orientated either east or west with dip that varies from -60° to sub vertical
</td></tr>
<tr><td>Sample security</td><td>
▪ Measures to provide sample security include:
– Adequately trained and supervised sampling personnel.
– Sea containers where samples are stored are locked with keys given to security department.
– Assay laboratory checks of sample dispatch numbers against submission documents.
</td></tr>
<tr><td>Audit and reviews</td><td>
▪ An internal MMG Geology Review was undertaken in 2013. No high risk issues were identified.
▪ Independent review conducted on sample preparation laboratory in 2012 by Richard Carver. All recommendations were actioned and laboratory passed audit.
</td></tr>
</table>

Section 2 Reporting of Exploration Results

Mineral tenement and land tenure status

- The Kinsevere Mining Licence (PE 528) is located approximately 27km north of Lubumbashi, the provincial capital of the Katanga Province, in the southeast of the Democratic Republic of the Congo (DRC). It covers an area of approximately 5.94km^2 as shown in Figure 36.

Figure 36 Location of the Kinsevere Mining Permit



- The mineral rights of PE 528 are held by La Générale des Carrierés et des Mines (Gécamines), the DRC state-owned copper mining company. Anvil mining, via its subsidiary AMCK Mining s.p.r.l. (AMCK, a joint venture between Anvil (95%) and Mining Company of Katanga s.p.r.l. (5%) has a Contrat d'Amodiation[3] (Lease Agreement) with Gécamines to mine and process ore from PE 528 until 2024, followed by a 15 year extension. Anvil Mining sold the Kinsevere project to MMG in 2012. The PE 528 permit covers the three major deposits of Tshifufiamashi, Tshifufia and Kinsevere Hill/Kilongo. The Tshifufiamashi deposit is locally referred to as "Mashi". The Tshifufia deposit comprises Tshifufia North, Central and South, and is locally referred to collectively as "Central". The Kilongo deposit extends north westwards from the Kinsevere Hill deposit and is sometimes referred to as "Kinsevere Hill Extended". PE 528 encloses the area for the planned mines, process plant, tailings storage facility and other unmovable infrastructure. In January 2007, Gécamines made an application to the Cadastre Minière (CAMI) to have PE 528 extended to cover the then recently defined extensions to mineralisation and provide space for the mine infrastructure (tailings dam, Stage I EAF, accommodation camps, etc.). This application was approved
- A royalty of 2.5% of gross revenue were adopted in January 2009 to reflect revised royalty payments after the Government reviewed all the mining contracts in 2008 where the terms of the Lease Agreement were amended.

Exploration done by other parties

- In the 1990's, Gécamines started the first surface exploration works (trenches, pits) in the Kinsevere tenement and that was followed by drilling programs on a joint venture contract between Gécamines and Exaco before the involvement of Anvil Mining in 2004. Table 46 summarises the previous exploration work.

Table 46 Summary of Previous Exploration Work by Gécamines and EXACO

Deposit	Pitting	Trenching		Drilling	
	No (m depth)	No. (metres)	Significant Grades	No. holes (metres)	Significant Grades
Tshifufiamashi	11	16 (1,304 m)	5.8% Cu 0.2% Co over 50 m	37 (846 m)	10.5% Cu 0.72% Co over 22.2 m
Tshifufia Central	-	17 (1,106 m)	7.6% Cu 0.3% Co over 15 m	19 (950 m)	6.3% Cu 0.6% Co over 23 m
Tshifufia South	-	39 (278 m)	7.2% Cu 0.3% Co over 40 m	11 (497 m)	
Kinsevere Hill	7 (44 m max.)	11 (625 m)	6.6% Cu 0.2% Co over 20 m	10 (1,021 m)	3.99% Cu 0.22% Co over 14.6 m

- In 2004, Anvil Mining carried out intense exploration in five phases to well define the deposits in Kinsevere.

---

[3] a *Contract d'Amodiation* is provided for under the DRC Mining Code, enacted by law No 007/2002 of July 11, 2002.

Geology

- The Kinsevere copper deposit is hosted in moderately to steeply dipping Neoproterozoic sedimentary formation of the Roan group of the Katanga stratigraphy in the Mine Series (R2) subgroup of Katangan Copperbelt.
- On surface, the Kinsevere copper deposit has been mapped as made of three separate Mine Series fragments (large breccia clasts of the Mine Series) whereby the first two fragments are situated along a major north-south oriented fracture and separated by a sinistral strike-slip fault, while the third fragment, called Kinsevere Hill, is situated along major northwest-southeast fracture and separated from the other fragments by another sinistral strike-slip fault. All these fragments are affected by fractures and breccias.
- The sulphide and oxide mineralisation in the Kinsevere copper deposit are either disseminated in recrystallized layers or infilling bedding plans, reactivated bedding, fractures and joints. The sulphides include: pyrite, chalcopyrite, bornite and chalcocite. Although in the supergene zone, sulphides are partially or completely replaced by malachite and other copper oxide minerals.

Table 47 Kinsevere Mine Series stratigraphy

Formation	Unit	Lithology	Comments	Mineralisation	Thickness
Kambove Dolomite *CMN* R2.3	Upper R2.3.2.	Pale coloured dolostone;	Stromatolitic & cherty		80-120m
	R2.3.2.	Cyclic dolomite & pale olive shale towards base	Pink brown-white massive; minor anhydrite; mineralised. evaporitic breccia	THIRD OREBODY (lenticular)	
	R2.3.1.	Grey or black dolostone & shales	Laminated, locally carbonaceous.		<50m
R2.2 Dolomitic Shales	*SD*	Where fresh, mostly graphitic shale and siltstone with minor dolomitic shale with evaporitic texture. Flaggy siltstone at base	BOMZ & SDB not defined or developed at Kinsevere. More dolomitic towards top	UPPER OREBODY	60-90m
R2.1	*RSC*	Silicified dolomite	Vuggy; stromatolitic	ABSENT AT KINSEVERE	
	RSF	Finely banded laminated argillaceous dolostone	Weakly silicified at Kinsevere	LOWER OREBODY	<2m
	DStrat	Fine >coarsely banded, planar bedded shaley dolomite	Distinct 1-5cm nodules replaced by silica/dolomite or sulphides.		3-4m
	Grey RAT	Chloritic & dolomitic sandy argillite, siltstone	Massive, weakly sandy. Reducing environment. Basal facies less mineralised		8-20m
R1	Red & Undifferentiated *RAT*	Massive to poorly bedded and silty argillite	Pink, maroon to white & chloritic	Minor superficial oxide mineralisation	>200m?

Drillhole information

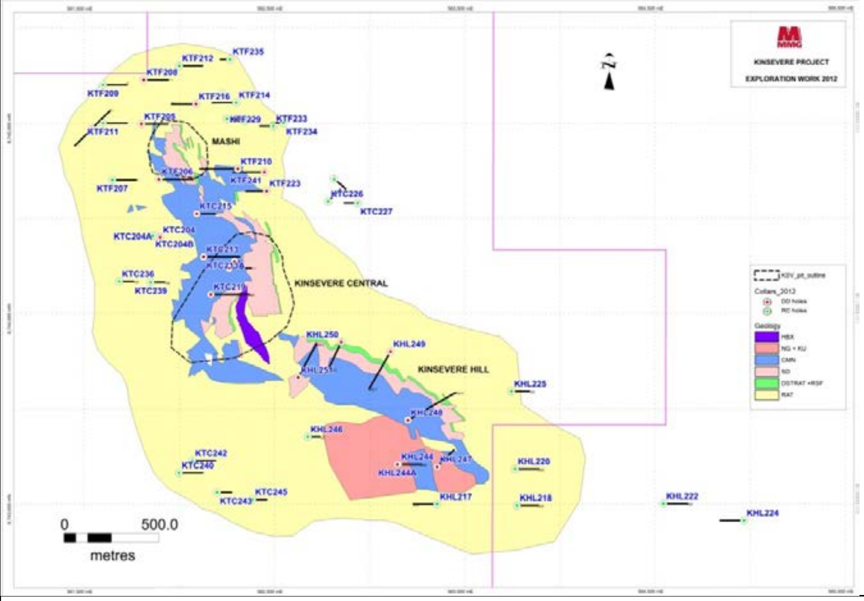
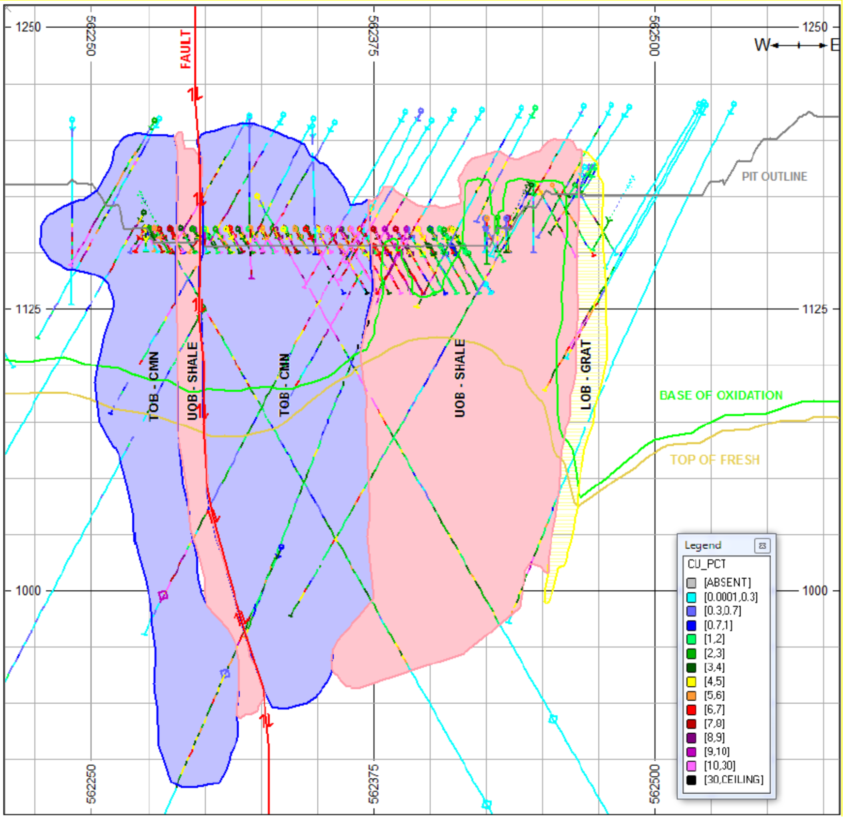
- 1,698 drillholes including diamond, RC, air-core, holes and associated data are held in the database.

Data aggregation methods

- No metal equivalents were used in the Mineral Resource estimation

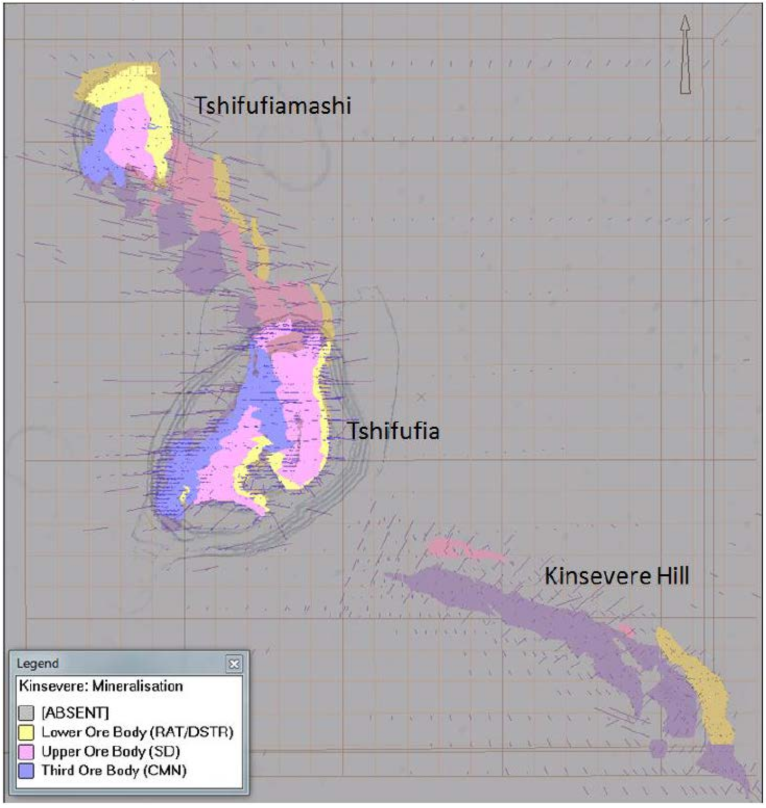
Relationship between mineralisation width and intercepts lengths

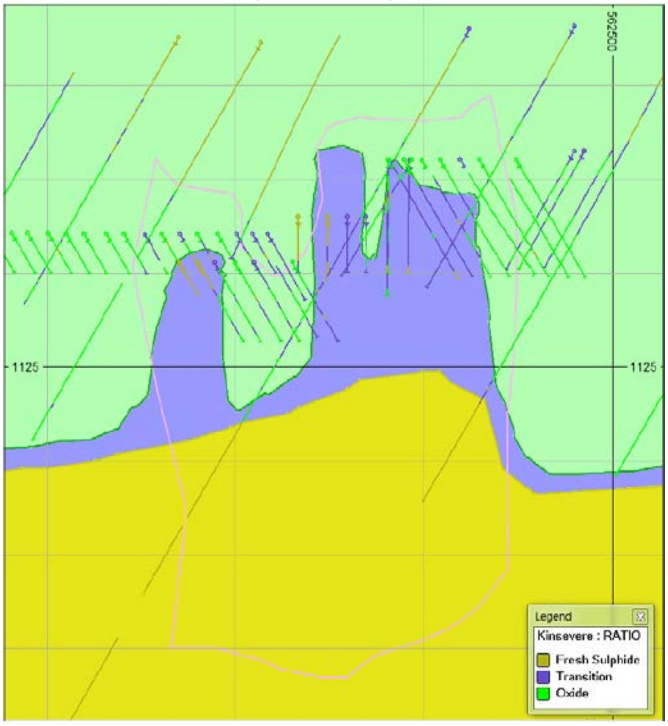
- Mineralisation true widths are captured by interpreted mineralisation 3D wireframes.
- Most drilling was at 50 to 60 degrees angles in order to maximise true width intersections.

Diagrams	**Figure 37 Plan view of the Kinsevere deposit** 
Balanced reporting	**Figure 38 Typical section through Tshifufia pit showing drilling interceptions (TCu%)** 
Other substantive exploration data	▪ Calcium estimates have been generated to assist the long term scheduling of the Mineral Resource in optimising the acid consumption required for economical copper extraction. Tabulation of the in situ calcium is provided in Section 3 "Metallurgical factors or assumptions" of this table.
Further work	▪ Airborne EM survey planned for September 2013 will cover the Kinsevere licence. ▪ Additional drilling is planned in the Kinsevere Hill to constrain the 0.54% Cu grade-shell and close off open-ended copper mineralisation. ▪ Drilling targets in the Northern part of Kinsevere Licence will be guided by the proposed airborne EM survey.

Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	- The complete drillhole database (RC grade control and diamond drilling) data is stored in two SQL databases using the DataShed and GBis front end management systems. - The grade control data is stored in DataShed, which is managed by the onsite Geology Group. - The exploration/resource (diamond drilling) database is stored in a GBis database. Management of this database is performed by the Melbourne Exploration Group. - All data in the database is exposed to standard logging codes and validation processes. - All drillhole data was exported to .csv format and desurveyed in Datamine. - Visual checks of collar, down hole survey, lithology and assay values done in Datamine in both section and plan view. - Any data errors were communicated to the Database group to be fixed in GBis/Datashed.

Site visits	- The Competent Person visited site on various occasions during 2012 and 2013. Site visits involvement with:
	  – Updating of mineralisation wireframes.
	  – Daily open pit visits and core yard visits and discussion with mine and exploration geologists on Kinsevere geology and mineralisation.
	  – Assist in updating the open pit grade control system.

Geological interpretation	- Wireframes solids and surfaces were created for the domains and zones of similar weathering, stratigraphy and style of mineralisation.
	- String envelopes were digitised along drill sections using a 0.3% total copper cut-off. The 0.3% total copper cut-off is a good indicator and marker for the mineralisation domains.
	- Geological logging was also used in determining the mineralisation domains and accordingly:
	  – **Lower Ore Body (LOB)** is associated with the *Dolomite Stratifiee* (DStrat), the *Roche Siliceuse Feuilletee* (RSF) and *Roche Argilo-Talqueuse* (GRAT).
	  – **Upper Ore Body (UOB)** is associated with the *Shale Dolomitiques* (SD).
	  – **Third Ore Body (TOB)** is associated with the *Calcaire a Mineraux* Noirs (CMN).
	- Figure 39 shows a plan view of the Kinsevere deposit showing the LOB, UOB and TOB domains.
	Figure 39 Plan view of the Kinsevere mineralisation domains
	
	
	- Each domain of mineralisation is influenced by weathering, oxidation and structural features such as faulting. The mineralisation domains were further subdivided into a soil, weathering, oxide, transition and fresh sulphide zone.

	▪ Weathering surface wireframes are also constructed to define the oxide, transition and primary surfaces (Figure 40). The magnitude of the acid soluble copper/total copper (ASCu/TCu) ratio has been used as an important criterion for modelling continuous zones of oxide, transition and fresh sulphide mineralisation. The following ratios have been used: – Oxide > 0.8 – Transition between 0.3 and 0.8 – Fresh < 0.3 **Figure 40 Cross-sectional view looking north showing the oxide, transition and fresh domains**  ▪ The resulting weathering, oxide, lithology and mineralisation domains were combined to code the drillhole data and empty block model used for estimation.
Dimensions	▪ The Kinsevere Project is located in the Katanga Province, in the southeast of the Democratic Republic of the Congo. It is situated approximately 27km north of the provincial capital, Lubumbashi, at latitude S 11° 21' 30" and longitude E 27° 34' 00". ▪ The strike length of mineralisation is approximately 1.3km for the Tshifufia and Tshifufiamashi deposits and dipping sub vertical. ▪ Mineralisation extends to 400m at depth and it can be up to 300m in width.
Estimation and modelling techniques	▪ Resource modelling was done using Datamine software. ▪ Mineralisation wireframes and surfaces of the topography, soil, base of weathering, oxide, transition and fresh are used to tag the drillholes by the mineralisation domain used for statistical analysis and grade estimation. ▪ Drillhole composite to 2m intervals. ▪ Grade-capping was done post compositing. Values greater than the selected cap value were set to the grade cap value and used in the estimation. ▪ Separate variography was performed for total copper % (TCu %), cobalt % (Co%) and calcium (Ca%). ▪ Search parameters for TCu%, Co% and Ca% estimate derived from mineralisation domain variography. ▪ Acid soluble copper (ASCu%) search and estimation parameters are the same as the TCu%. ▪ TCu%, ASCu%, Co% and Ca% were estimated using Ordinary Kriging (OK). ▪ Iron (Fe%), Magnesium (Mg%), Molybdenum (Mo%), Sulphur (S%) and Uranium (U%) were estimated as Inverse Distance Squared (ID^2) ▪ Fe%, Mg%, Mo%, S% and U% search parameters were based on a generic search of 45m x 20m x15m and using the deposit code (e.g. Mashi). ▪ Grade estimation done using a combination of hard and soft boundaries. The UOB transition and oxide and UOB fresh and transition use a soft boundary. The LOB and TOB use hard boundary estimation.

	▪ The soft boundary is one way (UOB transition can use UOB oxide samples but UOB oxide cannot use UOB transition). ▪ Estimation parameters for OK based on variography of drilling data. ▪ First estimation pass search radius uses the variogram range (98% of the Mineral Resources are interpolated during the first pass). ▪ Second search set to twice the variogram range (less than 2% of the Mineral Resources are interpolated during the second pass). ▪ Third pass used to estimate any unestimated values. This was set to five times the variogram range and is unchanged from previous estimates and needs to be reviewed for future work. 0.3% of the Mineral Resources are interpolated during the third pass. ▪ The Datamine Dynamic Anisotropy (DA) method was used to honour the mineralisation strike and dip variations thus improving the quality of the local estimate. ▪ Minimum of 8 samples and a maximum of 30 to 48 (depending on domain) were required for an estimate. ▪ Estimation was limited to a maximum of 5 or 8 samples depending on domain per drillholes for the TCu%, ASCu% and Co%. ▪ Octant search was used for domains that have a combination of grade control data and surface drilling data (for TCu%, ASCu% and Co%). ▪ Octant search and a minimum number of drillholes “restriction” was not used for estimating Ca%, Fe%, Mg%, Mo%, S% and U% due to lack of data. ▪ Unestimated Ca% values in blocks that have ASCu% >0.3 have been assigned the Ca% mean value of the domain and flagged accordingly. ▪ Parent block size in the grade control volume model was set to 5m x 10m x 5m with sub-blocking down to 2.5m. ▪ The rest of the Kinsevere block model is 10m x 20m x 5m with sub-blocking down to 2.5m. ▪ Estimation into parent block. ▪ Discretisation of 4 (X points) X 8 (Y points) X 2 (Z points). ▪ Kriging variance (KV), kriging efficiency (KE) and kriging slope of regression slope (SOR) were calculated during the estimate. These in conjunction with the drilling density we used to construct wireframes to select and assign the Mineral Resource classification.
Moisture	▪ Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	▪ The oxide Mineral Resource has been reported on an acid soluble copper grade of 0.75%. The current cut-off used for mining is 1% ASCu. ▪ The primary sulphide Mineral Resource has been reported on a total copper cut-off grade of 0.75%. This cut-off represents material that has a reasonable prospect for eventual economic extraction at some point within the next 15 years.
Mining Factors or assumptions	▪ No mining factors or assumptions have been applied to the Mineral Resource.

<table>
<tr><td>Metallurgical factors or assumptions</td><td>

- No metallurgical factors or assumptions have been applied to the Mineral Resource.
- A calcium estimate has been generated to assist Long Term Planning in scheduling more realistic acid consumption (and cost) to the Ore Reserves. Calcium values are tabulated in Table 48 and Table 49 for both the oxide and primary in situ' Mineral Resource.

Table 48 30 June 2013 oxide Mineral Resource (in situ' & depleted) 0.75% acid soluble copper cut-off

<table>
<tr><th></th><th></th><th>Tonnes (MT)</th><th>ASCu (%)</th><th>TCu (%)</th><th>Co (%)</th><th>Ca (%)</th></tr>
<tr><td>Oxide Mineral Resource</td><td>Measured</td><td>12.2</td><td>3.2</td><td>4.0</td><td>0.2</td><td>0.3</td></tr>
<tr><td></td><td>Indicated</td><td>12.0</td><td>2.5</td><td>2.9</td><td>0.1</td><td>1.3</td></tr>
<tr><td></td><td>Inferred</td><td>0.8</td><td>2.0</td><td>2.5</td><td>0.1</td><td>1.0</td></tr>
<tr><td></td><td>Total (M+I+I)</td><td>25</td><td>2.8</td><td>3.5</td><td>0.2</td><td>0.8</td></tr>
</table>

Table 49 30 June 2013 sulphide Mineral Resource (in situ' & depleted) 0.75% total copper cut-off

<table>
<tr><th></th><th></th><th>Tonnes (MT)</th><th>TCu (%)</th><th>ASCu (%)</th><th>Co (%)</th><th>Ca (%)</th></tr>
<tr><td>Sulphide Mineral Resource</td><td>Measured</td><td>1.5</td><td>2.7</td><td>0.9</td><td>0.2</td><td>1.1</td></tr>
<tr><td></td><td>Indicated</td><td>10.1</td><td>2.7</td><td>0.6</td><td>0.2</td><td>1.1</td></tr>
<tr><td></td><td>Inferred</td><td>10.9</td><td>2.2</td><td>0.3</td><td>0.1</td><td>1.5</td></tr>
<tr><td></td><td>Total (M+I+I)</td><td>22.5</td><td>2.5</td><td>0.5</td><td>0.1</td><td>1.5</td></tr>
</table>
</td></tr>
<tr><td>Environmental factors or assumptions</td><td>

- No environmental factors have been applied to the Mineral Resource estimate.
</td></tr>
<tr><td>Bulk Density</td><td>

- Density values were assigned to the block model per lithology and weathering or oxide domain.
- Assigned values were determined from 1,696 diamond core density measurements, four in-pit bulk sample measurements and a series of twelve in-pit measurements from specific lithologies having different degrees of weathering.
</td></tr>
<tr><td>Classification</td><td>

- The Measured and Indicated Mineral Resource classification wireframes have remained unchanged from 2012. The Inferred wireframe has been adjusted to include results from the 2013 surface drilling program in the primary sulphide mineralisation.
- These wireframes are based on a combination of confidence in assayed grade, geological continuity, resulting kriged estimate and their efficiencies. Kriging variance, efficiency and "slope of regression" have been calculated for the Mineral Resource. These have been used to assist in the creation of the Mineral Resource wireframes that are used to assign the classification to the block model.
</td></tr>
<tr><td>Audits or reviews</td><td>

- Internal MMG per review conducted by Anna Lewin (Senior Resource Geologist) in July 2013. The following recommendations were raised. None of these are considered material to the Resource with a very small percentage of the Resource blocks (less than 1%) effected:
  - Review the Mineral Resource classification. Including downgrading blocks that are unestimated due to new search parameters.
  - Adjust the classification of blocks that have an average Ca value assigned to them.
  - Eliminate gaps in the block model that have no blocks due to mineralisation wireframes cross overs.
  - Review the current parent block size and determine if a larger size is more appropriate.
</td></tr>
<tr><td>Discussion of relative accuracy / confidence</td><td>

- Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support.
</td></tr>
</table>

5.5 Ore Reserves - Kinsevere

5.5.1 Results

The 2013 Kinsevere Ore Reserves are based on the 2013 Mineral Resource model.

The 2013 Kinsevere Ore Reserves are summarised in Table 50.

Table 50 2013 Kinsevere Ore Reserves tonnage and grade (as at 30 June 2013)

Kinsevere Ore Reserves

				Contained Metal	
	Tonnes (Mt)	Copper (%TCu) *	Copper (%ASCu) *	**Copper ('000 t)**	**Copper ASCu * ('000 t)**
Proved	10	4.8	3.9	470	380
Probable	11	2.8	2.2	310	240
Total Ore Reserves	**21**	**3.8**	**3.0**	**790**	**620**

Ore Reserves are generally rounded and reported to 2 significant figures to reflect confidence in estimates. Totals may differ due to rounding

Contained metal does not imply recoverable metal.

* TCu stands for Total Copper, ASCu stands for Acid Soluble Copper.

Details of relevant modifying factors used in estimating Ore Reserves are given in the Technical Appendix published on the MMG website

Competent Person:

Julian Poniewierski (Member of AusIMM (CP), employee of MMG)

The three major differences from the 2012 Ore Reserves are:

(i) Due to the uncertainty of the future site costs all material (Measured and Indicated) that is above 0.85% and below 1.6% ASCu (the direct ROM feed cut-off grade) was classified as Probable.
(ii) Break-even cut-off grade has increased from 0.55% ASCu to 0.85% ASCu as a result of increasing operating costs, in particular the power costs.
(iii) A new Mineral Resource model that has resulted in a decrease in both tonnes and grade.

The material that is classified as Measured and has a grade higher than 1.6% ASCu was classified as Proved.

Approximately 73% of Measured Mineral Resource has been converted to Proved Ore Reserves.

The Probable Ore Reserves from the pit contain approximately 3Mt of Measured Mineral Resource.

The deposit is well drilled out and there is very little (less than 100,000 tonnes) of Inferred economic Mineral Resource in the pit designs for the Ore Reserves. All inferred Mineral Resource was reported together with the waste.

The individual sources of the Ore Reserves are shown in Table 51.

Table 51 2013 Kinsevere Ore Reserves tonnage and grade (as at 30 June 2013) by major source

	Classification	**Tonnes (Mt)**	**%ASCu**	**%TCu**	**Contained Metal[†]** **Copper ('000t)**
Central	Proved	6.6	4.2	5.2	340
	Probable	4.7	2.2	3.4	160
Mashi	Proved	1.5	3.8	4.4	70
	Probable	1.2	2.1	2.6	30
Kinsevere Hill	Proved	0			
	Probable	3.2	2.9	3.2	100
Stockpiles	Proved	1.6	3.1	3.8	60
	Probable	1.9	1.0	1.2	20
Sub-Total	Proved	9.8	3.9	4.8	470
	Probable	11.0	2.2	2.8	310
	2013 Total	**21**	**3.0**	**3.8**	**790**

[*]Totals may differ due to rounding; [†]Contained metal does not imply recoverable metal. Ore Reserves are generally rounded and reported to 2 significant figures to reflect confidence in estimates.

5.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Julian Poniewierski, confirm that I am the Competent Person for the Kinsevere Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Kinsevere Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited since August 2012.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 767,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 October 2013 was $HKD 1.72).

I verify that the Kinsevere Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in the supporting documentation relating to Ore Reserves as compiled by Aurimas Karosas, Senior Mining Engineer in the Melbourne Group Office of MMG Limited under the supervision of Julian Poniewierski.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Kinsevere Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Julian Poniewierski – 26/11/13

Mauro Bassotti (Witness)

5.5.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 52.

In compiling the Ore Reserves the Competent Person has reviewed the supplied information for reasonableness, but has relied on this advice and information to be correct.

Table 52 Contributing experts – Kinsevere Mine Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Mauro Bassotti, Senior Resource Geologist MMG Ltd (Melbourne)	Mineral Resource model
Michael Hollitt, Group Manager – Technology MMG Ltd (Melbourne)	Processing parameters, Gangue Acid Consumption information
Tomasz Krolikowski, Commercial Manager MMG Ltd (Kinsevere)	Costs
Mike Turner, Consultant Turner Mining and Geotechnical Pty Ltd	Geotechnical parameters
Aurimas Karosas, Senior Mining Engineer MMG Ltd (Melbourne)	Mining and general information, Whittle optimisation and pit designs
Kinsevere Geology department	Production reconciliation
Knight Piésold	Tailings dam design
Gavin Marre, Senior Business Analyst MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

5.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

5.6.1 Pit Design

Pits were designed to follow the selected Whittle optimisation shells taking geotechnical parameters into consideration, straightening wall sections where required to encourage the stability of the walls.

The minimum mining width of 50m was applied to all cut backs.

The ultimate pits are based on revenue factor 1 pit shells. The penultimate Central pit (stage 4A) was based on the maximum discounted cash-flow shell. Mining during 2014 will be limited to stage 4A, as mining of stage4B will compromise the expected ultimate pit design for a potential sulphides pit operation at Kinsevere – which will be the subject of investigative studies in 2014.

The time value loss of stockpiled low grade ore was taken into account during the Whittle Optimisation process by increasing the break-even cut-off grade by 25%. The basis for this decision was that stockpiled low grade ore will average 5 years in the stockpile during which time its time value decreases to 75% of its value at the time of mining (at an 8% discount rate). Whittle software assigns the value of the low grade material in the optimisation process as its value at the time of mining, not the time of processing.

The resulting pit stage designs are shown in Figure 41, Figure 42, Figure 43 and Figure 44.

Figure 41 **Mashi Pit Stage Design**

Figure 42 Central Pit Stage Design

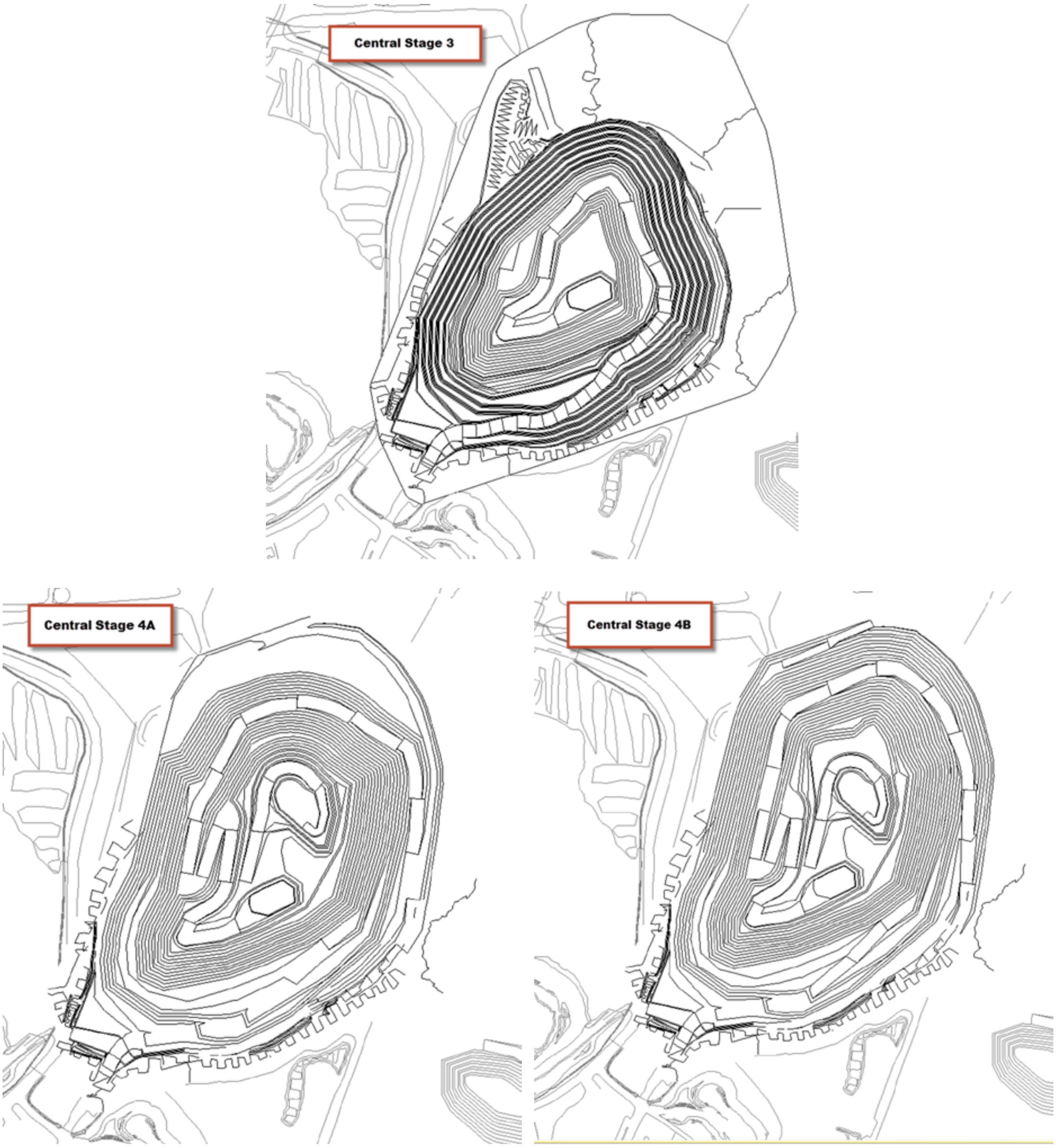


Figure 43 Kinsevere Hill North Pit Stage Design

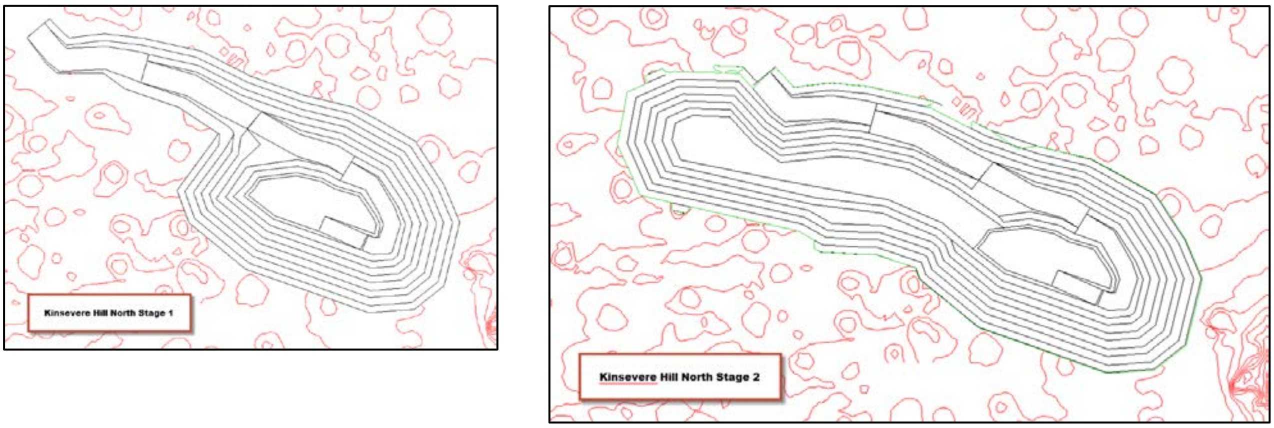
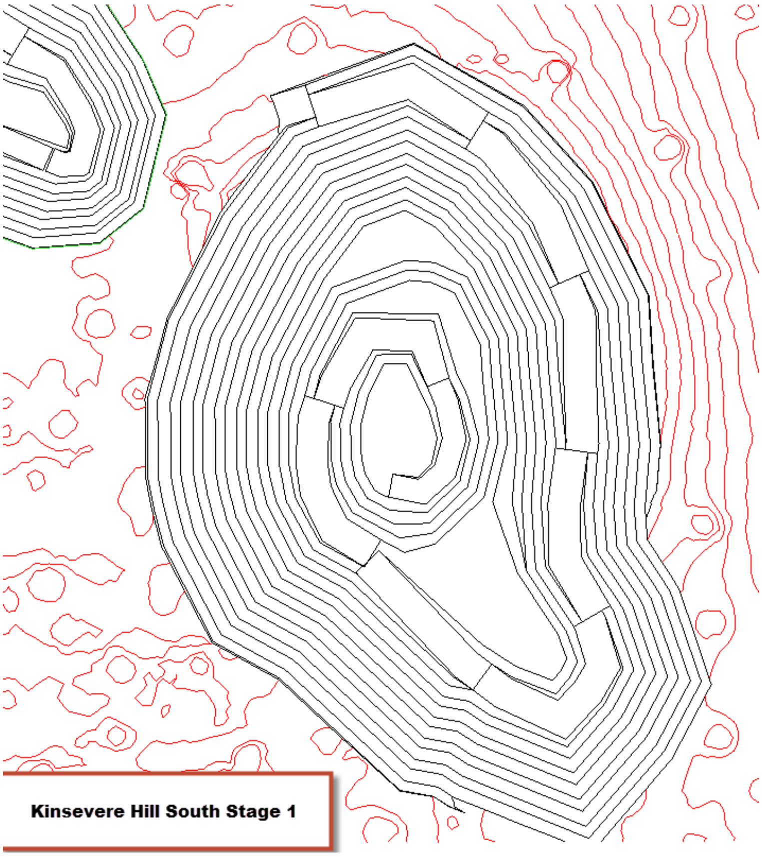


Figure 44 Kinsevere Hill South Pit Design



5.6.2 Geotechnical Parameters

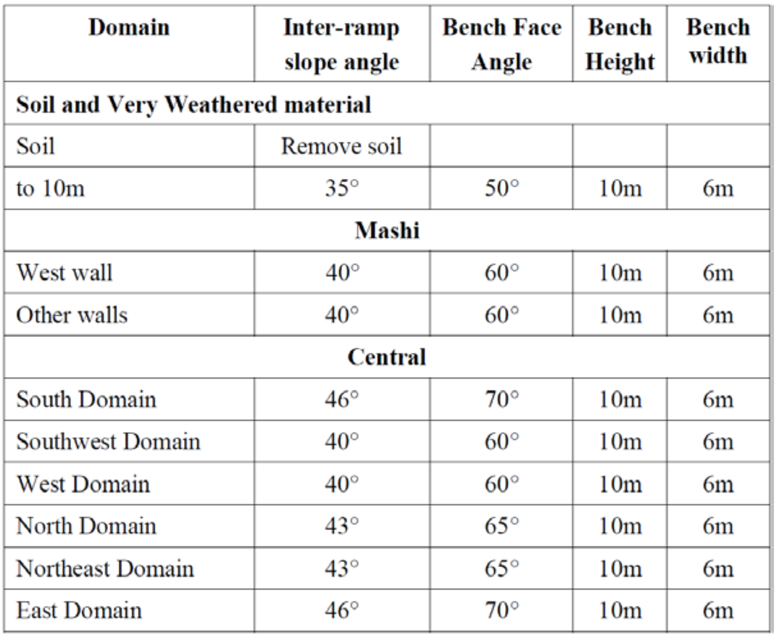
Optimisation and pit design geotechnical parameters were based on a report by consultant: Turner Mining and Geotechnical Pty Ltd. Turner Mining and Geotechnical Pty Ltd have been involved in Kinsevere geotechnical assessment and audits since the start of operations. To date there have been no stability issues.

The report issued by Turner Mining and Geotechnical Pty Ltd (TMG) detailing the findings of a site visit in November 2012 (TMG, January 2013) was used as the basis for the Ore Reserves design work.

Pit wall stability analysis was performed and it was found that the current pit designs fit well with the recommended slopes. The analysis showed that the current slopes appear to be safe with only some minor localised wall failures. As a result of the analysis some of the slope angles in selected regions have been increased. The angles have been increased to a safety factor of 1.25.

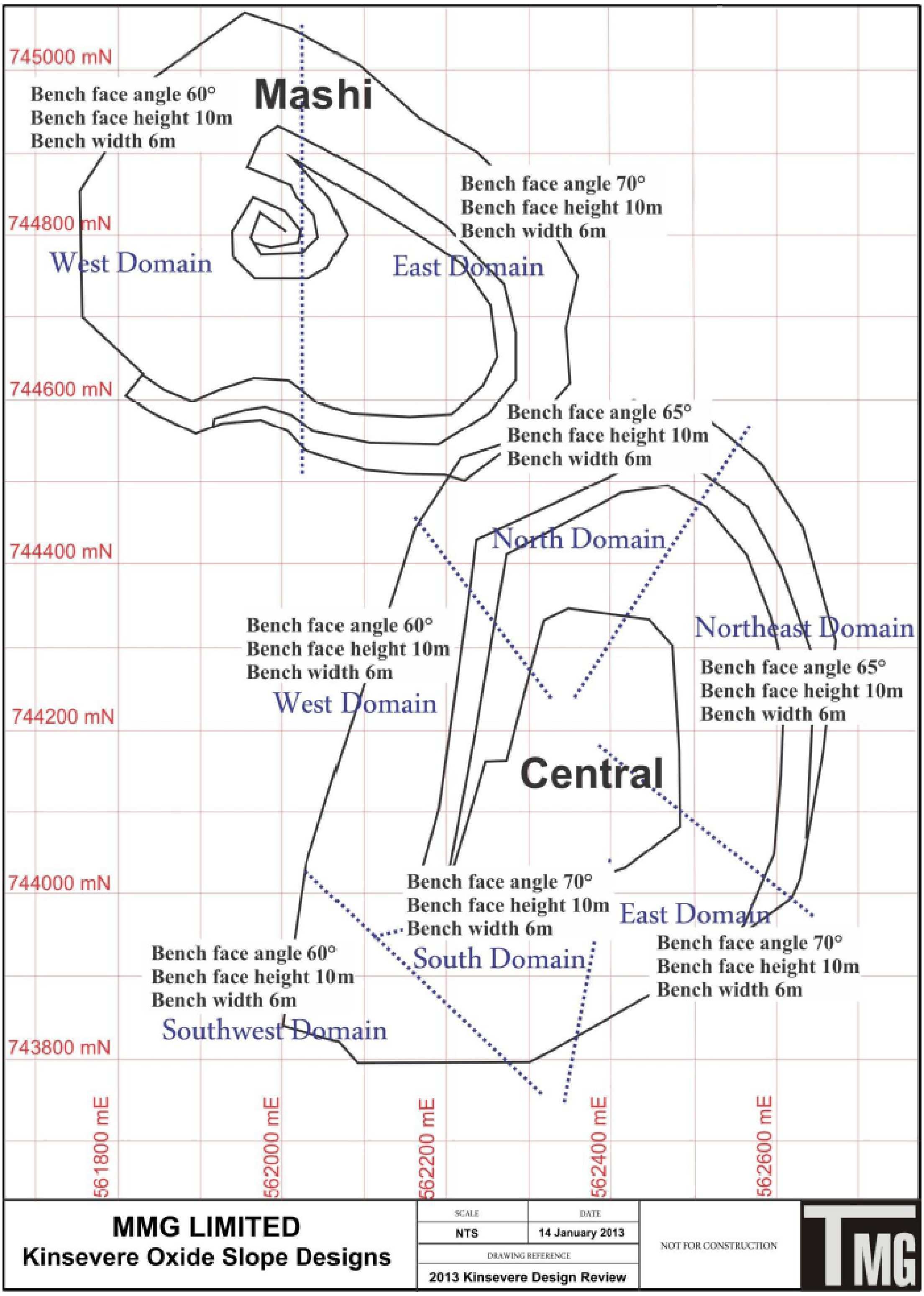
Slope angles by zone are presented in Table 53 and were used as input for both the pit optimisation and pit design processes. The slope design parameters for Central pit were adopted for Kinsevere Hill. Figure 45 shows the design sectors for the Central and Mashi Pits. In calculation of the slope angle and safety factor it was assumed the ground water is drawn down well ahead of the mining and the walls are not saturated.

Table 53 Kinsevere Oxide Pit Wall Angles



Domain	Inter-ramp slope angle	Bench Face Angle	Bench Height	Bench width
Soil and Very Weathered material				
Soil	Remove soil			
to 10m	35°	50°	10m	6m
Mashi				
West wall	40°	60°	10m	6m
Other walls	40°	60°	10m	6m
Central				
South Domain	46°	70°	10m	6m
Southwest Domain	40°	60°	10m	6m
West Domain	40°	60°	10m	6m
North Domain	43°	65°	10m	6m
Northeast Domain	43°	65°	10m	6m
East Domain	46°	70°	10m	6m

Figure 45 Oxide Pit Slope Zones



5.6.3 Processing (Metallurgical) Recovery Factors

The Kinsevere processing plant produces plated copper using acid leaching, solvent extraction and electrowinning as practised at many of the nearby mines on the Zambian Copperbelt.

Key design parameters for the Stage II plant included:

- Ore Treatment Rate 1.62Mtpa
- Ore Head Grade 4.02% Acid Soluble Cu
- Grind P80 212µm
- Plant Utilisations 92% (Crushing), 92% (Milling & Leaching), 94% (EW)
- Leach pulp density 18.5% w/w
- Leach Copper Recovery 94% (Acid Soluble)
- Leach Residence time 6 hours
- Leach pH 1.5

The processing plant was ramping up to full stable name plate capacity during 2012. The recoveries achieved are better than previously expected.

Recovery

Based on the historical data the average copper recovery is 98%.

For the Ore Reserves estimation, metallurgical recovery was estimated as a function of the Total Copper to Acid Soluble Copper ratio. For the ratio of more than 1.04 the recovery was set to 98% of Acid Soluble Copper. The recovery was set to proportionally decrease to the minimum of 94% at ratio of 1.00.

The reasons for the high recovery are a result of recognising that some of the non-acid soluble copper reports to the solution and is recovered in the later stages of the leach circuit. This effect is mostly due to residence time and available oxygen levels in the leach tanks.

During the feasibility study it was not recognised that the calcium (Ca) grades vary significantly in the ore body and that this will impact future gangue acid consumption (GAC). During the process of the Ore Reserves estimation the calcium grades have been included in the economic ore evaluation, allowing variable gangue acid consumption to be estimated. The equation for estimation of gangue acid consumption is a function of both calcium and manganese:

$$\text{GAC} = 46\times\text{Ca}\%+17\times\text{Mn}\%+6$$

In addition to being a major cost imposition on the processing costs, the plant has a physical limit to the gangue acid consumption that it can process. Blending of the feed material based on total gangue acid consumption will be required in the future.

A recognised ore type called “black shale” causes problems in the processing plant if fed at too high a proportion. In addition black shales with an acid-soluble to total copper ratio less than 0.5 were classified as non-processable, and were not included in the Ore Reserves.

Electrowinning

The electrowinning (EW) circuit defines the physical limit to the amount of plated copper that can be produced. This capacity is defined by the following equation:

Net plating capacity = No. of EW cells × Faraday's constant × Current Applied × Utilisation × Current Efficiency × Operating Hours/ 1,000,000

At Kinsevere the operating parameters are:

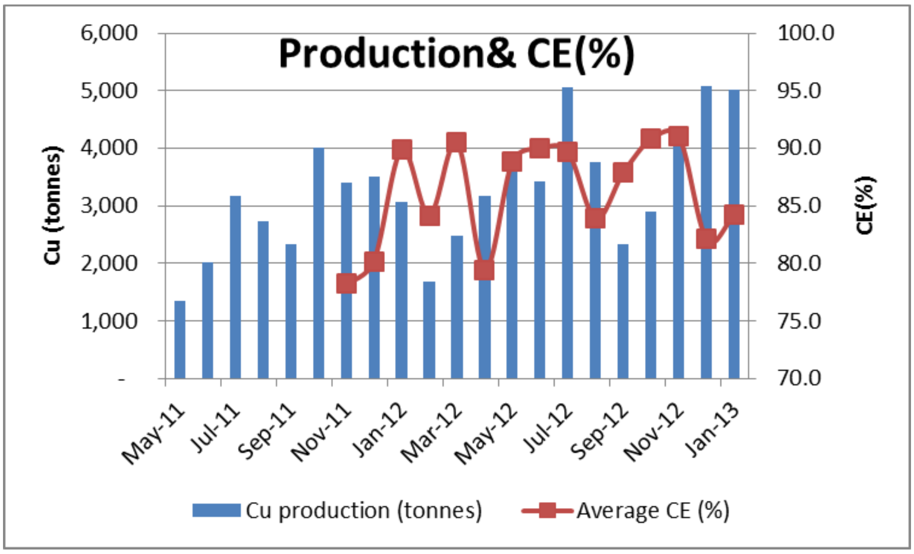
No of cells 78 per tank-house = 156 in total
Faraday's constant = 1.1853
Current of 27,000A per rectiformer for each tank-house = 54,000A
Current Efficiency = 88%
Utilisation = 92%

Using these parameters, the expected net plating capacity would be 70,800 tonnes.

For purposes of pit optimisation and cost analyses, 60,000 tonnes of copper plating has been assumed per annum.

The historical current efficiency is shown in Figure 46.

Figure 46 Kinsevere electro-winning circuit production and current efficiency - historical data



Solvent Extraction

In order for the EW circuit to be able to produce at the net plating capacity of 70,800 tonnes, the solvent extraction (SX) circuit needs to be able to extract that quantity of copper metal into solution.

The design flow capacity of the SX circuit is 641m^3/h. Currently the plant is running at an average flow of 614m^3/h on high grade ore and 515m^3/h on low grade ore. Increasing the flow to design capacity is currently a focus area of the site and MMG technical teams.

Comminution

The comminution circuit is currently considered a bottle-neck with respect to the tonnage that can be milled. That tonnage limit defines the grade material that needs to be fed to the mill in order to get the required copper into the SX circuit. Currently this bottleneck is circumvented to some extent by direct feed of copper in solution from a small heap leaching operation (being operated at the edge of the TSF, and treating scats left over from the Stage I processing).

Assuming operating parameters of 265 wtph, 90% utilisation, moisture content of 8%, and scat loss of 2%, the limiting tonnage that can be milled is approximately 1.9Mtpa.

5.6.4 Realised Revenue Factors (Selling Costs)

As the final product at site is the Copper Cathode there are no treatment charges or penalties.

Transportation charges used are as per the contracts in place with Trafigura for the cathode product off-take. These are listed below in Table 54.

Table 54 Cathode selling costs

	2013 costs (US$/tonne of cathode)
Copper cathode cartage	631.43
Assays costs	14.84
Total	646.27

5.6.5 Royalties and Obligations

Royalties payable by the operation are listed in Table 55.

Table 55 Royalties

Royalty Type	Royalty, (% of Gross Revenue)
DRC Government Royalty	2.0%
Gécamines Royalty	2.5%

The mineral rights to Kinsevere are held by Gécamines. MMG has a lease agreement with Gécamines to mine and process ore from Kinsevere until 2024, followed by an automatic fifteen year extension.

In February 2008, the Company (formerly AMCK) was advised by the Minister of Mines that, as a consequence of a Government commission, the terms of the Kinsevere Contrat d'Amodiation were under review. The Company completed its negotiations with Gécamines and the DRC Government in January 2009 and the key details of the amended agreement comprise:

- an increase in royalty payments to 2.5% of gross revenue; and
- the imposition of a further *pas de porte* (entry premium) of $15 million.

The *pas de porte* payments were made in July 2009 ($10 million) and January 2010 ($5 million).

Arising from the transaction with MMG, on the 10th February 2012, AMCK entered into a 'Clarification and Amendment Agreement' with Gécamines involving further payments as follows:

- a pre-payment of royalties, as previously determined, and at normal commercial terms; and
- a tonnage based cash payment ($35/t Cu) for new copper "Mineral Reserves"[4], in terms of contained copper metal, and in excess of the metal content reported as at 31st December 2010[5.]

With the acquisition of the Company by MMG, there are no further Project loan obligations to Trafigura. There are however, agreements in place with Trafigura[6] in respect of continuing cathode product off-take and the supply of acid for the processing plant.

With the completion of the Stage II Project construction and commissioning, there are essentially no further obligations to Ausenco in respect of the engineering, procurement and construction (EPC) agreement. Settlement is forthcoming on disputed claims and counter claims.

MCK has acknowledged and welcomed the acquisition of the Company by MMG, and has agreed to suspend historic claims against the Company for a period of six months from February 2012, in order to allow MMG to integrate the Company into its business. In return for MCK suspending its previous claims, Anvil has agreed to waive its pre-emptive rights, on a one time basis, should MCK elect to transfer its 5% interest in the AMCK joint venture to a third party. During the acquisition process MMG has bought out MCK's share.

5.6.6 Mining, Processing and Administration Costs

The site has been operating at current capacity since September 2011, and historical costs from that date have been used. The costs determined from an analysis of this historical data and used for determination of the Ore Reserves are shown in Table 56 to Table 59.

All costs are in USD.

For determination of costs that depend upon a throughput rate, the front end processing capacity is set to 1.6Mtpa.

---

[4] "Mineral Reserves" is a SAMREC Code term equivalent to "Ore Reserves" in the JORC Code

[5] Declared Ore Reserves as at 31st Dec 2010 have 712,100 tonnes of Acid Soluble Copper.

[6] Trafigura bought into the project and provided a loan for the project. Since the Anvil acquisition by MMG it has no remaining interest in the project.

Table 56 Ore tonnage based processing costs for oxides

INPUT	UNITS	2013 Costs
Plant - ROM Feed	$/t proc.	0.82
Plant - LT Stockpile Reclaim to ROM	$/t proc.	2.47
Plant - Power	$/t proc.	8.68
Sulphuric acid costs	$/t acid	520.00
Variable Sulphuric acid consumption @0.48% Ca: 46xCa%+17xMn%+6	$/t proc.	15.00
Crushing	$/t proc.	0.52
Grinding	$/t proc.	2.64
Leaching	$/t proc.	0.50
CCD/Clar'n / High grade Thickening	$/t proc.	0.85
Tailings Dam	$/t proc.	0.56
Reagents	$/t proc.	0.08
Services	$/t proc.	0.12
Assay Laboratory	$/t proc.	0.97
Met Laboratory	$/t proc.	0.01
Total	**$/t proc.**	**22.53**

Table 57 Metal tonnage based processing costs for oxides (SXEW)

SXEW		2013 Costs
Fixed Costs		
SXEW Fixed Time Costs (Labour)	$/t Cu	1,384
SXEW Plant - Power	$/t Cu	559
Variable Costs		
Solvent Extraction (SX)	$/t Cu	68
Electro-Winning (EW)	$/t Cu	133
Sub-Total	**$/t Cu**	**2,144**
	$/lb Cu	**0.97**

Table 58 Variable mining costs for oxides

	Units	2013 costs
Default Waste and Ore Mining - Contractor L&H average costs	$/bcm	6.88
Mining – Day-works	$/bcm	0.27
Mining - Drill Blast (average taking into account non blasted material)	$/bcm	0.83
Sub-Total	**$/bcm**	**7.98**
Differential Ore Mining Costs		
Mining - Grade Control/Laboratory	$/t proc.	0.74

Table 59 G&A fixed costs for oxides

	2013 costs M$/yr
Administration	31.2
Corporate Overheads	8.7
Social Development Projects	3.0
Technical Services	3.4
Plant	6.7
Salaries/Labour	12.4
Civils and Transport	8.2
Contractor Overheads	1.3
Mining Overheads	8.0
Dewatering	1.3
Sub-Total	**84.1 M$/yr**
G&A Time Cost	**1,384 $/t Cu** **0.63 $/lb Cu**

5.6.7 Mining Factors and Assumptions

The mining activities are undertaken utilising mining contractor that on average moves 6Mtpa. The ore is hauled either directly to the plant or to stockpiles in order to blend it to required grade of 3.9% of acid soluble copper for processing. During the lifetime of the mine significant low grade stockpiles will be built and they will be processed at the end of the lifetime when the high grade ore has been exhausted.

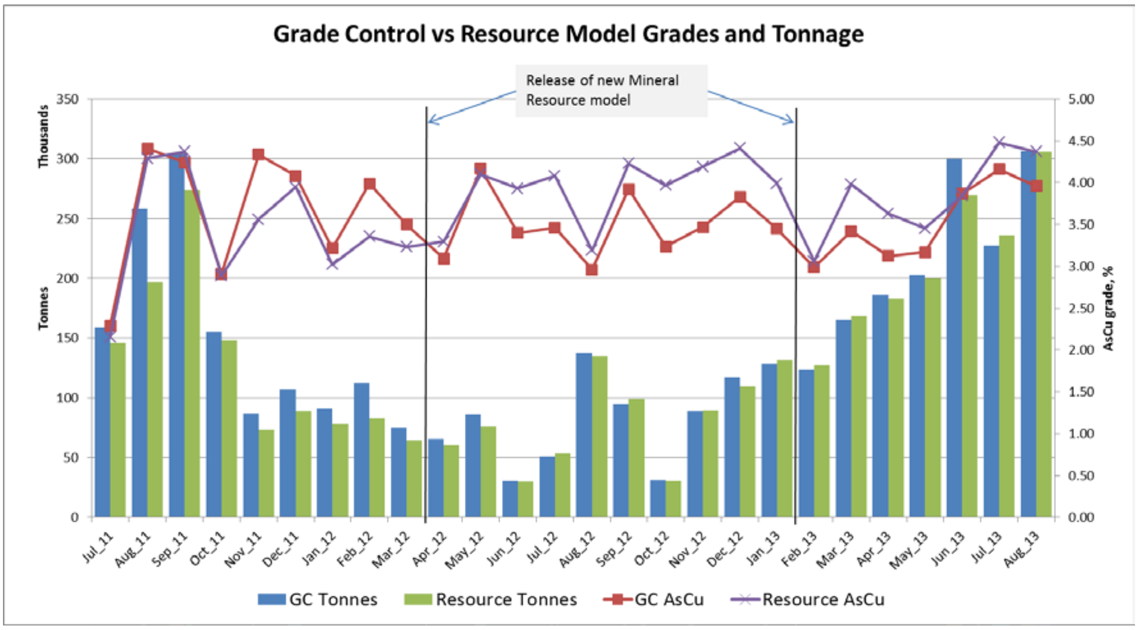
Mine Production Reconciliation

The mine undertakes a production reconciliation process between the grade control model and the Mineral Resource model.

Historically there have been significant issues with the measurement of density – a result of the ore not being homogeneous and with varying porosity and cavities. A lot of time and effort was spent in 2011 in order to improve the estimate of the density. These changes were incorporated in both the grade control model and the next Mineral Resource model released in April 2012. This reduced the variance between the two models. Further modifications were undertaken for the release of the 2013 Mineral Resource model.

Combined reconciliation results for the 3 year period are shown in Figure 47.

Figure 47 Grade control model vs. Mineral Resource model data



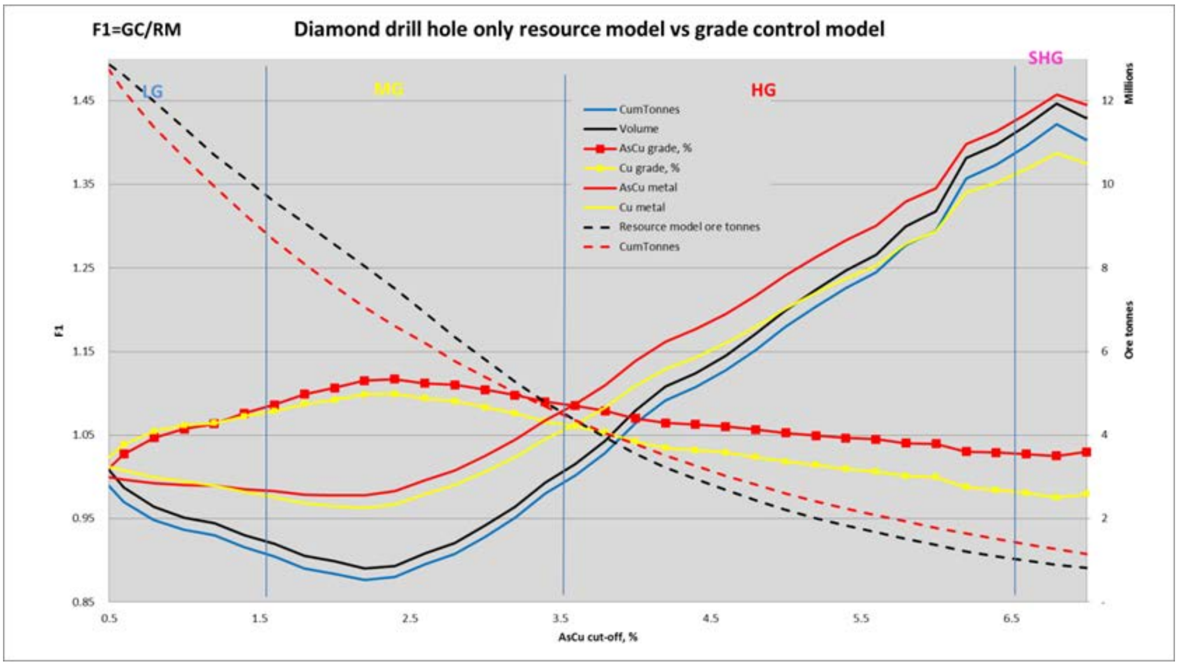
It will be noted in Figure 47 that the reconciliation is best at the time of release of each new Mineral Resource model. The Mineral Resource model used by the mine site incorporates the grade control drilling as at the date of compilation and thus it is expected that the two models should be close to each other in results. As time progresses and more grade control data is added to the grade control model the deviation between the two models increases.

Mineral Resource-Grade Control Models Grade Range Reconciliation

In order to understand the potential errors involved in using the Mineral Resource model for long term planning, a version of the Mineral Resource model was created without the use of grade control data – i.e. using only diamond drillhole data. This model was reconciled against the grade control model for the common volume of both models and for the resource above a range of cut-off grades.

The results of this reconciliation are shown in Figure 48. However during the reconciliation process it was also noted that the site controlled grade control model has a number of significant issues that are still to be resolved and as a result these reconciliation results are subject to considerable uncertainty. As such, the reconciliation results have not been used in any way to modify the stated Ore Reserves.

Figure 48 Grade control model vs. diamond drillhole data only version of Mineral Resource model over a range of acid soluble cut-off grades



Mill Production Reconciliation

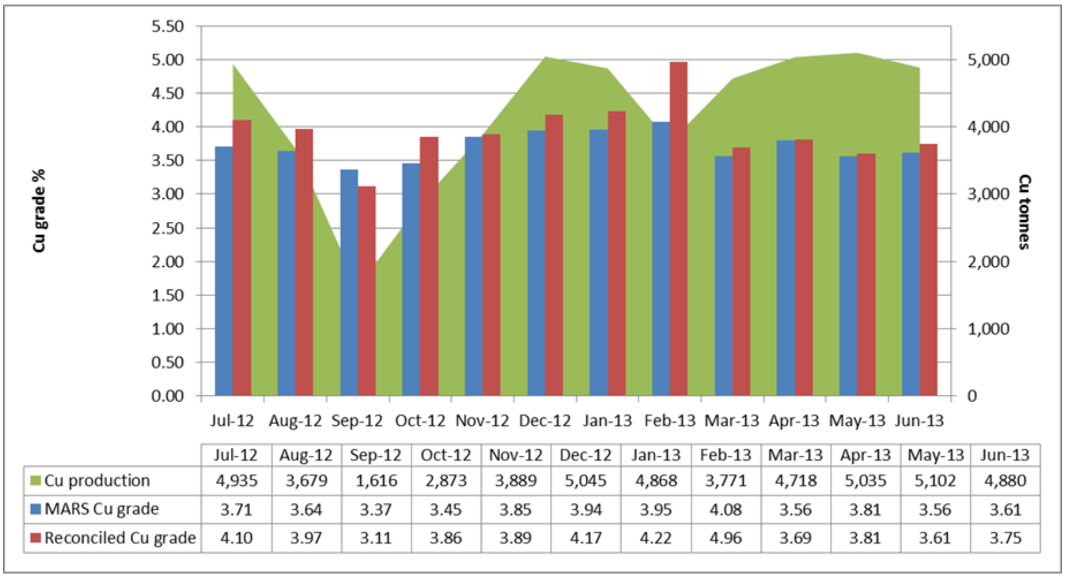
The mill reconciliation data from the last 12 months indicates that the daily estimated Cu mill feed grade has been underestimated over the same period of time. The degree of underestimation is not possible to be determined as the measurement of three critical variables (mill feed moisture, SG leach discharge liquor, and leach discharge density) has not been performed accurately.

Kinsevere uses the program MARS, which was developed on site, for metallurgical accounting purposes. MARS uses a combination of procedures which are linked to excel add-ins, sending and acquiring information from two sets of SQL databases (MARS Anvil Assays and MARS Anvil Production).

Monthly reconciliation is performed adjusting the estimated head grade in order to obtain 100% reconciliation in the monthly mass balance (Total Inputs / (Total Outputs + Δ Cu inventory)).

Figure 49 shows the comparison between copper head grade before and after the reconciliation. It is clear that the copper mill feed grade is being under estimated in the daily mill accounting system.

Figure 49 Mill head grade reconciliation



	Jul-12	Aug-12	Sep-12	Oct-12	Nov-12	Dec-12	Jan-13	Feb-13	Mar-13	Apr-13	May-13	Jun-13
🟩 Cu production	4,935	3,679	1,616	2,873	3,889	5,045	4,868	3,771	4,718	5,035	5,102	4,880
🟦 MARS Cu grade	3.71	3.64	3.37	3.45	3.85	3.94	3.95	4.08	3.56	3.81	3.56	3.61
🟥 Reconciled Cu grade	4.10	3.97	3.11	3.86	3.89	4.17	4.22	4.96	3.69	3.81	3.61	3.75

Dilution and Loss

For the purposes of pit optimisation, an ore dilution value of 5% at 0% grade has been assumed along with an ore loss value of 5%.

No dilution or loss factors have been applied to the Ore Reserves.

Moisture

In situ moisture assumption used was 8%. The tests determining this value were undertaken during the Feasibility Study and have not been changed or re-tested.

5.6.8 Infrastructure

Mining Infrastructure

Mining infrastructure currently on site includes:

- Mobile workshop
- Processing plant
- Site camp
- Admin offices
- Stage II offices
- Number of stockpiles
- TSF1 – used in Stage I HMS operation
- TSF2 – Stage II active tailings dam

Power

There is good access to the national hydroelectric power grid with two high tension power lines (120 and 220 kVA), operated by Société Nationale D'Eléctricite (SNEL, the DRC national supply authority), running adjacent to the Lubumbashi–Likasi national highway. A new 120 kVA power line has been built from Lubumbashi to Kinsevere, to connect the mine site to the 220 kVA national grid.

Whilst power outages are a common occurrence in Katanga Province, the incidence and duration of these appears to have increased in recent years. This is expected to be a function of increasing mining and industrial activity in the province, poor maintenance and an unreliable network, and SNEL over-commitments to customers.

The Stage II Project requires between 20 MW and 23 MW of power to allow for continuous operation. The overall site demand is between 23 MW and 25 MW. SNEL power outages as well as power restrictions have resulted in an average of only 10 MW being supplied to the site since November 2011.

Based on the high likelihood of a supply deficit and uncertainty about the longer term supply, a 16 MW power plant has been hired to supplement SNEL supplied power. It is expected that operating this power facility at 70% to 75% utilisation in addition to a 10 MW average from SNEL will provide sufficient power to achieve 60,000tpa cathode output. Output without the back-up power supply would be not more than 27,000tpa of cathode.

Basic details on the back-up power plant are as follows:

- size and specifications of units - 1.2 MW units for an online generation of 850 kW
- number of units - 21
- estimated diesel consumption - ~500,000L/week
- size of increased diesel storage facility ~ 600,000L

All 21 generators may be run for short periods of time when SNEL power is not available.

Operating at reduced capacity would not only result in direct revenue losses, but there could also be secondary losses due to anode and equipment damage arising from campaign operation of the plant and from constant power fluctuations. The cost of this additional power is in the order of $53 million per annum or $0.40/lb Cu.

Water

Process water is sourced from the TSF, internally circulated water, pit dewatering and make up water from the raw water dam. Process water is recycled back from the TSF and stored in the process water pond at the plant, which has a storage capacity of 60,000m^3 (100m x 100m x 6m). The raw water required by the plant as make up water is stored in a tank at the plant with an available capacity of 600m^3.

Raw water from boreholes is used for fire suppression water. A tank storage capacity of 600m^3 is maintained for emergency use at any time. An electric fire pump is used to supply water in the event of a fire. An additional diesel back-up fire pump is available in the event of a power outage.

There is also good water availability from the mine surrounds and/or the nearby Kifumashi River. Currently mine dewatering provides much of the make-up process water for Stage II.

Excess groundwater from the dewatering programme is discharged into the Kifumashi River via a discharge channel completed in late 2010 and located west of the Tshifufiamashi Pit.

Potable water is supplied by boreholes and an estimated 150 litres per person per day is used.

Communications

Mine site has a mobile and landline phone coverage that was established during Stage I construction. The internet connection to site is provided via satellite link.

Maintenance Workshops

Maintenance workshops are present on site for the mining contractor.

Airport

The main airport at Lubumbashi is used for access. Direct flights occur from Johannesburg (South Africa), Nairobi (Kenya), and the capital Kinshasa.

Road Access

The nearest major population centre is Lubumbashi, the capital of the Katanga Province, which is situated approximately 27km south of the property.

The Kinsevere property was previously accessed via a 22km, partially sealed road, which branches off the sealed Lubumbashi–Likasi national highway at the village of Kawama (11km northwest of the Lubumbashi International Airport turnoff). This road was refurbished as part of the Anvil Stage I Development Project. A new unsealed and wider access road has been constructed alongside the site power line route. This 24km length new road branches off the national highway at a point closer to the airport turnoff. Part of the road was sealed in 2013 with the additional work scheduled in 2013 and 2014.

The supply of most heavy operational spares and consumables originates from within southern Africa, although some equipment required for the Stage II plant construction came also from Australia, Asia and Europe.

Management of the Company's logistics function is coordinated from either site; an office in Lubumbashi or from a regional office in Johannesburg. Reputable trucking companies are used by the Company and transit times from Durban (a distance of 2,700km and from where equipment supplied from overseas is offloaded) is around 20 days, whilst equipment ex Johannesburg (a distance of 2,100km) is typically in transit for around 12 days.

Transport delays are often experienced at border crossings, most notably at Kasumbalesa on the DRC/Zambian border.

5.6.9 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 60. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 60 JORC Code Ore Reserves Assessment and Reporting Criteria for Kinsevere 2013 Ore Reserves

Assessment Criteria	Risk	Commentary
Mineral Resource estimate for conversion to Ore Reserves		The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The normal sub-celled Datamine Mineral Resource block model named "finmod_res_5x10x5_100.dm" dated 20-06-2013 was used for the optimisation purposes. Further details are discussed in the Mineral Resources Section of this report
Classification	Low	The Ore Reserves classification is based on the JORC requirements. The basis for the classification was the Mineral Resource classification and cut-off grade. Due to the uncertainty of the future site costs all material (Measured and Indicated) that is above 0.8% and below 1.6% ASCu was classified as Probable. The material that is classified as Measured and has a grade higher than 1.6% ASCu was classified as Proved.
Site visits	—	The Competent Person visited site on 19-26 September 2013 to inspect the site surface facilities and mining operations and liaise with site staff. He also visited site 25-31 October 2012.
Study status	Medium	The current mine and processing plant configuration have been in operation since September 2011. Ore Reserves inputs are based on actual historical performance data. Although full production ramp-up has been achieved for a number of months, further work is underway to ensure consistency of results and full realisation of the installed capacity.

Cut-off parameters	Medium	Estimated breakeven cut-off grade calculated as per historical practices is 0.85% Acid Soluble Copper at a US$2.8/lb copper price. This however is based on costs associated with full plating capacity of 60,000 tonnes per year. When treating the lower grade stockpiled ore this full plating capacity cannot be achieved without some kind of upgrading facility in place (currently the subject of further study in 2014). Thus the fixed time costs that are based on full plating capacity are under-estimating the tonnage related costs. To treat the low grade stockpiles therefore requires either a grade upgrading facility or a dramatic reduction in fixed costs. Examination of the fixed costs basis has determined that a large percentage of this required cost reduction can be achieved but is not by any means certain (relying heavily on dramatic reduction of expatriate labour force). Hence the low grade ore that has been or will be sent to the stockpiles have been downgraded to Probable Ore Reserves status.
Mining factors or assumptions	Medium	See Section 5.6.2 for details on geotechnical inputs. See Section 5.6.7 for details on dilution, loss and reconciliation.
Metallurgical factors or assumptions	Low	See Section 5.6.3 for details.
Environmental	Medium	ARD properties of the waste rock and black shales are unknown. Further work is planned to understand the properties of the rock and the required stockpile management policies. The property is not subject to any environmental liabilities. Following submission of the EIA (Consultants, July 2007), DRC Government approval of the Kinsevere Copper Project, Stages I and II, was issued by CAMI on 15th October 2007. Approval of a variance to the design and operation of the Stage II tailings storage facility was issued by the DPEM on 28th October 2008 (DPEM, Oct 2008). In relation to increased groundwater discharge arising from an expansion to the mine dewatering capacity, a Revision No. 1 to the 2007 EIA was prepared by Knight Piésold (Piésold, Dec 2010) and was submitted to the DPEM in December 2010, and finally approved in March 2011. An Environmental and Social Impact Assessment (ESIA) was prepared by KP (October, 2009) as a condition of the then proposed Project debt financing. Under debt financing circumstances, the lending institution must ensure that the Company complies with the internationally recognised Equator Principles (EP) and the International Finance Corporation (IFC) Principal Standards (PS). The ESIA document is intended to compliment the assessment information presented in the 2007 EIA. It does not overwrite any government approval or conditions of approval in the EIA of 2007 or any other regulatory requirements of the DRC Mining Code. To comply with the DRC Mining Regulations, it is necessary to manage surface water runoff in such a way that contaminated runoff is contained and sediment loadings (from disturbed catchments) are maintained at acceptable levels. In order to achieve this, a number of strategically placed Sediment Control Ponds (SCPs) and diversion channels will need to be implemented. As at October 2013 these changes have not been implemented, but there is work plan for it to be completed.
Infrastructure	High	See section 5.6.8 for details. The power situation rates this aspect as a high risk, with current mitigation by expensive on site diesel based power generation.
Costs	Medium	See Section 5.6.6 for details. Sustaining capital costs have been included in the pit optimisation. No further CAPEX was taken into account.
Revenue factors	Low	See Section 5.6.4 for details.

Market assessment	Low	There is off-take agreements with the trading company in place for all of the copper cathodes produced on site.
Economics	Medium	Costs detailed in this Appendix are based on historical actuals. Revenues are based on historical and contracted realisation costs and a realistic long-term metal price. The LOM financial model demonstrates the mine has a substantially positive NPV calculated at a discount rate of 8% and with respect to cash costs falls within the third quartile of copper producers. The value associated with the treatment of low grade stockpiles at the end of the project life is negative without substantial fixed costs reduction; however these do not substantially reduce the project NPV. No sensitivity analyses were undertaken for the Ore Reserves work.
Social	Low	Kinsevere site provides significant support to community with farming and other social projects financed by the site. It has strong support from the local community. Lubumbashi is a regional capital of the Katanga region. It has population of approximately 1.6M people (2012). Lubumbashi has a university that has some mining, geology and processing programs that prepare professionals. Personnel can be and are recruited from the local villages. The majority of these people are unskilled and require training. Skilled artisans and professional people can be and are recruited from Lubumbashi. Several hundred artisanal miners were previously active at Kinsevere before the Project commenced. Currently no artisanal miners are active in the area.
Audit or Reviews	Low	No external audits or reviews have been undertaken. This Ore Reserves statement is the first for MMG undertaken by MMG staff. The previous Ore Reserves statements were undertaken by Anvil Mining staff.
Discussion of relative accuracy/ confidence	—	The most significant factors affecting confidence in the Ore Reserves are: ▪ the ongoing issues of reliable power supply and the costs of that supply; ▪ the end of mine life cost reduction possible to enable economic treatment of the low grade stockpiles (however the value associated with these low grade Ore Reserves is minimal); ▪ the percentage of black shales that will be processable; ▪ estimates of gangue acid consumption that rely on calcium grade estimation; and ▪ volume variance effects of grade estimation highlighted by reconciliation work to date.
Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Low	Kinsevere is situated on the Central African Plateau at an elevation of 1,200 metres. The surrounding area gently slopes to the north towards the Kifumashi River, though more resistant parts of the Lower Roan stratigraphy form southeast-northwest trending, low, steep-sided ridges.
Climate	Low	Kinsevere has a distinct dry and wet season, with the wet season commencing in October and generally finishing by April. The average rainfall of the area is approximately 1,100mm, although this can range from 650mm to 1,500mm. Approximately 90% of the rainfall occurs during the wet season. Ongoing mining activities are not expected to be significantly affected during the wet season. Temperatures are generally mild and vary between 17°C and 26°C, with an average maximum around 28°C to 34°C in September and October, dropping by 2°C to 4°C with the onset of the rains. Winter maxima are around 21°C to 26°C but temperature can drop to as low as 5°C during the night in July and August.
Government Agreements	Low	The minerals rights to Kinsevere are held by Gécamines. MMG has a lease agreement with Gécamines to mine and process ore from Kinsevere until 2024, followed by an automatic fifteen year extension.
Hydrogeological Parameters	Low	Hydrogeological program is ongoing with Knight Piésold consultants managing the dewatering and water management programs.

Waste Storage (Including Tails Storage)	Low	The tailings storage facility was designed and planned by Knight Piésold (KP). KP has created a lift plan in stages that would last at least for a year at a 1.6Mtpa production rate. Next lift is expected to start in 2015 with subsequent lifts planned one each year.

6. GOLDEN GROVE UNDERGROUND OPERATIONS

6.1 Introduction and Setting

The Golden Grove mining operations is 100% owned and operated by MMG Limited through its Australian subsidiary Golden Grove Pty Ltd. The operations are located within the Yalgoo Local Government Area (Shire of Yalgoo) in the Mid-West Region of Western Australia.

Golden Grove is approximately 56km south of the township of Yalgoo, 375km north-northeast of Perth and 225km due east of the coastal port town of Geraldton. Access to site is via sealed roads from Perth to Paynes Find and from Geraldton to Yalgoo.

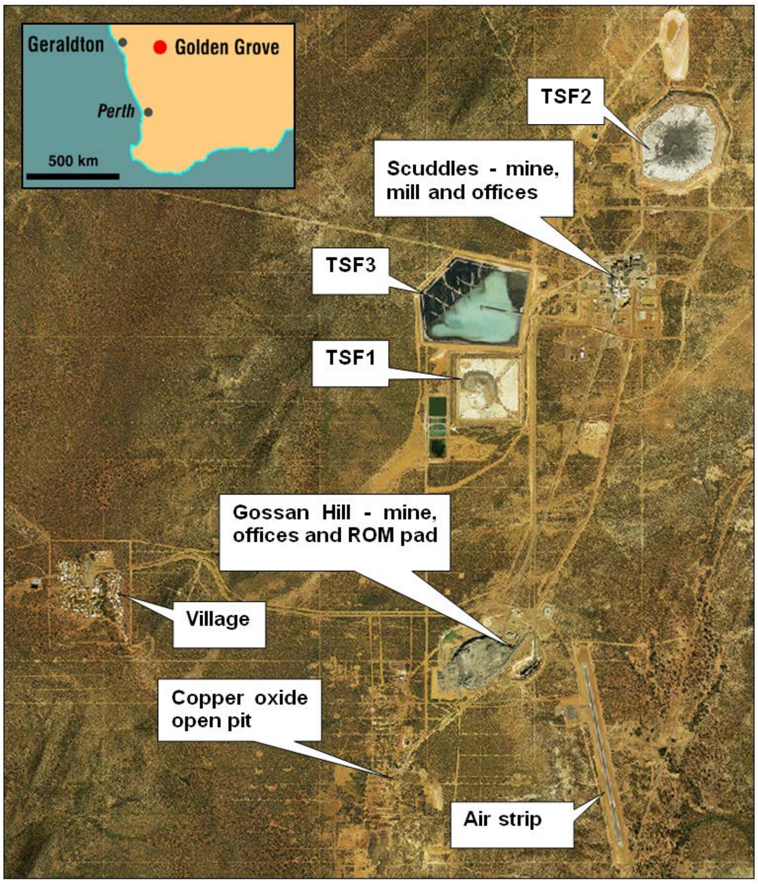
The Golden Grove operation comprises underground and surface operations at Gossan Hill and Scuddles, located 4km apart. Volcanogenic Hosted Massive Sulphide (VHMS) mineralisation was discovered at Gossan Hill in 1971 and at Scuddles in 1979. Scuddles underground operations began in 1990 and Gossan Hill underground operations started producing in 1998. Copper oxide ore is mined from an open pit at Gossan Hill that started in early 2012.

Figure 50 **Golden Grove Mine location**

The underground mines are operated by MMG employees and the open pit is operated by a mining contractor. Gossan Hill ore is trucked to surface and crushed at the Gossan Hill ROM pad, before being transported approximately 3km overland by conveyor to the treatment plant at Scuddles (refer Figure 51). Scuddles ore undergoes primary crushing underground before being hoisted to surface.

The treatment plant consists of a two-stage semi-autogenous grinding circuit followed by flotation using air agitation to recover the valuable minerals. Each ore type is treated separately to produce either zinc concentrate, heavy precious metals (HPM) concentrate, copper sulphide concentrate or copper oxide concentrate. These concentrates are transported by road-train to the storage and loading facility at Geraldton for shipment to smelters in Asia and Europe.

Figure 51 Aerial view of Golden Grove Operations showing location of key surface infrastructure



6.2 Geological Setting

The stratabound copper and zinc mineralisation is predominantly hosted in Golden Grove Member 6 (GG6) of the Golden Grove Formation. The mineralisation is massive to stringer style high-iron sphalerite and pyrite, with minor breccia mineralisation, continuous along strike and down dip. A second stratabound zinc mineralisation system is also found within the SC3 (Scuddles Formation Member 3) unit. This zinc mineralisation consists primarily of low-iron sphalerite and galena with high levels of gold and silver generally associated with the pyrite and galena mineralisation. These SC3 zinc lenses are approximately 1m to 7m thick and continuous along strike and down-dip.

Gossan Hill (Figure 52) and Gossan Valley (Figure 54) also contain copper mineralisation associated with magnetite in the GG4 stratigraphic unit. Gossan Valley also contains zinc mineralisation associated with the GG4 stratigraphic unit, which is not seen at either Gossan Hill or Scuddles.

Chalcopyrite/pyrite mineralisation has also been intersected in the footwall of the GG4 unit, often near its contact with either the GG2 or GG1 units. This mineralisation trends from stringer and breccia-style to sub-massive/massive.

The sequence is intruded by dolerite dykes, sub-vertical rhyolite dykes and a voluminous dacite body with associated feeder dykes which stope out portions of mineralisation.

At Gossan Hill, weathering and oxidation extend down approximately eighty metres from surface. Oxide/supergene copper and oxide gold Mineral Resources are located directly above the primary copper and zinc mineralisation respectively.

The main primary copper zone at Gossan Hill extends 700m along strike, 450m down-dip and is 80m wide. This mineralisation is hosted within the GG4 unit. Primary mineralisation occurs as chalcopyrite with various gangues. The system is broadly differentiated into a footwall massive sulphide zone, grading into a magnetite - sulphide zone, with stringer-style mineralisation throughout. Copper grades transgress sulphide/magnetite lens boundaries. A massive magnetite zone occurs in the hanging wall to the mineralisation.

Significant copper also occurs in the footwall of the GG6 unit zinc mineralisation at both Gossan Hill and Scuddles. This mineralisation comprises stringer to sub-massive chalcopyrite and pyrite with or without magnetite.

The main zinc mineralisation at Gossan Hill and Scuddles (Figure 53) is stratabound within the GG6 unit. The main sulphide types are pyrite, sphalerite, chalcopyrite, pyrrhotite and galena.

Figure 52 Long-section of mineralised zones at Gossan Hill, view is facing local grid west, grid squares are 500m

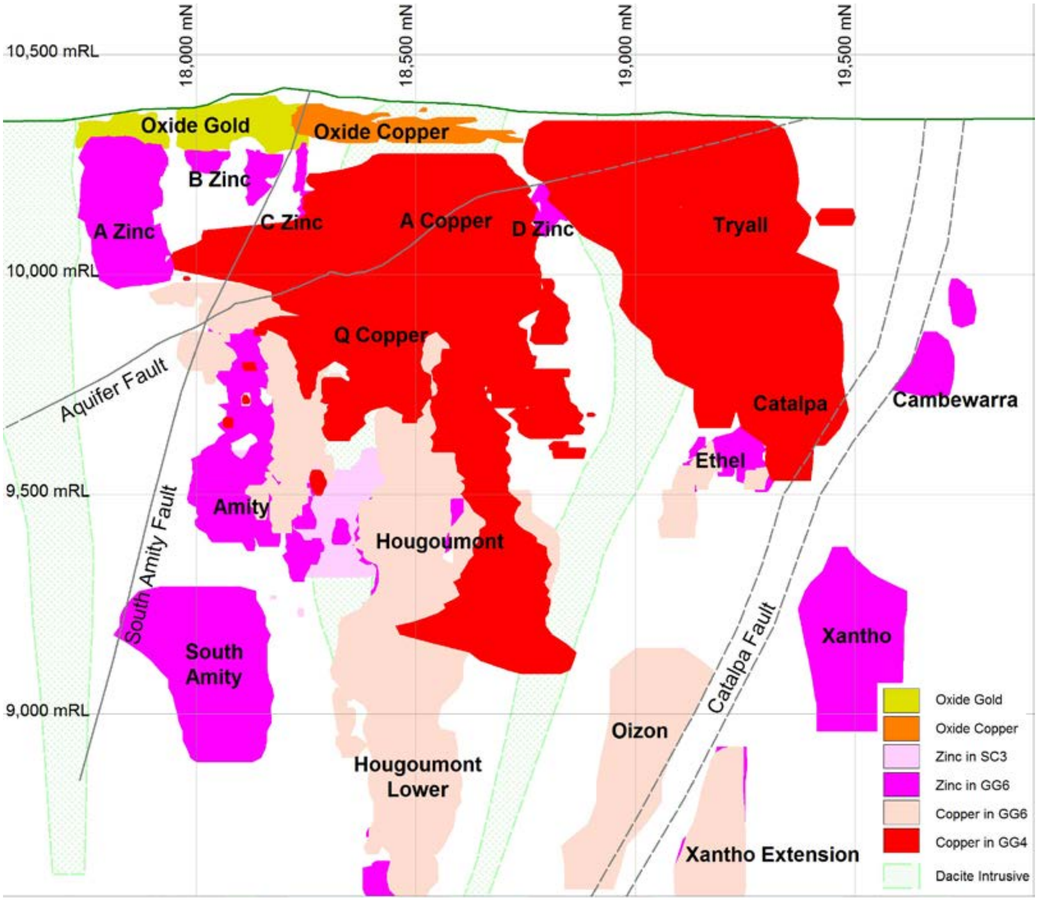


Figure 53 Long-section of mineralised zones at Scuddles, view is facing local grid west, grid squares are 500m

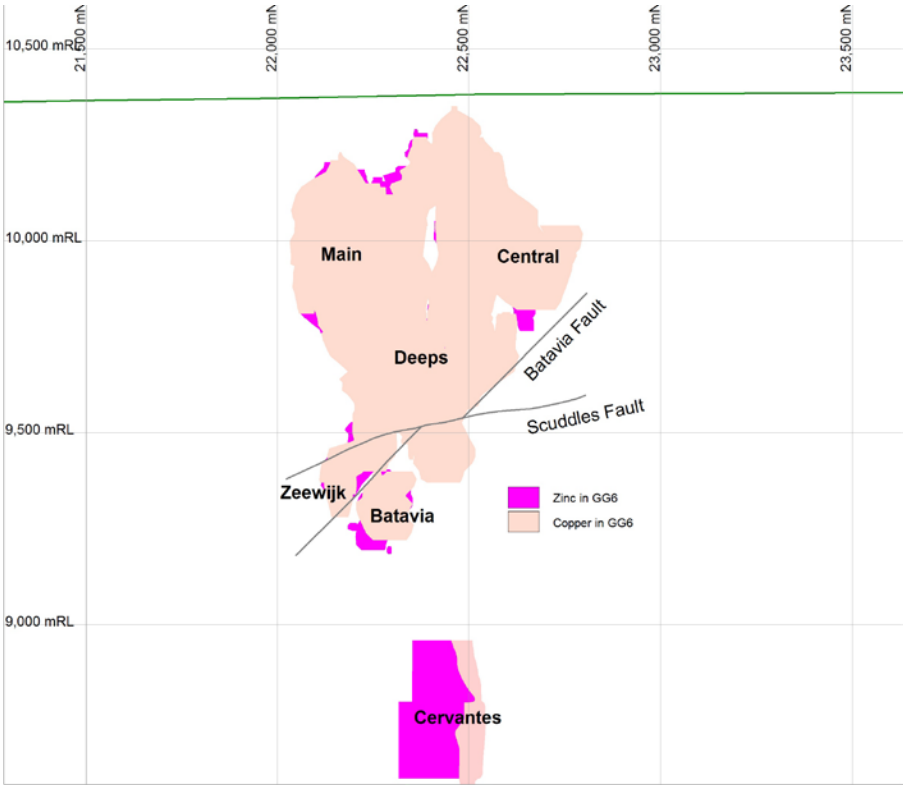
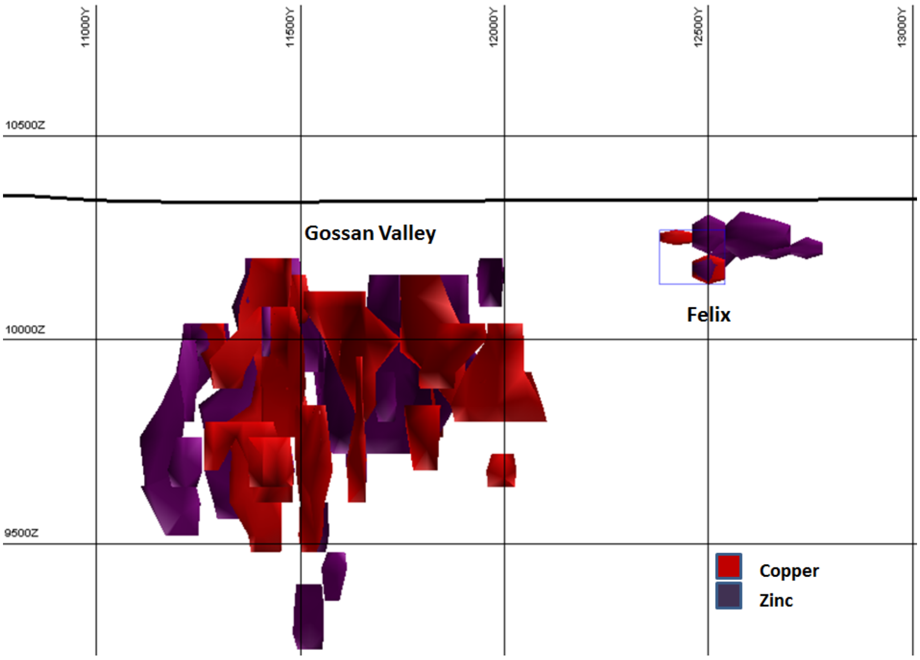


Figure 54 Long-section of mineralised zones at Gossan Valley / Felix, view is facing local grid west, grid squares are 500m.



6.3 Mineral Resources – Golden Grove Underground

6.3.1 Results

The Golden Grove primary sulphide July 2013 Mineral Resource Statement incorporates the primary zinc and primary copper Mineral Resources within the Gossan Hill, Scuddles and Gossan Valley deposits at the Golden Grove Mine Site. The Golden Grove oxide copper and oxide gold Mineral Resources are discussed in Section 7.2.

The Golden Grove Mineral Resource estimate as at June 30 2013 is summarised in Table 61 and Table 62.

Table 61 Golden Grove copper Mineral Resource as at June 30 2013

Golden Grove Primary Copper Mineral Resources (Inclusive of Ore Reserves)

Mine	Resource Category	Tonnes (Mt)	Cu %	Pb %	Zn %	Ag g/t	Au g/t	SG	NSR_LT $	Cu Metal (kt)
Gossan Hill Underground[1]	Measured	3.3	2.8	0.05	0.4	19	0.6	3.5	158	93
	Indicated	1.8	2.8	0.33	2.8	41	2.1	3.5	214	52
	Inferred	7.8	3.2	0.04	0.4	26	0.3	3.6	178	249
	TOTAL	**12.9**	**3.0**	**0.09**	**0.7**	**26**	**0.6**	**3.5**	**178**	**394**
Scuddles[1]	Measured	2.6	2.8	0.03	0.4	14	0.4	3.5	157	73
	Indicated	0.9	2.9	0.02	0.2	11	0.3	3.4	153	26
	Inferred	0.7	2.5	0.01	0.1	15	0.2	3.7	135	19
	TOTAL	**4.2**	**2.8**	**0.03**	**0.3**	**14**	**0.4**	**3.5**	**152**	**117**
Gossan Valley[1,2]	Measured	-	-	-	-	-	-	-	-	-
	Indicated	-	-	-	-	-	-	-	-	-
	Inferred	1.0	2.8	0.01	0.1	22	0.5	3.2	162	29
	TOTAL	**1.0**	**2.8**	**0.01**	**0.1**	**22**	**0.5**	**3.2**	**162**	**29**
Total Copper Resource		**18.1**	**3**	**0.07**	**0.6**	**23**	**0.5**	**3.5**	**171**	**352**

[1] nsr_lt>$95 (Net Smelter Return)

[2] Inc. Felix Orebody

Table 62 Golden Grove zinc Resource as at June 30 2013

Golden Grove Primary Zinc Mineral Resources (Inclusive of Ore Reserves)

Mine	Resource Category	Tonnes (Mt)	Cu %	Pb %	Zn %	Ag g/t	Au g/t	SG	NSR_LT $	Zn Metal (kt)
Gossan Hill Underground[1]	Measured	0.6	0.4	1.3	12.4	75	1.4	3.4	244	68
	Indicated	0.9	0.4	1.9	16.0	130	2.2	3.4	343	144
	Inferred	2.4	0.6	0.9	11.2	67	0.7	3.5	208	271
	TOTAL	**3.9**	**0.5**	**1.2**	**12.5**	**83**	**1.1**	**3.4**	**245**	**483**
Scuddles[1]	Measured	0.4	0.3	1.2	13.3	95	1	3.8	241	56
	Indicated	0.1	0.2	0.9	10.6	72	1	3.8	184	5
	Inferred	0.8	0.7	0.9	14.0	75	1	3.8	239	107
	TOTAL	**1.2**	**0.5**	**1**	**13.7**	**82**	**1**	**3.8**	**237**	**6**
Gossan Valley[1,2]	Measured	-	-	-	-	-	-	-	-	-
	Indicated	-	-	-	-	-	-	-	-	-
	Inferred	1.5	0.1	0.2	12.4	9	0.4	3.3	172	183
	TOTAL	**1.5**	**0.1**	**0.2**	**12.4**	**9**	**0.4**	**3.3**	**172**	**183**
Total Zinc Resource		**6.6**	**0.4**	**0.9**	**12.7**	**66**	**0.9**	**3.5**	**227**	

[1] nsr_lt>$95 (Net Smelter Return)

[2] Inc. Felix Orebody

All Mineral Resources quoted in this report were estimated from three dimensional block models created with Vulcan software. Mineral Resources are modelled using solid wireframes of geological boundaries guided by a 4% Zn cut-off and a 1% Cu cut-off. These cut-offs approximate the natural break between zinc or copper mineralisation and the background grades.

Individual block models are created for each mineralisation region at Golden Grove.

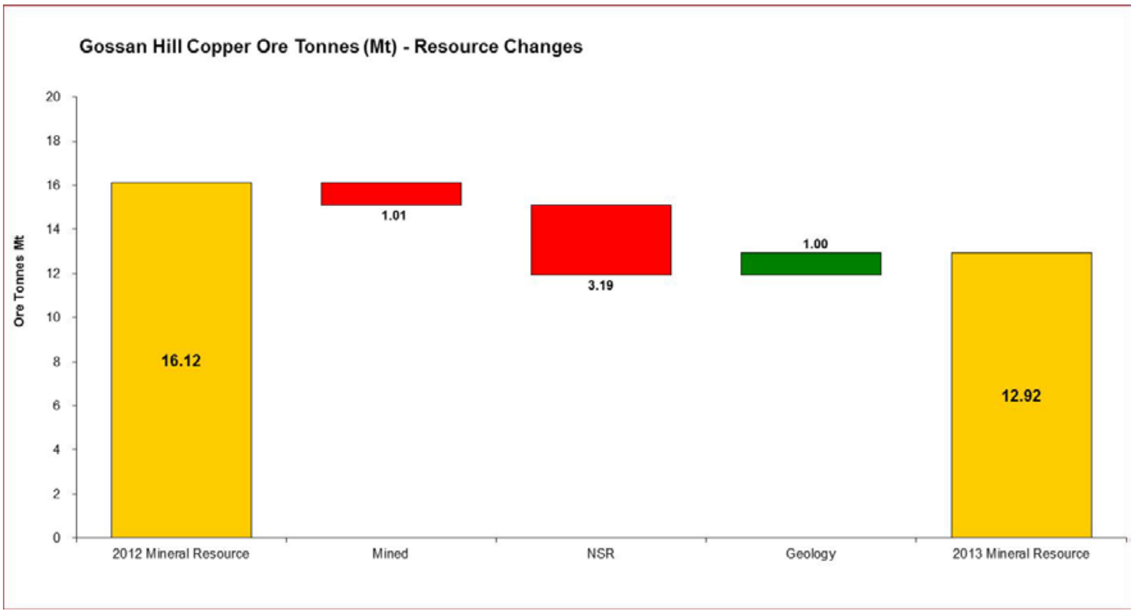
The reporting cut-off used for the primary zinc and copper Mineral Resource estimate approximates the site mining and processing break-even costs, taking into account metallurgical recovery, concentrate transport costs, concentrate treatment and refinery charges and royalties. Expressed as Net Smelter Return (NSR) or mine gate value, the cut-off NSR used for the Mineral Resource estimate is A$$95.00/t.

An average approximation of the NSR cut-off to grade can be expressed for mineralisation not containing precious metals and for mineralisation with precious metals is as follows:

- Copper or zinc mineralisation. Cu: 1.95%, Zn: 7.0%
- Zinc mineralisation with precious metals. Zn: 4.0%, Pb: 1.0%, Ag 50 g/t, Au 1.8 g/t.
-

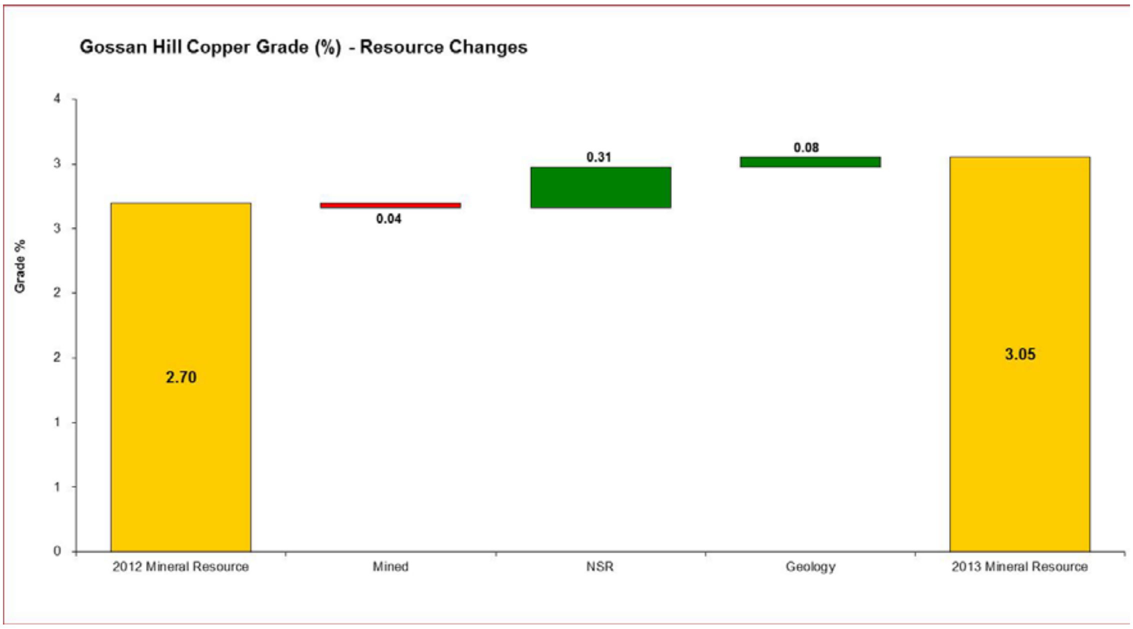
Previous Mineral Resources estimations are used in the validation of the 2013 estimation and are compared in waterfall charts. Waterfall charts for Gossan Hill copper and zinc are shown in Figure 55 to Figure 60.

Figure 55 Gossan Hill copper ore tonnes waterfall chart



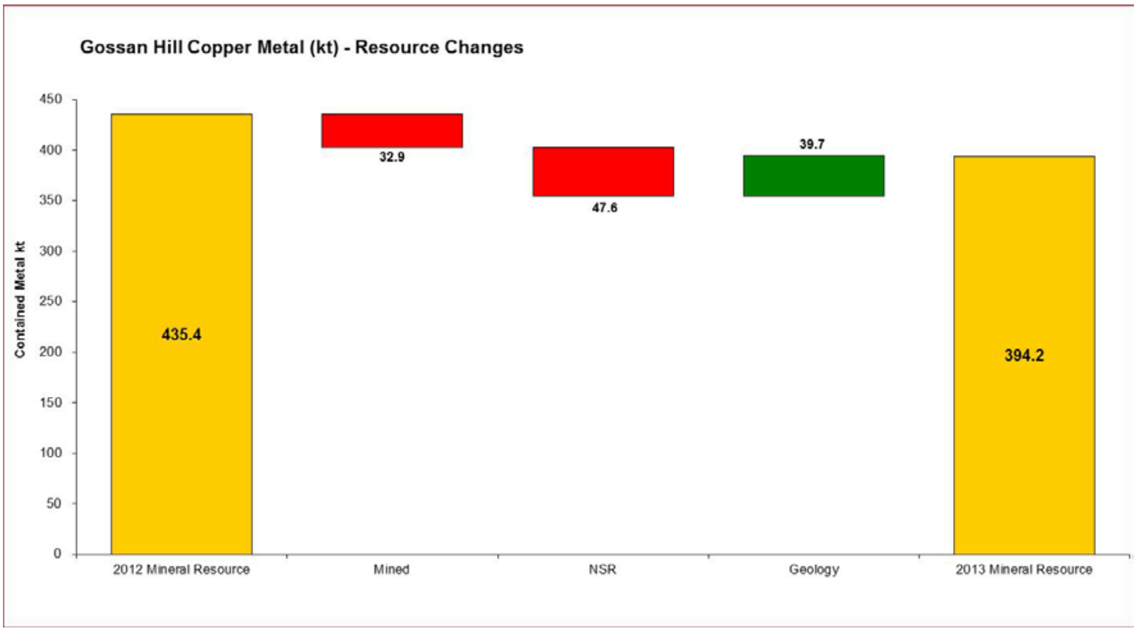
Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

Figure 56 Gossan Hill copper grade waterfall chart



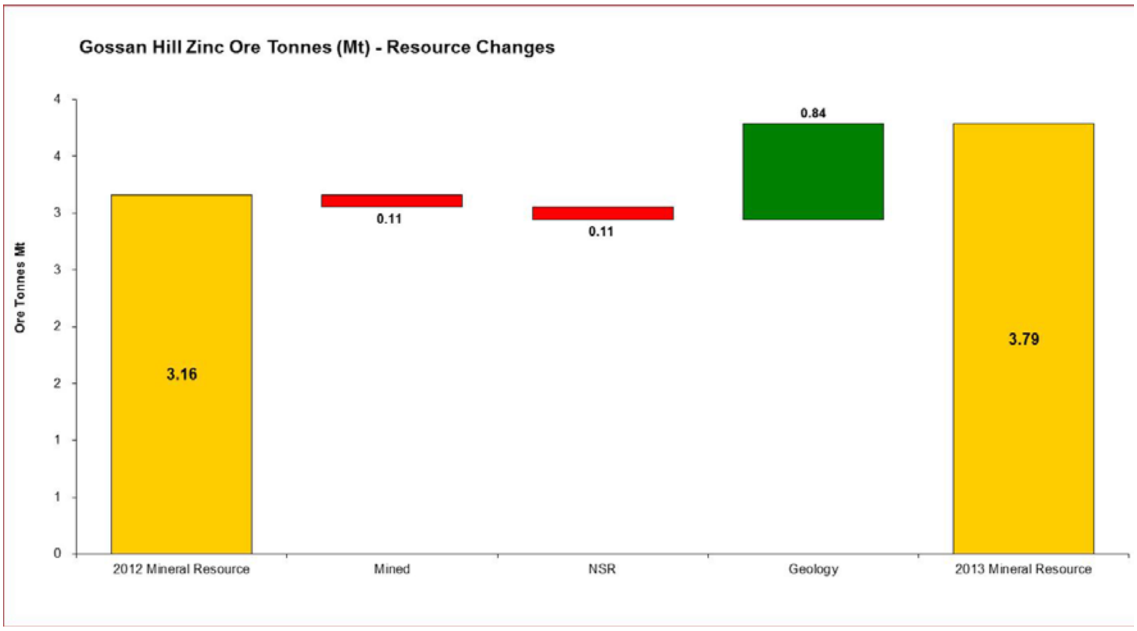
Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

Figure 57 Gossan Hill Copper contained metal waterfall chart



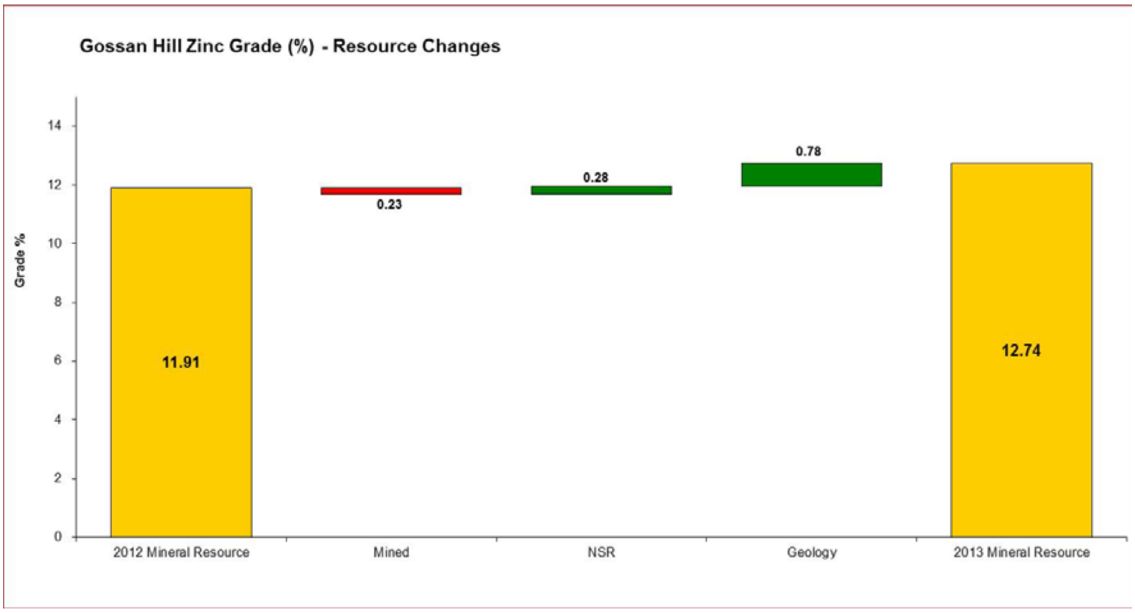
Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

Figure 58 Gossan Hill zinc ore tonnes waterfall chart



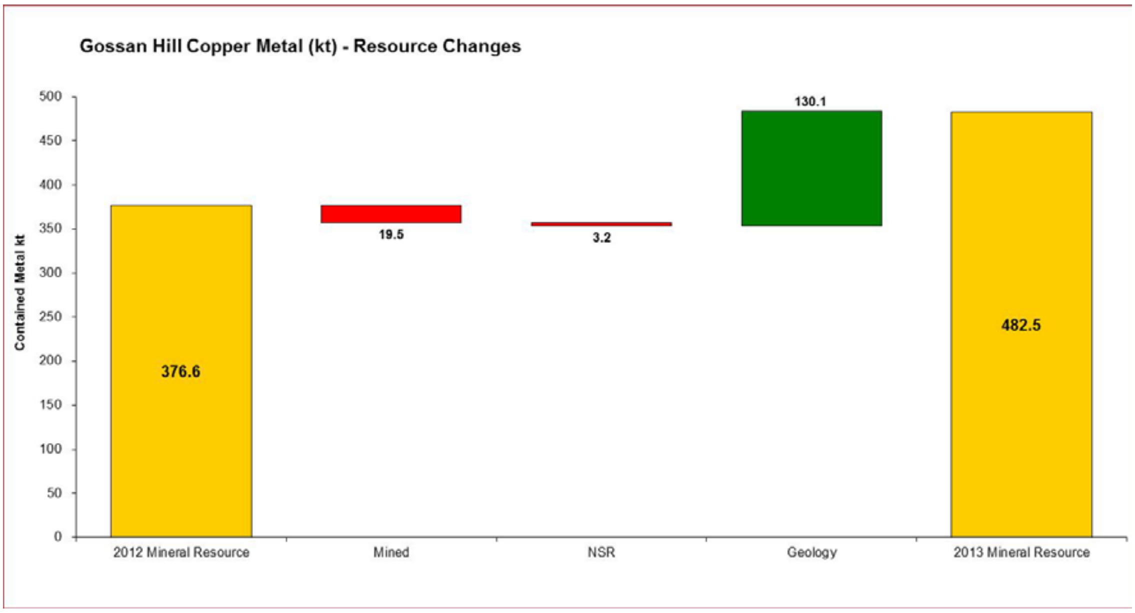
Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

Figure 59 Gossan Hill zinc grade waterfall chart



Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

Figure 60 Gossan Hill copper contained metal waterfall chart



Note: Only Gossan Hill tonnes are shown, Scuddles and Gossan Valley are not included.

6.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

We, Timothy Goodale and Lauren Stienstra, confirm that we are the Competent Persons for the Golden Grove underground operations Mineral Resources section of this Report and:

- We have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- We are Competent Persons as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I, Lauren Stienstra, is a Member of The Australasian Institute of Geoscientists
- I, Timothy Goodale, is a Member of The Australasian Institute of Mining and Metallurgy
- We have reviewed the relevant Golden Grove underground operations Mineral Resources section of this Report to which this Consent Statement applies.

We are full time employees of MMG Limited (at the time of estimation).

We have disclosed to the reporting company the full nature of the relationship between ourselves and the company, including any issue that could be perceived by investors as a conflict of interest.

We verify that the Golden Grove underground operations Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which we are responsible – the Golden Grove underground operations Mineral Resources - we consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Lauren Stienstra – 26/11/13

Stefan Gawlinski (Witness)

Tim Goodale – 26/11/13

Stefan Gawlinski (Witness)

6.4 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Golden Grove open pit Mineral Resources.

Table 63 Checklist of assessment and reporting criteria for Golden Grove underground Mineral Resource

<table>
<tr><th>Criteria</th><th>Status</th></tr>
<tr><td colspan="2">Section 1 Sampling Techniques and Data</td></tr>
<tr><td>Sampling techniques</td><td>Diamond drilling was used to obtain nominal 1m length half core samples which were submitted for analysis.
The samples range from 0.5m to 1.2m so as not to sample across lithological contacts.</td></tr>
<tr><td>Drilling techniques</td><td>Only diamond drill core and minor reverse circulation data was used in the Resource estimations for Gossan Hill and Scuddles. The total number of drillholes used are listed below:
- 6,388 drillholes were used in the Gossan Hill Resource model
- 3,026 drillholes were used in the Scuddles Resource model
- 361 drillholes were used in the Gossan Valley Resource model
The breakdown of Gossan Hill and Scuddles drilling by year and company is shown in Table 64 and Table 65.
Table 64 Breakdown of Gossan Hill drilling by year and company
<table>
<tr><th>Company</th><th>Years</th><th>BQ</th><th>HQ</th><th>LK48</th><th>LTK60</th><th>NQ</th><th>NX</th><th>PCD</th><th>PQ</th><th>UNK</th><th>NAVI</th><th>B</th><th>Total</th></tr>
<tr><td>Aztec, Amax, Esso and Production</td><td>1971-1978</td><td></td><td>2,626</td><td></td><td></td><td>6,436</td><td></td><td>80</td><td>1,413</td><td>2,591</td><td></td><td></td><td>13,147</td></tr>
<tr><td>Australian Consolidated Minerals</td><td>1979-1981</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>2,368</td><td></td><td></td><td>2,368</td></tr>
<tr><td>Normandy</td><td>1992-2001</td><td>3,245</td><td>4,119</td><td>58,745</td><td>13,339</td><td>26,616</td><td></td><td></td><td>718</td><td>43,277</td><td></td><td></td><td>150,058</td></tr>
<tr><td>Newmont</td><td>2002-2004</td><td>13,461</td><td>6,554</td><td>72,325</td><td>17,217</td><td>38,915</td><td></td><td></td><td>2,919</td><td>3,261</td><td></td><td></td><td>154,651</td></tr>
<tr><td>Oxiana</td><td>2005-2007</td><td>12,646</td><td>2,327</td><td>126,911</td><td>12,249</td><td>28,630</td><td></td><td></td><td>729</td><td></td><td>43</td><td></td><td>183,533</td></tr>
<tr><td>MMG</td><td>2009-2013</td><td>6,257</td><td>5,035</td><td></td><td>60,256</td><td>77,137</td><td>386</td><td></td><td>2,008</td><td>4,413</td><td>1,298</td><td>981</td><td>157,770</td></tr>
<tr><td>OZ Minerals</td><td>2008</td><td>1,294</td><td>4,177</td><td></td><td>21,665</td><td>18,105</td><td></td><td></td><td>1,123</td><td>19,163</td><td>27</td><td></td><td>65,554</td></tr>
<tr><td>Unknown</td><td>UNK</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>48,026</td><td></td><td></td><td>48,026</td></tr>
<tr><td>Total</td><td></td><td>36,904</td><td>24,838</td><td>257,980</td><td>124,725</td><td>195,838</td><td>386</td><td>80</td><td>8,909</td><td>123,099</td><td>1,367</td><td>981</td><td>775,108</td></tr>
</table>
Note: UNK = Unknown
Table 65 Breakdown of Scuddles drilling by year and company
<table>
<tr><th>Company</th><th>Years</th><th>BQ</th><th>BY</th><th>HQ</th><th>LK48</th><th>LTK60</th><th>NQ</th><th>NX</th><th>PCD</th><th>PQ</th><th>UNK</th><th>Total</th></tr>
<tr><td>Aztec, Amax, Esso and Production</td><td>1971-1978</td><td>972</td><td></td><td>13,260</td><td>34,475</td><td>8,569</td><td>28,373</td><td></td><td>165</td><td>2,379</td><td>2,664</td><td>90,856</td></tr>
<tr><td>Australian Consolidated Minerals</td><td>1979-1991</td><td></td><td>129</td><td></td><td></td><td></td><td>39</td><td></td><td></td><td></td><td>69,162</td><td>69,330</td></tr>
<tr><td>Normandy</td><td>1992-2001</td><td>175</td><td></td><td>1,910</td><td>25,120</td><td>688</td><td>11,587</td><td></td><td></td><td>301</td><td>123,395</td><td>163,176</td></tr>
<tr><td>Newmont</td><td>2002-2004</td><td>449</td><td></td><td>3,740</td><td>31</td><td></td><td>11,895</td><td></td><td></td><td>1,238</td><td></td><td>17,352</td></tr>
<tr><td>Oxiana</td><td>2007</td><td></td><td></td><td>247</td><td></td><td></td><td>1,920</td><td></td><td></td><td></td><td></td><td>2,167</td></tr>
<tr><td>OZ Minerals</td><td>2008</td><td></td><td></td><td></td><td></td><td>1,807</td><td>5,068</td><td></td><td></td><td></td><td></td><td>6,875</td></tr>
<tr><td>MMG</td><td>2010-2013</td><td>278</td><td></td><td>3,252</td><td></td><td>34,028</td><td>19,458</td><td>37</td><td></td><td>2,379</td><td>754</td><td>60,185</td></tr>
<tr><td>UNK</td><td>UNK</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>4,165</td><td>4,165</td></tr>
<tr><td>Total</td><td></td><td>1,873</td><td>129</td><td>22,409</td><td>59,626</td><td>45,091</td><td>78,339</td><td>37</td><td>165</td><td>6,296</td><td>200,140</td><td>414,105</td></tr>
</table>
</td></tr>
<tr><td>Drill sample recovery</td><td>Core recovery was greater than 99.5%.</td></tr>
<tr><td>Logging</td><td>All drill core/chips are geologically logged using codes set up for direct computer input into the Micromine Geobank™ database software package.
All diamond drillholes since 1999 have been photographed.
Diamond drillhole core is stored within the core yard facility.
Assay pulps are stored from delineation and exploration programs in the core yard facility.</td></tr>
<tr><td>Sub-sampling techniques and sample preparation</td><td>All sampled intervals in LTK60, BQTK and NQ diamond drillholes are ½ core sampled.
Core is orientated along the apical trace of the reference plane ellipse (typically bedding) and then half cored using a diamond core saw to ensure the sample is a true representative of the <i>in situ</i> mineralisation.
The optimal sample interval is 1m. Sample sizes can range from 0.5m to 1.2m.
Sample intervals do not cross lithological boundaries.
Samples undergo total pulverisation before being analysed for a basic suite of seven elements (Zn, Cu, Pb, Fe, S, Ag and Au).
The methodology for Zn, Cu, Pb, Fe, S, and Ag assay involves digesting a 0.25g pulverised sample in a 4-acid digest then analysed using ICP.
For gold, a nominal 30g sample is mixed with fluxing agents and fused. The gold is dissolved in aqua regia and the solution analysed by Atomic Absorption Spectroscopy (AAS).</td></tr>
</table>

Quality of assay data and laboratory tests	A certified matrix-matched MMG standard, of suitable grade, is inserted every 25th sample and as the last sample of the drillhole. Results for internal MMG standards indicate no significant bias. All internal MMG standards used at Golden Grove (Zn: GH1-GH5 and Cu: GHA-GHD) are certified for Cu, Zn, Au, Ag, Fe, Pb and S. From March 2011, every 50th sample is quarter core cut and submitted as a field duplicate. Results to date, based on 30% absolute difference, have performed poorly and are a topic for further investigation. A coarse blank sample is inserted every 50 samples. Results indicate acceptable laboratory performance.
Verification of sampling and assaying	Two laboratory audits have been carried out in the past 12 months, with no issues. No umpire laboratory was used during the reporting period.
Location of data points	All of the diamond drillhole collar locations and orientations are surveyed using an electronic theodolite and recorded in Geobank database. Down-hole surveying is performed using a Gyro tool for all underground holes.
Data spacing and distribution	Drillhole data spacing ranges from less than 10m x 10m in the active mining areas through to greater than 80m x 80m in other areas of the Mineral Resource. Drive mapping and surveyed photography are performed on all geologically important headings and drives. This provides a platform by which three dimensional spatially related digital maps are created to represent the deposits geology. This digitised mapping is used to guide the construction of triangulations used in the estimate.
Orientation of data in relation to geological structure	Drilling is conducted in east-west and west-east directions to correctly intercept the predominately north-south striking Golden Grove mineralisation.
Sample security	Measures to provide sample security included: Adequately trained and supervised sampling personnel. Half-cored samples are placed in numbered and tied calico sample bags. Bag and sample numbers are entered into the Micromine database. Samples are couriered to assay laboratory via truck in plastic bulker containers. Assay laboratory checks of sample dispatch numbers against submission documents.
Audits or Reviews	Regular auditing of external lab has been performed in the last 12 months. Regular laboratory audits have been completed by the Geological Database Administrator with support from Resource, Senior Mine, and Mine Geologists. No major areas of concern have been raised. The most recent laboratory audit was conducted in April, 2013.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	The mineral tenement and land tenure status of the Golden Grove operations are listed in Table 66.

Table 66 Mineral tenement and land tenure status for Golden Grove operations

Tenement No.	Prospect Name	Date Expires	Term Years	Date Granted
M59/03	Scuddles	08/12/2025	21	28/01/2005*
M59/88	Chellews	18/05/2030	21	20/04/2009*
M59/89	Coorinja	18/05/2030	21	20/04/2009*
M59/90	Cattle Well	18/05/2030	21	20/04/2009*
M59/91	Cullens	18/05/2030	21	20/04/2009*
M59/92	Felix	18/05/2030	21	20/04/2009*
M59/93	Flying Hi	18/05/2030	21	20/04/2009*
M59/94	Bassendean	18/05/2030	21	20/04/2009*
M59/95	Thundelarra	18/05/2030	21	20/04/2009*
M59/143	Bassendean	09/05/2031	21	21/04/2009*
M59/195	Gossan Hill	17/05/2032	21	17/06/2011*
M59/227	Crescent	07/05/2012	21	08/05/1990
M59/361	Badja	01/03/2016	21	02/03/1995
M59/362	Badja	01/03/2016	21	02/03/1995
M59/363	Badja	01/03/2016	21	02/03/1995
M59/543	Walgardy	04/02/2023	21	05/02/2002
M59/480	Marloo	01/07/2029	21	02/07/2008

* Renewal date

Exploration done by other parties	Original definition and exploration drilling was performed by Joshua Pitt, of Aztec Exploration, in 1971. From 1971 until 1992 multiple joint ventures and funding continued definition of the Mineral Resource, with highlights being the Scuddles, A Panel Zn, B Panel Zn, C Panel Zn and Cu discoveries. Parties involved include Amax Exploration, Esso Exploration, Australian Consolidated Minerals and Exxon. Newmont, Normandy, Oxiana, OZ Minerals and MMG have all been involved with the drilling and exploration of the Golden Grove leases since 1991. A table showing the companies, years, core size and meterage is shown in Table 64 and Table 65.
Geology	Host rock is a volcanogenic layered sediment system. The mineralisation style is that of a Sulphide VHMS system. The mineralisation is hosted within a series of sub-vertical lenses.
Drillhole information	The total drillhole registry for Golden Grove consists of over 22,000 drillholes comprising a range of different drilling types and core thicknesses. A series of tables containing company, year, drillhole diameter and meterage for holes included in the Mineral Resource estimation can be viewed in Section 1 of this table.

Data aggregation methods	No results of aggregated exploration data has been reported exclusively. Exploration drilling results for all orebodies are represented in block model tonnes and grade format only.
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	All data is stored in the Micromine database. All data is directly input into Micromine database using key field and data validation processes. Collar co-ordinates and drilling direction (azimuth and dip) are validated via comparison of planned data to surveyed data. Deviations of more than 1 degree over 30m of drillhole depth are flagged and evaluated for redrilling. The surveyed data is provided by direct measurement of each drillhole, at the commencement of each hole, using an electronic theodolite tool. Down-hole survey measurements are duplicated for each drillhole and results are compared. All data attributed to a given drillhole undergoes final validation and sign-off procedure.
Site Visits	Both Competent Persons work on-site at Golden Grove full-time on 8 days on, 6 days off roster.
Geological interpretation	Geological triangulations and grade shell triangulations are created by the Mine Geologists and Resource Geologists. Geological triangulation interpretations were formed from polygons snapped to drillholes. Primary zinc mineralisation triangulations were based on a high grade cut-off of ≥4% zinc and a low grade cut-off of ≥0.5% zinc. Primary copper mineralisation triangulations were based on a high grade cut-off of ≥1% copper and a low grade cut-off of ≥0.2% copper. Grades below these cut-offs were included in areas to honour the geology.
Dimensions	The main zinc mineralisation at Gossan Hill and Scuddles is divided into several zones, with each zone varying from 200m to 400m along strike, 200m to 700m down-dip and 3m to 20m in thickness.
Estimation and modelling techniques	Golden Grove Mineral Resources (Cu, Zn, Pb, Ag, Au and Fe) are estimated using Ordinary Kriging and Inverse Distance techniques as appropriate. Density was estimated using Ordinary Kriging (or inverse distance square from some domains), from data derived from bulk density measurements (refer to the Bulk Density Section of this table for details). Statistical analysis is performed using Snowden's Supervisor, whilst geological interpretation and block model estimation is performed in Vulcan. Net Smelter Return (NSR) block coding (discussed below); assumptions are made about the recovery of precious metals. These are discussed below in the metallurgical factors section. Iron is estimated in the block models, and although not essentially deleterious, it does impact the recovery of payable elements. This is discussed in the metallurgical factors section below. The block model extents were set up to capture all mineralisation. Initial parent cell, 20m x 50m x 50m with sub-cell, 1m x 2.5m x 2.5m, evaluation used. In areas of higher drill density, parent cells are limited to 10m x 25m x 25m or 5m x 5m x 5m depending on support available. In general a parent cell size equivalent to half the average drill spacing is used. Selective mining units approximate 1m (X), 4m (Y) and 4m (Z). These assumptions are reflected in the sub celling parameters of 1m x 2.5m x 2.5m, with consideration also given to geological resolution requirements. No direct correlation between elements is used in the estimation process. Geological interpretation is used to domain and code areas of significance. This is achieved by creating triangulations that represent the geology. These triangulations are used to subset estimation domains. Sample values of -99 were not ignored in mineralisation, instead all assay values ≤0 were over written with the nominal waste grade using the following assignments. Cu, Zn and Pb = 0.001%, Au = 0.001 g/t, Fe = 0.01% and Ag = 1g/t. This was done to avoid smearing of high grades into internal non-assayed waste zones. In some cases the decision was made to grade cap values for a given domain. The decision to grade cap was based on populations displaying coefficient of variance greater than 1, in conjunction with a clear disintegration of the population distribution as displayed in histogram format. Grade cap values, were based on the disintegration point of the population and were chosen with careful consideration to the per cent of the population effected. Variography was reviewed and updated for new interpretations and for existing domains materially affected by new drill data. Discretisation was set to 4 X 4 X 4. Generally the minimum number of samples per estimate was set to 10, with a maximum of 56. Searches employed are generally set at major 40m, semi-major 30m, minor 10m for the first pass, major 80m, semi-major 60m, minor 20m for the second and third passes, and major 160m, semi-major 120m, minor 40m for the fourth pass. This allows initial searches that are within and supported by variogram ranges and also well supported by data density. Block models were visually checked in Vulcan. Visual checks found that blocks adequately represented the drill grade data and that geological knowledge and continuity was captured within the models. Block and sample statistics were compared for all domains. Block statistics generally displayed a slightly lower mean and always showed reduced variance, in comparison to composite drill data.

<table>
<tr><td></td><td>Mining voids were used in the initial volume model creation:
- Mine void triangulations were used to sub cell the volume model and set the mined variable to 1. This was done using all voids up to 31st December 2012.
- Mine void triangulations, from the 1st January 2013 to 30th June 2013 were used to flag model blocks as mined =2. This occurred post processing to aid in the timely delivery of block models.
- All mined stopes at Gossan Hill are expanded 3m east and west to capture all unrecoverable mineralisation. Expanded stopes are given the variable name of nonrec=1.
- All mined stopes at Scuddles are expanded 5m north, south, east and west to capture all unrecoverable mineralisation. Expanded stopes are given the variable name of nonrec=2.
- Further to this, material that was deemed to be unrecoverable by the Mine Planning department was excluded from the model and assigned the variable of nonrec=3.
The block model was validated using the following techniques:
- Tonnes and grade, for each domain, was compared to previous years' results.
- Moving Window plots were created for all orebodies included in this Resource report. Analysis of plots showed good correlation of sample composite grades to block grades.
Waterfall comparison charts were constructed for each ore body and mining area. Waterfall charts for Gossan Hill mining area are displayed in Figure 55 to Figure 60 inclusive.</td></tr>
<tr><td>Moisture</td><td>All tonnages throughout the Mineral Resource, Ore Reserves and reconciliation process are reported as dry tonnes.</td></tr>
<tr><td>Cut-off parameters</td><td>Mineral Resources are reported to a cut-off NSR dollar value.
The NSR is a dollar value calculated from the grade and tonnes of a given block. Factors involved in the calculation include metallurgical recovery, milling cost, metal price and exchange rate financial assumptions, concentrate road and sea transportation costs (both dollar value and concentrate loss), royalties payable and refining charges.
The NSR cut-off for 2013 is A$$95/t.
Mineral Resources at Golden Grove are reported to 80% of the Ore Reserves cut-off. This provides a proxy for definition of Mineral Resource with eventual economic extraction potential.</td></tr>
<tr><td>Mining Factors or assumptions</td><td>Future mining factors and assumptions have been based on current mining practices.
Mining comprises long-hole open stoping and ore is hauled or hoisted to the surface.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>Metallurgical factors are incorporated into model block values via the calculation of a NSR value.
Recovery of payable minerals is dependent on iron ratios. Lower iron mineralisation is more amenable to copper and zinc recovery.
Recovery of precious metal mineralisation is dependent on zinc concentrations. Higher grade zinc mineralisation is amenable to better precious metal recoveries.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>No environmental assumptions have been used in the classification of the Golden Grove Mineral Resources.</td></tr>
<tr><td>Bulk Density</td><td>All samples have bulk density measurements taken in the core yard to be used as specific gravity (SG).
All blocks that did not have an SG estimated or assigned, or if the SG grade estimated was negative, were allocated an SG of 2.82.
All SG ≤ 2.82 values are updated with a calculated SG value. These values were calculated using the empirical formula below:
Calc_SG = (100/(35.294-([zn]*0.202)-([cu]*0.253)-([pb]*0.321)-([fe]*0.223)))</td></tr>
<tr><td>Classification</td><td>A multidisciplinary approach to Mineral Resource classification, involving geology, geostatistics and mining, was under taken. This was used in conjunction with an overriding consideration in grade confidence and geological continuity.
Drill density (as a proxy for data density), estimation run, number of samples and drillholes used in estimation for given block also influenced the classification. A summary of the guidelines are presented in Table 67.
Table 67 Quantitative Mineral Resource classification criteria
<table><tr><th>Classification</th><th>Drillhole spacing</th><th>Estimation run filled</th><th>Number of drillholes</th><th>Number of samples</th></tr><tr><td>Measured</td><td>10mx10m to 15mx15m</td><td>1-2</td><td>>=5</td><td>10-15</td></tr><tr><td>Indicated</td><td>20mx30m to 30mx30m</td><td>2-3</td><td>2-5</td><td>5-15</td></tr><tr><td>Inferred</td><td>Wider spacing</td><td>3-4</td><td><=2</td><td>1-15</td></tr></table>
As well as the quantitative approach taken to the Mineral Resource classification, subjective qualification was also used. This included decisions to include, or not, areas that did not strictly meet the quantitative criteria.</td></tr>
</table>

Audits or reviews	Internal audits were conducted in 2012 and 2013. Golden Grove 2012 Mineral Resource Checklist, conducted by Jared Broome, Anna Lewin. Golden Grove 2013 Mineral Resource Checklist, conducted by Jared Broome.
Discussion of relative accuracy/confidence	Assessment of model performance has been completed using design vs. claimed vs. mill reconciliation system. Reconciliation factors for the 2012/2013 financial year are summarised in Table 68, Table 69 and Table 70. Where: ▪ Design vs. Claim represents Mineral Resource model tonnes and grade/ geological grade control system tonnes and grade. ▪ Claim vs. Mill represents geological grade control system tonnes and grade/ concentrate back calculations of metal produced. ▪ Design vs. Mill represents Mineral Resource model tonnes and grade/ concentrate back calculations of metal produced.

Table 68 Reconciliation factors of tonnes by financial quarters and ore type

Year/quarter	**Ore type**	**Design vs. Claim**	**Claim vs. Mill**	**Design vs. Mill**
1201	Cu	0.99	0.97	0.96
	Zn	0.98	1.03	1.01
	Total	0.99	0.98	0.97
1202	Cu	0.93	0.99	0.92
	Zn	0.92	0.98	0.90
	Total	0.93	0.99	0.92
1203	Cu	0.95	0.92	0.87
	Zn	0.77	1.01	0.78
	Total	0.94	0.93	0.87
1204	Cu	0.95	0.94	0.89
	Zn	0.88	0.88	0.77
	Total	0.94	0.93	0.87
All Quarters	**Cu**	**0.95**	**0.96**	**0.91**
	Zn	**0.92**	**0.97**	**0.89**

Table 69 Reconciliation factors of grade by financial quarters and ore type

QTR	Ore Type	Design vs. Claim	Claim vs. Mill	Design vs. Mill
1201	Cu	1.16	0.90	1.05
	Zn	0.63	1.05	0.66
1202	Cu	0.99	0.86	0.85
	Zn	0.94	1.03	0.97
1203	Cu	1.06	0.91	0.96
	Zn	0.52	1.37	0.71
1204	Cu	0.92	1.02	0.94
	Zn	0.77	0.98	0.76
All Quarters	**Cu**	**1.04**	**0.90**	**1.05**
	Zn	**0.68**	**1.05**	**0.66**

Table 70 Reconciliation factors of metal by financial quarters and ore type

QTR	**Ore Type**	**Design vs. Claim**	**Claim vs. Mill**	**Design vs. Mill**
1201	Cu	1.15	0.88	1.01
	Zn	0.62	1.08	0.67
	Total	0.80	1.00	0.80
1202	Cu	0.92	0.85	0.79
	Zn	0.86	1.01	0.87
	Total	0.91	0.87	0.80
1203	Cu	1.00	0.84	0.84
	Zn	0.40	1.38	0.55
	Total	0.85	0.93	0.79
1204	Cu	0.88	0.96	0.84
	Zn	0.68	0.86	0.58
	Total	0.79	0.91	0.72
All Quarters	**Cu**	**0.99**	**0.87**	**0.86**
	Zn	**0.63**	**1.05**	**0.65**

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Mineral Resources.

6.5 Ore Reserves – Golden Grove Underground

6.5.1 Results

This Ore Reserves statement deals with the primary zinc and copper Mineral Resources at the underground operations of the Scuddles and Gossan Hill deposits. The open pit oxide Copper Ore Reserves are considered in a subsequent section of this Technical Appendix document and are not included in the Ore Reserves quoted in this section.

The Ore Reserves are derived from Mineral Resources using the geological database current as at 1 January 2013.

The Ore Reserves are based on detailed development and stoping designs and have been economically evaluated according to current metal price forecasts, recent operational costs, and mining and metallurgical recoveries.

The Golden Grove Underground Ore Reserves estimates as at 30 June 2013 are shown in Table 71 and Table 72. Changes in Ore Reserves from 2012 are primarily a result of mining depletion, new metallurgical recoveries, new financial parameters and updated geological information.

Table 71 Golden Grove underground operations Ore Reserves as at 30 June 2013

Mine/Commodity	Classification	Tonnes (Mt)	Cu %	Pb %	Zn %	Ag g/t	Au g/t
Gossan Hill - Zinc	Proved	0.4	0.7	1.3	9.8	90	1.6
	Probable	1.0	0.7	1.4	10.8	110	2.2
Scuddles - Zinc	Proved	0.2	0.3	1.1	11.9	89	1.0
	Probable	0.0	-	-	-	-	-
Total - Zinc	**Proved**	**0.6**	**0.6**	**1.2**	**10.5**	**90**	**1.4**
	Probable	**1.0**	**0.7**	**1.4**	**10.8**	**110**	**2.2**
	Total (Zinc)	**1.6**	**0.7**	**1.3**	**10.7**	**100**	**1.9**
Gossan Hill - Copper	Proved	1.8	2.3	0.0	0.3	16	0.5
	Probable	0.6	2.8	0.4	4.1	48	3.5
Scuddles - Copper	Proved	1.6	2.5	0.0	0.4	12	0.4
	Probable	0.6	2.4	0.0	0.2	9	0.2
Total - Copper	**Proved**	**3.4**	**2.4**	**0.0**	**0.4**	**14**	**0.5**
	Probable	**1.2**	**2.6**	**0.2**	**2.0**	**28**	**1.8**
	Total (Copper)	**4.6**	**2.4**	**0.1**	**0.8**	**18**	**0.8**

[*]Totals may differ due to rounding;

Table 72 Golden Grove underground operations Ore Reserves as at 30 June 2013, contained metal

Mine/Commodity	**Classification**	**Cu ('000t)**	**Contained Metal[†]**			
			Pb ('000t)	**Zn ('000t)**	**Ag (Moz)**	**Au ('000 Oz)**
Gossan Hill - Zinc	Proved	3	5	39	1.2	21
	Probable	7	14	109	3.4	72
Scuddles - Zinc	Proved	1	2	25	0.6	7
	Probable	0	0	0	0.0	0
Sub-Total (Zinc)	**Proved**	**3**	**7**	**65**	**1.8**	**27**
	Probable	**7**	**14**	**109**	**3.4**	**72**
	Sub-Total (Zinc)	**11**	**21**	**174**	**5.2**	**99**
Gossan Hill - Copper	Proved	43	1	6	1.0	32
	Probable	16	2	23	0.9	63
Scuddles - Copper	Proved	40	1	6	0.6	21
	Probable	14	0	1	0.2	4
Sub-Total (Copper)	**Proved**	**82**	**1**	**12**	**1.6**	**52**
	Probable	**30**	**3**	**24**	**1.0**	**67**
	Sub-Total (Copper)	**113**	**4**	**36**	**2.6**	**119**
Total Contained Metal[†]		**120**	**25**	**210**	**7.8**	**220**

[*]Totals may differ due to rounding; [†]Contained metal does not imply recoverable metal

6.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Wayne Ghavalas, confirm that I am the Competent Person for the Golden Grove underground operations Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Golden Grove underground operations Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 678,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 October 2013 was $HKD 1.72).

I verify that the Golden Grove underground operations Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Ore Reserves.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Golden Grove underground operations Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Wayne Ghavalas – 26/11/13

Michelle Wright (Witness)

6.5.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 73.

Table 73 Contributing Experts – Golden Grove Underground Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Kiki Kosmara, Senior Mining Engineer, MMG Ltd (Golden Grove)	Underground Mining
Nikki Dickinson, Mining Engineer MMG Ltd (Melbourne)	Underground Mining
Wayne Ghavalas, Underground Mine Manager MMG Ltd (Golden Grove)	Underground Mining
Tim Goodale, Senior Resource Geologist MMG Ltd (Golden Grove)	Geology, (2013 Mineral Resource Estimation – Gossan Hill)
Lauren Stienstra, Senior Mine Geologist MMG Ltd (Golden Grove)	Geology, (2013 Mineral Resource Estimation – Scuddles)
Dario Krmek, Project Metallurgist MMG Ltd (Golden Grove)	Metallurgy, NSRAR Metallurgical formulas
Stephen Ross, Commercial Manager MMG Ltd (Golden Grove)	Operating costs
Tim Goodale, Senior Resource Geologist MMG Ltd (Golden Grove)	Mineral Resource and geology reconciliation
Ben Ryan, Environment Superintendent MMG Ltd (Golden Grove)	Environment
Anthony Bennett, Geotechnical Engineer MMG Ltd (Golden Grove)	Geotechnical
Brooke Creemers, Community Relation Specialist MMG Ltd (Golden Grove)	Social and Community Relation Agreements
Gavin Marre, Senior Business Analyst MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

6.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

6.6.1 Mine Design

Stopes are designed using outline polygons created at 5m intervals, typically in cross-section. These are created based on the cut-off value, geotechnical parameters, design parameters and practical mining considerations.

Stopes are designed to selectively exclude barren intrusions and low grade zones where possible. Solid triangulations are generated from the polygons. These are interrogated against the resource block model to determine the contents of the stope designs. Where initial designs resulted in stope sections below the cut-off value, these sections are either redesigned or removed from design if practical.

A minimum mining width of 4 metres is used.

Level intervals are generally 30 metres.

Development is generally mined at 5.5 metres height by 5.5 metres width.

The layout of Gossan Hill mine is shown in long-section in Figure 61. The layout of Scuddles mine is shown in long-section in Figure 62.

Figure 61 Long-section of Gossan Hill Mine showing mining areas and stopes in the Ore Reserves

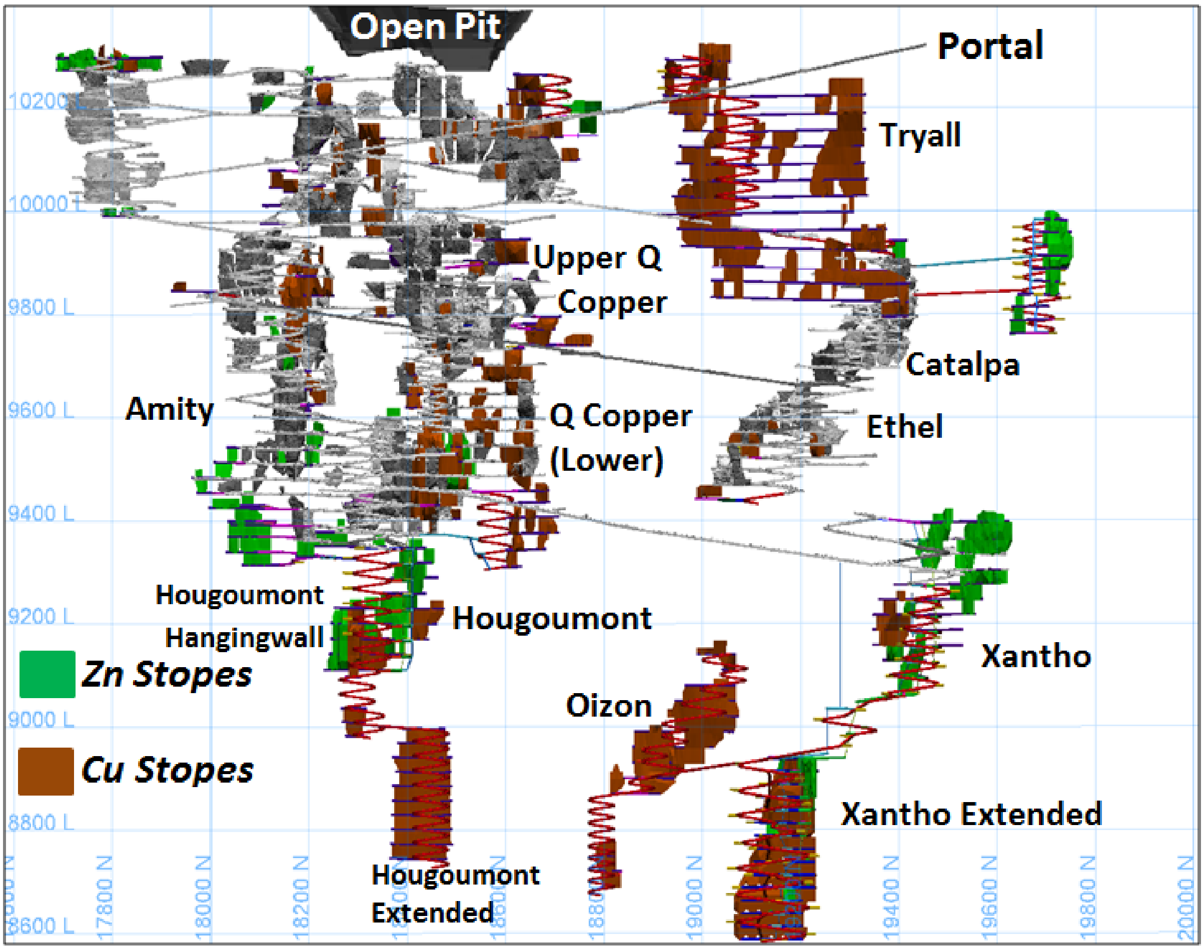
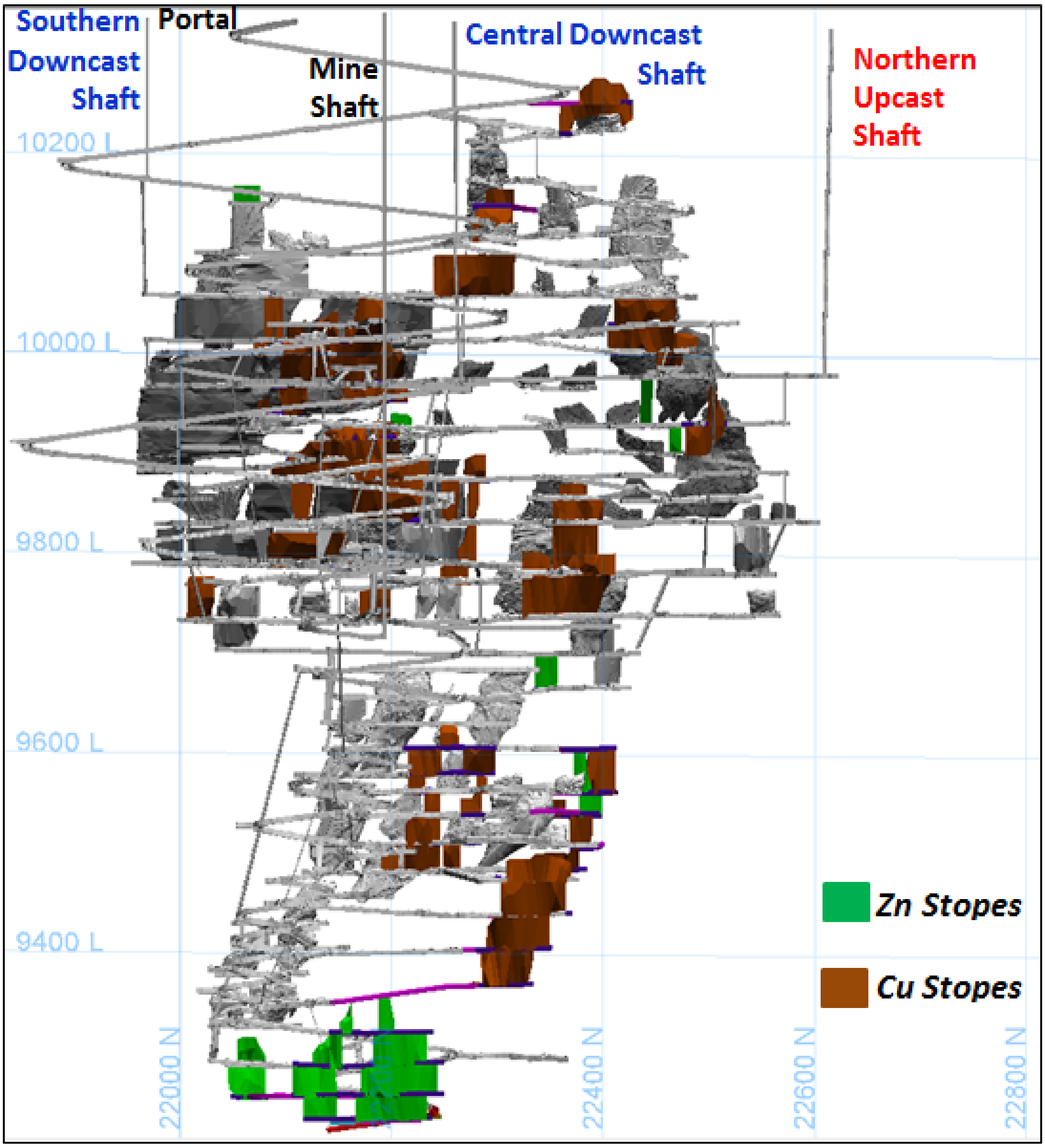


Figure 62 Long-section of Scuddles Mine showing mining areas and stopes in the Ore Reserves



6.6.2 Geotechnical Parameters

Stope Hydraulic Radius (HR) Guidelines

All stopes at Golden Grove are designed with geotechnical input. The Stability Graph Method as well as back analysis is used to determine stable spans. Numerical modelling is used to identify stress-related problem areas and define extraction sequences. The current practice is:

- Each stoping area is given an estimate of stable Hydraulic Radius, based on Q' values, experience with similar mine areas and numerical modelling (as per Table 74);
- Detailed mapping of geotechnical domains is then undertaken for each stope and the Stability Graph Method analysis is performed;
- Stress modelling is then carried out for each stope;
- All results are analysed and recommendations made in the 'Geotechnical Consideration' section of the Stope Design Record (SDR).

Forward looking risks are recognised as the following:

- As the mining horizon progresses deeper there is an increasing potential for the corresponding increase in in-situ stresses to have a deleterious impact on stope stability and ground support costs.
- The increase of in-situ stress will also affect the stope performance and the overall mining schedule may be impacted.
- Mining sequence could also be changed depending on stress characteristic and orientation.

Table 74 Base values of allowable hydraulic radius for different orebodies

Orebody	**Maximum HR, m**					
	Unsupported			**Supported**		
	HW	**FW**	**Crown**	**HW**	**FW**	**Crown**
Amity	9	9	5	11	11	6
Catalpa	8	9	5	9	12	7
Ethel	9	10	7	12	13	9
Hougoumont rhyodacite	8	8	5	9	10	6
Hougoumont dolerite	6	6	3	7	7	4
Hougoumont sediments	9	9	6	10	10	8
A Copper	9	9	7	11	11	9
Q Copper	12	12	8	15	15	10
Xantho	5	7	3	7	9	5

Ground Support (GS)

Golden Grove Ground Support standards are universally applied to both Gossan Hill and Scuddles mines. Ground Support standard implementation -

- Standard support above 800RL consists of mesh and friction bolts standards.
- Below 800RL generally supported with fibrecrete and friction bolts
- CHF headings supported with a combined of fibrecrete, mesh and friction bolts.
- Cable bolts are used in all intersections, stope brows and for mid-level hanging wall stability.

Future development in Xantho areas will require higher capacity (both static and dynamic) Ground Support. A number of options are being considered including chain-link mesh and yielding bolts. Due to increasing seismicity along faults in the upper areas of the mine as mined-out areas increase, this Ground Support system may also need to be implemented in identified high seismic risk areas.

Seismic System and Seismicity

Golden Grove currently has a seismic system at Gossan Hill mine only. The system consists of 14 geophones comprising of 11 tri-axial and 3 uniaxial sensors.

Gossan Hill is becoming increasingly seismically active. Of the 47,500 total events recorded since the seismic system was installed in 2006, over 17,000 of these have occurred in the last 12 months continuing the 2012 trend. Of these recent events, 36 major events (above M_L 0.0) and 3 events M_L 1.0 and above have occurred, one of which causing major damage to the Xantho Decline. This continued trend of seismicity reflects the deeper mining areas and requirement for dynamic ground support for which trials are underway.

6.6.3 Processing (Metallurgical) Recovery Factors

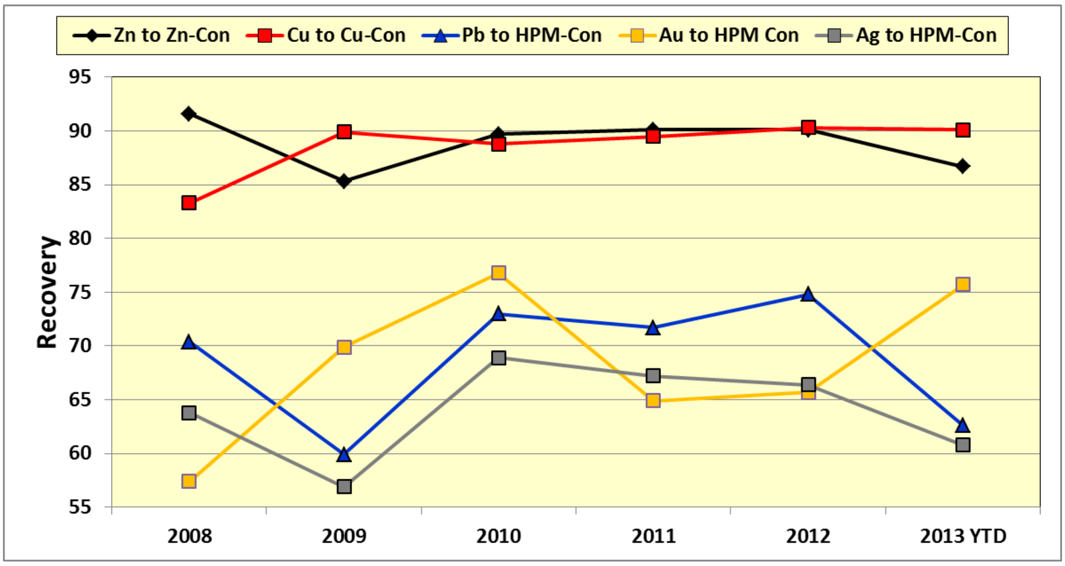
Operating parameters for the processing plant are determined annually by reviewing historical data, the mining forecasts (advising ore types and quantities available for processing) and anticipated metal prices. The historical mill recoveries for the last 5 year are shown in Figure 63. The average metallurgical recovery since 2008 is:

- 88.9% for zinc in the zinc concentrate
- 68.7% for lead in the lead (HPM) concentrate
- 64.0% for silver in the lead (HPM) concentrate
- 68.4% for gold in the lead (HPM) concentrate
- 88.6% for copper in the copper concentrate

Deleterious elements such as high iron could impact the recovery. Talc and Magnetite has not been estimated extensively in the block model used.

During copper sulphide campaigns, the processing plant sometimes experiences high levels of talc in the ore (gangue associations). Depending on the severity of the talc, the flotation circuit configurations can require alterations to counter the problem. The most common method of reducing the impact of talc in feed is to prefloat a talc rich concentrate and discard to tailings. Hence a saleable concentrate can be maintained, but unfortunately minor copper losses to the prefloat occur. Fluorine (F) is associated with talc and its presence is increased when talc is present. The fluorine (and chlorine) target in copper concentrate is 1500ppm combined (in any ratio). Ensuring a copper grade greater than 20.5% provides for satisfactory fluorine levels in the final concentrate.

Figure 63 Mill recoveries for last five years and 2013 YTD



6.6.4 Realised Revenue Factors (Net Smelter Return)

The realised revenue from the ore is expressed using a calculated Net Smelter Return After Royalty (NSRAR).

The price and foreign exchange assumptions are as discussed in Section 2.1. These prices are in real terms and based on the corporate economic assumptions as at 1 February 2013. Three sets are used, short-term, medium term, and long term. The relevant pricing environment depends upon the planned timing of extraction. The specific break-down by orebody is shown in Table 75.

The realisation costs for zinc concentrates are shown in Table 76, the realisation costs for copper concentrates are shown in Table 77, and the realisation costs for lead (HPM) concentrates are shown in Table 78.

Table 75 Golden Grove underground - price basis used for each orebody

NSRAR terms	Orebody
NSRAR_2013 (Short Term)	Amity Hougoumont Zinc above 1024 Level A Copper C Copper Q Copper D Zinc AB Zinc Ethel Catalpa
NSRAR_MT (Medium Term)	Hougoumont Copper above 1024 Level Scuddles Xantho
NSRAR_LT (Long term)	Cambewarra Hougoumont below 1024 Level Oizon Tryall Xantho Extended

Table 76 Golden Grove underground - NSR inputs for zinc concentrate realisation costs

Zinc		
Metal Paid - Zn (total)	85%	%
Minimum Deduction - Zn	8%	% dry
Base Treatment Charge - Zn	200	US$ / dmt con
TC Basis Price - Zn	2,000	US$ / t Zn
TC Escalator - Zn	0.050	US$ / (US$ / t)
TC Deflator - Zn	0.020	US$ / (US$ / t)
Silver		
Deduct - Ag	93.3	g / dmt con
Metal Paid - Ag (remainder)	65%	%
Penalties (Zn-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Road Freight & Port Costs	50	A$ / wmt con
Sea Freight	31.5	US$ / wmt con

Table 77 Golden Grove underground - NSR inputs for copper concentrate realisation costs

Copper		
Metal Paid - Cu (total)	97%	%
Minimum Deduction - Cu	1.0	% dry
Base Treatment Charge - Cu	80	US$ / dmt con
Refining Cost	0.08	US$ / lb
Silver		
Minimum Deduction - Ag	30	g / dmt con
Metal Paid - Ag (remainder)	90%	%
Refining Charge - Ag	0.35	US$/Oz payable
Gold		
Minimum Deduction – Au	0	g / dmt con
Metal Paid - Au (remainder)	95%	%
Refining Charge - Au	4.5	US$/Oz payable
Penalties (Cu-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Road Freight & Port Costs	50	A$ / wmt con
Sea Freight	31.5	US$ / wmt con

Table 78 Golden Grove underground - NSR inputs for lead (HPM) concentrate realisation costs

Lead			
Metal Paid - Pb (total)		95%	%
Minimum Deduction – Pb		3%	% dry
Base Treatment Charge – Pb	(CY14-16)	210	US$ / dmt con
	(CY17+)	175	US$ / dmt con
Silver			
Minimum Deduction - Ag		50	g / dmt con
Metal Paid - Ag (remainder)		95%	%
Refining Charge - Ag		2.5	US$/Oz payable
Gold			
Minimum Deduction – Au		2.0	g / dmt con
Metal Paid - Au (remainder)		95%	%
Refining Charge - Au		6.0	US$/Oz payable
Penalties (Pb-Con.)			
	No Penalties are Assumed		
Freight, Sampling and Insurance			
Road Freight & Port Costs		50.0	A$ / wmt con
Sea Freight (Antwerp)		42.5	US$ / wmt con

Concentrate moisture estimates assumptions are given in Table 79. Royalties payable are given in Table 80.

Table 79 Golden Grove underground - concentrate moisture assumptions

Concentrate	Moisture
Zinc	8.9%
Copper	9.0%
Lead	9.2%

Table 80 Golden Grove underground - royalties payable

Concentrate	Royalties
Zinc	5.0%
Copper	5.0%
Lead	5.0%
Gold	2.5%
Silver	2.5%

6.6.5 Mining Costs

The mining costs for the site were prepared using the actual cost data from January 2012 to March 2013. The unit operating cost is derived from the function of total cost and total volume of each contributing cost element. This cost model is also used to evaluate Ore Reserves estimation and cut-off grade calculation.

The mine operating costs per unit cost are listed in Table 81. Mining cost used for estimation is fixed for period of mine life.

As the mining horizon gets deeper, the production rate is expected to decrease as haulage tonnes-kilometres increases. Development rate will also be impacted due to changing to ground support requirement which could result on more delay for each cycle. Hence, mine operating cost is expected to generally increase with depth.

Table 81 Golden Grove underground - mining costs

Mining Costs	$/unit
Development (Bolt and mesh)	$4,372/m
Development (Fibrecrete)	$5,609/m
Stripping (Bolt and mesh)	$58/m
Stripping (Fibrecrete)	$75/m
Rehab (Bolt and mesh)	$2,679/m
Rehab (Fibrecrete)	$3,916/m
Cable bolting	$62/m
Raiseboring	$975/m
Long hole rise (upholes)	$3,086/m
Production Drilling	$74/m
Production Charging	$66/m
Bogging	$5.08/t
Hauling	$2.34/tkm
CHF Barricades	$13,050 each
CHF Filling	$31/m^3
Milling	$37.00/t
Mining Overhead	$21.60/t
Geology	$5.89/t
Site G&A	$18.30/t

General and Administration site cost based on the 15 months cost data is $18.30 per tonne ore. Mining overheads, geology costs and corporate charges have been re-estimated following a similar process. Although calculated, the corporate charges have not been used in the economic evaluation of the Ore Reserves estimate. The majority of these costs have increased compared to last year costs model (refer Table 82).

Table 82 Golden Grove underground - other costs

Costs	$/unit	
	2012	2013
General and Administration	$17.20/t	$18.30/t
Mining Overhead	$23.00/t	$21.60/t
Geology	$5.00/t	$5.89/t
Corporate	$5.00/t	$5.84/t

6.6.6 Mining Factors and Assumptions

Three different NSRAR values are used in the design of stopes: NSRAR (Short Term), NSRAR (Medium Term) and NSRAR (Long Term) – each calculated using the corresponding economic assumptions as discussed in Section 2.1. The NSRAR that was used for the cut-off for a particular stope is based on when the stope is expected to be mined.

Capital and operating development is assigned to one or more stopes for the purposes of distributing development costs as required by the economic evaluation process.

Reconciliation

Ore Reserves reconciliation is undertaken to determine the reliability of geological interpretations and grade models as well as the mining recovery and dilution assumptions. The reconciliation process includes daily monitoring of milled grades against visual estimates and planned grades based on firing design. Following the completion of each stope, a stope closure process is undertaken examining stope performance. Monthly production data is reconciled against the planned tonnage and grade from production firings. The monthly reconciliation reports are also produced comparing Claimed Ore Mined against Mill Reconciled Mined and Ore Reserves data.

Figure 64 and Figure 65 show the reconciliations for zinc and copper stopes mined in the last 12 months, comparing design stope performance to actual stope performance.

Figure 64 Copper stope reconciliation – July 2012 – June 2013

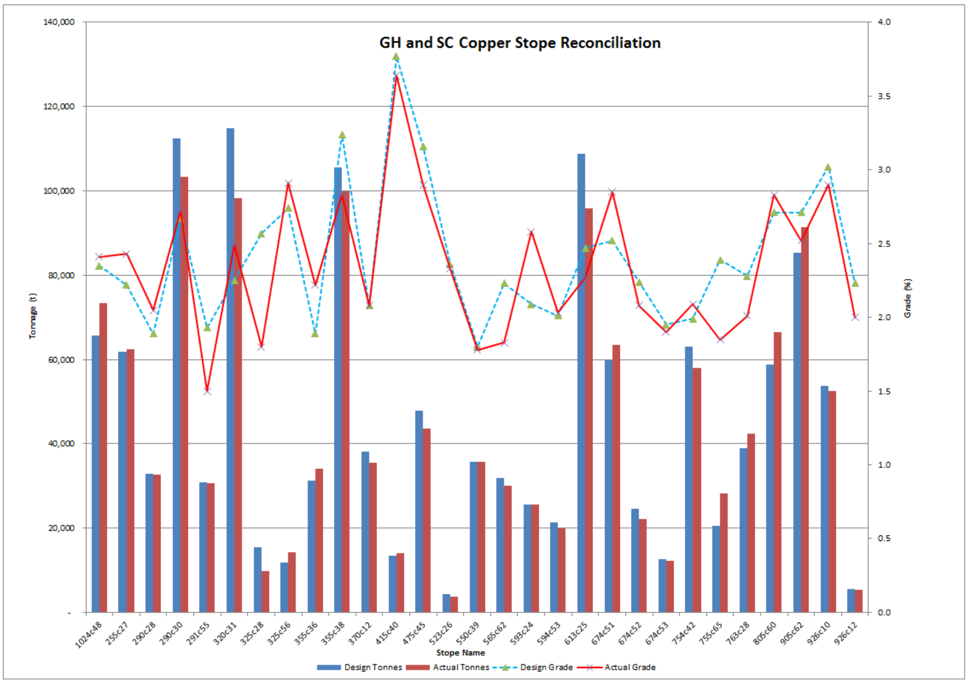
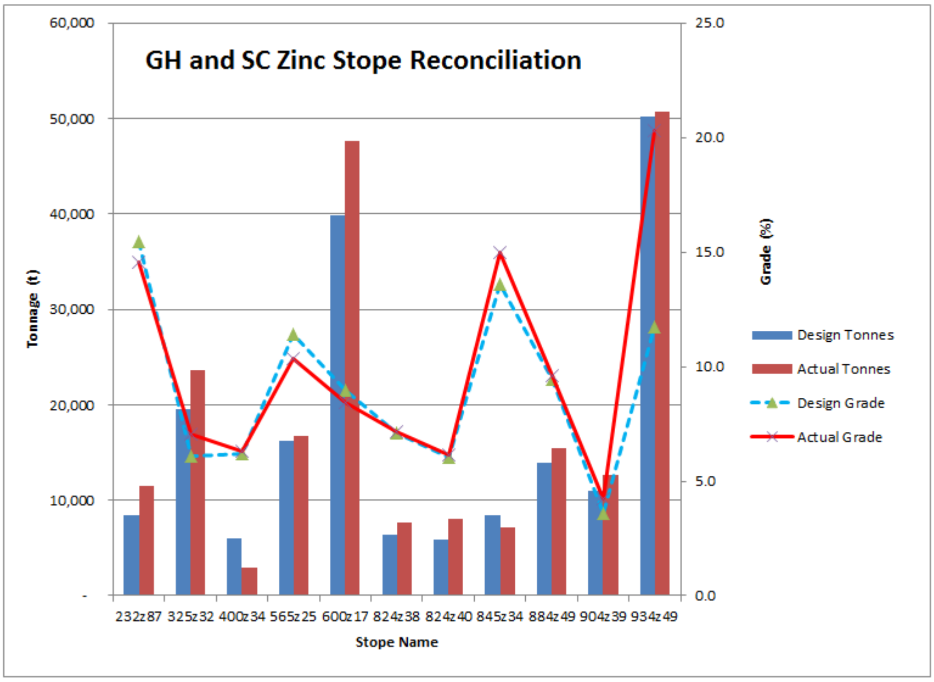


Figure 65 Zinc stope reconciliation – July 2012 – June 2013



Dilution and Recovery

Based on the reconciliation work, a 90% recovery factor has been applied to all stopes at Gossan Hill and Scuddles. The dilution factors that have been applied to the various orebodies are listed in Table 83.

Table 83 Golden Grove underground mining dilution factors

Domain	Dilution (Zn)	Dilution (Cu)	Dilution Grade*
Hougoumont and Hougoumont H/W	20%	10%	Zn Ore 1% Cu Ore 10%
Xantho Extended, Xantho and Oizon	20%	15%	Zn Ore 2% Cu Ore 10%
All other domains, includes Scuddles	10%	10%	Zn Ore 5% Cu Ore 10%

**Note the dilution grade is expressed a percentage of the original grade*

Mining dilution and recovery factor assumptions of new mining areas may change as soon as actual data is obtained. Changes to the mining method and sequence will also impact the mining recovery factor

6.6.7 Processing Costs

The current Golden Grove Processing operation caters for either a single concentrate production in the case of copper sulphide (CuS) or copper oxide (CuO) ore, or a 2-stage sequential concentrate production for lead (Pb)/zinc (Zn) ore.

The processing cost used in the Ore Reserves estimation needs to be reviewed after the Open Pit Oxide mine is completed as a result of higher distribution of unit cost compared to combined underground and open pit materials processing cost.

The processing cost was determined from data for recent months in which only underground ore was processed (i.e. no open pit ore was processed in those months). For the Ore Reserves calculation purposes the processing cost was estimated to be $37 per tonne ore.

6.6.8 Infrastructure

Mining Infrastructure

Existing major infrastructure at MMG Golden Grove includes:

- Scuddles underground mine with winder/headframe, decline portal, pump station and crusher station
- Scuddles main surface fan
- Radio hut, Plant air compressor and winder emergency hoist
- Gossan Hill underground mine with decline portal, pump station, ROM and crusher station
- Gossan Hill Surface fans (FSUC, MVR and CVR)
- Golden Grove Open Pit mine
- Mine site offices, mineral processing plant, laboratory, concrete batch plan, backfill plant, warehouse, fuel farms, washing pad facility, disposal facility, and workshops
- Surface magazine and Orica emulsion storage facility
- Land belt conveyor connecting Crusher station to mineral processing plant
- Accommodation village located 5km to the south-southwest of the mine offices and accessed via a sealed road
- 66kW overhead power transmission line linking the mine to the Western Power grid
- Groundwater bores for the supply of potable water
- Three tailings storage facilities (TSF1 is decommissioned)
- 27km mine water disposal pipeline to Lake Wownaminya
- Three of 1.15MW power generators
- Potable Water Reverse Osmosis (RO) treatment plant

Processing Plant

The mill has been operating continuously since 1990, processing both zinc and copper ores and producing zinc, copper and HPM concentrates.

Feed to the processing plant consists of pre-crushed sulphides ore or raw copper oxide ore. Pre-crushing of sulphide ore on the ROM is carried out by a contract company. The pre-crushed ore is fed to the mill to maximise throughput and offset the observed increase in ore hardness with mine depth. While previous work has been completed on the blasting of the ore underground resulting in better fragmentation, pre-crushing of the feed was reinstated in 2011 as the definitive means to increase production.

The ROM stocks are fed through the primary crusher, and this crushed product is conveyed overland to the Gossan Hill Stockpile. The Gossan Hill Stockpile has four vibrating feeders which allows for control of coarse to fine ore ratio blending, before being fed to the primary grinding circuit consisting of a primary SAG mill and two secondary ball mills. There is also the capacity to feed ore to the plant from the Scuddles Stockpile singularly or concurrently with the Gossan Hill Stockpile.

The Scuddles underground mine presents the mill with another feed source. The ore from is hoisted to the surface via shaft/winder mechanisms, and this is conveyed to the Scuddles stockpile. The Scuddles stockpile is campaigned as either Zn or Cu ore at any one time, depending on the mill campaigns, or volume of stopes being mined. This can result in Scuddles ore not being processed during some mill campaigns based on timing.

The ore is milled and close-circuit cycloned to produce a flotation feed target of <106µm, which is then fed through the flotation circuit. Any oversize material (>106µm) is directed back to the ball mills for further grinding, to ensure the economic mineral is liberated sufficiently, which in turn assists in increasing possible recovery downstream. There are two ball mills that can be used in the circuit, with the second ball mill being utilised if the throughput of the circuit is sufficiently high.

In the flotation circuit, depending on the ore type being processed, either a High Precious Metal (Pb HPM – a combined concentrate of lead, gold [Au] and silver [Ag]) and subsequent Zn concentrate, or Cu concentrate (sulphide or oxide), is produced. The copper oxide ore is unusual as it is sulphidised via sodium hydrosulphide prior to being floated, and at other dosing points throughout the circuit. This effectively alters the characteristics of the oxide to that of a sulphide, allowing a concentrate to be produced through normal flotation.

For all feed types, the ore slurry is passed through roughing, cleaning and scavenging flotation circuits with appropriate chemicals added to assist in flotation, in order to upgrade the economic mineral in the ore to a concentrate with saleable grade;

- >30% Pb HPM,
- >50% Zn and
- 20.5% Cu (sulphide)
- 20.5% Cu (oxide)

Power

Electricity is supplied from the WA grid through a southern distribution centre at Three Springs. Power consumption is typically around 14 MW, although the demand is expected to increase to 17 MW in the coming years. Standby generators are installed to enable essential services and underground fans to operate and to prevent bogging of tanks and thickeners.

Water

Water supply for the operations is secure with sufficient groundwater supply. Groundwater abstraction is approved by the Department of Water Licence to Take Water (GWL103574 (7)) with water abstraction for 2012 being 37% of the licence limit.

The majority of the groundwater is supplied through dewatering of both underground mines.

Potable water is supplied from groundwater bores and is treated through a reverse osmosis (RO) plant. Testing of the RO plant and potable water occurs monthly, reporting against Australian Drinking Water Guidelines (2011).

Communications

Communication facilities include:

- UHF Radio for surface area includes, processing plant, camp and ROM
- Radio - Leaky feeder for mainly underground Gossan Hill/Scuddles and surface area on certain channels
- Landline telephone for surface area and underground
- Mobile telephone for surface area includes camp
- Internet communication for surface area includes camp

Airport

An airstrip, located 5km to the south of the mine offices was sealed in 2007 providing all-weather access. It is serviced by flights from both Perth (one hour flight time) and Geraldton (45 minutes flight time).

Road Access

Access to Golden Grove is via sealed roads from Perth to Paynes Find and from Geraldton to Yalgoo. The Yalgoo to Paynes Find road is sealed between Yalgoo and Golden Grove while the remainder between Golden Grove and Paynes Find is a formed gravel road that can be closed to traffic during periods of wet weather.

6.6.9 Tenements

All the tenements at Golden Grove are held as Mining Leases, with a total strike length of 37km and an area of approximately 13.05 ha. The tenements have long tenure, as discussed in the Mineral Resources section.

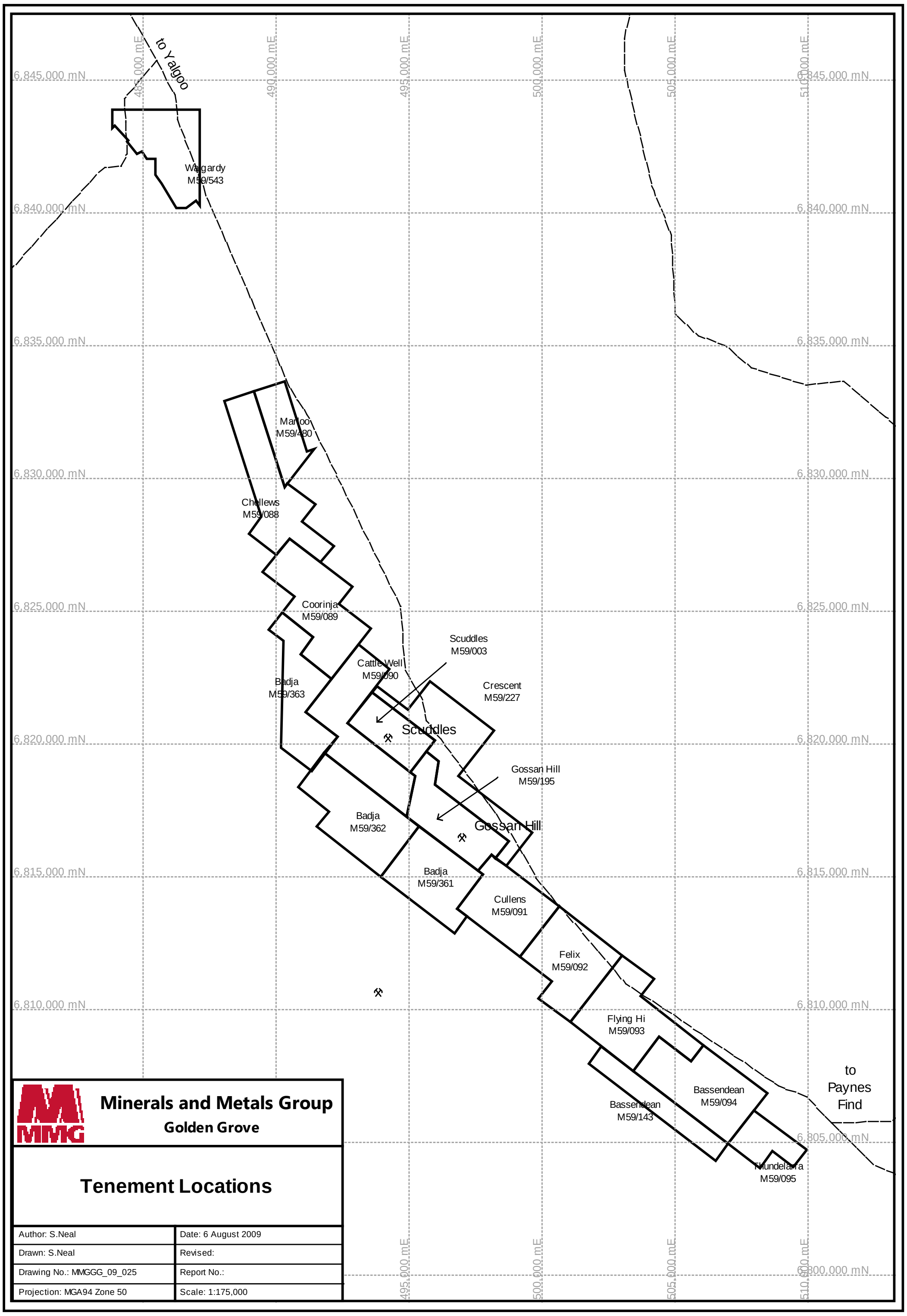
All currently known Mineral Resources are located on three tenements, M59/03 (Scuddles), M59/195 (Gossan Hill) and M59/92 (Gossan Valley and Felix). All tenement conditions have been met and those with shortfalls in expenditure have been granted exceptions. Table 66 in the Mineral Resource Section lists all the leases with their expiry date.

Golden Grove owns the Muralgarra Pastoral Station leasehold land which was purchased in 2007. Golden Grove currently has a de-stocking exemption on this pastoral lease until 2013 and is developing a strategic and diversified management plan with a focus on carbon sequestration and biodiversity offset project opportunities.

Table 84 Golden Grove mining tenements with their predominant mining activity and underlying pastoral lease

Tenement Lease	Activity
M59/195	Back Fill Plant, Gossan Hill Boxcut, ROM Pad & Crusher, Conveyor, Contractor workshops, Exploration
M59/362	Mining Village, Batch Reactor
M59/90	Borrow Pit, TSF1, TSF 3, Evaporation Pond A, B, C, Soil Depository.
M59/03	Scuddles Mine, Process Plant, TSF2, Admin, conveyor
M59/227	TSF2, Borrow Pit, Landfill, Gossan Mining Offices.
L59/22	Airstrip
L59/26	Camp Access Road
G59/19-23	Camp
G59/24	Camp Waste Water Treatment Facility

Figure 66 Tenement locations



6.6.10 Social Factors

MMG Golden Grove is located within the Shire of Yalgoo in the Murchison Region of Western Australia. The nearest community to Golden Grove is the Yalgoo Township, which is situated approximately 56km to the north of the site, with a population of approximately 100. The key stakeholders include the local government and community, pastoralists, employees and the Geraldton Port Authority.

MMG Golden Grove has maintained good partnership with neighbouring pastoral, traditional owner groups through various programs such as; Bayalgu Program, CHMA Badimia People, Life of Mine investment Agreement Shire of Yalgoo and GPA AQMP agreement.

Golden Grove is located in an area that is under claim by two Indigenous native title claimant groups. The Badimia People and the Widi Mob native title groups have registered claims (WC96/98) and (WC97/72) respectively. These claims intersect the MMG tenements from north to south with the Badimia claim overlaying the southern tenements and the Widi Mob claim overlaying the northern leases.

All Golden Grove tenements that overlay the abovementioned Native Title Claims were granted prior to the enactment of the Native Title Act (1993). As such result no formal land access agreements are required however, Golden Grove continues to fulfil its statutory heritage responsibilities in line with the Aboriginal Heritage Act (1972) and the Aboriginal and Torres Strait Islander Heritage Protection Act (ATSHIP 1984). Golden Grove ensures that relevant heritage surveys are conducted prior to the commencement of any new projects or changes at the site.

6.6.11 Environmental

The Gossan Hill and Scuddles underground mines operate under license L8593/2011/1 issued by the Western Australian Department of Environment and Conservation (DEC) as required by the *Environmental Protection Act 1986*. This license was issued 15 September 2011 and expires on 15 September 2014.

Golden Grove also has a license to take water issued under *Rights in Water and Irrigation Act 1914*. This license permits the extraction of up to 3.51 GL of ground water per year for the purposes of mine dewatering, dust suppression, ore processing and servicing the mining camp.

Golden Grove has a working Closure Plan that is reviewed annually.

Waste rock from the Golden Grove underground mines is typically neutral to alkaline with low soluble salt content when undisturbed. The waste rock contains sulphides and is therefore regarded as Potential Acid Forming (PAF). The waste rock is also host to silver, arsenic, bismuth, cadmium, cobalt, copper, lead, antimony, selenium and zinc which characterise the Volcanic Hydrothermal Massive Sulphides (VHMS) sequence.

Waste rock from underground preferentially remains underground where it is used to as a source of backfill. The waste rock that is transported to surface is either returned underground as road base, or is encapsulated in the dedicated ROM Pad PAF Encapsulation Facility.

Both Scuddles and Gossan Hill mines intercept productive groundwater aquifers that require dewatering to facilitate mining. Mine water from Scuddles and Gossan Hill dewatering operations is directed to the Mine Water Clarifier (MWC) where it is mixed with lime and flocculent to remove metals and sediment from solution. From here the water is directed into Evaporation Pond B where it passes through rock baffles and typha[7] (to remove further sediment) into Evaporation Pond A. From this point, the treated water is used either for:

- Mining operations such as drilling;
- Dust suppression (surface and underground operations);
- Mineral processing; and/or/
- Discharge to Lake Wownaminya.

---

[7] *A genus of monocotyledonous flowering plants in the family Typhaceae of wetland habitat.*

6.6.12 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 85. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 85 JORC Code Ore Reserves assessment and reporting criteria for Golden Grove underground operations 2013 Ore Reserves

Assessment Criteria	Risk Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	Low	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The 2013 Mineral Resource block model used for generation of the 2013 Ore Reserves. An average of 62% of the Measured and Indicated current Mineral Resources estimate was converted into Ore Reserves for both Gossan Hill and Scuddles mine (it should be noted that the Mineral Resources cut-off grade is A$95/t versus A$120/t.) Further details are discussed in the Mineral Resources Section of this report.
Classification	Low	The Proved Ore Reserves category is determined when Mineral Resources confidence level is "Measured" and satisfies the financial criteria. The Probable Ore Reserves category is determined when Mineral Resources confidence level is "Indicated" and satisfies the financial criteria.
Site visits	Low	The Competent Person is based on site.
Study status	Low	The mine is currently operating. Ore Reserves inputs are based on actual historical performance data.
Cut-off parameters	Low	Due to the polymetallic nature of Golden Grove, all the factors relating to the value of the ore (representative of cash costs to mine gate) are combined into a NSRAR value. Based the economic assumptions and cost review, the NSRAR cut-off is A$120/t. The cut-off was used as a guide to generate the Ore Reserves shapes; however each stope is assessed individually for the development, haulage distance, backfilling requirements and any other additional costs to ensure that it is profitable to mine. These costs are considered on an individual stope basis and stopes that make a loss with these costs included have not been included in the Ore Reserves.
Mining factors or assumptions	Low-Medium	See Section 6.6.6 for details. The geotechnical parameters and production rates at increasing depths are seen as the highest risks associated with the mining factors.
Metallurgical factors or assumptions	Low	See Section 6.6.3 for details.
Environmental	Medium	See Section 6.6.11 for details.
Infrastructure	Medium	See Section 6.6.8 for details. As mining horizon getting deeper, the haulage cost, production rates, ground stress and underground temperature will be impacted.
Costs	Low	See Section 6.6.5 for details on mining costs. See Section 6.6.7 for details on processing costs.
Revenue factors	Medium	See Section 6.6.4 for details. The long-term predicted increase in Zinc price is seen as the highest risk to the revenue factors used.
Market assessment	Medium	See Section 2.2 for details.

Economics	Medium	Golden Grove is an operating mine. Costs detailed in this Appendix are based on historical actuals. Revenues are based on historical and contracted realisation costs and a realistic long-term metal price. The LOM financial model demonstrates the mine has a positive NPV calculated at a discount rate of 8%.
Social	Medium	See Section 6.6.10 for details.
Audit or Reviews	Medium	No external audits were undertaken. An internal review was undertaken by the then Group Manager Mining (now Group Manager – Technical Governance).
Discussion of relative accuracy/ confidence		A qualitative risk assessment of each discussed item is included with each individual item in the second column of this table. Details of various risks are discussed in each relevant section.
<u>Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4</u>		
Topography	Low	Golden Grove is located within the Yalgoo biogeographic subregion, which is characterised by open woodlands and scrubs on earth or sandy earth plains The area surrounding Golden Grove is of low to moderate relief with long ranges separated by extensive plains. Elevation is generally around 350m above sea level with the highest point in the region being Minjar Hill at approximately 380m above sea level.
Climate	Low	Golden Grove is situated within the Yalgoo bioregion and has a variable climate with characteristics of semi-arid and Mediterranean climates and is prone to long periods of drought. Most rainfall occurs during the winter months, although more occasional major rainfall events, largely associated with tropical cyclone activity off the northwest shelf, occur in the summer months and can result in localised flooding. Average rainfall is 290.9mm annually. Monthly rainfall has seldom exceeded evaporation onsite. The region has relatively mild winter and very warm summer.
Government Agreements	Low	MMG Golden Grove has a number of mining, exploration and general purpose tenements extending over approximately 13,000ha. These tenements overlap 5 pastoral leases, one of which is own by Golden Grove.
Waste Storage (Including Tails Storage)		Current mining operations are predominantly at the Gossan Hill underground mine and Open Pit. A small amount of waste is also generated from development activities at Scuddles underground mine. Waste rock from underground preferentially remains underground as a stope backfill material. Waste material from the Open Pit is deposited in the dedicated ROM Pad PAF Encapsulation Facility. Some of the mill tailings are returned underground as cemented hydraulic fill (CHF) and the remainder is stored in purpose built tailing storage facilities.
Mineral tenement and land tenure status		See Section 6.6.9 for details of tenements involved in mining activities. The Mineral Resources Section (Table 66) lists all Golden Grove Mineral Leases and their expiry date.

7. GOLDEN GROVE OPEN PIT OPERATIONS

7.1 Introduction and setting

The open pit operations are an adjunct to the main underground operations of Golden Grove. The open pit operations area is located on the northern flank of Gossan Hill, directly over the current underground mining operations at Gossan Hill.

Mining is carried out using conventional truck and shovel methods using a mining contractor under the supervision of the MMG Golden Grove Open Pit department. When completed, the pit will extend over an area of approximately 15.7ha and reach a maximum vertical depth of approximately 120m.

Approximately 3.0Mt of copper ore and a further 17.8Mt of waste rock are expected to be mined over the project life. Completion is expected in mid-2014. The mined ore will comprise oxide, transitional and primary sulphide material.

Copper oxide ore currently being mined is stockpiled on the existing run-of-mine (ROM) pad adjacent to the pit and conveyed to the Scuddles processing plant for treatment on a campaign basis, supplementing sulphide ores from Gossan Hill and Scuddles underground operations. Sulphide ores will be mined in the later parts of the operation and will also be stockpiled separately.

Figure 67 Aerial view of open pit and waste dump development

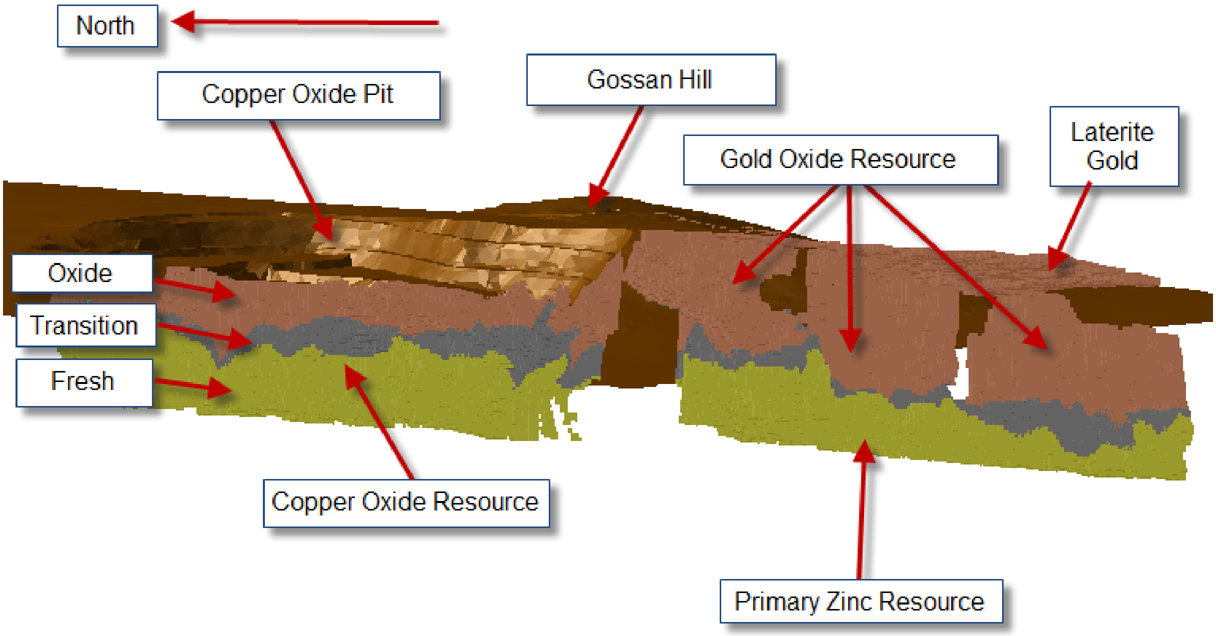
7.2 Mineral Resources – Golden Grove Open Pit

7.2.1 Results

The Golden Grove open pit Mineral Resource estimate for June 30 2013 was carried out by Optiro Pty Ltd utilising geological interpretations and data provided by MMG geologists. All Mineral Resources quoted in this report were estimated from three dimensional block models created with Vulcan software.

The Mineral Resource is reported in two parts; the copper Mineral Resource and the gold Mineral Resource (which includes primary zinc material beneath the gold deposit) as specified in Figure 68. The copper Mineral Resource is constrained by an optimised pit shell using long term metal prices, while the gold Mineral Resource is not constrained by an optimised open pit, but is reported above an elevation of 10,240mRL. Both areas include oxide, transitional and primary sulphide/fresh mineralisation material types.

Figure 68 Gossan Hill deposits views the north-west, below the surface



Mining of the copper Mineral Resource commenced in early 2011. Two reverse circulation grade control drill programs have been completed totalling 10,800m. The data from the two grade control programs was used to update the 2013 Mineral Resource.

Reportable Mineral Resources as estimated at 30 June 2013 for the copper mineral deposit and gold mineral deposit are shown in Table 86 and Table 87.

Copper Mineral Resource

The total copper Mineral Resource includes material above a cut-off of 0.7% Cu within the oxide, transitional and primary domains above the current final pit design and below June 30, 2013 pit topography surface.

0.8Mt @2.4% Cu stockpiles are entirely classified as Measured.

Table 86 Golden Grove Gossan Hill Copper Mineral Resource Table. Copper is reported above a cut-off grade of 0.7% Cu

Mineralisation type	Classification	Mt	Cu %
Stockpile oxide	Measured	0.8	2.4
Oxide	Indicated	1.2	2.3
Transitional	Indicated	0.6	2.2
Sulphide	Indicated	0.3	1.9
Total		**2.9**	**2.3**

Gold Mineral Resource

The gold Mineral Resource is reported above a 1.5g/t gold equivalent cut-off grade. The gold equivalent cut-off is calculated using the following formula:

$$Aueq = Au + Ag*1.5/80$$

Within the deposit, zinc and copper material has been modelled and reported. The zinc Mineral Resource is reported above a 3% Zn block cut-off grade. The copper Mineral Resource is reported above a cut-off grade of 0.7% Cu and below a cut-off grade of 3% Zn. Meaning, material was classed as zinc material if the material had a zinc grade greater than 3% Zn, if the material was less than 3% Zn (and greater than 0.7% Cu, the material was classed as copper material.

Material of less than 3% Zn and less than 0.7% Cu, but above 1.5g/t gold equivalent was classed as gold material.

All Mineral Resources have been reported above 10,240mRL.

Due to the variable weathering profile within the deposit, oxide material may contain transitional and primary/sulphide material.

Table 87 Golden Grove Gossan Hill gold Mineral Resource

				Grades				
	Mineralisation type	**Classification**	**Mt**	**Au ppm**	**Ag ppm**	**Cu %**	**Pb %**	**Zn %**
Gold/Silver	Oxide	Indicated	0.52	3.3	105			
	Transitional	Indicated	0.17	2.4	194			
	Sulphide	Indicated	0.08	1.4	81			
Gold/Silver	Oxide	Inferred	0.23	2.2	50			
	Transitional	Inferred	0.06	1.5	113			
	Sulphide	Inferred	0.06	0.4	119			
Zinc	Sulphide	Indicated	0.36	1.6	109	0.3	1.0	10.5
Zinc	Sulphide	Inferred	0.12	0.4	59	0.1	0.4	7.1
Copper	Sulphide	Indicated	0.17	0.7	20	1.5	0.0	0.3
Copper	Sulphide	Inferred	0.26	0.2	5	1.3	0.0	0.0

7.2.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Rob Oakley, confirm that I am the Competent Person for the Golden Grove open pit Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Golden Grove open pit Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Golden Grove open pit Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Golden Grove open pit Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Rob Oakley - 27/11/13

Bradley Bornshin (Witness)

7.3 Mineral Resource JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Golden Grove open pit Mineral Resources.

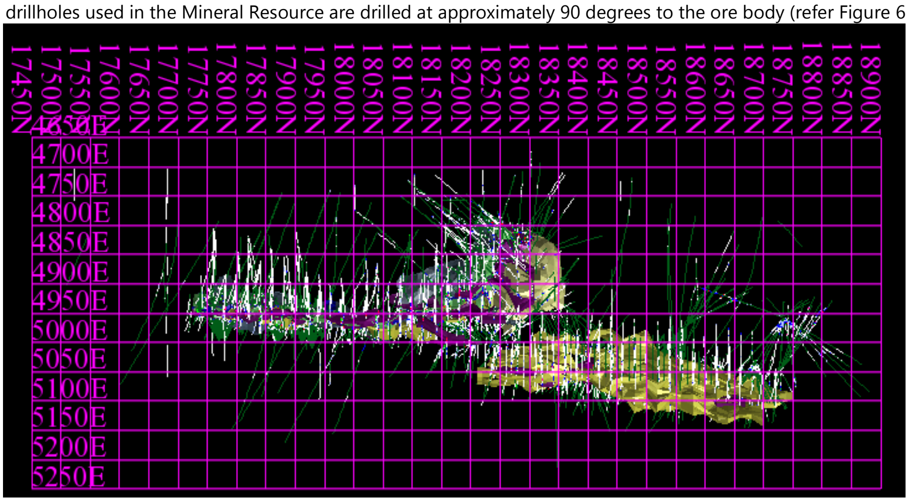
Table 88 Checklist of assessment and reporting criteria for Golden Grove underground Mineral Resource

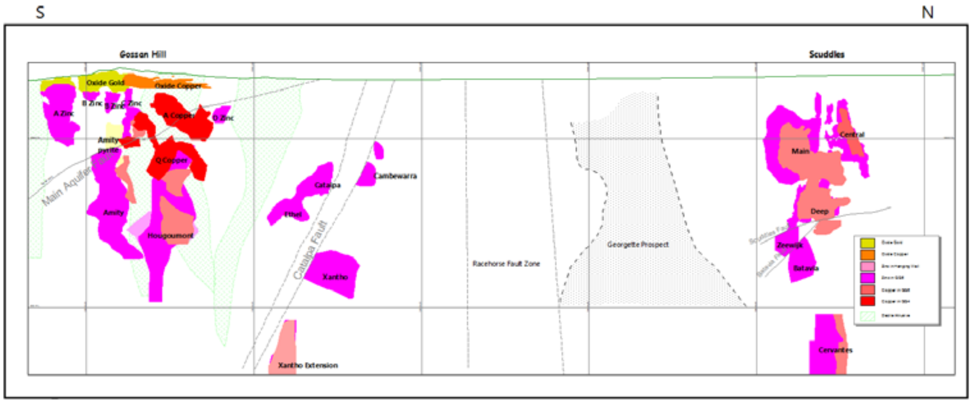
Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	▪ Aircore drilling samples captured in a bag attached to the cyclone, samples were collected/split using a spear (40mm or 50mm PVC pipe). ▪ Grade control RC 2m of sample is captured in the drill rigs cyclone, the sample is split through a cone splitter. ▪ DD core was split at geological boundaries. The core was cut in half using a diamond saw. The breakdown of Gossan Hill drilling by year and company is shown in Table 89.
Drilling techniques	▪ Aircore drilling approximately 4 1/2 inch drillhole diameter. ▪ RC drilling approximately 4 1/2 to 5 1/2inch drillhole diameter. ▪ Surface exploration DD: mainly NQ including PQ, HG and triple tube. ▪ Underground DD: BQ, NQ, LTK48 and LTK60.

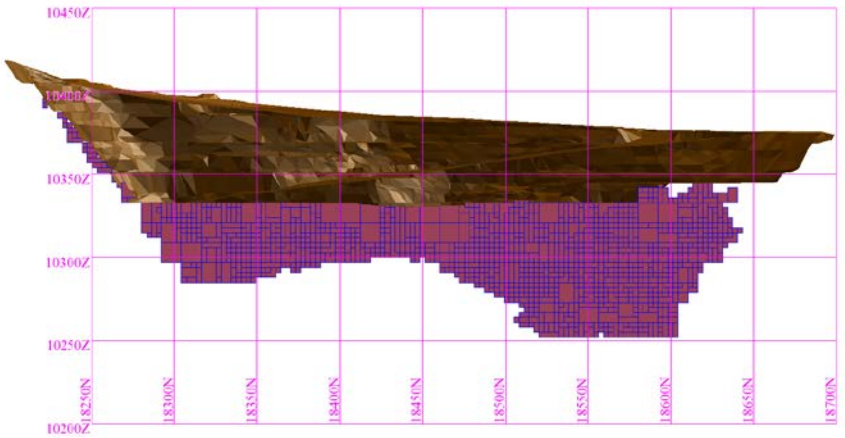
Table 89 Breakdown of Gossan Hill drilling by year, company and drilling type

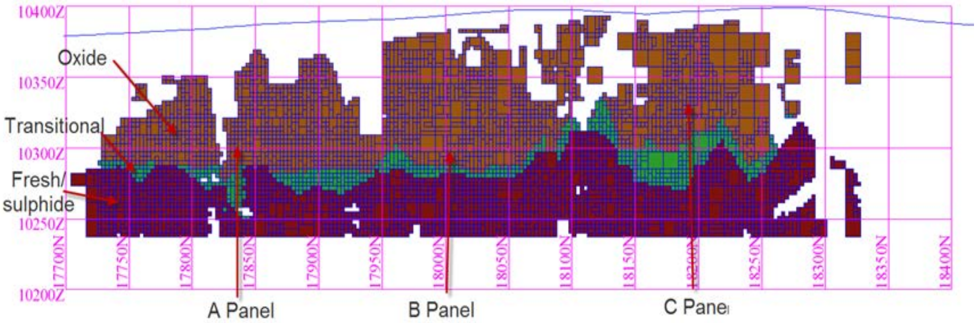
Years	Company	Drill type	Metres
1970 to 1993	MZC	Air Core	5,851
1970 to 1993	MZC	DD	84,228
1970 to 1993	MZC	RC	6,449
1994 to 2001	Normandy	DD	36,372
1994 to 2001	Normandy	RC	22,052
2002 to 2004	Newmont	DD	115,74
2002 to 2004	Newmont	RC	1,950
2005 to 2007	Oxiana	DD	360
2008 to 2009	OZ minerals	DD	7,870
2008 to 2009	OZ minerals	RC	162,68
2010 to 2012	MMG	DD	2,757
2010 to 2012	MMG	RC	11,912

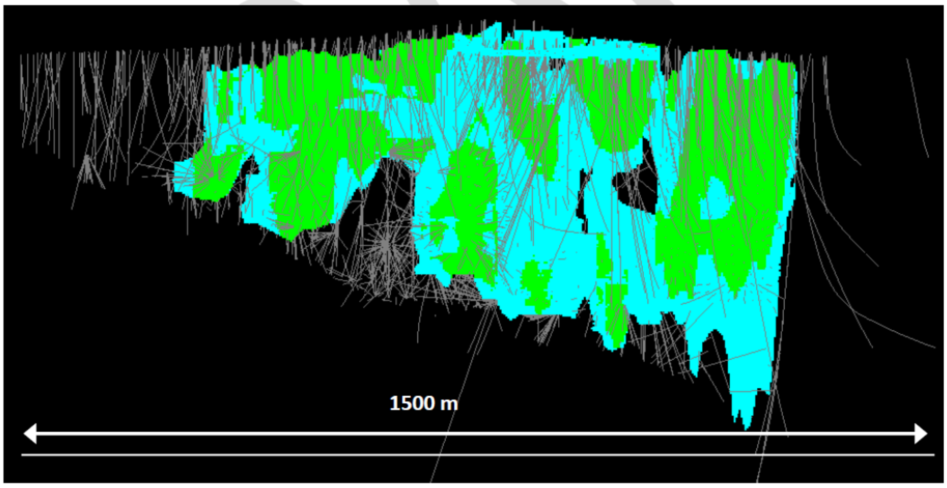
Criteria	Status
Drill sample recovery	▪ Limited recovery data is contained in the database. ▪ All drillholes pre-2000 have no recovery data. ▪ Recovery data for the 1994 RC program is contained in the database.
Logging	▪ Drillholes have been geologically logged. ▪ Geology has not been used to define grade boundaries with exception of the intrusive rocks.
Sub-sampling techniques and sample preparation	▪ Aircore drilling: samples captured in a bag attached to the cyclone, samples were collected/split using a spear (40mm or 50mm PVC pipe). ▪ Early RC (pre-1994): samples captured in a bag attached to the cyclone, samples were collected/split using a spear (40mm or 50mm PVC pipe). ▪ Post-1994 RC samples: captured in a bag attached to the cyclone, samples were split using a triple stage riffle splitter, 5m composites were collected using a spear (40mm or 50mm PVC pipe). ▪ Grade control RC: 2m of sample was captured in the cyclone attached to the drilling rig; the sample is split through a cone splitter. ▪ DD core: split at geological boundaries the core was cut in half using a diamond saw.
Quality of assay data and laboratory tests	▪ Certified standards have been used in drilling programs since 1993. Pre-2003 standards and duplicates were routinely inserted in all drilling samples, data reviewed and any issued identified rectified. These data, however, have not been recorded in the database. ▪ Current grade control programs (post-2012) have certified standards inserted at a rate of 1 in 20, blanks 1 in 50 and duplicate samples 1 in 50. All data is reviewed and any issued identified rectified. Various assay methods have been used: ▪ Resource drillholes, base metals assay method: 4-acid digest followed by ICP MS/ICPOES. ▪ Resource drillholes, gold and silver assay method: fire assay, AAS FA-AAS. ▪ Grade control RC program (April 2012), base metals: 4-acid digest followed by ICP MS/ICPOES.

	- Grade control RC program (October 2012), base metals: XRF. - Between the two grade control campaigns there was an overlap of approximately 6m in the Z direction. All overlapping copper data was checked for bias between the two methods, no bias was found.
Verification of sampling and assaying	- In June, 2012 a twinned drillhole program was undertaken. Good correlation between the historical and 2010 program was found. - Due to multiple phases of drilling, drillholes with significant intersections tend to have been twined and scissored, verifying results. - Underground drillholes (directly below the open pit), drilled from the footwall (generally surface drillholes are drilled from the hangingwall) confirm mineralisation widths, and significant intersections. - Use of both DD and RC indicates there is no significant bias between drilling methods.
Location of data points	- Collars before 1990 were recorded in the database, survey method unknown. - Collars after 1990 were picked up via DGPS or theodolite by qualified mine site surveyors. - All grade control collars (2012 onwards) were picked up via DGPS by qualified mine site surveyors.
Data spacing and distribution	- Drill spacing is 20m along strike and 10m down-dip. - The data spacing and distribution is sufficient to establish geology and grade continuity. - Underground mapping (below the surface deposits) confirm along strike geological and mineralisation continuity. - Grade control drill spacing is on a 10m x 10m grid, with alternative easting's offset by 5m. - The higher resolution grade control drilling has identified short-range, high-grade structures not defined in the Resource drilling or Mineral Resource model.
Orientation of data in relation to geological structure	- The mineralisation generally strikes north-south. - Most drillholes used in the Mineral Resource are drilled at approximately 90 degrees to the ore body (refer Figure 69).  **Figure 69 Plan view of the drillholes intersecting the surface deposits on Gossan Hill**
Sample security	Measures to provide sample security included: - Adequately trained and supervised sampling personnel. - Half-cored samples were placed in numbered and tied calico sample bags. - Bag and sample numbers were entered into the Micromine database. - Samples were couriered to the assay laboratory via truck in plastic bulker containers. - Assay laboratory checks of sample dispatch numbers against submission documents.
Audit and reviews	Audits and reviews on the Gossan Hill Project: - Arnold, 1997. Gossan Hill Project, Oxide Gold and Supergene Copper Mineralisation. Audit of data collection and Geological Interpretation Procedures. A report by Chris Arnold Resource Consultants (DRAFT), April 1997 (CARC/9703) for Murchison Zinc Company Pty Ltd. - Paul Blackney, David Gray, 2012. Data review of the Gossan Hill Oxide Gold Data Laboratory audits and reviews: - Regular laboratory audits have occurred historically with no major concerns identified. - The most recent laboratory audit was conducted in April, 2013. No major concerns were identified.

Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The Gossan Hill operation is covered by Mining Lease M59/195. For further information regarding the mineral tenements and land tenure status of the Golden Grove operations please refer to Table 66.
Exploration done by other parties	▪ Original definition and exploration drilling was performed by Joshua Pitt, of Aztec Exploration, in 1971. ▪ From 1971 until 1992 multiple joint ventures and funding continued definition of the Mineral Resource, with highlights being the Scuddles, A Panel Zn, B Panel Zn, C Panel Zn and Cu discoveries. Parties involved include Amax Exploration, Esso Exploration, Australian Consolidated Minerals and Exxon. Newmont, Normandy, Oxiana, OZ Minerals and MMG have all been involved with the drilling and exploration of the Golden Grove leases since 1991. ▪ A table showing the companies, years, core size and meterage is shown in Table 64 and Table 65.
Geology	▪ The Gossan Hill and Scuddles zinc and copper deposits are located within the same stratigraphic position but are situated approximately 4km apart. ▪ The mineralisation at Gossan Hill is hosted within lithology units GG4 to SC3 whereas the mineralisation at Scuddles is hosted within the lithology unit GG6. ▪ The copper oxide mineralisation is located near surface above the GG4 unit on the northern flank of Gossan Hill. ▪ Gold mineralisation is strata bound: – the footwall is comprised of thinly to thickly bedded siltstone, sandstone and polymictic pebble breccia, – the upper zone contains chert and chemical sediments, and is host to zinc and copper mineralisation, – the hangingwall is comprised of rhyodacite lava and breccia and massive dacite. ▪ The low grade laterite gold deposit on the western flank of Gossan Hill, is confined to the laterite it is unknown whether physical or chemical processes are responsible for the deposit.
Drillhole information	▪ 484 drillholes and associated data are held in the database. No individual hole is material to the Mineral Resource and individual drillhole information is not supplied.
Data aggregation methods	▪ No metal equivalents were used in the copper oxide Mineral Resource estimation. ▪ A metal equivalent has been used for the gold Mineral Resource estimate where: Aueq = (Au g/t + Ag g/t*1.5/80)
Relationship between mineralisation width and intercepts lengths	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Diagrams	**Figure 70 Generalised north-south long-section of the Gossan Hill and Scuddles deposits** 
Balanced reporting	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	▪ All DD, aircore and RC drillhole information was considered for this Mineral Resource estimation. RAB and other open holes were not considered.

Further work	**Copper Resource** - The surface defining the boundary between transition and fresh material is to be updated using the presence of sulphur assays from the grade control drilling as well as geological logging. - As the presence of pyrite in the transition material can be problematic for processing, metallurgical work is ongoing to determine if the transition material should be treated as oxide or primary. **Gold Mineral Resource** - Additional geological information gained through in-pit mapping and grade control drilling will be used to closely monitor the amount of copper within the oxide gold zone, the presence of copper in this material has processing implications.
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	- Historical data has been entered into databases by hand, and some historical data (assay methods) have been lost during transfer from the original Micromine text "DAT" database to an Explorer 3 database. - In 2008, all data was transferred to the Golden Grove Micromine database, which has recently been updated to Geobank™. - Currently, all data is directly input into Micromine database using key field and data validation processes. Collar co-ordinates and dip are validated via comparison of planned data to surveyed data.
Site visits	- The Competent Person worked on-site at Golden Grove from 1997 to 1999, and since 2011. The Competent Person works as a full-time employee on-site.
Geological interpretation	- Geological triangulations and grade shell triangulations are created by the Mine Geologists and Resource Geologists. Geological triangulation interpretations were formed from polygons snapped to drillholes. *Copper Mineral Resource* - Copper mineralisation triangulations were based on a cut-off of 0.2% Cu. - In the oxide zones secondary mineralisation crosses lithological boundaries. - In the transitional and fresh zones the mineralisation is constrained by grade and lithology. - Grades below these cut-offs were included in areas to honour the geology. *Gold Mineral Resource* - Separate triangulations were created for each element contained within the gold deposit. The following domain cut-off's were used: – Copper 0.2% Cu – Gold 0.3g/t Au – Silver 15g/t Ag – Lead 0.4% Pb – Zinc 0.11% Zn
Dimensions	**The copper mineralisation:** - Consists of 2 lenses with a strike length of 100m, joined by a 100m long low grade zone. - The depth of the copper mineralisation is constrained by a large, flat-lying dolerite beneath the pit. Figure 71 shows the extents of the copper Mineral Resource. **Figure 71 Gossan Hill copper pit, looking west. Pit outline as of 30 June, 2013 (brown), the material below the pit-shell are yet to be mined (pink) and represent the 2013 copper Mineral Resource** 

	The gold mineralisation - Is approximately 600m long. - All reported Mineral Resources were reported above the 10200mRL Figure 72 shows the extents of the gold mineralisation. All 3 panels in Figure 72 (A, B, and C) are approximately 150m x 150m. **Figure 72 Long-section looking west of the 2013 gold Mineral Resource, blue line is the current surface** 
Estimation and modelling techniques	- The drillhole data was composited to 1m down-hole intervals. 1m down-hole compositing ensured good resolution across domain boundaries and has honoured original sample lengths (>50% of original sample lengths are one metre). - Data distributions for gold, copper and iron within the mineralised domains all exhibit reasonably well constrained histograms, with a limited range in values, and limited evidence for domain mixing. Both lead and zinc (and possibly silver) have mixed distributions indicating that, for future Resource estimates, additional domaining should be investigated. - Experimental variograms were calculated and variogram models were interpreted. Variogram parameters from the July 2012 estimate were used for bulk density estimation. Variogram models were reasonable for the majority of metals within zones AB, C and GG4 with clearly defined nugget values and well defined structure. - Grade caps were established by investigating the univariate statistics, histograms, log-probability plots of the composited sample data per domain and consideration of the capped and uncapped mean. - Grade caps were applied to reduce the influence of high-grade outliers during block grade estimation. If the grade of a sample exceeded this value, the grade was reset to the grade-cap value. - Grade caps were not applied to all metals in all domains and no grade cap was applied to iron (a lower cut grade of 0.2% was applied to iron). - The block model has a parent block size of 6mE x 12mN x 12mRL. The parent blocks were allowed to sub-cell to 1.5mE x 3mN x 3mRL to more accurately represent the domain geometries and volumes. - Grades for gold, silver, lead, copper, zinc and iron and density values were estimated into parent blocks of a domain coded block model using Ordinary Kriging. Each domain was estimated separately and parent blocks were discretised 3 x 6 x 6 (X, Y, Z). - Block estimates were controlled by the original parent block dimension (i.e. parent cell estimation). - Kriging efficiency and slope of regression values were calculated during block grade estimation which was used to assist in classification of the Mineral Resource. - Estimation parameters were based on the modelled grade continuity, the geological continuity and the average spatial distribution of data. The first pass search radius for mineralised domains was set to two thirds of the variogram range. However where metals had shorter ranges of continuity, the first search radius was set to the variogram range. - Most blocks were estimated in the first search pass. The second search pass radii was set to twice the first and the third search radii were set to ensure that remaining blocks within the mineralised domain were assigned a grade. - Within each mineralised domain 8 to 12 samples were required for a single block estimate with a maximum of 24 to 28 samples. Estimates were limited to a maximum of five samples per drillhole. - All domain boundaries with waste material were treated as hard grade boundaries. - Block densities were estimated below the base of weathering, with the exception of intrusives which were hard coded. Average density values were assigned to the material above the base of weathering.
Moisture	- All tonnages in the Mineral Resource are reported as dry tonnes.
Cut-off parameters	- The copper Mineral Resource is reported above a 0.7% Cu cut-off. - The gold Mineral Resource is reported above a 1.5g/t Aueq cut-off where: Aueq = (Au g/t + Ag g/t*1.5/80)

Mining Factors or assumptions	▪ The copper Mineral Resource is reported within the final, long-term pit-shell. ▪ The gold Mineral Resource is reported above 10,240mRL. ▪ No other mining factors or assumptions have been taken into account.
Metallurgical factors or assumptions	▪ No metallurgical factors or assumptions have been applied to the Mineral Resource.
Environmental factors or assumptions	▪ No environmental factors or assumptions have been applied to the Mineral Resource.
Bulk Density	▪ Density data is available for the majority (84%) of the assayed intervals. ▪ Density data is obtained from wet/dry weight method and down-hole gamma methods. ▪ The density may be overestimated for the oxide material as no sealing techniques have been used for the core. ▪ Density values was assigned to drillhole intervals (using an algorithm) where: – density data was missing, or – mineralised intervals had a measured density value of less than 2.8 t/m^3 or greater than 4.2 t/m^3, or – material was above the base of weathering. ▪ The density algorithm used for assigning density values is as follows: $SG = (100/(35.294-(Zn \times 0.202)-(Cu \times 0.253)-(Pb \times 0.321)-(Fe \times 0.223)))$ ▪ The density data (both measured and derived from a density algorithm) was used for block density estimation below the base of weathering, with the exception of intrusives, for which average density values were assigned.
Classification	▪ Classification of the Mineral Resource was based on confidence in the assayed grade, geological continuity and efficiency/slope of regression of the kriged estimate. The Mineral Resource has been classified and reported with consideration of the guidelines of the JORC Code (JORC, 2012). ▪ Geological confidence is supported by underground exposures including geological mapping and drillhole data. ▪ The Indicated Mineral Resources are located where drilling is closely spaced. Inferred Mineral Resources are outbound of the Indicated Mineral Resources where drilling density is sparser (Figure 73). There is no in-situ Measured Mineral Resource material. ▪ **Figure 73 Long-section view (looking east) of the estimated gold deposit block model showing Mineral Resource classification categories (green = Indicated, blue = Inferred)** 
Audits or reviews	▪ Internal audits were conducted in 2012 and 2013. – Golden Grove 2012 Mineral Resource Checklist, conducted by Jared Broome, Anna Lewin. No material issues were identified. – Golden Grove 2013 Mineral Resource Checklist, conducted by Jared Broome. No material issues were identified.
Discussion of relative accuracy / confidence	▪ Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support. ▪ Reconciliation has not been completed as mined material has been stockpiled, with very little of it processed through the mill. Once enough mill data is available reconciliation will occur.

7.4 Ore Reserves – Golden Grove Open Pit

7.4.1 Results

The estimated Ore Reserves for the Open Pit operation at Golden Grove is summarised in Table 90.

Table 90 Total (in-pit and stockpiled) Ore Reserves as at 30 June 2013

	Oxide Ore Reserves		Transitional and Sulphide Ore Reserves		Total* Ore Reserves		Contained Metal†
	Tonnes (Mt)	Grade (%Cu)	Tonnes (Mt)	Grade (%Cu)	Tonnes (Mt)	Grade (%Cu)	Copper ('000t)
Proved	0.8	2.4	–	–	0.8	2.4	19
Probable	0.9	2.7	0.6	2.5	1.6	2.7	41
Total*	1.8	2.6	0.6	2.5	2.4	2.6	60

**Totals may differ due to rounding; †Contained metal does not imply recoverable metal*

Table 91 In-pit Ore Reserves as at 30 June 2013

	Oxide Ore Reserves		Transitional and Sulphide Ore Reserves		Total* Ore Reserves		Contained Metal†
	Tonnes (Mt)	Grade (%Cu)	Tonnes (Mt)	Grade (%Cu)	Tonnes (Mt)	Grade (%Cu)	Copper ('000t)
Proved	–	–	–	–	–	–	–
Probable	0.9	2.7	0.6	2.5	1.6	2.7	41
Total*	0.9	2.7	0.6	2.5	1.6	2.7	41

**Totals may differ due to rounding; †Contained metal does not imply recoverable metal*

Stockpiles exist on site for high chlorine ore, low chlorine ore and low grade ore (Table 92). A cut-off of 150ppm chlorine is used to define low and high chlorine ore. Stockpiles are classed as Proved Ore Reserves.

Table 92 Ore stockpiles (oxide) as at 30 June 2013

Stockpile Type	Tonnes (Mt)	Grade (%Cu)
ROM ore High Chlorine >1.5% Cu	0.30	4.3
ROM ore Low Chlorine > 1.5% Cu	0.02	1.5
Low Grade ore (0.7% to 1.5%)	0.49	1.2

7.4.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Chris Lee, confirm that I am the Competent Person for the Golden Grove open pit operations Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Golden Grove open pit operations Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 749,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 Oct 2013 was $HKD 1.72).

I verify that the Golden Grove open pit operations Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Ore Reserves.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Golden Grove open pit operations Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Chris Lee – 26/11/13

Peter Jasper (Witness)

7.4.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 93.

Table 93 Contributing experts – Golden Grove oxide pit Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Rob Oakley, Senior Mine Geologist, MMG Ltd (Golden Grove)	Geological Mineral Resources
Stephen Ross, Manager Commercial, MMG Ltd (Golden Grove)	Commercial Input
Peter O'Bryan, Consultant, Peter O'Bryan & Associates	Geotechnical
Geoffrey Senior, Group Manager Metallurgy, MMG Ltd (Melbourne)	Metallurgy
Trung Huynh, Plant Metallurgist, MMG Ltd (Golden Grove)	Metallurgy
Gavin Marre, Senior Business Analyst, MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

7.5 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

7.5.1 Pit Design

Overall pit design and progress are illustrated in Figure 74 and Figure 75.

Figure 74 View of pit digital terrain model as at 30 Jun 2013 - looking North to South with ore block mark-up on 10336 bench

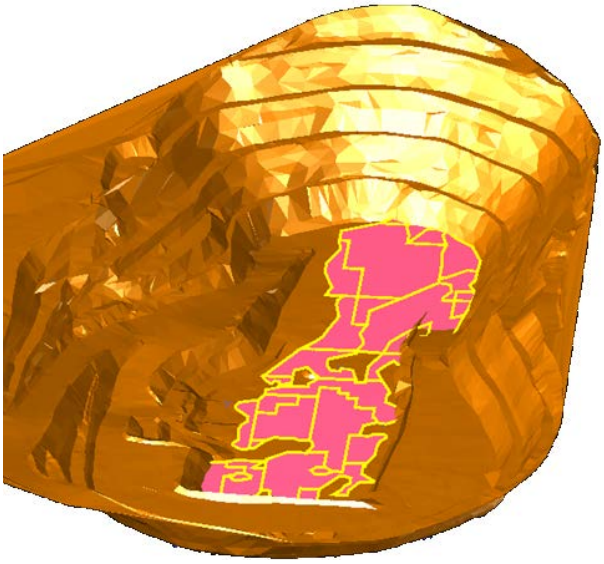
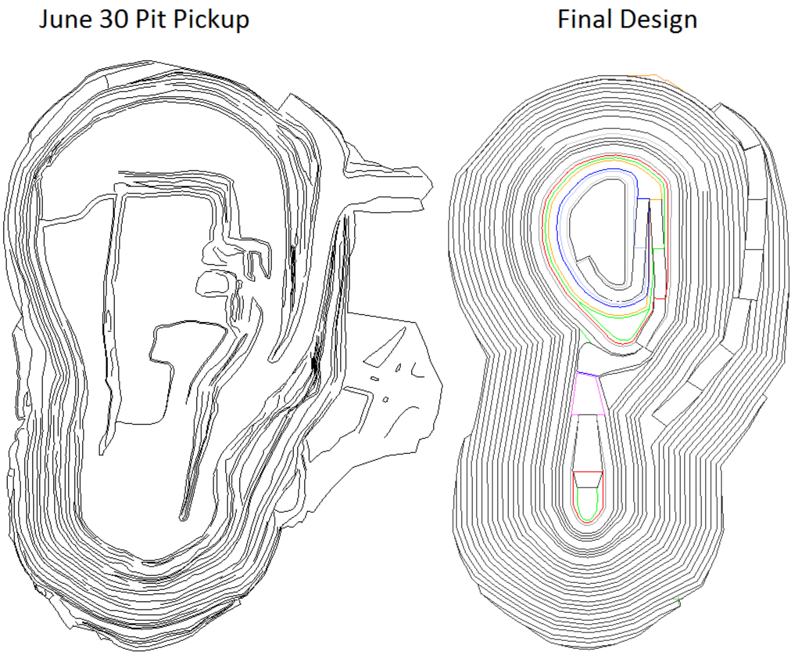


Figure 75 Strings of pit status as at 30 Jun 2013 and Final Pit Design



June 30 Pit Pickup

Final Design

7.5.2 Realised Revenue Factors

The realised revenue is a function of commodity price, exchange rate and revenue related costs such as freight, shipping insurance, treatment and refining charges.

For the short-term (calendar year 2014) copper price assumed at 3.50 US$/lb and an exchange rate of 0.99 US$/A$, the Australian price for copper is 3.54 A$/lb. The calculation of the reduction from commodity price to the realised revenue component of that price is shown in Table 94. This results in a realisation of 2.84 A$/lb of contained copper sold in concentrate.

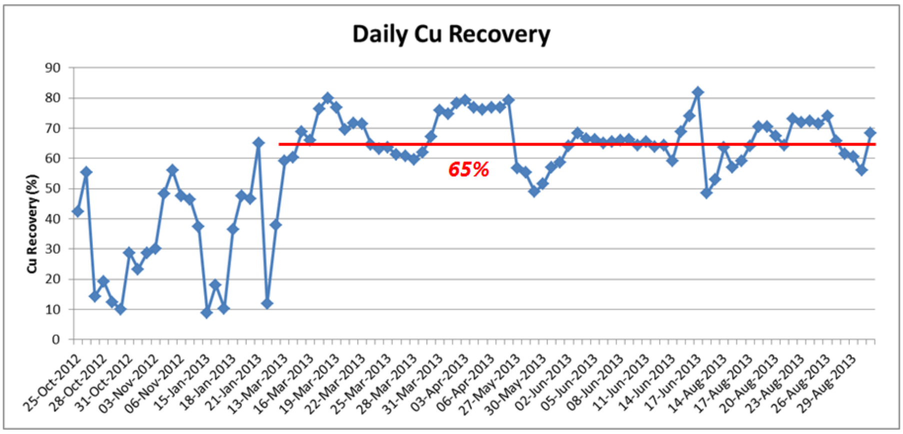
Table 94 Realised revenue calculation

Revenue Calculation		
Copper price	$/lb	$3.54
Copper metal	t	29,200
Conc grade	%Cu	20.50
Tonnes of concentrate		142,439
Wet tonnes	16% moist	165,229
Royalty	%Cu value	5
Freight to Port	$50/dt	$7,121,951
Shipping Port Ins	$31.50US/wt	$5,966,612
Treatment charges	$105US/wt	$6,563,274
payable %	96	28,032
Refining charge	$0.07US/lb	$4,550,481
Actual copper	metal lbs	64,356,800
	value	***$227,823,072***
Deductions	metal lbs	2,574,272
	$	$9,112,923
Charges	$	$24,202,318
Royalty	$	$11,391,154
Total deductions	***value***	***$44,706,395***
As a percent of original value		***19.6%***

7.5.3 Processing (Metallurgical) Recovery Factors

For the copper oxide ore, a recovery of 65% has been used - consistent with recent milling campaigns, as shown in Figure 76. This compares adversely with the expected 80% in the 2011 Feasibility study.

Figure 76 Copper oxide actual milling campaign recoveries to end August 2013



Transition and sulphide ore recovery has been estimated bench by bench using the formula below, as supplied by Geoff Senior (Group Manager Metallurgy, MMG). The recovery increases with head grade using the following formula.

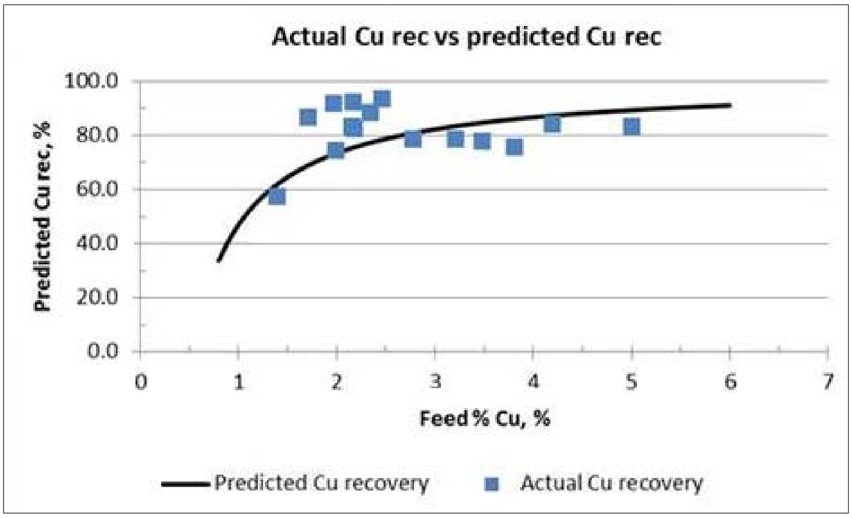
Cu rec (%) = 100 - 53/feed % Cu

The recovery for the transition and sulphide ore is a significant downgrade on that used in the feasibility where 87% had been used based on limited test work.

Test work results to date and the recovery algorithm are shown in Figure 77. A series of recent test results in the 1.7% to 2.5%Cu grade range has shown a better recovery that that using the recovery algorithm, however this is thought to be possibly a result of deeper more primary ore being tested.

The use of the recovery algorithm is therefore thought to be reasonable for the transitional material and probably conservative for the primary material.

Figure 77 Sulphide Ore Recovery Test Work Results



Using the recovery algorithm, the recovery for the remaining transitional and sulphide material in the pit has been estimated by bench as per Table 95.

Table 95 Estimate of transition and sulphide material recovery by bench

Bench	Transitional & Primary Sulphide Material at a 1.3% Cu Cut-Off Grade			Calculated Recovery
	Volume	Tonnes	Grade (%Cu)	
10309	10,400	30,800	3.53	85
10303	26,200	69,500	2.64	80
10297	46,700	123,000	2.87	81
10291	35,300	101,400	2.66	80
10285	31,500	95,400	2.66	80
10279	21,200	61,500	2.12	75
10273	17,800	51,300	2.09	75
10267	15,800	44,400	1.98	73
10261	14,400	40,700	2.00	74
Total	**219,400**	**618,134**	**2.55**	**79**

7.5.4 Processing (Metallurgical) Deleterious Elements

During processing of the first ore from the pit it became apparent that there was a major issue with the presence of chlorine as a deleterious element.

Chlorine is a deleterious element that adversely affects the marketability of the concentrate and hence the financial return. The impact of the chlorine is difficult to definitely define, as the nature of upgrade into the concentrate is not well understood - actual mill data showing the upgrade into concentrate from the feed grade is shown in Figure 78. Further complicating the issue is that there are multiple sources of chlorine - in addition to its presence in the ore it is also present in varying concentrations in the process water used.

Chlorine was not defined in the original Mineral Resource model and therefore cannot be defined in the Ore Reserves. From grade control drilling for which chlorine is being tested, it has been possible to define and separate low chlorine and high chlorine ore (see Table 96). These produce concentrates of varying chlorine grade that depending on the customer may be able to be blended with each other.

In calculating the cut-off grades, no allowance has been made for chlorine due to the absence of chlorine estimation values in the Mineral Resource model.

The latest information from marketing department is that the lower chlorine concentrates (<2000 ppm) can be sold at about 32 US$/dry metric tonne (dmt) penalty, with a penalty increase of around 3.5 US$/dmt/100ppm increase in chlorine grade. A 7000 ppm chlorine concentrate will therefore be sold at a penalty of around 210 US$/dmt. (*Source: Tim Roberts, Marketing Department, e-mail 25 September 2013*). By 30 June 2013 the majority of the expected high chlorine ore had been mined.

Figure 78 Chlorine upgrade from ore feed to concentrate - daily data from mill campaigns 3, 4 and 5

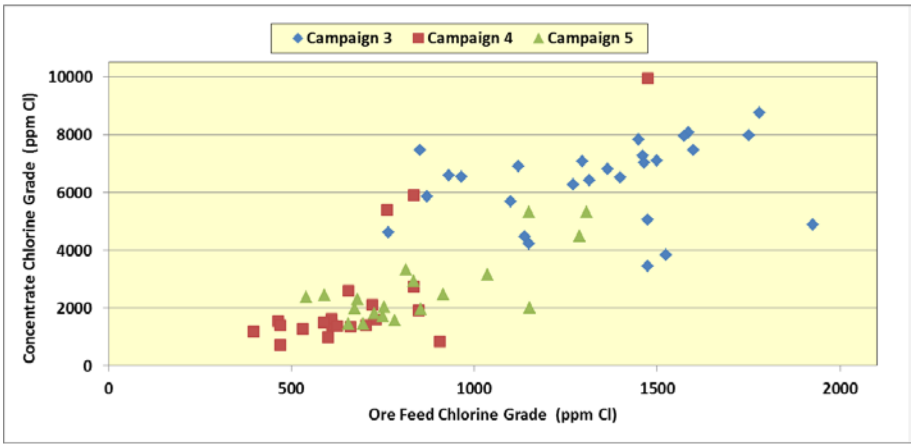


Table 96 Grade control drilled ore classified by chlorine as at 30 June 2013

	Tonnes	Grade (%Cu)
Low Chlorine (<150 ppm)	1,441,000	2.62
High Chlorine (>150 ppm)	176,000	3.50
Total	**1,617,000**	**2.71**

(Note: includes ore already mined and ore already milled so no longer in Ore Reserves)

For the ore that has been delineated by grade control drilling as at 30 June 2013, the estimated penalties to be paid as a result of chlorine are given in Table 97. This suggests an overall penalty equivalent ot A$2.55/dry tonne of ore.

Table 97 Estimate of penalties to be paid due to the presence of chlorine in concentrate

	Penalty Rate	Concentrate Tonnes	Penalty Amount
High Chlorine Concentrate	210 $/dmt	8,842	$ 1,860,000
Low Chlorine Concentrate	32 $/dmt	70,719	$ 2,260,000
Total		79,561	$ 4,120,000
Average Penalty per Tonne of Ore			**$ 2.55/ t-ore**

7.5.5 Mining Costs

The open pit is being mined by a contractor using a schedule of rates. Consequently mining costs are well defined. Expected costs by bench are given in Table 98.

Table 98 Contract mining costs by bench

Bench	Load & Haul (A$/bcm)	Drill & Blast (A$/bcm)	Fixed (A$/bcm)
10345	7.36	2.02	3.75
10339	7.55	2.02	3.75
10333	7.62	2.02	3.75
10327	7.74	2.02	3.75
10321	7.75	2.23	3.75
10315	7.78	2.73	3.75
10309	8.09	2.73	3.75
10303	8.50	2.73	3.75
10297	8.87	3.06	3.75
10291	8.99	3.40	3.75
10285	9.18	3.5	3.75
10279	9.47	3.61	3.75
10273	9.89	3.61	3.75
10267	10.41	4.18	3.75
10261	10.82	4.18	3.75

7.5.6 Cut-Off Grade

Variable cut-off grades have been used for each ore type. Transition and sulphide ores are expected to achieve lower recoveries than used in the Feasibility study. The cut-off grades will need to be reviewed once actual data is available for the transition and sulphide ore; this includes both costs and revenue factors.

The cut-off grade calculations are shown in Table 99 and Table 100 (which specifically shows the effects of the recovery algorithm on the calculation of the cut-off grade).

In practice, the mine is able to deliver more ore to the mill than is required to keep the mill operating. Hence a value stockpiling strategy is used with the oxide ore, the transition and sulphide ore split into a direct feed ROM grade ore and low stockpiled ore grade ore. Direct feed ROM ore grade cut-off for transitional and sulphide material is expected to be set at 1.6%Cu – however this will be better defined when grade control data is available.

Table 99 Cut-off grade calculation

		Oxide	Sulphide
Recovery	%	65	58
Milling unit cost	$/t milled	$36.00	$36.00
Administration unit cost	$/t milled	$10.00	$10.00
		$46.00	$46.00
Revenue $/t of cu metal	2.84	6,261	6,261
Recovered value per tonne		4,070	3,631
$/ per cent of copper		$40.70	$36.31
Cut-off grade	Cu	1.13	1.27

Table 100 Illustration of sulphide recovery algorithm effects on cut-off grade calculation

Grade (%Cu)	Calculated recovery (%)	Value $/t-ore (Revenue - Cost)	Processing costs A$/t-ore	Revenue A$/t-ore	Realised revenue A$/t-Cu
1.00	47.0	-16.6	46	29.43	6,261
1.05	49.5	-13.4	46	32.56	6,261
1.10	51.8	-10.3	46	35.69	6,261
1.15	53.9	-7.2	46	38.82	6,261
1.20	55.8	-4.1	46	41.95	6,261
1.25	57.6	-0.9	46	45.08	6,261
1.26	58.1	-	46	46.00	6,261
1.30	59.2	2.2	46	48.21	6,261
1.35	60.7	5.3	46	51.34	6,261
1.40	62.1	8.5	46	54.47	6,261
1.45	63.4	11.6	46	57.60	6,261

7.5.7 Ore Reserves Economics

The economics of the remaining pit have been evaluated in summary as shown in Table 101 and Table 102.

It can be seen that there is a large positive cash-flow for the remaining pit. Realised revenue from copper would need to drop below $1.52 A$/lb before the remaining pit became uneconomic. Alternatively the oxide recovery would need to drop to 12%. Sulphide recovery could drop to zero and the project is still cash-flow positive.

Table 101 Recovered copper by bench calculation for remaining pit as at 30 June 2013

Bench	Oxide Tonnes (t)	Oxide Grade (%Cu)	Oxide Recovery (%)	Transitional & Sulphide Tonnes (t)	Transitional & Sulphide Grade (%Cu)	Transitional & Sulphide Recovery (%)	Recovered Copper (t)
10345	400	1.14	65%	-	-	-	3
10339	4,500	1.84	65%	-	-	-	54
10333	10,000	1.92	65%	-	-	-	125
10327	215,300	2.91	65%	-	-	-	4,068
10321	215,000	2.85	65%	-	-	-	3,980
10315	190,000	2.69	65%	-	-	-	3,326
10309	169,000	2.59	65%	30,800	3.53	85	3,762
10303	104,000	2.61	65%	69,500	2.64	80	3,229
10297	31,200	2.55	65%	123,000	2.87	82	3,396
10291	3,900	1.65	65%	101,400	2.66	80	2,205
10285	1,600	1.48	65%	95,400	2.66	80	2,043
10279	300	1.39	65%	61,500	2.12	75	983
10273	0	0.00	65%	51,300	2.09	75	798
10267	0	0.00	65%	44,400	1.98	73	645
10261	0	0.00	65%	40,700	2.00	73	598

Table 102 Cash flow by bench calculation for remaining pit as at 30 June 2013

Bench	Mining Costs M$	G&A Costs $10/t M$	Mill Costs $36/t M$	Total Costs M$	Realised Revenue M$	Net Cash Flow M$	Cumulative Cash Flow M$
10345	0.05	0.00	0.02	0.07	0.02	-0.05	-0.05
10339	1.06	0.05	0.16	1.26	0.34	-0.9	-1.0
10333	1.59	0.10	0.36	2.06	0.78	-1.3	-2.2
10327	3.38	2.15	7.75	13.28	25.48	12.2	9.9
10321	3.32	2.15	7.74	13.21	24.92	11.7	21.7
10315	3.17	1.90	6.84	11.91	20.82	8.9	30.6
10309	2.73	2.00	7.19	11.91	23.55	11.6	42.2
10303	2.53	1.74	6.25	10.51	20.23	9.7	51.9
10297	2.27	1.54	5.55	9.36	21.14	11.8	63.7
10291	1.55	1.05	3.79	6.39	13.79	7.4	71.1
10285	1.35	0.97	3.49	5.82	12.79	7.0	78.1
10279	1.03	0.62	2.22	3.87	6.15	2.3	80.3
10273	0.74	0.51	1.85	3.11	5.03	1.9	82.3
10267	0.65	0.44	1.60	2.70	4.03	1.3	83.6
10261	0.54	0.41	1.47	2.42	3.77	1.4	**84.9**

7.5.8 Geotechnical Parameters

The open pit design incorporates geotechnical parameters that have been provided by an external consultant: Peter O'Bryan of Peter O'Bryan & Associates.

This consultant routinely is engaged every six months to confirm actual performance versus expected.

To date the pit has performed well geotechnically with some very minor issues encountered. As the pit is moving into more competent material with depth no significant geotechnical issues are expected.

The geotechnical parameters used in the pit design are as per the Feasibility Study geotechnical report "Golden Grove Copper Oxide Pit Geotechnical Assessment", (report 10008) dated July 2010.

The stability of the open pit is expected to be governed predominantly by the presence, attitude and shear strength of geological structures within and/or in close proximity (behind) the pit walls. The relatively low strength of the highly weathered rocks through which much of the pit is being mined will also affect the pit wall stability.

The wall rocks have been and are expected continue to be dry due to the influence of existing underlying underground working.

The geotechnical domains for the open pit are fundamentally defined by lithology (sediments and volcanics). Further sub-division is on the basis of extent of weathering. Geotechnical parameters have been modified (slightly steeper angles) for pit end walls which cross the stratigraphy in comparison to the major stratigraphy-parallel walls.

The wall design parameters used in the pit design are as follows:

Eastern Wall:

Face height:	≤ 20m	
Face angle	55°	surface to 10340 mRL
	60°	10340 mRL to 10300 mRL
	65°	10300 mRL to pit floor
Berm width	5m	for the uppermost berm at 10400 or 1038 mRL in the south and 10360 mRL in the north
	7m	at 20m vertical intervals below the uppermost berm
Overall angle	48.5 °	

Western Wall:

Face height:	≤ 20m	
Face angle	55°	surface to 10340 mRL
	60°	10340 mRL to 10320 mRL
	65°	10320 mRL to pit floor
Berm width	5m	at 10360 mRL (and 1038mRL where developed)
	7m	at 20m vertical intervals from 10340 mRL
Overall angle	48.1 °	exclusive of ramps 10390 to 10260mRL

Northern End Wall:

Face height:	≤ 20m	
Face angle	55°	surface (10370 MRL ± 3m) to 10340 mRL
	60°	10340 mRL to 10300 mRL
	65°	10300 mRL to pit floor
Berm width	5m	at 10360 mRL
	7m	at 20m vertical intervals from 10340 mRL
Overall angle	50.2 °	exclusive of ramps 10370 to 10260mRL

Southern End Wall:

Face height:	≤ 20m	
Face angle	55º	surface (variable elevation) to 10400 mRL
	60º	10400 mRL to 10360 mRL
	65º	10360 mRL to pit floor
Berm width	5m	at 10400 mRL
	7m	at ≤20m vertical intervals from 10410 mRL
Overall angle 52.4 º	exclusive of ramps 10415 to 10280mRL	

7.5.9 Mining Factors and Assumptions

Minimum Mining Width

The orebody has a northern (80m wide X 150m long) and southern lode (50m wide X 100m long) joined by a section of lower grade material. The ore bodies are relatively wide and have sharp visible contacts.

No minimum mining widths have been applied as the orebody is physically larger than what would be required as a minimum mining width. It should be noted that as the pit deepens and moves from supergene oxide into primary ore that the orebody tonnes per vertical metre and grade drop off. So whilst the orebody maybe 100m wide now it will not be so in another 50 vertical metres never the less it still remains at a width that can be comfortably mined using the existing equipment on site.

Pit Design

The pit design is based on shells selected using Whittle Four-X software to run pit optimisations. This work was undertaken by AMC Consultants as part of the Feasibility Study in 2011. Whittle uses the Lerchs-Grossman algorithm to determine a maximum depth for an economic pit and then a set of nested shells that evaluate variations in costs or revenue. The revenue factor 1.0 shell (optimum case) was selected as the basis of the pit design as there was little difference between the undiscounted and discounted cash flows and little variation in tonnes and grades in a number of nested shells either side of this revenue shell.

Pit design was carried out by AMC Consultants in Datamine and was based on 10m batter heights (generally mined at 2.5m height – except in the overburden where it is mined in 5m flitches). Batter wall angles and berm widths were as per geotechnical consultant recommendations and vary according to expected rock mass conditions.

Mining Dilution

As reported in the 2011 Feasibility Study, mining dilution and ore loss was simulated by regularisation of the block model and determined to be 12% and 8%, respectively. Dilution was assumed to have no grade.

No additional operational dilution or loss is applied. The impact of the modelling process is assumed to compensate for any additional impact from either dilution or ore loss.

The nature of the orebody and the mining practices result in a low probability of dilution and loss. Visual boundaries, an almost vertical dip to the orebody and the common presence of a low grade halo at the edges of the orebody make the orebody simple to mine and the very favourable geometry reduces the likelihood of dilution and loss. Every blast is monitored for movement and ore mark outs adjusted accordingly. No ore is mined on nightshift and all ore is mined under geological supervision. Angled reverse circulation drilling is carried out on a tight pattern to define ore for 42 vertical metres at a time.

The positive reconciliation of the Grade Control model to the Mineral Resource model indicates that there has been no need to apply any further dilution or loss adjustments.

Reconciliation

Grade Control drilling to date has consistently indicated a positive reconciliation to the 2011 Mineral Resource model as illustrated with the results for the 10345 to 10315 mRL section of the pit shown in Table 103. No adjustments have been made for this positive reconciliation.

Table 103 Grade control to Mineral Resource model reconciliation (10345 to 10315 mRL)

For 345rl to 315rl	t	%Cu	Metal
Grade Control	758,632	2.72	20,612
Reserve	654,815	2.81	18,424
Variance	16%	-3%	12%

The new updated model built by Optiro Pty Ltd during 2012 has improved reconciliation of the Resource Model against the grade control model, but is still under reporting the tonnes and grade, suggesting that the Ore Reserves is conservative.

The Competent Persons expectations are that the Ore Reserves as stated may underestimate the tonnes of the ore and hence the contained metal. It is expected this difference in metal to be less than 10%.

There is currently no reconciliation available for as mined against as milled. This reconciliation is hindered by complications associated with significant stockpiling of ore.

7.5.10 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 104. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 104 JORC Code Ore Reserves Assessment and Reporting Criteria for Golden Grove Open Pit Operations 2013 Ore Reserves

Assessment Criteria	Risk Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	Low	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The Mineral Resources include the oxide and transitional copper mineralisation at Gossan Hill and limited sulphide mineralisation directly beneath the oxide deposit. The Mineral Resource model used as the basis for 2013 Ore Reserves estimation was the 2012 Mineral Resource block model prepared by Optiro Pty Ltd – being an update of the 2011 model to include the first rounds of grade control data. This was the same model used to declare the Mineral Resources included this statement. The Ore Reserves includes ore on stockpiles.
Classification	Low	The in-pit Ore Reserves are classed as Probable only, in line with Mineral Resources classification of Indicated. Stockpiled ore has been classified as Proved, in line with Mineral Resources classification of Measured. Inferred Mineral Resources are not included in the Ore Reserves. It is noted that Inferred Mineral Resources amounts to less than 1% of the Mineral Resource within the pit.
Site visits	-	The Competent Person (Chris Lee) is based at the Golden Grove site in his capacity as Manager Mining – Open Pit.
Study status	Low	The Golden Grove Open Pit is currently an operating mine. Mining commenced on the 9th January 2012 and is scheduled for completion in July 2014. As at 30 June 2013, 71.6% of the ultimate pit volume had been mined. A Feasibility Study was completed in February 2011. No change to pit design has been undertaken given the advanced state of mining the pit. The Ore Reserves quoted are the results of an internal MMG re-assessment of the Mineral Resources contained within the designed pit for changes in the modifying factors since the February 2011 Feasibility Study and using the end of June 2103 topographic surface.
Cut-off parameters	Medium	See Section 7.5.6 for details. Oxide cut-off grade is well established, but there will be a need to review cut-off grade for transition and sulphide ore based on actual operating data once available.

Mining factors or assumptions	Low	See Section 7.5.8 for details. See Section 7.5.9 for details relating to minimum mining width, pit design, dilution and loss, and reconciliation.
Metallurgical factors or assumptions	Medium	See Section 7.5.3 for details.
Environmental	Low	A detailed analysis of waste mined has been undertaken since the pit commenced. Every 50,000BCM of waste, a sample is collected and despatched for PAF test work. This work has not identified any PAF material. Whilst this is an after the event method, proactively grade control holes have been assayed for S and it is intended to use these assays to define PAF in the future. A PAF storage facility has been constructed adjoining the ROM pad and is ready to receive any PAF waste. The waste dump has been progressively rehabilitated and regrowth on the outer batters is already evident. Construction involved removing vegetation and topsoil, build the outer walls first, install a perimeter drain and sediment trap, profile the outer walls to a gentle slope, push the topsoil and vegetation back over the slope, and contour rip. As waste has been tipped into the middle the remaining vegetation and topsoil has been reclaimed and placed directly onto the top of the dump. Outstanding work to do includes building an abandonment bund, completing the perimeter fence, and seeding and fertilising the dump. Open pit operations are conducted under Operating Licence L5175/1988/9.
Infrastructure	Low	No additional site infrastructure is required to realise the open pit Ore Reserves. All necessary infrastructures were established prior to mining and or milling of the open pit commencing, as a result of the underground mine operating at this site since 1980.
Costs	Medium	The open pit is being mined by a contractor using a schedule of rates. Consequently mining costs are very well defined – see Section 7.5.5 for details. Milling and administration costs have been supplied by site's commercial department. A cost of $36/t of ore for milling and $10/t of ore for administration has been used. It is noted that site accounting practices do not allow for separation of activity based costing of different milling concentrate products.
Revenue factors	Low	Revenue factors were based on ultra-short-term pricing as discussed in Section 2.1 (CY14 prices and exchange rates). As the pit will be completed mid 2014 this is appropriate. The remaining Ore Reserves are highly profitable, copper price would need to drop more than 45% to make it cash flow negative
Market assessment	High	See Section 2.2 for details. There is a limited market for the oxide product requiring careful liaison between site, marketing, and customers. The high chlorine content results in a cost penalty as discussed in Section 7.5.4.
Economics	Low	See Section 7.5.7 for details.
Social	Low	No known issues. MMG Golden Grove is located within the Shire of Yalgoo in the Murchison Region of Western Australia. The nearest community to Golden Grove is the Yalgoo Township, which is situated approximately 56km to the north of the site, with a population of approximately 100. The key stakeholders include the local government and community, pastoralists, employees and the Geraldton Port Authority. MMG Golden Grove has maintained good partnership with neighbouring pastoral, traditional owner groups through various programs such as; Bayalgu Program, CHMA Badimia People, Life of Mine investment Agreement Shire of Yalgoo and GPA AQMP agreement. MMG Golden Grove is located in an area that is under claim by two Indigenous Native Title claimant groups.
Audit or Reviews	Medium	No external audits or reviews have been undertaken. An informal internal review was undertaken by Julian Poniewierski (Group Manager – Technical Governance) during compilation of this report. No internal audits have been undertaken.
Discussion of relative accuracy/ confidence		A qualitative risk assessment of each discussed item is included with each individual item in the second column of this table. Whilst there are a number of parameters for which there is low confidence, the impact of this uncertainty on the remaining Ore Reserves is such that the likelihood of destroying the robust economics of the remaining Ore Reserves is extremely low.

Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Low	Golden Grove is located within the Yalgoo biogeographic subregion, which is characterised by open woodlands and scrubs on earth or sandy earth plains The area surrounding Golden Grove is of low to moderate relief with long ranges separated by extensive plains. Elevation is generally around 350m above sea level with the highest point in the region being Minjar Hill at approximately 380m above sea level.
Climate	Low	Golden Grove is situated within the Yalgoo bioregion and has a variable climate with characteristics of semi-arid and Mediterranean climates and is prone to long periods of drought. Most rainfall occurs during the winter months, although more occasional major rainfall events, largely associated with tropical cyclone activity off the northwest shelf, occur in the summer months and can result in localised flooding. Average rainfall is 290.9mm annually. Monthly rainfall has seldom exceeded evaporation onsite. The region has relatively mild winter and very warm summer. Greatest risk to open pit operations would be a major rain event from the tail end of a cyclone which potentially would require some dewatering of the pit.
Government Agreements	Low	Mining Proposal ID 29469 covers this project located on tenement M59/195 expiry date May 2032.
Hydrogeological Parameters	Low	Pit is dry as it has an underground mine below it. Maybe have to dewater in a major rain event. Have a high head diesel pump near the pit ready to be deployed and have already run polyline into pit base for such an event.
Waste Storage (Including Tails Storage)	Low	The tailings dam has sufficient capacity out to late 2014 or mid-2015. The open pit will be finished in July 2014. The dam has been designed for additional lifts and it is likely that not all the open pit ore will have been milled prior to the next lift occurring

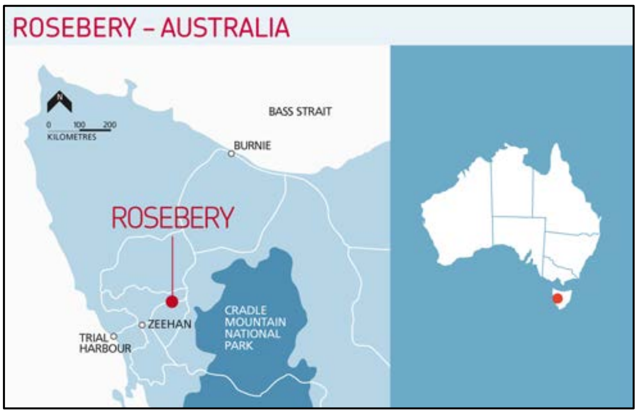
8. ROSEBERY

8.1 Introduction and Setting

MMG Limited holds title to the Rosebery Mine Lease (ML 28M/1993 – 4,906ha) which covers an area that includes the Rosebery, Hercules and South Hercules base and precious metal mines.

Mining Lease 28M/93 is located on the West Coast of Tasmania approximately 120km south of the port city of Burnie. The main access route to Rosebery Township from Burnie is via the B18 and the Murchison Highway (A10), which connect 8km east of Waratah. The mining lease encompasses the township of Rosebery.

Figure 79 Rosebery Mine location



Rosebery is a mechanised long-hole open-stope underground operation with footwall ramp access. The mine currently employs a benching mining method, but has historically also used a cut and fill stoping method. The mine has historically produced approximately 800,000 tonnes of ore per year with plans to increase this to over 900,000 tonnes per year going forward. Mine production in 2012 was the highest achieved for the site at 856,958 tonnes. The ore is processed into separate concentrates for zinc, lead and copper. The mine also produces a gold/silver doré bullion.

8.2 Geological Setting

The Rosebery and Hercules deposits are hosted within the generally north - south trending Cambrian age Mount Read Volcanics in Western Tasmania. The Mount Read Volcanics are comprised of an assemblage of lavas, volcaniclastics and volcanic derived sediments deposited within the Dundas Trough. The Mount Read Volcanics also host the Cambrian aged Hellyer and Que River base metal deposits, the Henty gold deposit, and the Mt. Lyell copper and gold deposits.

In the immediate mine area the host sequence is composed of the following from bottom to top:

(i) Footwall Pyroclastics - >800m of feldspar-phyric pumice breccias
(ii) Crystal lithic volcaniclastic sediments (the ore host) - 35m thick
(iii) Black Slate - the immediate hangingwall - up to 30m thick
(iv) Hangingwall Pyroclastics (epiclastics) – 50m to 200m of feldspar-quartz-phyric volcaniclastics
(v) Mount Black Volcanics (dacitic to andesitic lavas) - >1000m thick

The sulphide mineralisation forms tabular sheets up to 10m or more thick, dipping at 45°E over a strike length of 4km north-south, extending down dip to a depth of more than 1.5km.

The low angle (approximately 45°), east dipping Rosebery Thrust Fault forms the western boundary to this sequence. A similar low angle, east dipping structure, the Mt. Black Thrust Fault, separates the Hangingwall sequence from the Mt. Black Volcanics. Both of these structures are thought to be Devonian in age.

8.3 Mineral Resources - Rosebery

The MMG Rosebery Mine Mineral Resource statement has been upgraded in June 2013 after some 42,000m of Resource infill diamond drilling.

The June 2013 Rosebery Mineral Resource estimate was completed entirely by Rosebery site personnel, with updated 2013 Net Smelter Return after Royalties (NSRAR) calculations provided by MMG Melbourne office.

Upgraded geological and Mineral Resource block models were constructed for K, N, P, W (using the Ordinary Kriging algorithm), X and Y Lenses (using Inverse Distance squared method) using data from the current resource infill diamond drilling campaign. Models presented for M/Q, R/S, T, U and V have updated NSRAR only.

All models were created using Datamine Studio 3 (version 3.21), with standardised macros used to create and define block model boundaries, outline grade domains, assign the Mineral Resource category and allocate NSRAR values based on 2013 economics. These macros also serve as a record of process and files used. All files used and created in the modelling process are saved for future reference.

8.3.1 Results

There are no new first time reports, but there has been material change to the stated Mineral Resources compared to that reported in 2012. Table 105 reports the Rosebery Mineral Resources as at June 30th 2013 above a Net Smelter Return After Royalties (NSRAR) cut-off of $122.50. A reduction in total Mineral Resources of 5.0 Mt from last year is largely attributed to the reclassification of X lens (-2.6 Mt) from Inferred to unclassified, and similarly for one domain of W lens (-1.3 Mt). Mining depletion of Mineral Resources amounts to 516,000t for the year. The remaining reduction in total tonnage is the result of cut-off grade increases resulting from the 2013 revision of economic parameters which have increased cost per tonne, whilst keeping the NSRAR cut-off fixed at $122.50.

Table 105 Rosebery Mineral Resource as at June 30 2013

Rosebery Mineral Resources

Cut-off grade is based on the Net Smelter Return value of A$122.5 per tonne

							Contained Metal				
	Tonnes (Mt)	Zinc (% Zn)	Copper (% Cu)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Rosebery											
Measured	8.1	13	0.4	3.9	120	1.6	1,100	30	316	32	0.42
Indicated	4.9	10	0.3	3.4	130	1.4	500	15	167	20	0.22
Inferred	5.3	10	0.6	3.2	110	2.1	530	31	170	19	0.36
Total	**18**	**11**	**0.4**	**3.6**	**120**	**1.7**	**2,100**	**76**	**650**	**71**	**1.0**
South Hercules											
Net Smelter Return cut-off of A$105 per tonne											
Measured	0.7	3.7	0.1	2	160	2.9	26	0.81	14	3.7	0.07
Indicated	0.1	2.5	0.1	1.2	160	2.9	3	0.13	1.2	0.5	0.01
Inferred											
Total	**0.8**	**3.6**	**0.1**	**1.9**	**160**	**2.9**	**29**	**0.94**	**15**	**4.2**	**0.08**
Total Contained Metal							**2,100**	**77**	**670**	**75**	**1.1**

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

As Rosebery is a polymetallic mine, NSR is used as a cut-off to capture the correct value of the contained metal.

Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.

Competent Person:

Mark Aheimer (Member of AusIMM, employee of MMG)

8.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (“2012 JORC Code”).

Competent Person Statement

I, Mark Aheimer, confirm that I am the Competent Person for the Rosebery Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Rosebery Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Rosebery Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Rosebery Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Mark Aheimer – 27/11/13

Stuart Dawes (Witness)

8.4 Mineral Resource JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Mineral Resources.

Table 106 Checklist of assessment and reporting criteria for Rosebery Mineral Resource

<table>
<tr><th>Criteria</th><th>Status</th></tr>
<tr><td colspan="2">Section 1 Sampling Techniques and Data</td></tr>
<tr><td>Sampling techniques</td><td>Selected diamond drill core samples are analysed at the ALS laboratory in Burnie, Tasmania.
Diamond core is selected, marked and ID tagged for sampling by the logging geologist. Sample details and ID are stored in the database for correlation with returned geochemical assay results.</td></tr>
<tr><td>Drilling techniques</td><td>The most recent underground diamond drilling was LTK48, LTK60, NQ, BQTK and BQ in size.
Historical drillholes are a mixture of sizes from AQ, LTK (TT), BQ, NQ, HQ and PQ (refer Table 107).
Table 107 Historical drillhole size by date and location
<table>
<tr><th>Hole size</th><th>Date range</th><th>Location</th></tr>
<tr><td>AQ</td><td>1938</td><td>Upper mine 4 level</td></tr>
<tr><td>BQ</td><td>1937 - 1969</td><td>Upper mine and declines</td></tr>
<tr><td>TT46</td><td>1986 - 1992</td><td>15, 18 level</td></tr>
<tr><td>TT46, NQ/BQ</td><td>1992 - 1996</td><td>Upper mine, J lens</td></tr>
<tr><td>6.5”, 4.5”</td><td>1996 - 1999</td><td>Open cut</td></tr>
<tr><td>TT48, HQ, NQ/BQ, BQ</td><td>1997 - 1998</td><td>Upper mine, J & T lens, 17 level</td></tr>
<tr><td>TT46, TT48, PQ, HQ, NQ/BQ, BQ</td><td>1998 - 2007</td><td>Lower mine below 35K</td></tr>
<tr><td>6.5”, 4.5”</td><td>1999</td><td>Northern open pit</td></tr>
<tr><td>BQ, NQ, NQ/BQ, HQ/NQ</td><td>2007 - 2010</td><td>Lower mine</td></tr>
<tr><td>TT48</td><td>2007</td><td>47W VAD1820mN</td></tr>
<tr><td>TT48</td><td>2009</td><td>50K FWD1310mN</td></tr>
<tr><td>LTK60, TT48, TT60, BQ, HQ, PQ/HQ/NQ</td><td>2010 - 2012</td><td>Lower mine</td></tr>
<tr><td>LTK48, LTK60, BQTK, BQ, NQ/BQ,</td><td>2012 - 2013</td><td>Lower mine</td></tr>
</table></td></tr>
<tr><td>Drill sample recovery</td><td>Long term diamond drill sample recoveries are 98% based on 21,000 samples. Drill crews mark and define lost core intervals with blocks. Sample recovery is measured and recorded in the database.</td></tr>
<tr><td>Logging</td><td>All drillholes are logged using laptop computers, with Corelog 2005 software from 2010 to present, and Corelog 2000 software from 1996 to 2010.
Prior to 1996 diamond drillholes were logged using Lotus spread sheets or on paper.</td></tr>
<tr><td>Sub-sampling techniques and sample preparation</td><td>Geological samples are prepared as per Rosebery Work Instruction Diamond Drill Core Sample Preparation; core is half sawn for sampling where directed by the logging geologist.
The remaining half is kept and stored in the original sample tray. Unsampled core is discarded.
The standard sampling length is one metre of half core in areas of mineralisation; the minimum length is 40cm; the maximum 1.5m. Sampling is not allowed to extend over domain boundaries (checked by database algorithm).
From 2005 geological samples have been processed in the following manner:
Dried
Crushed
Pulverised to 75µm
3-Acid Partial Digest (considered suitable for base metal sulphides)
Analysis of Pb, Zn, Cu, Ag, Fe by Atomic Absorption Spectrometry (AAS)
Au values are determined by fire assay
Sizing analysis is carried out on 1:20 pulps to ensure 90% passing 75µm
From 2010 geological samples have been processed in the following manner:
Dried
Crushed
Pulverised to 75µm
Lithium Borate Fusion and 3-Acid Partial Digest
Analysis of Ag by Atomic Absorption Spectrometry (AAS)
Au values are determined by fire assay
Analysis of Pb, Zn, Cu and Fe by X-Ray Fluorescence (XRF)</td></tr>
</table>

	▪ Sizing analysis is carried out on 1:20 pulps to ensure 90% passing 75µm
Quality of assay data and laboratory tests	▪ ALS laboratory Burnie releases quarterly QAQC data to MMG for analysis of internal ALS standard performance. The performance of ALS internal standards appears to be satisfactory, but the ALS standards used are for concentrate Pb, Zn and Cu levels and not for drill core. A request has been made for ALS to insert standards more appropriate for drill core assays for further monitoring of laboratory performance. ▪ Routine insertion of matrix-matched standards, dolerite blanks and duplicates occurs at 1:25 ratio to normal assays. ▪ Blanks are inserted to check crush and pulverisation performance. ▪ Duplicates are taken after crush and pulverisation (under review). ▪ Independent audit of the ALS Burnie laboratory and MMG Rosebery sample preparation area undertaken in April 2013 by Coffey Mining Pty Ltd. The key findings for the MMG Rosebery sample preparation area were: – Drill core is cut in half and prepared for chemical analysis; "site duplicates" (¼ core samples) are not collected to establish in-situ variances. – Preparation duplicates are not collected to monitor the splitting fraction or verify the proportion of the splits by the Boyd crusher. – Trays containing crushed drill core are stacked on top of each other, and there is the potential for cross contamination of samples during manual handling and preparation for milling. – Attempts should be made to reduce the level of sample loss during the milling cycle. – The coarse grind size of the geological samples, as evident from laboratory sizing analysis for every one in twenty (5%) drill core samples, requires attention. – Final pulp splitting by spooning from the mill pot is inappropriate and should be replaced with proper pulp splitting equipment such as a rotary pulp splitter. – The Rosebery geology QAQC results indicate a bias in the zinc, lead and copper results reported by the ALS laboratory, Burnie. The key findings for the audit of the ALS laboratory at Burnie were: ▪ The laboratory is working towards ISO 17025 accreditation. ▪ There is an apparent bias in the zinc, lead and copper assays for Rosebery geological and metallurgical samples that require further investigation. Whilst laboratory QA/QC data appeared acceptable (given that it is used to control the analysis), laboratory performance on Rosebery geology reference standards and blanks is unacceptable. The laboratory QA/QC database for Rosebery samples needs verification. ▪ Visits were undertaken by senior Rosebery geology staff to discuss QAQC results and inspect facilities in February and September 2013. Requests were made for ALS to insert standards at levels in the vicinity of ore grade cut-off (close to 5% Zn and 5% Pb) given the continued negative bias of zinc XRF assays; for ALS to release internal laboratory repeat results to MMG; and to confirm samples were processed in sequence to ensure maximum effectiveness of blanks and MMG standards. All these requests were complied with. ▪ QAQC program instigated in 1996 consisting of inserting locally-sourced internal reference material (IRM) as check samples. Three IRMs were chosen; HBM-01, LBM-01 and LBM-02 with agreed values determined by two laboratories (Pasminco and Analabs). ▪ Certified matrix-matched reference materials, duplicates and blanks introduced to QAQC program in 2008. ▪ In 2007-2008 QAQC standards analysed by AAS were noted to be biased low for Pb, Cu, Ag, Au and Fe; zinc precision and accuracy were acceptable and free from bias. ▪ Change to XRF analysis recommended in November 2009 after a standards review in August 2009 confirmed AAS results were biased low and inaccurate. New XRF analyses commenced in early 2010. ▪ QAQC protocols defined by Rosebery Work Instructions currently under review for inclusion in a digital Document Repository system. ▪ Monthly QAQC review commenced in January 2013. Key observations made from the compilation of these reports were: – Insertion of control samples (blanks, duplicates and standards) meets or exceeds the required 1:25 ratio required by the MMG Sampling and QAQC Work Instruction. – The current low grade MMG standard is certified at 9.4% Zn close to the high grade ore cut-off, but no low grade/waste cut-off standard exists. Attempts are being made to source a matrix-matched standard at 5% Zn for inclusion in routine sample analysis. • – The current MMG 'duplicate' process is not providing a true duplicate of sample preparation stages. A duplicate splitter for the Boyd crusher has been ordered and is expected to be installed in November 2013. The

	current MMG duplicate process is actually a pulp repeat – the results of which show >90% of samples within 10% relative difference for Pb, Zn, Ag and Fe. Duplicates for Ag and Au fail to achieve the standard; however, Ag generally falls outside the 10% criterion at low concentrations below 10ppm, whereas Au is poor across the range. These errors are thought to be due to sample inhomogeneity and poor duplicate aliquot selection methods at the sample preparation stage. Ultimately, the entire "duplicate" process needs review. ▪ ALS Burnie assay results for MMG standards are consistently biased low for zinc and iron, but within 3% on average from the certified MMG standard values. The main driver behind this error is thought to be that MMG standards are ICP/INAA certified, whereas the routine ALS analysis method used is XRF. MMG standards were sent to the Brisbane ALS laboratory for umpire ICP check analysis and returned assays within 1% of the certified values. ▪ MMG standards have also been sent to two other independent umpire laboratories for XRF analysis to check and compare the MMG standard performance at ALS Burnie. Results have not yet been received at the time of writing (October 2013). Notwithstanding this, the accuracy of the standards and the ALS assay results within 3% is considered to be sufficient for Mineral Resource estimation. ▪ A new supply of dolerite was obtained for blank material in May 2013 as the preceding supply was exhausted. ▪ Evidence suggests that some smearing or contamination of analytes occurs, particularly for lead and copper from preceding samples, but that the amount of contamination seen is less than 10 times the detection limit on average. ▪ Sizing analysis shows that 96% of samples pass the MMG Guideline of 80% passing 75µm; the more stringent Rosebery requirement for 90% passing 75µm has a 77% pass rate. Sizing data for 2013 reveals an average 92.9% passing 75µm with a standard deviation of 5.7% and a 95% confidence interval of 81.7 to 100%. In March 2013, the Boyd crush size was changed from 3mm to 2mm, and the pulveriser bowl and puck were replaced resulting in a significant improvement in sizing performance from an average 92.4% to 94.6% passing 75µm. ▪ Discussions are currently taking place at management level to move to ICP analysis for all Rosebery evaluation drill core assays in order to remove the abovementioned bias between the XRF analysis and the ICP certified standards.
Verification of sampling and assaying	▪ All data is stored in the Core Log data base which is constructed from a series of MS Access tables and queries run from a Visual Basic front end. The database was purpose built by an external company for geological data collection and manipulation; the version currently in use at Rosebery is version CoreLog2005 version 20 of 19 Aug2013. ▪ Drilling, core logging and sampling data is entered by geologists; assay results are entered by the Senior or Resource geologist after data is checked for outliers, sample swaps, performance of duplicates, blanks and standards, plus significant intersections are checked against core log entries and core photos. Errors are rectified before data is entered into the database. ▪ Core Log validation algorithms are run to check data integrity before verification can take place and data is released for modelling use. Data is able to be flagged manually as excluded by the modelling geologist; the standard Datamine macros only allow verified and unexcluded data to be used for modelling. ▪ During the evaluation drill program, drillholes are planned, where appropriate, to target areas where drillholes have penetrated mineralisation at significant depth from surface, or where drillhole length exceeds 350m and there is potential for the down-hole survey to be in question. The later drill data can then be used as a modelling check for wireframe construction.
Location of data points	▪ All current diamond drillholes are down-hole single-shot surveyed using a Reflex Ezi-shot tool at 30m intervals. ▪ Collar positions of underground drillholes are picked up by Rosebery Mine Surveyors using a Leica TPS 1200. ▪ Collar positions of surface drillholes are picked up by contract Surveyors using differential GPS. ▪ Selected surface exploration holes have been down-hole surveyed using a SPT north seeking gyro (parent holes only). ▪ Multi-shot down-hole surveys were completed to check single-shot survey performance on selected drillholes from 15/2/13 to 13/4/13. A total of 37 drillholes had multi-shot data recorded and compared to single-shot in 3D space. ▪ Grid system used is the Cartesian Rosebery Mine Grid, offset from Magnetic North by 24°33' with mine grid origin at AMG E= 378870.055, N= 5374181.69, and mine grid levels equal to AHD + 1.490m + 3048.000m.

Data spacing and distribution	▪ The Rosebery mineral deposit is drilled on variable spacing dependent on lens characteristics. Drill spacing typically ranges from 100mx100m (Inferred) to 40mx40m (Indicated) to 20mx20m (Measured) and is considered sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserves (MROR) estimation processes and classifications applied.
Orientation of data in relation to geological structure	Drillhole orientation is typically planned at 90 degrees to lens strike in vertical radial fans. Drill fan spacing and orientation is planned to provide evenly spaced, high angle intercepts of the mineralised lenses where possible. Where drillholes from surface or older holes longer than 400m exist, attempts are made to confirm mineralised intercepts by twinning from newly developed drives.
Sample security	Samples are stored in locked compound with restricted access during preparation. Despatch to ALS Burnie via contract transport provider in sealed containers. Receipt of samples acknowledged by ALS by email. Assay data returned separately in both MS xlsx and PDF formats.
Audit and reviews	Coffey Mining Pty Ltd completed an audit of the core sample preparation area in April 2013.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ Rosebery Mine Lease ML 28M/1993 includes the Rosebery, Hercules and South Hercules polymetallic mines. It covers an area of 4,906ha. ▪ ML28M/93 located was granted to Pasminco Australia Limited by the State Government of Tasmania in May 1994. This lease represents the consolidation of 32 individual leases that previously covered the same area. ▪ Tenure is held by MMG Australia Ltd for 30 years from 1st May 1994. ▪ Lease expiry date is 1st May 2024. ▪ The consolidated current mine lease includes two leases; (consolidated mining leases 32M/89 and 33M/89). These were explored in a joint venture with AngloGold Australia under the Rosebery Extension Joint Venture Heads of Agreement. This agreement covered two areas, one at the northern and the other at the southern end of the Rosebery Mine Lease, covering a total of 16.07 sq. km. The joint venture agreement was between the EZ Corp of Australia (now MMG Rosebery Mine) and Shell Company of Australia Limited (now AngloGold Australia Metals Pty. Ltd., formerly Acacia Resources (formerly Billiton)). A Heads of Agreement was signed on 16th May 1988 with initial participating interest of 50% for each party. Other partners in the joint venture are Little River Resources Ltd. and Norgold Ltd. They have a combined net smelter return royalty of 2.3695%, payable on production from the Rosebery Extension Joint Venture area. AngloGold withdrew from the joint venture on the 31st December 2001.
Exploration done by other parties	▪ Significant exploration completed over 70 years of mine life to date. ▪ No exploration drilling carried out in the 2013 reporting period.
Geology	▪ The Rosebery Volcanogenic Massive Sulphide deposit is hosted within the Mt. Read Volcanics, a Cambrian assemblage of lavas, volcaniclastics and sediments deposited in the Dundas Trough between the Proterozoic Rocky Cape Group and the Tyennan Block. ▪ Sulphide mineralisation occurs in stacked stratabound lenses between the Rosebery Thrust Fault and the Mt. Black Thrust Fault; the host lithology and the adjoining faults all dip approximately 45 degrees east.
Drillhole information	▪ No exploration results to report for the 2013 reporting period.
Data aggregation methods	▪ No exploration results to report for the 2013 reporting period.
Relationship between mineralisation widths and intercept lengths	▪ No exploration results to report for the 2013 reporting period.
Diagrams	▪ No exploration results to report for the 2013 reporting period.
Balanced reporting	▪ No exploration results to report for the 2013 reporting period.
Other substantive exploration data	▪ No exploration results to report for the 2013 reporting period.
Further work	▪ Ongoing drill programs will be planned to increase deposit confidence as the need arises.

Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	▪ All Rosebery drillhole data is stored in the Corelog2005 database (SQL Express managed MS Access). ▪ A major database validation project was undertaken in March 2010, with a number of relatively minor errors found. ▪ Validation routines in the Corelog program check for overlapping sample, lithological and alteration information, as well as reject criteria such as logging information past EOH depth.
Site visits	▪ The 2013 Competent Person for the Rosebery site is a permanent employee of MMG based at Rosebery full-time.
Geological interpretation	▪ Mineralisation at Rosebery consists of a series of massive sulphide lenses hosted within felsic volcanics, sandstones and siltstones (Host Sequence). ▪ The Host package and sulphide lenses have an approximate 45 degree dip to the east, with some localised variation. ▪ Geological modelling of the mineralisation follows geological and Pb+Zn>5% grade boundaries. Each lens is interpreted separately as several mineralogical, metallurgical and physical differences occur between lenses relating to structure, alteration and primary mineralogy. ▪ The broad stratiform nature and continuity of the lenses is confirmed by underground mapping of the mining and development exposures.
Dimensions	▪ The Rosebery mineral deposit extends from 400E-1800E, 2500N to -1100N, 3400RL-1900RL (Rosebery Mine grid co-ordinates) and is currently open to the north, south and at depth. Individual lenses vary in size from a few hundred metres up to 1000m along strike and/or down-dip.
Estimation and modelling techniques	▪ Datamine software has been used to estimate Mineral Resources from 1999 to present. ▪ For current Mineral Resource estimations a parent block size of 5m x 10m x 10m is used. Historically smaller block sizes have been used. ▪ Various estimation techniques have been used historically at Rosebery including polygonal, nearest neighbour, inverse distance and ordinary kriging. Current block models use a mixture of inverse distance squared and ordinary kriging, guided by the wireframed geological and mineralogical domains. ▪ Separate block models are created for each lens. ▪ Separate grade caps are applied to both low grade and high grade material. ▪ One metre assay composites are used for all estimation work. ▪ Grades are estimated into the parent block only. ▪ Discretisation is set to 2x4x4 (XYZ) points per parent cell volume. ▪ Octant search methods were not used. ▪ The search ellipse is given a 1.5x expansion and a 3x expansion on the second and third searches respectively for all models except for P lens, which had a second search radius factor of two. ▪ No dilution or recovery factors are taken into account during the estimation of Mineral Resources. ▪ Block models are validated by: – Visual inspections for true fit with the high and low grade wireframes (to check for correct placement of blocks and sub-blocks). – Block model to wireframe volume differences are checked. – Visual comparison of block model grades against composite file grades. – Global statistical comparison of the estimated block model grades against the composite statistics and raw length-weighted data.
Moisture	▪ Tonnes have been calculated on a dry basis.
Cut-off parameters	▪ NSRAR (Mineral Resource cut-off grade) ▪ Rosebery has used a Net Smelter Return After Royalty (NSRAR) as its cut-off grade for the 2013 Mineral Resource modelling to reflect the multi-commodity nature of the Rosebery mineralisation. ▪ The NSRAR cut-off value used for reporting the 2013 Rosebery Mineral Resource is $122.5 (70% of the AU$175 NSR used as the Ore Reserves cut-off). The NSRAR cut-off value used for reporting the 2013 South Hercules Mineral Resource is AU$105 (70% of the AU$150 NSR used as the Ore Reserves cut-off). ▪ In situ geological boundary wireframing was introduced in 2010-2011 to replace the fluctuating metal price dependent cut-off grade. Wireframing is constructed around massive and semi-massive sulphide mineralisation as well as disseminated mineralisation with Pb+Zn>5%. ▪ Long-term economic assumptions for 2013 Mineral Resource Reporting are shown in ▪ ▪ Table 108.

Table 108 Long-term economic assumptions

Description	Value
A$ - US$ Exchange Rate	0.84
Pb Metal Price (US$/t)	2,469
Zn Metal Price (US$/t)	2,601
Cu Metal Price (US$/t)	6,173
Ag Metal Price (US$/oz)	20.0
Au Metal Price (US$/oz)	1,200

The metal price and exchange rate parameters used above are long-term forecasts as issued by MMG Group Office in July 2013.

Mining Factors or assumptions	No mining factors are applied.
Metallurgical factors or assumptions	Recoveries are based on historical recoveries provided by the Rosebery concentrator and monitored in monthly reconciliation reports.
Environmental factors or assumptions	No environmental factors are applied.
Bulk Density	Rosebery uses a formula to determine the dry bulk density (DBD), based on lead, zinc, copper and iron assays, and assuming a certain partition of the iron species between chalcopyrite and pyrite. A study was conducted in August 1999 to examine the accuracy of the above formula compared to measured values. This study concluded the algorithm is accurately representing DBD for the Rosebery mineralisation. The formula is set out below. Galena%= Pb%/0.8658= 1.1550Pb Sphalerite% = Zn%/0.6315789 = 1.5833Zn Chalcopyrite% = Cu%/0.343 = 2.9155Cu Pyrite% = (Fe%-(sp%*0.04)-(cp%*0.3043))/0.467 Therefore; Pyrite% = 2.1413Fe%-0.1356Zn%-1.8997Cu% Non-sulphide gangue (nsg) = 100-(gn%+sp%+cp%+py%) Therefore; Nsg = 100- 1.155Pb-1.5833Zn-2.9155Cu-2.1413Fe+0.01356Zn+1.8997Cu SG = (gn%*0.075)+(sp%*0.04)+(cp%*0.042)+(py%*0.05)+(nsg*0.0265) Gn%*0.075 = 0.866Pb% Sp%*0.04 = 0.0633Zn% Cp%*0.042 = 0.1224Cu% Py%*0.05 = 0.1071Fe%-0.0068Zn%-0.0950Cu% nsg*0.0265 = 2.65 –0.0306Pb% –0.0384Zn%-0.0269Cu%-0.0567Fe% Therefore; SG =2.65+0.0560Pb%+0.0181Zn%+0.0005Cu%+0.0504Fe%

Classification	- Mineral Resource Classifications were based on drillhole sample spacing. *Inferred Mineral Resources* - Mineralisation of demonstrable continuity based on geological interpretation between more than one drillhole. Drillhole spacing nominally 100mx100m or less. *Indicated Mineral Resources* - Drillhole spacing is a nominal 40m along strike and 40m up and down-dip in the plane of mineralisation or less. There is evidence to suggest geological and grade continuity. *Measured Mineral Resources* - The drillhole spacing is a nominal 20m along strike and 20m up and down-dip or less. Knowledge of the geology and grade distribution of the mineralisation is sufficiently high to allow detailed stope definition. In areas of a multi lens nature or complex geology closer drillhole spacing may be required (For example P lens and remnant mining areas).
Audits or reviews	- An internal review of geological and resource modelling processes was completed by Jared Broome and Mauro Bassotti of MMG Limited on 9th September 2013.
Discussion of relative accuracy / confidence	- Twelve month rolling reconciliation figures for Ore Reserves model to Metallurgical Balance are within 5% for all metals except Pb (+12%) and Ag (+6%). - Mining and development mapping by mine geologists shows good general correlation between modelling and actual geology. - The combination of Mineral Resource model, mapping, stope commentaries and inspections provides reasonably accurate grade estimations for stockpile replenishment and mill feeds.

8.5 Ore Reserves - Rosebery

8.5.1 Results

The total Ore Reserves as at 30 June 2013 for MMG Rosebery Mine is summarised below in Table 109.

Table 109 2013 Rosebery Ore Reserves Tonnage and Grade (as at 30 June 2013)

Classification	Tonnes (Mt)	Zinc %Zn	Copper %Cu	Lead %Pb	Silver Ag (g/t)	Gold Au g/t
Proved	2.8	11.8	0.3	3.5	110	1.5
Probable	2.9	8.9	0.3	3.4	130	1.5
2013 Total*	**5.7**	**10.3**	**0.3**	**3.5**	**120**	**1.5**

Totals may differ due to rounding;

Table 110 2013 Rosebery Ore Reserves Contained Metal (as at 30 June 2013)

Classification	Contained Metal†				
	Zinc ('000t)	Copper ('000t)	Lead ('000t)	Silver (Moz)	Gold (Moz)
Proved	330	9	99	10	0.14
Probable	260	9	98	12	0.14
2013 Total*	**590**	**17**	**197**	**22**	**0.27**

Totals may differ due to rounding; †Contained metal does not imply recoverable metal

The major differences from the 2012 Ore Reserves are:

- Mining Depletion.
- Changes in Mineral Resource model.
- Updates and corrections to the NSRAR calculation (including the removal of a previous double counting of silver recovered into the copper concentrate).
- Correction of stope Ore Reserves classification system. Previously a stope average Mineral Resource category expressed as an numeric "grade" was used which allowed Mineral Resources to be upgraded to an Ore Reserves classification above the appropriate classification for the underlying Mineral Resource (e.g. a stope which contained 5% Indicated and 95% Measured Mineral Resource would have been classified in 2012 as a 100% Proved Ore Reserves tonnage rather than as a 95% Proved and 5% Probable Ore Reserves tonnage). The difference in Ore Reserves tonnage in 2013 using the different approaches was 11%.

8.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Julian Poniewierski, confirm that I am the Competent Person for the Rosebery Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Rosebery Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited since August 2012.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 767,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 October 2013 was $HKD 1.72).

I verify that the Rosebery Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in the supporting documentation relating to Ore Reserves as compiled by MMG staff under the supervision of Julian Poniewierski.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Rosebery Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Julian Poniewierski – 26/11/13

Mauro Bassotti – (Witness)

8.5.3 Expert Input Table

A number of persons have contributed key inputs to the Rosebery Ore Reserves determination. These are listed below in Table 111.

Table 111 Contributing Experts – Rosebery Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Adrian Hills, Senior Mining Engineer, MMG Ltd (Melbourne)	Mining Engineering
David Brown, Technical Services Manager, MMG Ltd (Rosebery)	Mining Engineering
Peter Murray, Senior Mining Engineer MMG Ltd (Rosebery)	Mining Engineering
William Bennett, Senior Consulting Engineer, Mining Plus Pty Ltd	Mining Engineering
Jope Nawai, Senior Consulting Engineer, Mining Plus Pty Ltd	Mining Engineering
Mark Aheimer, Geology Superintendent, MMG Ltd (Rosebery)	Resource Geology
Darrin Evans, Resource Geologist, MMG Ltd (Rosebery)	Resource Geology
Stuart Dawes, Resource Geologist, MMG Ltd (Rosebery)	Resource Geology
Ben Reimers, Metallurgical Superintendent, MMG Ltd (Rosebery)	Metallurgy
Willard Zirima, Senior Geotechnical Engineer, MMG Ltd (Rosebery)	Geotechnical Engineering
Gavin Marre, Senior Business Analyst, MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

8.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

8.6.1 Mine Design

All current mining production is carried out by long-hole open stoping. The majority is longitudinal retreat sequence while some limited areas are by wider transverse stopes.

Lenses are divided into panels and are mined using a bottom-up sequence in a continuous 45 degree retreating front towards the level access drives. The nature of this mining sequence causes fluctuations in the grade profile of the short term schedules. Stoping panels contain between 3 and 5 sublevels with crown pillars left in-situ between the backs of up-hole stopes and the lowest sill drive of the panel above.

Stopes are backfilled with unconsolidated rock using trucks with ejector trays or cemented rock fill (CRF) using a system where cement slurry is mixed with waste rock in a sump on the level and placed using a loader. All rock fill is sourced from access development. Up-hole retreat stopes are left as an open void due to lack of access for fill placement. Within large areas of CRF a local pillar is left between every three stopes.

Stope design is carried out using the Mineable Shape Optimiser (MSO) process within CAE mining software (Datamine Mine2-4D). The length of each block used in MSO was set at five metres with each stope being a combination of three or four of these blocks giving a stope strike length of 15m or 20m. Stope strike lengths of 15m were used in W and X Lens while the others lenses used 20m. The height was set to 20m (floor to floor) and the minimum mining width to 3.5m. This was adjusted to 4.65m for horizontal width to allow for the low dip of the ore body and to achieve the 3.5m true width.

8.6.2 Geotechnical Parameters

The ore body is sandwiched between the Mount Black Fault (in the hanging-wall) and Rosebery Fault (in the foot-wall). Sub-vertical, North-South trending faults splay-off these main faults running sub-parallel to ore drives and footwall drives and are exposed at an average of a fault every 20m-25m.

The intersection of these minor fault splays with the general 45 to 60 degrees east dipping foliation/joints creates potentially unstable/convex wedges requiring deeper (cable bolting) support regime in north-south trending drives. On a stope-by-stope basis, geotechnical engineers consider the collected geological structural mapping information to determine stable stope designs to minimize excessive dilution from hanging-wall failures.

Rock property testing is undertaken on a semi-regular basis. In February 2011 N and W lenses basic rock properties tests were done to assist with the decision to reduce inter-level spacing from 25m to 20m.

Seven programs of stress measurements across nine locations have been conducted at Rosebery since 1983. Stress magnitudes and orientations assist in determining favourable stoping direction i.e. managing the stress front by continuous retreat push the stresses into the abutments.

Geotechnical data is collected from all diamond drillholes at Rosebery. The data includes RQD (rock quality designation) and core recovery data.

All headings are mapped for geology and structures after completion of every cut. This data is used by the geotechnical engineers to determine the appropriate ground support regime.

Rosebery recently introduced an upgraded ground control system to cope with changing ground conditions as the mine developed deeper.

A seismic event, which caused some damage in the 51K area in June 2012, resulted in the acceleration of that introduction, and all new development is being supported using the new system.

The original ground control system of friction bolts and mesh that had been used throughout the mine has now been replaced by a ground control system that uses resin-grouted rebar bolts with fibre reinforced shotcrete (FRS) as surface support. This new system provides increased support capacity, which is more suitable for the weak ground and changing stress conditions.

8.6.3 Processing (Metallurgical) Recovery Factors

The metallurgical recoveries for each product are outlined below in Table 112.

Table 112 Rosebery Recovery to Product (Concentrate/Doré)

Product	**Recoveries (%)**				
	Zn	**Pb**	**Cu**	**Au**	**Ag**
Gold Doré	–	–	–	21%	*NB: (1)*
Copper Concentrate	–	–	min(91, 20.933*Cu +54.267)/100%	33%	33%
Zinc Concentrate	min(96, 0.2401*Zn+ 87.632)/100%	–	–	*NB: (2)*	*NB: (2)*
Lead Concentrate	3.7%	min(92, 0.9507*Pb+ 76.804)/100%	–	17.5%	42.1%

Notes:

1) *Silver is calculated as a constituent ratio to gold in the Doré. Silver is set to 0.35 against gold being 0.60.*
2) *There is currently no relationship for gold and silver reporting to Zinc concentrate.*

Table 113 Rosebery Concentrate Grades

Concentrate	**Grade**
Zinc	55.5 %Zn
Copper	20.0 %Cu
Lead	65.0 %Pb

The mill throughput strategy is designed around a series of bottlenecks, at which point a particular section of the flotation plant has reached its maximum capability to handle concentrate. Target feed rate is therefore dependant on the feed grade entering the plant. The bottlenecks are, in descending order of importance: 13 metal tonnes per hour of zinc, 5 metal tonnes per hour of lead and 0.6 metal units per hour of copper. A maximum feed rate of 110 tonnes per hour of ore is targeted if the feed assay is too low to reach any of the bottlenecks.

8.6.4 Realised Revenue Factors (Net Smelter Return)

The input values used for generating revenue towards the Net Smelter Return After Royalty (NSRAR) are based on economic assumptions in place as of 1st February 2013 and discussed in detail in Section 2.2.

Stopes that are scheduled within the first three years were assessed on the medium term price assumptions and if not economic are “turned-off” in the scheduling package. The remaining stopes scheduled in 2017 onwards have their economics assessed on the long-term price assumptions.

The source of revenue for Rosebery is the sales of three separate concentrate products, being zinc, lead and copper, along with a doré product. The terms relating to the transportation and sales of these products have been used to calculate a Net Smelter Return (NSR) and applicable royalties are applied to produce a Net Smelter Return After Royalty (NSRAR).

Concentrate moisture assumptions are 8% for all concentrate products.

High costs on copper concentrate are due to it being considered a dirty concentrate.

For Copper concentrate there are penalties applicable for high levels of combined lead and zinc, arsenic, antimony and bismuth. Penalties paid are currently insignificant to the revenue calculations for the Ore Reserves.

Arsenic levels are however becoming an increasing problem within the Copper circuit. Indications are that the mineralogy may preclude the production of an arsenic content below 5000 ppm. Both elements are expected to increase in concentration as the mine deepens. Currently arsenic is not modelled in the geological block models; further investigation needs to be carried out to determine what level is reporting to concentrate and what penalty impact has been encountered in the past. Test work has commenced in relation to future ore with respect to future metallurgical performance.

Elevated iron levels can hamper efforts to produce the required concentrate zinc grade. This revolves around the amount of iron that is present in sphalerite along with other iron bearing minerals, such as pyrite.

While the process environment and method has not changed, other than planned capacity, there has not been any testing of future ore where the ratio of metals differs to what is being currently supplied. Some future areas indicate there is value from NSRAR calculation but zinc is low at less than 4%Zn and silver is 300g/t plus. This material requires further test work to ensure the value calculation is correct.

Table 114 Rosebery - NSR inputs for zinc concentrate realisation costs

Zinc		
Metal Paid - Zn (total)	85%	%
Minimum Deduction – Zn	8%	% dry
Base Treatment Charge - Zn	200	US$ / dmt con
TC Basis Price – Zn	2,000	US$ / t Zn
TC Escalator – Zn	0.050	US$ / (US$ / t)
TC Deflator – Zn	0.020	US$ / (US$ / t)
Penalties (Zn-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Rail Freight & Port Costs	23.22	A$ / wmt con
Sea Freight	40.0	US$ / wmt con

Table 115 Rosebery - NSR inputs for copper concentrate realisation costs

Copper		
Metal Paid - Cu (total)	96.5%	%
Minimum Deduction - Cu	1.4%	% dry
Base Treatment Charge - Cu	295	US$ / dmt con
Refining Cost	0.295	US$ / lb
Silver		
Minimum Deduction - Ag	30	g / dmt con
Metal Paid - Ag (remainder)	85%	%
Refining Charge - Ag	0.65	US$/Oz payable
Gold		
Minimum Deduction – Au	0	g / dmt con
Metal Paid - Au (remainder)	92%	%
Refining Charge - Au	6.0	US$/Oz payable
Penalties (Cu-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Rail Freight & Port Costs	23.22	A$ / wmt con
Sea Freight (containerised)	120.0	US$ / wmt con

Table 116 Rosebery Mine - NSR inputs for lead (HPM) concentrate realisation costs

Lead			
Metal Paid - Pb (total)		95%	%
Minimum Deduction – Pb		3%	% dry
Base Treatment Charge – Pb	(CY14-16)	210	US$ / dmt con
	(CY17+)	175	US$ / dmt con
Zinc			
Metal Paid - Zn (total)		90%	%
Minimum Deduction – Pb		8.5%	% dry
Silver			
Minimum Deduction - Ag		50	g / dmt con
Metal Paid - Ag (remainder)		95%	%
Refining Charge - Ag		0.65	US$/Oz payable
Gold			
Minimum Deduction – Au		1.0	g / dmt con
Metal Paid - Au (remainder)		90%	%
Refining Charge - Au		6.0	US$/Oz payable
Penalties (Pb-Con.)			
No Penalties are Assumed			
Freight, Sampling and Insurance			
Rail Freight & Port Costs		23.22	A$ / wmt con
Sea Freight		45.0	US$ / wmt con

Table 117 Rosebery Mine - NSR inputs for gold doré realisation costs

Gold in Doré		
Recovery to Doré	21%	%
Average Gold Percentage in Doré	60%	%
Doré Refining Charge	0.82	A$ / Oz
Silver in Doré		
Average Silver Percentage in Doré	35%	%

Royalty

The royalties payable to the government of Tasmania are based on a mix of net sales value and profit. The equation for the royalty payment is:

$$R = (0.019 \times N) + \left(\frac{0.4 \times p^2}{N}\right)$$

where –

R is the royalty;

N is the yearly net sales of the mineral for the immediately preceding year;

P is the yearly profit as defined in regulation 8, if any, for the immediately preceding year.

Further details of the royalty payment and explanation of the royalty formula can be found at the Tasmanian government website:

http://www.thelaw.tas.gov.au/tocview/index.w3p;doc_id=%2B58%2B2006%2BAT%40EN%2B20131001110000.

As the actually royalty paid relies on the profit which is not known at the time of analysis, the historical rates of royalty payment were reviewed. Over the past seven years, the rate of royalty paid ranged between 1.98% and 3.16%, averaging 2.4%. For the purposes of the Ore Reserves estimation a conservative royalty rate of 3.0% has been assumed.

8.6.5 Mining Costs and Cut-Off Value

Costs used in assisting with setting of the cut-off value used for the Ore Reserves estimation were based on an assessment of actual costs for the first six months of 2013. This analysis is presented in Table 118.

Table 118 Rosebery - cost breakdown for first six months of 2013

Cost Categories	**YTD Cost June 2013 (A$ M$)**
Total Mining Cash Costs	$50.8
Mine Technical Services (excl. Geology)	$1.6
Mine Technical Services - Geology	$1.0
Asset Management - Mine Maintenance	$8.9
Mining Costs Deferred (Capital Mine Development)	-$14.8
Total Mining Costs (with Capital Mining Costs excluded)	**$ 47.6**
Total Mill Operating Costs	$ 9.5
Asset Management - Mill Maintenance	$ 6.6
Total Mill Costs	**$ 16.1**
Total Support Costs	**$10.1**
Total Site Op. Cash Costs (with Capital Mining Costs excluded)	**$73.8**
Ore Tonnes to Surface	419,117
Indicative Break-Even NSRAR $/t (with Capital Mining Costs excluded)	**$176**

The indicated break-even NSRAR from Table 118 is consistent with the cut-off value that was applied in last year's Ore Reserves estimate.

The actual cut-off value used for the 2013 Ore Reserves estimate was an NSRAR value of 170$/t (rather than the calculated 176 $/t shown in Table 118). This change was taken following a review of the unit costs for 2013 year to date and comparing this to the original 2013 budget and re-forecast.

A "Hill-of-Value" optimisation study was undertaken in association with AMC Consultants and was completed in early 2013. The study indicated that Rosebery's NPV would be increased by increasing the cut-off value. Planning for increasing the cut-off value is expected to be undertaken during 2014.

8.6.6 Mining Factors and Assumptions

Reconciliation

Currently reconciliation is between design, final outcome and the Mineral Resource block model. Work has commenced on the creation of a Grade Control Model. Review and reconciliation of stopes during 2013 has been less than previously undertaken due to on-going staffing issues during 2012 and 2013.

Production reconciliation for financial; year 2012/13 is summarised in Table 119.

Table 119 Rosebery– Ore Reserves to mill reconciliation 1 July 2012 to 30 June 2013

	Tonnes	Pb %	Zn %	Cu %	Ag %	Au (g/t)	Fe %
Ore Reserves	830,000	2.9	9.6	0.3	101	1.3	7.7
Mill Production	846,000	3.2	9.7	0.3	107	1.3	8.5
Variance	16,000	0.3	0.1	0.0	6	0.0	0.8
% Variance	*2%*	*+10%*	*+1%*	*0%*	*+6%*	*0%*	*+10%*

Dilution and Recovery

Stope grades are calculated from design grades multiplied by a tonnage at zero grade dilution factor (T factor) and tonnage recovery factor (R factor) to the planned stope shapes.

The rates that have been used for dilution and recovery in the 2013 Ore Reserves are derived from previous stope reconciliations as 2012/13 stope reconciliations are substantially incomplete. The factors applied are recognised as being problematic and requiring significant work to better reflect the effects of differing stope parameters. The factors are applied as a percentage and will not accurately reflect potential effects of differing stope widths. Additionally the factors have been generated from historical analysis of stopes predominantly with a strike length of 20m and no allowance for shorter strike stopes now being designed in some of the lenses.

A study was commenced at the start of 2013 to understand the relationship of dilution to the stope size and to create an equivalent linear over-break value, however no usable relationships were uncovered from this analysis. Further geotechnical based analysis of the dilution data will be undertaken in the coming year.

The factors that were used for the application of dilution and recovery to the 2013 Ore Reserves are outlined in Table 120, and are the same as used in 2012 (which were based on reviews undertaken in 2011 and early 2012).

Table 120 Rosebery - dilution and recovery factors used for Ore Reserves

Lens	Stope Type	T Factor	R Factor
K	DHS Longitudinal	1.1	0.9
	DHS Transverse	1.15	0.95
	UHS	1.1	0.9
N	DHS	1.1	0.9
	UHS	1.2	0.8
P	DHS	1.1	0.9
	UHS	1.2	0.8
W	DHS	1.12	0.9
	UHS	1.2	0.8
X	DHS	1.12	0.9
	UHS	1.2	0.8
Y	DHS	1.12	0.9
	UHS	1.2	0.8
Development		1.12	1

Future planning is to introduce paste for stope filling to replace Cemented Rock Fill. No allowance in design or any modifying factors have been applied in relation to the usage of Paste Fill.

8.6.7 Infrastructure

Mining Infrastructure

With mining activity taking place underground at Rosebery, access to the operating areas is by the main decline, “The Fletcher Decline”. Prior to the decline connecting through to surface and becoming the main haulage route, ore was hoisted up the No. 2 shaft, extending from 17L through to discharge on 7 Level. Truck haulage through the decline commenced in March 2003 and the shaft was decommissioned in mid-May 2003.

The Rosebery primary ventilation circuit is essentially a series circuit where airflow accumulates airborne contaminants and heat as it progresses deeper into the mine and finally reporting to the return airways and exhausting to surface. The current primary ventilation system supplies approximately 540m^3/s of air to the underground mine. The system comprises of three primary fan installations on the surface and two booster fan installations underground.

Concentrator

The MMG Rosebery concentrator is located on the edge of the town of Rosebery. The concentrator treats approximately 800,000 t of ore per annum. Ore is primarily sourced from the adjacent underground mine and in the past has also been supplemented by ore from external sources. The concentrator consists of a jaw, cone and rolls crushers followed by primary and secondary ball milling with hydro-cyclone classification. A three stage sequential froth flotation circuit produces copper/gold, lead/silver and zinc concentrates. The concentrates are pressure filtered and railed to the coast at Burnie and then shipped to smelters in Hobart, Port Pirie and elsewhere. Gravity concentration, high intensity cyanidation and electrowinning produce gold doré which is refined in Perth.

The Rosebery Concentrator has operated continuously since its commissioning in 1936.

Tailings Storage Facility

Tailings from the ore treatment are placed in a Tailings Storage Facility (TSF) located to the north of Rosebery at Bobadil. This facility has capacity at current rates through to late 2017. Beyond 2017 there are two options for tailings storage;

(i) upgrading the disused 2/5 Dam TSF, and
(ii) establishing a new location.

The 2/5 Dam can be used as an interim measure between the end of the existing dam’s life and constructing a new TSF. While draft plans exist for a new TSF, permitting for EPA approvals have not been submitted as yet.

Tailings storage capacity is critical to the site as it involves major capital works with extensive lead times.

Potential to implement paste fill to underground operations will have an impact on the life of the current TSF and plans for future sites.

Power

Power supply is contracted with the Electrical Supply Authority for the region. The Supply Authority's substation currently has an N-1 arrangement which ensures that supply is maintained in the event of a loss of critical equipment (e.g. transformer). Works are currently underway to provide an upgrade to the substation infrastructure, the result of which will provide a significant increase to the security of the supply to the site.

Water

Fresh water for the site is currently sourced from Lake Pieman with an allotment of 5,500ML. An allotment of 1,647ML from the Stitt River was handed back to Cradle Mountain Water during Quarter 1 of 2013. A further major source of input water is from precipitation and runoff, accounting for 3,106ML.

Information for 2012 shows 3,540ML being sourced from Lake Pieman out of a total site input volume of 9,019ML.

Communications

Primary communication from the Rosebery Mine site is by phone along with surface mobile phone coverage, provided by Telstra. Along with the phone system there is connection to email and internet services through a wireless system.

Airport

The nearest airport is a commercial airport at Wynyard (Burnie), some 1.5 hours' drive from Rosebery.

Road Access

Rosebery is accessed via the Murchison highway which passes through the mine lease and within 100m of the mine site. The Rosebery mine is located on the north western edge of the town of Rosebery.

Road access to the site is good with sealed roads to both the North Coast area and south through to Hobart. While these are sealed roads there can be issues during winter with snow and ice making travel hazardous.

Rail Access

Concentrate is transported using the Emu Bay Railway which is a freight only rail line that connects the West Coast area to the port in Burnie.

8.6.8 Environmental

Waste Water

The waste water management at Rosebery involves collecting all potentially contaminated water including storm water, mine water and mill tailings at the Effluent Treatment Plant (ETP), where lime is added prior to pumping the whole volume of treated water to the Bobadil TSF via the Flume (an open concrete channel flowing under gravity to the TSF). After the final polishing stage, water is subsequently discharged to the Pieman River.

The ETP hydraulic capacity is approximately 600 l/sec and the plant is capable to receive 335 l/sec of site mine water with remaining limited spare capacity of approximately 265 l/sec to treat the site surface rain or storm water.

Environmental Legacy Sites

There is a range of environmental legacy sites that are indirectly related to Rosebery that are being managed by Group Office. While these are not directly related to the current operations they are located either on the mine lease or are in the local region.

The historic Hercules and South Hercules area has a large impact on the land area along with major water issues. This would be the current leading legacy site. Smaller historic legacy sites include the Zeehan Smelter site and historic mines numbering at least ten known sites, such as Jupiter's, along with a number of suspected workings.

Waste Rock

Currently there is no noted differentiation in waste rock classification. Work has commenced to collate information in relation to acid forming potential which is ongoing.

The majority of waste rock produced is retained underground and used for stope filling, either as straight Rock Fill or as Cemented Rock Fill. Any surplus waste rock is trucked to the surface and unloaded at the current waste rock dump, referred to as Assay Creek.

Approval for a new waste rock dump (EPN 8815/1) has been gained which will be located within the existing open pit. Aspects of the approval for this waste dump have implications in regards to the management of potentially acid forming (PAF) waste rock, hence the work being undertaken to understand the acid forming potential of waste rock.

A further area of work being investigated which will impact on waste rock management is the utilisation of paste fill. This will impact on the amount of waste rock required for filling and the amount brought to surface.

Environmental Standards

Environmental matters are managed on site under ISO 14001 certified document: 2004 Environmental Management Standard which was last audited in June 2012.

8.6.9 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 121.

Table 121 JORC Code Ore Reserves Assessment and Reporting Criteria for Rosebery 2013 Ore Reserves

Assessment Criteria	**Risk Assessment**	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	–	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The Ore Reserves estimate has been generated by applying the metallurgical, social, environmental and financial aspects of the operations (the modifying factors) on that portion of the Mineral Resource Estimate, classified as "Measured" and "Indicated". Further details are discussed in the Mineral Resources Section of this report
Classification	Low – Medium	There is a mixture within each stope of blocks of multiple Mineral Resource classes. These have been separated out into their individual constituents such that no Inferred Mineral Resources are included and no Indicated Mineral Resources are upgraded to Proved Resources. All Proved Ore Reserves derive from Measured Mineral Resources. All Probable Ore Reserves derive from Indicated Mineral Resources.
Site visits		The Competent Person Julian Poniewierski visited the Rosebery site during 2013 on the dates of 3-6 September 2013.
Study status	Low	The mine is an operating site with on-going detailed Life-of-Mine planning.
Cut-off parameters	Low	See Section 8.6.5 for details.
Mining factors or assumptions	Medium	See Section 8.6.1 for details on mine design. See Section 8.6.2 for details on geotechnical parameters. See Section 8.6.6 for details on dilution, recovery and reconciliation.
Metallurgical factors or assumptions	Low	See Section 8.6.3 for details.
Environmental	Medium	See Section 8.6.8 for details.
Infrastructure	Medium	Old infrastructure, mill commissioned in 1936. Ongoing rehabilitation requirements. See section 8.6.7 for details.
Costs	Medium	Operating costs were taken from actual costs for the year to date and compared to the Budget and a later forecast. Further work will be completed in this area for improved cost breakdown and allocation. Maintaining control of costs with increases in ground support and other underground activities. Any reductions in mining cost will have largest impact across site.
Revenue factors	Low	See Section 8.6.4 for details.
Market assessment	Low	See Section 2.2 for details.
Economics	Low	The mine is profitable and life-of-asset economic modelling shows that the Ore Reserves are economic.
Social	Low	The West Coast area of Tasmania has a strong long history with mining. There are a large number of people employed by the mine from the town of Rosebery and the local area. Community issues and feedback associated with the Rosebery mine are generally received through the MMG Community Liaison Office in Agnes Street, Rosebery. All issues are reported on a Communication and Complaints form and forwarded to the Stakeholder Relations Officer for action as soon as practicable. The Stakeholder Relations Officer makes direct contact with the complainant to understand the issue. Once details are understood the Stakeholder Relations Officer then communicates with the department concerned to resolve the matter. All complaints are registered on Stake Tracker (formally RIMS), where if required, corrective actions are initiated and monitored. During the 2012/13 reporting period, a total of ten feedback/complaints were received regarding noise. As a result, the Stakeholder Relations Officer conducted two meetings with the local residents and a door knocking exercise in the area of complaints. A noise impact assessment has been completed by Aurecon concluding the Concrete Batching Plant is the main source of nuisance noise. MMG is progressing with the recommended actions from the impact assessment. As an intermediate measure, the concrete batching plant has restrictions on operation times to specified times during the day only.
Audit or Reviews	–	At the start of the Ore Reserves process there was a review of the NSRAR calculation method by Group Technical Services. This fixed an error in revenue calculation for Zinc concentrate and changed field naming to avoid unintended duplicate application of values. During the review of the calculation the Marketing Department was extensively consulted to verify the included assumptions.

Assessment Criteria	Risk Assessment	Commentary
		The Geology Department at Rosebery also spent time working with the NSRAR script to ensure correct operation for each model. Detail has been added to the script and background document to track when and who has made changes. Mineral Resource block models had verification processes run over them during the design and evaluation process. There have been no independent internal or external review or audit carried out on the Ore Reserves process during the past year.
Discussion of relative accuracy/ confidence	–	A qualitative risk assessment of each discussed item is included with each individual item in the second column of this table.
Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Medium - Low	The Rosebery mine is located on the west coast of Tasmania on the edge of the central highland plateau. The area immediately surrounding the mine is characterised by glacial valleys and steep mountainous terrain. The mine surface infrastructure is located at an approximate elevation of 100m above sea level. The major risk the topography poises is that it limits the area available for expansion of facilities.
Climate	Low	The climate is wet temperate with approximately 2000mm of rainfall annually. Summers may be mild to warm with maximum temperatures in the mid-30^{O}C, while the winters are cool to cold with occasional snow and ice. Risks associated with climate are low as mine has operated in this climate for 75+ years.
Government Agreements	Low	Stable government environment, receptive to revenue from mining.
Waste Storage (Including Tails Storage)	Medium - High	See Section 8.6.8 for discussion of waste rock. See Section 8.6.7 for discussion of TSF. There is limited capacity in current TSF and no decision for a new site has yet been made.

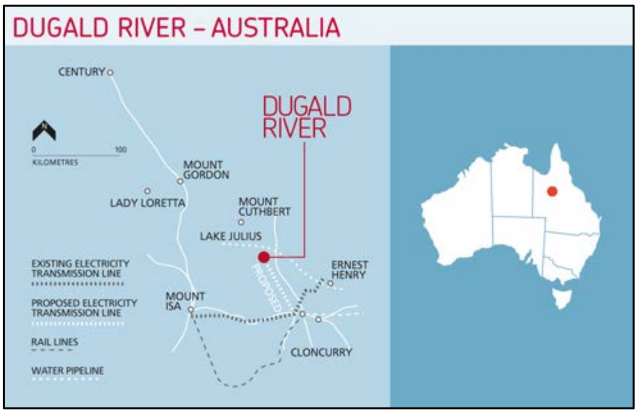
9. DUGALD RIVER PROJECT

9.1 Introduction and setting

The Dugald River project is located in northwest Queensland approximately 65km northwest of Cloncurry and approximately 85km northeast of Mt. Isa. It is approximately 11km (by the existing access road) from the Burke Developmental Road, which runs from Cloncurry to Normanton (**Error! Reference source not found.**).

It is one of the world's largest undeveloped zinc-lead-silver deposits containing a Mineral Resource of 63Mt at 12% Zn, 1.8% Pb, 31g/t Ag and 0.8% Mn (Table 122) and is wholly owned by a subsidiary of MMG Limited. The project is not currently operational; however investigative development work is currently underway allowing access to the orebody for further test-work and potentially trial stoping.

Figure 80 Dugald River project location



9.2 Geological setting

The Dugald River deposit is located within a 3km-4km wide north-south trending high strain domain named the Mt. Roseby Corridor. The Mt. Roseby Corridor has experienced complex polyphase deformation and metamorphism during the Isan Orogeny, which has resulted in widespread alteration and transposition of both stratigraphy and pre-existing structural fabrics.

The main Dugald River lode is hosted within a major north-south striking steeply west dipping shear zone which cross cuts the strike of the Dugald River Slate stratigraphy at a low angle. All significant zinc-lead-silver mineralisation is restricted to the main lode. Lesser-mineralised hanging wall and footwall lenses are present. Three main mineralisation textures/types are recognised, including banded, slaty breccia, and massive breccia.

The mineralogy of the Dugald lode is typical of a shale-hosted base metal deposit. The main sulphide minerals are sphalerite, pyrite, pyrrhotite and galena with minor arsenopyrite, chalcopyrite, tetrahedrite, pyrargyrite, marcasite and alabandite. The gangue within the lode is composed of quartz, muscovite, carbonates, K-Feldspar, clays, graphite, carbonaceous matter and minor amounts of calcite, albite, chlorite, rutile, barite, garnet, and fluorite. The metamorphic grade of the sulphides is upper greenschist facies as indicated by few sphalerite grains achieving sphalerite/pyrrhotite/pyrite equilibrium and the graphitisation of original amorphous carbonaceous material.

The strike length of the mineralised zone is approximately 2,400m between 13350mN and 15750mN, striking north-south and dips between 85° and 45° to the west. A south plunging flexure zone of shallow dip occurs at 14400mN (at surface). The true thickness of the majority of the Mineral Resource is between 3m and 30m, with the thickest zones occurring around a north plunging shoot.

9.3 Mineral Resources – Dugald River

9.3.1 Results

The June 2013 Mineral Resource estimate for the Dugald River deposit is shown in Table 122.

The Mineral Resource is reported at a cut-off grade of 6% zinc. This grade defines mineralisation which is prospective for future economic extraction and is unchanged from the 2012 Mineral Resource. The Mineral Resource has been depleted to account for mining of ore by way of underground development of ore drives (Table 123).

Table 122 June 2013 Dugald River Mineral Resource at a 6% zinc cut-off

June 2013 Dugald River Mineral Resource at Zn > 6% Cut-off (Depleted as of 30/05/2013)								
	TONNES (MT)	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Mn (%)**	**Fe (%)**	**S (%)**	**Tot. C (%)**
Measured	3	14	1.9	61	0.5	12	17	3.8
Indicated	31	12	1.9	46	0.6	11	14	2.4
Inferred	29	12	1.7	13	0.9	11	15	0.4
	TONNES (MT)	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Mn (%)**	**Fe (%)**	**S (%)**	**Tot. C (%)**
Measured + Indicated	***34***	***12***	***1.9***	***47***	***0.6***	***11***	***14***	***2.5***
Inferred	***29***	***12***	***1.7***	***13***	***0.9***	***11***	***15***	***0.4***
	TONNES (MT)	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Mn (%)**	**Fe (%)**	**S (%)**	**Tot. C (%)**
Total (M+I+I)	***63***	***12***	***1.8***	***31***	***0.8***	***11***	***15***	***1.6***

Table 123 Material Mined as of 30/05/2013

Material Mined as of 30/05/2013								
TONNES ('000 t)	**DENSITY**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Mn (%)**	**Fe (%)**	**S (%)**	**Tot. C (%)**
116	3.1	9	1.1	40	0.3	10	13	3.5

All Mineral Resources quoted in this report were estimated from three dimensional block models created with Datamine software. Mineral Resources are modelled using solid wireframes of geological boundaries and also looking at the natural grade break between the high and low grade mineralisation domains. The "natural" grade break was determined by looking at both the geological logging and zinc grade distribution and selecting a domain contact that is representative of the high-grade massive sulphide mineralisation.

Zn, Pb, Ag, Mn, Fe, S and Ctot (total carbon) grades were interpolated using an ordinary kriging algorithm. Variogram and estimation parameters were defined using Supervisor Software. The drillholes were unfolded into a flat plane which then allowed variography of the data to be performed in an unfolded space, thus allowing more accurate variograms to be generated.

Estimates were modelled on geological domains and density estimated in the model. Blocks where a density was not estimated, a stoichiometric formula was used to inform the blocks.

Increased underground drilling and mapping of exposed underground mineralisation has resulted in a better understanding of mineralisation continuity and geometry at Dugald River. The Dugald River Mineral Resource classification has changed from the 2012 Mineral Resource to reflect increased understanding of the mineral deposit.

The 2012 Mineral Resource is shown in Table 124. Changes between the 2013 and 2012 Mineral Resource tonnes and grades are shown in Table 125.

The Measured Mineral Resource tonnage has been reduced by 85% and is now constrained to locations that are supported by underground drilling at 20m x 20m nominal drill spacing, ore drive development and associated geological mapping and robustness of the grade estimate. The robustness of the estimate was determined by assessing the distribution of the kriging variance, efficiency and slope of regression in the estimated model and then create 3D wireframes that were used to select the Mineral Resource categories.

The Indicated Mineral Resource has increased by 39% resulting in a 21% total reduction of Measured and Indicated combined.

The Inferred Mineral Resource has increased by 130%, with this change accounting for previously classified Indicated material reclassified as Inferred.

The global Mineral Resource has increased by 13%.

Breakdown of changes between the 2013 and 2012 Mineral Resource are illustrated in the waterfall charts below (Figure 81 and Figure 82) for total tonnes and zinc metal tonnes.

Table 124 June 2012 Dugald River Mineral Resource at a 6% zinc cut-off

November 2012 Dugald River Mineral Resource at Zn > 6% Cut-off	**TONNES (MT)**	**DENSITY**	**Zn (%)**	**Pb (%)**	**Ag g/t**	**Mn (%)**	**Fe (%)**	**S (%)**
Measured	21	3.2	13	2.1	58	0.7	12	16
Indicated	22	3.2	13	2.0	26	0.9	12	15
Inferred	13	3.2	12	2.0	16	1.0	11	15
Measured + Indicated	***43***	***3.2***	***13***	***2.0***	***42***	***0.8***	***12***	***16***
Inferred	*13*	*3.2*	*12*	*2.0*	*16*	*1.0*	*11*	*15*
Total (M+I+I)	***56***	***3.2***	***13***	***2.0***	***36***	***0.8***	***12***	***15***

Table 125 Comparison between 2012 and 2013 Mineral Resources

2013 vs. 2012 Mineral Resource % Differences by Classification – Zn > 6% Cut Off	**TONNES**	**DENSITY**	**Zn (%)**	**Pb (%)**	**Ag g/t**	**Mn (%)**	**Fe (%)**	**S (%)**
Measured	-85%	3%	3%	-7%	5%	-29%	5%	7%
Indicated	39%	0%	1%	0%	83%	-20%	2%	-1%
Inferred	130%	0%	2%	-14%	-18%	-7%	0%	0%
Measured + Indicated	**-21%**	**0%**	**1%**	**-2%**	**17%**	**-15%**	**2%**	**0%**
Total (M+I+I)	**13%**	**0%**	**-1%**	**-8%**	**-10%**	**-5%**	**0%**	**-1%**

Figure 81 Waterfall chart of changes in tonnes in the June 2013 Mineral Resource

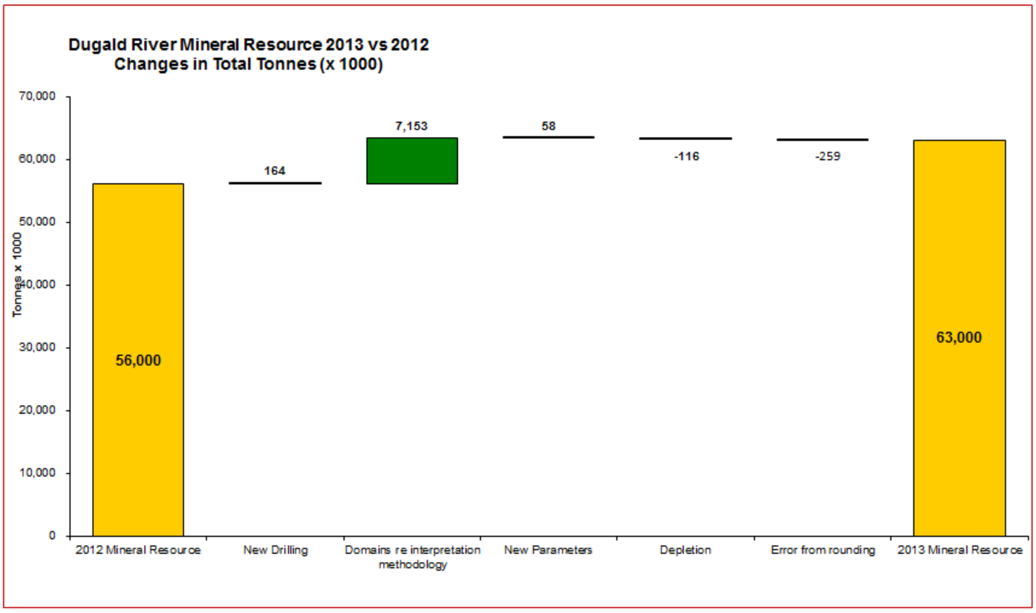
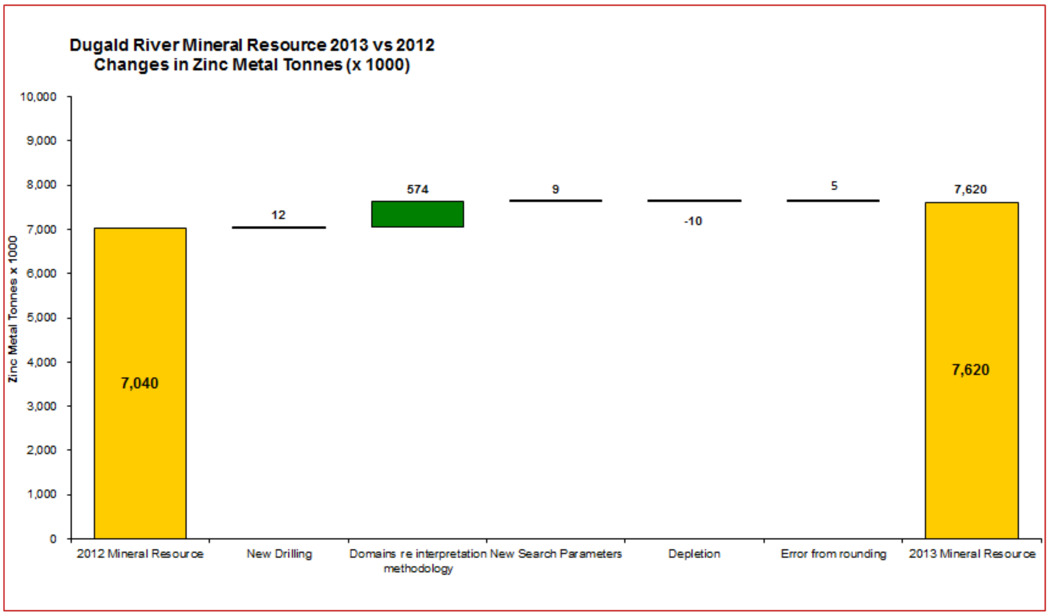


Figure 82 Waterfall chart of changes in zinc metal tonnes in the June 2013 Mineral Resource



9.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (“2012 JORC Code”).

Competent Person Statement

I, Mauro Bassotti, confirm that I am the Competent Person for the Dugald River Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Dugald River Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Dugald River Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Dugald River Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Mauro Bassotti – 26/11/13

Anna Lewin (Witness)

9.4 Mineral Resource JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of Dugald River Mineral Resources.

Table 126 Checklist of assessment and reporting criteria for Dugald River Mineral Resource

Criteria	Status
	Section 1 Sampling Techniques and Data
Sampling techniques	▪ Diamond core was sampled either whole, ¾, ½, ¼, or sliver for the PQ core. ▪ 7% of the dataset was sampled using reverse circulation (RC). ▪ Table 127 illustrates the various sampling techniques as captured in the database. The tabulation is only for drillholes used in the 2013 Mineral Resource. A large portion of the drillholes data (72%) does not have a drill sample type value inserted in the database. ▪ Half core splits of NQ2 or LTK60 was collected from underground diamond drilling in 2013 drilling.

Table 127 Drill sampling by type

Drill sampling type	Metres	% of Total
¾ core (undifferentiated)	409	0.27%
Diamond drilling (undifferentiated)	158	0.11%
½ core	344	0.23%
½ core BQ	111	0.07%
½ core HQ	518	0.35%
½ core HQ3	5,113	3.43%
½ core LTK60	482	0.32%
½ core NQ	185	0.12%
½ core NQ2	13,890	9.32%
½ core NQ3	1,189	0.80%
No record (in database)	***107,648***	***72.20%***
Pulps (re-assays)	442	0.30%
¼ core	296	0.20%
¼ core NQ2	47	0.03%
RC	***10,489***	***7.04%***
Unknown	664	0.45%
Whole core	7,055	4.73%
Whole core BQ	23	0.02%
Whole core NQ	34	0.02%
TOTAL	**149,097**	**100%**

Criteria	Status
Drilling techniques	▪ A number of drilling techniques were used which are; tabulated in Table 128 with number of metres drilled for each technique and proportion in percentage of the database as used in the 2013 Mineral Resource. ▪ 54% of surface diamond drilling does not have an entry in the database that allows separation of the data by drillhole diameter. This data is pre-2007. ▪ Post-2007 data has correctly been captured in the database. ▪ 4% of the drilling data used in the Mineral Resource is RC type. ▪ 2013 underground drilling data is predominantly NQ2 with some LTK60.

Table 128 Drilling techniques

Hole type	Hole diameter	Metres	% of total
DD	HQ	4,119	2.1%
DD	HQ2	501	0.3%
DD	HQ3	19,300	9.7%
DD	NQ2	35,327	17.8%
DD	NQ3	2,962	1.5%
DD	PQ	361	0.2%
DD	***Unknown***	***107,831***	***54.3%***
DD_UG	LTK60	1,461	0.7%
DD_UG	NQ2	18,763	9.5%
RC	***Unknown***	***7,828***	***3.9%***
TOTAL		**198,453**	**100%**

DD = Surface diamond drilling. DD_UG = Underground diamond drilling

Drill sample recovery

- Recovery recorded during core logging was generally 100%, with minor losses in broken / sheared and faulted ground. There is no relationship between core loss and mineralisation or grade (Table 129).

Table 129 Recovery (%) in mineralisation units and hangingwall shear zone

Mineralisation/Lithology	Recovery % (average)
Banded ore	99.4
Carbonate mineralisation	100.0
Massive sulphides	100.0
Pyrrhotite slaty breccia	99.9
Siliceous mineralisation	99.4
Slaty breccia	99.9
Sulphide stringers	99.0
Hanging wall shear zone	*90.7*

Logging

- Core logging recorded geological and geotechnical information including lithology, stratigraphy, weathering, alteration, geotechnical characteristics.
- Mineralised core is stored at -4°C in refrigerated containers to minimise oxidation for metallurgical testing. Unmineralised drillcore is stored on pallets in the yard.
- Core photographs are available for most drillholes. All drillholes post-2008 have been photographed both wet and dry.

Sub-sampling techniques and sample preparation

- Sample intervals were selected based on geological contacts.
- Sample lengths were 1m while still respecting the geological contacts.
- Core was cut by diamond saw.
- Core samples were then bagged, numbered and dispatched to assay laboratories.
- Samples were generally 2kg to 3kg in weight.
- Sample preparation was completed by ALS Mount Isa:
  - Original 3kg bulk sample received and jaw crushed to a coarse 9mm.
  - Sample is then split (riffle split) with:
    - Half returning back to site for storage (coarse rejects).
    - Half fine crushed (Boyd crusher) to 70% nominal passing 2mm.
  - Sample is then split (rotary splitter).
  - Sample pulverised 500g to 800g to 85% passing 75µm.
  - Pulps sent to Brisbane ALS for analysis.

Quality of assay data and laboratory tests

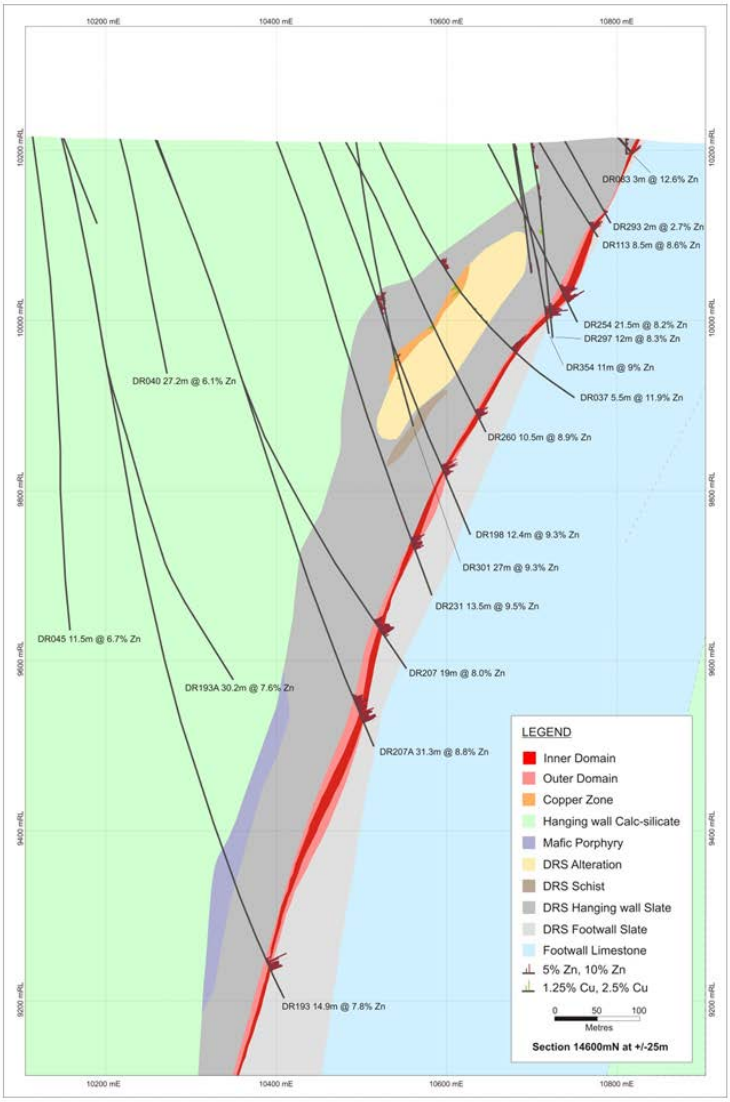
- 2013 ALS Lab visits were undertaken by MMG site geologist and the Competent Person. No major issues were identified in both the sample preparation and analysis laboratories.
- 2013 March ALS Brisbane Laboratory audit completed by IO Global (for MMG Exploration) identified a number of recommendations:
  - Balance bench is made of wood and is not stable when leaned on. It was observed that a weight fluctuation of up to 0.006g was caused through leaning on bench.
  - Balances used for MMG analysis are 3 decimal place top loaders (Mettler or AND). There is some draught in the room due to the air conditioning system.

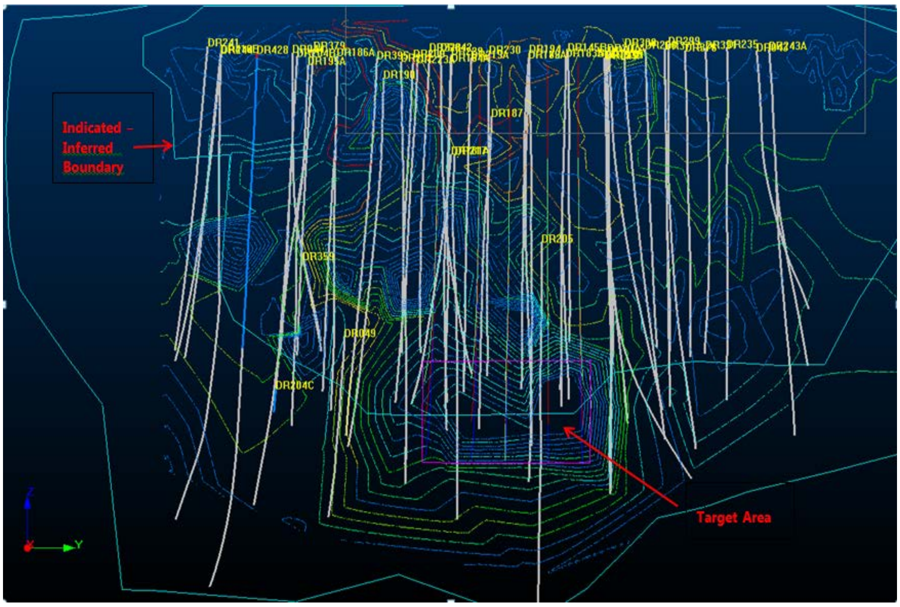
No critical issues were identified by the IO Global audit.

- Base metals 4-acid digests analysis was used. The digestion process was as follows:
- Approximately 0.25g of sample catch weighed into a Teflon test tube.
- HNO_3 and $HClO_4$ were added and digested at 115°C for 15 minutes.

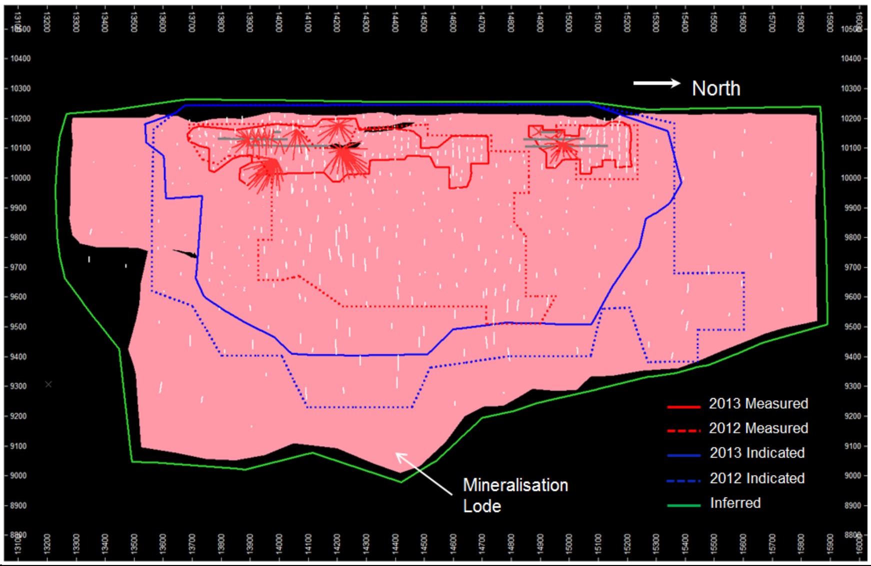
	▪ HF is added and digested at 115°C for 5 minutes. ▪ The tubes were then digested at 185°C for 145 to 180 minutes. Taking the digest to incipient dryness. Digest is not "baked". ▪ 50% HCl was added and warmed. ▪ Made to 12.5ml using 9.5ml 11% HCl. – Acid volumes were added using dispensers that are calibrated daily by weight. – Failed despatches were analysed for Znppm, Pbppm, Agppm and Mnppm. Despatches were failed when the returned assay value is above 3 standard deviations. – A mix of internal and external standards and coarse blanks were submitted with every batch of samples: o Standards inserted at approximately 1:10. o Blanks inserted at approximately 1:20. o Duplicate samples were selected from returned coarse rejects. These were dispatched along with standards and blanks.
Verification of sampling and assaying	▪ Assay results were visually verified against logging and core photos. ▪ Core logging data was recorded directly into a Database (GBis) by experienced geologists (geological information such as lithology and mineralisation) and field technicians (geotechnical information such as recovery and RQD). ▪ 50% of the original sample that was jaw crushed to 9mm was sent back from ALS for storage onsite. The sample was split from the original by way of a riffle splitter. Approximately 5% of the coarse rejects were sent to an independent laboratory (Genalysis). ▪ Duplicate samples were selected from coarse rejects returned from the lab. ▪ Standards used in sampling were cross checked before dispatching. This process was as follows: – Geologists select the standards and where they are inserted amongst the samples. – Cut sheets were printed for the field technicians. – The field technicians sign off on each standard and blank that is inserted (requires two people to check). – The signed cut sheets were stored in folders that are kept in the core shed. ▪ Dispatches where a number of standards were returned with results greater than 3 standard deviations away from the certified mean were failed and the whole batch is re-assayed.
Location of data points	▪ All drillhole collar surveys were undertaken by a licensed surveyor. ▪ All surface collar points were surveyed in MGA94 and then converted into local mine grid. ▪ All underground collar points were surveyed in local mine grid using total station. ▪ Strong local magnetic fields associated with pyrrhotite mineralisation within the deposit reduce the effectiveness of conventional down-hole survey tools, therefore; – All underground drillholes were gyroscopically surveyed. – 181 surface drillholes have also been gyroscopically surveyed. ▪ Drillholes that have not been gyroscopically surveyed rely on single-shot down-hole camera readings.
Data spacing and distribution	▪ Drill spacing varies across the strike and dip of the mineralisation lode. ▪ The highest drill density in the orebody is 20m x 20m while the lowest drill density is greater than 100m x 100m spacing. ▪ Underground mapping of faces was digitised and used in the interpretation and wireframing process. ▪ Drillhole data are predominately located in the top 300m of the Mineral Resource. This is due to the difficulty and cost involved in drilling deeper sections.
Orientation of data in relation to geological structure	▪ Geological mapping (both underground and surface) and interpretation show that the mineralisation is striking north-south and dips between 85 and 45 degrees towards the west. Hence drilling is conducted on east-west and west-east directions to intersect mineralisation across strike. ▪ Drilling orientation is not considered to have introduced any sampling bias. ▪ Drillholes that have been drilled down-dip and semi-parallel to the mineralisation have been excluded from the estimate.
Sample security	Measures to provide sample security included: ▪ Adequately trained and supervised sampling personnel. ▪ Well maintained sampling sheds. ▪ Cut core samples stored in numbered and tied calico sample bags. ▪ Calico sample bags transported by courier to assay laboratory. Assay laboratory checks of sample dispatch numbers against submission documents.
Audit and reviews	▪ Audit of the ALS Sample Preparation Laboratory in Mount Isa was conducted in 2013 by site geologists. No critical issues were identified.

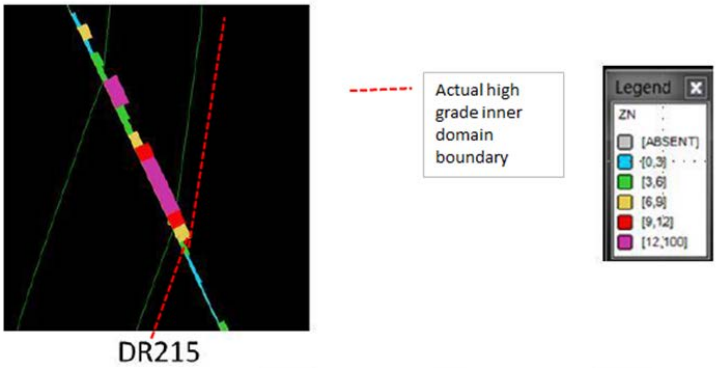
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The Dugald River Mining Leases are wholly owned by a subsidiary of MMG Limited. ▪ MMG holds one exploration lease and one mineral development lease in addition to the mining leases on which the Dugald River Mineral Resource is located. EPM12163 consists of 6 sub-blocks and covers an area of 20sqkm to the west of the Dugald River deposit. MDL 79 overlaps the north-western area of the EPM12163.
Exploration done by other parties	▪ There is currently no exploration done by other parties.
Geology	▪ The Dugald River deposit is hosted by steeply dipping mid-Proterozoic sediments of the Mary Kathleen Zone in the Eastern Succession of the Mt. Isa Inlier. ▪ The host sequence is composed of the Knap dale Quartzite and the Mt. Roseby Schist Group (which includes the Hangingwall calc-silicate unit, the Dugald River Slate and the Lady Claire Dolomite). The sequence is an interbedded package of greenschist to amphibolite grade metamorphosed carbonate and siliclastic lithologies. ▪ Mineralised widths vary from 3m to 30m. The mineralised zone extends approximately 2.4km in strike length and up to 1.35km down-dip.
Drillhole information	▪ 796 diamond drillholes and associated data are held in the database. No individual hole is material to the Mineral Resource estimate and hence this geological database is not supplied.
Data aggregation methods	▪ No metal equivalents were used in the Mineral Resource estimation.
Relationship between mineralisation width and intercepts lengths	▪ Mineralisation true widths were captured by 3D modelled wireframes with intercept angles ranging from 90 to 45 degrees (Figure 83).

Diagrams	**Figure 83 Cross-section looking north** 
Balanced reporting	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

Further work	▪ No future regional exploration programs are currently planned. ▪ 12,500m of down-dip infill drilling is planned for early in 2014 (Figure 84). The program is planned to be drilled from surface targeting the thick and high-grade zone to the south. The aim is to convert this Inferred material to Indicated and to confirm the thickness of mineralisation. **Figure 84 Long-section indicating proposed Indicated – Inferred drilling target area** 
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	▪ All data are stored in a GBis Relational Database onsite. ▪ The database is replicated every 24hrs to the Melbourne Server for backup. ▪ All logging are digital and directly entered into the onsite GBis database via a wireless connection (at the core shed). ▪ Data integrity is managed by internal GBis validation checks/routines that are administered by the Melbourne Database Group and/or the site Geology Team. ▪ Data integrity was also checked externally by running Datamine macros on the drillhole file to check for EOH, and sample overlaps. ▪ Manual checks were carried out by reviewing the drillhole data in plan and section views.
Site visits	The Competent Person visited site on various occasions through 2013. Site visits included involvement with: ▪ Assist with implementation of wireframe/modelling procedures. ▪ Assist with wireframe interpretation and methodology as applied in the 2013 Mineral Resource work. ▪ Assist with generation of underground level plan sections. ▪ Inspection of geological mapping plans. ▪ Inspection of underground workings. ▪ Training of site resource geologist.
Geological interpretation	▪ The mineralisation zone is modelled within two domains, the outer and inner domain. ▪ The inner domain is defined as the high-grade zinc domain and defines a continuous zone of massive and breccia sulphide textures. ▪ The outer zone defines the surrounding lower-grade mineralisation with its associated assemblage of sulphide stringers and shoots of discontinuous massive and breccia sulphide textures. ▪ Separate wireframes have been constructed in Datamine for the inner and outer domains. ▪ Where possible a low-grade (internal dilution) domain has been identified and modelled within the high-grade domain. ▪ Selection of the low/high-grade domain was based on geological observations and assay results. Zinc grade histograms in combination with geological logging were used to assist in selecting this contact. ▪ Underground mapping of development drives for both access and ore drives were also used in assisting with the geological interpretation.
Dimensions	▪ The main Dugald lode is hosted within a major north-south striking steeply west dipping shear zone which cross-cuts

	the strike of the Dugald River Slate stratigraphy at a low angle. ▪ The strike length of mineralisation is approximately 2,400m between 13350mN and 15750mN. ▪ Dip varies between 85 and 45 degrees to the west. ▪ The true thickness of the majority of the Mineral Resource is between 3m and 30m with the thickest zones occurring to the south.
Estimation and modelling techniques	▪ Mineral Resource modelling was done using Datamine software. ▪ Zonal composite was done at a nominal 1m interval with residual composite intervals absorbed evenly into the composites resulting in no loss of sample intervals. ▪ Grade capping was completed post compositing. Values greater than selected cap value were set to the grade cap value and used in the estimation. ▪ The Datamine UNFOLD process was used to unfold the drilling data and allow variography analysis to be performed in the unfolded space. ▪ The generated variography and search parameters were then applied to the estimate. ▪ Separate variography and estimation were performed for Zn%, Pb%, Agppm, Mn%, Fe%, S% and C_tot% (total carbon). ▪ Grade estimation was performed in both unfolded space and using the dynamic anisotropy method. Blocks which were un-estimated in the unfold estimate were populated by estimated values generated from the dynamic anisotropy method. The overall estimation methodology was: – Ordinary kriging (OK) estimate in the unfolded space. – Ordinary kriging and inverse distance squared (ID^2) estimate using the dynamic anisotropy (DA) method. – Unfolded and DA models combined. – Blocks not estimated in the unfolded process were assigned an OK value from the DA model. – If the OK estimate was not available in the DA model (due to lack of drilling data) ID^2 values were assigned to the blocks. ID^2 and unfold estimated values were flagged in the combined model to allow easy identification. – Hard boundary contacts were used to select samples used to estimate blocks. An incremental search ellipse was used with the maximum search radii based on maximum anisotropic variogram ranges. – Parent block size was set at 2.5m x 12.5m x 12.5m with sub-cells x=0.5m, y=0.5m, z=0.5m. Justification of the small sub-cell size was based on the need to have some detailed granularity of the mineralisation domains contacts. – Grade interpolation was based on ordinary kriging (OK) or Inverse Distance Squared (ID^2) for un-estimated OK blocks. – Sub-celled blocks were assigned the same grade as the estimated grade of the parent block. – Block discretization was done at 2 x 4 x 4. – Octant method was applied to the estimate. A minimum of 2 octants was required for the estimate with a minimum of 2 samples per octant and a maximum of 6. – A minimum number of 4 drillholes were used in the estimate. – Minimum number of 8 samples with a maximum of 20 samples was used in the estimate. – A number of block models were generated: unfold grade (with OK), anisotropy grade (with OK and ID^2) and a lithology model. – The final model was assembled by combining these models. Highest priority was given to the unfolded estimate, followed by the anisotropy OK estimate. Un-estimated blocks were assigned an ID^2 estimate from the anisotropy model.
Moisture	▪ Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	▪ Mineral Resource has been reported on a zinc greater than 6% cut-off (unchanged since 2012). ▪ This cut-off represents material that has a reasonable prospect for eventual economic extraction at some point within the next 15 years. ▪ Tabulations and comparisons against the 2012 Mineral Resource have been done at a zinc >0%, >6% and >9% cut-offs. ▪ Grade tonnage curves have been generated for Zn, Pb, Ag and Mn.
Mining Factors or assumptions	▪ No mining factors have been applied to the Mineral Resource. ▪ Underground development is taking place at Dugald River to allow better drilling and mapping access to the mineralisation.
Metallurgical factors or assumptions	▪ Manganese percentage in the zinc concentrate algorithm is calculated by way of the following algorithm: Mn%ZnCon = -0.79857 + (0.09192*Fe%) + (1.57170*Mn%) + (0.76522*C_Tot%) - (0.04902*(Fe%*C_Tot%)) Mn%ZnCon is Mn (%) in zinc concentrate

	C_Tot% = Total Carbon
Environmental factors or assumptions	- No environmental factors or assumptions have been applied to the Mineral Resource. - Dugald River operates under Environmental Authority EPML00731213 issued by the Department of Environment Heritage Protection on 12 August 2012 and amended on 7 June 2013.
Bulk Density	- Bulk density estimated using Inverse Distance Squared (ID^2). - Density estimation constrained within the defined mineralisation domains. - Un-estimated blocks were assigned a density value based on the 2012 Bulk Density Calculation: Bulk Density = (3.8*A/100) + (7.3*B/100) + (4.6*C/100) + (2.573*D/100) where: Sphalerite content A = 1.5*Zn% Galena content B = 1.15*Pb% Pyrrhotite/Pyrite content C = (Fe%-(0.15*Zn%))*1.5 Gangue D = 100-A-B-C SG of sphalerite = 3.8 SG of Galena = 7.3 SG of Pyrrhotite/pyrite = 4.6 SG of gangue = 2.573 Fe content in Sphalerite = 10%
Classification	- Mineral Resource Classification has changed from the 2012 Mineral Resource Model. - 2013 Classification incorporates a combination of Kriging variance (KV), Kriging efficiency (KE), Kriging slope of regression (SOR), drilling density and location of underground development (presence of underground geological mapping). - The CP reviewed the distribution of KV, KE and SOR in long-section view and then generated 3D wireframes to select Measured, Indicated and Inferred blocks. These wireframes also take into consideration the location of the underground development and presence of geological mapping and the 20m x20m underground drilling. - The generation of these wireframes was necessary to remove the "spotty dog" in the classification of the 2013 Mineral Resource. - Drilling density used for Mineral Resource classification is: – Measured <=20m x 20m – Indicated <=100m x 100m – Inferred >100m x 100m - Figure 85 shows the Dugald River Mineral Resource block model with the Measured, Indicated and Inferred wireframes used in selecting the Mineral Resource classification. **Figure 85 Long-section looking east Mineral Resource block model** 
Audits or reviews	- No external audits or reviews have been carried on the current 2013 Mineral Resource estimate. - The 2013 MMG IPR (Independent Peer Review) was focused on the 2012 Mineral Resource. Key findings of the IPR were: – The comparison and reconciliation of the 2010 and 2012 data and models has demonstrated significant change in

	local Mineral Resource estimation where the drill data is closed down to 25m grid spacing. As part of the 2013 Mineral Resource update it is recommended MMG review the criteria used for Mineral Resource classification. – Underground geological mapping should be converted into "ore body" and structural knowledge to be used in the 2013 Mineral Resource. – The Competent Person should review the QA/QC procedures and results to ensure the reliability of the methods and results prior to endorsing the work in the Mineral Resource CP Report. ▪ A 2012 Mineral Resource review by Lewis Mineral Resource Consultants Pty Ltd (LMRC) identified a number of areas of the 2012 estimate that needed to be reviewed: – Selection of the high grade domain should be done using a more natural grade break and not the zinc 9% used in the 2012 model. This natural break should be determined by looking at both the mineralisation logging and zinc grade distribution down the drillhole. Figure 86 shows an example on how this boundary should be determined. **Figure 86 Example of the determination of the high-grade domain**  – Unfolding of the drilling data should be done to allow more robust variography to be performed in the unfolded space. – Density should be estimated. Blocks that have un-estimated density values should be assigned a value based on the 2012 equation. All of these recommendations have been incorporated in the 2013 Mineral Resource estimate.
Discussion of relative accuracy / confidence	▪ Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support. ▪ The Measured and Indicated Mineral Resource category have been adjusted to align classification of the Mineral Resource to take into account current mineralisation knowledge obtained from recent underground drilling and geological mapping of underground development.

9.5 Ore Reserves – Dugald River

The June 2011 Ore Reserves statement was the first statement to be released for the Dugald River deposit and was determined as a part of the Definitive Feasibility Study completed in December 2008. The June 2012 Ore Reserves statement was an update, based on the 2009 Mineral Resource model, and re-evaluating some of the dilution criteria calculations.

Further detailed geotechnical investigations have been undertaken since the June 2012 Ore Reserves statement and have resulted in a considerably different view of the geotechnical stability associated with previous mine designs delivering into the Ore Reserves evaluation.

The 2013 Ore Reserves are based on the available detailed design for a Sub-Level Open Stoping (SLOS) option using 20 metre development spacing and 15 metre stope strike lengths. Further studies on mining methods and dimensions are currently underway and the chosen mining configuration is likely to change.

9.5.1 Results

The June 2013 Dugald River Ore Reserves are summarised in Table 130.

Table 130 2013 Dugald River Ore Reserves tonnage and grade (as at 30 June 2013)

Dugald River Ore Reserves

					Contained Metal		
	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
Proved							
Probable	24	12.5	2.0	41	3,100	500	32
Total Ore Reserves	**24**	**12.5**	**2.0**	**41**	**3,100**	**500**	**32**

Ore Reserves are generally rounded and reported to 2 significant figures to reflect confidence in estimates. Totals may differ due to rounding. Contained metal does not imply recoverable metal.

Details of relevant modifying factors used in estimating Ore Reserves are given in the Technical Appendix published on the MMG website.

Competent Person:

Julian Poniewierski (Member of AusIMM (CP), employee of MMG)

The 2012 Ore Reserves were based on stopes designed in 2008 at the time of the 2008 Feasibility Study to a 10.8% zinc equivalent. These stope designs were based on a 25 metre development sub-level spacing and a 25 metre hangingwall strike length. The 2013 Ore Reserves are based on a redesign of the stopes to a 20 metre development sub-level spacing and a 15 metre hangingwall strike length – resulting from a geotechnical reassessment of stope hangingwall span stability and dilution.

After assignment of dilution, any stopes not exceeding a 2013 Long Term pricing Net Smelter Return (NSR) of A$215/t were excluded from the Ore Reserves. The NSR was calculated using prices discussed in Section 2.1. Approximately 22% of the Ore Reserves is from development (at an NSR cut-off value of A$85/t) – the remainder (78%) is from stoping.

9.5.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Ore Reserves statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Julian Poniewierski, confirm that I am the Competent Person for the Dugald River Ore Reserves section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Dugald River Ore Reserves section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Limited since August 2012.

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest. Specifically, I currently have a grant to options vesting in 2016 for 767,000 MMG Limited shares at an exercise price of $HKD 2.62 (price at 16 October 2013 was $HKD 1.72).

I verify that the Dugald River Ore Reserves section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in the supporting documentation relating to Ore Reserves as compiled by various MMG staff and consultants under the supervision of Julian Poniewierski.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Dugald River Ore Reserves - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Julian Poniewierski – 26/11/13

Mauro Bassotti – (Witness)

9.5.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 131.

Table 131 Contributing Experts – Dugald River Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Mauro Bassotti, Senior Resource Geologist MMG Ltd (Melbourne)	Geological Mineral Resources
AMC Consultants Pty Ltd (Brisbane)	Costs Input
“*2008 Feasibility Study*”	Metallurgy
Max Lee (MMG), Geotechnical Specialist, MMG Ltd (Melbourne)	Geotechnical
AMC Consultants Pty Ltd (Brisbane)	Geotechnical
AMC Consultants Pty Ltd (Brisbane)	Mining
Gavin Marre, Senior Business Analyst, MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

9.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information is provided to comply with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the code.

9.6.1 Mine Design

The orebody is split into a North and South mine, due to its 2km strike length and a low-grade zone in the centre of the orebody. The selected mining method is Sublevel Open Stoping (SLOS).

The North mine is narrow (average ~5m true width) and sub-vertical. The South mine is wider than the North mine with a flexural zone in the centre. The South mine is narrow and steep in the upper zone (~top 200m from surface) and lower zone (~below 700m from surface). The central zone is flatter and thicker than the upper and lower zones.

AMC Consultants Pty Ltd (AMC) were contracted during the first half of 2013 to undertake a Mining Methods Review and to produce an update to the life-of-mine plan (LOMP) for Dugald River based on stope dimensions for stoping areas below the 200m level of 20m sub-level spacing and 15m hangingwall strike span. Areas already developed at a 25m level spacing will be stoped at that sub-level spacing. The Ore Reserves are based on this updated design (which included economic Inferred Mineral Resources that have not been included in the Ore Reserves).

The stopes were broken into the following categories:

- Longitudinal SLOS, for any stopes less than 8m wide horizontally.
- Transverse SLOS, made up of 15m strike SLOS mined full width of the orebody. Continuous retreat transverse SLOS has been assumed.
- Crown SLOS, for the top level of each panel where stoping occurs directly below a previous mined area.

The ore and waste will be hauled using trucks.

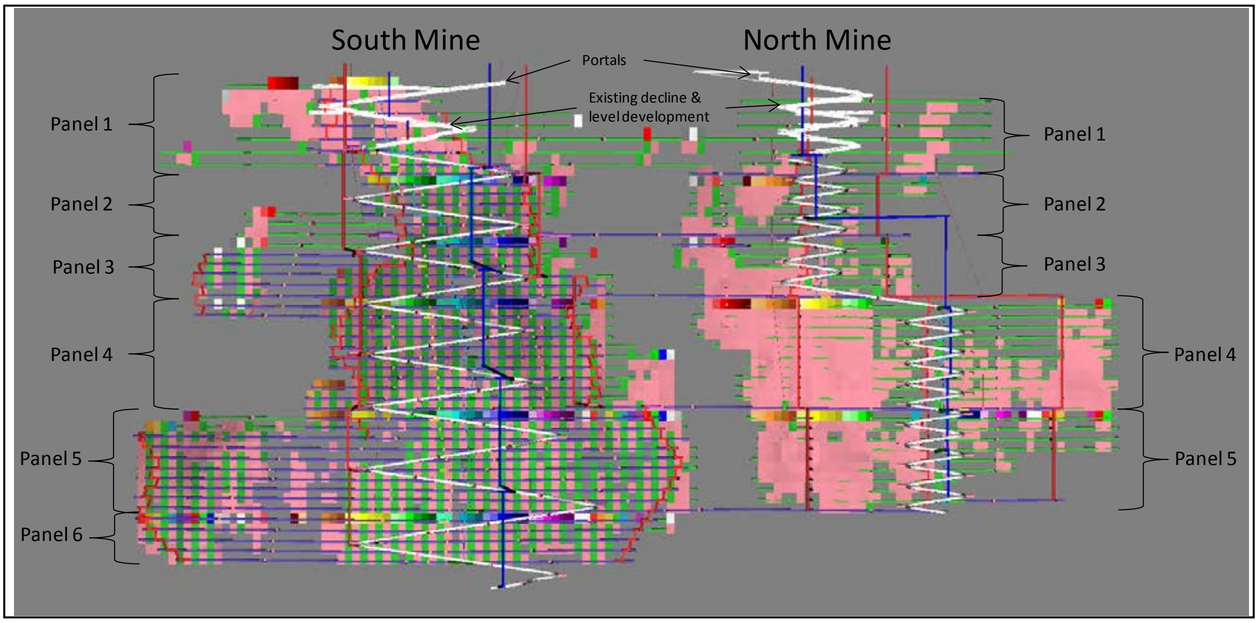
Stopes will be filled with paste fill to allow complete orebody extraction.

A minimum stoping width of 2.5m for SLOS was adopted.

Both mines were divided into 140m high panels, or working areas, based on the orebody properties and expected decline advance rate. At the base of each panel an allocation for a footwall drive was included. It is proposed these drives would be used for services and potentially a trucking route for one-way haulage. For transverse stopes, footwall drives and waste crosscuts were included on each sublevel.

An overview of mine design layout is shown in Figure 87.

Figure 87 Stope and development design – looking west



The average annual production rate for the LOMP (including the economic inferred Mineral Resource mining inventory) is approximately 1.6Mt over 15 years of steady-state production. The total mine life is 21 years. The annual ore production and NSR values are shown in Figure 88.

Figure 88 Annual ore production and NSR values

9.6.2 Geotechnical Parameters

The most significant matter to affect a change to the reported Ore Reserves is as a result of a re-evaluation of the geotechnical parameters used in the analysis of stope stability and stope dilution. This has been a major focus of an ongoing technical evaluation program that was initiated in November 2012. The technical evaluation program involved structural re-analysis of core from 668 available surface diamond drillholes, and construction of a geotechnical model using a re-interpretation of the data with the knowledge gained by the underground development exposures.

This technical evaluation program has indicated that ground conditions at the Dugald River project are likely to be less favorable than previously assumed in the 2008 Feasibility Study, potentially causing stability problems and more dilution in the proposed stopes. Specifically it has been noted that the Dugald ore zones and hangingwall have been highly affected by the presence of various shear zones including a pervasive hangingwall shear zone (HWSZ). These shear zones vary from 10cm to in excess of 10m in thickness. These ground conditions are reflected by three key geotechnical parameters:

- poor RQD/fractures/slickensides, resulting in low N' (N-Prime)[8] in drill core;
- poor drilling recovery, sometimes losing up to 5m core sections, with zero core recovered; and
- underground failures in development that have been experienced (prior to adoption of development in-cycle shotcreting).

Using the new geotechnical model a detailed engineering based evaluation of stope stability and dilution parameters was undertaken. The method adopted for this evaluation was based on a detailed dilution and stability study undertaken at the George Fisher mine – a shale hosted zinc-lead orebody with some similar characteristics to Dugald River orebody. This particular study was published as a PhD thesis by Geoff Capes in 2009[9].

The geotechnical parameters used for assessment of stope stability and dilution were based on rock quality (N') and geotechnical domain thicknesses. A formula was used for determining the hangingwall dilution, based on an equivalent length of overbreak sloughing (ELOS) for a 15m stope span and the hangingwall N'.

$$\text{For 15m stope span: ELOS (m)} = 2.20 \times \text{EXP}(-0.36 \times N') \ \ldots\ldots\ 4.2 > N' > 1.0$$

For N' <1.0 the span is unstable and it is expected it will collapse. For N' >4.2 the span will be stable and only 0.5 metres of hangingwall dilution is applied. In between these values of N' the following process was used to create the mining inventory (from which the Ore Reserves are extracted):

(i) Based on the N' in the geotechnical block model, the ELOS was estimated and depth of failure (DoF) for the high grade lode and the high grade lode to the HWSZ were determined using DoF = 1.5 x ELOS + 1.

(ii) If the hangingwall shear zone was thicker than 1.5m (irrespective of the location) it could not be allowed to be penetrated by the estimated DoF due to the potential for the HWSZ to unravel and lead to unacceptable failure propagation upwards into a significant number of higher levels.

(iii) If the thickness of the zone between the high grade ore and HWSZ was >DoF and N'>1, then the stope would be stable, and there would be no requirement to leave an ore skin on the HW side of the high grade lode. This meant that HWSZ would not be exposed, and therefore the grades for the zone between the high grade ore and HWSZ were used for dilution calculations.

(iv) If N'<1 for the zone between the high grade ore lode and HWSZ (irrespective of how thick), then the stope would be unstable and there was a requirement for an ore skin to be applied, based on the DoF of the high-grade lode. The high-grade lode grades would then be used for dilution calculations.

(v) If the thickness of the zone between the high grade ore lode and HWSZ was <DoF and N'>1, then the stope would be unstable and there was a requirement for an ore skin (based on a combination of the zone between the high grade ore lode and HWSZ and the high-grade lode itself). Based on the DoF of the zone between the high grade ore lode and HWSZ, the ore skin thickness was determined to be DoF of the high-grade lode minus thickness of the zone between the high grade ore lode and HWSZ. The dilution tonnes and grade were attributed as a proportion of ore skin the high-grade lode and the zone between the high grade ore lode and HWSZ.

---

[8] *Mathews et al. (1981) developed an empirical relationship between the stability number N, and the shape factor, S, of a stope surface. N' (N-Prime) is modified version of that stability number*

[9] *Open Stope Hangingwall Design Based on General and Detailed Data Collection in Rock Masses with Unfavourable Hangingwall Conditions.* , PhD Thesis, University of Saskatchewan, Geoff Capes, April 2009.

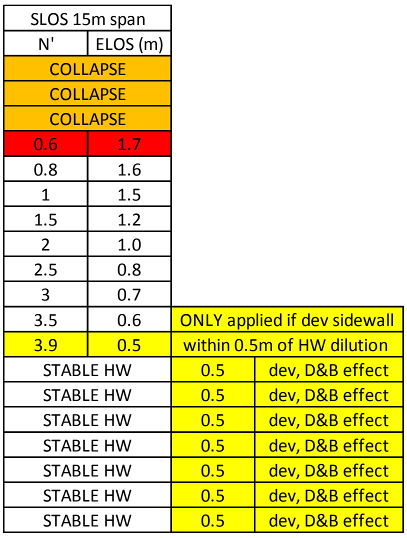
(vi) If both the HWSZ and the zone between the high grade ore lode and HWSZ were less than 1.5m thick, then the development and stoping was located to include the HWSZ.

It was assumed the footwall (FW) dilution will be 0.5m ELOS, consistent with the 2008 Feasibility Study.

The expected ELOS dilution results for calculating dilution for 15m stope spans based on the hangingwall N′ value are shown below in Figure 89.

The resulting range of dilutions determined for the Ore Reserves stopes as a percentage of the in-situ stope tonnage (less development tonnes) is shown in Figure 90. The distribution of the dilution percentage in terms of the number of stopes is shown in Figure 91. The range of stope tonnages in various dilution percentage ranges is summarized in Table 132, with the overall tonnes-weighted average dilution percentage being 22%.

Figure 89 Stope dilution ELOS as a function of modified Mathews Stability Number N′



SLOS 15m span			
N'	ELOS (m)		
COLLAPSE			
COLLAPSE			
COLLAPSE			
0.6	1.7		
0.8	1.6		
1	1.5		
1.5	1.2		
2	1.0		
2.5	0.8		
3	0.7		
3.5	0.6	ONLY applied if dev sidewall	
3.9	0.5	within 0.5m of HW dilution	
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect
STABLE HW		0.5	dev, D&B effect

Figure 90 Stope dilution percentage as a function of in-situ stope tonnage

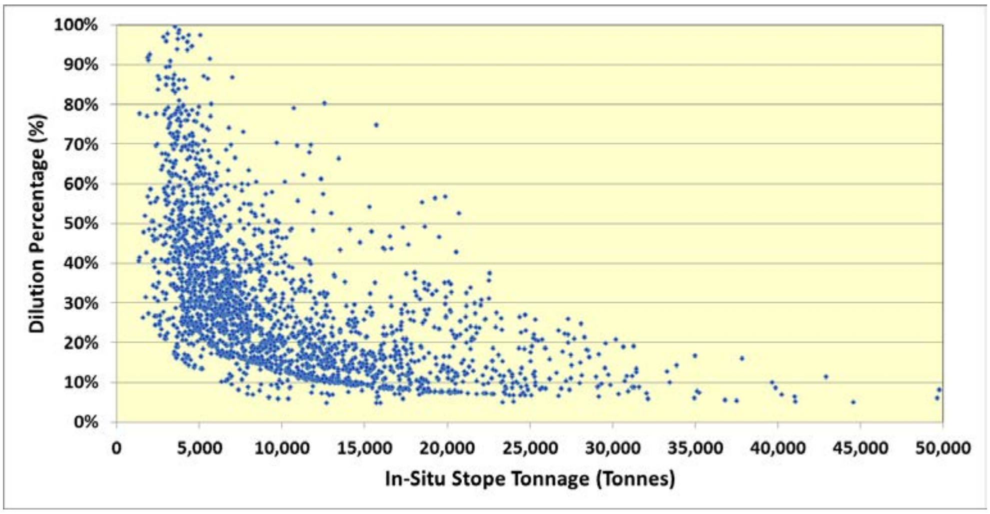


Figure 91 Stope frequency distribution of dilution percentage

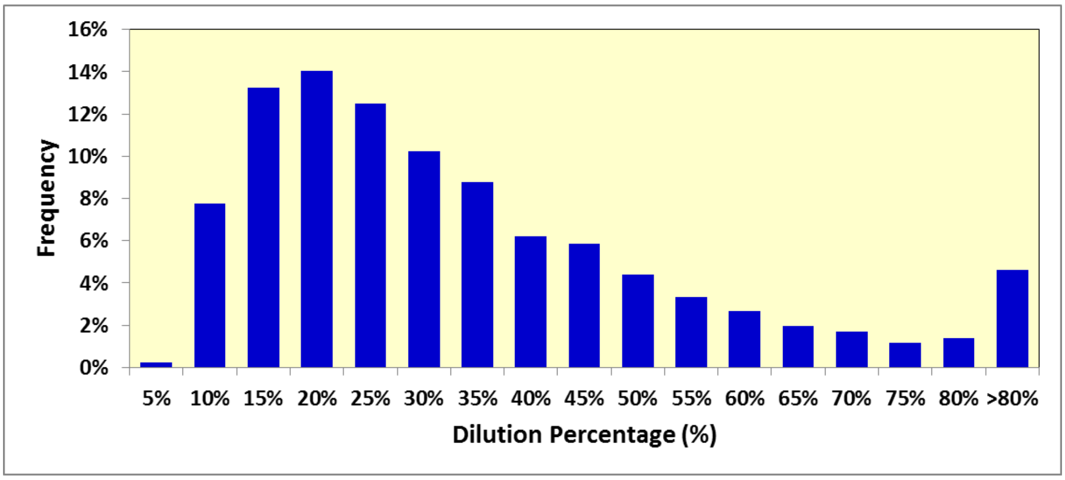


Table 132 Tonnes weighted distribution of stope dilution percentage in the Ore Reserves

Dilution %	Stope Tonnage %
<10%	16%
10-15%	20%
15-20%	17%
20-25%	13%
25-30%	9%
30-35%	7%
35-40%	4%
40-45%	4%
45-50%	3%
≥50%	8%
Tonnes-Weighted Average	22%

9.6.3 Mill Design

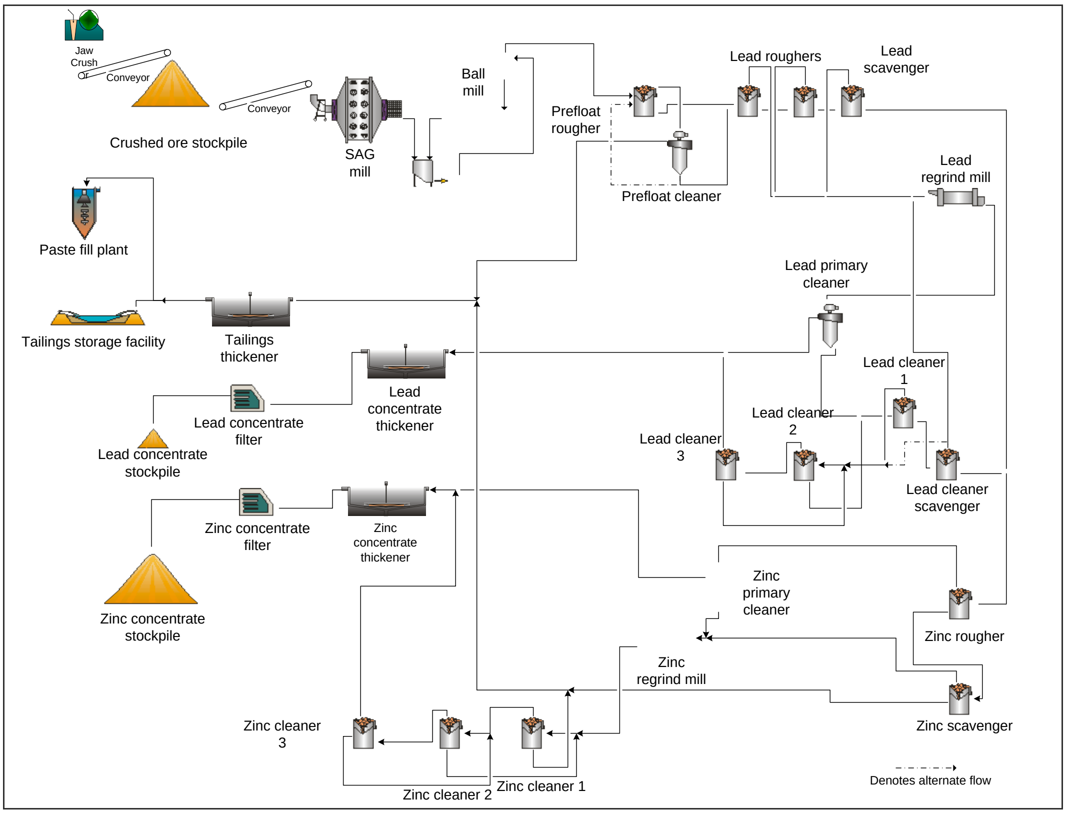
The mill has yet to be constructed and is in engineering design phase.

The process plant design was based on the results of comprehensive metallurgical bench-scale test programs, by Pasminco in 1998-2001 and OZ Minerals in 2008, which were conducted on composite samples representing the main ore types to be processed (Massive Breccia, Slate Breccia and Banded) as well as the then current life-of-mine blends. The samples included footwall and hangingwall dilution material. Further testwork was carried out in 2010 by MMG as an extension of the OZ Minerals 2008 program.

The proposed process plant design is based on a metallurgical flowsheet with unit operations and equipment that are well proven in base metals flotation plants, particularly in the Mt. Isa inlier. It uses a conventional lead-zinc processing flowsheet and industry standard equipment to produce separate lead concentrate and zinc concentrate.

A schematic flowsheet is shown in Figure 92.

Figure 92 Dugald River Processing Plant Flowsheet



The processing costs used in the Ore Reserves determination were based on estimates prepared by Ausenco, with MMG providing the base rates for the labour, utilities and consumables costs to Ausenco.

Processing (Metallurgical) Recovery Factors

The metallurgical recovery factors used in the Ore Reserves determination are shown in Table 133. These are based on the test work dating back to the 2008 Feasibility Study.

Table 133 Metallurgical recoveries and concentrate grades used

	Metal	Mill Recoveries	Concentrate Grade
Zinc Concentrate	Zn	87.8%	54%
	Pb	0.0%	–
	Ag	0.0%	–
Lead Concentrate	Zn	1.0%	–
	Pb	75.0%	70%
	Ag	35.0%	–

9.6.4 Realised Revenue Factors (Net Smelter Return)

The realised revenue from the ore is expressed using a calculated Net Smelter Return (NSR). In this case the NSR does not include Royalty – in alignment with the commercial group definition of NSR[10].

Commodity price and exchange rates used were the long-term forecasts stated in Section 2.1.

The realisation costs for zinc concentrates are shown in Table 134, and the realisation costs for lead concentrates are shown in Table 135.

The NSR (expressed as A$/t of ore) was calculated for a range of zinc, lead and silver grade combinations and was able to be simplified back to a value related to only the ore grades present; the equation being:

$$\text{NSR} = 1{,}699 \times \text{Zn}(\%) + 1{,}729 \times \text{Pb}(\%) + 0.26 \times \text{Ag}(g/t)$$

---

[10] An alternative form of NSR that includes Royalty, referred to as NSRAR (NSR after Royalty), was subsequent to this work adopted as the future technical standard for the group, and is used in other sites reported in this document.

Table 134 Dugald River NSR inputs for zinc concentrate realisation costs

Zinc		
Metal Paid - Zn (total)	85%	%
Minimum Deduction - Zn	8%	% dry
Base Treatment Charge - Zn	200	US$ / dmt con
TC Basis Price - Zn	2,000	US$ / t Zn
TC Escalator - Zn	0.030	US$ / (US$ / t)
TC Deflator - Zn	0.020	US$ / (US$ / t)
Silver		
Deduct - Ag	93.3	g / dmt con
Metal Paid - Ag (remainder)	65.0%	%
Penalties (Zn-Con.)		
Penalties - Zn Concentrate - Fe	1.50	US$/dmt/%Fe > Penalty Trigger
Penalties - Zn Concentrate - Fe Trigger Level	8.0%	%Fe
Penalties - Zn Concentrate – Mn	5.00	US$ / dmt
Freight, Sampling and Insurance		
Road Freight & Logistics - Export	47	A$ / wmt con
Road Freight & Logistics - Townsville Refinery	71	A$ / wmt con
Sea Freight	45	US$ / wmt con

Table 135 Dugald River NSR inputs for lead concentrate realisation costs

Lead		
Metal Paid - Pb (total)	95%	%
Minimum Deduction - Pb	3%	% dry
Base Treatment Charge - Zn	175	US$ / dmt con
Silver		
Minimum Deduction - Ag	50	g / dmt con
Metal Paid - Ag (remainder)	95%	%
Refining Charge - Ag	0.31	US$/Oz payable
Penalties (Pb-Con.)		
No Penalties are Assumed		
Freight, Sampling and Insurance		
Road Freight & Logistics - Export	47	A$ / wmt con
Road Freight & Logistics – Local Aust.	71	A$ / wmt con
Sea Freight	45	US$ / wmt con

From test work undertaken for the 2008 Feasibility Study it was determined that over the life of the project silver grades in zinc concentrate range from 33g/t to 88g/t, and average about 65g/t and at no time exceed the payment threshold of 93g/t (3oz/t).

Manganese in concentrate is determined by the linear relationship Mn-*in-Con=1.617 * Mn% in Mineral Resource + 0.51*.

Iron in concentrate is determined by the linear relationship: *Fe-in-Con = -0.5915 x 54%Zn Grade + 41.7%*

Concentrate moisture estimates and concentrate transport loss assumptions are given in Table 136.

Table 136 Concentrate moisture and transport loss assumptions

Concentrate	Moisture	Transport Loss
Zinc	8.9%	0.25%
Lead	9.5%	0.25%

9.6.5 Royalties

Queensland State Government royalties payable are prescribed by the Minerals Resources Regulation 2013 and are based on a variable *ad valorem* rate between 2.5% to 5.0% depending on metal prices, advised quarterly and calculated on payable metal. They are published by the Queensland Government Department of Mines and Energy and can be found at the web-site of the "Office of State Revenue":

https://www.osr.qld.gov.au/royalties/rates.shtml.

For the long term prices used in the Ore Reserves estimation at the time of evaluation, the relevant rates were 3.22% for zinc, 5.00% for lead, and 5.00% for silver.

A royalty discount applies for base minerals processed within Queensland to a particular metal content, as prescribed by Section 51 of the Mineral Resources Regulation 2013. This discount is 35% for zinc and 25% for lead. Economic evaluations of Dugald River have assumed that 22.5% of concentrates will be sold locally in Queensland.

9.6.6 Mining Costs and Cut-Off Value

Mining costs for the 20m x 15m SLOS mining method that is the basis for these Ore Reserves are provided in Table 137 (Approximate in that they are a combination of fixed and variable costs that differ on an annual basis).

AMC has used the cost modelling tool, MCost, developed by AMC, to estimate costs. MCost is a first-principles cost modelling tool that allows users to transparently see the assumptions upon which cost estimates are built, and allows further development and calibration of the model around site experience.

The NSR cut-off value used for stoping based on these costs was $215/tonne as per Table 138, allowing for approximately a $50/tonne margin above break-even value – a margin value indicative of that required for the best NPV from previous modelling work. For development, the cut-off value used for the Life of Mine Plan (LOMP) was an NSR value of $85 per tonne.

Table 137 Approximate mine operating costs

Operating Cost Component	Unit Cost ($/t ore)
General	25.3
Development	13.9
Drill and Blast – Production	11.3
Pastefill	10.1
Trucking – Production	7.8
Bogging - Production	3.5
Trucking – Development	2.1
Total Operating Cost ($/t)	**74.0**

Table 138 Costs for cut-off value calculation

Element	Value
Prod Rate (Mtpa)	1.6
Mining Cost ($/t)	74
G&A Cost ($/t)	28
Milling Cost ($/t)	64
Margin ($/t)	50
Cut-off Value ($/t)	**216 (215 Used)**

9.6.7 Mining Factors and Assumptions

Mining Dilution and Recovery

Two aspects of dilution were considered, fill dilution and hangingwall dilution.

The fill dilution and stope production recoveries used are shown in Table 139.

Table 139 20m ×15m SLOS fill and recovery factors

		Fill Dilution		Stope		Modified
STOPE	TYPE	Floor (m)	Walls (m)	RECOVERY	PILLAR (m)	RECOVERY
Crown	C	0.15	0.00	85%	5	57%
Longitudinal	L	0.15	0.30	95%	0	95%
Transverse	T	0.15	0.50	95%	0	95%

The hangingwall dilution was calculated for each stope based on the geotechnical conditions and thicknesses of the hangingwall materials. The method used was based on the results of dilution analysis undertaken at George

Fisher mine (a similar shale hosted Zn-Pb orebody also located within the Mount Isa Inlier) as part of a PhD thesis by Geoff Capes submitted in 2009[11]. The method is outlined in Section 9.6.2.

Based on these geotechnical requirements a mining inventory and development physicals were calculated; and the outlying stopes and associated levels were evaluated to determine the total number of economic stopes above cut-off NSR value. The Ore Reserves comprise the Measured and Indicated stoping tonnes from this mining inventory plus the Measured and Indicated development tonnage associated with the stope areas developed.

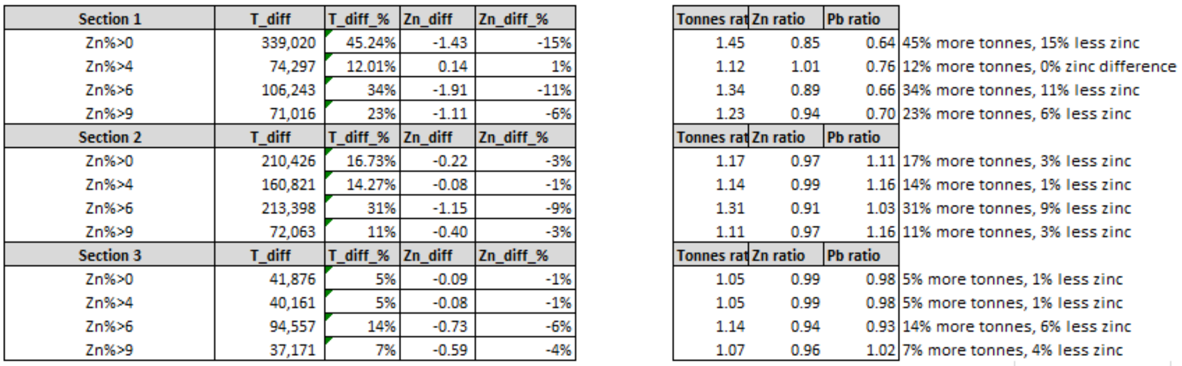
Reconciliation

No production has yet been undertaken to reconcile against.

As part of the Mineral Resource improvement a program of infill drilling is in place, and during 2012 and 2013 a number of in-fill diamond drillholes were drilled and assayed.

Reconciliation was undertaken to examine the change to the Mineral Resource Model due to the extra in-fill drilling in the sections covered by the extra drilling. A summary of the resulting reconciliation factors is given in Table 140.

Table 140 **Tonnes and grades reconciliation factors between in-fill drilled sections of the 2010 and 2012 Mineral Resource models**



Section 1	T_diff	T_diff_%	Zn_diff	Zn_diff_%	Tonnes rat	Zn ratio	Pb ratio	
Zn%>0	339,020	45.24%	-1.43	-15%	1.45	0.85	0.64	45% more tonnes, 15% less zinc
Zn%>4	74,297	12.01%	0.14	1%	1.12	1.01	0.76	12% more tonnes, 0% zinc difference
Zn%>6	106,243	34%	-1.91	-11%	1.34	0.89	0.66	34% more tonnes, 11% less zinc
Zn%>9	71,016	23%	-1.11	-6%	1.23	0.94	0.70	23% more tonnes, 6% less zinc
Section 2	**T_diff**	**T_diff_%**	**Zn_diff**	**Zn_diff_%**	**Tonnes rat**	**Zn ratio**	**Pb ratio**	
Zn%>0	210,426	16.73%	-0.22	-3%	1.17	0.97	1.11	17% more tonnes, 3% less zinc
Zn%>4	160,821	14.27%	-0.08	-1%	1.14	0.99	1.16	14% more tonnes, 1% less zinc
Zn%>6	213,398	31%	-1.15	-9%	1.31	0.91	1.03	31% more tonnes, 9% less zinc
Zn%>9	72,063	11%	-0.40	-3%	1.11	0.97	1.16	11% more tonnes, 3% less zinc
Section 3	**T_diff**	**T_diff_%**	**Zn_diff**	**Zn_diff_%**	**Tonnes rat**	**Zn ratio**	**Pb ratio**	
Zn%>0	41,876	5%	-0.09	-1%	1.05	0.99	0.98	5% more tonnes, 1% less zinc
Zn%>4	40,161	5%	-0.08	-1%	1.05	0.99	0.98	5% more tonnes, 1% less zinc
Zn%>6	94,557	14%	-0.73	-6%	1.14	0.94	0.93	14% more tonnes, 6% less zinc
Zn%>9	37,171	7%	-0.59	-4%	1.07	0.96	1.02	7% more tonnes, 4% less zinc

9.6.8 Infrastructure

Underground Infrastructure

The underground mine is accessed via two declines. The mine is split into two parts – North and South and thus it has two separate declines for the UG access.

The construction of the portal box-cuts was commenced in October 2011, with the first firings of the two exploration declines occurring in early February 2012. As at 30 June 13 there was 1,722 metres of decline in place. In addition there was 5,503 metres of lateral development in place.

Currently two ventilation shafts are in place, the Southern Fresh Air Raise (FAR) – at 3.5 metres diameter and 143 metres depth; and the Northern FAR at 3.5 metres diameter and 172 metres depth.

Two escape raises are in place: South 50 (1.8m dia) – 40m and North 75 (1.8m dia) – 56m.

The expected total underground development structure for the Life-of-Mine is summarised in Table 141.

---

[11] *Open Stope Hangingwall Design Based on General and Detailed Data Collection in Rock Masses with Unfavourable Hangingwall Conditions.* , PhD Thesis, University of Saskatchewan, Geoff Capes, April 2009.

Table 141 Underground development infrastructure

Description	Length	Tonnes/Material
Decline	13km	1.1Mt of waste
Access and ancillary horizontal development	30km	2.5Mt of waste
Vertical development	8km	0.4Mt of waste
Footwall drives	31km	2.6Mt of waste
Cross-cuts	53km	4.4Mt of waste
Ore development	60km	4.8Mt of ore

Surface Infrastructure

Existing surface infrastructure includes: a gravel access road; a temporary camp for construction phase; a temporary contractors mobile equipment facility; ore and waste stockpile pads; contaminated run-off water storage dams; a core shed; a fuel farm and gensets for power generation; bore water fields; and office buildings including emergency medical facilities.

Major infrastructure yet to be built includes: a permanent camp; a processing plant; a tailings storage facility; a permanent mobile equipment workshop; recreational facilities; power supply lines; and raw water supply pipe line.

Northwest Queensland is not connected to the state electricity grid. A Queensland semi-government owned electrical power generation company owns and operates the Mica Creek gas fired power station on the southern outskirts of Mount Isa. Plans are to connect to the Mount Isa grid in the future. Power is currently generated on-site using diesel gensets.

The main source of raw water will be Lake Julius, with average demand estimated to be 692 ML/y assuming a recovery of approximately 50% decant water recovered from the TSF. The total water input to the processing plant's grinding and flotation circuits will be approximately 1,170 m^3/h including water returning from the thickeners, TSF decant, the raw water circuit, gland water and water contained in the ore and reagents. The pipeline connecting the Lake Julius–Ernest Henry pipeline to the Dugald River site will be sized for the total plant demand.

Scheduled regular commercial air services operate between Brisbane and Mt. Isa with at least one daily jet service to Brisbane and other services operate to Townsville. Cloncurry airport is used by commuter aircraft operating to Townsville, Cairns and Brisbane and serves as the fly-in–fly-out (FIFO) airport for Glencore Xstrata Limited's Ernest Henry mine and also has commercial services to Brisbane and Townsville.

The 11km access from the Burke Developmental Road is currently being upgraded and will include an emergency airstrip for medical and emergency evacuation use.

9.6.9 Ore Reserves Assessment and Reporting Criteria Table

The remaining assessment and reporting criteria required by the 2012 JORC Code – consistent with "Table 1 Section 4" of the code are given in the following Table 142. Each of the items in this table has been summarised as the basis for the assessment of overall Ore Reserves risk in the table below, with each of the risks related to confidence and/or accuracy of the various inputs into the Ore Reserves qualitatively assessed.

Table 142 JORC Code Ore Reserves Assessment and Reporting Criteria for Dugald River Project 2013 Ore Reserves

Assessment Criteria	Risk Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	Medium	The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The Mineral Resource model used was the MMG December 2012 Mineral Resource model ("*mrl_5_20121217.dm*"). Risks associated with the model are related to ore body complexity seen underground but not reflected in the Mineral Resource model due to the spacing of the drillholes that inform the model.
Classification	High	Ore Reserves are all reported as Probable. Due to uncertainties with a number of the modifying factors, no Proved Ore Reserves have been declared. Only Measured and Indicated Mineral Resources have been used to inform the Ore Reserves. No Inferred Mineral Resources are included in the Ore Reserves.

Assessment Criteria	Risk Assessment	Commentary
Site visits	-	The Competent Person undertook two site visits during 2013; 12-14 March 2013, and 13-16 September 2013.
Study status	Medium	The initial mine design was detailed in a Feasibility Study undertaken in 2008 and released in January 2009. With physical access into the orebody occurring in 2012 it was recognised that the orebody was more complex than modeled from drilling results and that the geotechnical conditions of the orebody hangingwall were more challenging for dilution control than assumed in the 2008 Feasibility Study. In November 2012 a major geotechnical study was commenced involving re-examination and re-logging of all diamond drill core and re-analysis of the geotechnical parameters of the ore-zone and hangingwall zones. Results of this geotechnical re-analysis were fed back into a number of conceptual level studies for various Sub-Level Open Stoping layouts (various development level spacing and stope strike span lengths). Detailed updated design work including scheduling and cost modeling was undertaken by AMC Consultants Pty Ltd for one option that was felt to be technically viable from a stope stability viewpoint: 20m development level spacing x 15m stope strike length. This detailed design was used as the basis for this Ore Reserves statement being the only scenario option for which sufficient detail is available to support an Ore Reserves. Further studies on mining methods and dimensions are currently underway and the chosen mining configuration is likely to change.
Cut-off parameters	Medium	See Section 9.6.6 for cut-off value discussion.
Mining factors or assumptions	High	Mine design parameters are discussed in detail in Section 9.6.1. Geotechnical parameters, specifically the dilution, are discussed in detail in Section 9.6.2. Other mining factors are discussed in Section 9.6.7.
Metallurgical factors or assumptions	High	See Section 0 for details.
Environmental	Low	Dominant vegetation comprises remnant woodlands and there are no major watercourses on the site however there are several minor ephemeral tributaries.
Infrastructure	Low	See Section 9.6.8 for details.
Costs	Medium	See Section 9.6.6 for details.
Revenue factors	Medium	See Section 9.6.4 for details.
Market assessment	Medium	See Section 2.2 for details. There is a concern with potential marketability of some of the Dugald product due to manganese content.
Economic	High	Current economic modelling of the current Ore Reserves shows positive annual operating costs. However repayment of expected invested capital is only possible on an undiscounted cash flow basis.

Assessment Criteria	Risk Assessment	Commentary
Social	Low	The nearest major population centre to the project is Cloncurry with a population of approximately 4,000 and the largest employers are mining, mining related services and grazing. In terms of Native Title, the Kalkadoon # 4 People filed a claim in December 2005 covering an area which includes the project area, water pipeline corridor and part of the power line corridor. This claim over 40,000 square kilometres of land was granted in 2011. MMG has concluded a project agreement with the Kalkadoon People dated 6 April 2009. Under this agreement, the claimant group for the Kalkadoon are contractually required to enter into an s31 Native Title Agreement pursuant to the Native Title Act 1993. The agreement sets out the compensation payments and MMG's obligations for training, employment and business development opportunities if/when the project is commissioned. MMG has developed an excellent working relationship with the Kalkadoon claimant group. An official 'Welcome to Country' ceremony was held for MMG in late March 2012. MMG has registered an indigenous Cultural Heritage Management Plan (CHMP) which covers the entire project area and has undertaken all necessary surveys and clearances for all ground disturbing work undertaken on site to date without any issues or complications. The CHMP was developed in consultation with the Kalkadoon # 4 People.
Audit or Reviews	Low	No external or internal audits were undertaken. New personnel in the company have been heavily involved in reviewing the project along with further studies by AMC Consultants Pty Ltd. An Independent Peer Review was undertaken on the whole project rather than the Ore Reserves *per se*.
Discussion of relative accuracy/ confidence	-	A number of key high risk factors that affect the project remain, and are indicated in the "Risk Assessment" column of this table.
Additional Factors believed to be relevant but not specifically listed by the JORC Code Table 1 Section 4		
Topography	Low	The topography of the site is undulating with a dominant ridgeline running through the central portion of the project area
Climate	Low	The hottest months are November to January when the mean monthly maximum ranges from 37ºC to 38ºC. The coolest months are June to August when the mean monthly minimum ranges from 10ºC to 12ºC. The region experiences a distinctive wet season between November and April. January and February exhibit the highest mean monthly rainfall, averaging 140mm and 118mm respectively. The driest month of the year is July, recording an average of just 3.5mm with less than one day of rain for the month. Winds are predominantly from the southeast. Maximum wind gusts of up to 145km/h have been recorded.
Hydrogeological Parameters	Low	Based on hydrogeology assessments, the mine has been assumed to be relatively dry. Experience to date supports this supposition.
Waste Storage (Including Tails Storage)	Low	Underground waste will largely be used as fill underground. Temporary surface storage facilities have been built that include drainage and sediment controls. A tailings storage facility (TSF) has been designed and is located in a relatively long narrow valley on the western edge of the project area in the foothills of the Knapdale Ranges. The proposed tailing containment is almost entirely provided by the valley topography and the initial retaining structure will consist of a single geotextile lined engineered embankment on the western side. The main embankment will be augmented during the life of the project by two downstream lifts in operating years 5 and 14. Smaller embankments will be required at the northern extremity in year 14 of operations. Stability studies were conducted to ensure the design of proposed embankment was such that the

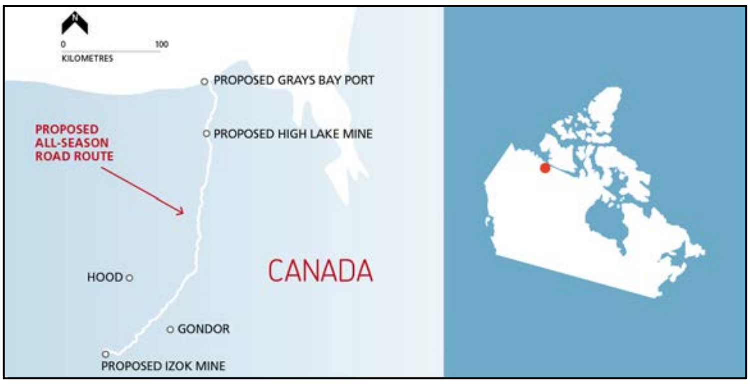
Assessment Criteria	Risk Assessment	Commentary
		integrity of the structure would be preserved under static and seismic loading conditions and that factors of safety met or exceeded the allowable factors.

10. IZOK LAKE

10.1 Introduction and setting

The Izok deposit is located in the West Kitikmeot Region of Nunavut Territory in the Canadian Arctic (Figure 93).

Figure 93 Izok Deposit location



10.2 Geological Setting

The Izok volcanogenic massive sulphide (VMS) deposit occurs within the west-central Slave structural province of the Canadian Shield. The first geological mapping of the west-central Slave structural province was conducted by C.H. Stockwell in 1932 and J.A. Fraser in 1959, both of the Geological Survey of Canada. These surveys provided the first outlines of the Archean volcano-sedimentary belts, granitic terrain, and Aphebian supracrustal rocks of the area.

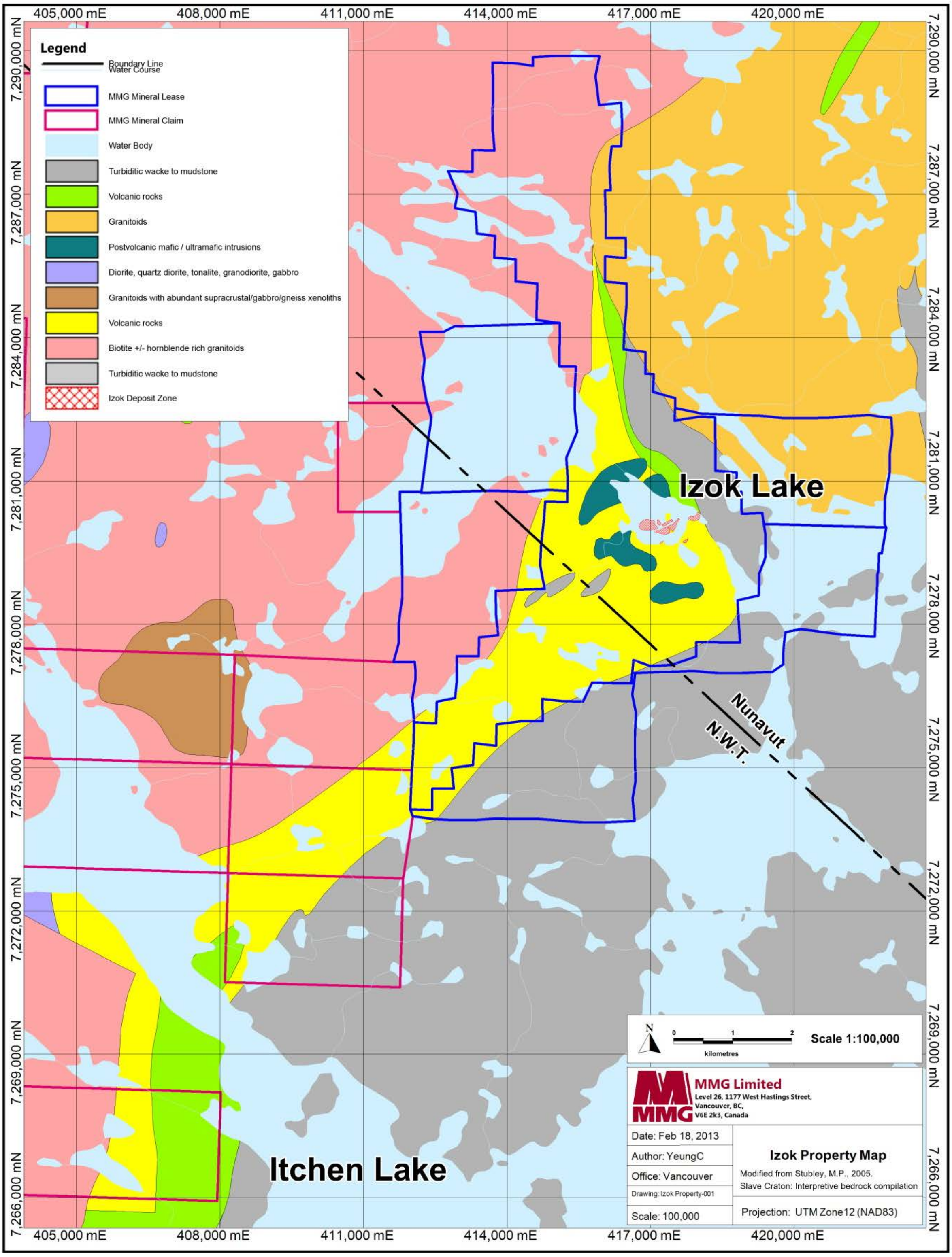
The Archean volcano-sedimentary rocks were assigned to the Yellowknife Supergroup, which is locally divided into the lower Point Lake Formation (a suite of mafic tholeiitic to felsic calc-alkaline metavolcanic rocks and derived metasedimentary rocks) and the upper Contwoyto Formation (a series of iron formation-bearing greywacke turbidites). Both formations are intruded by late Archean granitic bodies and crosscut by north-northwest-trending diabase dikes belonging to the Helikian-aged Mackenzie Swarm. The volcanic rocks in the Izok area are described as being primarily a series of felsic to mafic tuffs, flows and metasediments with some calcareous rocks, partly bedded and clearly water lain. Indicators of stratigraphic tops are scarce and the rocks are extensively deformed. The felsic volcanic rocks of the Point Lake Formation were considered to be dominantly rhyolite flows, typically plagioclase porphyritic with or without quartz phenocrysts. Rhyolite samples in the Izok Lake area have been dated using U-Pb zircon geochronology at 2623 ± 20 Ma and 2680.5 +7/-3 Ma.

The Izok massive sulphide deposit is hosted by the Point Lake Formation, within and near the stratigraphic top of a succession of dominantly felsic volcanic rocks with lesser intermediate and mafic metavolcanic and derived metasedimentary rocks. This suite, which forms an arcuate belt approximately 18km in length and ranges between 1 and 5.5km in width, is informally referred to as the “Izok Lake belt.” The north limb of the belt strikes northwest and dips steeply to the northeast, whereas south of Izok Lake, the belt abruptly swings to strike southwest and dip steeply to the southeast. The belt merges at its south end with the north–south-trending Point Lake Formation. The volcanic stratigraphy youngs to the east, as indicated by pillow facing directions, and is conformably overlain by turbiditic sedimentary rocks of the Contwoyto Formation.

Rocks in the Izok area have undergone three phases of deformation and amphibolite grade (high temperature and low pressure) metamorphism.

The Izok property area was last glaciated between 10,000 and 8,000 years ago. Erosion by the continental glaciation denuded the terrain into bare, extensive areas of fresh rock with a partial veil of till cover.

Figure 94 **Geology of the Izok Property**



10.3 Mineral Resources

10.3.1 Results

The Izok Mineral Resource estimate was developed from a drillhole database which included 362 drillholes for a total of 48,207 metres and 9,670 assays. Since the previous Mineral Resource estimate, an additional 65 drillholes totalling 12,130 metres were completed in and around the Izok deposit. A total of 2,322 assays were collected from these recent drillholes.

A set of three dimensional models to constrain the estimation of the Izok Mineral Resource, is based upon dividing the material into massive sulphide ("MSS") (with semi-massive sulphide ("PMS"), together) and lower grade stringer ("STR") sulphide mineralisation. For the Izok Mineral Resource models, initial quick models were created in Leapfrog 3D geological modelling software to use as a guide for detailed modelling. Detailed three dimensional models for the Izok deposit were then completed in Gemcom GEMS 6.4.1 software.

Cu, Pb Zn, Au and Ag grades were interpolated using an Ordinary Kriging (OK) interpolation method and inverse distance squared (ID2) interpolation method. Variogram and estimation parameters were defined using Gemcom GEMS 6.4.1 software and Supervisor Software. Estimates were modelled on geological domains and density derived from whole core using the Weight in Air/Weight in Water (WW/WA) method.

The Izok Mineral Resource estimate as at June 30 2013 is summarised in Table 143.

The updated Mineral Resource for the Izok deposit, inclusive of Inukshuk, is reported at an economic cut-off grade of 4.0% ZnEq.

The equivalency calculations are based on metal prices of US$1,200/oz for gold, US$20/oz for silver, US$2.80/lb for copper, US$1.18/lb for zinc and US$1.12/lb for lead, and assumes metal recoveries of 75% for gold, 83% for silver, 89% for copper, 93% for zinc and 81% for lead. Other factors used in the equivalency calculations include capital and operating costs. Note that metal prices and recoveries and operating costs may differ from those used for the cash flow model.

Table 143 Total Izok Mineral Resource

Izok Lake Mineral Resources

							Contained Metal				
4% Zn equivalent cut-off grade	Tonnes (Mt)	Zinc (% Zn)	Copper (% Cu)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Measured	-	-	-	-	-		-	-	-	-	
Indicated	13	13	2.4	1.4	73	0.18	1,790	324	194	32	0.1
Inferred	1.2	11	1.5	1.3	73	0.21	120	18	16	2.8	0.01
Total Mineral Resources	**15**	**13**	**2.3**	**1.4**	**73**	**0.18**	**1,910**	**342**	**209**	**34**	**0.1**

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.
Competent Person:
Allan Armitage (Member Association of Professional Geoscientists of Alberta, employee of MMG)

Notes:

(i) ZnEq% = Zn + (Cu*3.3123) + (Pb*1.0856) + (Au*1.8662) + (Ag*0.0328) using Gold at $1,200/oz, Silver at $20/oz, Copper at $2.80/lb, Lead at $1.12/lb, Zinc at $1.18/lb; Metal Recoveries – Gold 75%, Silver 83%, Copper 89% Lead 81% and Zinc 93%.
(ii) Indicated and Inferred Mineral Resources are inclusive of Ore Reserves
(iii) Mineral Resource grade, tonnage and contained metal in the table have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
(iv) Mineral Resources reported to comply with the 2012 JORC code.
(v) Mineral Resources that are not Ore Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Ore Reserves.

10.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Allan Armitage, confirm that I am the Competent Person for the Izok Lake Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta - a 'Recognised Professional Organisation' (RPO) for the purposes of JORC Code reporting.
- I have reviewed the relevant Izok Lake Mineral Resources section of this Report to which this Consent Statement applies.

I was a full time employee of MMG (at the time of estimation).

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Izok Lake Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)
With respect to the sections of this report for which I am responsible – the Izok Lake Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Allan Armitage – 26/11/13

Susan Ball (Witness)

10.4 Mineral Resource JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of the Izok Lake Mineral Resources.

Table 144 Checklist of assessment and reporting criteria for Izok Lake Mineral Resource

<table>
<tr><th>Criteria</th><th>Status</th></tr>
<tr><td colspan="2">Section 1 Sampling Techniques and Data</td></tr>
<tr><td>Sampling techniques</td><td>Diamond drilling was used to obtain 0.1m up to 5.1m length (average 1.33m) half core samples that were submitted for analysis.</td></tr>
<tr><td>Drilling techniques</td><td>Diamond drilling was used to produce AQ, NQ or HQ size diamond core.
The resource database contains 362 diamond holes for 48,207m (all drillholes were collared from surface) (Table 145).
Table 145 Drillholes by drilling company, year, type and length
<table>
<tr><th>Year</th><th>Operator</th><th>No. of Holes</th><th>Core Size</th><th>Length (m)</th><th>Drilling Company</th></tr>
<tr><td>1975 - 1977</td><td>Texasgulf</td><td>115</td><td>AQ</td><td>12,207</td><td>Bradley Brothers</td></tr>
<tr><td>1992</td><td>Minnova</td><td>111</td><td>NQ</td><td>14,230</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>1993</td><td>Metall</td><td>71</td><td>NQ</td><td>9,640</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>2007</td><td>Wolfden</td><td>1</td><td>NQ</td><td>182</td><td>Major Drilling</td></tr>
<tr><td>2008</td><td>Zinifex</td><td>48</td><td>NQ</td><td>8,001</td><td>Major Drilling</td></tr>
<tr><td>2010</td><td>MMG</td><td>1</td><td>NQ</td><td>588</td><td>Major Drilling</td></tr>
<tr><td>2011</td><td>MMG</td><td>6</td><td>NQ</td><td>2,080</td><td>Major Drilling</td></tr>
<tr><td>2012</td><td>MMG</td><td>9</td><td>HQ</td><td>1,279</td><td>Major Drilling</td></tr>
<tr><td>Total</td><td></td><td>362</td><td></td><td>48,207</td><td></td></tr>
</table></td></tr>
<tr><td>Drill sample recovery</td><td>Recovery recorded during core logging was generally good to excellent with minor losses in broken ground. Core recovery was not well documented historically; however, it is not believed that core loss is a significant bias in sampling.</td></tr>
<tr><td>Logging</td><td>Historic data is minimal.
Recent core logging recorded geological and geotechnical information including lithology, alteration strength, mineralogy, RQD, fracture frequency, degree of breakage, weathering/alteration, core recovery.
Drill core storage buildings located at Ham Lake approximately 6.5km Northwest of Izok Lake housed all the mineralized zones on core racks. The remainder of the drill core boxes were well organized in numeric order on outside racks.
Core photographs are available for most drillholes.</td></tr>
<tr><td>Sub-sampling techniques and sample preparation</td><td>Core was split in half by diamond saw. Sample lengths were cut to various lengths within mineralized zones while respecting geological contacts.
Core samples were then bagged, numbered and dispatched to assay laboratories.
Sampling techniques varied between drill campaigns
Laboratory process also followed various techniques between drill campaigns but are believed to have followed industry standards
1975 to 1977 drilling, no detailed description of the sample preparation is available.
1992 to 1993, samples after being sawn in half were crushed on site to 6mm and the other half left in the box for future review. The crushed sample of approximately 250g was then riffled out, bagged and sent out for assay.
2007 and 2012, at the lab, samples were jaw crushed to 70% passing 2mm. A 250g sub-sample is taken from this material and pulverized to 85% passing 75µm.</td></tr>
<tr><td>Quality of assay data and laboratory tests</td><td>Laboratory analysis is considered to be total with the following methods applied at various times:
Texasgulf core from the years of 1975 to 1977 was assayed at Bondar-Clegg in Ottawa and no detailed description of the assay procedures is available
Analyses of the 1992 and 1993 drilling were conducted at Technical Services Laboratory (TSL) in Saskatoon. Atomic absorption spectroscopy using either a Varian 1275 or Varian Spectr. 5 determined the elements
Minnova and Metall conducted a check assaying program to examine the accuracy of the assaying for their 1992 and 1993 drilling programs and of the results from the Texasgulf drilling from 1975 to 1977.
Between 2007 and 2012, ALS Chemex in Vancouver was the primary laboratory. Zinc, copper, lead, silver and 57 other elements are assayed on a 0.25g sub-sample by total acid digestion ($HF–HNO_3–HClO_4$) and ICP finish (ALS Chemex code ME-MS61r). Samples reporting greater than 1% Zn, Cu or Pb, or greater than 100 g/t Ag by ICP are re-digested by total acid, diluted, and finished by AA (ALS Chemex code OG62). Silver samples reporting greater than 1,500 g/t are re-assayed by fire assay with a</td></tr>
</table>

	gravimetric finish. Gold is assayed by standard fire assay methods with an ICP finish on a 30g sub-sample (ALS Chemex code Au-ICP21). Total carbon and total sulphur were analysed by combustion furnace (ALS Chemex codes C-IR07 and C-IR08). Mercury is analysed by Aqua Regia Digestion (ALS Chemex code ME-MS41). ▪ 1992 to 1993, one standard from Certified Reference Materials Standards or from in-house standards and a sample repeat were inserted every 20 samples. ▪ 2007 and 2012 included the insertion of blanks, standards and duplicates into assay sample batches. Procedures varied slightly over the years but typically standards made up ~5% of a batch of samples, blank 2%-3% and Duplicates ~5%. The results of the 2007 to 2012 QA/QC program indicate this dataset has no material issues and is fit for use in the Mineral Resource estimate presented.
Verification of sampling and assaying	▪ Assay results were verified against, assay certificates, logging and core photos. ▪ Routine twinning of holes was not carried out. Infill drilling was carried out to improve confidence in the Mineral Resource and upgrade it from Inferred to Indicated categories. ▪ Core logging data was recorded in Excel spread sheets by experienced geologists. ▪ Drill logs were loaded into a site Access Database up until 2009. The Access database was then transferred into a GIBIS database on the MMG Server with subsequent Excel drill logs were loaded into it directly.
Location of data points	▪ All drillhole coordinates are in Mine Grid Plane Projection. ▪ Prior to 2007, drillhole locations were spotted based on a local mine grid ▪ Down hole surveying in the historic drilling was largely limited to a few dip measurements and no down-hole azimuth readings. ▪ 2007 to 2012, all drillholes were initially spotted with a Garmin 60CX hand held GPS. Final drillhole locations were surveyed using ether the Trimble R8 RTK system (drillhole location) or the Reflex APS system (drillhole location and true north azimuth). ▪ All holes were surveyed with EZ-shot single readings every 50m-100m and some holes were also surveyed with a Maxibor wire line survey with readings every 3m.
Data spacing and distribution	▪ Drill spacing approximately <= 50m for Indicated areas of Mineral Resource. ▪ Drill spacing approximately > 50m for Inferred areas of Mineral Resource. ▪ The data spacing and distribution Cu, Pb, Zn, Au and Ag is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource estimation procedures and classifications applied. ▪ The quantity of assays for deleterious elements such as Bi, As, Cd and Hg is limited to assaying commencing in 2011. Prior programmes were not assayed for these elements. For this reason confidence in the content and distribution of deleterious elements is low. It is assumed that production sampling with blending and processing strategies will be sufficiently implemented to manage levels of these deleterious elements reporting to concentrates thus maintaining the appropriateness of the Mineral Resource classifications applied.
Orientation of data in relation to geological structure	▪ The Izok deposit is a complexly zoned cluster of five composite massive sulphide lenses. ▪ Geological mapping and interpretation show that the mineralisation forms elongate (east-west) near surface trough-shaped flat to south dipping, east plunging lenses. Drilling was conducted on vertical to steep north-south directions to intersect mineralisation across-strike. ▪ Drilling orientation is not considered to have introduced any sampling bias.
Sample security	▪ In the recent drill campaigns, sample security was reasonably maintained ▪ Measures to provide sample security included: – Adequately trained and supervised sampling personnel – Shipped in sealed containers via air freight to the assay laboratories. – Assay laboratory checks of sample dispatch numbers against submission documents
Audit and reviews	▪ A number of reviews were undertaken of the Izok drill database over the years. Reviews found that many of the processes and systems set up by the various companies were industry best and/or good-practice at the time the work was completed. However, several issues with the drill data were identified and recommendations were made for future work. Recommendations included improved QAQC processes be implemented, a greater volume of dry bulk density measurements to be taken, improved assay techniques, more strict criteria used to classify Indicated and Inferred Mineral Resources. More recent drill programs (2011 and 2012) have improved the integrity of the database. ▪ An internal MMG review was undertaken in early 2012. Similar issues were identified.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The Izok Mineral Resource is located within the bounds of Mining lease (ML) 3163. ▪ Mining Lease 3163 covers an area of 10,945 acres (4,429 hectares) with the north half in Nunavut Territory and south half located in Northwest Territories. ▪ ML 3163 is 100% owned MMG and have an expiry date on the 19/10/2026. ▪ Xstrata, now Glencore, is entitled to a net smelter return royalty equal to 3% of the net smelter returns realized, or deemed to be realized, from the sale or other disposition of ore or concentrates mined from any orebody located on the property.

<table>
<tr><td>Exploration done by other parties</td><td>

1975 to 1977 drilling completed by Texasgulf, 15,000m (151 holes).
1992 to 1993 drilling completed by Matall/Minova, 55,208 (114 holes).
2008 drilling was completed by Zinifex, 13,310m in 68 holes (Zinifex acquired Wolfden which later became OZ Minerals in 2008).
In 2009 China Minmetals bought almost all mining assets of Oz Minerals which included Izok Lake and High Lake properties. These collective assets became MMG, a wholly owned subsidiary of China Minmetals.

Table 146 Exploration drillholes by year, drilling company, size, type and length
<table>
<tr><th>Year</th><th>Operator</th><th>No. of drillholes</th><th>Core size</th><th>Length (m)</th><th>Drilling Company</th></tr>
<tr><td>1975</td><td>Texasgulf</td><td>42</td><td>AQ</td><td>5,094</td><td>Bradley Brothers</td></tr>
<tr><td>1976</td><td>Texasgulf</td><td>97</td><td>AQ</td><td>9,923</td><td>Bradley Brothers</td></tr>
<tr><td>1977</td><td>Texasgulf</td><td>7</td><td>AQ</td><td>688</td><td>Bradley Brothers</td></tr>
<tr><td>1992</td><td>Minnova</td><td>121</td><td>NQ</td><td>17,713</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>1993</td><td>Metall</td><td>89</td><td>NQ</td><td>19,406</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>1994</td><td>Metall</td><td>7</td><td>NQ</td><td>4,585</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>1995</td><td>Metall</td><td>6</td><td>NQ</td><td>4,228</td><td>Longyear (Boart) Canada</td></tr>
<tr><td>2007</td><td>Wolfden</td><td>5</td><td>NQ</td><td>2,789</td><td>Major Drilling</td></tr>
<tr><td>2008</td><td>Zinifex</td><td>70</td><td>NQ</td><td>13,310</td><td>Major Drilling</td></tr>
<tr><td>2010</td><td>MMG</td><td>5</td><td>NQ</td><td>2,240</td><td>Major Drilling</td></tr>
<tr><td>2011</td><td>MMG</td><td>42</td><td>NQ</td><td>15,108</td><td>Major Drilling</td></tr>
<tr><td>2012</td><td>MMG</td><td>23</td><td>NQ/HQ</td><td>7,821</td><td>Major Drilling</td></tr>
<tr><td>Total</td><td></td><td>514</td><td></td><td>102,905</td><td></td></tr>
</table>
</td></tr>
<tr><td>Geology</td><td>

The Izok volcanogenic massive sulphide (VMS) deposit occurs within the west-central Slave structural province of the Canadian Shield.
Hosted within the Archean volcano-sedimentary rocks assigned to the Yellowknife Supergroup primarily near the top of the lower Point Lake Formation (a suite of mafic tholeiitic to felsic calc-alkaline metavolcanic rocks and derived metasedimentary rocks).
Rocks in the Izok area have undergone three phases of deformation and amphibolite grade (high temperature and low pressure) metamorphism.
The Izok deposit is a complexly zoned cluster of five composite massive sulphide lenses: Northwest, North, Central (Central West and Central East), and Inukshuk.
Individual massive sulphide lenses are further subdivided into three main classes of sulphide mineralisation and subsequently divided into six main types of sulphide mineralisation namely:
(i) Polymetallic – a) polymetallic and b) sphalerite-galena
(ii) Zinc – c) pyrite-sphalerite and d) sphalerite-pyrite
(iii) Copper – e) pyrite-chalcopyrite and f) chalcopyrite-pyrrhotite
Individual lenses vary from 350m to 570m in length, 50m to 200m wide, 10m up to 125m thick and extend from surface up to 400m depth.
The shape of the massive sulphide lenses partly reflects structural deformation (Izok Lake antiform), but probably also reflects a primary volcanic dome feature.
A mappable zone of strong hydrothermal alteration with an exposed surface area in excess of 14km^2 encompasses the deposit.

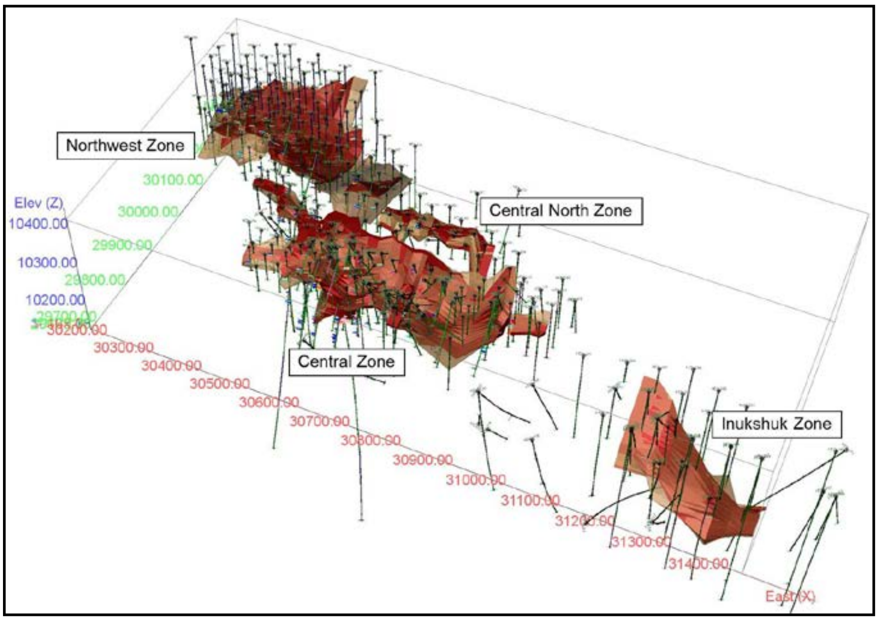
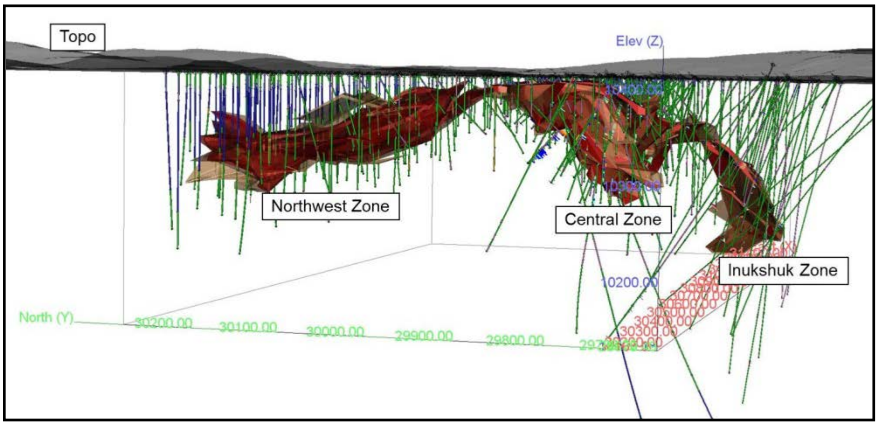
</td></tr>
<tr><td>Drillhole information</td><td>

362 diamond drillholes and associated data are held in the database and used to define the Mineral Resource. No individual hole is material to the Mineral Resource estimated and hence this geological database is not supplied.
This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

</td></tr>
<tr><td>Data aggregation methods</td><td>

Mineral Resources are reported on a ZnEq% basis using the following information: metal grades, metal prices, metal recoveries, smelting and refining terms, operating costs depending on mine method and capital costs. Where:
ZnEq% = Zn + (Cu*3.3123) + (Pb*1.0856) + (Au*1.8662) + (Ag*0.0328) using Gold at $1200/oz, Silver at $20/oz, Copper at $2.80/lb, Lead at $1.12/lb, Zinc at $1.18/lb; Metal Recoveries – Gold 75%, Silver 83%, Copper 89% Lead 81% and Zinc 93%.
Note these metal prices and recoveries may differ from those used for the cash flow models in the Feasibility Study.
This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

</td></tr>
</table>

Relationship between mineralisation width and intercepts lengths	Geological mapping and interpretation show that the mineralisation forms elongate (east-west) near surface trough-shaped flat to south dipping, east plunging lenses. Drilling was conducted on vertical to steep north-south directions to intersect mineralisation across-strike. Holes have been drilled vertically in order to intersect the ore lenses perpendicularly, thereby giving an approximate true thickness.
Diagrams	 **Figure 95 Isometric view looking northwest showing the Izok Mineral Resource models**  **Figure 96 Isometric view looking east showing the Izok Mineral Resource models** ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Balanced reporting	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	▪ All diamond drillhole information was considered for this Mineral Resource estimation. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	▪ No future work program is currently planned. ▪ The project is currently in the final stages of a Feasibility Study.
Section 3 Estimating and Reporting of Mineral Resources	

Database Integrity	▪ All data was stored in a customised access database and was converted to the MMG GBis database by the MMG Exploration Department in 2009/10. ▪ All logging was entered into Microsoft Excel and loaded into the database. ▪ Assay data was loaded from Microsoft Excel directly into database pre 2009. Post 2009 laboratory files were directly loaded into GBis. ▪ Data integrity was validated for EOH depth and sample overlaps. ▪ Manual checks were carried out by plotting and review of sections and plans. ▪ Typographical errors in assay values, supporting information on source of assay values, and finally a review of standards, blanks, and duplicates was completed.
Site visits	The Competent Person has not visited the site. The Competent Person was directed towards Resource estimate completion prior to undertaking a site visit. His employment ceased before a site visit could be arranged.
Geological interpretation	▪ The Izok massive sulphide deposit is hosted by the Point Lake Formation, within and near the stratigraphic top of a succession of dominantly felsic volcanic rocks with lesser intermediate and mafic metavolcanic rocks, and derived metasedimentary rocks. ▪ The Izok Lake Belt rocks are folded into an antiform, with the south limb striking northeast and dipping steeply to the southeast while the north limb strikes southeast and dips steeply to the northeast. ▪ The Izok deposit is a complexly zoned cluster of five composite massive sulphide lenses: Northwest, North, Central (Central West and Central East), and Inukshuk. ▪ Individual massive sulphide lenses are further subdivided into three main classes of sulphide mineralisation and subsequently divided into six main types of sulphide mineralisation namely: (i) Polymetallic – a) polymetallic and b) sphalerite-galena (ii) Zinc – c) pyrite-sphalerite and d) sphalerite-pyrite (iii) Copper – e) pyrite-chalcopyrite and f) chalcopyrite-pyrrhotite ▪ Individual lenses vary from 350m to 570m in length, 50m to 200m wide, 10m up to 125m thick and extend from surface up to 400m depth ▪ The shape of the massive sulphide lenses partly reflects structural deformation (Izok Lake antiform), but probably also reflects a primary volcanic dome feature. ▪ A mappable zone of strong, hydrothermal alteration with an exposed surface area in excess of 14km^2 encompasses the deposit. ▪ Archean formations were intruded by late Archean granitic bodies and pegmatites and crosscut by north to northwest trending diabase dykes that belong to the Helikian-aged Mackenzie Swarm. ▪ A set of 3D models to constrain the Mineral Resource estimation of the Izok mineralisation, is based upon dividing the material into massive sulphide (“MSS”) (with semi-massive sulphide (“PMS”), together) and lower grade stringer (“STR”) sulphide mineralisation. ▪ 3D models were completed in Gemcom Gems 6.4.1 software on north-south mine grid sections, generally 15m-30m apart, for each of the Northwest, North Central, Central and Inuksuk zones. Each zone was completed in two models with separate models for each of the MSS and a lower grade STR material. ▪ The MSS models were generally created to conform to the defined boundaries of MSS and PMS from the lithology logs. Locally the boundary was modified by the sample results. For instance, where the MSS interval included 2%-8% ZnEq material at its periphery, the contact may have been moved in 1m to 3m to align with the true contact of increased grade. This means that the MSS solid is not entirely a pure geologic solid, but has been slightly modified to reflect ZnEq grades near the margins of the units. These adjustments are minor and do not have a material effect on the models. ▪ The STR models were generally created to a lower cut-off of 4% ± 1% ZnEq. This means that the STR model is essentially a grade shell of ZnEq. Although this does at times eliminate some volume of 2%-4%ZnEq material, most commonly the grade ramps up quickly. ▪ Each model was then clipped to topography/overburden surface and selected waste zones which cross-cut mineralisation including diabase dykes and pegmatite dykes. ▪ In addition to the mineralisation models, wireframe models of geology/waste were created and included models of the diabase dykes, gabbro, granite, pegmatite, dacite and china rhyolite. Material not modelled is interpreted to be primarily comprised of undifferentiated metamorphosed felsic to intermediate volcanic rocks. ▪ Confidence in geological interpretation of Inferred mineralisation is at a lower level than Indicated mineralisation due to the limited sampling in these areas, hence implied but not verified geological and grade continuity occurs.
Dimensions	▪ The Northwest lens forms a flat to shallow-dipping body with dimensions of approximately 450m x 300m x 22m thick that subcrops under Izok Lake and extends to a depth of ~130m.

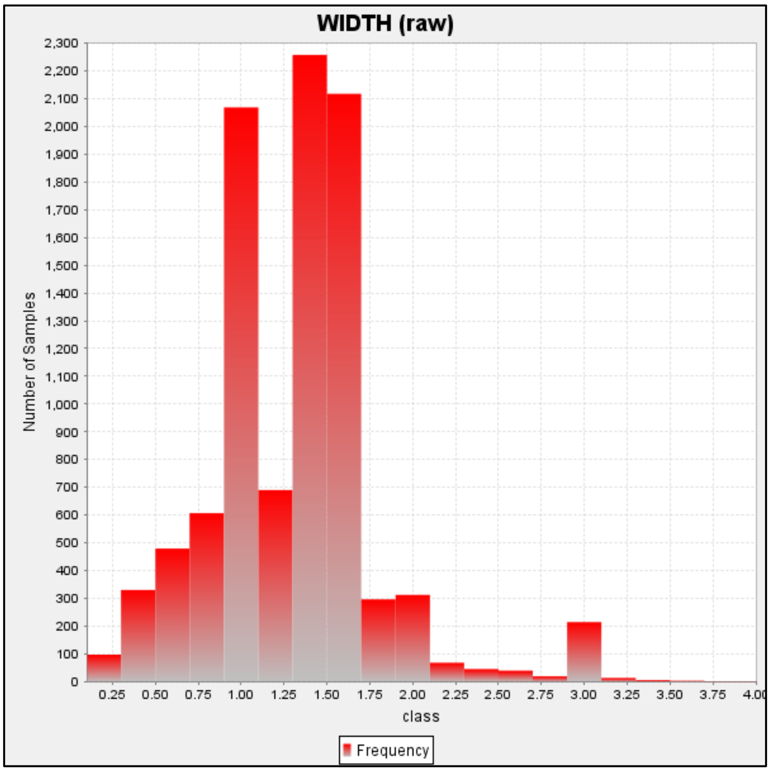
	- The North lens is a small, near-surface, elongate body with dimensions of approximately 200m x 50m x 15m thick. This deposit lies along strike from the Northwest lens. - The Central lens forms a shallow east-plunging, south dipping trough-shaped body with overall dimensions of approximately 570m x 200m x up to 125m thick. - Inukshuk is a dipping and plunging elongate body, defined over a strike length of at least 300m, a dip length of 140m, and an average thickness of 10m. - Mineralisation extends between (local mine grid): – 30200mE to 31000mE – 29700mN to 30300mN – 10260mRL to 10410mRL
Estimation and modelling techniques	- Zinc, copper, lead, silver and gold were estimated for each zone in Izok deposit. - To generate grade within the blocks Ordinary Kriging (OK) algorithm was used for the Northwest and Central zones. The algorithm was inverse distance squared (ID^2) for the North Central and Inukshuk zones, due to the lack of data. For the Northwest and Central zones, check validation models were carried out using ID^2 with very similar results in the global Mineral Resource numbers at various cut-off grades. - Block size was set to 5m x 5m x 5m. The selective mining unit is approximately 5m x 5m x 5m as recommended by engineering studies. - For the estimate, sample intervals were composited to 1.5m which approximates the average sample width of 1.33m for 9,670 assay samples. Approximately 68% of the samples were 1.5m or less and ~95% of the samples were 2m or less (Figure 97). **Figure 97 Width of raw drillhole samples**  - The estimate of each element was undertaken using hard domain boundaries and a series of elliptical search passes orientated generally in the plane of mineralisation. - The long axis of the search ellipses is generally oriented to reflect the observed preferential long axis (geological trend) of the Mineral Resource models. The short Y direction reflects the roughly 1/2 to 1/3 distance of the model in this direction relative to the longer axis. The dip axis of the search ellipse was set to reflect the observed trend of the mineralisation down dip. - Two passes were used to interpolate grade in all of the blocks in the wireframe. The first pass interpolated grade into blocks in the Indicated category and the second pass interpolated grade into blocks in the Inferred Category. - A summary of the search parameters for each Zone of the deposit is listed in Table 147. - The first estimation search pass employed a minimum of 8 and maximum of 20 samples. Estimates were also limited to a maximum of 10 samples from any given hole. Additional passes used more relaxed criteria to estimate the less well informed blocks.

Table 147 Summary of search parameters for each Zone of the deposit

Parameter	**Northwest Zone**		**Central Zone**		**North Central Zone**		**Inukshuk Zone**	
	Indicated	**Inferred**	**Indicated**	**Inferred**	**Indicated**	**Inferred**	**Indicated**	**Inferred**
Search type	Ellipsoid		Ellipsoid		Ellipsoid		Ellipsoid	
Principle azimuth	7°		170°		90°		315°	
Principle dip	0°		-40°		0°		47°	
Intermediate azimuth	97°		80°		0°		67°	
Anisotropy X	55	110	55	110	55	110	55	110
Anisotropy Y	55	110	55	110	55	110	55	110
Anisotropy Z	15	45	26	52	15	30	24	48
Min. samples (MSS)	8	4	8	4	4	2	4	2
Max. samples (MSS)	20	20	20	20	12	12	12	12
Min. samples (LG)	4	2	4	2	4	2	4	2
Max. samples (LG)	12	12	12	12	12	12	12	12
Min. drillholes	2	1	2	1	2	1	2	1

- Statistical analysis between estimated blocks and input data was reviewed.
- Visual checks of block grades and drillhole data in plan and section.
- Interpolation distances in general are 20 m-50m but occur up to 60m-80m in less well drilled area.

Moisture	▪ Bulk density measurement was conducted on dried samples and tonnes in the model have been estimated on a dry basis.
Cut-off parameters	▪ Mineral Resources have been reported at a cut-off grade of 4.0% ZnEq. ▪ This Mineral Resource cut-off represents material that has reasonable prospects for eventual economic extraction within approximately the next 15 years. ▪ The cut-off grade is based on the expectation that the Izok deposit will be mined by open pit methods. ▪ The Inukshuk deposit is reported at a cut-off grade of 4.0% ZnEq. The deposit is not currently expected to be included in Ore Reserves, but still meets the requirements of being prospective for eventual economic extraction.
Mining Factors or assumptions	▪ Mining factors or assumptions have not been applied to the Mineral Resource. ▪ The project is currently undergoing a Feasibility Study and mining methods are being investigated.
Metallurgical factors or assumptions	▪ Metallurgical factors or assumptions have not been applied to the Mineral Resource. ▪ Metallurgical test work was recently completed for the Indicated areas of the Mineral Resource and selected areas of the Inferred Mineral Resource. Test work included mineralogy, comminution tests, flowsheet development, variability tests, production estimation and flotation product testing. ▪ The key assumption for the Izok project is that it will be mined in conjunction with the High Lake deposit and the production of concentrate on a year-by-year basis, will assume an approximate blend of 67% Izok and 33% High Lake ore.
Environmental factors or assumptions	▪ Environmental factors or assumptions have not been applied to the Mineral Resource. ▪ MMG is currently in the process of completing a Feasibility study and baseline data collection is ongoing for the approvals process for development of the Izok Project.
Bulk Density	▪ Bulk density used 1992 and 1993 was determined based on an estimate of the volume of the constituent sulphide minerals, plus gangue, for a particular sample interval during logging. After receipt of the base metal assays, the proportions of the minerals chalcopyrite, sphalerite, and galena were calculated from the assay results. The density of each sample was estimated from the proportion and specific gravity of the constituent phases (total of 7,069 samples). ▪ Bulk Density measurements were determined by Minnova Inc. on 292 mineralized samples from eleven holes drilled in 1992 by the weight in air (oven dried) /weight in water technique and compared results to the calculated densities. Analysis found that that the calculated densities may be slightly too low by 3.5%. ▪ The updated MMG database for the Izok deposit totals 3,292 samples of mineralised and unmineralised material. This included analysis of historic and recent drill core. The density analysis for samples from the metallurgical test holes were completed by ALS in the lab by either the WW/WA on whole core or on pulverized material using a pycnometer. The pycnometer method is an acceptable method as there are no or limited naturally occurring voids or cavities. ▪ A single SG value was used for each mineralisation and geological model based on an analysis of the SG data within each domain. The SG values for each mineralised and geological model were determined based on an analysis of density data within each model.
Classification	▪ Classification is based on data spacing and distribution relative to the distribution and continuity of Cu, Pb, Zn, Ag and Au mineralisation which is often coincident with geological contacts. ▪ The Mineral Resource estimate in areas with drill spacing of 50m or less is classified as Indicated and in areas with drill densities of greater than 50m is classified as Inferred.

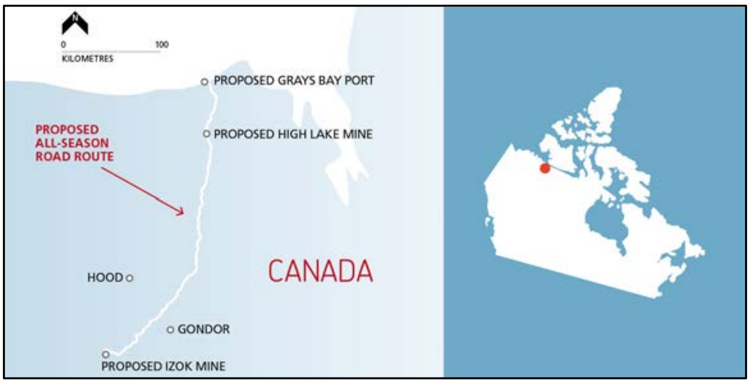
Audits or reviews	▪ Reviews of the current Mineral Resource estimate was completed by Optiro Pty Ltd out of Perth Australia (October and December 2012) as well as MMG personnel (Jared Broome, January 2013. ▪ Audits of previous Mineral Resource Estimate were undertaken by Hatch (2008. Izok Project Pre-Feasibility Study, Volume I – Technical), Behre Dolbear Australia (BDA) (2011), Amec (2011, Izok Lake Prefeasibility Study), and MMG (Jared Broome, spring of 2012). ▪ Few issues with the drill data were identified and recommendations were made for future work. Recommendations included improved QAQC processes be implemented, a greater number of dry bulk density measurements to be taken, improved assay techniques, more strict criteria used to classify Indicated and Inferred Mineral Resources. More recent drill programs have improved the integrity of the database.
Discussion of relative accuracy / confidence	▪ Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support. ▪ Previous drilling (Texasgulf, 1975 to 1977) with small-diameter holes produced samples that were less representative than later, larger-diameter holes. ▪ Texasgulf sampling and analysis protocols are not known. ▪ Historic sampling programmes did not include the insertion of standards, blanks and duplicates. Approximately 33% of the assay database which is used to define the Mineral Resource comes from the Texasgulf drilling. ▪ Minnova/Metall conducted a check assaying program to examine the accuracy of the assaying for their 1992 and 1993 drilling programs and Texasgulf's 1975 to 1977 drilling programs. It was concluded from the historic check sampling that both the zinc and copper assay databases, which are essentially based on TSL assay results, have a small but persistent bias towards high values. This is estimated to be in the range of 0.2% to 0.3% for Zn and around 0.05% for Cu for the overall deposit. The magnitude of this bias is smaller than the accuracy of the overall estimating procedure. Approximately 43% of the assay database which is used to define the Mineral Resource comes from the Minnova/Metell drilling. ▪ Assay QA/QC procedures for drilling conducted between 2007 and 2012 included the insertion of blanks, standards (from a certified laboratory) and field duplicates into assay sample batches. Procedures varied slightly over the years, but typically standards, blanks and duplicates made up about 5%, 2%-3% and about 5% of a batch sample, respectively. Standards certified for Cu, Pb, Zn, Ag and Au, were plotted and examined for deviations from acceptable limits (i.e. ± 2-3 standard deviations). Blank samples (typically silica sand) were plotted on scatter plots and examined for deviations from background levels (i.e. samples which assayed > 2 times an elements detection limit). Field duplicates were plotted on scatter plots to compare sample repeatability. The results of the 2007 to 2012 QA/QC program indicate there are no material issues with the drill core assay data. ▪ Down-hole surveying in the historic drilling was largely limited to a few dip measurements and no down-hole azimuth readings thus limiting the confidence of the down-hole location of drillholes (predominantly angled holes) and the location of the deposit boundaries. This is less of an issue for vertical holes. ▪ It is the opinion of the Competent Person that the exploration work was professionally managed and used procedures meeting or exceeding generally accepted industry best practices. After review, the Competent Person is of the opinion that the exploration data is sufficiently reliable to interpret with confidence the boundaries of the base metal mineralisation for the Izok deposit and support evaluation and classification of Mineral Resources in accordance with the 2012 JORC code.

11. HIGH LAKE

11.1 Introduction and setting

The High Lake deposit is located in the West Kitikmeot Region of Nunavut Territory in the Canadian Arctic (Figure 98).

Figure 98 High Lake Deposit location



11.2 Geological setting

The High Lake volcanogenic massive sulphide (VMS) deposit is hosted within the High Lake greenstone belt in the northern part of the Slave structural province approximately 40km south of Coronation Gulf. The High Lake greenstone belt, located in the northern Slave province, is a north-south striking Archean greenstone belt approximately 80km long, and ranges from 5 to 10km wide in the north to 25km wide in the south.

The belt is subdivided into three domains: the Western, Central, and Eastern, based on lithology, mineralisation, and geochronology. The Western domain consists primarily of intermediate and felsic volcanic rocks with minor mafic volcanic rocks and numerous massive sulphide occurrences, many containing significant gold. Zircons from the dacitic rocks in the Western domain returned U-Pb dates of 2695 ± 3 Ma to 2705 ± 1 Ma. The Central domain is a sedimentary rock-dominated package of typically chemical carbonate sedimentary rocks, slates, siltstones, greywackes, volcaniclastic rocks and minor conglomeratic sedimentary rocks with lesser, commonly interbedded felsic and mafic volcanic flows. The main mineral occurrences in this domain consist of gold veins with anomalous arsenic contents such as the ULU deposit. Younger U-Pb ages of 2616 ± 3 Ma and 2612 ± 3 Ma have been determined for the Central domain. The Eastern domain is characterized by predominantly mafic to intermediate volcanic rocks with only minor sedimentary rocks, and has a geology and metallogeny similar to the Western domain. Zircons from a dacite porphyry in the Eastern domain returned a U-Pb age of 2671 ± 3 Ma.

The High Lake deposits are located within the Western domain where a zircon from a felsic volcanic rock returned a U-Pb age of 2705 ± 1 Ma.

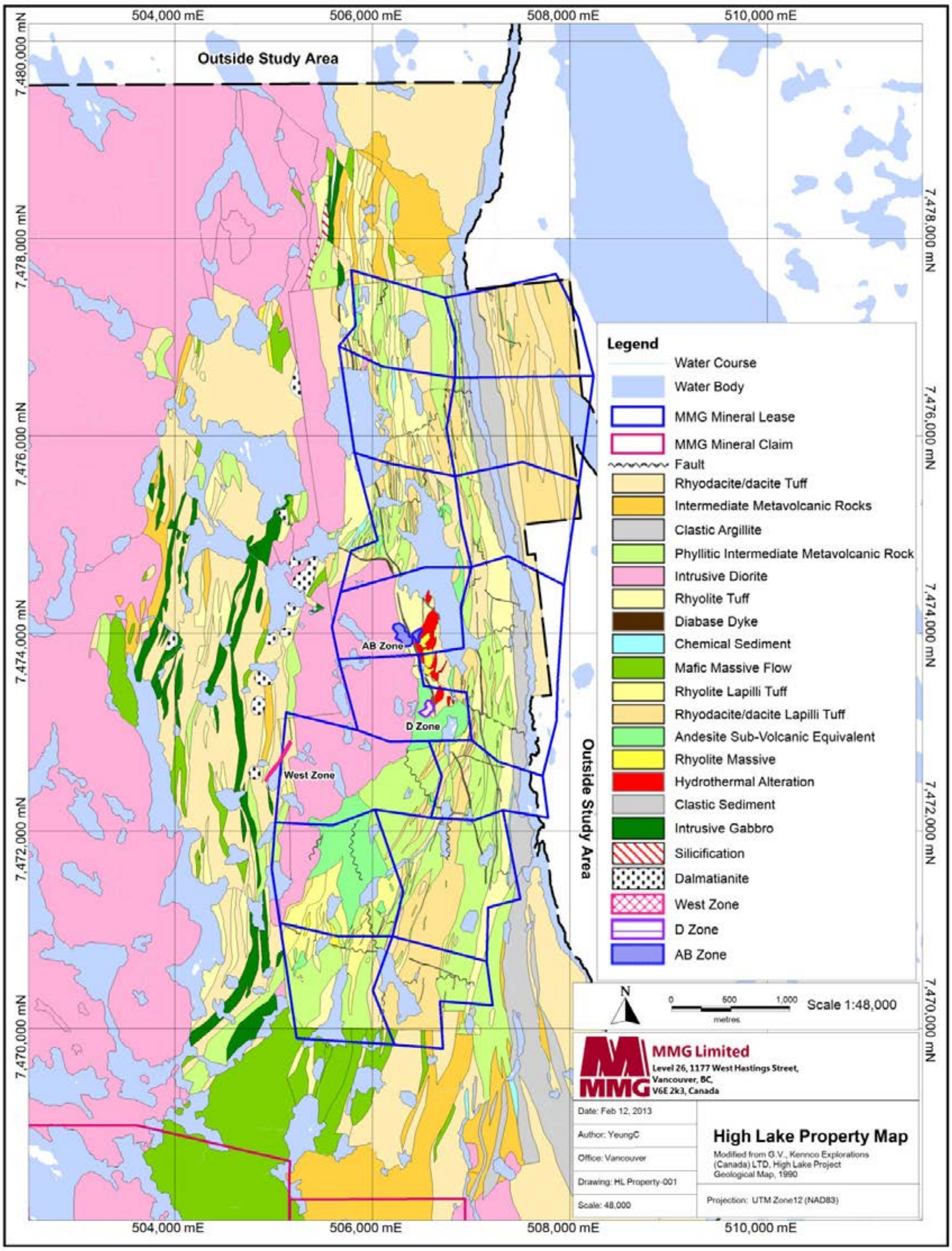
The central part of the High Lake property is underlain by north-trending Archean aged (2.69Ga-2.60Ga) basaltic to rhyolitic flows and fragmental volcanics. Intercalated with the rhyolitic volcanics, and at their eastern contact with andesitic rocks, are numerous carbonate-rich exhalite lenses. Argillites and greywacke underlie the easternmost part of the property. A large mass of Late Archean plutonic rocks intrudes the supracrustal units in the western part of the property. Several prominent northwest and north-south trending brittle faults, including the regional High Lake fault, variably displace granitoid and volcanic units before the emplacement of the diabase dikes.

The High Lake VMS deposits are within the felsic volcanic sequence with the AB and D zones at or near the contact with granodiorite intrusion. Stratigraphic tops for the AB and D zones are interpreted to face west.

Four sets of structures are recognized within the supracrustal rocks of the belt, interpreted to have formed during separate deformational events. Regional metamorphic grade in the High Lake belt is predominantly greenschist facies, as indicated by the assemblage quartz-chlorite-carbonate-epidote in andesites. Contact metamorphic aureoles are documented around granitic intrusions, with metamorphic grades that reach amphibolite facies.

Past drilling has identified the West, AB and D mineral zones. The West zone is on the western side of the diorite/granodiorite intrusion, about 1.3km west of the D zone, and is comprised of three mineralized lenses. The D zone is about 560m south of the AB zone. It is a banded polymetallic massive sulphide comprised of four separate lenses of mineralisation. The largest is about 150m long, dips down 320m and is up to 35m thick. The A zone is larger, comprised of stringer type sulphide mineralisation stratigraphically below the more massive sulphide B zone mineralisation. The A zone is also discordant with the host felsic lapilli tuffs interbedded with ash tuffs and crystal tuffs. The B zone is stratigraphically above the A zone.

Figure 99 Geology of the High Lake Property



11.3 Mineral Resources

11.3.1 Results

The updated High Lake Mineral Resource was developed from a drillhole database which included 286 drillholes for a total of 80,869m and 10,747 assays. Since the last Mineral Resource estimate, an additional 17 drillholes totalling 3,907m were completed in and around the High Lake deposits. A total of 1,786 assays were collected from these recent drillholes.

A set of 3D models to constrain the estimation of the High Lake Mineral Resource, is based upon dividing the material into massive sulphide ("MSS") (with semi-massive sulphide ("PMS"), together) and lower grade stringer ("STR") sulphide mineralisation. Detailed 3D Mineral Resource models for the High Lake deposit were completed in Gemcom GEMS 6.4.1 software.

Cu, Pb Zn, Au and Ag grades were interpolated using an Ordinary Kriging (OK) interpolation method and inverse distance squared (ID2) interpolation method. Variogram and estimation parameters were defined using Gemcom GEMS 6.4.1 software and Supervisor Software. Estimates were modelled on geological domains and density derived from whole core using the Weight in Air/Weight in Water (WW/WA) method.

The High Lake Mineral Resource estimate as at June 30 2013 is summarised in Table 148. The updated Mineral Resources for the High Lake deposit are reported at various cut-off grades due to the fact that the AB and D zones will be mined by open pit mining methods and the West zone will be mined by underground mining methods.

The Mineral Resources, inclusive of Ore Reserves, are reported in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves – The JORC Code (2012 Edition) and estimated by a Competent Person as defined by the JORC Code. Mineral Resources that are not Ore Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Ore Reserves.

The equivalency calculations (note ☐(i) below) are based on metal prices and metal recoveries. Other factors used in the equivalency calculations include capital and operating costs. Note that metal prices, recoveries and operating costs may differ from those used for the cash flow model.

Table 148 Global High Lake Mineral Resource

High Lake Mineral Resources

							Contained Metal				
3% Cu equivalent cut-off grade	Tonnes (Mt)	Zinc (% Zn)	Copper (% Cu)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	7.9	3.5	3.0	0.3	83	1.3	279	239	25	21	0.3
Inferred	6.0	4.3	1.8	0.4	84	1.3	256	108	25	16	0.3
Total Mineral Resources	**14**	**3.8**	**2.5**	**0.4**	**84**	**1.3**	**536**	**347**	**50**	**37**	**0.6**

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Details of relevant inputs for estimating Mineral Resources are given in the Technical Appendix published on the MMG website.
Competent Person:
Allan Armitage (Member Association of Professional Geoscientists of Alberta, employee of MMG)

(i) CuEq% = Cu + (Zn*0.3019) + (Pb*0.3278) + (Au*0.5634) + (Ag*0.0099) using Gold at $1200/oz, Silver at $20/oz, Copper at $2.80/lb, Lead at $1.12/lb, Zinc at $1.18/lb; Metal Recovery assumptions are Gold 75%, Silver 83%, Copper 89%, Lead 81% and Zinc 93%.

(ii) Indicated and Inferred Mineral Resources are inclusive of Ore Reserves

(iii) Mineral Resource grade, tonnage and contained metal in the table have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding

(iv) Mineral Resources reported to comply with the 2012 JORC code.

(v) Mineral Resources that are not Ore Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Ore Reserves.

11.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (“2012 JORC Code”).

Competent Person Statement

I, Allan Armitage, confirm that I am the Competent Person for the High Lake Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta - a ‘Recognised Professional Organisation’ (RPO) for the purposes of JORC Code reporting.
- I have reviewed the relevant High Lake Mineral Resources section of this Report to which this Consent Statement applies.

I was a full time employee of MMG (at the time of estimation).

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the High Lake Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the High Lake Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Allan Armitage – 26/11/13

Susan Ball (Witness)

11.4 Mineral Resource JORC 2012 Assessment and Reporting Criteria

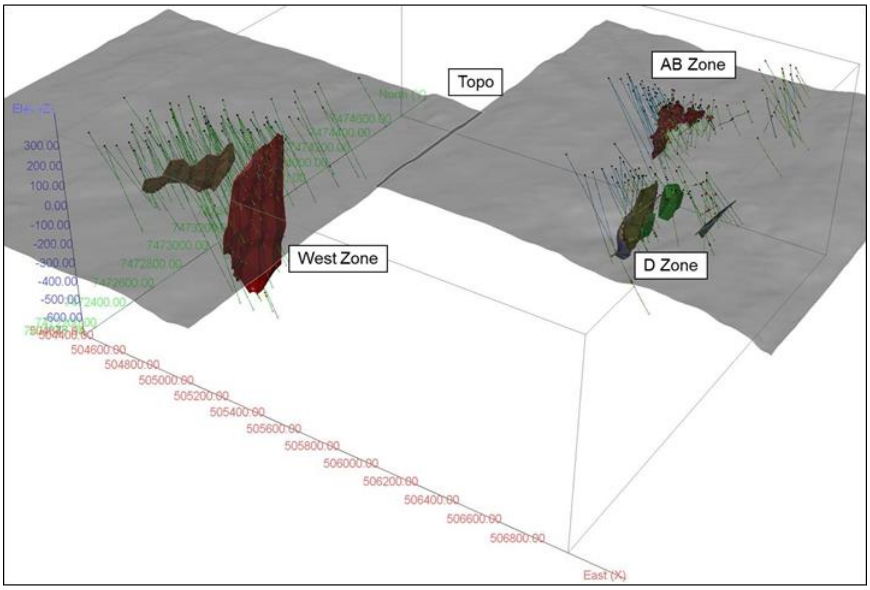
The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of the High Lake Mineral Resources.

Table 149 Checklist of assessment and reporting criteria for High Lake Mineral Resource

<table>
<tr><th>Criteria</th><th>Status</th></tr>
<tr><td colspan="2">Section 1 Sampling Techniques and Data</td></tr>
<tr><td>Sampling techniques</td><td>Diamond drilling was used to obtain 0.1m up to 5.5m length (average 1.09m) half core samples that were submitted for analysis (~89% of the samples were < 1.5m).</td></tr>
<tr><td>Drilling techniques</td><td>Table 150 Drillholes by drilling company, year, type and length
<table>
<tr><th>Year</th><th>Company</th><th>Holes</th><th>Metres</th><th>Samples</th><th>Type</th></tr>
<tr><td>1956</td><td>Kennecott</td><td>31</td><td>3,844</td><td>563</td><td>AX</td></tr>
<tr><td>1957</td><td>Kennecott</td><td>20</td><td>3,207</td><td>456</td><td>AX</td></tr>
<tr><td>1992</td><td>Aber/Kennecott</td><td>17</td><td>3,638</td><td>851</td><td>NQ</td></tr>
<tr><td>1993</td><td>Aber/Kennecott</td><td>23</td><td>4,501</td><td>960</td><td>NQ</td></tr>
<tr><td>2001</td><td>Wolfden</td><td>16</td><td>3,147</td><td>662</td><td>NQ</td></tr>
<tr><td>2002</td><td>Wolfden</td><td>22</td><td>6,978</td><td>1,081</td><td>NQ</td></tr>
<tr><td>2003</td><td>Wolfden</td><td>48</td><td>16,009</td><td>1,497</td><td>NQ</td></tr>
<tr><td>2004</td><td>Wolfden</td><td>46</td><td>17,781</td><td>1,266</td><td>NQ</td></tr>
<tr><td>2005</td><td>Wolfden</td><td>37</td><td>14,808</td><td>1,158</td><td>NQ</td></tr>
<tr><td>2006</td><td>Wolfden</td><td>4</td><td>879</td><td>87</td><td>NQ</td></tr>
<tr><td>2007</td><td>Zinifex</td><td>5</td><td>2,170</td><td>380</td><td>NQ</td></tr>
<tr><td>2012</td><td>MMG</td><td>9</td><td>2,033</td><td>505</td><td>NQ</td></tr>
<tr><td>2012</td><td>MMG</td><td>8</td><td>1,874</td><td>1,281</td><td>HQ</td></tr>
<tr><td>Total</td><td></td><td>286</td><td>80,869</td><td>10,747</td><td></td></tr>
</table></td></tr>
<tr><td>Drill sample recovery</td><td>▪ Recovery recorded during core logging was generally good to excellent with minor losses in broken ground. Core recovery averages 97% for all mineralized zones. It is not believed that core loss has introduced significant bias in sampling.</td></tr>
<tr><td>Logging</td><td>▪ Historic logging information is not well documented.
▪ Recent core logging recorded geological and geotechnical information including lithology, alteration strength, mineralogy, RQD, fracture frequency, degree of breakage, weathering/alteration, core recovery.
▪ All drill core is stored at the High Lake camp located in the south-central area of the property, on High Lake near the AB and D zones.
▪ Core photographs are available for drillholes except for the 1956 to 1957 holes.</td></tr>
<tr><td>Sub-sampling techniques and sample preparation</td><td>▪ Core was split in half by diamond saw. Sample lengths were cut to various lengths within mineralized zones while respecting geological contacts.
▪ Core samples were then bagged, numbered and dispatched to assay laboratories.
▪ Sampling techniques varied between drill campaigns
▪ Laboratory process also followed various techniques between drill campaigns but are believed to have followed industry standards
▪ During 1955 to 1957 samples were processed at Kennecott’s own laboratory in Salt Lake, Utah. No sample preparation or assay procedures are available from this period.
▪ In the 2001 drill campaign samples were crushed to 70% passing -6mm. A riffle splitter was used to cut a <250g sample which was then pulverized to 85% passing 75µm.
▪ In 2002 samples were crushed to -8 mesh and a 400g sample was extracted using a riffle splitter. Samples were pulverized to 95% passing -150 mesh.
▪ In 2003 and 2004 half core was dried and then crushed in two stages to -6mm and then to -2mm. The sample was then split in a riffle splitter to obtain a sub-split of 250g-300g. The subsplit was then pulverized in a ring and puck mill to produce a final product that was 95% minus 200 mesh (- 75µm).
▪ 2007 and 2012, at the lab, samples were jaw crushed to 70% passing 2mm. A 250g sub-sample is taken from this material and pulverized to 85% passing 75µm.</td></tr>
<tr><td>Quality of assay</td><td>▪ Laboratory analysis by drill campaign is described below:</td></tr>
</table>

data and laboratory tests	– During 1955 to 1957 samples were processed at Kennecott's own laboratory in Salt Lake, Utah. No sample preparation or assay procedures are available from this period. It is assumed that the methodologies used by Kennecott would result in reliable assays. – In 1991 to1992 samples were analysed by the Acme Analytical Laboratories Ltd., Vancouver. Au and Ag assays were via fire assay using a 1 assay-ton sample (29.17g). A 0.5g sample was assayed with aqua regia digestion and AAS finish for Cu, Zn, Pb. Standard samples were inserted by the laboratory at a rate of one for each 25 sample batch. Standard samples were inserted by the laboratory at a rate of one for each 25 sample batch. – Assaying during 1993 was completed by Chemex Laboratories Ltd of North Vancouver, B.C. Samples were tested via an aqua regia digestion followed by determination of Cu, Zn, Pb and Ag by flame AAS. Gold was assayed using a 30g sample and fire assayed followed by an AAS finish. Samples containing more than 10g/t Au were reassayed using a 29.2g (1 assay ton) sample and fire assay with gravimetric finish. Insertion of standards is reported however the results were not available for review. – In the 2001 drill campaign samples were analysed at ALS Chemex Ltd, Vancouver (ISO 9002 certification). Assaying of Cu, Zn, Pb and Ag were completed using aqua regia digestion followed by AAS. Gold assays were based on a 20g pulp sample using fire assay and an AAS finish. Insertion of control samples were reported to have occurred however results were not available for review. – In 2002 Accurassay Laboratories in Thunder Bay, Ontario (ISO/IEC 17025 accreditation) were used. Samples were assayed for Cu, Pb, Zn and Ag using aqua regia digestion and AAS. Gold assays were based on a 20g pulp sample using fire assay and an AAS finish. Insertion of control samples were reported to have occurred however they were not available for review. – In 2003 and 2004, Global Discovery Laboratories (GDL), a division of Teck Cominco Ltd., was used to assay the High Lake samples. Samples were assayed for Cu, Pb, Zn and Ag using aqua regia digestion and AAS. Gold assays were based on a 30g pulp sample using fire assay and an AAS finish. – Between 2007 and 2012, ALS Chemex in Vancouver was the primary laboratory. Zinc, copper, lead, silver and 57 other elements are assayed on a 0.25g sub-sample by four acid "near-total" digestion (HF–HNO$_3$–HClO$_4$) and ICP finish (ALS Chemex code ME-MS61r). Samples reporting greater than 1% Zn, Cu or Pb, or greater than 100 g/t Ag by ICP are re-digested by four acid digest, diluted, and finished by AA (ALS Chemex code OG62). Silver samples reporting greater than 1,500 g/t are re-assayed by fire assay with a gravimetric finish. Gold is assayed by fire assay with an ICP finish on a 30g sub-sample (ALS Chemex code Au-ICP21). Total carbon and total sulphur were analysed by combustion furnace (ALS Chemex codes C-IR07 and C-IR08). Mercury is analysed by Aqua Regia Digestion (ALS Chemex code ME-MS41). ▪ QAQC protocols employed from 2007 to 2012 included the insertion of blanks, standards and duplicates into assay sample batches. Procedures varied slightly over the years but typically standards made up ~5% of a batch of samples, blank 2%-3% and Duplicates ~5%. The results of the 2007 to 2012 QA/QC program indicate there are no material issues with the drill core assay data. The results of the 2007 to 2012 QA/QC program indicate this dataset has no material issues and is fit for use in the Mineral Resource estimate presented.
Verification of sampling and assaying	▪ Assay results were verified against assay certificates, logging and core photos. ▪ Routine twinning of holes was not carried out. Infill drilling was carried out to improve confidence in the Mineral Resource and upgrade it from Inferred to Indicated categories. ▪ Core logging data was recorded in Excel spread sheets by experienced geologists. ▪ Drill logs were loaded into a site Access Database up until 2009. The Access database was then transferred into a GIBIS database on the MMG Server with subsequent Excel drill logs were loaded into it directly.
Location of data points	▪ All drillhole coordinates are in the UTM system NAD 83, Zone 12 Projection. ▪ During early drill programs, drillhole locations were spotted based on a surface grid established in NAD 27 Zone 12 grid system. ▪ Wolfden re-surveyed drillholes in 2005. All drillholes were surveyed in UTM NAD27 Zone 12. ▪ 2007 to 2012, all drillholes were initially spotted with a Garmin 60CX hand held GPS. Final drillhole locations were surveyed using ether the Trimble R8 RTK system (drillhole location) or the Reflex APS system (drillhole location and true north azimuth). ▪ In 2012, Ollerhead & Associates Ltd. ("Ollerhead") was contracted to re-survey the High Lake deposit drillhole collar locations in UTM NAD83 Zone 12 using a dual frequency Leica Vivas GPS System 1200. Of 288 historic and recent drillholes in the High Lake deposit areas, including the 2012 drillholes, Ollerhead was able to find the original collars for 246 holes. Overall, drillholes completed from 1956 to 2002 shifted by an average of 0.3m east, -4.0m north and -10.8m elevation. This average difference was added to all 1956 to 2002 holes not surveyed, which totalled 34 holes. All other holes - if not re-surveyed were not changed. ▪ Acid tests were used to determine down-hole deflection in the initial 1956 and 1957 drill phases.

	▪ Acid tests were also used for the 1992, 1993 and 2001 drill campaigns. ▪ Drill campaigns conducted in 2002 onwards were surveyed for deviation from intended target with a standard acid testing apparatus. ▪ Drillholes in the West Zone were tested for deviation using the Reflex Maxibor system which measures rod deviation every three metres. Check surveys using the Reflex Easy Shot system were used as backup to the Maxibor surveys. ▪ In 2012, all drillholes were surveyed with EZ-shot single readings every 50m-100m and some holes were also surveyed with a Maxibor wire line survey with readings every 3m.
Data spacing and distribution	▪ Drillhole spacing ranges from 15m to greater than 50m ▪ The data spacing and distribution Cu, Pb, Zn, Au and Ag is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource estimation procedures and classifications applied. ▪ The quantity of assays for deleterious elements such as Bi, As, Cd and Hg is limited to assaying commencing in 2008. Prior programmes did not assay for these elements. For this reason confidence in the content and distribution of deleterious elements is low. It is assumed that production sampling with blending and processing strategies will be sufficiently implemented to manage levels of these deleterious elements reporting to concentrates thus maintaining the appropriateness of the Mineral Resource classifications applied.
Orientation of data in relation to geological structure	▪ The High Lake deposit comprises the West Zone, D Zone and AB Zones. ▪ Geological mapping and interpretation show that the mineralisation forms a series of elongate north-south trending, steep west dipping, lenses of massive, semi-massive and stringer mineralisation. Drilling was conducted predominantly on shallow to steep east directions to intersect mineralisation across-strike. ▪ Drilling orientation is not considered to have introduced any sampling bias.
Sample security	▪ Sample security for historic drill campaigns is not well documented. ▪ In the recent drill campaigns (2002 onwards), sample security was well maintained. ▪ Measures to provide sample security included: – Adequately trained and supervised sampling personnel – Shipped in sealed containers via air freight to the assay laboratories – Assay laboratory checks of sample dispatch numbers against submission documents
Audit and reviews	▪ A number of reviews were undertaken of the High Lake Mineral Resource over the years. Reviews found that many of the processes and systems set up by the various companies were industry best and/or good-practice at the time the work was completed. However, several issues with the drill data were identified and recommendations were made for future work. Recommendations included improved QAQC processes be implemented, a greater volume of dry bulk density measurements to be taken, improved assay techniques, more strict criteria used to classify Indicated and Inferred Mineral Resources. More recent drill programs have improved the integrity of the database. ▪ An internal MMG review was undertaken in early 2012. Similar issues were identified.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The High Lake property consists of 15 leases, covering 1,730ha and a portion of CO-29 lands which cover 6,171ha. The leases are mainly within Land Claim CO-29, which encompasses both surface and subsurface rights to Nunavut Tunngavik Incorporated (NTI). The mining leases, however, are grandfathered and are exempt from NTI ownership as long as tenure is maintained; they are, therefore, subject to the Canada Mining Act. ▪ A/B zone is situated on mining lease ML 2381, the D zone is situated on mining lease ML 2374, and the West Zone is on the boundary of ML 2377 and the IOL CO-29 parcel. ▪ The leases are 100% owned MMG and have an expiry date on the 16/04/2034. ▪ Kennecott, now Rio Tinto, retained a 1.5% net smelter royalty interest in the High Lake leases.
Exploration done by other parties	▪ 1956 to 1957 drilling completed by Kennecott, 7,050m in 51 drillholes. ▪ 1992 to 1993 drilling completed by Aber/Kennecott, 10,693m in 60 drillholes. ▪ 2001 to 2006 drilling was completed by Wolfden, 75,731m in 133 drillholes. ▪ 2007 to 2008 drilling was completed by Zinifex and Oz, 4,454m in 13 drillholes (Zinifex acquired Wolfden which later became OZ Minerals in 2008). ▪ In 2009 China Minmetals bought almost all mining assets owned by Oz Minerals which included Izok Lake and High Lake properties. These collective assets became MMG.
Geology	▪ The High Lake VMS deposit is hosted within the High Lake greenstone belt in the northern part of the Slave structural province. ▪ The central part of the High Lake property is underlain by north-trending Archean aged (2.69Ga-2.60Ga) basaltic to rhyolitic flows and fragmental volcanics. Intercalated with the rhyolitic volcanics and at their eastern contact with andesitic rocks, are numerous carbonate-rich exhalite lenses. Argillites and greywacke underlie the

	easternmost part of the property. A large mass of Late Archean plutonic rocks intrudes the supracrustal units in the western part of the property. ▪ Four sets of structures are recognized within the supracrustal rocks of the belt, interpreted to have formed during separate deformational events. Regional metamorphic grade in the High Lake belt is predominantly greenschist facies. Contact metamorphic aureoles are documented around granitic intrusions, with metamorphic grades that reach amphibolite facies. ▪ Geological mapping and interpretation show that the mineralisation forms a series of elongate north-south trending, steep west dipping, lenses of massive, semi-massive and stringer mineralisation. Drilling was conducted predominantly on shallow to steep east directions to intersect mineralisation across-strike. ▪ Individual lenses vary from 150m to 300m in length, 20m to 45m wide, and extend from surface up to 300m to 400m up to 900m depth ▪ The shape of the massive sulphide lenses partly reflects structural deformation, but probably also reflects a primary volcanic dome feature. ▪ Mappable zones of strong hydrothermal alteration with an exposed surface area encompass the deposit.
Drillhole information	▪ 286 diamond drillholes and associated data are held in the database and used to define the Mineral Resource. No individual hole is material to the Mineral Resource estimated and hence this geological database is not supplied. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Data aggregation methods	▪ Mineral Resources are reported on a CuEq basis using the following information: metal grades, metal prices, metal recoveries, smelting and refining terms, operating costs depending on mine method and capital costs ▪ CuEq% = Cu + (Zn*0.3019) + (Pb*0.3278) + (Au*0.5634) + (Ag*0.0099) using Gold at $1200/oz, Silver at $20/oz, Copper at $2.80/lb, Lead at $1.12/lb, Zinc at $1.18/lb; Metal Recoveries – Gold 75%, Silver 83%, Copper 89% Lead 81% and Zinc 93%. ▪ Note these metal prices and recoveries may differ from those used for the cash flow models in the Feasibility Study. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Relationship between mineralisation width and intercepts lengths	▪ Geological mapping and interpretation show that the mineralisation forms a series of elongate north-south trending, steep west dipping, lenses of massive, semi-massive and stringer mineralisation. Drilling was conducted predominantly on shallow to steep east directions to intersect mineralisation across-strike. Whenever possible holes have been drilled in order to intersect the ore lenses at a high angle, thereby giving an approximate true thickness.
Diagrams	**Figure 100** **Isometric view looking northwest showing the High Lake Mineral Resource models**  ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Balanced reporting	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is

	provided for this section.
Other substantive exploration data	▪ All diamond drillhole information was considered for this Mineral Resource estimation. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	▪ No future work program is currently planned. ▪ The project is currently in the final stages of a Feasibility Study.

Section 3 Estimating and Reporting of Mineral Resources

Database Integrity	▪ All data was stored in a customised Microsoft Access database and was converted to the MMG GBis database by the MMG Exploration Department in 2009/2010. ▪ All logging was entered into Microsoft Excel and loaded into the database. ▪ Assay data was loaded from Microsoft Excel directly into database pre 2009. Post 2009 laboratory files were directly loaded into GBis. ▪ Data integrity was checked and validated for EOH depth and sample overlaps. ▪ Manual checks were carried out by plotting and review of sections and plans. ▪ Typographical errors in assay values, supporting information on source of assay values and finally a comparison of standards, blanks, and duplicates was completed.
Site visits	▪ The Competent Person has not visited the site. The Competent Person was directed towards Resource estimate completion prior to undertaking a site visit. His employment ceased before a site visit could be arranged.
Geological interpretation	▪ A set of 3D models to constrain the Mineral Resource estimation of the High Lake mineralisation, is based upon dividing the material into massive sulphide ("MSS") (with semi-massive sulphide ("PMS"), together) and lower grade stringer ("STR") sulphide mineralisation. ▪ 3D models were completed in Gemcom Gems 6.4.1 software on east-west sections, generally 12.5m -25m apart, for each of the West, AB and D Zones. The West zone was completed with separate models for each of the MSS and a lower grade STR material. ▪ The AB and D Zones were modelled in separate lenses of variable mixed massive sulphide, semi-massive sulphide and stringer sulphides. The models were constructed based on the distribution of the base metal mineralisation in the 0.5% to 1.0% CuEq range. A minimum width of approximately 3m was used, which resulted in the occasional incorporation of hanging wall or foot wall material. ▪ Each model was then clipped to topography/overburden surface and selected waste zones which cross-cut mineralisation including diabase dykes. ▪ In addition to the mineralisation models, wireframe models of geology/waste were created and included models of the diabase dykes, granite, and mixed volcanic rocks. ▪ Confidence in geological interpretation of Inferred mineralisation is at a lower level than Indicated mineralisation due to the limited sampling in these areas, hence implied but not verified geological and grade continuity occurs.
Dimensions	▪ The West Zone forms a steep-dipping body with dimensions of approximately 275m long, extends about 900m down dip and is up to 40m thick. ▪ The D Zone comprises 4 separate steep west-dipping lenses. The largest is about 150m long, extends 320m down-dip and is up to 35m thick. ▪ The AB Zone is also a steep west dipping zone of mixed massive, semi-massive and stringer mineralisation defined over a strike length of 175m, a dip length of 375m, and a thickness ranging from 10m up to 130m. ▪ For the West Zone mineralisation extends between (NAD 83, Zone 12): – 504700mE to 505075mE – 7472200mN to 7472800mN – -525mRL to 325mRL ▪ For the AB and D Zones mineralisation extends between (NAD 83, Zone 12): – 506250mE to 506800mE – 7473150mN to 7474000mN – -50mRL to 325mRL
Estimation and modelling techniques	▪ Zinc, copper, lead, silver and gold were estimated for each zone in the High Lake deposit. ▪ To generate grade within the blocks, the algorithm was inverse distance squared (ID^2) for all zones. Check validation models were carried out using Ordinary Kriging which returned very similar global results for all models. Global Mineral Resource numbers at a 2% CuEq cut-off grade. ▪ Block size was set to 2.5m x 2.5m x 2.5m for the West Zone and 2.5m x 5m x 5m for the AB and D Zones. The block size was selected in order to accommodate the more closely spaced drilling and the open pit and

underground mining block size (standard mining unit) as recommended by engineering studies.

- For the estimate, sample intervals were composited to 1.0m which approximates the average sample width of 1.09m for 10,747 assay samples. Of the total assay population ~72% of the samples were 1.0m or less and ~89% of the samples were 1.5m or less.
- The estimate of each element was undertaken using hard domain boundaries and a series of elliptical search passes orientated generally in the plane of mineralisation. These search orientations and sizes were supported by variography analysis Gemcom GEMS 6.4.1 software.
- The long axis of the search ellipses is generally oriented to reflect the observed preferential long axis (geological trend) of the models of the mineralisation. The short Y direction reflects the roughly 1/2 to 1/3 distance of the model in this direction relative to the longer axis. The dip axis of the search ellipse was set to reflect the observed trend of the mineralisation down dip.
- Two passes were used to interpolate grade in all of the blocks in the wireframe. The first pass interpolated grade into blocks in the Indicated category and the second pass interpolated grade into blocks in the Inferred Category. Table 151 summarises the search parameters used in the estimation.

Table 151 Summary of search parameters for each Zone of the deposit

Parameter	**West Zone**		**AB Zone**		**D Zone**	
	Indicated	**Inferred**	**Indicated**	**Inferred**	**Indicated**	**Inferred**
Search Type	Ellipsoid		Ellipsoid		Ellipsoid	
Principle Azimuth	38°		322°		222°	
Principle Dip	39°		-64°		-71°	
Intermediate Azimuth	210°		174°		188°	
Anisotropy X	55	110	35	70	55	110
Anisotropy Y	55	110	35	70	55	110
Anisotropy Z	15	30	12.5	25	12.5	25
Min. Samples (MSS)	4	2	4	2	4	2
Max. Samples (MSS)	12	12	12	12	12	12
Min. Drillholes	2	1	2	1	2	1

- Statistical analysis between estimated blocks and input data was reviewed.
- Visual checks of block grades and drillhole data in plan and section.
- Extrapolation distances in general are 20m to 50m but occur up to 110m in less well drilled areas.

Moisture	▪ Bulk density measurement was conducted on dried samples and tonnes in the model have been estimated on a dry basis.
Cut-off parameters	▪ Mineral Resources have been reported at cut-off grades of between 2.0 and 4.0% CuEq. ▪ The cut-off grades represent material that has reasonable prospects for eventual economic extraction approximately within the next 15 years. ▪ The cut-off grades are based on the expectation that the High Lake deposits will be mined by underground and open pit methods.
Mining Factors or assumptions	▪ Mining factors or assumptions have not been applied to the Mineral Resource. ▪ The project is currently in the final stages of a Feasibility Study. ▪ The West Zone would be mined by underground mining methods and the AB and D zones would be mined by open pit mining methods.
Metallurgical factors or assumptions	▪ Metallurgical factors or assumptions have not been applied to the Mineral Resource. ▪ Metallurgical test work was recently completed for the areas of Indicated Mineral Resource and selected areas of the Inferred Mineral Resource. Test work included mineralogy, comminution tests, flowsheet development, variability tests, production estimation and flotation product testing. ▪ The key assumption for the High Lake deposits is that they will be mined in conjunction with the Izok deposit and the production of concentrate on a year-by-year basis, will assume an approximate blend of 67% Izok and 33% High Lake ore.
Environmental factors or assumptions	▪ Environmental factors or assumptions have not been applied to the Mineral Resource. ▪ MMG is currently in the process of completing a Feasibility Study and baseline data collection is ongoing for the approvals process for development of the High Lake Project and the Izok Corridor Project.
Bulk Density	▪ Approximately 200 pulp samples were collected in 2005 and were comprised of samples derived from each of the mineralisation zones. SG determinations were conducted on pulps using a pycnometer. This is an acceptable method as there are no or limited naturally occurring voids or cavities.

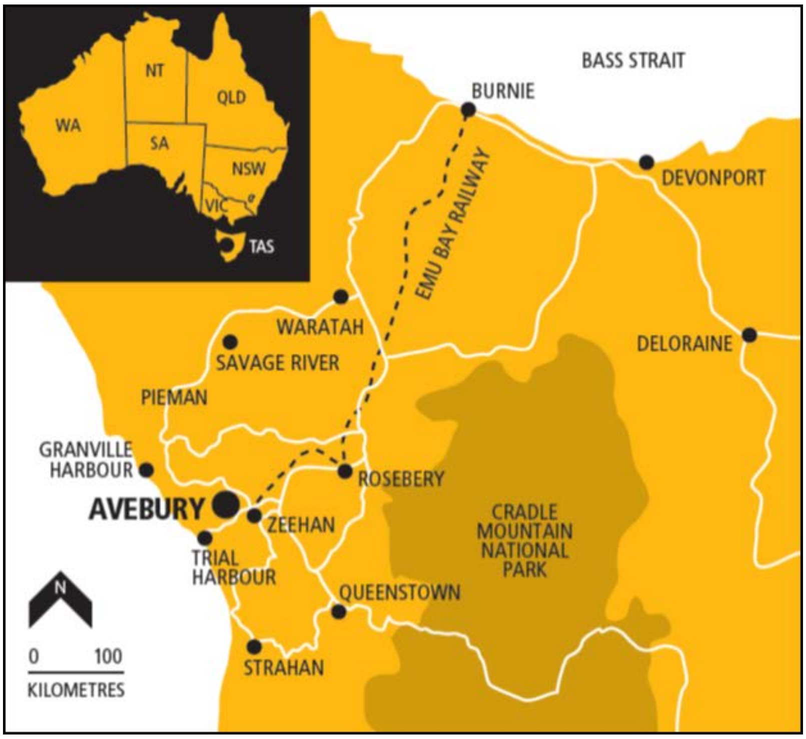
	- The updated MMG database for the High Lake deposit totals 1,469 samples of mineralized and unmineralised material. The SG analysis for samples, including samples from the metallurgical test holes, were completed by ALS in the lab by either the WW/WA on whole core or on pulverized material using a pycnometer. Of the 1,469 samples, densities for 1,094 or 74% of samples were determined by the WW/WA method. - A single SG value was used for each mineralisation and geological model based on an analysis of the SG data within each domain. The SG values for each mineralised and geological model were determined based on an analysis of density data within each model.
Classification	- Classification is based on data spacing and distribution relative to the distribution and continuity of Cu, Pb, Zn, Ag and Au mineralisation which is often coincident with geological contacts. - The Mineral Resource estimate in areas with drill spacing of 30m to 50m or less is classified as Indicated and in areas with drill densities of greater than 50m is classified as Inferred.
Audits or reviews	- Reviews of the current Mineral Resource estimate was completed by Optiro Pty Ltd out of Perth Australia as well as MMG personnel. - Audits of previous Mineral Resource Estimate were undertaken by Hatch, Behre Dolbear Australia (BDA), and MMG personnel. - Few issues with the drill data were identified and recommendations were made for future work. Recommendations included improved QAQC processes be implemented, a greater number of dry bulk density measurements to be taken, improved assay techniques, more strict criteria used to classify Indicated and Inferred Mineral Resources. More recent drill programs have improved the integrity of the database.

12. AVEBURY

12.1 Introduction and Setting

The Avebury Nickel Sulphide Mine is located 10km west of Zeehan on the west coast of Tasmania. This estimate covers all Mineral Resources contained on Mining Lease (ML) 3M/2003 and ML 6M/2007, which are held by Allegiance Metals, a wholly owned subsidiary of MMG.

Figure 101 Location of the Avebury Nickel Mine



12.2 Geological Setting

The Avebury Nickel Sulphide deposit is hosted in moderately to steeply dipping Cambrian ultramafic intrusive rocks belonging to the McIvor Hill Mafic-Ultramafic Complex.

The whole sequence has undergone moderate contact metamorphism to hornfels accompanied by mild to strong metasomatism during the intrusion of the Heemskirk Granite at the end of the Devonian Tabberabberan Orogeny. Variable metasomatism of the ultramafic rock has formed two distinctly different gangue mineral assemblages; a serpentinite-magnetite gangue (SERP) or an intensely metasomatised tremolite-diopside-magnetite gangue (SKSP). The ultramafic shows a moderately tight antiform geometry gently plunging to the west. Most of the nickel sulphide mineralisation is located within the ultramafic immediately adjacent to its margins. Nickel grades diminish toward the interior of the ultramafic body. Mineralisation is dominated by a pentlandite-pyrrhotite-magnetite assemblage with much lesser millerite, gersdorfite and niccolite.

Mineralised true widths vary from 4m to 40m and average around 10m true width. Mineralised lenses are generally around 50m-600m in length and can extend over 400m down-dip.

12.3 Mineral Resources

12.3.1 Results

All Mineral Resources quoted in this report were estimated from three dimensional block models created with Datamine software. Mineral Resources are modelled using solid wireframes of geological boundaries and/or a minimum 0.4% Ni cut-off boundary which approximates the natural break between nickel mineralisation and background grades.

Ni, As, Co, MgO, FeO and S grades were interpolated using an ordinary kriging algorithm. Variogram and estimation parameters were defined using Supervisor Software. Estimates were modelled on geological domains and density derived from a FeO and MgO indexed formula based on Archimedes method bulk density tests and ICP chemical analysis.

The Avebury Mineral Resource estimate as at June 30 2013 is summarised in Table 152. The Mineral Resource estimate remains unchanged since June 2011 and incorporates all drilling results received until June 2011. No further drilling has been completed since the June 2011 estimate.

Table 152 Avebury Mineral Resource as at June 30 2013 reported above a 0.4% Ni cut-off grade

0.4% cut-off *	Tonnes (Mt)	Ni %	As ppm	Co ppm	MgO %	FeO %	S %	SG
Measured	3.8	1.1	410	245	28	11.2	1.4	3.2
Indicated	4.9	0.9	352	244	25	11.9	1.4	3.2
Inferred Avebury	8.5	0.9	378	216	25	8.5	1.3	3.1
Inferred East Avebury	12.2	0.8	241	227	32	11.3	0.8	2.9
Inferred	20.7	0.8	297	223	29	10.2	1.0	3.0
Total	**29.3**	**0.9**	**321**	**229**	**28**	**10.6**	**1.1**	**3.0**

*All Mineral Resources quoted as total nickel, a nickel recovery of 74% is expected using conventional flotation processes.

12.3.2 Statement of Compliance with JORC Code Reporting Criteria and Consent to Release

This Mineral Resource statement has been compiled in accordance with the guidelines defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("2012 JORC Code").

Competent Person Statement

I, Peter Carolan, confirm that I am the Competent Person for the Avebury Mineral Resources section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining.
- I have reviewed the relevant Avebury Mineral Resources section of this Report to which this Consent Statement applies.

I was a full time employee of MMG (at the time of estimation).

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Avebury Mineral Resources section of this Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Competent Person Consent

Pursuant to the requirements Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

With respect to the sections of this report for which I am responsible – the Avebury Mineral Resources - I consent to the release of the 2013 Mineral Resources and Ore Reserves Statement as at 30 June 2013 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

Peter Carolan – 18/4/13

Jared Broome (Witness)

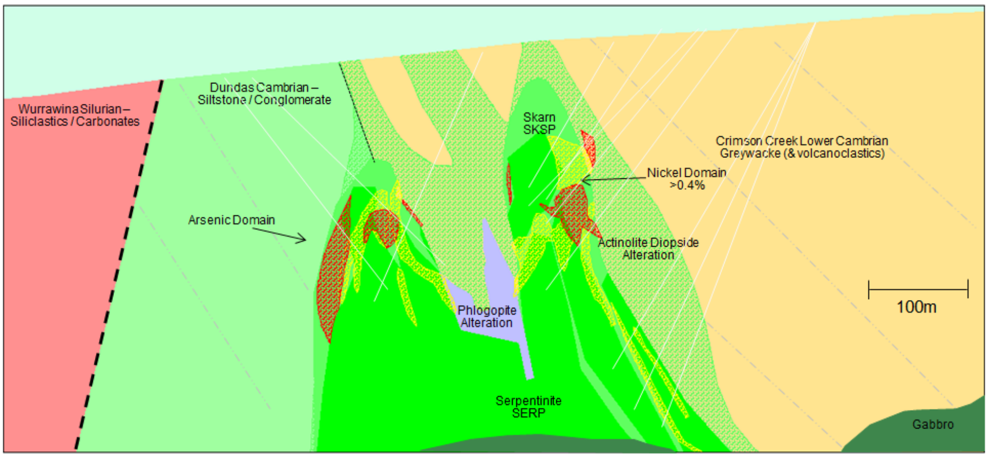
12.4 Mineral Resource JORC 2012 Assessment and Reporting Criteria

The following table follows the requirements of JORC TABLE 1 sections 1, 2 and 3 as applicable for the estimation and reporting of the Avebury Mineral Resources.

Table 153 Checklist of assessment and reporting criteria for Avebury Mineral Resource

Criteria	**Status**
Section 1 Sampling Techniques and Data	
Sampling techniques	▪ Diamond drilling was used to obtain nominal 1m length (+/- 0.5m) half core samples that were submitted for analysis.
Drilling techniques	▪ Diamond drilling was used to produce NQ or equivalent LTK60 size diamond core. ▪ The database contains 456 diamond holes for 118,000m (156 holes were collared from surface with the remainder collared from underground).
Drill sample recovery	▪ Recovery recorded during core logging was generally 100%, with minor losses in broken ground. There is no relationship between core loss and mineralisation or grade.
Logging	▪ Core logging recorded geological and geotechnical information including lithology, stratigraphy, weathering, alteration, strength, RQD, number of defects, defect type, defect to core angle, roughness and infill material. ▪ Core is stored in a core shed in the township of Zeehan. ▪ Core photographs are available for most drillholes.
Sub-sampling techniques and sample preparation	▪ Core was split in half by diamond saw. Sample lengths were cut as close to 1m as possible while respecting geological contacts. ▪ Core samples were then bagged, numbered and dispatched to assay laboratories. ▪ Samples were generally 2kg to 3kg in weight. ▪ Laboratory process followed drying, crushing, milling and homogenising entire sample to 80% passing 0.75µm.
Quality of assay data and laboratory tests	▪ Laboratory analysis is considered to be total with the following methods applied at various times: ▪ Pre-2005: 4-acid digest and analysis of Ni, As, Co, S by ICP_AES at Analabs laboratories, Townsville. ▪ 2005 to 2009: Pressed powder XRF analyses for Ni, As, Co, S, FeO, MgO at Burnie Research Laboratories, Burnie. ▪ Post-2009: 4-Acid digest with ICPAES analysis at ALS laboratories Perth. ▪ Internal standards and pulp duplicates were submitted with every batch of samples. ▪ Approximately 1 in every 10 submissions was sent for independent laboratory analysis (Amdel laboratories, Adelaide, ALS Laboratories Perth). ▪ A re-assay program of 2008 drill samples (pressed powder XRF method) by 4-acid ICP analysis identified a +6% Ni bias and a -21% As bias in the XRF results. ▪ The XRF analysis method for samples returning values less than 100ppm As is considered inaccurate. The mean ICP result of repeat analysis for <100ppm As was 24ppm As. ▪ Corrections for the sub 100ppm As XRF values and the As bias identified were factored into a run of the estimation process. This produced a result showing of a 10% increase in As metal in the Arsenic domain areas of the Measured Mineral Resource. Overall a difference of <1% in As grade occurred for the entire Mineral Resource. This factor has not been applied to the Mineral Resource Estimate.
Verification of sampling and assaying	▪ Assay results were verified against logging and core photos. ▪ Routine twinning of holes was not carried out. Infill drilling was carried out to improve confidence in the Mineral Resource and upgrade it from Inferred to Indicated and Measured categories. ▪ Core logging data was recorded in Excel spread sheets by experienced geologists. ▪ Drill logs were loaded into a site Access Database up until 2009. The Access database was then transferred into a GIBIS database on the MMG Server with subsequent Excel drill logs were loaded into it directly.
Location of data points	▪ All drillhole collar surveys by were undertaken by a licensed surveyor. ▪ All coordinates are in Mine Grid Plane Projection (a close approximation of AGD66). ▪ Strong local magnetic fields associated with Avebury mineral deposit reduce the effectiveness of conventional down-hole survey tools. ▪ Post-2005 surface drillholes were gyroscopically surveyed. ▪ Pre-2005 surface drillholes surveyed by Maxibor or had azimuth corrected from nearby Maxibor holes. ▪ Down-hole dip taken was recorded from digital survey tool or Eastman single shot cameras. ▪ Underground drillhole azimuth was recorded as collar azimuth for holes collared east of 345350 E. ▪ Underground drillhole azimuth was recorded as collar azimuth +1° per 50m down hole distance for holes collared west of 345350 E.

Data spacing and distribution	▪ Drill spacing approximately < 50m x 20m for Measured areas of Mineral Resource. ▪ Drill spacing approximately < 60m x 40m for Indicated areas of Mineral Resource. ▪ Drill spacing approximately 100m x 100m for Inferred areas of Mineral Resource. ▪ The distribution and continuity of nickel mineralisation is often is coincident with geological contacts, these features identified in drilling are demonstrated by underground mapping. The data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource estimation procedures and classifications applied. ▪ The distribution of the deleterious arsenic content is not as regular as the nickel distribution, and is less well supported by geological indicators. For this reason confidence in the arsenic content and distribution is lower than for nickel. It is assumed that production sampling with blending and processing strategies will be sufficiently implemented to manage arsenic levels reporting to concentrates thus maintaining the appropriateness of the make the Mineral Resource classifications applied.
Orientation of data in relation to geological structure	▪ Geological mapping and interpretation show that the mineralisation forms in antiformal setting striking east-west. Hence drilling is conducted on north-south and south-north directions to intersect mineralisation across-strike. ▪ Drilling orientation is not considered to have introduced any sampling bias.
Sample security	▪ Measures to provide sample security included: – Adequately trained and supervised sampling personnel – Core yard facility with security fence and well maintained sampling sheds – Cut core samples stored in numbered and tied calico sample bags – Calico sample bags transported by courier to assay laboratory – Assay laboratory checks of sample dispatch numbers against submission documents
Audit and reviews	▪ Review of the 2007 Mineral Resource estimate was undertaken by AMC Consultants. This review found that many of the processes and systems set up by Alligiance Mining were industry best and/or good-practice. No fundamental or high risk issues were identified. Recommendations included improved QAQC processes be implemented, and a greater volume of dry bulk density measurements to be taken. ▪ An internal MMG Scoping Study Review was undertaken in 2009. No high risk issues were identified.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	▪ The Avebury Mineral Resource is located within the bounds of Mining lease (ML) 3M/2003 and ML 6M/2007. Mineral Resources within ML 6M/2007 are identified as "East Avebury" in Mineral Resource table. ▪ ML 3M/2003 and ML 6M/2007 are held by Allegiance Mining a subsidiary of MMG and have an expiry date on the 16/10/2024. ▪ A royalty of 2% Net Smelter Return from mining within ML 3M/2003 and ML 6M/2007 payable to Royalty Gold Incorporated applies. ▪ A royalty rate of 5.5% payable to the Tasmanian Government applies. ▪ An Agreement for the Purchase and Sale of Nickel Concentrates is in place with Jinchuan Group Ltd and is applicable to all nickel concentrates produced from the Avebury Nickel Project.
Exploration done by other parties	▪ A zinc exploration joint venture between CRA Exploration Pty Limited and Allegiance Mining over the period 1991 to 1997 identified elevated nickel in stratigraphic exploration drillholes targeting magnetic anomalism. ▪ In January 1998 Allegiance Mining drilled the discovery hole A001 into the Central Avebury Orebody.
Geology	▪ The Avebury Nickel Sulphide deposit is hosted in moderately to steeply dipping Cambrian ultramafic intrusive rocks belonging to the McIvor Hill Mafic-Ultramafic Complex. ▪ The whole sequence has undergone moderate contact metamorphism to hornfels accompanied by mild to strong metasomatism during the intrusion of the Heemskirk Granite at the end of the Devonian Tabberabberan Orogeny. Variable metasomatism of the ultramafic rock has formed two distinctly different gangue mineral assemblages; a serpentinite-magnetite gangue (SERP) or an intensely metasomatised tremolite-diopside-magnetite gangue (SKSP). The ultramafic shows a moderately tight antiform geometry gently plunging to the west. Most of the nickel sulphide mineralisation is located within the ultramafic immediately adjacent to its margins. Nickel grades diminish toward the interior of the ultramafic body. Mineralisation is dominated by a pentlandite-pyrrhotite-magnetite assemblage with much lesser millerite, gersdorfite and niccolite. ▪ Mineralised widths vary from 4m to 40m and average around 10m true width. Mineralised lenses are generally around 50m to 600m in length and can extend over 400m down-dip.
Drillhole information	▪ 456 diamond drillholes and associated data are held in the database. No individual hole is material to the Mineral Resource estimated and hence this geological database is not supplied. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

Data aggregation methods	▪ No metal equivalents were used in the Mineral Resource estimation ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Relationship between mineralisation width and intercepts lengths	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Diagrams	**Figure 102** **Generalised north-south cross-section facing west of the Avebury deposit**  ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Balanced reporting	▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	▪ All diamond drillhole information was considered for this Mineral Resource estimation. ▪ This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	▪ No future work program is currently planned. ▪ The operation is on care and maintenance.
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	▪ All data was stored in customised access database and was converted to the MMG GBis database by the MMG Exploration Department in 2009/10. ▪ All logging was entered into Microsoft Excel and loaded into the database. ▪ Assay data was loaded from Microsoft Excel directly into database pre 2009. Post 2009 laboratory files were directly loaded into GBis. ▪ Data integrity was validated for EOH depth and sample overlaps. ▪ Manual checks were carried out by plotting and review of sections and plans. ▪ Drillhole A007 was been removed from the database due to inaccurate survey results.
Site visits	▪ The Competent Person visited site on various occasions through 2010/11. Site visits included: – Involvement in the planning and execution of exploration / extensional drilling programs – Inspection of core handing, logging, sampling procedures and of facilities – Inspection of geological mapping plans – Inspection of underground workings – Inspection of Burnie Research Laboratory analysis facilities
Geological interpretation	▪ Mineralisation is hosted in Middle Cambrian ultramafic bodies intruding Cambrian volcaniclastic sediments. ▪ Both host volcaniclastic and ultramafic intrusions are steeply north dipping in an overturned south facing sequence. ▪ The stratigraphy and intrusions broadly strike east-west. ▪ Devonian Granite intrusion has strongly hornfelsed and locally metasomatised the host sequence. ▪ Gangue mineralogy is either black serpentinite-magnetite or a pale grey-green diopside-tremolite-magnetite.

	▪ Mineralisation consists of coarse disseminated and stringer pentlandite with minor pyrrhotite. ▪ Nickel arsenides (Niccolite, Maucherite, Gersdorfite), although sparse are contributors to the penalty element. They occur in elevated concentrations in discreet zones both parallel with and cross-cutting the main Ni bearing mineralisation. ▪ Mineral Resource estimation was made using Datamine Software. Separate Nickel, Arsenic and Ultramafic domains were wireframe modelled using north-south cross sections, respecting geological contacts and down-hole geochemical data. ▪ Nickel domains are delineated on the SKSP/SERP to Volcanoclastics contact and a 0.4% Ni cut-off which is the natural break between background ultramafic Nickel and elevated Nickel sulphides. Coarse pentlandite mineralisation is visible above 0.4% Ni. ▪ Separate wireframes were modelled for high arsenic (>300ppm) domains. ▪ Although confidence in geometries defined by Measured drill spacing is adequate for mining assessment, infill drilling for development and stope margin definition is required prior to mining. This is carried out on a 25m x 15m or closer spacing. ▪ Confidence in geological interpretation of Inferred mineralisation is at a lower level than Indicated and Measured mineralisation due to the limited sampling in these areas, hence implied but not verified geological and grade continuity occurs.
Dimensions	▪ Mineralised lenses are located on the flanks of the antiformal ultramafic body. True widths vary from 4m to 40m and average around 10m true width. Lenses are between 50m - 600m in length and can extend over 400m down-dip. ▪ Mineralisation extends between: – 353700mE to 355900mE – 5357100mN to 5357750mN – 1550mRL to 2200mRL
Estimation and modelling techniques	▪ The Avebury Mineral Resource is located within the bounds of ML 3M/2003 and ML 6M/2007. Mineral Resources within ML 6M/2007 are identified as "East Avebury Inferred" in Mineral Resource table. ▪ Model attributes were interpolated using an ordinary kriging algorithm. ▪ Parent block size was set to 10m x 10m x 10m with sub blocks 1.25m x 2.5m x 1.25m. The selective mining unit is approximately 25m x 25m x 5m. ▪ For the estimate sample intervals were composited to approximately 1m so that no residuals were created. ▪ 40 Nickel domains based on SKSP/SERP to Volcanoclastics contact and cut-off of 0.4% Ni. Domains do include internal dilution. Domains at times consist of 2 to 3 lenses. Lenses range in size from 50m x 50m x 4m up to 300^{+}m x 200m x 20^{+}m. These domains were used for the estimation of Ni, S and Co. ▪ 24 Arsenic domains based on a 300ppm cut-off where >4m width, Arsenic samples outside of these domains were top cut to 5000ppm (0.2% of samples). ▪ Background ultramafic, skarn and host rock domains were used in the estimation of MgO and FeO and background Ni, Co, S and As grade. In these background domains un-estimated blocks were assigned the following grades to assist with density calculations; Ni%= 0.01, MgO% = 16.52 , FeO% = 8.36, S%= 0.4, As ppm =5, Co ppm =5. ▪ The estimate of each element was undertaken using hard domain boundaries and a series of elliptical search passes orientated in the plane of mineralisation. These search orientations and sizes were supported by variography analysis. ▪ The first estimation search pass was 120m x 80m x 40m, additional larger passes were used to estimate less well informed blocks. ▪ The first estimation search pass employed a minimum of 8 and maximum of 32 samples and a minimum of 3 octants with a minimum of 1 and maximum of 16 samples per octant. Estimates were also limited to a maximum of 4 samples from any given drillhole. Additional passes used more relaxed criteria to estimate the less well informed blocks. ▪ Statistical analysis between estimated blocks and input data was reviewed. ▪ Visual checks of block grades and drillhole data in plan and section. ▪ Extrapolation distances in general are 25m to 50m but occur up to 100m in less well drilled area.
Moisture	▪ Bulk density measurement was conducted on oven dried samples and tonnes in the model have been estimated on a dry basis.
Cut-off parameters	▪ Mineral Resources have been reported at a 0.4% Ni cut-off. This represents all material within the mineralised Nickel domains. ▪ This Mineral Resource cut-off represents material that has reasonable prospects for eventual economic extraction

at some point within the next 15 years.

- While in operation for the selected mining method the Mineral Resource (at a 0.4% Ni cut-off) to Ore Reserves conversion was approximately 40%
- For comparison purposes, the Mineral Resource at a 0.7% Ni block grade cut-off is presented in Table 154.

Table 154 Avebury Mineral Resource 0.7% Ni block grade cut-off

0.7% cut-off *	Tonnes (Mt)	Ni %	As ppm	Co ppm	MgO %	FeO %	S %	SG
Measured	3.4	1.2	412	255	28	11.2	1.5	3.2
Indicated	3.8	1.0	383	262	25	11.8	1.5	3.2
Inferred Avebury	6.1	1.1	451	243	24	8.6	1.5	3.1
Inferred East Avebury	6.5	0.9	308	267	29	11.2	1.0	3.0
Inferred All	12.6	1.0	378	255	26	9.9	1.2	3.1
Total	**19.8**	**1.0**	**385**	**256**	**26**	**10.5**	**1.3**	**3.1**

Mining Factors or assumptions

- No mining factors have been applied to the Mineral Resource.
- While in production decline access mining involved 25m sub-level open stoping using transverse and longitudinal methods of widths 5m to 10m. Mined stopes were filled with rock fill and Cemented Rock Fill (CRF).
- The site is currently on care and maintenance as a result of becoming uneconomic during the 2008 Global Financial Crises.

Metallurgical factors or assumptions

- Metallurgical test work was completed for the Measured and Indicated areas of the Mineral Resource and selected areas of the Inferred Mineral Resource. Test work included standard variability, comminution, grinding and float tests and the treatment of bulk samples from selected mineralisation types.
- The metallurgical processing plant containing crushing, grinding and floatation stages to produce nickel sulphide concentrate operated between mid-2008 and early 2009. The plant has a nameplate design of 900ktpa at 79% Ni recovery to a 20%+ Ni in concentrate grade.
- The plant is currently on care and maintenance.
- A small portion of nickel concentrate produced during the period of operation contained arsenic levels which reached unsaleable levels.
- It is assumed that production scheduling, blending, and processing strategies will enable the sale of future concentrates. It is assumed that concentrate limits for deleterious elements will not change.
- All Mineral Resources are quoted as total nickel.

Environmental factors or assumptions

- Avebury operates under Land use permit (DA P7/2004) issued by the Tasmanian Environmental Protection Authority (EPA) dated 29 June 2005.
- Environmental Protection Notice (EPN 7446/2) for mining to 900ktpa on ML 3M/2003 was issued by the EPA in July 2009.
- An application for an EPN for mining on ML 6M/2007 has been submitted to the EPA but has not progressed by either party due to suspension of operations.
- Licence exceedance for water discharge is an ongoing issue which has been recognised by the EPA to be caused by inappropriate licence conditions. MMG has received formal notification from the EPA that the discharge is not causing additional environmental harm.

Bulk Density

- Bulk density measurements are undertaken by the weight in air (oven dried) /weight in water technique. No sealing of core was undertaken as core porosity is low. The density measurements were compared against elemental compositions to generate Indexed density formulas for SERP and SKSP rock types.
- The shown Indexed density formulas were applied to the estimated blocks grades in each domain to calculate the resultant dry bulk density

$$\text{SKSP} = 0.029 * (-0.85 * \text{FeO\%} + \text{MgO\%} * 0.6) + 3.4$$

$$\text{SERP} = 0.065 * (0.3 * \text{FeO\%} + 0.6 * \text{S\%} + 0.1 * \text{Ni\%}) + 2.44$$

$$\text{HOST} = 2.89$$

Classification

- Mineral Resource classification is based on data spacing and distribution relative to the distribution and continuity of nickel mineralisation which is often coincident with geological contacts. These features identified in drilling are demonstrated by mapping of underground development exposures.
- Measured Mineral Resource areas contain a drill spacing of < 50m x 20m and are also accessed by mine level development.
- Indicated Mineral Resource areas contain a drill spacing of < 60m x 40m.

	▪ Inferred Mineral Resource areas contain a drill spacing of approximately 100m x 100m. ▪ Classification is supported by reconciliation of production results against past Mineral Resource estimates. At the end of February 2009 reconciliation against Mill figures of the previous 6 month production period showed that the then current Mineral Resource model estimated to within +-10% for Nickel and Arsenic grades, for metal content and for tonnes (assumptions on dilution grade were made). ▪ The distribution of the deleterious arsenic content is not as regular as the nickel distribution, and is less well supported by geological indicators. For this reason confidence in the arsenic content and distribution is lower than for nickel. It is assumed that production sampling with blending and processing strategies will be sufficiently implemented to manage arsenic levels reporting to concentrates thus maintaining the appropriateness of the Mineral Resource classifications applied.
Audits or reviews	▪ No audit or review has been carried out on the current Mineral Resource estimate. ▪ A full audit of 2005 Mineral Resource Estimate was undertaken by AMC Consultants. ▪ Review by Behre Dolbear Australia (BDA) took place in 2007. ▪ A review of the 2007 Mineral Resource estimate was undertaken by AMC Consultants. This review found that many of the processes and systems set up by Allegiance Mining were industry best and/or good-practice. No fundamental flaws or high risk issues were identified. Recommendations included a review of variography and a review of the classification procedure to take into account variography, interpretation from drilling and from underground exposures and mining reconciliations. ▪ An internal MMG Scoping Study Review was undertaken in 2009. No high risk issues were identified.
Discussion of relative accuracy / confidence	▪ Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support. ▪ Reconciliation over a 6 month production period showed that the then current Mineral Resource model estimated to within +-10% for Nickel and Arsenic grades, for metal content and for tonnes (assumptions on dilution grade were made). This supports the fundamentals of the support data and estimation process. This provides an indication of the expected relative accuracy of the Mineral Resources classified as Measured in the estimate.

13. EXTERNAL REFERENCES

Smith, S. G., Olberg, D., and Manini, A. J., 2005, The Sepon gold deposits, Laos: exploration, geology and comparison to Carlin-type gold deposits in the Great Basin: Geological Society of Nevada Symposium, Reno, Nevada, 2005.

Kelso, I., Briggs, T., Basford, P., 2001. The Century Zinc Deposit – Geological Update. AIG Journal, 2001, 03).

Broadbent, G.C., Andrews, S.J., Kelso, I.J. 2002. A Decade of New Ideas: Geology and Exploration History of the Century Zn-Pb-Ag Deposit, Northwestern Queensland, Australia – Society of Economic Geologists, Chapter 6.