MMG LIMITED

五礦資源有限公司

(Incorporated in Hong Kong with limited liability)

(HKEX STOCK CODE: 1208)
(ASX STOCK CODE: MMG)

MINERAL RESOURCES AND ORE RESERVES STATEMENT AS AT 30 JUNE 2017

This announcement is made by MMG Limited (Company or MMG and, together with its subsidiaries, the Group) pursuant to rule 13.09(2) of the Rules Governing the Listing of Securities on The Stock Exchange of Hong Kong Limited (Listing Rules) and the Inside Information Provisions (as defined in the Listing Rules) under Part XIVA of the Securities and Futures Ordinance (Chapter 571 of the Laws of Hong Kong).

The board of directors of the Company (Board) is pleased to report the Group's updated Mineral Resources and Ore Reserves Statement as at 30 June 2017 (Mineral Resources and Ore Reserves Statement).

The highlights of the Mineral Resources and Ore Reserves Statement as at 30 June 2017 include:

- The Group's Mineral Resources (contained metal) have increased for lead (3%) and have decreased for copper (10%), zinc (10%), silver (10%), gold (14%) and molybdenum (3%).
- The Group's Ore Reserves (contained metal) have increased for zinc (28%), lead (41%), silver (2%) and molybdenum (1%) and decreased for copper (6%) and gold (11%).
- Mineral Resources tonnes at Kinsevere and Dugald River have increased by 4.5Mt and 3.9Mt respectively.
- Ore Reserves tonnes at Dugald River and Rosebery increased by 10.3Mt and 0.2Mt respectively.
- Las Bambas Mineral Resources tonnes have decreased by 250Mt.
- Las Bambas Ore Reserves tonnes have decreased by 0.5Mt.
- Mineral Resources and Ore Reserves for Golden Grove and Avebury have been removed from the Mineral Resources and Ore Reserves Statement due to divestment of these assets.

All data reported here is on a 100% asset basis, with MMG's attributable interest shown against each asset within the Mineral Resources and Ore Reserves tables (pages 4 to 8).

MINERAL RESOURCES AND ORE RESERVES STATEMENT

A copy of the executive summary of the Mineral Resources and Ore Reserves Statement is annexed to this announcement.

The information referred to in this announcement has been extracted from the report titled Mineral Resources and Ore Reserves Statement as at 30 June 2017 published on 18 October 2017 and is available to view on www.mmg.com. The Company confirms that it is not aware of any new information or data that materially affects the information included in the Mineral Resources and Ore Reserves Statement and, in the case of estimates of Mineral Resources or Ore Reserves, that all material assumptions and technical parameters underpinning the estimates in the Mineral Resources and Ore Reserves Statement continue to apply and have not materially changed. The Company confirms that the form and context in which the Competent Person's findings are presented have not been materially modified from the Mineral Resources and Ore Reserves Statement.

By order of the Board
MMG Limited
Jiao Jian
CEO and Executive Director

Hong Kong, 18 October 2017

As at the date of this announcement, the Board comprises nine directors, of which two are executive directors, namely Mr Jiao Jian and Mr Xu Jiqing; three are non-executive directors, namely Mr Guo Wenqing (Chairman), Mr Gao Xiaoyu and Mr Zhang Shuqiang; and four are independent non-executive directors, namely Dr Peter William Cassidy, Mr Leung Cheuk Yan, Ms Jennifer Anne Seabrook and Professor Pei Ker Wei.

MMG Limited

MINERAL RESOURCES AND ORE RESERVES STATEMENT

30 June 2017

EXECUTIVE SUMMARY

Mineral Resources and Ore Reserves for MMG have been estimated as at 30 June 2017, and are reported in accordance with the guidelines in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (2012 JORC Code) and Chapter 18 of the Listing Rules. Mineral Resources and Ore Reserves tables are provided on pages 4 to 8, which include the 30 June 2017 and 30 June 2016 estimates for comparison. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources that convert to Ore Reserves. All supporting data is provided within the Technical Appendix, available on the MMG website.

Mineral Resources and Ore Reserves information in this statement has been compiled by Competent Persons (as defined by the 2012 JORC Code). Each Competent Person consents to the inclusion of the information in this report that they have provided in the form and context in which it appears. Competent Persons are listed on page 9.

MMG has established processes and structures for the governance of Mineral Resources and Ore Reserves estimation and reporting. MMG has a Mineral Resources and Ore Reserves Committee that regularly convenes to assist the MMG Governance and Nomination Committee and the Board of Directors with respect to the reporting practices of the Company in relation to Mineral Resources and Ore Reserves, and the quality and integrity of these reports of the Group.

Key changes to the Mineral Resources (contained metal) since the 30 June 2016 estimate have been mostly related to depletion[1] together with lower price assumptions which have impacted Las Bambas and resulted in a net decrease in contained copper metal. The divestment of Golden Grove has also contributed to the global copper metal decrease in Mineral Resources. The contained zinc metal in the Mineral Resources has decreased almost entirely due to the divestment of Golden Grove. The sale of the Avebury asset in Tasmania has resulted in the removal of nickel from the Mineral Resources statement.

The MMG Ore Reserves (contained metal) have increased since the 30 June 2016 statement for zinc and lead principally due to increases at Dugald River and Rosebery. Decreases in Ore Reserves (contained metal) for copper are the result of depletion[1] at Las Bambas, Sepon and Kinsevere combined with the divestment of Golden Grove. Decreases of Indicated Mineral Resources at Sepon have resulted in a reduction of available material for Ore Reserves conversion.

Total tonnes of Mineral Resources and Ore Reserves have decreased with depletion and divestment. In addition, Mineral Resources have also decreased due to copper price assumptions. Las Bambas Mineral Resources and Ore Reserves have decreased by 250Mt and 0.5Mt respectively. Dugald River Mineral Resources and Ore Reserves tonnes have increased by 4Mt and 10Mt respectively. Sepon Mineral Resources and Ore Reserves have decreased by 2.3Mt and 5.5Mt respectively, while Kinsevere Mineral Resources have increased by 4.5Mt and Ore Reserves have decreased by 4.4Mt.

Pages 10 and 11 provide further discussion of the Mineral Resources and Ore Reserves changes.

[1] Depletion in this report refers to material treated by the mill and depleted from the Mineral Resources and Ore Reserves through mining.

MMG Limited

MINERAL RESOURCES AND ORE RESERVES STATEMENT

30 June 2017

MINERAL RESOURCES[1]

All data reported here is on a 100% asset basis, with MMG's attributable interest shown against each asset within brackets.

	2017							2016						
Deposit	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**
Las Bambas (62.5%)														
Ferrobamba Oxide Copper														
Indicated	9.3	2.0						16.8	2.0					
Inferred	0.6	2.5						0.7	1.9					
Total	**9.9**	**2.0**						**17.4**	**2.0**					
Ferrobamba Primary Copper														
Measured	542	0.64			3.0	0.06	204	529	0.68			3.3	0.06	198
Indicated	546	0.60			2.8	0.05	211	527	0.59			2.7	0.05	191
Inferred	263	0.60			2.4	0.04	158	397	0.57			2.1	0.03	146
Total	**1,351**	**0.62**			**2.8**	**0.05**	**198**	**1,453**	**0.62**			**2.7**	**0.05**	**181**
Ferrobamba Total	**1,361**							**1,471**						
Chalcobamba Oxide Copper														
Indicated	6.1	1.5						6.5	1.5					
Inferred	0.7	1.5						0.9	1.5					
Total	**6.8**	**1.5**						**7.3**	**1.5**					
Chalcobamba Primary Copper														
Measured	85	0.37			1.1	0.01	148	94	0.40			1.2	0.01	148
Indicated	195	0.67			2.5	0.03	141	196	0.63			2.4	0.03	145
Inferred	36	0.52			1.8	0.02	141	48	0.47			1.6	0.02	131
Total	**315**	**0.57**			**2.0**	**0.03**	**143**	**338**	**0.55**			**1.9**	**0.02**	**144**
Chalcobamba Total	**322**							**345**						
Sulfobamba Primary Copper														
Indicated	85	0.67			4.7	0.02	170	103	0.60			4.1	0.02	162
Inferred	100	0.58			6.5	0.02	119	201	0.44			4.0	0.02	119
Total	**184**	**0.62**			**5.7**	**0.02**	**142**	**304**	**0.50**			**4.0**	**0.02**	**133**
Sulfobamba Total	**184**							**304**						
Oxide Copper Stockpile														
Indicated	5.5	1.0						3.4	0.9					
Total	**5.5**	**1.0**						**3.4**	**0.9**					
Primary Copper Stockpile														
Measured	0.2	0.85			4.5		148	0.37	0.7			3.1		214
Total	**0.2**	**0.85**			**4.5**		**148**	**0.37**	**0.7**			**3.1**		**214**
Las Bambas Total	**1,873**							**2,124**						

[1] S.I. units used for metals of value; Cu=copper, Zn=zinc, Pb=lead, Ag=silver, Au=gold, Mo=molybdenum, Ni=nickel.

MINERAL RESOURCES

	2017							2016						
Deposit	Tonnes (Mt)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Mo (ppm)	Tonnes (Mt)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Mo (ppm)
Kinsevere (100%)														
Oxide Copper														
Measured	3.0	4.4						3.1	4.6					
Indicated	13.6	3.0						13.7	3.1					
Inferred	2.8	2.3						3.5	2.4					
Total	**19.4**	**3.1**						**20.3**	**3.2**					
Transition Mixed Copper Ore														
Measured	0.27	2.7						0.7	3.4					
Indicated	1.4	2.3						2.0	3.0					
Inferred	0.12	2.1						0.2	2.2					
Total	**1.8**	**2.4**						**2.9**	**3.0**					
Primary Copper														
Measured	0.40	2.5						0.4	3.1					
Indicated	23.8	2.2						18.5	2.6					
Inferred	2.2	1.7						2.2	2.0					
Total	**26.4**	**2.2**						**21.2**	**2.5**					
Copper Stockpiles														
Measured														
Indicated	7.9	2.5						6.8	2.4					
Total	**7.9**	**2.5**						**6.8**	**2.4**					
Kinsevere Total	**55.5**							**51.2**						
Sepon (90%)														
Oxide Gold														
Measured														
Indicated	1.5					3.1		1.6					3.0	
Inferred	0.21					2.3		0.4					2.1	
Total	**1.7**					**3.0**		**2.0**					**2.8**	
Partial Oxide Gold														
Measured														
Indicated	1.1					4.3		1.3					4.2	
Inferred	0.05					3.2		0.1					2.9	
Total	**1.1**					**4.3**		**1.3**					**4.1**	
Primary Gold														
Indicated	7.1					3.9		7.8					4.0	
Inferred	0.11					3.0		0.1					3.5	
Total	**7.2**					**3.9**		**7.9**					**4.0**	
Supergene Copper														
Indicated	5.5	4.7						12.9	3.5					
Inferred	1.5	3.3						0.3	3.5					
Total	**7.0**	**4.4**						**13.3**	**3.5**					
Primary Copper														
Indicated	7.1	1.0						5.0	1.2					
Inferred	5.2	1.2						3.3	1.1					
Total	**12.2**	**1.1**						**8.4**	**1.2**					
Copper Stockpiles														
Measured														
Indicated	6.1	1.4						5.7	1.6					
Total	**6.1**	**1.4**						**5.7**	**1.6**					
Sepon Total	**25.4**							**38.6**						

MINERAL RESOURCES

Deposit	2017 Tonnes (Mt)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Mo (ppm)	2016 Tonnes (Mt)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Mo (ppm)
Dugald River (100%)														
Primary Zinc														
Measured	8.1		13.1	2.4	70			5.5		14.2	2.0	64		
Indicated	28.9		12.3	2.3	40			27.1		12.9	2.2	50		
Inferred	27.8		11.4	1.9	10			28.5		12.0	1.7	13		
Total	**64.8**		**12.0**	**2.2**	**31**			**61.1**		**12.6**	**1.9**	**34**		
Primary Copper														
Inferred	4.4	1.8				0.2		4.4	1.8				0.2	
Total	**4.4**	**1.8**				**0.2**		**4.4**	**1.8**				**0.2**	
Zinc Stockpiles														
Measured	0.23		10.8	1.7	49									
Dugald River Total	**69.4**							**66.0**						
Rosebery (100%)														
Rosebery														
Measured	6.0	0.26	9.3	3.3	118	1.4		5.4	0.25	8.1	2.9	107	1.3	
Indicated	6.2	0.26	7.9	2.6	112	1.3		5.7	0.25	7.6	2.6	102	1.2	
Inferred	6.5	0.30	7.4	2.7	90	1.4		11.2	0.26	8.0	2.7	95	1.4	
Total	**18.6**	**0.27**	**8.2**	**2.9**	**106**	**1.4**		**22.4**	**0.26**	**7.9**	**2.7**	**100**	**1.3**	
Rosebery Total	**18.6**							**22.4**						
High Lake (100%)														
Measured														
Indicated	7.9	3.0	3.5	0.3	83	1.3		7.9	3.0	3.5	0.3	83	1.3	
Inferred	6.0	1.8	4.3	0.4	84	1.3		6.0	1.8	4.3	0.4	84	1.3	
Total	**14.0**	**2.5**	**3.8**	**0.4**	**84**	**1.3**		**14.0**	**2.5**	**3.8**	**0.4**	**84**	**1.3**	
Izok Lake (100%)														
Measured														
Indicated	13.5	2.4	13.3	1.4	73	0.2		13.5	2.4	13.3	1.4	73	0.2	
Inferred	1.2	1.5	10.5	1.3	73	0.2		1.2	1.5	10.5	1.3	73	0.2	
Total	**14.6**	**2.3**	**13.1**	**1.4**	**73**	**0.2**		**14.6**	**2.3**	**13.1**	**1.4**	**73**	**0.2**	

MMG Limited

MINERAL RESOURCES AND ORE RESERVES STATEMENT

30 June 2017

ORE RESERVES[1]

All data reported here is on a 100% asset basis, with MMG's attributable interest shown against each asset within brackets.

	2017							2016						
Deposit	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**
Las Bambas (62.5%)														
Ferrobamba														
Primary Copper														
Proved	497	0.68			3.2	0.06	206	492	0.71			3.4	0.07	201
Probable	326	0.71			3.6	0.06	207	340	0.71			3.5	0.06	202
Total	**823**	**0.69**			**3.4**	**0.06**	**207**	**832**	**0.71**			**3.5**	**0.06**	**201**
Chalcobamba														
Primary Copper														
Proved	59	0.53			1.8	0.02	141	53	0.51			1.7	0.02	151
Probable	143	0.72			2.7	0.03	132	136	0.75			2.8	0.03	135
Total	**202**	**0.66**			**2.5**	**0.03**	**134**	**188**	**0.68**			**2.5**	**0.03**	**140**
Sulfobamba														
Primary Copper														
Proved														
Probable	60	0.80			5.9	0.03	161	66	0.78			5.5	0.03	176
Total	**60**	**0.80**			**5.9**	**0.03**	**161**	**66**	0.78			**5.5**	0.03	**176**
Primary Copper Stockpile														
Proved	0.17	0.85			4.5		148	0.37	0.72			3.1		**214**
Total	**0.17**	**0.85**			**4.5**		**148**	**0.37**	**0.72**			**3.1**		**214**
Las Bambas Total	**1,085**							**1,086**						
Kinsevere (100%)														
Oxide Copper														
Proved	2.6	4.5						2.9	4.5					
Probable	8.1	3.5						9.8	3.5					
Total	**10.7**	**3.7**						**12.7**	**3.7**					
Copper Stockpiles														
Proved														
Probable	2.5	3.6						4.9	2.2					
Total	**2.5**	**3.6**						**4.9**	**2.2**					
Kinsevere Total	**13.2**							**17.6**						
Sepon (90%)														
Supergene Copper														
Probable	3.5	4.7						8.0	3.5					
Total	**3.5**	**4.7**						**8.0**	**3.5**					
Primary Copper														
Probable	0.35	1.1						2.3	0.84					
Total	**0.35**	**1.1**						**2.3**	**0.84**					
Copper Stockpiles														
Probable	5.6	1.4						4.6	1.7					
Total	**5.6**	**1.4**						**4.6**	**1.7**					
Sepon Total	**9.4**							**14.9**						

[1] S.I. units used for metals of value; Cu=copper, Zn=zinc, Pb=lead, Ag=silver, Au=gold, Mo=molybdenum.

ORE RESERVES

	2017							2016						
Deposit	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**	**Tonnes (Mt)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Au (g/t)**	**Mo (ppm)**
Dugald River (100%)														
Primary Zinc														
Proved	7.9		11.8	2.1	62			4.6		12.3	1.7	55		
Probable	24.9		11.9	2.2	39			17.8		12.1	2.0	48		
Total	**32.8**		**11.9**	**2.2**	**44**			**22.5**		**12.2**	**2.0**	**50**		
Dugald River Total	**32.8**							**22.5**						
Rosebery (100%)														
Proved	3.8	0.25	9.0	3.4	119	1.4		3.2	0.25	8.8	3.1	110	1.3	
Probable	1.8	0.21	7.6	3.0	131	1.3		2.2	0.22	7.5	3.0	118	1.3	
Total	**5.6**	**0.24**	**8.6**	**3.3**	**123**	**1.4**		**5.4**	**0.24**	**8.3**	**3.0**	**113**	**1.3**	
Rosebery Total	**5.6**							**5.4**						

COMPETENT PERSONS

Deposit	Accountability	Competent Person	Professional Membership	Employer
MMG Mineral Resources and Ore Reserves Committee	Mineral Resources	Rex Berthelsen[1]	FAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Ore Reserves	Nan Wang[1]	MAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Metallurgy: Mineral Resources / Ore Reserves	Reinhardt Viljoen[1]	MAusIMM	MMG
Las Bambas	Mineral Resources	Rex Berthelsen[1]	FAusIMM(CP)	MMG
Las Bambas	Ore Reserves	Yao Wu[1]	MAusIMM	MMG
Las Bambas	Metallurgy: Mineral Resources / Ore Reserves	Amy Lamb[1]	MAusIMM	MMG
Sepon	Mineral Resources	Chevaun Gellie[1]	MAusIMM	MMG
Sepon	Ore Reserves	Jodi Wright[1]	MAusIMM(CP)	MMG
Sepon	Metallurgy: Mineral Resources / Ore Reserves	Kevin Rees	MAusIMM	MMG
Kinsevere	Mineral Resources	Douglas Corley[1]	MAIG R.P.Geo.	MMG
Kinsevere	Ore Reserves	Jodi Wright[1]	MAusIMM(CP)	MMG
Kinsevere	Metallurgy: Mineral Resources / Ore Reserves	Nigel Thiel[1]	MAusIMM(CP)	MMG
Rosebery	Mineral Resources	Anna Lewin	MAusIMM(CP)	MMG
Rosebery	Ore Reserves	Karel Steyn[1]	MAusIMM	MMG
Rosebery	Metallurgy: Mineral Resources / Ore Reserves	Kevin Rees	MAusIMM(CP)	MMG
Dugald River	Mineral Resources	Douglas Corley[1]	MAIG R.P.Geo.	MMG
Dugald River	Ore Reserves	Karel Steyn[1]	MAusIMM	MMG
Dugald River	Metallurgy: Mineral Resources / Ore Reserves	Nigel Thiel[1]	MAusIMM(CP)	MMG
High Lake, Izok Lake	Mineral Resources	Allan Armitage	MAPEG[2] (P.Geo)	Formerly MMG

The information in this report that relates to Mineral Resources and Ore Reserves is based on information compiled by the listed Competent Persons, who are Members or Fellows of the Australasian Institute of Mining and Metallurgy (AusIMM), the Australian Institute of Geoscientists (AIG) or a Recognised Professional Organisation (RPO) and have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as Competent Persons as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (2012 JORC Code). Each of the Competent Persons has given consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

[1] Participants in the MMG Long-Term Incentive Plans which may include Mineral Resources and Ore Reserves growth as a performance condition
[2] Member of the Association of Professional Engineers and Geoscientists of British Columbia

SUMMARY OF SIGNIFICANT CHANGES

MINERAL RESOURCES

Mineral Resources as at 30 June 2017 have changed since the 30 June 2016 estimate for a number of reasons with the most significant changes outlined in this section.

Mineral Resources (contained metal) have increased for lead (3%) while a decrease has occurred for zinc (10%), copper (10%), gold (14%), silver (10%) and molybdenum (3%).

Variations to Mineral Resources (contained metal) on an individual site basis are discussed below:

Increases:

Increases to the Mineral Resources (contained metal) for lead (19%) and zinc (2%) at Dugald River are due to a significant update of the Mineral Resource model from a major drilling campaign completed since the last model. No depletion has occurred at Dugald River during the reporting period.

Decreases:

The decreases in Mineral Resources (contained metal) are due to:

- divestment[1] of Golden Grove (copper, zinc, lead, silver, gold);
- divestment[2] of Avebury (nickel);
- depletion, drilling and modelling at Sepon (copper 18%);
- depletion, lower metal price and higher cost assumptions at Las Bambas (copper 9%); and
- three factors at Rosebery (copper 11%, zinc 14%, lead 12%, silver 11% and gold 13%) – 80% of tonnes as a result of a determination that the Inferred material around remnant stopes in the upper mine area has no foreseeable prospects for eventual economic extraction and the remaining 20% as a result of depletion and cut-off grade increases.

[1] Golden Grove divested Mineral Resources (metal) = 380kt copper, 1156 kt zinc, 89kt lead, 28Moz silver and 650koz gold.

[2] Avebury divested Mineral Resource (metal) = 260kt nickel

ORE RESERVES

Ore Reserves as at 30 June 2017 (contained metal) have increased for zinc (28%), lead (41%), silver (2%) and molybdenum (1%) and decreased for copper (6%) and gold (11%).

Variations to Ore Reserves (contained metal) on an individual site basis are discussed below:

Increases:

- Dugald River Ore Reserves have increased (10.3Mt) resulting from conversion of Mineral Resources by infill drilling, modelling, and increased planned mill throughput. These changes have resulted in an increase in zinc (42%), lead (62%) and silver (30%) metal in Ore Reserves. No depletion has occurred at Dugald River in 2017.

- Rosebery Ore Reserves have increased (0.2Mt), more than replacing depletion due to drilling and Mineral Resources conversion. There is an increase of copper (5%), zinc (8%), lead (11%), silver (13%) and gold (14%) metal compared to 2016 Ore Reserves.

Decreases:

A net reduction in Ore Reserves (metal) for copper and gold due to:

- depletion at all producing operation;

- a further reduction at Sepon due to a decrease in Indicated Mineral Resources available for conversion, combined with depletion has resulted in a reduction of copper metal of 34%;

- a further reduction at Kinsevere due to removal of uneconomic stockpiles. This change combined with depletion has resulted in a 15% reduction of copper metal for the site;

- a further reduction at Las Bambas due to a small reduction (0.02% Cu) in copper grade; and

- divestment[1] of Golden Grove accounts for almost all the reduction in gold metal (300koz).

[1] Golden Grove divested Ore Reserves (metal) = 82kt copper, 247 kt zinc, 32kt lead, 7.7Moz silver and 300koz gold.

KEY ASSUMPTIONS

PRICES AND EXCHANGE RATES

The following price and foreign exchange assumptions, set according to the relevant MMG Standard as at January 2017, have been applied to all Mineral Resources and Ore Reserves estimates. Price assumptions for all metals have changed from the 2016 Mineral Resources and Ore Reserves statement.

Table 1 : Price (real) and foreign exchange assumptions

<table>
<tr><th></th><th>Ore Reserves</th><th>Mineral Resources</th></tr>
<tr><td>Cu (US$/lb)</td><td>2.96</td><td>3.40</td></tr>
<tr><td>Cu (US$/lb) (Sepon only)</td><td>2.73</td><td>3.28</td></tr>
<tr><td>Zn (US$/lb)</td><td>1.19</td><td>1.43</td></tr>
<tr><td>Pb (US$/lb)</td><td>0.95</td><td>1.14</td></tr>
<tr><td>Au US$/oz</td><td>1200</td><td>1400</td></tr>
<tr><td>Ag US$/oz</td><td>17.5</td><td>20.4</td></tr>
<tr><td>Mo (US$/lb)</td><td>8.3</td><td>9.5</td></tr>
<tr><td>USD:CAD</td><td>1.18</td><td rowspan="3">As per Ore Reserves</td></tr>
<tr><td>AUD:USD</td><td>0.80</td></tr>
<tr><td>USD:PEN</td><td>3.10</td></tr>
</table>

CUT-OFF GRADES

Mineral Resources and Ore Reserves cut-off values are shown in Table 2 and Table 3 respectively.

Table 2 : Mineral Resources cut-off grades

Site	Mineralisation	Likely Mining Method[1]	Cut-Off Value	Comments
Las Bambas	Oxide Copper	OP	1% Cu	Cut-off is applied as a range that varies for each deposit and mineralised rock type at Las Bambas. *In-situ* copper Mineral Resources constrained within US$3.40/lb Cu pit shell.
	Primary Copper	OP	0.16 – 0.5% Cu	
Sepon	Oxide Gold	OP	0.9 – 1.7 g/t Au	Approximate cut-off grades shown in this table. Variable cut-off grade based on a net value calculation which accounts for costs, recoveries and metal prices within US$1,400/oz pit shells.
	Partial Oxide	OP	1.7 – 4.2 g/t Au	
	Primary Gold	OP	1.3 – 2.6 g/t Au	
	Supergene Copper – Carbonate	OP	1.4 – 1.6% Cu	Approximate cut-off grades shown in this table. Variable cut-off grade based on a net value calculation which accounts for costs, recoveries and metal prices within US$3.28/lb pit shells.
	Supergene Copper - Chalcocite	OP	1.5 – 1.6 % Cu	
	Primary Copper	OP	0.5 – 0.6% Cu	
Kinsevere	Oxide Copper & Stockpiles	OP	0.6% ASCu[2]	*In-situ* copper Mineral Resources constrained within a US$3.40/lb Cu pit shell.
	Transition Mixed Copper	OP	1.1% TCu[3]	
	Primary Copper	OP	0.8% TCu[3]	
Rosebery	Rosebery (Zn, Cu, Pb, Au, Ag)	UG	A$166/t NSR[4]	Remnant upper mine areas A$179/t NSR[4]
Dugald River	Primary Zinc (Zn, Pb, Ag)	UG	A$134/t NSR[4]	
	Primary Copper	UG	1%Cu	
High Lake	Cu, Zn, Pb, Ag, Au	OP	2.0% CuEq[5]	CuEq[5] = Cu + (Zn×0.30) + (Pb×0.33) + (Au×0.56) + (Ag×0.01): based on Long-Term prices and metal recoveries at Au:75%, Ag:83%, Cu:89%, Pb:81% and Zn:93%.
High Lake	Cu, Zn, Pb, Ag, Au	UG	4.0% CuEq[5]	CuEq[5] = Cu + (Zn×0.30) + (Pb×0.33) + (Au×0.56) + (Ag×0.01): based on Long-Term prices and metal recoveries at Au:75%, Ag:83%, Cu:89%, Pb:81% and Zn:93%.
Izok Lake	Cu, Zn, Pb, Ag, Au	OP	4.0% ZnEq[6]	ZnEq[6] = Zn + (Cu×3.31) + (Pb×1.09) + (Au×1.87) + (Ag×0.033); prices and metal recoveries as per High Lake.

[1] OP = Open Pit, UG = Underground
[2] ASCu = Acid Soluble Copper
[3] TCu = Total Copper
[4] NSR = Net Smelter Return
[5] CuEq = Copper Equivalent
[6] ZnEq = Zinc Equivalent

Table 3 : Ore Reserves cut-off grades

Site	Mineralisation	Mining Method	Cut-Off Value	Comments
Las Bambas	Primary Copper Ferrobamba	OP	0.19 – 0.27%Cu	Range based on rock type recovery.
	Primary Copper Chalcobamba		0.21 – 0.27%Cu	
	Primary Copper Sulfobamba		0.24 – 0.26% Cu	
Sepon	Supergene Copper [1]	OP	1.1% Cu	Approximate cut-off grades shown in this table. Variable cut-off grade based on net value script. Low grade float refers to stockpile reclaim.
	Supergene Copper [1]- low grade float[2]		0.9% Cu	
	Primary Copper		0.5% Cu	
Kinsevere	Copper Oxide	OP	0.9% ASCu[3]	Approximate cut-off grades shown in this table. Variable cut-off grade based on net value script.
		OP	0.9% ASCu[3]	Stockpile reclaim.
Rosebery	(Zn, Cu, Pb, Au, Ag)	UG	A$166 NSR[4] /t	
Dugald River	Primary Zinc	UG	A$134 NSR[4] /t	

[1] Supergene copper refers to carbonate and chalcocite ore types.
[2] Low grade float refers to stockpile reclaim
[3] ASCu = Acid Soluble Copper
[4] NSR = Net Smelter Return

PROCESSING RECOVERIES

Average processing recoveries are shown in Table 4. More detailed processing recovery relationships are provided in the Technical Appendix.

Table 4: Processing Recoveries

Site	Product	Recovery						Concentrate Moisture Assumptions
		Copper	Zinc	Lead	Silver	Gold	Mo	
Las Bambas	Copper Concentrate	86%	-	-	69%	64%		10%
	Molybdenum Concentrate						55%	5%
Rosebery	Zinc Concentrate		87%		9%	6%		8%
	Lead Concentrate		7%	80%	39%	13%		7%
	Copper Concentrate	67%			43%	36%		8%
	Doré[1] (gold and silver)				0.2%	28%		
Dugald River	Zinc Concentrate	–	86%		30%	–		10%
	Lead Concentrate	–		75%	27%	–		12%
Sepon	Copper Cathode	83%	–	–	–	–		–
Kinsevere	Copper Cathode	85% (95% ASCu[2])	–	–	–	–		–

The Technical Appendix published on the MMG website contains additional Mineral Resources and Ore Reserves information (including the Table 1 disclosure).

[1] Silver in Rosebery doré is calculated as a constant ratio to gold in the doré. Silver is set to 0.17 against gold being 20.7

[2] ASCu = Acid Soluble Copper

MMG Mineral Resources and Ore Reserves Statement

as at 30 June 2017

Technical Appendix

18 October 2017

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APPROVALS PAGE

Signature	Name	Position	Date
	Rex Berthelsen	Group Manager Technical Governance	18/10/17
Signature	**Name**	**Position**	**Date**
	Nan Wang	Group Manager Technical Services – ASEA / Africa	18/10/17
Signature	**Name**	**Position**	**Date**
	Reinhardt Viljoen	Group Manager Technical Services – Sth America	18/10/17
Signature	**Name**	**Position**	**Date**

The above signed endorse and approve this Mineral Resources and Ore Reserves Statement Technical Appendix.

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 change 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 Resources and Ore Reserves release document, which outlines the supporting details to the Mineral Resources 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 became 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 judgment 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

The economic analysis undertaken for each Ore Reserves described in this document and for the whole Company has resulted in positive net present values (NPVs). MMG uses a discount rate appropriate to the size and nature of the organisation and individual deposits.

2.1 Commodity Price Assumptions

The price and foreign exchange assumptions used for the 2017 Mineral Resources 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

	Ore Reserves	Mineral Resources
Cu (US$/lb)	2.96[1]	3.40[23]
Zn (US$/lb)	1.19	1.43
Pb (US$/lb)	0.95	1.14
Au US$/oz	1200	1400
Ag US$/oz	17.5	20.4
Mo (US$/lb)	8.3	9.5
AUD:USD	0.80	As per Ore Reserves
USD:PEN	3.10	

[1] Short run pricing of $2.73/lb Cu was used for Sepon Ore Reserves
[2] Short run pricing of $3.28/lb Cu was used for Sepon Mineral Resources
[3] 2017 Resource Copper price compared with $3.50/lb in 2016

2.2 Competent Persons

Deposit	Accountability	Competent Person	Professional Membership	Employer
MMG Mineral Resources and Ore Reserves Committee	Mineral Resources	Rex Berthelsen[1]	FAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Ore Reserves	Nan Wang[1]	MAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Metallurgy: Mineral Resources / Ore Reserves	Reinhardt Viljoen[1]	MAusIMM	MMG
Las Bambas	Mineral Resources	Rex Berthelsen[1]	FAusIMM(CP)	MMG
Las Bambas	Ore Reserves	Yao Wu[1]	MAusIMM	MMG
Las Bambas	Metallurgy: Mineral Resources / Ore Reserves	Amy Lamb[1]	MAusIMM	MMG
Sepon	Mineral Resources	Chevaun Gellie	MAusIMM	MMG
Sepon	Ore Reserves	Jodi Wright[1]	MAusIMM(CP)	MMG
Sepon	Metallurgy: Mineral Resources / Ore Reserves	Kevin Rees	MAusIMM	MMG
Kinsevere	Mineral Resources	Douglas Corley[1]	MAIG R.P.Geo.	MMG
Kinsevere	Ore Reserves	Jodi Wright[1]	MAusIMM(CP)	MMG
Kinsevere	Metallurgy: Mineral Resources / Ore Reserves	Nigel Thiel[1]	MAusIMM(CP)	MMG
Rosebery	Mineral Resources	Anna Lewin	MAusIMM(CP)	MMG
Rosebery	Ore Reserves	Karel Steyn[1]	MAusIMM	MMG
Rosebery	Metallurgy: Mineral Resources / Ore Reserves	Kevin Rees	MAusIMM(CP)	MMG
Dugald River	Mineral Resources	Douglas Corley[1]	MAIG R.P.Geo.	MMG
Dugald River	Ore Reserves	Karel Steyn[1]	MAusIMM	MMG
Dugald River	Metallurgy: Mineral Resources / Ore Reserves	Nigel Thiel[1]	MAusIMM(CP)	MMG
High Lake, Izok Lake	Mineral Resources	Allan Armitage	MAPEG[2] (P.Geo)	Formerly MMG

The information in this report that relates to Mineral Resources and Ore Reserves is based on information compiled by the listed Competent Persons, who are Members or Fellows of the Australasian Institute of Mining and Metallurgy (AusIMM), the Australian Institute of Geoscientists (AIG) or a Recognised Professional Organisation (RPO) and have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as Competent Persons as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (2012 JORC Code). Each of the Competent Persons has given consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

[1] Participants in the MMG Long-Term Incentive Plans which may include Mineral Resources and Ore Reserves growth as a performance condition.
[2] Member of the Association of Professional Engineers and Geoscientists of British Columbia.

3 LAS BAMBAS OPERATION

3.1 Introduction and Setting

Las Bambas is a world class copper gold (Cu-Au) mine located in the Andes of southern Peru approximately 75km south-southwest of Cusco, approximately 300km north-northwest of Arequipa, and approximately 150km northeast of Espinar (also named Yauri). Las Bambas is readily accessible from either Cusco or Arequipa over a combination of paved and good quality gravel roads. Road travel from Cusco takes approximately 6 hours, while road travel from Arequipa takes approximately 9 hours.

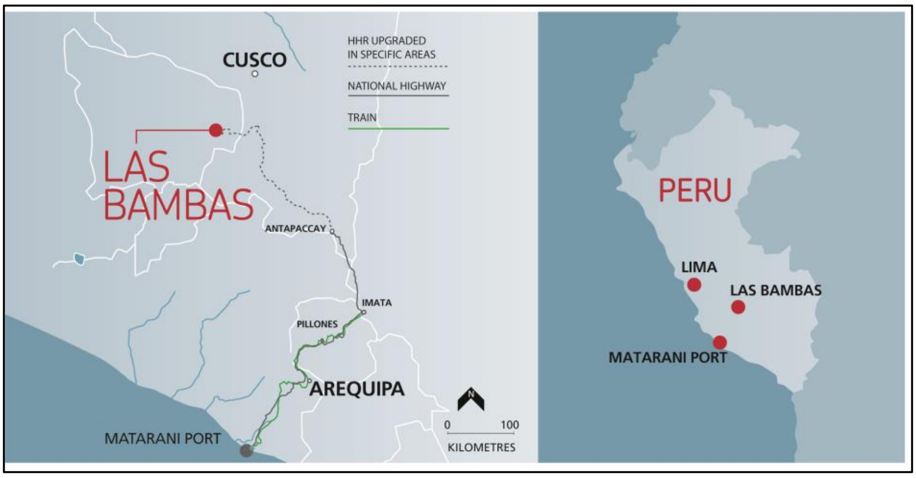


Figure 3-1 Las Bambas Mine location

Las Bambas is a truck and excavator mining operation with a conventional copper concentrator. Copper production commenced in the fourth quarter 2015 with the first concentrate achieved on 26 November. The first shipment of product departed the Port of Matarani for China on 15 January 2016. The mine has achieved full capacity, running at over 400,000 tonnes per day. Both lines of the copper concentrate plant have been commissioned. The primary crusher was commissioned during the fourth quarter of 2015 and the overland conveyor has now reached 100% capacity of 8,000 tonnes per hour. The tailings dam reached planned levels with discharge and water recirculation back to plant fully operational. Commercial production was declared on July 1, 2016.

Las Bambas is a joint venture project between the operator MMG (62.5%), a wholly owned subsidiary of Guoxin International Investment Co. Ltd (22.5%) and CITIC Metal Co. Ltd (15.0%).

The Mineral Resources and Ore Reserves have been updated with additional drilling completed in 2016 for the June 2017 release. The 2017 Mineral Resources estimation includes updated geological interpretation and estimation parameters.

3.2 Mineral Resources – Las Bambas

3.2.1 Results

The 2017 Las Bambas Mineral Resources is summarised in Table 2. The Las Bambas Mineral Resources is inclusive of the Ore Reserves. All data reported here is on a 100% asset basis. MMG's attributable interest in Las Bambas is 62.5%.

Table 2 2017 Las Bambas Mineral Resources tonnage and grade (as at 30 June 2017)

Las Bambas Mineral Resource (62.5%)									
						Contained Metal			
Ferrobamba Oxide Copper[1]	**Tonnes (Mt)**	**Copper (% Cu)**	**Silver (g/t Ag)**	**Gold (g/t Au)**	**Mo (ppm)**	**Copper (kt)**	**Silver (Moz)**	**Gold (Moz)**	**Mo (kt)**
Indicated	9.3	2.0				188			
Inferred	0.6	2.5				14			
Total	**9.9**	**2.0**				**202**			
Ferrobamba Primary Copper[2]									
Measured	542	0.6	3.0	0.06	204	3,485	53	1.0	110
Indicated	546	0.6	2.8	0.05	211	3,258	49	0.9	115
Inferred	263	0.6	2.4	0.04	158	1,572	20	0.3	42
Total	**1,351**	**0.6**	**2.8**	**0.05**	**198**	**8,315**	**122**	**2.2**	**267**
Ferrobamba Total	**1,361**	0.6				**8,517**	**126**	**2.3**	**269**
Chalcobamba Oxide Copper[1]									
Indicated	6.1	1.5				89			
Inferred	0.7	1.5				10			
Total	**6.8**	**1.5**				**99**			
Chalcobamba Primary Copper[3]									
Measured	85	0.4	1.1	0.01	148	312	3	0.04	13
Indicated	195	0.7	2.5	0.03	141	1,301	16	0.19	28
Inferred	36	0.5	1.8	0.02	141	187	2	0.03	5
Total	**315**	**0.6**	**2.0**	**0.03**	**143**	**1,800**	**21**	**0.26**	**45**
Chalcobamba Total	**322**					**1,899**	**22**	**0.27**	**46**
Sulfobamba Oxide Copper[1]									
Inferred									
Total									
Sulfobamba Primary Copper[4]									
Indicated	85	0.67	4.7	0.02	170	571	13	0.1	14
Inferred	100	0.58	6.5	0.02	119	582	21	0.1	12
Total	**184**	**0.62**	**5.7**	**0.02**	**142**	**1153**	**34**	**0.1**	**26**
Sulfobamba Total	**184**					**1,153**	**34**	**0.1**	**26**
Oxide Stockpiles									
Indicated	5.5	1.0				56			
Sulphide Stockpiles									
Measured	0.2	0.85	4.5		148	1.5	0.03		0.03
Total Contained	**1,873**					**11,625**	**182**	**2.7**	**341**

Notes:

1. 1% Cu cut-off grade contained within a US$3.4/lb pit shell for oxide material.
2. Average (0.17% Cu) cut-off grade contained within a US$3.4/lb pit shell for primary material.
3. Average (0.18% Cu) cut-off grade contained within a US$3.4/lb pit shell for primary material.
4. Average (0.20% Cu) cut-off grade contained within a US$3.4/lb pit shell for primary material.

Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Contained metal does not imply recoverable metal.

3.2.2 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 3 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 3 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Las Bambas Mineral Resources 2017

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
Sampling techniques	• Diamond drilling (DD) was used to obtain an average 2m sample that is half core split, crushed and pulverised to produce a pulp (95% passing 105µm). Diamond core is selected, marked and numbered for sampling by the logging geologist. Sample details are stored in an acQuire database for correlation with returned geochemical assay results. • Samples for analysis are bagged, shuffled, re-numbered and de-identified prior to dispatch. • Whole core was delivered to the Inspectorate Laboratory in Lima (2005-2010) and Certimin Laboratory in Lima (2014 to 2015) for half core splitting and sample preparation. From mid-2015 core samples were cut and sampled at an ALS sample preparation laboratory on-site. Samples are then sent to ALS Lima for preparation and analysis. • There are no inherent sampling problems recognised. • Measures taken to ensure sample representivity include the collection, and analysis of coarse crush duplicates.
Drilling techniques	• The drilling type is wireline diamond core drilling from surface. Drill core is not oriented. All drillholes used in the Mineral Resource estimates have been drilled using HQ size.
Drill sample recovery	• Recovery is estimated by measuring the recovered core within a drill run length and recorded in the acQuire database. Run by run recovery has been recorded for 385,133m of the total 395,140m of diamond drilling in the data used for Mineral Resources estimation for the Sulfobamba, Chalcobamba and Ferrobamba deposits. Diamond drill recovery average is about 97% for all deposits (98% for Sulfobamba, 97% for Chalcobamba and Ferrobamba deposits). • The drilling process is controlled by the drill crew and geological supervision provides a means for maximising sample recovery and ensures suitable core presentation. No other measures are taken to maximise core recovery. • There is no detectable correlation between recovery and grade which can be determined from graphical and statistical analysis. Preferential loss/gains of fine or coarse materials are not significant and do not result in sample bias as the nature of mineralisation is stockwork and disseminated sulphides. Diamond core sampling is applied and recovery is considered high.
Logging	• 100% of diamond drill core used in the Mineral Resource estimates has been geologically and geotechnically logged to support Mineral Resources estimation, mining and metallurgy studies. • Geological logging is qualitative and geotechnical logging is quantitative. All drill core is photographed.

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
Sub-sampling techniques and sample preparation	• All samples included in the Mineral Resource estimates are from diamond drill core. Drill core is longitudinally sawn to provide half-core samples within intervals directed by the logging geologist. The remaining half-core is kept and stored in the original sample tray. The standard sampling length is 2m for PQ core (minimum 1.2m) and HQ core (minimum 1.2m, maximum 2.2m) while NQ core is sampled at 2.5m (minimum 1.5m). Sample intervals do not cross geological boundaries. • From 2005 until 2010 geological samples have been processed in the following manner: Dried, crushed, pulverised to 95% passing 105µm. Sizing analysis is carried out on 1 in 30 samples. • From 2010 geological samples have been processed in the following manner: Dried, crushed, pulverised to 95% passing 105µm. Sizing analyses are carried out on one in 10-15 samples. • Representivity of samples is checked by duplication at the crush stage one in every 40 samples. No field duplicates are taken. • Twelve month rolling Quality Assurance / Quality Control (QAQC) analysis of sample preparation techniques indicate the process is appropriate for Las Bambas samples. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Las Bambas mineralisation (porphyry and skarn Cu-Mo mineralisation) by the Competent Person.
Quality of assay data and laboratory tests	• From 2005 until 2010 the assay methods undertaken by Inspectorate (Lima) for Las Bambas were as follows: o Cu, Ag, Pb, Zn, Mo - 0.5g of sample. Digestion by 4-Acids. Reading by Atomic Absorption Spectrometry (AAS). o Acid soluble - 0.2g sample. Leaching by a 15% solution of H_2SO_4 at 73°C for 5 minutes. Reading by AAS. o Acid soluble - 0.2g of sample. Digestion by a citric acid solution at 65°C for 15 minutes. Reading by AAS. o Au - Cupellation at 950°C. Reading by AAS. Above detection limit analysis by gravimetry. o 35 elements - Digestion by aqua regia and reading by ICP. • From 2010 to 2015 routine assay methods undertaken by Certimin (Lima) for Las Bambas are as follows: o Cu, Ag, Pb, Zn, Mo - 0.5g of sample. Digestion by 4-Acids. Reading by Atomic Absorption Spectrometry (AAS). o Acid soluble copper – 0.2g sample. Leaching by a 15% solution of H_2SO_4 at 73°C for 5 minutes. Reading by AAS. o Acid Soluble copper - 0.2g of sample. Digestion by a citric acid solution at 65°C for 15 minutes. Reading by AAS. o Au - Fire assay with AAS Finish. Over-range results are re-assayed by Gravimetric Finish. o 35 elements - Digestion by aqua regia and reading by ICP. • From 2015 to present routine assay methods undertaken by ALS (Lima) for Las Bambas are as follows:

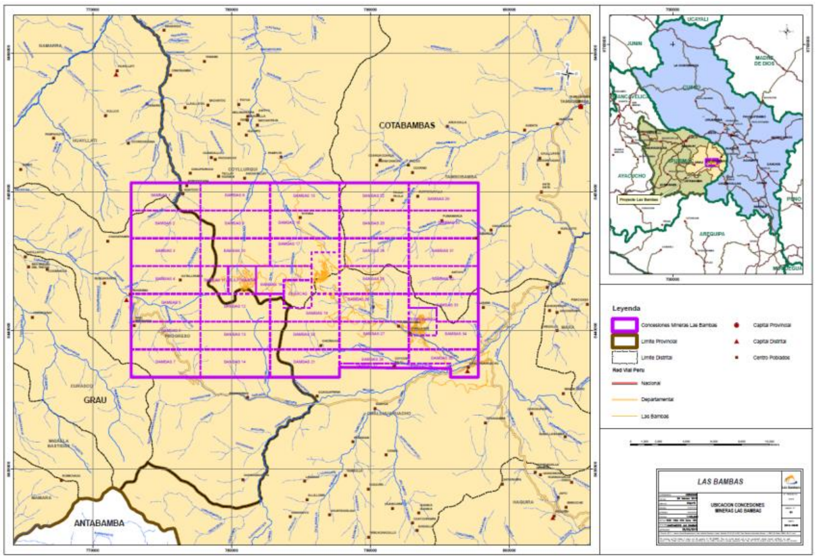
Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
	○ Cu, Ag, Pb, Zn, Mo - 0.5g of sample. Digestion by 4-Acids. Reading by Atomic Absorption Spectrometry (AAS). ○ Acid soluble copper – 0.5g sample. Leaching by a 5% solution of H_2SO_4 at ambient temperature for 1 hour. Reading by AAS. ○ Au - Fire assay with AAS Finish. Over-range results are re-assayed by Gravimetric Finish. ○ 35 elements - Digestion by aqua regia and reading by ICP. • All of the above methods with the exception of the acid soluble copper are considered total digest. • Since 2013, composited pulps have been submitted to Certimin Laboratory for sequential copper analysis. This method produces results of acid soluble (H_2SO_4), then cyanide soluble followed by residual copper in sequence. This analysis is used for geometallurgical modelling. • No geophysical tools, spectrometers or handheld XRF instruments have been used in the analysis of samples external to the ALS laboratory for the estimation of Mineral Resources. • Assay techniques are considered suitable and representative; independent umpire laboratory checks occurred routinely between 2005-2010 using the ALS Chemex laboratory in Lima. Check samples were inserted 1 in every 25 samples (2005-2007), every 50 samples (2008) and every 40 samples (2010). For the 2014, 2015 and 2016 programs duplicated samples were collected at the time of sampling and stored. Samples are then sent to the Inspectorate Laboratory, Lima, for third party (umpire) analysis. The samples were selected at a rate of 1:40. Results received indicate a good correlation between datasets and show no bias for copper, however the check assays for molybdenum were considered biased low after reviewing of the standards inserted with the batches. • Inspectorate, Certimin and ALS release quarterly QAQC data to Las Bambas for analysis of internal laboratory standard performance. The performance of the laboratory internal standards is within acceptable limits. • Las Bambas routinely insert: ○ Primary coarse duplicates: Inserted at a rate of 1:25 samples (2005-2007), 1:50 samples (2008), and 1:40 samples (2010- present). ○ Coarse blank samples: Inserted after a high-grade sample (coarse blank samples currently make up about 4.2% of all samples analysed). ○ Pulp duplicates samples: Inserted 1:25 samples (2005-2007), 1:50 samples (2008), and 1:40 samples (2010-present). ○ Pulp blank samples: Inserted before the coarse blank sample and always after a high grade sample (pulp blank samples currently make up about 4.2% of all samples analysed). ○ Certified Reference Material (CRM) samples: Inserted at a rate of 1:50 samples (2005-2006), 1:40 samples (2007) and 1:20 samples (2008 to present).

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
	• QAQC analysis has shown that: o Blanks: a minimum level of sample contamination by copper was detected during the sample preparation and assay. o Duplicates: the analytical precision is within acceptable ranges when compared to the original sample, i.e., more than 90% of the pairs of samples are within the error limits evaluated for a maximum relative error of 10% (R^2>0.90). These results were also repeated in the external ALS check samples. o Certified Reference Material: acceptable levels of accuracy and precision have been established. • Sizing test results are not routinely analysed.
Verification of sampling and assaying	• Verification by independent personnel was not undertaken at the time of drilling. However, drilling, core logging and sampling data is entered by geologists; assay results are entered by the resource geologist after data is checked for outliers, sample swaps, performance of duplicates, blanks and standards, and significant intersections are checked against core log entries and core photos. Errors are rectified before data is entered into the database. • Apart from 20 metallurgical drillholes drilled in 2007 which twinned Mineral Resource Ferrobamba drillholes, no twinned drillholes have been completed. • All drillholes are logged using laptop computers directly into the drillhole database (acQuire). Prior to November, 2014 diamond drillholes were logged on paper and transcribed into the database. Assay results are provided in digital format (both spreadsheet and PDF) by the laboratories and are automatically loaded into the database after validation. All laboratory primary data and certificates are stored on the Las Bambas server. • The database has internal validation processes which prevent invalid or unapproved records from being stored. Additional manual data validation occurs in Vulcan software before data is used for interpretation and Mineral Resources modelling. Unreliable data is flagged and excluded from Mineral Resources estimation work. • No adjustments have been made to assay data – if there is any doubt about the data quality or location, the drillhole is excluded from the estimation process.
Location of data points	• In 2005 collar positions of surface drillholes were picked up by Horizons South America using Trimble 5700 differential GPS equipment. From 2006, the Las Bambas engineering personnel have performed all subsequent surveys using the same equipment. Since 2014, drillholes are set out using UTM co-ordinates with a hand held Differential Global Positioning System (DGPS) and are accurate to within 1m. On completion of drilling, collar locations are picked up by the onsite surveyors using DGPS (Trimble or Topcon). These collar locations are accurate to within 0.5m. • During the 2014 due diligence process (2014) RPM independently checked five collar locations at Ferrobamba and Chalcobamba with a handheld GPS and noted only small differences well within the error limit of the GPS used. RPM did not

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
	undertake independent checking of any Sulfobamba drillholes. The collar locations are considered accurate for Mineral Resources estimation work. • In 2005 the drilling contractor conducted downhole survey's using the AccuShot method for non-vertical drillholes. Vertical holes were not surveyed. If the AccuShot arrangement was not working, the acid test (inclination only) was used. Since 2006, all drillholes are surveyed using Reflex Maxibor II equipment units which take measurements every 3m. The downhole surveys are considered accurate for Mineral Resources estimation work. • The datum used is WGS 84 with a UTM coordinate system zone 19 South. • In 2006 Horizons South America surveyed the topography at a scale of 1:1000 based on aerophotogrammetric restitution of orthophotos. A digital model of the land was generated every 10m and, using interpolation, contour lines were obtained every metre. The maps delivered were drafted in UTM coordinates and the projections used were WGS 84 and PSAD 56. A triangulated surface model presumably derived from this survey is in current use at site and is considered suitable for Mineral Resources and Ore Reserves modelling purposes.
Data spacing and distribution	• The Las Bambas mineral deposits are drilled on variable spacing dependent on rock type (porphyry vs. skarn). Drill spacing typically ranges from 100m x 100m to 25m x 25m and is considered sufficient to establish the degree of geological and grade continuity appropriate for Mineral Resources estimation processes and Mineral Resources classifications applied. • Drillhole spacing of approximately 25m x 25m within skarn hosted material and 50m x 50m within porphyry hosted is considered sufficient for long term Mineral Resources estimation purposes based on a drillhole spacing study undertaken in 2015. While the 25m spacing is suitable for Mineral Resource estimation, the Las Bambas deposits tend to have short scale 5m - 10m variations within the skarn that are not captured by the infill drilling at this spacing. This localised geological variability is captured by mapping and drillhole logging. • Diamond drillhole samples are not composited prior to routine chemical analysis; however the nominal sample length is generally 2m. All sequential copper analysis is undertaken on pulps that are composited most commonly to 8m but sample lengths as small as 1m are contained in the database.
Orientation of data in relation to geological structure	• Overall drillhole orientation is planned at 90 degrees to the strike of the mineralised zone for each deposit. Drillhole spacing and orientation is planned to provide evenly spaced, high angle intercepts of the mineralised zone where possible, thus minimising sampling bias related to orientation. However, in some locations namely the east and western areas of Ferrobamba containing skarn, due to the orientation of the drilling grid some drillholes are orientated along strike, yet still manage to intersect the ore zones at moderate angles. • Drilling orientation is not considered to have introduced sampling bias.

Sample security	• Measures to provide sample security include:
	  o Adequately trained and supervised sampling personnel.
	  o Samples are stored in a locked compound with restricted access during

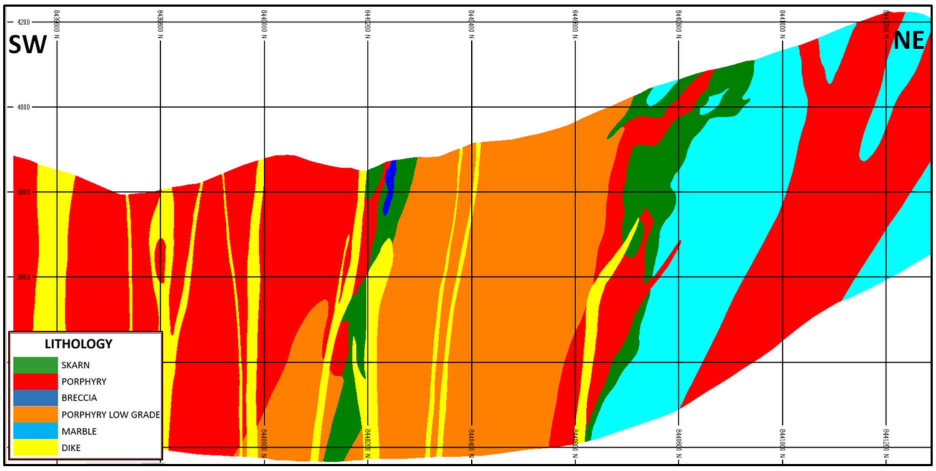
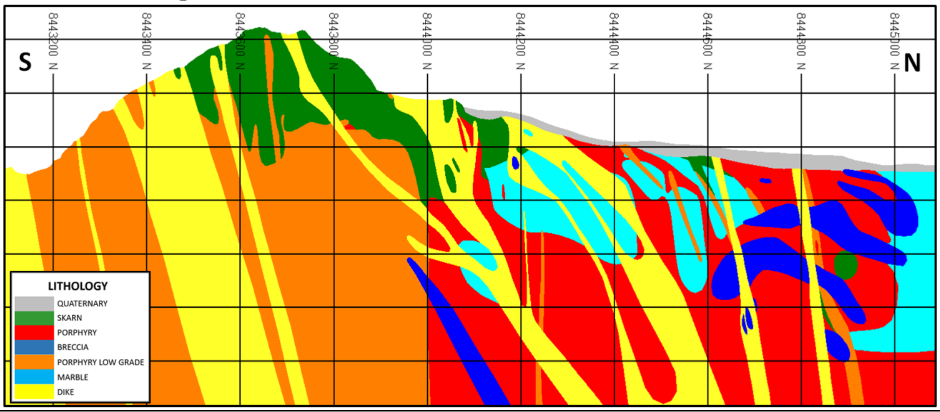
Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
	preparation. o Dispatch to various laboratories via contract transport provider in sealed containers. o Receipt of samples acknowledged by receiving analytical laboratory by email and checked against expected submission list. o Assay data returned separately in both spreadsheet and PDF formats.
Audit and reviews	• In 2015, an internal audit, checking 5% of the total samples contained in the acQuire database was undertaken comparing database entry values to the original laboratory certificates for Cu, Ag, Mo, As and S. No material issues were identified. • Internal audits of the Inspectorate and Certimin laboratories have occurred twice a month by Las Bambas personnel. Historically, any issues identified have been rectified. Currently, there are no outstanding material issues. • The Competent Person has visited the both the Certimin and ALS laboratories in Lima.

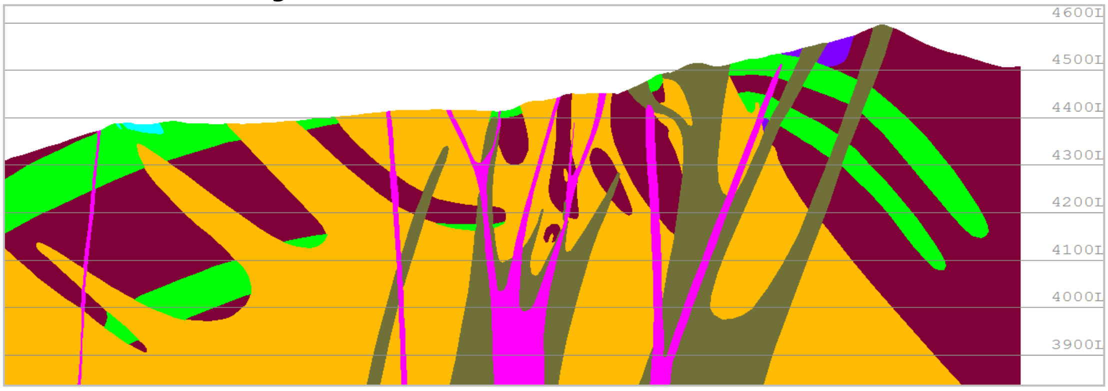
Assessment Criteria	Commentary
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	• The Las Bambas project has tenure over 41 Mineral Concessions. These Mineral Concessions secure the right to the minerals in the area, but do not provide rights to the surface land. • Property of surface land is acquired through a separate process. The below map outlines the 41 Mineral Concessions and the mine property owned by MMG.  • Tenure over the 41 Concessions is in good standing. There are no known

Assessment Criteria	Commentary
Section 2 Reporting of Exploration Results	
	impediments to operating in the area.
Exploration done by other parties	• The Las Bambas project has a long history of exploration by the current and previous owners. • Exploration commenced in 1966 with around 450km of surface diamond drilling drilled to date. • Initial exploration was completed by Cerro de Pasco followed by Cyprus, Phelps Dodge, BHP, Tech, and Pro Invest prior to Xstrata Resources definition drilling which commenced in 2005. All historical drilling is outlined in the table below. • Glencore and Xstrata merged to form Glencore plc. • In 2013, MMG Ltd, Guoxin International Investment Corporation. Limited and CITIC Metal Co., Ltd enter into an agreement to purchase the Las Bambas project from Glencore plc.

Company	Year	Deposit	Purpose	Type	# of DDH	Drill size	Metres Drilled
Cerro de Pasco	1996	Chalcobamba	Exploration	DDH	6	Un-known	906.4
Cyprus	1996	Chalcobamba	Exploration		9		1,367.3
Phelps Dodge	1997	Ferrobamba	Exploration	DDH	4	Un-known	737.8
		Chalcobamba			4		653.4
BHP	1997	Ferrobamba	Exploration	DDH	3	Un-known	365.8
		Chalcobamba			4		658.6
Pro Invest	2003	Ferrobamba	Exploration	DDH	4	HQ	738.0
		Chalcobamba			7		1,590.0
Xstrata	2005	Ferrobamba	Resource Evaluation	DDH	109	HQ	26,839.9
		Chalcobamba			66		14,754.1
		Sulfobamba			60		13,943.0
	2006	Ferrobamba	Resource Evaluation	DDH	125	HQ	51,004.2
		Chalcobamba			95		27,982.9
		Sulfobamba			60		16,971.5
		Charcas			8		2,614.1
		Azuljaja			4		1,968.9
	2007	Ferrobamba	Resource Evaluation	DDH	131	HQ	46,710.4
		Chalcobamba			134		36,617.6
		Sulfobamba			22		4,996.6
	2008	Ferrobamba	Resource Evaluation	DDH	118	HQ	46,773.8
		Chalcobamba			90		22,096.6
	2010	Ferrobamba	Resource Evaluation	DDH	91	HQ	28,399.9
MMG	2014	Ferrobamba	Resource Evaluation	DDH	23	HQ	12,609.7
	2015	Huancarane	Sterilisation	DDH	5	HQ	772.6
	2015	Ferrobamba	Resource Evaluation	DDH	154	HQ	53,771.7
	2016	Ferrobamba	Resource Evaluation	DDH	114	HQ	31,206.2
		Chalcobamba	Resource Evaluation	DDH	13		1,880.3
			Total				**448,931.3**

Assessment Criteria	Commentary
Geology	• Las Bambas is located in a belt of Cu (Mo-Au) skarn deposits associated with porphyry type systems situated in south-eastern Peru. This metallogenic belt is controlled by the Andahuaylas-Yauri Batholith of Eocene- Oligocene age, which is emplaced in Mesozoic sedimentary units, with the Ferrobamba Formation (Lower-to-Upper Cretaceous) being of greatest mineralising importance. • The porphyry style mineralisation occurs in quartz-monzonite to granodiorite rocks.

Assessment Criteria	Commentary
Section 2 Reporting of Exploration Results	
	Hypogene copper sulphides are the main copper bearing minerals with minor occurrence of supergene copper oxides and carbonates near surface. The intrusive rocks of the batholith in contact with the Ferrobamba limestones gave rise to contact metamorphism and, in certain locations, skarn bodies with Cu (Mo-Au) mineralisation.
Drillhole information	• Historical exploration drillholes (prior to 2005) have been excluded from the Mineral Resource estimates. • Drillhole data is not provided in this report as this report is for the Las Bambas Mineral Resources which use all available data and no single hole is material for the Mineral Resource estimates.
Data aggregation methods	• Historical exploration drillholes (prior to 2005) have been excluded from the Mineral Resource estimates. • No metal equivalents were used in the Mineral Resources estimation.
Relationship between mineralisation width and intercepts lengths	• No exploration diamond drillholes have been completed in the 2017 reporting period. Historical exploration drillholes (prior to 2005) have been excluded from the Mineral Resource estimates. • Mineralisation true widths are captured by interpreted mineralisation 3D wireframes. Drillholes are drilled to achieve intersections as close to orthogonal as possible.
Diagrams	 **Section Through Ferrobamba** 

Assessment Criteria	Commentary
Section 2 Reporting of Exploration Results	
	Section Through Chalcobamba  **Section Through Sulfobamba**
Balanced reporting	• All drilling completed during the 2017 reporting period completed at Ferrobamba is infill in nature. Historical exploration drillholes (prior to 2005) have been excluded from the Mineral Resource estimates. A small number of holes were drilled at Chalcobamba for the purpose of hydrogeology, geotechnical and infill.
Other substantive exploration data	• No substantive exploration diamond drillholes have been completed in the 2017 reporting period. Historical exploration drillholes (prior to 2005) have been excluded from the Mineral Resource estimates. • In the past year several orebody knowledge studies have been carried out including skarn zonation, vein densities and a large age dating program. Results from these studies are assisting with improving the understanding of the orebodies. This work will continue.
Further work	• An ongoing program of regional and deposit scale mapping, cross sectional studies, infrared spectral analyses, isotopic and petrographic studies and soil sampling. • A program of exploration assessment and targeting is currently underway to identify exploration options for the Las Bambas leases. • Permitting for regional exploration drilling is underway. • Ongoing infill programs are planned to increase deposit confidence.

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	• The following measures are in place to ensure database integrity: ○ All Las Bambas drillhole data is stored in an SQL database (acQuire) on the Las Bambas site server, which is backed up at regular intervals. ○ Geological logging is entered directly into laptop computers which are uploaded to the database. Prior to November 2014, diamond drillholes were logged on paper logging forms and transcribed into the database. From November 2015 logging was entered directly into a customised

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	interface using portable tablet computers. o Assays are loaded directly into the database from digital files provided from the assay laboratory. • The measures described above ensure that transcription or data entry errors are minimised. • Data validation procedures include: o A database validation project was undertaken in early 2015 checking 5% of the assayed samples in the database against original laboratory certificates. No material issues were identified. o The database has internal validation processes which prevent invalid or unapproved records to be stored.
Site visits	• The Competent Person has undertaken numerous site visits to Las Bambas since acquisition. In the view of the Competent Person there are no material risks to the Mineral Resources based on observations of site practices. • Several site visits to the Ferrobamba area and the Chalcobamba area have been conducted but due to local community restrictions, the Competent Person has been unable to visit Sulfobamba to date. • The site previously employed a practice of 'double blind' sample randomisation at the laboratory. It essentially guarantees the secrecy of the results from the operating laboratory. It does however pose a minor risk of compromising sample provenance, although the risk is probably low.
Geological interpretation	• There is good confidence on the geological continuity and interpretation of the Ferrobamba and Chalcobamba deposits. Confidence in the Sulfobamba deposit is considered moderate due to limited drilling. • The 2017 geological interpretation was undertaken on paper sections orientated perpendicular to the established structural trend of each deposit. Section spacing for the interpretations varied between deposits from 25m at Ferrobamba to 50m at Sulfobamba. The geological logging, assay data and surface mapping were used in the interpretation. The drilling and surface mapping were checked against each other to ensure they were concordant. • No alternative interpretations have been generated for the Las Bambas mineralisation and geology. • Exploratory data analysis (EDA) indicated that the lithological characterisation used for the 2010 geological interpretation were for the most part valid (with minor changes) and were applied for the 2017 modelling. Each lithological unit was modelled according to age, with the youngest modelled first. Structural considerations such as plunge of the units, folding and faults were taken into consideration (where information existed). Orthogonal sections were also interpreted to ensure lithological continuity. • Oxidation domains were produced for the Ferrobamba and Chalcobamba models. Oxidation domains were based on logged mineral species and acid soluble copper to total copper assay ratios.

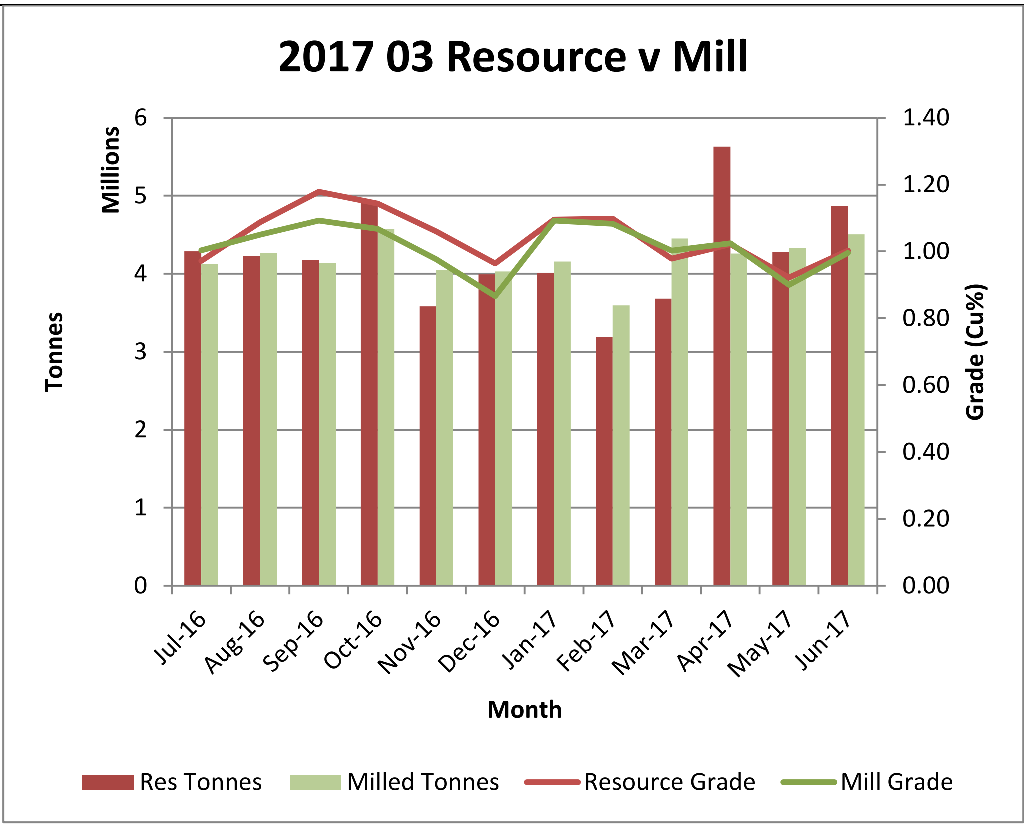
Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	• Geological interpretations were then modelled as wireframe solids (based on the paper sections) and were peer reviewed within the Las Bambas Geology department and by the Competent Person. • Specific grade domains (copper and molybdenum) were not created, with the exception of interpreted, spatially coherent high-grade shoots at Sulfobamba. A domain cut-off of 0.8% Cu was used for the high-grade domain. The introduction of the high-grade domain was supported by EDA, contact plots, and change of support analysis.
Dimensions	• The Las Bambas Mineral Resources refers to three distinct deposits; each have been defined by drilling and estimated: o Ferrobamba Mineral Resources occupies a footprint which is 2500m N-S and 1800m E-W and over 900m vertically. o Chalcobamba Mineral Resources occupies a footprint of 2300m N-S and 1300m E-W and 800m vertically o Sulfobamba Mineral Resources is 1800m along strike in a NE direction and 850m across strike in a NW direction and 450m deep.
Estimation and modelling techniques	• Mineral Resources estimation for the three deposits has been undertaken in Vulcan (Maptek) mining software with the following key assumptions and parameters: o Ordinary Kriging interpolation has been applied for the estimation of Cu, Mo, Ag, Au, As, Ca, Mg, S, CuAS (acid soluble copper), CuCN (cyanide soluble copper) and density. This is considered appropriate for the estimation of Mineral Resources at Las Bambas. o The Ferrobamba, Chalcobamba and Sulfobamba block models utilised sub-blocking. Models were then regularised for use in mine planning purposes. o Extreme grade values were managed by upper grade capping based on statistical assessment evaluated for all variables and domains. Consideration was also given to the metal content above the top cap value. o All elements were estimated into lithology domains. At Ferrobamba five different orientation domains were identified in the skarn and were used in the interpolation. The boundaries between each orientation domain were treated as semi-soft boundaries. At Sulfobamba high-grade skarn shoots were identified and were used in the interpolation of copper only. The boundaries between the low and the high-grade skarn were treated as hard boundaries. o Data compositing for estimation was set to 4m, which matches two times the majority of drillhole sample lengths (2m), provides good definition across interpreted domains, and adequately accounts for bench height. CuAS and CuCN was composited to 8m which matches the majority of composite sample lengths. o Variogram analysis was updated for all deposits. Variogram analysis was undertaken in Vulcan software (Ferrobamba, Sulfobamba) and Supervisor (Snowden) software (Chalcobamba). o No assumptions have been made about the correlation between variables.

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	All variables are comparably informed and independently estimated. ○ Interpolation was undertaken in three to four passes. • Check estimates using Discrete Gaussian change of support modelling have been performed on all models. Block model results are comparable with previous Mineral Resources estimations after changes due to drilling and re-modelling by the site. • Reconciliation of the last 12 months of production indicates that the Mineral Resources block model is over-calling the Mill claimed copper metal by 3.7%. This comprises a 3% overcall of grade and a 0.7% overcall of tonnage. These results demonstrate that the Mineral Resources block model is robust and fit for purpose for use in mine planning and Ore Reserves estimation. In the graph below. • Assumptions about the recovery of by-products is accounted in the net-smelter return (NSR) calculation which includes the recovery of Mo, Ag and Au along with the standard payable terms. • Arsenic is considered a deleterious element and has been estimated. Sulphur, calcium and magnesium are also estimated to assist in the determination of NAF (non-acid forming), PAF (potentially acid forming) and acid neutralising material. • Block sizes for all three deposits were selected based on Quantitative Kriging Neighbourhood Analysis (QKNA). The Ferrobamba block size was 20m x 20m x 15m with sub-blocks of 5m x 5m x 5m. Chalcobamba block size was set to 30m x 30m x 15m, with sub-blocks of 10m x 10m x 5m. The block size at Sulfobamba was set to 50m x 50m x 15m (sub-blocked to 25m x 25m x 7.5m) which roughly equates to the drill spacing. The search anisotropy employed was based on both the ranges of the variograms and the drill spacing. All models were regularised to 20m x 20m x 15m for use in Ore Reserve estimates. • The selective mining unit is assumed to be approximately 20m x 20m x 15m (x,y,z) which equates to the Ferrobamba block model block size. • Block model validation was conducted by the following processes – no material issues were identified: ○ Visual inspections for true fit for all wireframes (to check for correct placement of blocks and sub-blocks). ○ Visual comparison of block model grades against composite sample grades. ○ Global statistical comparison of the estimated block model grades against the declustered composite statistics. ○ Change of support analysis on major lithological domains. ○ Swath plots and drift plots were generated and checked for skarn and porphyry domains.
Moisture	• All tonnages are stated on a dry basis.
Cut-off	• The Mineral Resources is reported above a range of cut-offs based on material type and ore body. The cut-off ranges from 0.16% Cu cut-off grade for hypogene

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
parameters	material to 0.23% Cu for marble/calc-silicate hosted material at Ferrobamba. Oxide material has been reported above a 1% Cu cut-off grade. The reported Mineral Resources have also been constrained within a US$3.40/lb pit shell with revenue factor=1. • The reporting cut-off grade is in line with MMG's policy on reporting of Mineral Resources which considers current and future mining and processing costs and satisfies the requirement for prospects for future economic extraction.
Mining factors or assumptions	• Mining of the Las Bambas deposits is undertaken by open pit mining method, which is expected to continue throughout the life of mine. Large scale mining equipment including 300 tonne trucks and 100 tonne face shovels will be used for material movement. • During block regularisation, internal dilution is included to produce full block estimates. • Further information on mining factors is provided in Section 4 of this table. • No other mining factors have been applied to the Mineral Resources.
Metallurgical factors or assumptions.	• Currently the processing of oxide copper mineralisation has not been studied to pre-feasibility or feasibility study level. The inclusion of oxide copper Mineral Resources is based on the assumption that processing of very similar ores at Tintaya was completed successfully in the past. A head grade of greater than 1.5% Cu was required for favourable economics. This assumption has been used at this stage for the oxide copper mineralisation. • Sulphide and partially oxidised material is included in the Mineral Resources which is expected to be converted to Ore Reserves and treated in the onsite concentrator facilities. • No other metallurgical factors have been applied to the Mineral Resources.
Environmental factors or assumptions	• Environmental factors are considered in the Las Bambas life of asset work, which is updated annually and includes provision for mine closure. • Geochemical characterisation undertaken in 2007 and 2009 indicate the majority of the waste rock from Ferrobamba and Chalcobamba deposits to be Non-Acid Forming (NAF) and that no acid rock drainage from the waste rock dumps from these two pits should be expected. Waste rock samples from Sulfobamba were found to contain higher concentrations of sulphur and that 30% to 40% of waste rock could be Potentially Acid Forming (PAF). Suitable controls will be implemented for all PAF waste rock, including investigating opportunities for backfill into pit voids. Additional geochemical characterisation work is ongoing. • Tailings generated from processing of Ferrobamba and Chalcobamba were determined to be NAF. Geochemical characterisation of tailings generated from processing of Sulfobamba ores is currently under assessment, however for environmental assessment purposes it was assumed to have PAF behaviour. Current Life of Asset schedules have Ferrobamba tailings processing scheduled for several years after Sulfobamba tailings are processed. Effectively encapsulating the Sulfobamba tailings in the Tailings Storage Facility(TSF) . A closure plan was submitted and approved by the regulator in 2016.

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	• Based on the current TSF design and the design assumptions for dry settled density and beach angle, the TSF at Las Bambas has a final capacity of 784Mt of tailings from processing 800Mt. Three studies have been conducted looking at increasing tailings storage capacity at Las Bambas: o Tailings characterisation test work to assess final settled density and beach slope in current TSF. o Options assessment to increase capacity at TSF currently under construction. o Pre-feasibility study for an additional TSF.
Bulk density	• Bulk density is determined using the Archimedes' principle (weight in air and weight in water method). Samples of 20cm in length are measured at a frequency of approximately one per core tray and based on geological domains. The density measurements are considered representative of each lithology domain. • Bulk density measurement occurs at the external, independent assay laboratory. The core is air dried and whole core is immersed in wax prior to bulk density determination to ensure that void spaces are accounted for. • Density values in the Mineral Resources models are estimated using Ordinary Kriging within the lithology domain shapes. Un-estimated blocks were assigned a density value based on an expected value of un-mineralised rock within each geological domain.
Classification	• Mineral Resource classifications used criteria that required a certain minimum number of drillholes. The spatial distribution of more than one drillhole ensures that any interpolated block was informed by sufficient samples to establish grade continuity. As well, drillhole spacing specific to rock type (skarn vs. porphyry) were used to classify each Mineral Resources category. • Drillhole spacing for classification were based on an internal Ferrobamba drillhole spacing study undertaken in 2015. Results from the study indicate: o Measured Mineral Resources: 25m x 25m drillhole spacing in the skarn, 50m x 50m drillhole spacing for the porphyry. o Indicated Mineral Resources: 50m x 50m drillhole spacing in the skarn, 100m x 100m drillhole spacing for the porphyry. o Inferred Mineral Resources are generally defined as twice the spacing of Indicated Mineral Resources with regard to each rock type. • Only copper estimated values were used for classification. Estimation confidence of deleterious elements such as arsenic was not considered for classification purposes. • The Mineral Resource classification applied appropriately reflects the Competent Person's view of the deposit.
Audits or reviews	• Historical models have all been subject to a series of internal and external reviews during their history of development. The recommendations of each review have been implemented at the next update of the relevant Mineral Resource estimates. • Several extensive reviews were undertaken as part of the MMG due diligence

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	process for the purchase of Las Bambas. These reviews included work done by Runge Pincock Minarco (RPM), which resulted in the published Competent Person report in 2014. In addition significant review work was carried out by AMEC. No fatal flaws were detected in these reviews and all recommendations were considered and addressed in the 2015 Mineral Resources update and all subsequent updates. • A self-assessment of all 2017 Mineral Resource modelling was completed by the Competent Person using a standardised MMG template. No fatal flaws were detected in the review. Areas previously identified for improvement have been addressed and include: o Mineral Resource classification for the Ferrobamba block model uses a wireframe shape to constrain the final Mineral Resource category. o Acid soluble copper results are used to model an oxidation type domain. This is in turn used to constrain the acid soluble copper estimate.
Discussion of relative accuracy / confidence	• There is high geological confidence of the spatial location, continuity and estimated grades of the modelled lithologies within the Mineral Resources. Minor, local variations are expected to occur on a sub-25m scale that is not detectable by the current drill spacing. Global declustered statistics of the composite databases on a domain basis were compared against the block model. Block model estimates were within 10% of the composite database. Local swath plots were undertaken for each deposit. All plots showed appropriate smoothing of composite samples with respect to estimated block grades. • The Las Bambas Mineral Resource estimates is considered suitable for Ore Reserves and mine design purposes. The Mineral Resources model was evaluated using the discrete Gaussian change of support method for copper in most domains. Based on the grade tonnage curves generated, the Mineral Resources model should be a reasonable predictor of tonnes and grade selected during mining. • Reconciliation of the last 12 months of production indicates that the Mineral Resources block model is over-calling the Mill claimed copper metal by 3.7%. This comprises a 3% overcall of grade and a 0.7% overcall of tonnage. These results demonstrate that the Mineral Resources block model is robust and fit for purpose for use in mine planning and Ore Reserves estimation. In the graph below.

Criteria	Status
Section 3 Estimating and Reporting of Mineral Resources	
	 • The accuracy and confidence of this Mineral Resource estimates is considered suitable for public reporting by the Competent Person.

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

This Mineral Resources 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, Rex Berthelsen, confirm that I am the Competent Person for the Las Bambas 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 and Metallurgy
- I have reviewed the relevant Las Bambas Mineral Resources section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Ltd. at the time of the 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 Las Bambas 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 the Las Bambas 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 Las Bambas Mineral Resources - I consent to the release of the 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Rex Berthelsen F.AusIMM(CP) (#109561)

18/10/2017

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Signature of Witness:

Anna Lewin
Melbourne, VIC

Witness Name and Residence: (e.g. town/suburb)

3.3 Ore Reserves – Las Bambas

3.3.1 Results

The 2017 Las Bambas Ore Reserves are summarised in Table 4. All data reported here is on a 100% asset basis. MMG's attributable interest in Las Bambas is 62.5%.

Table 4 2017 Las Bambas Ore Reserves tonnage and grade (as at 30 June 2017)

Las Bambas Ore Reserves (62.5%)						Contained Metal			
Ferrobamba Primary Copper[1]	**Tonnes (Mt)**	**Copper (% Cu)**	**Silver (g/t Ag)**	**Gold (g/t Au)**	**Mo (ppm)**	**Copper (kt)**	**Silver (Moz)**	**Gold (Moz)**	**Mo (kt)**
Proved	497	0.68	3.2	0.06	206	3,366	51	1.0	103
Probable	326	0.71	3.6	0.06	207	2,308	38	0.7	67
Total	**823**	**0.69**	**3.4**	**0.06**	**207**	**5,674**	**89**	**1.7**	**170**
Chalcobamba Primary Copper[2]									
Proved	59	0.53	1.8	0.02	141	312	3	0.04	8
Probable	143	0.72	2.7	0.03	132	1,026	13	0.16	19
Total	**202**	**0.66**	**2.5**	**0.03**	**134**	**1,338**	**16**	**0.20**	**27**
Sulfobamba Primary Copper[3]									
Probable	60	0.80	5.9	0.03	161	480	11	0.1	10
Total	**60**	**0.80**	**5.9**	**0.03**	**161**	**480**	**11**	**0.1**	**10**
Sulphide Stockpiles									
Proved	0.2	0.85	4.5	NA	148	1.5	0.03	NA	0.03
Total	**0.2**	**0.85**	**4.5**	**NA**	**148**	**1.5**	**0.03**	**NA**	**0.03**
Total Contained Metal						**7,494**	**117**	**1.9**	**207**

1 0.19% to 0.27% Cu cut-off grade based on rock type and recovery
2 0.21% to 0.27% Cu cut-off grade based on rock type and recovery
3 0.24% to 0.26% Cu cut-off grade based on rock type and recovery

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

Contained metal does not imply recoverable metal

3.3.2 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 5 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code. 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 5 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Las Bambas Ore Reserve 2017

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td>Mineral Resource estimates for conversion to Ore Reserves</td><td>
Mineral Resource block models have been provided by Group Technical Services (GTS). The block models contain descriptions for lithology, Mineral Resources Classification, mineralisation, ore type, and other variables described in model release memorandums. These block models were used for the pit optimisation purpose using reasonable assumptions for cost and metal prices, GEOVIA Whittle was the software package used for this purpose.
<table>
<tr><th>MR block models</th><th>Ferrobamba</th><th>Chalcobamba</th><th>Sulfobamba</th></tr>
<tr><td>Completed by</td><td>Rex Berthelsen</td><td>Rex Berthelsen</td><td>Anna Lewin / Rex Berthelsen</td></tr>
<tr><td>Memorandum date</td><td>11 April 2017</td><td>21 April 2017</td><td>31 May 2017</td></tr>
<tr><td>Block model file</td><td>lb_fe_mor_1703 v2.bmf</td><td>lb_ch_mor_1701 v5.bmf</td><td>lb_sb_1704_mor_v2.asc</td></tr>
<tr><td>Block size (m)</td><td>20 x 20 x 15</td><td>20 x 20 x 15</td><td>20 x 20 x 15</td></tr>
<tr><td>Model rotation</td><td>35°</td><td>0°</td><td>0°</td></tr>
</table>
The Measured and Indicated Mineral Resources quantities are inclusive and not additional to the Ore Reserves reported.
</td></tr>
<tr><td>Site visits</td><td>
The Competent Person has visited the Las Bambas site four times in the past year. Each visit consisted of discussions with relevant people associated with Ore Reserve modifying factors including Geology, Grade Control, Geotechnical, Mine Planning and Mining Operations, Metallurgy, Tailings and Waste Storage, and Environmental Areas. The outcomes from the visits have included reaching a common understanding in those areas, and achieving the specific purpose of each trip, such as improving the short term mine plan. Site visits were also carried out by contributing experts listed in the expert input table at the end of this document.
</td></tr>
<tr><td>Study status</td><td>
The Las Bambas Ore Reserve estimates were prepared on the basis of Feasibility and Pre-Feasibility level studies that include the following:
Bechtel Feasibility Study 2010; and
Las Bambas Mine Site 3 TSF Prefeasibility Study, MWH, 2015.
Additional work/studies include:
Glencore Mineral Resources and Ore Reserves Report 2013;
Audit of Las Bambas Ore Reserves 2013, by Mintec, Inc in October 2013;
MMG Competent Person Report prepared by Runge Pincock Minarco (RPM), June 2014;
</td></tr>
</table>

Assessment Criteria	Commentary
	○ MMG Las Bambas cut-Off Grade Report 2017; ○ Geotechnical guidance by Piteau (2009-2010); ○ Geotechnical work conducted by site personnel and Itasca, 2015; ○ Sulfobamba Metallurgy Testing, 2015; ○ Tailings Storage Facility – Initial review of options to extend filing life, ATCW, 2015; ○ Technical review of future TSF, Khlon Crippen Berger, 2016; and ○ Conceptual Development of New Tailings Storage Facility, Ausenco 2017. • 2017 Life of Asset Plan (LoA) Low Case was produced as part of the MMG planning cycle demonstrates this is technically achievable and economically viable, and that material Modifying Factors have been considered.
Cut-off parameters	• MMG Board approved metal prices for the cut-off calculation have been provided by MMG Group Technical Services in accordance with the MMG MROR Standard. • Costs were estimated based on information provided by the Las Bambas Finance Department. • The breakeven cut-off (BCoG) 2017 has been calculated with updated metal prices and costs, and is applied to the CuT%. (Source: 2017 Las Bambas CoG Report). • Cut-off grade has been determined for each ore-type within each deposit:

Cut-off grades by ore-type for Ferrobamba:

COG Component	Ferrobamba by Ore Type					
	FSSL	FSSM	FPSL	FPSM	FMSL	FBRE
BCOGinpit	0.19%	0.22%	0.20%	0.24%	0.27%	0.25%

Cut-off grades by ore-type for Chalcobamba:

COG Component	Chalcobamba by ORE TYPE						
	CSSL	CSSM	CSML	CSMM	CPSL	CPSM	CBRE
BCOGinpit	0.21%	0.24%	0.21%	0.22%	0.21%	0.26%	0.27%

Cut-off grades by ore-type for Sulfobamba:

COG Component	Sulfobamba by ORE TYPE				
	SSSL	SSSM	SPSL	SPSM	SBRE
BCOGinpit	0.24%	0.26%	0.24%	0.25%	0.25%

Assessment Criteria	Commentary
Mining factors or assumptions	• The method of Ore Reserves estimation included pit optimisation, final pit and phase designs, consideration of mining schedule and all modifying factors. Addition information is provided in this section. • The mining method selected for the Las Bambas operation is open cut mining, which enables bulk mining of these large low to moderate grade mineral deposits that are outcropping to sub-cropping. Three deposits, Ferrobamba, Chalcobamba and Sulfobamba are each mined in separate open pits. Mining is by way of conventional truck and shovel operation. This method and selected mining equipment are appropriate for these deposits.

Assessment Criteria	Commentary

- The geotechnical recommendations were provided by the Geotechnics & Hydrogeology team at Las Bambas in coordination with MMG Group Technical Services. These recommendations are based on studies performed by Piteau (2009-2010), site personnel and Itasca (2015). The pits are sectored by structural zones, taking into account the last recommendations for Bench Face Angle (BFA) which indicates 65° and 70° for upper and lower benches respectively. Mine designs slope angles are shown below in the corresponding tables.
- The geotechnical parameters used for the slopes are as follows:

Geotechnical recommendations for Ferrobamba

Sector	Level (m)	BFA (°)	H (BFA)	Catch Bench	IRA (°)	Height 70°& 65°	Decoupling Height	Decoupling Width	Angle By Zone	OA (°)	Total Height (m)	Bench (number)
N	3465-3975	70	30	15	49.2	510	210	35	**47.6**	46.3	780	26
	3975-4245	65	30	15	46	270	210	35	**43.9**			
NE	3465-3975	70	30	15	49.2	510	210	35	**47.6**	46	795	27
	3975-4260	65	30	15	46	285	210	35	**43.3**			
E	3390-3810	70	30	17.1	47	420	210	35	**46.9**	45.1	630	21
	3810-4020	65	30	17.1	44	210	210	35	**41.7**			
SE	3390-3810	70	30	14	50.3	420	210	35	**49.7**	47.8	570	19
	3810-3960	65	30	14	47	150	120	35	**43.0**			
S	3390-3675	70	15	8.5	47.1	285	120	35	**45.2**	43.1	420	28
	3675-3810	65	15	8.5	44.1	135	120	35	**39.1**			
SW	3525-3720	70	15	9.7	44.7	195	120	35	**42.5**	40.7	315	21
	3720-3840	65	15	9.7	41.9	120	120	30	**38.0**			
NW	3465-3975	70	30	15	49.2	510	210	35	**47.6**	46	795	27
	3975-4260	65	30	15	46	285	210	35	**43.3**			
CE	3390-3660	70	30	11.2	53.6	270	210	35	**52.0**	52	270	9

Geotecnical recommendations for Chalcobamba:

Sector	Level (m)	BFA (°)	H (BFA)	Catch Bench	IRA (°)	Height Zone (m)	Decoupling Height	Decoupling Width	Decoupling Number	Angle By Zone	Angle +decoupling	OA (°)	Total Height (m)	Bench (number)
CH-S2	4330-4450	70	15	8	48.1	120	90	35	1	45.7	**50.3**	44.1	210	8
	4450-4540	65	15	8	45	90				42.1	**47.7**			6
CH-SE	4255-4465	70	15	8	48.1	210	90	35	1	46.7	**49.3**	45.2	300	14
	4465-4555	65	15	8	45	90				42.1	**47.7**			6
CH-E	4165-4435	70	15	8	48.1	270	105	35	2	44.1	**45.9**	43.6	375	18
	4435-4540	65	15	8	45	105				42.5	**47.3**			7
CH-N	4165-4360	70	15	8	48.1	195	105	35	1	46.6	**49.4**	45.1	300	13
	4360-4465	65	15	8	45	105				42.5	**47.3**			7
CH-NW	4165-4285	70	15	8	48.1	120	90	35	1	45.7	**50.3**	44.1	210	8
	4285-4375	65	15	8	45	90				42.1	**47.7**			6
CH-W	4165-4330	70	15	8	48.1	165	90	35	1	46.3	**49.7**	44.8	255	11
	4330-4420	65	15	8	45	90				42.1	**47.7**			6
CH-SW	4315-4435	70	15	8	48.1	120	90	35	1	45.7	**50.3**	44.1	210	8
	4435-4525	65	15	8	45	90				42.1	**47.7**			6
CH-S1	4315-4450	70	15	8	48.1	135	90	35	1	45.9	**50.0**	44.4	225	9
	4450-4540	65	15	8	45	90				42.1	**47.7**			6

Geotechincal recommendations for Sulfobamba

Sector	Zone	C_Lito	Level (m)	BFA (°)	H (BFA)	Catch bench	IRA (°)	Height Zone (m)	Decoupling Height	Decoupling Width	Decoupling Number	Angle by zone	OA (°)	Total Height (m)	Bench (number)
SU-S	Sup.	71	4565 - 4475	65	15	8	42	90	150	35	1	**38**	41	255	6
	Inf.	40, 47	4475 - 4310	70	15	8	45	165				**42**			11
SU-E	Sup.	71	4565 - 4445	65	15	8	42	120	150	35	1	**38**	42	255	8
	Inf.	40, 47	4445 - 4310	70	15	8	45	135				**42**			9
SU-NE	Sup.	71	4420 - 4345	65	15	8	42	165	150	35	1	**38**	42	255	11
	Inf.	40	4345 - 4165	70	15	8	45	90				**42**			6
SU-N	Sup.	81	4460 - 4310	65	15	8	44	150	150			**45**	45	150	10
SU-W	Sup.	40	4565 - 4505	65	15	8	42	60	150	35	1	**38**	41	195	4
	Inf.	80, 81	4505 - 4370	70	15	8	44	135				**41**			9

- To improve the reliability and safety of the pit design at Ferrobamba decoupling berms have been included on all sectors.
- The 2017 Mineral Resource models for Ferrobamba and Chalcobamba have been used for the updated 2017 Ore Reserves. The Mineral Resource model for Sulfobamba remained the same as 2016 and 2015 but was regularised to

Assessment Criteria	Commentary
	20m x 20m x 15m block sizes for Ore Reserves estimation. • The pit optimisation was developed for the three open pits based on the 2017 Mineral Resource block models, the strategy for the final pit selection was based on the NPV by pit shell at revenue factor 1 (RF=1.0). RF 1.0 pit shell is used in all assets across the MMG Group. Final pit designs that incorporate more practical mining considerations were carried out using those optimisation shells. • Dilution and recovery have been accounted for in the regularised block model used for the Ore Reserves estimate. Hence, the Ore Reserves estimate has applied no further factoring. Additional studies for dilution and recovery will be undertaken when more reconciliation data is available. • In the pit, the minimum mining width is 70m; the minimum SMU has been set at 10m x 10m x 15m. • Inferred Mineral Resource material has not been included in the pit optimisation or in the Ore Reserves estimates. • The main mining infrastructure includes; crusher, overland conveyor, tailings and waste storage facilities, stockpiles, roadways/ramps, workshops and so forth. • All infrastructure requirements are established for Ferrobamba. Further capital investment is required for infrastructure to support the needs of Chalcobamba and Sulfobamba. • The required infrastructure for Chalcobamba pit have been identified and included in the current approved Environment Impact Assessment (EIA), with 50% of the north waste dump not located within the property boundary. Another location that is fully within the property boundary has been identified and used in the LoA planning; however, it is yet to be evaluated by environmental/legal/exploration areas and will be included in the 4th EIA amendment. • The planned Sulfobamba infrastructure has been identified within the Las Bambas mining concession, however the infrastructure and deposit are not located within the area of MMG land ownership.
Metallurgical factors or assumptions	• The metallurgical process is a conventional froth flotation concentrator and thickener to produce two separate Cu and Mo concentrates and is appropriate for the style of mineralisation. • Metallurgical copper concentration process comprises the following activities; crushing, grinding and flotation, producing copper and molybdenum concentrates. Copper concentrates contain gold and silver as byproducts. Las Bambas Project commenced commercial production on 1 July 2016. • Extensive comminution and flotation test work has been conducted and metallurgical recoveries determined for different rock types and different mining areas. • Bulk samples and pilot scale tests have been conducted on representative samples of the deposits. For the Ferrobamba deposit, nearly all of the tests were completed by the G&T laboratory in Canada as part of Feasibility

Assessment Criteria	Commentary
	Study, though a small number of additional confirmatory tests were included from more recent testing by ALS in Peru. For Chalcobamba, all of the tests were completed by G&T and reported in the Feasibility Study. For Sulfobamba, the data analysed were those from testing at G&T in 2015. • The ore contains no deleterious elements which impact concentrate quality. • The recovery equations have been provided by the Metallurgical Group at Las Bambas in coordination with MMG Group Technical Services. • The equations were generated based on metallurgical test work, from diamond drilling sample data in each pit. It should be noted that the copper recovery is a function of the ratio of acid soluble copper (CuAS) to total copper (CuT), which is a determining factor for the recovery. The copper recovery is determined by the following equations. ***Ferrobamba:*** For all the materials except marble: *Cu Recovery (%) = (96.0-94.0*(CuAS/CuT) + 1.5* For marble: *Cu Recovery (%) = 89.0 – 205.0*(CuAS/CuT) + 1.5* ***Chalcobamba:*** *Cu Recovery (%) = 94.4-90.0*(CuAS/CuT) + 1.5* ***Sulfobamba:*** *Cu Recovery (%) = 89.2 - 80.4*(CuAS/CuT) + 1.5* An improvement in recovery of 1.5% has been added to account for ongoing metallurgical improvement work since the start of operation. • The recovery of Mo, Ag, Au, has been provided by Metallurgical Group at Las Bambas.

Metal		Ferrobamba	Chalcobamba	Sulfobamba
Mo	%	55.5	55.5	55.5
Ag	%	72.7	54.3	54.3
Au	%	65.0	56.6	56.6

Assessment Criteria	Commentary
	• Benchmarking of data from four other mines in South America has been completed and the recovery algorithms that describe performance at these mines compared with those used for the three Las Bambas deposits. The four other mines were Tintaya, Toquepala, Cerro Verde and Antamina.
Environmental	• The Environmental Impact Study for the Las Bambas Project was approved on 7 March 2011 by the Peruvian Government with directorial resolution Nº073-2011-MEM/AAM. • The construction of the project processing facilities including Tailings Storage Facility at Las Bambas was approved 31 May 2012 by the General Directorate of Mining through Resolution N°178-2012-MEM-DGM/V. • The Mine Closure Plan for the Las Bambas Project was approved 11 June

Assessment Criteria	Commentary
	2013 through Directorial Resolution N°187-2013-MEM-AAM. As per the Peruvian regulatory requirements, an update of the Mine Closure Plan was approved 28 September 2016, through Directorial Resolution N°288-2016-MEM-DGAAM. • A first amendment to the Environmental Impact Study was approved 14 August 2013 through Directorial Resolution N°305-2013-MEM-AAM, whereby amendments to the Capacity of the Chuspiri water reservoir and changes to the environmental monitoring program were approved. • On 26 August 2013 the relocation of the molybdenum circuit (molybdenum plant, filter plant and concentrate storage shed) from the Antapaccay region to the Las Bambas project area was approved by the environmental regulator through Directorial Resolution Nº319-2013-MEM-AAM, after assessment of the technical report showed that the environmental impacts of the proposed changes were not significant. • On 6 November 2013, through Directorial Resolution N°419-2013-MEM-DGM/V, the General Directorate of Mining approved the amendment of the construction permit for the processing facilities to allow the construction of the molybdenum circuit at the Las Bambas project area. • Minor changes to the project layout were approved 13 February 2014 through Directorial Resolution N°078-2014-MEM-DGAAM and 26 February 2015, through Directorial Resolution N°113-2015-MEM-DGAAM. • On 17 November 2014, the second modification of the Study of Environmental Impact was approved with Directorial Resolution Nº 559-2014-EM/DGAAM, whereby changes to the water management infrastructure and changes in the transport system from concentrate slurry pipeline to bimodal transport (by truck and train) were approved. • A second amendment to the construction permit for processing facilities was approved through Directorial Resolution N°419-2015-MEM-DGM/V to allow changes to the design of the tailings storage facility and changes to the water management system and auxiliary infrastructure. • Environmental approval for minor changes to project layout was approved 1 June 2016 through Directorial Resolution N°177-2016-MEM-DGAAM. • Geochemical characterisation studies on waste rock samples were conducted in 2007 and 2009/2010 as part of the Feasibility and Environmental Impact Studies, conclusions from these studies indicate that it should be expected that less than 2% of the waste rock from the Ferrobamba and Chalcobamba pits should be Potentially Acid Forming (PAF). No acid rock drainage from the waste rock dumps from these two pits should be expected. Waste rock samples from Sulfobamba were found to contain higher concentration of sulphur and that 30% to 40% of waste rock could be PAF. • Further geochemical characterisation studies on waste rock and tailings samples are currently underway to support the Environmental Impact Study Modification. • The operation of the Ferrobamba waste rock dump was approved on 29th September 2015 by the General Directorate of Mining through Directorial

Assessment Criteria	Commentary
	Resolution N°1780-2015-MEM/DGM. • Currently, Las Bambas has four water use licenses: ○ License for underground water obtained with Directorial Resolution 0519-2015-ANA / AAA.XI.PA, for a volume up to 9,460.800 m3 / year. ○ License for non-contact water obtained with Directorial Resolution 0518-2015-ANA / AAA.XI.PA, for a volume up to 933.993 m3 / year. ○ License for contact water obtained with Directorial Resolution 0520-2015-ANA / AAA.XI.PA, for a volume up to 4,730,400 m3 / year. License for contact water obtained with Directorial Resolution 0957-2016-ANA / AAA.XI.PA, for a volume up to 4,730,400 m3 / year. ○ License for fresh water obtained with Directorial Resolution 0778-2016-ANA / AAA.XI.PA, for a volume up to 23,501,664 m3 / year.
Infrastructure	• Las Bambas has the following infrastructure established on site: ○ Concentrator currently in operation. ○ Based on the current TSF design and the design assumptions for dry settled density and beach angle, the TSF currently under construction at Las Bambas has a final capacity of 784Mt of tailings from processing 800Mt. Three studies have been conducted looking at increasing tailings storage capacity at Las Bambas: – Tailings characterization test work to assess final settled density and beach slope in current TSF. – Options assessment to increase capacity at TSF currently under construction. – Pre-feasibility study for an additional TSF at Tambo valley. ○ Camp accommodation for staff ○ Water supply is sufficient for site and processing, sourced from the following: rainfall and runoff into Chuspiri dam, groundwater wells, contact waters, recirculating water in the process plant. Pump station from Challhuahuacho River is being considered for the future. ○ Transport of the copper concentrate is performed by trucks, covering a distance of 380km, to the Imata Village, then it is transported by train, covering a distance of 330km, up to Matarani Port in Arequipa region. Transport of the molybdenum concentrate will be performed by trucks all the way from Las Bambas site to Matarani Port, covering a distance of 710Km. This method is also used temporarily for some of the copper concentrate. ○ There are principal access roads that connect Las Bambas and national routes, Cotabambas to Cusco and Cotabambas to Arequipa. ○ High voltage electrical power is sourced from the national grid

Assessment Criteria	Commentary
	Cotaruse – Las Bambas, with a capacity of 220Kv. o The majority of staff working at the operation are from the region immediately surrounding the project. o Technical personnel are mostly sourced from within Peru. The operation has a limited number of expatriate workers. Additional support is provided by MMG office's in Lima and Melbourne Group office personnel. o Chalcobamba pit operation requires additional purchase of land to the North side of the lease (Waste dump 2). However, an alternative location is being evaluated that does not require land purchase; this is what the LoA planning is based on currently. Sulfobamba pit operation requires additional purchase of land for the pit and other infrastructure.
Costs	• Las Bambas Project commenced commercial production on 1 July 2016; future additional capital costs such as TSF 2 expansion are based on pre-feasibility studies. The operating costs used for Ore Reserves estimation are based on the 2017 Budget (2017-2019) and 2016 Life of Asset (LoA) (2020 onwards) as per Corporate (MMG) guidelines and other considerations. Specifically: o Average costs are calculated by using the first 3 years budget plus remaining LoA estimated costs year by year; o Necessary adjustments required for the prices and consumption rates, updated during the budget process, are made to establish connection between the budget and LoA; and o Approved cost savings from identified initiatives and improvements to be delivered over the life of mine are incorporated. • No deleterious elements are expected in the concentrates that would result in smelter penalties. • Metal prices and exchange rates are the same as those reported in the section for cut-off parameters. These Board approved prices and rates are provided by MMG Corporate and are based on external company broker consensus and internal MMG strategy. • Transportation charges are based on quotations from local companies. • Treatment and refining charges (TC/RC's) are based on marketing actuals and projections. Royalties payable are based on information provided by the Finance and Commercial Group at Las Bambas. • Sustaining capital costs, together with mining costs and processing costs have been included in the pit optimisation. The sustaining capital costs are principally related to TSF construction and equipment replacement. The inclusion or exclusion of these costs in the Ore Reserves estimation process has been done following MMG guidelines according the objective of each capital expenditure in the operation.
Revenue factors	• All mining input parameters are based on the Ore Reserves estimate LoA Low Case production schedule. All cost inputs are based on tenders and

Assessment Criteria	Commentary
	estimates from contracts in place as with net smelter returns (NSR) and freight charges. These costs are comparable with the regional averages. • The gold and silver revenue is via a credit at the refinery. • TC/RC's have been included in the revenue calculation for the project.
Market assessment	• MMG considers that the outlook for the copper price over the medium and longer term is positive, supported by further modest demand growth which is expected to exceed increases in supply. • Global copper consumption growth will continue to be underpinned by rising consumption in China and the developing countries in Asia as these nations invest in infrastructure such as power grids, commercial and residential property, motor vehicles and transportation networks and consumer appliances such as air conditioners. • Supply growth is expected to be constrained by a lack of new mine projects ready for development and the requirement for significant investment to maintain existing production levels at some operations. • Las Bambas has life of mine agreements in place with MMG South America and CITIC covering 100% of copper concentrate production which is sold to these parties at arms' length international terms. These agreements ensure ongoing sales of Las Bambas concentrate.
Economic	• The costs are based on the 2017 LoA Low Case projections based on actual costs, 2017 Budget, and other sources including tendered prices for consumables and estimated quantities of consumables, labour, and services. • The financial model of the Mine Plan shows that the mine has a substantially positive NPV. The discount rate is in line with MMG's corporate economic assumptions. • Various sensitivity analyses on the key input assumptions were undertaken during mine optimisations and financial modelling. All produce robust positive NPVs.
Social	• Las Bambas project is located in the Apurimac region that has a population of approximately 456,000 inhabitants (Census 2014). The region has a University located in the city of Abancay, with mining programs that provide professionals to the operation. The project straddles two provinces of the Apurimac region, Grau and Cotabambas. • Las Bambas Project contributes to a fund - "FOSBAM" - that promotes activities for the sustainable development of the disadvantaged population within the project's area of influence, comprising the provinces Grau and Cotabambas – Apurimac. • Las Bambas Project and the local community entered into an agreement for the resettlement of the rural community of Fuerabamba, Convenio Marco. The construction the township of Nuevo Fuerabamba was completed in 2014 and the resettlement of all community families were completed in 2016. Currently, in joint work with the community, MMG are executing the commitments described in the framework agreement - Convenio Marco. • During the extraordinary general meeting in January 2010, Fuerabamba

Assessment Criteria	Commentary
	community approved the agreements contained in the so-called "compendium of negotiating resettlement agreements between the central negotiating committee of the community of the same name and representatives of Las Bambas mining project" that considers 13 thematic areas. • Las Bambas Project provides important support to the community in the areas of agriculture, livestock and health, education, and other social projects, which is well received. • Las Bambas has had periodic social conflict with communities along the public road used for concentrate transport and logistics. In response to these concerns Las Bambas has committed to key aspects of the Cotabambas Social Development Plan, has formalised dialogue processes with communities along the road; and is working to systematically improve the road conditions and reduce the impacts, while also maximising the social development opportunities available to these communities. • Las Bambas applies ICMM and other world class social standards.
Other	• Las Bambas owns 7,718Ha of land within the mining project. • The mining concession totals an area of 35,000 hectares, which includes the area containing the three mineral deposits and their corresponding infrastructures. • Only 10% of the concession of Las Bambas has been explored year to date. • According to Directorial Resolution N°187-2013-MEM-DGM/V, dated May 2nd, 2013, the Pit Mining Plan and the waste dump was approved for the Las Bambas project. • Approval for the exploitation of the Ferrobamba pit was granted on 30th September 2015 through Directrorial Resolution N° 1780-2015-MEM/DGM. • The title for the Beneficiation Concession and approval for the operation of the concentrator plant was granted on November 30th, 2015 through Directorial Resolution N° 2536-2015-MEM/DGM. • It is reasonable to expect that the future land acquisition and community issues will be materially resolved and government approvals will be granted within the required timeframe.
Classification	• The classification of Ore Reserves is based on the requirements set out in the JORC code 2012, based on the classification of Mineral Resources and the cut-off grade. The material classified as Measured and Indicated Mineral Resources within the final pits and is above the breakeven cut-off (BCoG CuT%) grade is classified as Proved and Probable Ore Reserves respectively. • The Competent Person considers this appropriate for the Las Bambas Ore Reserves estimate. • No Probable Ore Reserves have been derived from Measured Mineral Resources.
Audit or Reviews	• The 2014 Ore Reserves were reviewed by Runge Pinock Minarco for the MMG Competent Person's Report.

Assessment Criteria	Commentary
	• The 2017 Ore Reserve estimates have been reviewed and validated by Edgard Mendoza, Las Bambas Long Term Planning Superintendent; in coordination with Group Technical Services. • No external reviews have been conducted for the 2017 Ore Reserves estimate.
Discussion of relative accuracy/ confidence	• The principal factors that can affect the confidence on the Ore Reserves are: o Proved Ore Reserves are considered to have a relative accuracy of +/- 15% within a volume equivalent to 3 months of production while Probable Ore Reserves are considered to have a relative accuracy of +/- 15% within a volume equivalent to 12 months of production. o Geotechnical risk related to slope stability due to uncertainties in the geomechanical / hydrology models, rock mass blast damage, and increase in mining rate. o Metallurgical recovery model uncertainty due to operational variability. In the best case scenario, this would represent variability of +/- 2% and in the worst scenario +/- 5% to metal recovery. o Increases in rising operating costs for mining and processing. o Increase in selling cost due to the transportation (truck and rail) cost increases. o Capital costs variation for the new or expansion of current TSF. Also land acquisition and/permitting delays for additional TSF needs. o Mining dilution and recovery adjustments as results of future reconciliation studies when more operational data become available. o Change in environmental legislation, could be more demanding.

3.3.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 6.

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 6 Contributing experts – Las Bambas Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Rex Berthelsen, Group Manager Technical Governance, MMG Ltd (Melbourne)	Mineral Resources model
Amy Lamb, Principal Metallurgist, MMG Ltd (Melbourne)	Updated processing parameters and production record
Luis Tejada, Superintendent Geotechnics and Hydrogeology, MMG Ltd (Las Bambas)	Geotechnical parameters
Christian Holland, Principal Geotechnical Engineering, MMG Ltd (Melbourne)	
Edgard Mendoza, Long Term Planning Superintendent, MMG Ltd (Lima)	Cut-off grade calculations Whittle/MineSight optimisation and pit designs
Yerko Delgado, Technical Services Manager (acting), MMG Ltd (Las Bambas)	Production reconciliation
David Machin, Principal Water and Tailings Engineer, MMG Ltd (Melbourne)	Tailings dam design
Giovanna Huaney, Environmental Superintendent, MMG Ltd (Las Bambas)	Environmental/Social/Permitting
Ignacio Ishizawa, Social Development Manager, MMG Ltd (Las Bambas)	
Liz Vergara, Land Access and Resettlement Manager, MMG Ltd (Las Bambas)	
Ramiro Zuñiga, Business Evaluation Specialist, MMG Ltd (Lima)	Economics and Evaluation
Steve Whitehead, General Manager Marketing, MMG Ltd (Melbourne)	Marketing

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

This Ore Reserve 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, Yao Wu, confirm that I am the Competent Person for the Las Bambas Ore Reserve 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 Fellow of The Australasian Institute of Mining and Metallurgy
- I have reviewed the relevant Las Bambas Ore Reserve section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Ltd. at the time of the 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 Las Bambas Ore Reserve 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 the Las Bambas Ore Reserve.

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 Las Bambas Ore Reserves - I consent to the release of the 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

18/10/2017

Yao Wu MAusIMM(#108391)

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Karel Steyn
Melbourne, VIC

Signature of Witness:

Witness Name and Residence: (e.g. town/suburb)

4 KINSEVERE OPERATION

4.1 Introduction and setting

Kinsevere is located in the Katanga Province, in the southeast of the Democratic Republic of Congo (DRC). It is situated approximately 27km north of the provincial capital, Lubumbashi (Figure 4-1), at latitude S 11° 21′ 30″ and longitude E 27° 34′ 00″.

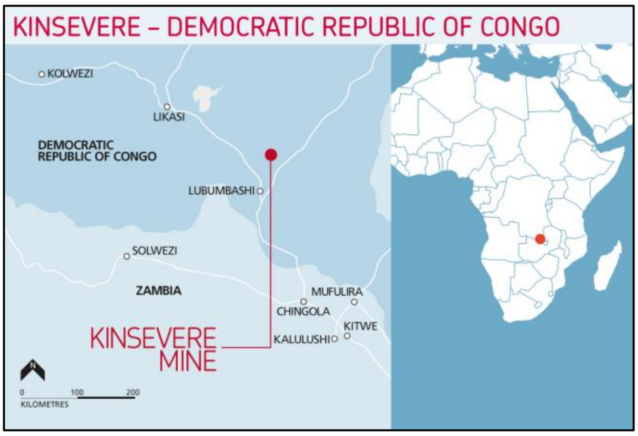


Figure 4-1 **Kinsevere Mine location**

Kinsevere is a conventional truck and excavator operation with atmospheric leaching of the oxide ore using a solvent extraction electro-winning (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% Cu. The HMS was decommissioned in June 2011 when the Stage II SXEW plant was commissioned.

4.2 Mineral Resources - Kinsevere

4.2.1 Results

The 2017 Kinsevere Mineral Resources are summarised in Table 7. The Kinsevere oxide Mineral Resources is inclusive of the Ore Reserves.

Table 7 2017 Kinsevere Mineral Resources tonnage and grade (as at 30 June 2017)

Kinsevere Mineral Resources					
				Contained Metal	
Oxide Copper[2]	**Tonnes (Mt)**	**Copper (% Cu)**	**Copper (AS % Cu)**	**Copper ('000)**	**Copper AS ('000)**
Measured	3.0	4.4	3.9	132	115
Indicated	13.6	3.0	2.6	405	347
Inferred	2.8	2.3	1.9	63	54
Total	**19.4**	**3.1**	**2.7**	**601**	**517**
Transition Mixed Ore (TMO) Copper[3]					
Measured	0.3	2.7	1.0	7	3
Indicated	1.4	2.3	0.9	33	13
Inferred	0.1	2.1	0.8	3	1
Total	**1.8**	**2.4**	**0.9**	**43**	**17**
Sulphide Copper[4]					
Measured	0.4	2.5	0.4	10	2
Indicated	23.8	2.2	0.2	533	36
Inferred	2.2	1.7	0.1	38	2
Total	**26.4**	**2.2**	**0.2**	**581**	**40**
Stockpiles					
Indicated - Oxide	5.7	2.3	1.8	132	101
Indicated - TMO	2.4	3.0	1.4	73	33
Total	**8.1**	**2.5**	**1.6**	**206**	**134**
Kinsevere Total	**55.7**	**2.6**	**1.3**	**1431**	**708**

[1] AS stands for Acid Soluble
[2] 0.6% Acid soluble Cu cut-off grade
[3] 1.1% Total Cu cut-off grade
[4] 0.8% Total Cu cut-off grade
All Mineral Resources except stockpiles are contained within a US$3.40/lb Cu pit shell
Contained metal does not imply recoverable metal.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

4.2.2 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 8 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 8 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Kinsevere Mineral Resources 2017

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
Sampling techniques	• The Mineral Resource estimates use a combination of reverse circulation (RC) drilling diamond drilling (DD). The RC drilling is predominately collected for grade control and the DD is used for exploration and resource delineation work. • DD core is sampled mostly as 1m intervals while samples in un-mineralised zones are sampled over 4m lengths. Sampling is predominantly performed by cutting half core, with half retained on site for future reference. For PQ drilling undertaken 2015-2016, quarter core was submitted for sampling. • Grade control drilling (RC) is composited into 2m samples collected after riffle splitting. • Each sample is crushed and pulverised to produce a pulp (>85% passing 75µm) prior to analysis at the site SGS laboratory. • Measures taken to ensure sample representivity include orientation of the drillholes as close as practical to perpendicular to the known mineralised structure. In addition field duplicates have been taken and analysed. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Kinsevere mineralisation (sediment hosted base metal) by the Competent Person.
Drilling techniques	• RC drilling was used to obtain 2m composited RC chip samples. 226,687m or 75% of the sample data used in the Mineral Resource estimates were from RC samples (5.5" hammer), of that 172,179m (57%) was from grade control drilling. • PQ and HQ size DD core was used to obtain nominal 1m sample lengths. DD core was not routinely oriented. 74,152m or 24% of the sample data used in the Mineral Resource estimates were from DD samples. • 24,685m of RC grade control drilling and 1,240 of RC drilling was completed since the 2016 estimation and utilised in the 2017 estimate. • 9,368m of DD drilling was completed since the 2016 estimation and utilised in the 2017 estimate. The drilling is dominantly PQ with minor HQ. The recent DD drilling (2015/2016) was drilled to inform a sulphide Scoping and Pre-Feasibility Study (PFS).

Assessment Criteria	Commentary
	• In the view of the Competent Person sampling is of a reasonable quality to estimate the Mineral Resources.
Drill sample recovery	• DD core recovery recorded was typically above 90%, with only minor losses in competent ground (recovery average 97.3% for all drilling, and over 98.7% within ore zones). As expected, the recovery fell in unconsolidated ground such as weathered material close to surface and in vuggy zones of dolomitic rocks (average recovery approximately 85%, in this area). The vuggy zones are generally controlled by major structures. Triple tube core barrels were used to maximise core recovery. DD core recovery and run depth are verified and checked by a geological technician at the drill site. This data is recorded and imported in the database. • RC drilling has been observed for sample recovery with adequate sample volume being returned. However, no quantitative measurements of recovery have been recorded. • There is no observed relationship between core loss and mineralisation or grade - no preferential bias has occurred due to any core loss.
Logging	• RC chips are logged by geologists directly into an Excel logging template, geological information captured includes: lithology, stratigraphy, weathering, oxidation, colour, texture, grain size, mineralogy and alteration. This data is then imported into the database. For DD core samples both geological and geotechnical information is logged which includes: lithology, stratigraphy, mineralisation, weathering, alteration, geotechnical parameters: strength, RQD, structure measurement, roughness and infill material. • All RC chip and DD core samples (100%) have been geologically logged to a level that can support appropriate Mineral Resource estimation. • Logging captures both qualitative descriptions such as geological details (e.g. rock type, stratigraphy) with some quantitative data (e.g. ore mineral percentages). Core photography is not known to have occurred prior to MMG ownership. Since MMG took control of the site all DD core is photographed. • The total length and percentage of the relevant intersections logged is 100%.
Sub-sampling techniques and sample preparation	• DD core was split in half (NQ) or quartered (PQ) using a diamond saw. Sample lengths were cut as close to 1m as possible while also respecting geological contacts. Samples were generally 2kg to 3kg in weight. • RC samples are collected from a cyclone by a trained driller's assistant. If the sample was dry the sample was passed through a riffle splitter and collected into a pre-numbered calico bag. Residual material was sampled and sieved for collection into chip trays for logging and the remainder returned to the larger poly-weave bag.

Assessment Criteria	Commentary
	The splitter was cleaned using compressed air or a clean brush and tapped using a rubber mallet. If the sample was wet the sample was dried in the laboratory oven before being split according to the procedure above (for dry samples). • Samples from individual drillholes were sent in the same dispatch to the preparation laboratory. • Representivity of samples was checked by sizing analysis and duplication at the crush stage. • Field duplicates were inserted at a rate of approximately 7% to ensure that the sampling was representative of the in-situ material collected. Field duplicates in current RC programs have shown acceptable levels of repeatability across all elements analysed. • These practices are industry standard and are appropriate for the grain size of the material being sampled. • RC and DD samples were prepared on-site by the geology department, who provide pulp samples to the SGS analytical facility also on site at Kinsevere. The samples were oven dried at approximately 80°C, crushed to 85% passing -2mm using a jaw crusher and milled to 85% passing 75µm using one of three single sample LM2 vibratory pulverising mills. • Since 2015, DD drilling core was processed at the onsite Exploration core yard. Sample preparation was conducted at this facility through an ALS managed Containerised Preparation Laboratory (CPL). Pulp samples where then sent to ALS Johannesburg for analysis. • The sample size for both RC and DD is considered appropriate for the grain size of the material being sampled.

Quality of assay data and laboratory tests	• RC ore control samples are currently assayed at the onsite SGS Laboratory.
	  o Following preparation, 50g pulp samples are routinely analysed for total and acid soluble copper, cobalt and manganese.
	  o A 3-acid digest with AAS finish was used to analyse for total values.
	  o A sulphuric acid digest with AAS finish was used to analyse for acid soluble copper.
	• All DD core samples prior to 2011 were assayed at:
	  o ALS Chemex Laboratory, Johannesburg.
	  o McPhar Laboratory, Philippines.
	  o ACTLabs Laboratory, Perth.
	  o 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.

Assessment Criteria	Commentary
	• From 2011, prepared samples were submitted to the ISO 17025 accredited SGS Laboratory in Johannesburg with the following assay scheme: o ICP-OES method with a 4-acid digest analysing 32 elements including copper from 0.5ppm to 1%. o ICP-OES method using alkali fusion is applied to over-range copper results. o ICP-AES with a 4-acid digest was used for calcium and sulphur analysis. o XRF was used for uranium analysis. o Acid soluble copper using a sulphuric acid digest and AAS finish. • Since 2015 DD drilling, prepared samples were submitted to the ALS Laboratory in Johannesburg with the following assay scheme: o ICP-OES (ME-ICP61) method with a 4-acid digest analysing 33 elements plus OG triggers when Cu greater than 1% (OG62) o LECO analysed Total Carbon (C-IR07), Organic Carbon (C-IR17), Total Sulphur (S-IR08) and Sulphide Sulphur (S-IR07) o Acid soluble copper using a Sequential Leach (Cu-PKGPH06) finish. • No geophysical tools, spectrometers or handheld XRF instruments have been used in the analysis of samples external to the ALS laboratory for the estimation of Mineral Resources. • QAQC employs the insertion of Certified Reference Material (CRM) every 25 samples; blanks, field duplicates, coarse duplicates and pulp duplicates are taken/ inserted within every batch of 50 samples; and umpire laboratory checks are submitted for every batch of 20 samples to check accuracy, precision and repeatability of the assay result. Acceptable levels of accuracy and precision have been established. If control samples do not meet an acceptable level the entire batch is re-analysed. • The analysis methods described above are appropriate for the style and type of mineralisation.
Verification of sampling and assaying	• Significant intersections are verified by alternate company personnel following receipt of assay results and during the geological modelling process. • Twinned pre-collars are present in the database. These were used to confirm and check geological intervals and/or assay intervals. Twin drillholes are not used in the Mineral Resource estimates. • Data is collected in Excel spread sheets and imported into industry standard databases that have built in validation systems and QAQC reporting systems. Raw data is imported in the database as received

Assessment Criteria	Commentary
	by the laboratory. • Where data was deemed invalid or unverifiable it was excluded from the Mineral Resource estimation. • There are no adjustments to the assay data.
Location of data points	• Prior to 2011 all drillhole collars were located using a hand held GPS. Accuracy of GPS is +/- 5m for x and y coordinates and has poor accuracy of the z (elevation) coordinates. Elevations of these holes were later adjusted by using a LIDAR survey method. • RC and DD holes collared post-2011 are surveyed by qualified surveyors. Downhole surveys have been carried out using Eastman single-shot cameras or Reflex EZ tools. Surveys are taken at variable intervals and stored in the database. • Coordinates are in Kinsevere Mine Grid, which is related to WGS84 by the following translation: -8000000m in northing and -22.3m in elevation. • A LIDAR survey was used to generate a topographic surface. This surface was also used to better define the elevation of drillhole collars. The LIDAR survey is considered to be of high quality and accuracy for topographic control.
Data spacing and distribution	• Grade control RC drill pattern spacing is 5m x 15m, which is sufficient to adequately define lithology and mineralisation domain contacts and transition zones. • The overall DD pattern spacing is between 25m and 100m, which is sufficient to establish the required degree of geological and grade continuity that is appropriate for the Mineral Resources. 2016/2017 DD drilling aimed to infill target areas to 40m x 40m spacing or better. • DD samples are not composited prior to being sent to the laboratory however the nominal sample length is generally 1m. RC samples are 2m intervals but compositing up to 4m has occurred in the past (mainly in areas of known waste).
Orientation of data in relation to geological structure	• The mineralisation strikes between north and north-west at Mashi / Central pits, and to, the east south east at Kinsevere Hill. All drillholes are oriented such that drillholes have a high angle of intersection with the dominant strike and dip of bedding and structures, with the local scale of mineralisation also considered. All drillholes are either oriented east or west with dips of 60° to sub-vertical. • The combination of both east and west orientations likely minimises sampling bias, which, if present, is not considered material.

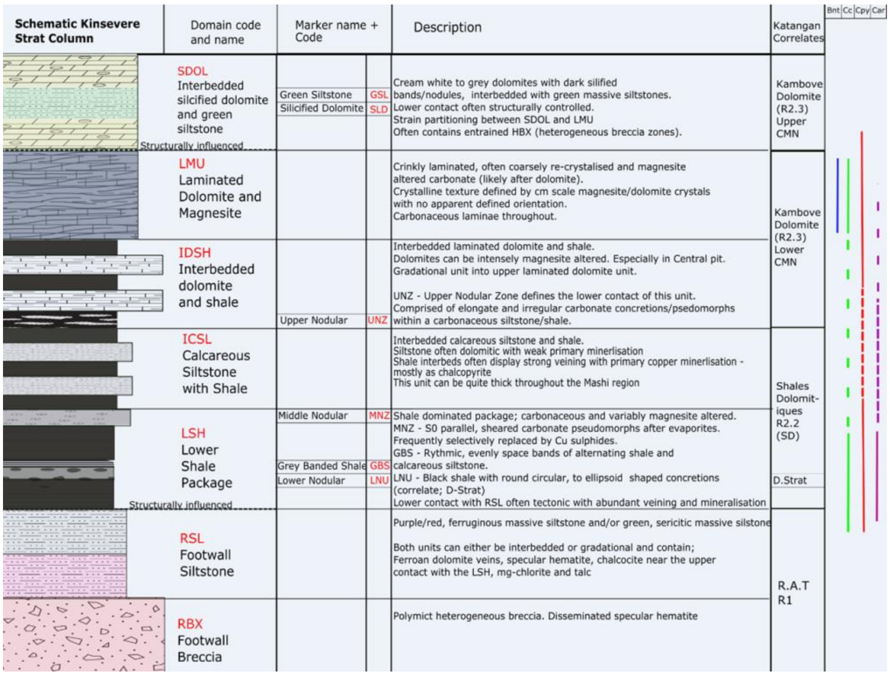
Sample security	• Measures to provide sample security include:
	    ○ Adequately trained and supervised sampling personnel.
	    ○ Sea containers used for the storage of samples are kept

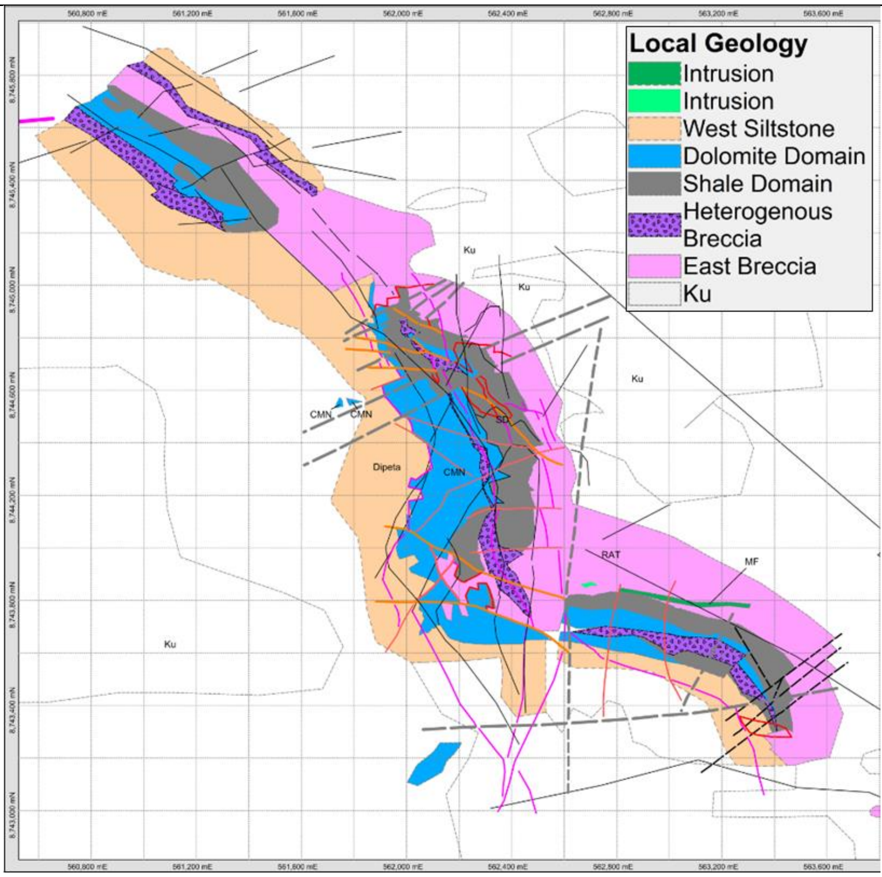
Assessment Criteria	Commentary
	locked with keys held by the security department. o Assay laboratory checks of sample dispatch numbers against submission documents.
Audit and reviews	• An independent audit of the Mineral Resource model was completed in October 2016, by Pennywise Pty Ltd as part of an audit on the results of the Sulphide Scoping Study and was commissioned by MMG Limited. No material errors were found. • Internal visits by the Competent Person and MMG Group Office geologists to the site laboratory, sample preparation area and drill locations are undertaken at least annually. These inspections have not identified any material risks.
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). • The mineral rights of PE 528 is held by La Générale des Carrierés et des Mines (Gécamines), the DRC state-owned copper mining company. Anvil mining, via its subsidiary MMG Kinsevere SARC has a Contrat d'Amodiation (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. • A royalty of 2.5% of gross revenue was 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. • There are no known impediments to operating in the area.
Exploration done by other parties	**Summary of Previous Exploration Work by Gecamines and EXACO**

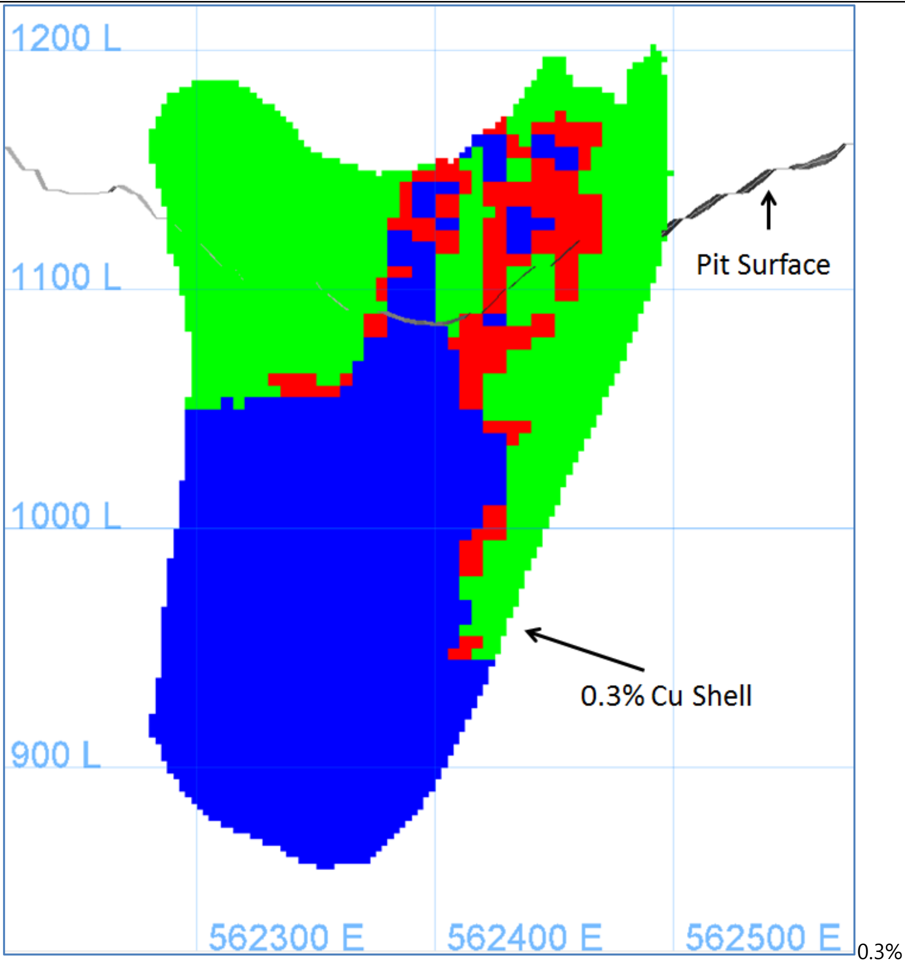
Deposit	Pitting	Trenching		Drilling	
	No. (m depth)	No. (metres)	Significant grades	No. holes (metres)	Significant grades
Tshifufiamashi	11	16 (1,304m)	5.8% Cu 0.2% Co over 50m	37 (846m)	10.5% Cu 0.72% Co over 22.2m
Tshifufia Central	-	17 (1,106m)	7.6% Cu 0.3% Co over 15m	19 (950m)	6.3% Cu 0.6% Co over 23m
Tshifufia South	-	39 (278m)	7.2% 0.3% Co over 40m	11 (497m)	
Kinsevere Hill	7 (44m max)	11 (625m)	6.6% Cu 0.2% Co over 20m	10 (1,021m)	4.0% Cu 0.22% Co over 14.6m

• In 2004 Anvil Mining carried out intensive exploration drilling to

Assessment Criteria	Commentary
	define the deposits in Kinsevere. • In 2012 MMG continued exploration aimed at identifying additional mineralisation beyond the Anvil Mining Mineral Resources. • In 2013/2014 MMG Exploration have been conducting works around the Mine Lease within a 50km radius of the known deposit to explore for additional high-grade oxide material. • In 2015 MMG conducted a Scoping Study on the potential to process the copper sulphide ore at Kinsevere located beneath the current oxide Ore Reserves. As part of this study, 5 DD holes were drilled in early 2015. In August 2015, DD drilling re-commenced as part of a follow on Pre-Feasibility Study to increase confidence in the copper sulphide Mineral Resources. This drilling was completed at the end of 2016 and included in the 2017 Mineral Resource estimates. • Drilling commenced in May 2017 to inform the Sulphide Feasibility Study.
Geology	• The Kinsevere Copper deposit is a sedimentary hosted copper deposit. The deposit is hosted in moderately to steeply dipping Neoproterozoic sedimentary formation of the Roan group of the Katanga stratigraphy in the Mine Series subgroup of Katangan African Copper belt. • 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 N-S oriented fracture and separated by a sinistral strike-slip fault, while the third fragment, called Kinsevere Hill, is situated along major NW-SE fracture and separated from the other fragments by another sinistral strike-slip fault. All these fragments are affected by fractures and breccias. • The sulphide, transitional and oxide mineralisation in the Kinsevere copper deposit are either disseminated in recrystallised layers or infilling bedding plans, reactivated bedding, fractures and joints. Sulphide mineralisation includes chalcopyrite, bornite, chalcocite and pyrite. Oxide mineralisation is dominated by malachite with lesser chrysocolla. A transitional zone exists between the primary and oxide zones with both a horizontal trend, controlled by ground water movements, and a sub vertical trend controlled by bedding and structures. Transitional copper species include chalcocite, cuprite, covellite and native copper. This zone is known as the TMO (transitional/mixed ore) zone.

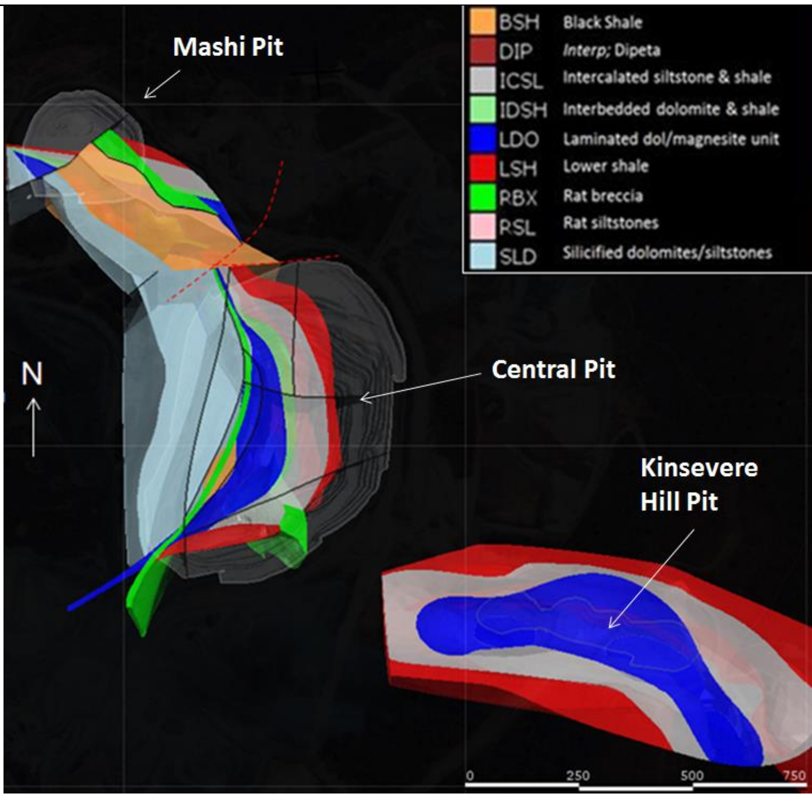
Assessment Criteria	Commentary
	Kinsevere Mine Series Stratigraphy 
Drillhole information	• Within the database used, there are 876 exploration drillholes (285 DD, 31 RC with DD tail and 560 RC) and 6,309 grade control drillholes (all RC). • No individual drillhole is material to the Mineral Resource estimates and hence this geological database is not supplied.
Data aggregation methods	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section. • No metal equivalents were used in the Mineral Resources 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° angles in order to maximise true width intersections. • Geometry of mineralisation is interpreted as sub- vertical to vertical and as such current drilling allows true width of mineralisation to be determined.

Assessment Criteria	Commentary
Diagrams	 **Plan view of the Kinsevere deposit**

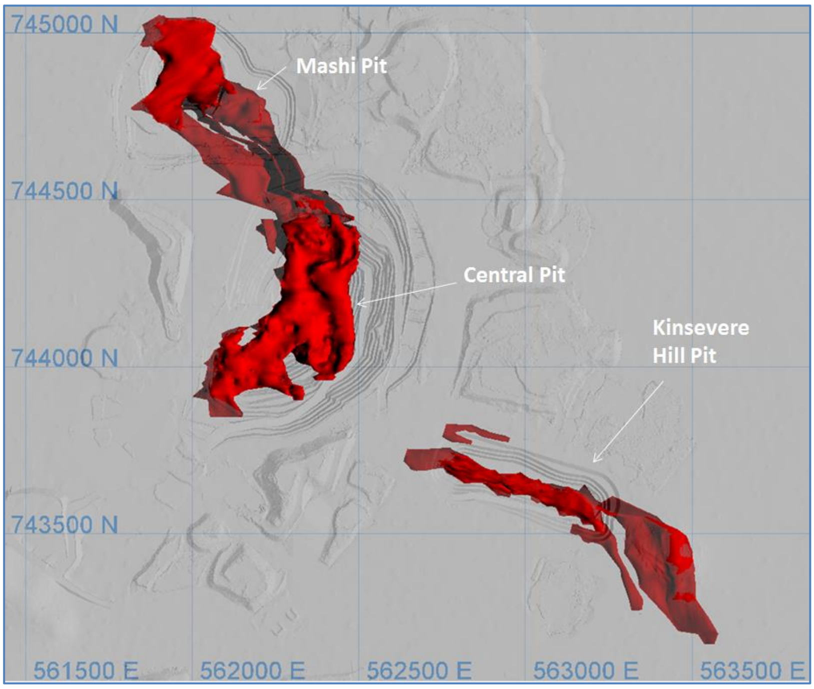
Assessment Criteria	Commentary
	 Cu Shell coded by Oxidation state (Green – Oxide / Red – TMO / Blue – Primary) **Typical cross section through Tshifufia (Central) pit showing mineralisation zones**
Balanced reporting	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	• The exploration focus will be within the Mine Lease and within a 50km radius of the known deposit to explore for additional high-grade oxide material. • Drilling as part of a Sulphide Feasibility study was ongoing during the completion of the 2017 Mineral Resource estimates. It is due for completion in late 2017.

Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	• The following measures are in place to ensure database integrity: ○ The complete drillhole database (RC grade control and DD) data is stored in a SQL databases using the Geobank software front end management systems: ○ The grade control data (RC) is managed by the onsite Geology team and the Group Technical Services database team in Melbourne. ○ The exploration/resource (DD) data is managed by the Group Technical Services database team in Melbourne. ○ Data is collected in Excel templates and imported into the database. Import routines check for data consistency and errors before the import is successful, thus maintaining data integrity. ○ The databases offer secure storage and consistent data which is exposed to validation processes, standard logging and data recording codes. • The measures described above ensure that transcription or data entry errors are minimised. • Data validation procedures include: ○ Internal database validation systems and checks. ○ Visual checks of exported drillholes in section and plan view, checking for accuracy of collar location against topography, and downhole trace de-surveying. ○ External checks in Vulcan software prior to the data used for Mineral Resources. Checks on statistics, such as negative and unrealistic assay values. • Any data errors were communicated to the Database team to be fixed in Geobank. Data used in the Mineral Resources has passed a number of validation checks both visual and software related prior to use in the Mineral Resources.
Site visits	• The Competent Person visited on two occasions during 2016/17 (Nov 16 and May 17). Site visit work included: ○ Visits to the ROM stockpiles, open pit mine, core yard, sample preparation and on-site assay laboratory. ○ Discussions with geologists (mine and exploration), mine planning engineers and metallurgists.
Geological interpretation	• The geological interpretation is based on a combination of geological logging and assay data (total copper %). There is a relatively high level of confidence in both geological and grade continuity within the upper zone of the deposit that is drilled to grade control density. There is less certainty in the geological interpretation in the lower portions of the Mineral Resources purely due to a lower drilling density however due to recent infill drilling and geological

	interpretation/knowledge the interpretation is considered reliable. • Both grade control RC and exploration DD and RC holes were used in the interpretation of the geological domains that are used in the Mineral Resources. • No alternative interpretations of the Mineral Resources have been used. However, an Indicator Kriging approach was used to construct weathering domains (within the mineralised zone). Mg high grade domains (using a 6% Mg cut-off) and Ca high grade (using a 9% Ca cut-off) domains were constructed using Leapfrog software. Lithological domains were constructed using Leapfrog software. • Wireframe solids were created for the copper mineralisation (using a 0.3% Cu cut-off). String envelopes were digitised along drill sections and were used to generate the wireframe surfaces. • Geological logging and geochemical data analysis was used to determine the lithological domains. • The magnitude of the acid soluble copper/total copper (AsCu/Cu) ratio has been used as an important criterion for the determination of the oxide, TMO and primary sulphide zones. The following ratios have been used to delineate the respective zones: o Oxide > 0.5 o Transition and mixed (TMO) between 0.3 and 0.5 o Primary < 0.3 • The resulting weathering, lithology, mineralisation domains were combined to code the drillhole data and the block model used for estimation. • On a local scale grade continuity is affected by minor stratiform lower grade zones that in places have been incorporated within the main deposit. This internal waste is better defined during the grade control drilling, and mined accordingly. Larger internal waste zones have been defined by wireframe interpretation. • Structural features (faults / fractures) provide an important control on the mineralisation and grade continuity.

<table>
<tr><td></td><td>


Plan View of Kinsevere Lithology Domains



Plan View of Kinsevere Mineralised Domains
</td></tr>
<tr><td>Dimensions</td><td>

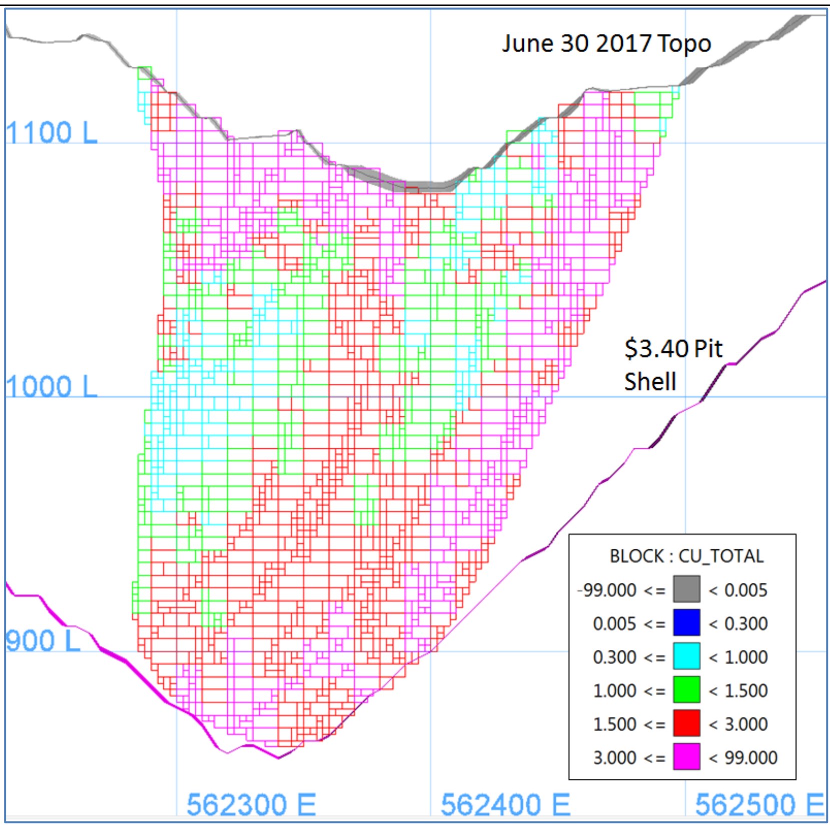
- The mineralisation strike length is approximately 1.3km for the Tshifufia and Tshifufiamashi deposits while Kinsevere Hill has a 1km strike length. The mineralisation dips sub-vertically. Mineralisation extends to 400m at depth and it can be up to 300m in width.
- The mineralisation outcrops on Kinsevere Hill, and at the Tshifufiamashi deposit.

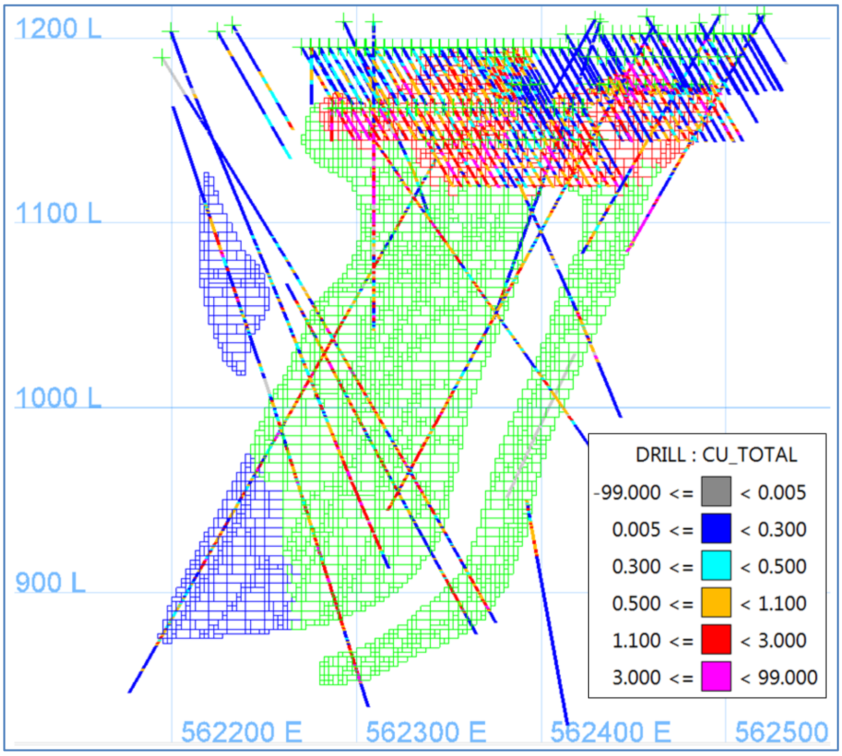
</td></tr>
</table>

Estimation and modelling techniques	- Estimation applied mostly kriging interpolation within domains as outlined further in this section, and is considered appropriate for this style of mineralisation. - Mineral Resources modelling was conducted using Vulcan software. - Variograms updated for major elements including Cu, AsCu, Ratio, Ca and Mg. Variograms from 2016 were reviewed and unchanged for other elements. - The key estimation assumptions and parameters are as follows: - Cu, AsCu, AsCu/Cu (RATIO), Co, Ca, Fe, Mg, Mn and S were estimated using Ordinary Kriging (OK). Uranium (U) was estimated by using an Inverse Distance to the power of 2 method (ID2). - Indicator Kriging (IK) was used to determine oxide, mixed and primary sulphide domains, based on the AsCu/Cu ratio. Leapfrog software was used to construct high grade domains for Ca and Mg. - RATIO was only estimated where the sample had a Cu > 0.2% and both a Cu and AsCu value were present, if criteria not met RATIO value was recorded as absent. - Extreme grade values were managed by grade capping which was performed post compositing. Values greater than the selected cut value were set to the top cut (cap) value and used in the estimation. - Wireframes and surfaces of the topography; mineralised domains, together with IK domain are used to tag the drillholes and are used for statistical analysis and grade estimation. - Grade estimation was completed using a combination of hard and soft boundaries. A soft boundary is used between the estimation domains used to direct the search anisotropy locally within the mineralised domain. Soft boundaries were used to estimate RATIO. A hard boundary is used for the high grade Ca and Mg domains, and generally between the oxide, mixed and primary domains; however depending on the geostatistical analysis, some domains were combined for the estimation of specific elements. - A series of local estimation domains were generated to honour the mineralisation strike variations thus improving the quality of the local estimate. - A composite length of 2m was used applied. Any residual intervals less than half the composite interval were appended to the previous sample interval. - No assumptions have been made about the correlation between variables. All variables are comparably informed and independently estimated. - Search parameters for Cu, AsCu, RATIO, Co, Ca, Fe, Mg estimate were derived from mineralisation and waste domain

	variography and on Quantitative Kriging Neighbourhood Analysis (QKNA). U search parameters based on a generic search of 400m x 400m x 400m, U grades higher than 250ppm were distance limited to 20m. o First estimation pass search radius uses 80% of the variogram range. Over 90% of the blocks are informed in the first pass. The second search was set to two times the variogram range. o Minimum of 5 to 14 and a maximum of 12 to 30 samples (depending on element and/or domain) for each estimate. o The search neighbourhood was also limited to a maximum of 4 to 7 samples per drillhole depending on the domain to be estimated. o Discretisation was set to 4 x 8 x 2 (X, Y, Z). o Kriging variance (KV), Kriging efficiency (KE) and Kriging regression slope (RS) of the copper estimate were calculated during the estimation. • The 2016 and 2017 in-situ Mineral Resources models has been compared and shows no material difference with metal content 2% higher for the 2017 model mainly due to drilling and subsequent re-interpretation in some areas. • The Comparison between the Mineral Resources and the mill feed grade is complicated by the operational strategy of treating high-grade ore and stockpiling lower-grade ore for later treatment. In early 2017 a stockpile adjustment occurred based on detailed survey pick-ups. Generally there was a volume and metal reduction. • Kinsevere does not produce any by-products hence no assumptions regarding the recovery of by-products are made in the estimate or cut-off and reporting. • Parent block size of the Kinsevere block model is 10m x 20m x 5m with sub-blocking down to 2.5m. Estimation was into the parent block. The size of the blocks is appropriate to the spacing of drillholes. • No further assumptions have been made regarding modelling of selective mining units. • The block model and estimate has been validated in the following ways: o Visual checks in section and plan view against the drillholes. o Grade trend plots comparing the model against the drillholes.
Moisture	• Tonnes in the model have been estimated on a dry basis.

Cut-off parameters	• The oxide Mineral Resource has been reported above an acid soluble copper cut-off grade of 0.6% and an acid soluble to total copper ratio (RATIO) greater than or equal to 0.5. This is unchanged from the 2016 Mineral Resources. • For TMO Mineral Resources type, that defines transitional and mixed copper species, has been reported above a total copper cut-off grade of 1.1% and a RATIO greater than or equal to 0.3 and less than 0.5. The cut-off grade has dropped from 2016 (1.5%) due mainly to lower assumed processing costs and higher assumed recoveries based on a potential ferric leach operation considered as part of the Sulphide PFS study. There is no Ore Reserves for TMO material. • The sulphide Mineral Resources has been reported above a total copper cut-off grade of 0.8% and a RATIO less than 0.3. This cut-off has dropped from the 2016 Mineral Resources which was 1.1% Cu due mainly to lower assumed processing costs. • The sulphide Mineral Resources (primary copper) is defined as having a RATIO less than 0.3 and this has changed from last year where less than 0.2 was used to define sulphides. The change is based on the geological and metallurgical learnings from the PFS study that showed that sulphides are the dominant copper species between a RATIO of 0.2 and 0.3 and most suitable to flotation concentrating. There is no Ore Reserves for the primary copper. • The reported Mineral Resources have also been constrained within a US$3.40/lb pit shell. Both the sulphide and TMO cut-offs have reduced this year due to a combination of lower assumed processing costs and increased recovery based on a potential ferric leach operation and all cut-offs were assessed against a block by block Net Value calculation to confirm suitability. The reporting cut-off grade and the pit-shell price assumption are in line with MMG's policy on reporting of Mineral Resources which is prospective for future economic extraction.

	 **Cross-section of Copper Mineral Resources model contained within the US$3.40lb pit shell**
Mining factors or assumptions	• Mining of the Kinsevere deposits is undertaken by the open pit method, which is expected to continue throughout the life of mine. • Mining selection has been considered in the calculation of cut-off grade parameters and in the constraint of Mineral Resources within the US$3.40/lb pit shell. • No mining factors have been applied to the Mineral Resources.
Metallurgical factors or assumptions	• The metallurgical process applied at the Kinsevere Operation applies acid leaching coupled with solvent extraction electro-winning (SXEW) technology to produce copper cathode plates for sale. • Mineral Resources cut-off grade reporting criteria is based on potential future economic extraction influenced by a ferric leach and flotation operation based on findings on a PFS study. • Consideration of metallurgy has been included in the cut-off grade calculation, material type and in the construction of the US$3.40/lb pit shell. • No metallurgical factors have been applied to the Mineral Resources.
Environmental factors or assumptions	• Environmental factors are considered in the Kinsevere life of asset work, which is updated annually and includes provision for mine closure. • The property is not subject to any environmental liabilities. • Potentially Acid Forming (PAF) and Non-Acid Forming (NAF) criteria is controlled by the acid neutralising capabilities of the dolomitic CMN

	unit and the potential acid forming potential of the shale rich SD which is known to contain pyrite where a sulphur cut-off is utilised.
Bulk density	• In-situ dry bulk density values are determined from 6,839 diamond core density measurements, 4 in-pit bulk sample measurements and 12 in-pit measurements from specific lithologies. • Bulk sample and in-pit measurements account for void spaces. • Bulk density was calculated using the wet and dry method: *Bulk Density = Dry Sample Weight/(Dry Sample Weight – Wet Sample Weight)* • Average in-situ bulk density values were assigned to the blocks within each domain. • There have only been minor immaterial changes from the values assigned for the 2016 Mineral Resource estimates.
Classification	• Wireframes used for Mineral Resources classification are based on a combination of confidence in assayed grade, geological continuity and Kriging metrics (Kriging variance, efficiency and slope of regression). • In general, drilling within Measured Mineral Resource regions is spaced 15m x 15m, Indicated is 35m x 35m and Inferred Mineral Resource ranges up to 80m x 80m. • The Mineral Resources classification reflects the Competent Persons view on the confidence and uncertainty of the Kinsevere Mineral Resources.  **Cross section showing Kinsevere Mineral Resources classification and drilling density (Red-Measured, Green-Indicated, Blue-Inferred)**

Audits or reviews	• An external Mineral Resources audit was conducted by Jeremy Witley from MSA (The MSA Group) in June 2014. Overall the review stated that the estimate has been conducted in a competent and professional manner. Recommendations were incorporated into the 2015 Mineral Resources. • An independent audit of the Mineral Resources model was completed in October 2016, by Pennywise Pty Ltd as part of an audit on the results of the Sulphide Scoping Study and was commissioned by MMG Limited. No material errors were found. • MMG conducts annual internal reviews of Mineral Resource estimates. No significant issues have been identified.
Discussion of relative accuracy / confidence	• Close-spaced grade control drilling within the Measured Mineral Resources areas provides suitable estimation on a local scale and supports the requirements of mining selectivity for the Kinsevere operation. • Estimates in the deeper primary copper mineralisation will not be as locally accurate, due to wider spaced drilling however the geological and grade interpretations are robust due to a high understanding of geological controls. The level of uncertainty is captured by the Indicated Inferred Mineral Resources category. • Estimation of dry bulk density values needs to be evaluated to determine if achievable with the current number of samples. The method of assigning bulk density values is similar to the 2016 Mineral Resources and is not considered to have any material impact on the reported tonnages.

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

This Mineral Resources 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, Douglas Corley, 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 Geoscientists
- 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 Ltd. at the time of the 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 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 the Kinsevere 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 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Douglas Corley MAIG R.P.Geo. (#1505)

18/10/2017

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Signature of Witness:

Anna Lewin
Melbourne, VIC

Witness Name and Residence: (e.g. town/suburb)

4.3 Ore Reserves - Kinsevere

4.3.1 Results

The 2017 Kinsevere Ore Reserves is based on the 2017 Mineral Resources model.

The 2017 Kinsevere Ore Reserves are summarised in Table 9.

Table 9 2017 Kinsevere Ore Reserves tonnage and grade (as at 30 June 2017)

Kinsevere Ore Reserves				Contained Metal	
Central Pit	**Tonnes (Mt)**	**Copper (% Cu)**	**Copper (%ASCu)**	**Copper ('000 t)**	**Copper AS[1] (' 000)**
Proved	1.0	4.2	3.6	44	38
Probable	2.3	4.5	3.9	103	89
Central Pit Total	3.3	4.4	3.8	148	126
Mashi Pit					
Proved	1.4	4.2	3.7	57	51
Probable	2.7	3.0	2.6	80	70
Mashi Pit Total	4.1	3.4	3.0	138	120
Kinsevere Hill					
Proved	0.16	8.1	7.2	13	12
Probable	3.1	3.2	2.9	101	89
Kinsevere Hill Total	3.3	3.5	3.1	114	101
Stockpiles					
Probable	2.5	3.6	2.8	91	71
Stockpiles Total	2.5	3.6	2.8	91	71
Kinsevere Total	13.2	3.7	3.2	490	418

[1] AS= Acid Soluble
Cut-off grade were calculated at a US$2.96/lb copper price and are based on a Net Value Script taking into account gangue acid consumption. The cut-off grade approximates 0.9% ASCu under current operating conditions and 0.9% ASCu at the cessation of mining activities.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Contained metal does not imply recoverable metal.

The main difference from the 2016 Ore Reserves is:

(i) An updated Mineral Resources model, including additional drilling, has seen a refinement of the interpretation for the Mashi and Kinsevere Hill Mineral Resources. This resulted in an increase in both total and acid soluble contained copper metal.
(ii) Mine and stockpile depletion.
(iii) Exclusion of 2.3Mt of marginal stockpile.

4.3.2 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 10 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code. 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 10 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Kinsevere Ore Reserves 2017

Assessment Criteria	Commentary
Mineral Resource estimates for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Ore Reserves. • The Ore Reserves include ore on stockpiles. • The sub-celled Mineral Resources block model named "est_2017_v4.bmf.dm" and dated 30-06-2017 was used for optimisation purposes. • Mineral Resources block model based on Ordinary Kriging interpolation has been applied for the estimation of all elements. It has a parent block size of 10m x 20m x 5m with sub blocking down to 2.5m.
Site visits	• The Competent Person visited the site in July and October, 2016 and in February and June, 2017. • Each visit consisted of discussions with relevant people associated with Ore Reserves modifying factors including geology, grade control, geotechnical parameters, mine planning and mining operations, metallurgy, tailings and waste storage, and environmental disciplines. The outcomes from the visits have included achieving a common understanding, review of assumptions, calculation of cut-off grade and developing the Life-of-Asset mine plan. • A site visit was also carried out by Group Manager Technical Services- in June 2017. • The 2015 Competent Person Ore Reserves undertook a site visit in February 2017.
Study status	• The current mine and processing plant configuration has been in operation since September 2011. Ore Reserves are based on actual historical performance data and projected estimates for cost and performance. • Life-of-Asset Low Case was produced as part of the MMG planning cycle. This Low Case informs the Ore Reserves – it demonstrates it is technically achievable, economically viable, while considering material Modifying Factors.

Cut-off parameters	• Breakeven cut-off grades were calculated at a US$2.96/lb copper price and acid soluble to total copper ratio greater than or equal to 0.5. At a fixed gangue acid consumption of 16.5kg/t, the following approximate COG's are applied:
	    o 0.9% ASCu under current operating conditions.
	    o 0.9% ASCu under post mining conditions.
	• The COG estimates are based on a Net Value Script (NVS) calculation

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>
that takes into account price assumptions, gangue acid consumption calculation and costs associated with current operating conditions.

- The NVS is run over the Mineral Resources Model to identify material that is suitable for processing. This material must be economic during processing to be defined as Ore Reserves.
- For the cost assumptions please see the “Costs” section.
- For the price assumptions please see the “Revenue factors” section.
</td></tr>
<tr><td>Mining factors or assumptions</td><td>

- The method of Ore Reserves estimation included pit optimisation, final pit and phase designs, consideration of mine and mill schedule and all modifying factors.
- Kinsevere mine is an open pit operation that is mining and processing oxide copper ore. The operation uses a contract mining fleet of excavators and both fixed and articulated dump trucks along with a fleet of ancillary equipment.
- This mining method is appropriate for the style and size of the mineralisation.
- The slope geotechnical guidelines used are as follows:

<table>
<tr><th>Design Sector</th><th>Weathering Code</th><th>Overall Wall Angle (degrees)</th><th>Interamp Angle (degrees)</th><th>Bench Face angle (degrees)</th><th>Bench height (metres)</th><th>Bench width (metres)</th></tr>
<tr><td rowspan="4">West</td><td>4</td><td>-</td><td>35</td><td>50</td><td>10</td><td>6</td></tr>
<tr><td>3</td><td><u>no greater</u> than 27 degrees</td><td>-</td><td>45</td><td>10</td><td><u>minimum of 6m</u> - adjust bench width to meet Overall WALL angle criteria</td></tr>
<tr><td>2</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>1</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td rowspan="4">Northeast</td><td>4</td><td>-</td><td>35</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>3</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>2</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>1</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td rowspan="4">East/Southeast /South/ Northwest</td><td>4</td><td>-</td><td>35</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>3</td><td>-</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td>2</td><td>-</td><td>43</td><td>65</td><td>10</td><td>6</td></tr>
<tr><td>1</td><td>-</td><td>46</td><td>70</td><td>10</td><td>6</td></tr>
</table>

  - These guidelines take into account the potential failure mode/s at bench, interamp and overall wall scale.
  - Limit equilibrium analysis was conducted on the final slope designs to ensure that the Factors of Safety were >1.2.
- The pit optimisation was based on the 2017 Mineral Resources block model and the strategy for the final pit selection was based on a revenue factor 1 (RF=1.0). RF 1.0 pit shell is used across all assets in the MMG Group. Final pit designs incorporating further practical mining considerations were carried out using these optimisation shells.
- Assumed mining dilution is 5% and mining recovery is 95%. Dilution
</td></tr>
</table>

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>modelling and reconciliation data supports these assumptions.
• Minimum mining width (bench size) is ~40m.
• No Inferred Mineral Resource material has been included in optimisation and/or Ore Reserves reporting.
• All required infrastructure is in place. There is a planned increase in mining rate in 2018 that is considered to be within the capacity of the site mining contractors.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>• Kinsevere is an operating mine. The metallurgical process is a hydrometallurgical process involving grinding, tank leaching, counter-current decantation (CCD) washing, solvent extraction and electrowinning.
• The 2007 Pre-feasibility Study (PFS) tested composite and variability samples from Central, Mashi & Kinsevere Hill. The test work found:
  ○ Sulphuric acid leach tests on the master composites indicated that acid soluble copper recoveries were high (>95 % at pH 1.5 212µm for 8 hour leach duration);
  ○ Acid soluble copper leach kinetics were fast with leaching virtually complete after 4 hours;
  ○ Moderate gangue acid consumption with all samples below 21kg H2SO4/tonne; and
  ○ Assay analysis of 12 hour leach solutions did not detect significant levels of any deleterious elements.
• The acid leach process has been operating successfully since start-up in September 2011.
• Copper recovery is determined by the equation:
Cu recovery (%) = (0.9513*ASCu)/Cu
where ASCu refers to the acid soluble copper content of the ore which is determined according to a standard test. The ASCu value is typically about 80% of the total copper value though the exact percentage varies with the ore type. Much of the non-acid soluble copper is present in sulphides which are not effectively leached in the tank leaching stage. Hence, the aforementioned ratio (ASCu/Cu) criteria of >=0.5 has been applied, to classify ore.
• The reconciliation between expected and actual recovery is checked each month. The following table summarizes the outcomes for the last four quarterly periods.
<table>
<tr><th rowspan="2">Period</th><th colspan="2">Recovery of Acid Soluble Copper (%)</th></tr>
<tr><th>Predicted</th><th>Actual</th></tr>
<tr><td>Q3 2016</td><td>95.1</td><td>93.4</td></tr>
<tr><td>Q4 2016</td><td>95.1</td><td>95.9</td></tr>
<tr><td>Q1 2017</td><td>95.1</td><td>94.6</td></tr>
<tr><td>Q2 2017</td><td>95.1</td><td>95.4</td></tr>
</table></td></tr>
</table>

Assessment Criteria	Commentary
	• The same equation for copper recovery has been applied for Mashi and Kinsevere Hill oxide ores. Ore from both of these pits has been processed in the past 12 months with no observed change in copper recovery. Similar leach performance was found in the 2007 PFS test work, and geological drilling and interpretation has not recognised differences in oxide copper mineralisation to that in Central. • The main deleterious components of the ore are carbonaceous (black) shales which increase solution losses in the washing circuit and dolomite which increases acid consumption in the leaching process. • The effect of black shale is currently controlled by blending which is used to limit the percentage of this component in the feed to less than 25%. • Total gangue acid consumption for Central and Kinsevere Hill South ore has been estimated based on the historic average consumption rates of 16.5kg/t of oxide ore. • Kinsevere Hill North and Mashi contains higher levels of dolomite, and hence results in a higher acid consumption than Central ore. For this reason, based on laboratory testwork, the gangue acid consumption (kg/t ore) for Kinsevere Hill North and Mashi, where Ca >=1%, is calculated as 43.8*Ca – 27.3. For ore containing <1% Ca, the gangue acid consumption is assumed to be equal to the historical average of 16.5kg/t. • To prevent process issues, ore feed to the mill is blended to ensure the gangue acid consumption does not exceed 35kg/t. • For Ore Reserves, a processing capacity of 2.3Mtpa of ore and an electrowinning capacity of 80ktpa of copper cathode has been assumed. Both mill throughput and cathode production rates have been demonstrated as sustainable. • Kinsevere mine does not produce any by-products.
Environmental	• The DRC Mining Regulations require an operating site to have an approved Environmental and Social Impact Assessment (ESIA) and a supporting Environmental Management Plan of the Project (EMPP). A site's operations are limited to those that are described in the ESIA. The ESIA approval process requires community and public consultation. An ESIA and associated EMPP are valid for five years. • An update to the 2012 ESIA and 2015 Addendum was submitted to the regulator before 31 August 2017. It covers the current oxide operation, including existing practices and activities undertaken as part of operational ramp up in 2017 (including extension to the black shale waste rock dump, changes to topsoil stockpile locations, construction of a new truck maintenance workshop, bush clearance for non-acid forming (NAF) waste dump extension works, and an improved surface water management strategy to manage water discharges). Some aspects of the Kinsevere Primary Copper Project are included within the 2017 ESIA Update.

Assessment Criteria	Commentary
	• The two yearly external audit against Article 459, DRC Mining Regulations has been completed and submitted to the regulator. • Geochemical properties of the waste rock and black shales have been characterised through preliminary studies, with more detailed studies ongoing. Field observations indicate that sulphides present in the black shale stockpiles are oxidising and generate heat. A mineral waste management plan outlines the operational controls, considering field observations, and results of studies. • Potential for waste rock acid generation is currently estimated based on the sulphur grade. Waste is separated based on its potential for acid generation, with Potentially Acid Forming (PAF) material being encapsulated in PAF cells within waste rock dumps. Non-acid forming (NAF) materials are used for encapsulation or to construct landforms. • Existing tailings storage facility has capacity to meet current Ore Reserves requirements.

Infrastructure	• The Kinsevere mine site is well established with the following infrastructure in place:
	  ○ The plant is operational.
	  ○ Labour is mostly sourced from Lubumbashi and surrounding villages with some expatriate support. There is an existing accommodation facility onsite.
	  ○ There is sufficient water for the processing.
	  ○ Copper cathode is transported off-site by truck.
	  ○ Site has an access road that is partially sealed.
	  ○ There is power supply from the national grid and from onsite generators.
	  ○ The Ore Reserves do not require any additional land for expansion.
	  ○ Tailings Storage Facility in place and future lifts are accounted for.
	• Grid power in country can be intermittent; mitigation management is through diesel based power generation. Future grid power availability is forecast to improve.
	• Timely dewatering of the mining areas continues to be an important aspect of mining operations.

Costs	• Kinsevere is an operating mine and has historical costs that have been used to inform the 2017 Kinsevere Budget (January 2017 to December 2017), with the exception of the contract mining costs. • Mining costs are based on contract mining costs. • The Ore Reserves estimation has been based on these costs in accordance with JORC 2012 code. • Transportation charges used in the evaluation are based on the actual

Assessment Criteria	Commentary
	invoice costs that MMG are charged by the commodity trading company per the agreement. • The processing costs include calculated gangue acid consumption. • The final product contains no deleterious elements. • US dollars have been used thus no exchange rates have been applied. • Weathering profiles have been updated to model in-pit blasting to inform the mining costs. • Since the final product is copper cathode (Grade A non LME registered) there are no additional treatment or refining or any other similar charges. • Sustaining capital costs have been included in the pit optimisation. The sustaining capital costs are principally related to the tailings storage facility lift construction and the process plant. The inclusion or exclusion of these costs in the Ore Reserves estimation process has been done in accordance with the JORC 2012 code. • Allowances have been made for royalties. • A cash flow model was produced based on the detailed mine and processing schedule. This model includes the aforementioned costs as well as all sustaining capital that is needed to realise the Ore Reserves.
Revenue factors	• For cost assumptions see section above – "Costs" • The assumed copper price is US$2.96/lb, which is the same as that reported in the cut-off parameters section. These prices are provided by MMG corporate, approved by the MMG Board, and are based on external company broker consensus and internal MMG strategy.
Market assessment	• MMG considers that the outlook for the copper price over the medium and longer term is positive, supported by further modest demand growth which is expected to exceed increases in supply. • Global copper consumption growth will continue to be underpinned by rising consumption in China and the developing countries in Asia as these nations invest in infrastructure such as power grids, commercial and residential property, motor vehicles and transportation networks and consumer appliances such as air conditioners. • Supply growth is expected to be constrained by a lack of new mine projects ready for development and the requirement for significant investment to maintain existing production levels at some operations. • There is a life of mine off-take agreement with a trading company in place for all of Kinsevere's copper cathode production. The off-take arrangement has been in place since the commencement of cathode production at site and has operated effectively. There is no reason to expect any change to this in future.
Economic	• The costs are based on historical actuals, the 2017 Kinsevere Budget and current contractor mining costs. • Revenues are based on historical and contracted realised costs and

Assessment Criteria	Commentary
	copper price as reported in the cut-off parameters section. • The Ore Reserves financial model demonstrates the mine has a substantially positive NPV. • The discount rate is in line with MMG's corporate economic assumptions and is considered to be appropriate for the location, type and style of operation. • Standard sensitivity analyses were undertaken for the Ore Reserves work and support the robustness of the Ore Reserves estimate.
Social	• Kinsevere provides a range of social development support to the communities surrounding the operation, with a specific focus on United Nations Sustainable Development Goals 1-6. • The centrepiece of the social investment program is the Ubuntu Farm, a multipurpose large scale farm designed to support food and economic security while also providing a centre of agricultural excellence and research. • Household surveys have been undertaken in 2017 demonstrating that Kinsevere is contributing positively to socio-economic outcomes in the region. This survey also showed that in-migration to the region from other communities continues, and that as people become progressively wealthier they chose to relocate from Kilongo or other near mine communities to Lubumbashi, where health, education and other infrastructure is more readily available. • Community engagement was undertaken as part of the Social Impact Assessment process. During the course of these discussions it became clear that while Kinsevere has provided education and training and employment opportunities to near mine communities that there are ongoing expectations that the level of local participation will increase should any expansion or extension activity be undertaken. • Local villagers and artisanal miners have increased their levels of activity on and around the Kinsevere lease. In response to this, actions have been undertaken with the support of Government to progressively relocate people living and farming on the Kinsevere lease and to fully compensate them for their assets. Officials have also been engaged in the removal of artisanal miners from the region and site has improved security management in response to any ongoing risk of incursion.
Other	• MMG has a Contrat d'Amodiation (Lease Agreement) with Gécamines to mine and process ore from the Kinsevere Project until 2024. A renewal application process for 15 year extension shall be submitted to DRC regulatory at least 1 year and no more than 5 years before the expiry date. • The PE 528 permit covers the three major deposits of Tshifufiamashi, Tshifufia and Kinsevere Hill/Kilongo. • A Contract d'Amodiation is provided for under the DRC Mining Code, enacted by law No 007/2002 of July 11, 2002.

Assessment Criteria	Commentary
Classification	• The Ore Reserves classification is based on the JORC 2012 requirements. The basis for the classification was the Mineral Resources classification and cut-off grade. The ex-pit material classified as Measured and Indicated Mineral Resources and is above 0.9% ASCu with an acid soluble to total copper ratio greater than or equal to 0.5, and is demonstrated to be economic to process, is classified as Proved and Probable Ore Reserves respectively. • Stockpile material at Kinsevere is classified as Indicated. Indicated Mineral Resources above 0.9% ASCu with an acid soluble to total copper ratio greater than or equal to 0.5, and is demonstrated to be economic to process, is classified as Probable Ore Reserves. • The Ore Reserves do not include any Inferred Mineral Resources.
Audit or Reviews	• An external Ore Reserves audit was completed in 2013. The work was carried out by SRK Consultants African branch. Even though some minor improvements have been suggested, no major issues were identified. The audit continues to be valid considering no significant changes in Ore Reserves estimation methodology has occurred between the 2014 to 2017 Ore Reserves estimate. • The next external Ore Reserves audit is planned for completion in 2018 on the 2017 Ore Reserves.
Discussion of relative accuracy/ confidence	• The most significant factors affecting confidence in the Ore Reserves are: o Reliability of the grid power supply. o Any updates to the estimates of gangue acid consumption. o Increase in operating costs for mining and processing. o Geotechnical risk related to slope stability. o Effective management of surface water.

4.3.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 11.

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 11 Contributing experts – Kinsevere Mine Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Doug Corley, Principal Resource Geologist MMG Ltd (Melbourne)	Mineral Resources model Stockpile Tonnes and Grade
Nigel Thiel, Principal Metallurgist, MMG Ltd (Melbourne)	Metallurgy
Christian Holland, Principal Geotechnical Engineering, MMG Ltd (Melbourne)	Geotechnical parameters
Dean Basile, Principal Mining Engineer, Mining One Consultants (Melbourne)	Whittle optimisation, mining costs, pit designs, mine and mill schedules, review of Ore Reserves estimate
Kinsevere Geology department	Production reconciliation
David Machin, Principal Water and Tailings Engineer, MMG Ltd (Melbourne)	Tailings Capacity
Knight Piésold	Tailings dam design
Claire Beresford, Senior Analyst, Business Evaluation, MMG Ltd (Melbourne)	Economic Assumptions and evaluation
Natalie Shade, Group Manager Environment, MMG Ltd (Melbourne)	Environment
Melanie Stutsel, General Manager Safety Environment and Social Performance, MMG Ltd (Melbourne)	Social
Steve Whitehead, General Manager Marketing, MMG Ltd (Melbourne)	Marketing

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

This Ore Reserve 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, Jodi Wright, 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 Pty Ltd.

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 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 the Kinsevere 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 Kinsevere Ore Reserves - I consent to the release of the 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

18/10/2017

Jodi Wright MAusIMM(CP) (#209552)

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Nigel Theil
Melbourne, VIC

Signature of Witness:

Witness Name and Residence: (e.g. town/suburb)

5 SEPON – COPPER AND GOLD OPERATIONS

5.1 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 5-1).

Figure 5-1 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. Sepon operations are located east of Savannakhet via National Route 9 then northward from Ban Nabo along National Route 28A.

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

5.2 Mineral Resources - Sepon

5.2.1 Results

The 2017 Sepon Mineral Resources are summarised in Table 12. The Sepon Mineral Resources are inclusive of the Ore Reserves. All data reported here is on a 100% asset basis. MMG's attributable interest in Sepon is 90%.

Table 12 2017 Sepon Mineral Resources tonnage and grade (as at 30 June 2017)

Sepon Mineral Resources (90%)					
				Contained Metal	
Supergene Copper[1]	Tonnes (Mt)	Copper (%)	Gold (g/t)	**Copper ('000 t)**	**Gold (Moz)**
Measured					
Indicated	5.5	4.7		259	
Inferred	1.5	3.3		50	
Total	**7.0**	**4.4**		**309**	
Copper Stockpiles					
Indicated	6.1	1.4		86	
Total	**6.1**	**1.4**		**86**	
Primary Copper[1]					
Measured					
Indicated	7.1	1.0		73	
Inferred	5.2	1.2		63	
Total	**12.2**	**1.1**		**136**	
Oxide Gold[2]					
Measured					
Indicated	1.5		3.1		0.1
Inferred	0.2		2.3		0.02
Total	**1.7**		**3.0**		**0.2**
Partial Oxide Gold[2]					
Measured					
Indicated	1.1		4.3		0.1
Inferred	0.05		3.2		0.01
Total	**1.1**		**4.3**		**0.2**
Primary Gold[2]					
Measured					
Indicated	7.1		3.9		0.9
Inferred	0.1		3.0		0.01
Total	**7.2**		**3.9**		**0.9**
Gold Stockpiles					
Indicated	0.9		1.7		0.05
Total	**0.9**		**1.7**		**0.05**
Total Contained Metal				**531**	**1.3**

1 Variable cut-off grade based on net value script accounting for costs, recoveries and metal prices within US$3.28/lb pit shells.

2 Variable cut-off grade based on net value script accounting for costs, recoveries and metal prices within US$1,400/oz pit shells.

Contained metal does not imply recoverable metal.

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

5.2.2 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 13 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 13 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Sepon Copper and Gold Mineral Resources 2017

Criteria	Commentary
Section 1 Sampling Techniques and Data	
Sampling techniques	• Reverse Circulation (RC) drilling, mostly used for grade control, was sampled at 1m (3kg-5kg) intervals for analysis. Diamond drilling (DD) was sampled as nominal 1m half-core lengths modified (up to +/-0.5 m) to geological boundaries as appropriate. • Dry RC samples pass through a rotary splitter on the bottom of the cyclone. Wet RC samples are hand-split into quadrants. Sample weights range between 1.75kg and 3.75kg. • Samples are crushed and pulverised to produce a pulp (>85% passing 85µm). • Measures taken to ensure sample representivity include orientation of the drill holes as close as practical to perpendicular to the known mineralised structure, and collection, and analysis of field duplicates. • In the view of the Competent Person, sampling is of a reasonable quality sufficient to estimate the Mineral Resources.
Drilling techniques	• RC and DD are used in the Sepon Mineral Resource estimates. There are two types of RC referred to in this document. Prior to 2006 RC drilling was used to provide information for resource estimation, however due to ongoing issues with wet samples a change was made to DD core for resource work. RC has also been used for grade control purposes for the life of mine. • Previously the grade control RC drilling has not been used for resource modelling. Since the 2016 reporting period grade control data has been used in the Khanong Copper Mineral Resource estimate only. • All DD drilling used triple tube core barrels, core diameter is largely HQ3 but PQ3 is also common in the clayey near surface zones. • All competent DD core was orientated.
Drill sample recovery	• DD recovery averages 95% (based on length) and RC recovery averages 80% (based on mass). • DD sample recoveries were recorded as the length of core recovered per metre of drilling and are stored in the database. RC sample recoveries are recorded as sample weight in the database and a recovery calculated based on expected weight given a particular volume and density. • The drilling process is controlled by the drill crew and geological supervision provides a means for maximising sample recovery and ensures suitable core presentation. No other measures are taken to maximise core recovery. • Preferential loss/gains of fine or coarse materials are not considered significant. • Sample recovery is better and generally of lower grade in competent primary rock than in the softer and higher grade transitional and oxide material as expected. • Recovery loss at Sepon is not deemed material in the estimation process in the view

Criteria	Commentary
	of the competent person. • In places, RC holes have been twinned with DD to determine if any bias is detected regarding recovery between ore and waste, to date it has not been deemed materially biased.
Logging	• All RC and DD core was logged on paper log sheets and entered manually into the Sepon database until 2015. • Post 2015 logging is carried out on tablets and automatically linked to the database. Several quality control (QC) triggers are set in the database to validate data as entered. • All Sepon RC and DD drill core has been geologically and geotechnically logged to support mineral resource estimation, mining and metallurgy studies. • Geological logging is qualitative, using a set of pre-determined Sepon tables for; lithology, structure, mineralisation, geotech, oxidation, alteration and a site developed metcode (metallurgical domain). • Core is photographed and stored digitally. • Most drill cores are stored at the Sepon core shed. Rationalisation has occurred since 2015 where core that cannot be identified has been removed. • A total of 1,315,902m of DD and RC drilling data is contained in the resource database, of this 84% is geologically logged, and 98% of sampled intervals contain gold and copper assays (15% of the RC data was assayed but not logged). • Only samples within the mineralisation or geological domains are used in the estimations. There is also some drilling within block model extents not used in estimations as they fall outside of the areas of interest, or have suspect assay results. • A total of 29,277 grade control RC holes for 505,079m and 1,956 resource drill holes for 152,043m were used to inform the 2017 Khanong block model. • A total of 7,436 DD and RC drill holes for 567,464m, were used to inform the Thengkham region block model.
Sub-sampling techniques and sample preparation	• DD core was orientated along the apical trace of the reference plane. Half-core samples were taken using a diamond core saw for competent core or sampled by hand using a spatula or blade for clay-rich or rubbly material. • RC (pre 2006 resource) samples were collected from a cyclone and, if dry, put through a three stage riffle splitter for a 12.5% sub-sample. A 3kg-5kg (1m) sample was collected into pre-numbered sample bags for analysis. Before 2006, if RC samples were wet, then sampling was by quartering. After 2006 wet RC samples were no longer taken. • Grade Control RC drilling is sub-sampled using a cyclone splitter if dry and if wet is split into 4 quadrants and one quadrant is sent for analysis. • The RC and DD sample preparation techniques are considered appropriate as sample preparation techniques. • All samples contain a waterproof sample ID tag in numbered calico bags and are weighed. The samples were stacked and wrapped on a pallet before being transported by truck to the laboratory.

Criteria	Commentary

	• Resource sample processing and gold fire assaying takes place at the ALS laboratory Vientiane. A small proportion of the data has been assayed at the site assay laboratory using similar sample methods as part of a trial to determine if the site assay laboratory can accommodate all resource drilling internally.
	• Grade control sample preparation and analysis is carried out by the internal site laboratory.
	• Upon laboratory receipt of samples they were sorted, barcode tagged for tracking and then weighed. The samples were oven dried at 110°C (for core samples, minimum of 12 hours drying or longer until the sample has completely dried, allowing it to pass through a crusher without pelleting). The entire sample was crushed in a Jaw Crusher with 70% passing 2mm. The sample was rotary split to 3kg if required, then pulverised using an LM5 to 85% passing 85µm.
	• Representivity of samples was checked by:
	  o Sizing analysis.
	  o Duplication at the crush stage.
	• Measures taken to ensure sampling is representative of the insitu material collected include:
	  o Field duplicates were taken as an additional 12.5% split every 15m for RC drilling. DD field duplicate samples were taken as quarter core every 15m (but at times have been sampled every 20m). Duplicate samples were collected and analysed at both the coarse crush split and the pulverised split stages.
	  o Replication of the duplicates is considered satisfactory.
	• Sample size is considered appropriate for the disseminated gold and copper grain size for both the RC and DD samples.
	• The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Sepon mineralisation (sediment hosted base metal) by the Competent Person.

Quality of assay data and laboratory tests	• Following sample preparation a 110g pulp aliquot for Au Fire Assay and 20g pulp aliquot for ICP multi element was taken. The 20g pulp aliquots were transported to other ALS laboratories (usually ALS Brisbane) for ICP and Leco furnace analysis. The analytical procedure is as follows:
	  o If Au grade > 10g/t Au, re-assayed by Fire Assay Gravimetric.
	  o If Au grade > 0.4g/t Au, re-assayed using CN Leachwell technique.
	  o Detection limit for Fire Assay is 0.01ppm.
	• A multi-element suite (varying through time from 30 – 40 elements, but always including Cu, Ag, S, Mo) was analysed by ICP-AES.
	• The current copper multi-element suite analysed by ICP-AES (ALS laboratory code ME-ICP61) contains Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W and Zn.
	• Copper samples above 0.2% Cu are sent for sequential copper analysis.
	• If Cu > 0.5% Cu, the sample was re-assayed using an ore grade technique (either

Criteria	Commentary
	AAS or diluted ICP). These methods are considered total methods. • For some samples total sulphur, sulphide sulphur, sulphate sulphur, total carbon, carbonate carbon and organic carbon were analysed by Leco Furnace following appropriate digestion, this sample selection is based on a Cu trigger that has varied overtime. • No geophysical tools, spectrometers or handheld XRF instruments have been used in the analysis of samples external to the ALS laboratory for the estimation of Mineral Resources. • The quality control system adopted for each drillhole includes: o grade and matrix matched certified standard material (CRM) o coarse and pulp blanks o field duplicates (up until 2016) o pulp repeats. o Photographs of all sample bags, in order, is taken prior to shipment to the laboratory. • At a minimum, every drillhole contains at least one coarse blank, pulp blank and CRM standard. At a minimum 1 in 15 samples is a control sample (earlier programmes vary from 1 in 25 to 3 in 25). • Checks of the laboratory results and data import procedures are undertaken to identify any spurious results for verification and re-assay. Acceptable levels of accuracy and precision have been established. Any suspect data is excluded from the Mineral Resource estimate. • Independent / round robin laboratory checks were conducted on a quarterly or half yearly basis until 2010. The results were generally unbiased with respect to each other, for example, Khanong has an overall relative precision of +/- ~6%. Since 2010 no independent laboratory checks have been undertaken, this was re-instated in January 2016. • The use of an external umpire lab commenced in January 2016. To date 2,297 samples have been sent for umpire review. • In December 2015 a three month trial of the Sepon lab was carried out, the results showed that due to turnaround times, inability to complete all desired analysis and detection limits the ALS lab would continue to be used for Mineral Resource analysis.
Verification of sampling and assaying	• Verification by independent or alternative company personnel was not undertaken at the time of drilling. However, significant assay results are compared to drillhole logging and photos on a random basis. • Monthly internal reviews are carried out for all assay batches returned. Any CRM samples that exceed 3 standard deviations in a row from the expected value will have its batch returned for re-analysis in grade control and resources. Resources also fail batches where 2 results > 2 standard deviations on the same side of the expected value exist. • QAQC failures for resource drilling during the reporting period totaled 6 blanks and 29 standards, most were attributed as mis-labelled standards. The lab was notified

Criteria	Commentary
	of significant fails but no re-assays were requested in the database.

	Grade control samples	Resource Samples	
	KHN only	Sepon Lab	ALS lab
Number of samples sent to Lab for analysis	3,739	2,436	10,210
Number of QAQC samples that failed protocols.	76	NR	9
Blanks	36	274	1,070
Standard	40	137	535
Duplicates	0	152	445
Number of batches (DPOs) sent back for Reanalysis.	23	NR	3

- Resource drilling within the reporting period accounts for a total of 2% of samples within the database and any issues with the sample QAQC protocols over the past year is not considered significant to the resource estimate.
- Sample swaps, mislabeling and incorrect control sample insertion related to human error have been identified by checking procedures and photos and are rectified.
- There are no known deficiencies in the assay data quality from ALS and Sepon laboratories that affect the resource confidence. The repeats show a higher variance than would be expected, however the limitations on the sampling make representative samples difficult to achieve within the clay material. The pulp duplicates do not show large variance.
- Twinned drilling using DD has been completed for parts of the Mineral Resources to validate RC drilling. These areas are largely mined out and do not represent a large part of the insitu Mineral Resource. In general, twinning found that some RC drilling within wet conditions has resulted in smearing and positive grade bias. Current practice is only to use DD in wet drilling conditions. RC drillholes with suspected smearing have not been excluded from the mineral resource estimation.
- Laboratory result files are directly uploaded into the database with no manual data entry.
- Below detection limit assay results are stored in the database as the detection limit (negative) with appropriate metadata. No other modification of the assay results is undertaken. When exported from the database it is shown as half detection limit.
- Where data was deemed invalid or unverifiable it was excluded from the Mineral Resource estimation.

Criteria	Commentary
Location of data points	• Drillhole collars locations are located by differential GPS or total station survey instrument. Downhole surveys have been carried out using Eastman single-shot cameras or Reflex EZ tools. Surveys were taken at depths of 12m, 30m and then every 30m to the bottom of hole. • All drillhole collars are converted from UTM / India-Thai 1960 projection to SPG06 local grid coordinate systems. • In 2008, a LIDAR (Light Detection and Ranging) survey was completed providing an

Criteria	Commentary
	accurate topographic surface. Drillhole collar locations have been validated through a process of database and spatial checking for both historical and recent data and by comparing the collar locations to the LIDAR topographic surface. A number of holes were identified as having suspected locations and resolved prior to modelling of the data. • GC drillholes are marked out by the survey department. Collars are picked up by survey post drilling to account for any movement between planned and actual collar location.
Data spacing and distribution	• Drillhole spacing generally ranges from 100m to 25m for resource DD. GC drilling occurs on a 7.5m grid pattern. • The data spacing and distribution is considered sufficient due to reconciliation and variogram analysis to establish the degree of geological and grade continuity appropriate for mineral resource estimation and classification methods used at Sepon for Inferred and Indicated categories only. • It is acknowledged that closer space drilling could allow identification of further short range structures, within the 25m and would need to be undertaken if the Mineral Resource was classified to a Measured category. • DD samples are not composited prior to being sent to the laboratory. RC samples are 1m intervals but compositing up to 4m has occurred in the distant past.
Orientation of data in relation to geological structure	• Geological mapping and interpretation show that mineralisation generally strikes 060^0-090^0 (deposit dependent); hence Resource drilling is conducted on north-south sections to intersect the mineralised zone at a high angle. Most drillholes dip -60^0 to -90^0, depending on the expected dip of the target mineralisation and surface site access for drill pads. • In parts of the Thengkham (TKM) and Thengkham North (TKN) model areas, drillholes were drilled at -60^0 along 090^0 or 270^0 from 50m spaced sections in order to reduce the need for vegetation clearance and ground disturbance in areas of steep topography. • Drilling orientation is not considered to have introduced any sampling bias. This has been confirmed through variogram reviews (2015) using the varying orientation data.

Sample security	• Measures to provide sample security includes:
	  ○ Adequately trained and supervised sampling personnel.
	  ○ Core yard facility with security fence and well maintained sampling sheds.
	  ○ Cut core is sampled and stored in calico bags tied and clearly numbered in sequence.
	  ○ Calico sample bags are transported on wrapped pallets to the assay laboratory.
	  ○ The laboratory checks sample dispatch numbers against submission documents and advises of any discrepancies.
	  ○ Sample bags are photographed prior to shipment
	  ○ Assay data returned separately in both spreadsheet and PDF formats.

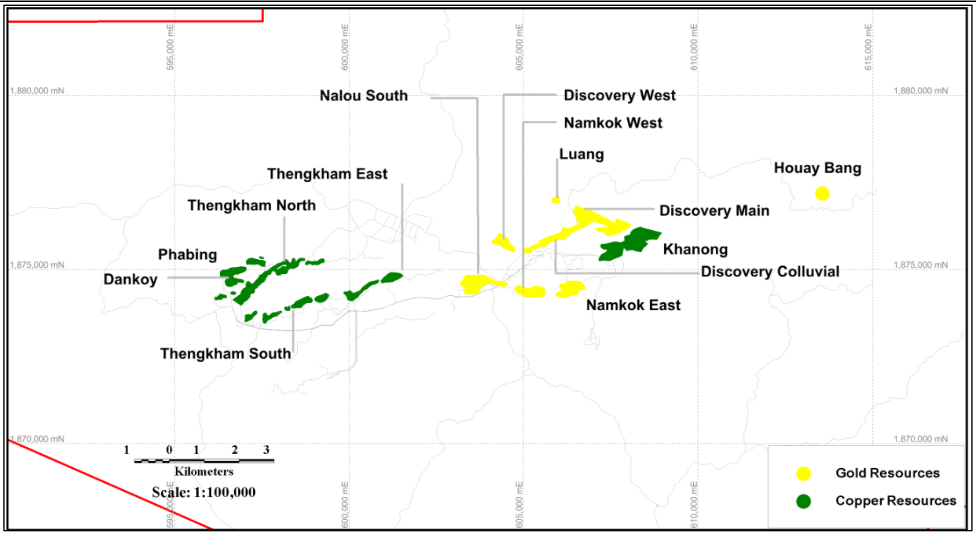
Audit and	• A trial of the Sepon Laboratory for Resource sample analysis was undertaken in

Criteria	Commentary
reviews	2015-2016. All samples >=0.2% Cu and QAQC samples were sent to ALS laboratory for comparison during the trial period. The decision was made to remain with ALS laboratory on the basis of available analytical methods, better QA/QC results and turn-around time. • REFLEX Geochemistry completed a QC review on data from 1 January 2011 – 31 May, 2014. The conclusions indicate that the control samples have provided a satisfactory guide to the accuracy and precision of the analyses. • The ALS laboratory in Vientiane has been audited on a quarterly basis by site personnel and/or Competent Person when possible. No material issues have been identified at the laboratory. The last audit was conducted in February 2017 by site personnel and not the Competent Person. • The ALS laboratory in Brisbane was audited in March 2016 by the Competent Person. No material issues were identified. • In 2008 a QC review of assay data at the Thengkham South deposit and Phabing area was undertaken (Hackman & Associates) and found that there were no obvious grade biases in the dataset, there were however quality discrepancies that required follow up. These have been addressed. • A 2008 external audit (IO Global) of the database found post-2006 analytical data to be of appropriate integrity. • In 2007 a twin drillhole study undertaken by QG comparing RC samples to DD samples, found that the use of all the available RC drilling is likely to be biased and overestimate tonnes above a gold cut-off. This was due to the presence of wet RC samples. Measures have been taken since this report to exclude wet RC samples from the estimate. Excluded drillholes are flagged in the audit table of the database and is reviewed and updated annually.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	• These Mineral Resources are located within the bounds of the Mineral Exploration and Production Agreement (MEPA), a direct agreement with the Laos Government. The MEPA provides for exploration, development and extraction of any Mineral Resources discovered. • The MEPA provides for exploration, development and extraction of any Mineral Resources discovered. The Sepon MEPA occupies portions of both Savannakhet Province, and Khammouane Province to the immediate north. • The Sepon MEPA originally occupied 5212km^2. Various relinquishments have occurred since it was granted in 1993, the most recent relinquishment in early 2005 has resulted in the current retained area of 1247km^2. • A royalty is payable to the Government of Laos, representing 4.5 % of the FOB value of minerals received by LXML. The Lao Government also owns a 10% share in LXML. Tenement rental is payable at US$500 per square kilometre per annum (for operational and mining areas) in accordance with Annex D of the MEPA. Taxes and other obligations are set out in Article 13 of the MEPA. • The operating period in accordance with the MEPA is thirty (30) years, from commencement of operations. As defined in Article 10, Paragraph 2 of the MEPA for Sepon Gold Stage 1, the operating period commenced on 1 March 2003.

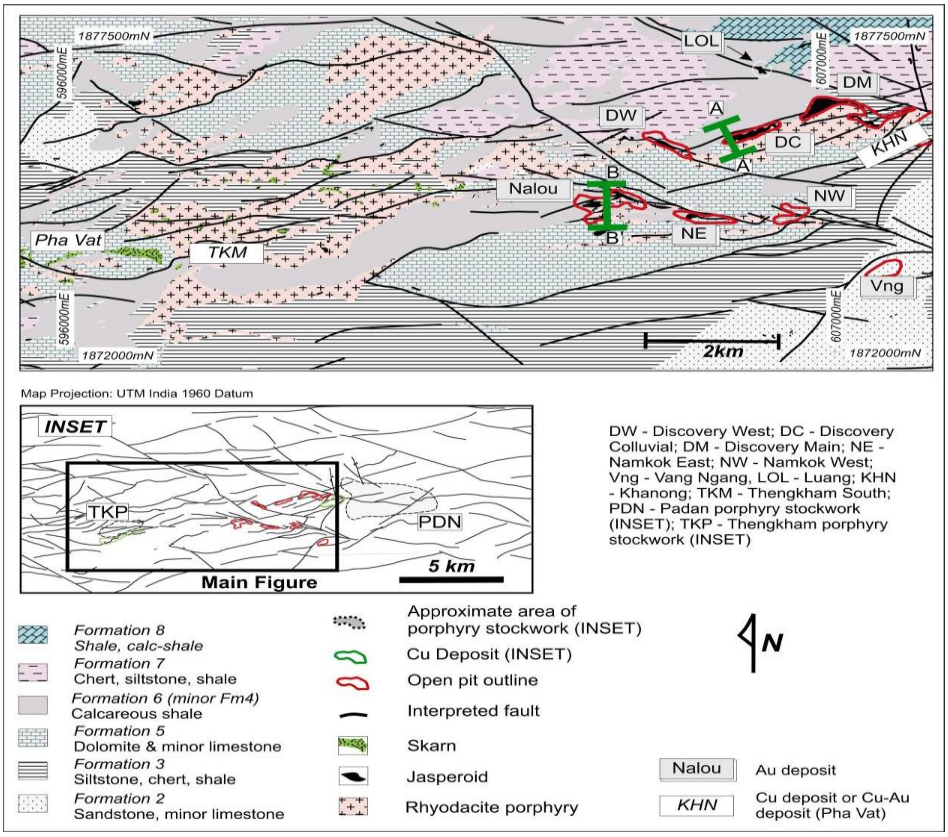
Criteria	Commentary
	• The terms of the agreement provides for the right to apply for two extensions of the operating period with each extension for a period of 10 years. • There are no known impediments to operating in the area.
Exploration done by other parties	• CRA Exploration first identified the Sepon Mineral District as an area of interest in 1990 and formed LXML as holder of the MEPA. • Between 1995 and 1999 Rio Tinto discovered and defined several gold only Mineral Resources and copper and gold Mineral Resources at the Khanong prospect. • Oxiana Resources Limited became manager of the Sepon Project in 2000 by buying 80% of LXML before later buying the remaining 20% interest from Rio Tinto. The Laos Government exercised its option to acquire a 10% interest in LXML in 2007. • In 2008 Oxiana merged with Zinifex Ltd to form OZ Minerals Limited. • In 2009 MMG acquired OZ Minerals Limiteds interest in LXML. • The Sepon exploration and resource geology groups have been maintained throughout the OZ Minerals and MMG takeovers. • MMG closed the Sepon exploration department in early 2017.
Geology	• 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. It lies within the Troungson geological region covering a broad spectrum of rocks ranging in age from Upper Proterozoic to Jurassic. The regional geology is dominated by an Upper Palaeozoic sedimentary belt of arkosic and feldspathic sandstone, variably calcareous and carbonaceous siltstone, shale and limestone which is variably dolomitised 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. • Several styles of mineralisation have been recognised within the Sepon Mineral District: porphyry-like Cu-Mo-Au mineralisation, skarnoid Cu-Mo-Au mineralisation adjacent to porphyry intrusive, distal skarn related Cu-Au-Ag+/- Pb+/-Zn massive sulphide veins, Carlin type carbonate hosted gold mineralisation and carbonate hosted Mississippi Valley type Pb-Zn-Ag mineralisation. In addition weathering and supergene re-mobilisation has created supergene copper, exotic supergene copper, oxide gold and alluvial gold in karst fill deposits. • All primary deposits are hydrothermal and, at least spatially, related to the RDP intrusive and skarns. Supergene copper mineralisation results from the oxidation, dissolution and transport of primary sulphide hosted copper mineralisation to sites where chemical conditions result in copper precipitation (reduced groundwater, replacement of sulphide, reaction with alkali carbonate). Supergene copper mineralisation occurs above and down slope of primary mineralisation. Chalcocite mineralisation replaces massive pyrite immediately above the skarns. Copper carbonate mineralisation occurs where copper rich groundwater reacts with carbonate rocks. The best supergene copper zones occur above higher grade zones of primary mineralisation and have a vertical profile with the best grades immediately above the base of weathering. • Gold mineralisation mostly occurs in the fault zones and adjacent to the fault zones

Criteria	Commentary
	at the contact between the dolomite of the Nalou formation and the overlying shales and nodular carbonate of the Discovery Formation. 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. Regional WNW-striking, steeply NNE-dipping normal faults are believed to have been the major pathway for ascending mineralising fluids. • Primary gold mineralisation occurs as Carlin style gold forming distally to the copper skarn systems. Mineralisation occurs in association with decalcification and partial silica replacement of calcareous mudstones ('jasperoid') along steep faults, and is typically best developed at the contact of the Nalou Formation (dolomite) and the overlying Discovery Formation (nodular calc-shale). • Oxide gold mineralisation shows further control by weathering processes with very high grade zones developed as karst fill (mineralised jasperoid boulders occurring within a matrix of decomposed rock and clays) on chemically weathered carbonate rocks.
Drillhole information	• No individual drillhole is material to the Mineral Resource estimate and hence this geological database is not supplied.
Data aggregation methods	• This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section. • No metal equivalents were used in the Mineral Resource estimation.
Relationship between mineralisation width and intercept lengths	• Mineralisation true widths are captured by interpreted mineralisation 3D wireframes. • Most drilling was at -60° to -90° dip angles in order to maximise the width of intersections. • Geometry of mineralisation is interpreted as sub-horizontal in the supergene and sub-vertical in the hypogene material and as such current drilling allows true width of mineralisation to be determined.
Diagrams	

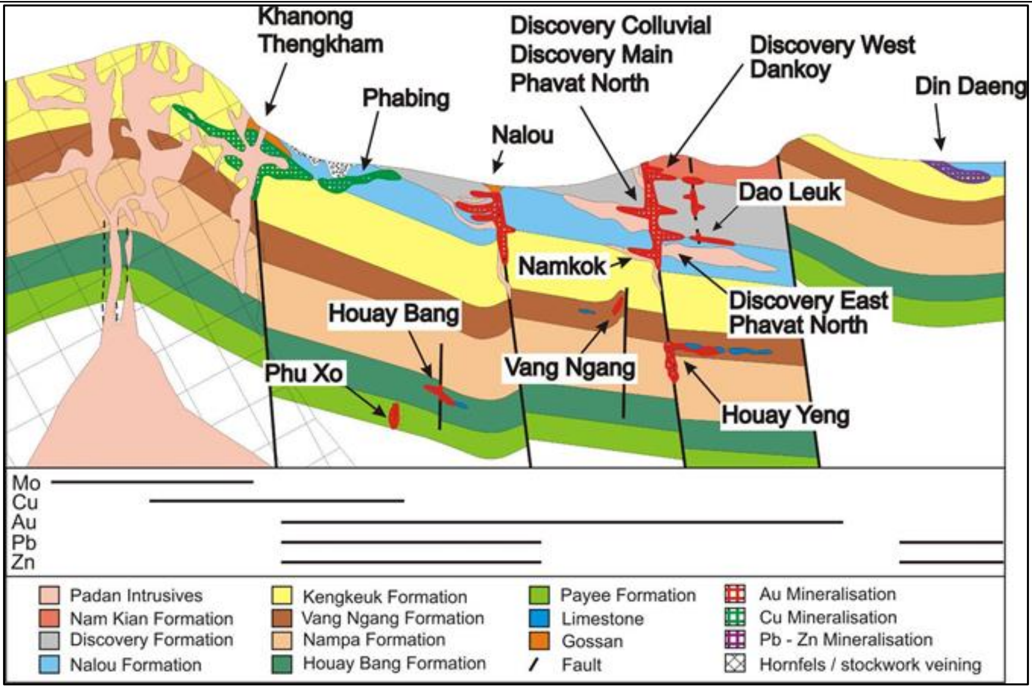
Criteria	Commentary



Location of Sepon gold and copper deposits (red line=MEPA)



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

Criteria	Commentary
	 **A generalised east - west cross section of Sepon deposit is shown above.**
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	• Exploration within the 2016/2017 drill season's primary focus is to: o Aid in the increase of the known Mineral Resource base for oxide copper through discovery of new copper deposits and definition of early stage targets to advanced exploration targets.

Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	• The following measures are in place to ensure database integrity: o A digital field logging system (with lookups and referential integrity (RI)) or paper based system for exploration both transferring to a data entry database (DataEntryDB), (with lookups and further RI) then transferring to the master database (LaosDB) where assays are loaded and approved. o The senior geologist reviews all new drill data logged in exploration logging view and sign off after all corrections are made prior to loading into LaoDB which is done via SQL Server stored procedures to detect and hold any errors on import. • Geobank software provides a front end to an SQL database for Mineral Resource and grade control data. Analytical data is uploaded directly from laboratory SIF

	files. • The measures described above ensure that transcription or data entry errors were minimised. • Data validation procedures include: ○ Validation routines by database personnel check for overlapping sample depths, lithological and alteration information, as well as reject criteria such as logging information past EOH depth. • Data used in the Mineral Resource has passed a number of validation checks including both visual and software related prior to use in the Mineral Resource. • The gold database used in the Mineral Resource is a merger of several databases and represents the best data for the gold mineral resources, which is separate to the master database. A number of major data transfers, with inherent risks, have occurred over the life of the project, several suspect holes (assays) exist within the master drillhole database. Further work is required to verify and update the master database. • There is spatial confidence in the occurrence of gold grades, however the accuracy of those grades is unknown. • The database currently being used for the gold estimates is deemed to be appropriate for an Indicated Mineral Resource.
Site visits	• The Competent Person has undertaken numerous (more than 20) visits to Sepon since 2013 in the course of providing Mineral Resources estimation, project management and mentoring to the site geologists. • All site visits include, core and logging review, drill site inspections, design of drill programs and review of geological modelling. When possible laboratory visits are also undertaken, this however did not occur within the 2016-2017 reporting period.
Geological interpretation	• Prior to Mineral Resources estimation an underlying three dimensional geological model (stratigraphy, structure and intrusives) was made for all deposits. All the domains used for estimation were interpreted using known controls on the domain variable with the geological model as a framework. For example the gold grade domains, whilst interpreted at a nominal gold grade, follow favourable stratigraphic contacts and controlling fault structures. Confidence in the geological (domain) interpretation for all Sepon Mineral Resource estimates is high. • For the copper deposits a surface to demarcate the base of supergene mineralisation (BOSM) is interpreted using logged drillhole data, core photos and assay data. BOSM is important for the carbonate oxide copper mineralisation as high grade copper is known to collect within depressions that are assumed to be related to structural controls. • The boundary/gradeshell cut-off for most copper deposits is 0.1% Cu, with a 1% Cu domain also used in the Khanong copper mineralisation as it shows bimodal characteristics and a 1.2% Cu domain within the Thengkham Ridge. For gold deposits a domain cut-off of 0.2g/t Au – 0.5g/t Au was used. These domain cut-offs were selected by geostatistically identifying population breaks in the sample data. As well, visual investigations ensuring that these cut-offs displayed reasonable continuity in three-dimensional space taking into account the local geology. Grade domains are constructed from polygon generated triangulations and/or leapfrog models. • In the supergene copper zones the logging of key minerals used to distinguish

	chalcocite mineralisation from copper carbonate mineralisation is at times incompatible with assay data. In these situations the assay data 'over-rides' the logging data. • In 2016-2017 review of all the geochemistry developed probabilistic algorithms that allow the probability of material being primary, chalcocite or carbonate to be calculated and stored in the database. These fields are also used in the construction of copper speciation domains. • Sulphur values are also used in the construction of the chalcocite mineralisation domains as there is an association between chalcocite and sulphur that is not present in the carbonate copper mineralisation, as a general rule chalcocite is modelled within the >=0.5% S region of the oxide mineralisation. The sulphur domain is constructed using IK modelling techniques. • The underlying geological models were largely interpreted from logged drilling data and deposit scale surface geological mapping. • Where geologically plausible alternative interpretations exist the Mineral Resource category was downgraded. • If new drill programs contradict the geological model, the model is updated to reflect new drill data. • The geological continuity of mineralisation and ore mineralogy is a key input into Mineral Resources classification with supergene copper mineralisation. This largely reflects reduced ore mineralogy continuity rather than ore grade continuity.
Dimensions	• Sepon hosts a number of deposits, the dimension of each deposit included in this Mineral Resource are listed below. Where block model extents overlap, wireframes and if necessary priorities are used to prevent double reporting. ○ Thengkham Ridge (combined Thengkham North, South and East, TKS, TKE and Songkham West) 15000mE-22900mE, 72800mN-75656mN, 0mRL-600mRL. ○ Phabing: 15950mE-17750mE, 74250mN-75470mN, 0mRL-500mRL. ○ Khanong: 26600mE-29125mE, 74748mN-76524mN, -50mRL-650mRL. ○ Discovery West: 23875mE-25555mE, 75250mN-75970mN, 0mRL-300mRL. ○ Discovery Main: 25500mE-28200mE, 75250mN-77110mN, 150mRL-450mRL. ○ Nalou: 22700mE-24500mE, 73730mN-75350mN, 150mRL-325mRL. ○ Namkok West: 24500mE-26060mE, 74000mN-75320mN, 0mRL-300mRL. ○ Phavat North - Dankoy: 15300mE-17460mE, 73700mN-75200mN, -0mRL-650mRL. ○ Vang Yang South: 27100mE-27820mE, 72400mN-73300mN, -0mRL-500mRL. ○ Houay Bang: 33200mE-33992mE, 76500mN-77304mN, 50mRL-450mRL. ○ Muang Luang: 25500mE-26250mE, 76700mN-7752mN, -330mRL-350mRL.

Estimation and modelling techniques	• Mineral Resource estimation was undertaken in Vulcan (Maptek) mining software with the following key assumptions and parameters: ○ Ordinary Kriging interpolation has been applied for the estimation of Cu, Au, Ag, Ca, Mg, Mn, Fe, Total S and Total C (TK_Ridge block model only) for the copper models, sulphide sulphur and carbonate carbon are also estimated in the gold models. Inverse distance to the power of two interpolation has been applied where there was insufficient data to model spatial continuity for kriging weights within the gold block model estimates. This is considered appropriate for the estimation of Mineral Resources at Sepon. ○ Extreme grade values were managed by upper grade capping. The typical upper-cap used is the 99th percentile, or a grade that shows a metal loss within the raw data of less than 5% metal to contain outliers, however this may vary depending on the results of geostatistical analysis. For some domains however, high-yield restrictions were applied to contain outliers. The high yield grade threshold varies depending on the results of geostatistical analysis and by domain. ○ The block models in the copper deposits use geostatistical domains comprised of various combinations of copper grade (nominal 0.1% and 1% or 1.2% Cu grade shell), lithology, oxidation (supergene / hypogene), sulphur grade (0.5% S grade shell in supergene). Structual domains are also modelled based on the orientation of the mineralisation. In the gold block models geostatistical domains comprised gold grade domains (0.2g/t Au – 0.5g/t Au), lithology, oxidation (base of complete oxidation, base of partial oxidation and primary domain) and orientation wireframes. ○ The estimates of copper and gold were predominantly 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. ○ Locally varying anisotropy (LVA) modelling was used instead of static searches within structural domains to estimate the copper oxide mineralisation within the Thengkham Ridge model. ○ The carbonate copper estimate at Khanong was estimated using a soft boundary between the 0.1% Cu and 1% Cu domain as they are considered to be associated. The soft boundary limits used: – 1% Cu soft contact (limited to 12.5m x 12m x 2.5m along domain boundary) and –0.1% Cu soft contact (limited to 25m x 24m x 5m along domain boundary) ○ A composite length of 1m downhole was chosen for 2016 copper models, and the compositing process validated. A 2m or 3m composite has been used in the gold models historically based on sample size. ○ Exploratory data analysis, variography and search neighbourhood optimisation for each domain was performed using Supervisor or Vulcan geostatistical software. ○ Ca, Mg and Mn within the dolomite unit have all used the same semi-variograms for estimation in the copper models. The dolomite unit has been generated using categorical Indicator Kriging and a probability shell.

	Dolomite and Manganese content strongly influence gangue acid consumption in the plant. The dolomite triangulation had further restriction applied for the Thengkham Ridge region and was modelled as "high probability dolomite" and "low probability dolomite" clays to account for the bimodal nature of the Ca and Mg within the oxide dolomite domains. This was probabilistically calculated using geochemistry signatures of the oxide material. o Total sulphur and carbon are estimated to assist with gold speciation and in the determination of non-acid forming (NAF), potentially acid forming (PAF) material. o The minimum and maximum number of composites allowable to interpolate a block was typically set at 4 / 10 and 16 / 32 respectively, based on a Quatitative Kriging Neighbourhood Assessment (QKNA). o Block model estimates of copper have not utilised octant searches and with the search distance set to approximately two thirds of the variogram range and doubled for pass 2 and 3 within the Thengkham model, and use the entire variogram range for estimation within the Khanong model. o Octant searches were used for the first two passes for ancillary elements only. • No assumptions about the recovery of by-products have been made. • Parent block sizes within the copper block models are 25m x 24m x 5m (XYZ), with limits of 12.5m x 12m x 5m limits applied to copper domains in the Thengkham Ridge model. The parent block size has considered: data spacing, mining methods and copper variography. Sub-blocks honouring relevant shapes and surfaces were used. The parent block size delineates ore zones without compromising the block variance. Using similar methods, parent blocks of 15m x 6m x 2.5m (XYZ) were used for the gold block models. • Block discretisation is 5 x 4 x 3 (x,y,z). • Search distances in general for copper estimates are the length of the first variogram structure in pass 1 and doubled in subsequent passes. Larger passes were used to interpolate less well informed blocks. However this varies from deposit to deposit. • Ancillary elements generally follow the rule of two thirds of the range of the variogram for pass 1 and doubled in subsequent passes. • No further assumptions have been made regarding modelling of selective mining units. • Block models are validated by: o Visual inspections for true fit with the high and low grade wireframes (to check for correct placement of blocks and sub-blocks). o Visual comparison of block model grades against composite file grades. o Global statistical comparison of the estimated block model grades against the composite statistics and raw data. o Global and local (on key sections) swath plots are used to check for bias. o Validation block models to determine the impact of each variable change. • Reconciliation with grade control block models (where available) were undertaken.

	Results indicate good global reconciliation, but at times significant monthly variances. This led to the decision: ○ To use grade control drill data in the Khanong estimate as the wide spaced Resource drilling was overstating grades and smoothing the estimate in areas that grade control demonstrated high variability within grade. ○ Create a high grade copper domain in the Thengkham estimate to minimise smoothing of grades and reflect the short scale variability of the mineralisation.
Moisture	• Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	• Since 2016 the Sepon Mineral Resources has been reported via a net value return (NVR) process, or margin calculation. The NVR script takes into account the impact gangue acid consumption has on the processing route material will take through the plant and the extra cost associated to process those parcels of material and recoveries and metal prices (as per the pit shell generated price assumptions). • Any block that has a margin value greater than $0 has been flagged as a reportable Mineral Resource block. Review of the NVR shows the value is equivalent to the following cut-offs. ○ Carbonate copper (Type 2) = ~1.4%-1.6% Cu. ○ Chalcocite copper = ~1.5%-1.6% Cu. ○ Primary sulphide copper = ~0.5%-0.6% Cu. ○ Oxide gold = 0.9 g/t-1.7 g/t Au. ○ Partial oxide gold = 1.7 g/t-4.2 g/t Au. ○ Primary gold = 1.3 g/t-2.6 g/t Au. • Copper Mineral Resources have also been reported within a US$3.28 pit shell. • Gold Mineral Resources have also been reported within a US$1,400 pit shell. • These Mineral Resource cut-off grade represents material that has reasonable prospects for eventual economic extraction. Areas that are remaining skins along historic copper mining have been removed from the Mineral Resource as it is not economic to rehab these areas to retrieve this material. • The reporting cut-off grade is in line with MMG's policy on reporting of Mineral Resources which considers current and future mining and processing costs and satisfies the requirement for prospects for future economic extraction.
Mining factors or assumptions	• Mining factors and assumptions as discussed in section above: Cut-off parameters
Metallurgical factors or assumptions	• Sepon is an operating mine. The metallurgical process is a hydrometallurgical process involving grinding, tank leaching, autoclave leaching, solvent extraction and electrowinning. The process has been operating successfully since start up in 2005. • Additional discussion of metallurgical factors is provided in Section 4 of this table.
Environmental factors or assumptions	• Environmental permitting in the Lao PDR is completed through environmental and social impact assessments (ESIA), in accordance with the MEPA, and other relevant legislation. ○ The ESIA process is similar in nature to the process followed around the

<table>
<tr><td></td><td>
world. In February 2010, the Lao PDR issued the decree on Environmental Impact Assessment, revised through regulation 8030/MONRE in December 2013.
o In November 2011, the Environmental Impact Assessment (EIA) guidelines were released. EIA guidelines provide an interpretation of the ESIA decree and outline government expectations for what will be in an ESIA and how ESIA should be conducted.
• These two documents outline the process for environmental assessments in the Lao PDR.
• The acid forming potential characteristics were assigned to a block model variable as per the table below. The values are based on a study by Environmental Geochemistry International.
<table>
<tr><th colspan="4">PAF and NAF values assigned in the modelling process</th></tr>
<tr><th>Lithology Domain</th><th>S1/2/3/4</th><th>Code</th><th>Description</th></tr>
<tr><td>All except Dolomite</td><td>>0.3% S</td><td>1</td><td>PAF</td></tr>
<tr><td>Dolomite</td><td>>0.3% S</td><td>2</td><td>NAF</td></tr>
<tr><td>All</td><td><0.3% S</td><td>3</td><td>NAF</td></tr>
</table>
Where PAF = Potentially Acid Forming, NAF = Non-Acid Forming
</td></tr>
<tr><td>Bulk density</td><td>
• Samples for bulk density determination are taken from diamond drill core every 5m-10m using the wax coated core immersion method.
• The bulk density determinations were estimated into the Thengkham ridge block model by omnidirectional ordinary kriging, estimation was based on supergene vs hypogene division and then further on lithology.
• In the other deposits where density data is sparse, density was assigned to the block model SG using average values within ore domains and lithological domains. Reconciled mined tonnes demonstrate these values are robust.
</td></tr>
<tr><td>Classification</td><td>
• Classification is determined by examination of the following criteria:
o Geological: mineralisation continuity including spatial configuration and spatial continuity.
o Sample quality: areas of wet RC drilling are downgraded.
o Statistical: kriging efficiency and kriging slope of regression.
o Data: the relative data density, distance of nearest composite and number of composites used.
• Classification is applied using classification wireframes constructed around aggregate areas generally conforming to the classification criteria.
• In the Competent Person's view the classification is appropriate to the deposits at Sepon given the available data for estimation.
</td></tr>
<tr><td>Audits or reviews</td><td>
• An external review carried out by AMC Consultants was completed in 2017 on the Thengkham Ridge Resource model due to the degree of change in metal between estimates sparking internal policy. The review made immediate recommendations for change:
o Replace the 1.5% Cu high grade domain with a 1.2% Cu domain.
o LVA modelling should use an upper and lower surface for estimation (Topo and BOSM).
</td></tr>
</table>

	◦ A full reconciliation between grade control and resource model to be completed. All of the above changes were completed and a secondary review was carried out by AMC Consultants. The review endorsed the model and agreed it was fit for purpose for estimating Mineral Resources and that and Indicate Mineral Resources category was appropriate for the deposit. • Internal review was carried out on the Khanong and Thengkham ridge models in 2016. All recommendations from this review were taken forward to produce the 2016 Mineral Resources. • In 2014 the TKM and KHN Copper Mineral Resource estimates were audited by H&S Consultants. Numerous recommendations were made and all were addressed in the 2015 Mineral Resource estimate. • Independent technical review on the copper and gold Mineral Resources was undertaken in 2010 by AMC Consultants.
Discussion of relative accuracy/ confidence	• Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support. • Reconciliation for Sepon during the past 12 months shows a total decrease of 18% in tonnes and 12% in metal from the Mineral Resource estimate to the grade control models, however an 8% increase in copper grade has been seen within the grade control model. Reconciliation shows a greater variation over shorter periods and for individual deposits. Overall differences within the reporting period fall within the acceptable limits for an Indicated Mineral Resource. • The Khanong region is now mining along the oxide to primary transition zone, traditionally labelled oxide material within the Resource model close to this boundary is performing better in the primary and low grade float circuit and is being labelled as primary in the Grade control model, this transition zone needs to be monitored and Resource estimations need to align with the Grade control criteria to ensure a better reconciliation. • Prior to the oxide gold plant shutting down in late 2013 reconciliations showed some variance on predicted mineral resource tonnage and grade compared to grade control models on a pit by pit basis. The primary gold material has never been mined and as a result, no reconciliation can be undertaken.

5.2.3 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, Chevaun Gellie, confirm that I am the Competent Person for the Sepon Mineral Resource 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 Mineral Resource section of this Report to which this Consent Statement applies.

I am a full time employee of MMG Ltd. at the time of the 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 Sepon Mineral Resource 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 the Sepon 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 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Chevaun Gellie MAusIMM (#311344)

18/10/2017

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Signature of Witness:

Rex Berthelsen
Melbourne, VIC

Witness Name and Residence: (e.g. town/suburb)

5.3 Ore Reserves – Sepon

5.3.1 Results

The 2017 Sepon Ore Reserves is based on the 2017 Mineral Resources model.

The 2017 Sepon Ore Reserves are summarised in Table 14. All data reported here is on a 100% asset basis. MMG's attributable interest in Sepon is 90%.

Table 14 2017 Sepon Ore Reserves tonnage and grade (as at 30 June 2017)

Sepon Ore Reserves (90%)			
			Contained Metal
	Tonnes (Mt)	**Copper (% Cu)**	**Copper ('000 t)**
Supergene Copper			
Probable	3.5	4.7	166
Total	**3.5**	**4.7**	**166**
Copper Stockpiles			
Probable	5.6	1.4	79
Total	**5.6**	**1.4**	**79**
Primary Copper			
Probable	0.35	1.1	4.0
Total	**0.35**	**1.1**	**4.0**
Sepon Total	**9.4**	**2.6**	**249**

Cut-off grades were calculated at a US$2.73/lb copper price. The following approximate COG's are applied:

Supergene (Carbonate and Chalcocite): 1.1% Cu

Supergene Low Grade Float (Carbonate and Chalcocite): 0.9% Cu

Primary: 0.5% Cu

The COG estimates are based on a Net Value Script (NVS) calculation that takes into account material type, price assumptions, process method, gangue acid consumption, distance to the crusher, and costs associated with current operating conditions.

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

Contained metal does not imply recoverable metal.

The main difference from the 2016 Ore Reserves is:

(i) An updated Mineral Resources model, including additional drilling, has seen a refinement of the interpretation of the Thengkham and Khanong Mineral Resources. This resulted in a decrease in total contained copper metal.

(ii) Mine and stockpile depletion.

(iii) Assumed copper price reduced to $2.73/lb in 2017 from $2.95/lb in 2016.

5.3.2 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 15 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code. 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 15 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Sepon Copper Ore Reserves 2017

Assessment Criteria	Commentary

Mineral Resource estimates for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Ore Reserves.
	• The Ore Reserves include ore on stockpiles.
	• Three sub-celled Mineral Resources block models were used for the optimisation purposes:
	  o Khanong (KHN): "SEP_KHN_May17.bmf".
	  o Thengkham (TKM): "SEP_TKM_MROR_2017C.bmf".
	  o Phabing (PHB): "PHB_MROR_2017.bmf"
	• Mineral Resources are modelled using solid wireframes for geological boundaries and grade boundaries for most copper deposits is 0.1% Cu, with a 1% Cu domain also used in the Khanong copper mineralisation as it shows bimodal characteristics and a 1.2% Cu domain within the Thengkham Ridge for the carbonate mineralisation.
	• Ordinary Kriging interpolation has been applied for the estimation of Cu, Au, Ag, Ca, Mg, Mn, Total S, Total C and Fe for the copper models. This is considered appropriate for the estimation of Mineral Resources at Sepon. Ca, Mg and Mn form the basis for the gangue acid consumption estimate which is a large proportion of the processing costs.

Site visits	• The Competent Person visited the site in July 2016 and February, March and May in in 2017. • Each visit consisted of discussions with relevant people associated with Ore Reserves modifying factors including geology, grade control, geotechnical parameters, mine planning and mining operations, metallurgy, tailings and waste storage, and environmental disciplines. The outcomes from the visits have included achieving a common understanding, review of assumptions, calculation of cut-off grade and developing the Life of Asset mine plan. • A site visit was also carried out by Group Manager Technical Services (ASEA/Africa)-MMG in February, March, April and June 2017.
Study status	• Sepon is an operating mine. The Ore Reserves are based on actual operating data for supergene material. • The Sepon Primary Feasibility Study was concluded in December 2015 and primary ore assumptions for copper cut-off grade have been estimated based on inputs from this study and the 2017 Budget. Primary ore processing is planned to commence in late 2019. • Life-of-Asset Low Case was produced as part of the MMG planning cycle. This Low Case informs the Ore Reserves and is technically achievable, economically viable, and, material Modifying Factors have been considered.

Assessment Criteria	Commentary
Cut-off parameters	• Break even cut-off grades were calculated for both the supergene and primary ores at a US$2.73/lb copper price. The following approximate cut-off grades are applied o Supergene (Carbonate and Chalcocite): 1.1% Cu. o Supergene Low Grade (LG) Float (Carbonate and Chalcocite): 0.9% Cu. o Primary: 0.5% Cu. • The cut-off grades estimates are based on a Net Value Script (NVS) calculation that takes into account material type, price assumptions, process method, gangue acid consumption calculation, distance to the crusher, and costs associated with current operating conditions. • An incremental COG is used to define Primary and LG float material. It does not include G&A and applies reduced sustaining capital costs. As such, primary and LG float material must be co-fed with supergene material to be economic. • The NVS is run over the Mineral Resources Model to identify material that is suitable for processing. This material must be economic during processing to be defined as Ore Reserves. • For the cost assumptions please see the "Costs" section. • For the price assumptions please see the "Revenue factors" section.
Mining factors or assumptions	• The method of Ore Reserves estimation included pit optimisation, final pit and phase designs, consideration of mine and mill schedule and all modifying factors. • Sepon mine is an open pit operation that is mining and processing copper ore. The operation uses a fleet of excavators and articulated trucks along with a fleet of auxiliary equipment. • This mining method is appropriate for the style and size of the mineralisation. • Slope design guidance took into account the potential failure modes at bench, interamp and overall wall scale. • Interamp angles varied from 18 to 40 degrees for saprolitic material and 35 to 50 degrees for hard competent rock. • Limit equilibrium analysis was conducted on all the optimised pit shapes to ensure that the Factor of Safety was greater than or equal to 1.2. • The pit optimisations were based on the 2017 Mineral Resources block models and the strategy for the final pit selection was based on a revenue factor 1 (RF=1.0). RF 1.0 pit shells are used across all assets in the MMG Group. Final pit designs were carried out on these shells. • Mining dilution was estimated to be 7.5% for TKM and 10.0% for KHN mine areas. Mining recovery was estimated to be 90.0% for TKM and 95.0% for KHN mine areas. Historical performance and dilution modelling was used to support these assumptions. • Minimum mining width (bench size) is ~20m.

Assessment Criteria	Commentary
	• No Inferred Mineral Resource material has been included in optimisation and/or Ore Reserves reporting. • All required infrastructure is in place. • The Ore Reserves are based on actual operating data and projected forecasts based on the 2017 Sepon Budget. Additional pits that are yet to be developed are similar in nature to the current mining environment.
Metallurgical factors or assumptions	• Sepon is an operating mine. The metallurgical process is a hydrometallurgical process involving grinding, tank leaching, autoclave leaching, solvent extraction and electrowinning. The process has been operating successfully since start up in 2005. • A low grade concentrate is also produced from flotation of stockpiled low grade or primary ore, which is also leached in the autoclave and leach circuit. The process has been operating successfully since start up in late 2015. • Copper recovery for the mill feed is determined by the equation: Cu recovery (%) = (Cu Grade – Tails Grade (0.32%) / Cu Feed Grade) – Soluble Loss (3.50%) • The median fixed tails grade of 0.32% is derived from analysis of historical data as well as from test work of drill core samples. Over the last 3 years actual recovery has averaged 85%. • Copper recovery for low grade stockpiled ore via flotation is: Cu recovery (%) = min(75, 24.84 + 17.46 x Ln(S)) • Copper recovery for primary flotation is: Cu recovery (%) = min(94, 100 – 76 x (ASCu/Cu) – 0.23 x (S/Cu) – 0.58 x (Mg) - 0.68 x (Fe/S)) • The main deleterious component in the ore is dolomite which increases acid consumption in the leaching process. • High gangue acid consumption (GAC) 'dolomite' ore (GAC >75kg/t, Cu<7.5%), has the option to pass through a scrubber screen to remove oversize material; reducing the average gangue acid consumption levels through the copper plant. Scrubber parameters comprise 1.1Mtpa throughput rate, mass rejection of 20% and copper losses of 10%. Gangue acid consumption is estimated for direct feed supergene chalcocite ore using the formulas: if Ca/Mg < 2.5 AND Ca>Mg 10.5 + 43.8 x % Ca + 16.5 x % Mn if Ca/Mg > 2.5 10.5 + 43.8 x % Mg x 1.65 + 16.5 x % Mn if Ca/Mg < 2.5 AND Mg>Ca 10.5 + 43.8 x % Mg x 0.1 + 16.5 x % Mn Gangue acid consumption is estimated for direct feed supergene carbonate ore using the formulas: if Ca/Mg < 2.5 AND Ca>Mg 30.5 + 43.8 x % Ca + 16.5 x % Mn if Ca/Mg > 2.5 30.5 + 43.8 x % Mg x 1.65 + 16.5 x % Mn

Assessment Criteria	Commentary
	if Ca/Mg < 2.5 AND Mg>Ca 30.5 + 43.8 x % Mg x 0.1 + 16.5 x % Mn Gangue acid consumption is estimated for post scrubber supergene chalcocite ore using the formulas: if Ca/Mg < 2.5 AND Ca>Mg 10.5 + 21.9 x % Ca + 16.5 x % Mn if Ca/Mg > 2.5 10.5 + 43.8 x % Mg x 1.65 + 16.5 x % Mn if Ca/Mg < 2.5 AND Mg>Ca 10.5 + 43.8 x % Mg x 0.1 + 16.5 x % Mn Gangue acid consumption is estimated for post scrubber supergene carbonate ore using the formulas: if Ca/Mg < 2.5 AND Ca>Mg 30.5 + 21.9 x % Ca + 16.5 x % Mn if Ca/Mg > 2.5 30.5 + 43.8 x % Mg x 1.65 + 16.5 x % Mn if Ca/Mg < 2.5 AND Mg>Ca 30.5 + 43.8 x % Mg x 0.1 + 16.5 x % Mn Net acid consumption (NAC) is then calculated using the formula: NAC (kg/t) = GAC (kg/t) + 5 • The NAC 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. • For concentrate produced from low grades ores, a gangue acid consumption of 5kg/t is assumed which is the historical average. • As required, blending of ore types is used to control the acid requirement based on their respective calculated gangue acid consumption values. • The inclusion of a term for Mn in the gangue acid equations reflects a small, but significant consumption of acid by manganese wad. • For Ore Reserves, a processing rate of up to: 2.59Mtpa of supergene ore, 0.85Mtpa of low grade stockpile and 0.6Mtpa of primary ore with an electrowinning capacity of 90ktpa of copper cathode. • Production rates have been demonstrated as sustainable over the previous years of the operation. Primary production rates have been estimated as part of the Sepon Sustain Primary Copper Feasibility Study.
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). An Environmental Monitoring and Management Plan (EMMP) is the guiding environmental document submitted to the Government every three years or when there is a material change to activities on site. An EMMP update was submitted in 2015 which is under discussion with the Lao regulator. An update to the EMMP and SMMP is planned for 2018. • Potential for waste rock acid generation is currently estimated based on the sulphur grade. Waste is separated based on its potential for acid generation, with Potentially Acid Forming (PAF) material being encapsulated in PAF cells within waste rock dumps. Non-acid forming (NAF) materials are used for encapsulation or to construct landforms. • Where practicable, waste rock is backfilled into sterilised pit voids.

Assessment Criteria	Commentary
	• Western Tailing Storage Facility (WTSF) tailings pipeline was fully replaced/repaired consistent with regulator expectations. • With an additional lift, the WTSF tailings storage facility will have capacity to contain the Ore Reserves tailings.

Infrastructure	• The Sepon mine site is well established with the following infrastructure in place:
	  o The plant is operational.
	  o The Primary processing facilities consist of existing decommissioned gold processing facilities, only minor work was required to tie these facilities into the existing plant.
	  o Labour is mostly sourced from Vientiane, Savannahket and surrounding villages with some expatriate support. There is an existing accommodation facility onsite.
	  o There is sufficient water for the processing.
	  o Copper cathode is transported off-site by truck.
	  o Site has access by sealed road and an all-weather airstrip.
	  o Power is supplied from Thailand by two 115kV transmission lines from the Mahaxa1 Substation, where three 50MVA transformers step down the voltage to 22kV and supply the mine and processing operations.
	  o The Ore Reserves do not require any additional land for expansion.
	  o Tailings Storage Facility in place and future lifts are accounted for.
	• No significant additional site infrastructure is required to realise the Ore Reserves other than a lift to the tailings storage facility.

Costs	• Sepon is an operating mine and has historical costs that have been used to inform the 2017 Sepon Budget (January 2017 to December 2017). • The Ore Reserves estimation has been based on these costs in accordance with JORC 2012 code. • The processing costs include calculated gangue acid consumption and vary by process stream. • The final product contains no deleterious elements. • US dollars have been used thus no exchange rates have been applied. • Weathering profiles have been updated to model in-pit blasting. • Since the final product is LME grade A and grade B copper cathode there are no applicable treatment or refining or any other similar charges. • Sustaining capital costs have been included in the pit optimisation. The sustaining capital costs are principally related to the tailings storage facility lift construction and the process plant. The inclusion or exclusion of these costs in the Ore Reserves estimation process has been done in accordance with the JORC 2012 code. • Allowances have been made for royalties.

Assessment Criteria	Commentary
	• A cash flow model was produced based on the detailed mine and processing schedule. This model includes the aforementioned costs as well as all sustaining capital that is needed to realise the Ore Reserves.
Revenue factors	• For cost assumptions see section above – "Costs" • The assumed copper price is US$2.73/lb which is the same as that reported in the cut-off parameters section. These prices are provided by MMG corporate, and approved by the MMG Board, and are based on external company broker consensus and internal MMG strategy. • The 2016 Ore Reserves was based on an assumed copper price of $2.95/lb.
Market assessment	• MMG considers that the outlook for the copper price over the medium and longer term is positive, supported by further demand growth which is expected to exceed increases in supply. • Global copper consumption growth will continue to be underpinned by rising consumption in China and the developing countries in Asia as these nations invest in infrastructure such as power grids, commercial and residential property, motor vehicles and transportation networks and consumer appliances such as air conditioners. • Supply growth is expected to be constrained by a lack of new mine projects ready for development and the requirement for significant investment to maintain existing production levels at some operations. • Sepon is well located within the large and growing copper consuming region of South East Asia with the 250,000tpa Thailand market being only one day trucking from site. Sepon has contracts with a range of customers within Thailand and also exports some cathode to China. Demand for copper is expected to continue to grow in these regions and this should ensure a continuing attractive market for Sepon cathode going forward.
Economic	• The costs are based on historical actuals and the 2017 Sepon Budget. Revenues are based on forecasted realised costs and copper price as reported in the cut-off parameters section. • The Ore Reserves financial model demonstrates the mine has a positive NPV. • The discount rate is in line with MMG's corporate economic assumptions and is considered to be appropriate for the location, type and style of operation. • Standard sensitivity analyses were undertaken for the Ore Reserves work. Sepon is most sensitive to changes in metal price and flattening of pit slopes.
Social	• The Social Management and Monitoring Plan (SMMP) is the guiding document that describes the strategies used by Sepon in cooperation with key stakeholders to manage the social impacts and opportunities for local communities affected by mining operations. 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. • Sepon provides a range of social development support to the communities

Assessment Criteria	Commentary
	surrounding the operation, with a specific focus on United Nations Sustainable Development Goals 1-6. • Activities at Sepon have been refocussed to maximise the opportunities for sustainable social and economic outcomes beyond the life of mine. These investments focus predominantly on community based education and health infrastructure, and support for the development of small to medium enterprises. • A household income survey was undertaken in 2016-17 which demonstrated the level of economic uplift achieved in a sample of households. The survey showed that whilst this change had been significant, that the ability of communities to sustain this post mining was uncertain. In response to this, social investments now focus on communities more closely located to the mining operation and engaged additional external delivery and microfinance partners to support increased socio-economic outcomes at the household level.
Other	• MMG (via it subsidiary Lane Xang Minerals Limited) has an agreement (Mineral Exploration and Production Agreement) with the Government of the People's Democratic Republic of Laos dated 15th June 1993. • A Mining License 408/MEM.DGM has been granted over the area of mining operations. A renewal application is required QTR3 2019.
Classification	• The Ore Reserves classification is based on the JORC 2012 requirements. The basis for the classification was the Mineral Resources classification and cut-off grade. Indicated Mineral Resources above cut-off grade, and is demonstrated to be economic to process, is classified as Probable Ore. • Stockpile material at Sepon is classified as Indicated Mineral Resource. Indicated Mineral Resources above cut-off grade, and is demonstrated to be economic to process, is classified as Probable Ore Reserves. • The Ore Reserves do not include any Inferred Mineral Resources.
Audit or reviews	• An Ore Reserves Method Audit was completed by Mining One Pty Ltd in April 2014. The 2014 Audit concluded that the operational practices were sound and comparable to industry standards. • Subsequent site visits by the Competent Person supports this. All recommendations from the audit were addressed in the 2016 Ore Reserves. There have been no significant changes to the Ore Reserves Estimation methodology; hence the 2014 Audit continues to be valid. • The next external Ore Reserves audit is planned for completion in 2018 on the 2017 Ore Reserves.

Discussion of relative accuracy/ confidence	• The most significant factors affecting confidence in the Ore Reserves are:
	  o Four month plant shutdown planned for 2020- ability to restart the plant without compromising costs or recoveries.
	  o Geotechnical risk related to slope stability.
	  o Metallurgical recovery model uncertainty for primary ore.
	  o Increase in operating costs for mining and processing.

5.3.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 16.

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 16 Contributing Experts – Sepon Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
EXPERT PERSON / COMPANY	**AREA OF EXPERTISE**
Chevaun Gellie, Principal Geologist MMG Ltd (Melbourne)	Mineral Resources model Stockpile Tonnes and Grade
Kevin Rees, Principal Metallurgist MMG Ltd (Sepon)	Metallurgy
Christian Holland, Geotechnical Engineering Specialist MMG Ltd (Melbourne)	Geotechnical parameters
Paul Harris, Operation Manager – Sepon MMG Ltd (Sepon)	Infrastructure and capital projects
Marc English, Principal Mining Engineer AMC Consultants Pty Ltd.	Whittle optimisation, mining costs, pit designs, mine and mill schedules, review of Ore Reserves estimate
David Machin, Principal Water and Tailings Engineer, MMG Ltd (Melbourne)	Tailings capacity
Knight Piésold	Tailings dam design
Richard Le, Senior Analyst Business Evaluation, MMG Ltd (Melbourne)	Economic assumptions and evaluation
Natalie Shade, Group Manager Environment, MMG Ltd (Melbourne)	Environment
Melanie Stutsel, General Manager Safety Environment and Social Performance, MMG Ltd (Melbourne)	Social
Steve Whitehead, General Manager Marketing, MMG Ltd (Melbourne)	Marketing

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

This Ore Reserve 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, Jodi Wright, 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 Pty Ltd.

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 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 the Sepon 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 Sepon Ore Reserves - I consent to the release of the 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Jodi Wright MAusIMM(CP) (#209552)

18/10/2017

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Signature of Witness:

Nigel Thiel
Melbourne, VIC

Witness Name and Residence: (e.g. town/suburb)

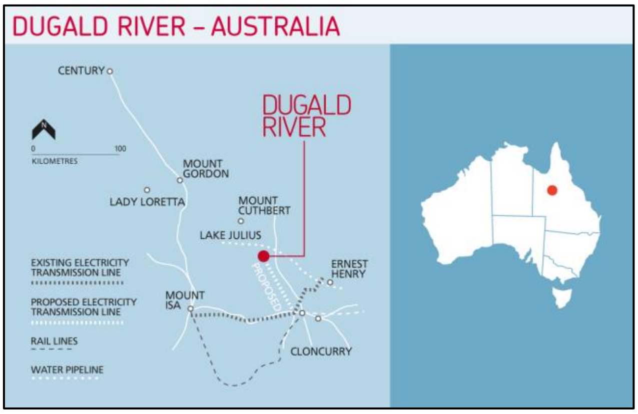
6 DUGALD RIVER PROJECT

6.1 Introduction and setting

The Dugald River project is located in northwest Queensland approximately 65km northwest of Cloncurry and approximately 85km northeast of Mount Isa (Figure 6-1) It is approximately 11km (by the existing access road) from the Burke Developmental Road, which runs from Cloncurry to Normanton.

It is one of the world's largest undeveloped zinc-lead-silver deposits containing 65Mt at 12% Zn, 2.2% Pb and 31 g/t Ag (as of 30 June 2017 at a $134/t NSR cut-off) and is wholly owned by a subsidiary of MMG Limited.

Figure 6-1 Dugald River project location



6.2 Mineral Resources – Dugald River

6.2.1 Results

The 2017 Dugald River Mineral Resources are summarised in Table 17. The Mineral Resources has been depleted to account for mining of ore by way of underground development of ore drives and stope production. The 2017 Mineral Resources has been reported above an A$134/t NSR *(net smelter return*) cut-off.

Table 17 2017 Dugald River Mineral Resources tonnage and grade (as at 30 June 2017)

Dugald River Mineral Resources								Contained Metal			
	Tonnes (Mt)	Copper (% Cu)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	Copper ('000)	Zinc ('000)	Lead ('000)	Silver (Moz)	Gold (MoZ)
Primary Zinc[1]											
Measured	8.1		13.1	2.4	70			1,065	197	18	
Indicated	29		12.3	2.3	40			3,567	669	38	
Inferred	28		11.4	1.9	10			3,176	531	8	
Total	**65**		**12.0**	**2.2**	**31**			**7,808**	**1,397**	**64**	
Stockpiles											
Measured	0.2		10.8	1.7	49			25	3.9	0.4	
Total	**0.2**		**10.8**	**1.7**	**49**			**25**	**3.9**	**0.4**	
Total Primary Zinc	**65**		**12.0**	**2.2**	**31**			**7,833**	**1,401**	**65**	
Primary Copper2											
Inferred	4.4	1.8				0.2	79				0.03
Total	**4.4**	**1.8**				**0.2**	**79**				**0.03**
Dugald River Total							**79**	**7,833**	**1,401**	**65**	**0.03**

[1] $134/t NSR Cut-off, in-situ (less depletion and oxide material)
[2] 1% Cu Cut-off, in-situ (less depletion and oxide material)
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Contained metal does not imply recoverable metal

6.2.2 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 18 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 18 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Dugald River Mineral Resources 2017

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

Diamond drilling (DD) was used to obtain an average 1m sample length while still respecting geological contacts. DD core was sampled either whole, ¾, ½, ¼. Once samples are selected by a geologist the samples are marked and the allocated sample ID's stored in the database.
Table below shows samples collected at Dugald River for use in the Mineral Resources model by drill type, hole size and sample type.

<table>
<tr><th>Drill Type</th><th>Hole Size</th><th>Sample Type</th><th>Metres</th><th>% of Total</th></tr>
<tr><td rowspan="23">DD & DD_UG</td><td rowspan="2">BQ</td><td>Whole Core</td><td>204</td><td>0.30%</td></tr>
<tr><td>1/2 Core</td><td>39</td><td>0.06%</td></tr>
<tr><td rowspan="2">LTK60</td><td>Whole Core</td><td>1,838</td><td>2.70%</td></tr>
<tr><td>1/2 Core</td><td>2,878</td><td>4.23%</td></tr>
<tr><td rowspan="3">NQ3</td><td>Whole Core</td><td>6</td><td>0.01%</td></tr>
<tr><td>1/2 Core</td><td>1,338</td><td>1.97%</td></tr>
<tr><td>Unknown</td><td>158</td><td>0.23%</td></tr>
<tr><td rowspan="3">NQ</td><td>Whole Core</td><td>3,182</td><td>4.68%</td></tr>
<tr><td>1/2 Core</td><td>206</td><td>0.30%</td></tr>
<tr><td>1/4 Core</td><td>42</td><td>0.06%</td></tr>
<tr><td rowspan="3">NQ2</td><td>Whole Core</td><td>5,243</td><td>7.71%</td></tr>
<tr><td>1/2 Core</td><td>31,060</td><td>45.67%</td></tr>
<tr><td>1/4 Core</td><td>288</td><td>0.42%</td></tr>
<tr><td rowspan="4">HQ</td><td>Whole Core</td><td>2,138</td><td>3.14%</td></tr>
<tr><td>3/4 core</td><td>409</td><td>0.60%</td></tr>
<tr><td>1/2 Core</td><td>1,041</td><td>1.53%</td></tr>
<tr><td>1/4 Core</td><td>296</td><td>0.43%</td></tr>
<tr><td>HQ3</td><td>1/2 Core</td><td>5,117</td><td>7.52%</td></tr>
<tr><td rowspan="2">PQ</td><td>Whole Core</td><td>255</td><td>0.37%</td></tr>
<tr><td>Unknown</td><td>7</td><td>0.01%</td></tr>
<tr><td rowspan="2">Unknown</td><td>Whole Core</td><td>513</td><td>0.75%</td></tr>
<tr><td>1/2 Core</td><td>444</td><td>0.65%</td></tr>
<tr><td colspan="2">DD TOTAL</td><td>56,703</td><td></td></tr>
<tr><td rowspan="4">RC</td><td>5 ¾ inch</td><td>Chips</td><td>1,772</td><td>2.61%</td></tr>
<tr><td>100 & 150 mm</td><td>Chips</td><td>1,732</td><td>2.55%</td></tr>
<tr><td>Unknown</td><td>Chips</td><td>7,801</td><td>11.47%</td></tr>
<tr><td colspan="2">RC TOTAL</td><td>11,305</td><td></td></tr>
<tr><td>TOTAL</td><td colspan="2"></td><td>68,008</td><td>100%</td></tr>
</table>

Approximately 16% of the dataset was sampled using RC drilling

</td></tr>
</table>

Assessment Criteria	Commentary
	techniques; however this is confined to pre-collar surface drilling and generally from regions outside of the mineralised zone. Approximately 22% of the total drilled meters were sampled.

- Since 2010, samples are bagged, numbered and dispatched to ALS Mt Isa laboratory:
  - Until 2016 ,the sample was jaw crushed, 50% split,
  - Crushed using a Boyd crusher 70% nominal passing 2mm.
  - The sample is rotary split with 500g-800g retained and pulverised to 85% passing 75µm.
  - All rejected material is collected and saved (Coarse – jaw crushed product, collected 2010 to 2016).
  - Pulps are then sent to ALS Brisbane for analysis.
- For the 2007/2008 program laboratory sample preparations involved drying, crushing (jaw and Boyd), and pulverising the ½ core sample to 85% passing 75µm.
- No detailed information can be found for laboratory preparation prior to 2007, however a similar procedure is assumed.
- Various laboratories have been used over time, and have been summarised in the table below (over 83% of all assays have been through the ALS laboratories.

Date Range	Laboratory	Number of Samples	%
2010 to current	ALS	43,292	63.66%
2001 to 2010	ALS	13,378	19.67%
	Amdel	239	0.35%
	Genalysis	439	0.65%
	Unknown	***139***	***0.20%***
Prior to 2001	AAL	267	0.39%
	Amdel	1,920	2.82%
	Aminya	238	0.35%
	Analabs	2,667	3.92%
	Pilbara	3,511	5.16%
	SGS	174	0.26%
	UNE	8	0.01%
	Unknown	***1,736***	***2.55%***
TOTAL		**68,008**	**100.00%**

- Since 2010 the four acid the digestion process used (ALS Brisbane) is as follows:
  - Approximately 0.25g of sample catch weighed into a Teflon test tube.
  - HNO_3 and $HClO_4$ are added and digested at 115°C for 15 minutes.
  - HF is added and digested at 115°C for 5 minutes.
  - The tubes are then digested at 185°C for 145 to 180 minutes. This

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>
takes the digest to incipient dryness (digest is not "baked")
o 50% HCl is added and warmed
o Made to 12.5ml using 9.5ml 11% HCl.
• The table below summaries the analytical method and digest used for all assays in the Mineral Resource estimate. Tthe majority of assays have been determined by using a four acid digest with an ICP OES read.
<table>
<tr><th colspan="2" rowspan="2">Base Metal Analysis</th><th colspan="4">Analytical Method</th><th rowspan="2">Total</th></tr>
<tr><th>ICP</th><th>AAS</th><th>XRF</th><th>Unknown</th></tr>
<tr><td rowspan="6">Digest</td><td>Four Acid</td><td>51,201</td><td>4,482</td><td>-</td><td>-</td><td>55,683</td></tr>
<tr><td>Aqua Regia</td><td>5,947</td><td>-</td><td>-</td><td>-</td><td>5,947</td></tr>
<tr><td>Mixed Acid</td><td>466</td><td>190</td><td>-</td><td>-</td><td>656</td></tr>
<tr><td>Perchloric</td><td>-</td><td>87</td><td>-</td><td>-</td><td>87</td></tr>
<tr><td>LBF (Fusion)</td><td>-</td><td>-</td><td>87</td><td>-</td><td>87</td></tr>
<tr><td>Unknown</td><td>131</td><td>275</td><td>8</td><td>1,238</td><td>1,652</td></tr>
<tr><td colspan="2">Total</td><td>57,745</td><td>5,034</td><td>95</td><td>1,238</td><td></td></tr>
</table>
• There are no inherent sampling problems recognised.
• Measures taken to ensure sample representivity include orientation of the drillholes as close as practical to perpendicular to the known mineralised structure, and the collection and analysis of field duplicates.
</td></tr>
<tr><td>Drilling techniques</td><td>
• Drilling used for the Mineral Resource estimates started in 1969 and continued until present. Within the database used there are 1388 drillholes (472 from surface {combination of Percussion / RC and DD} and 915 DD underground), summarised in the table below.
• Approximately 8% of the surface drilling data does not have drillhole diameters recorded.
<table>
<tr><th>Hole Type</th><th>Hole Diameter</th><th>Meters</th><th>% of Total</th></tr>
<tr><td>DD</td><td>BQ</td><td>2,993</td><td>0.97%</td></tr>
<tr><td>DD</td><td>HQ</td><td>26,976</td><td>8.73%</td></tr>
<tr><td>DD</td><td>HQ2</td><td>501</td><td>0.16%</td></tr>
<tr><td>DD</td><td>HQ3</td><td>19,786</td><td>6.41%</td></tr>
<tr><td>DD</td><td>LTK60</td><td>1,856</td><td>0.60%</td></tr>
<tr><td>DD</td><td>NQ</td><td>44,246</td><td>14.33%</td></tr>
<tr><td>DD</td><td>NQ2</td><td>41,525</td><td>13.45%</td></tr>
<tr><td>DD</td><td>NQ3</td><td>2,968</td><td>0.96%</td></tr>
<tr><td>DD</td><td>PQ</td><td>703</td><td>0.23%</td></tr>
<tr><td>DD</td><td>Unknown</td><td>14,908</td><td>4.83%</td></tr>
<tr><td>DD_UG</td><td>LTK60</td><td>16,620</td><td>5.38%</td></tr>
<tr><td>DD_UG</td><td>NQ2</td><td>108,613</td><td>35.17%</td></tr>
<tr><td>DD_UG</td><td>NQ3</td><td>148</td><td>0.05%</td></tr>
<tr><td>PD</td><td>5 "</td><td>83</td><td>0.03%</td></tr>
<tr><td>PD</td><td>6 ¼ "</td><td>12</td><td>0.004%</td></tr>
</table>
</td></tr>
</table>

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>

PD	**Unknown**	**1,296**	**0.42%**
RC	100 mm	1,612	0.52%
RC	150 mm	114	0.04%
RC	5 ¾ "	15,972	5.17%
RC	**Unknown**	**7,912**	**2.56%**
TOTAL		**308,844**	**100%**

DD = Surface diamond drilling, **DD_UG** = Underground diamond drilling, **PD**= Percussion drilling

- Some Historical holes (drilled prior to 1969) and other listed drillholes were not used for this estimate due to poor sample quality and reliability.
</td></tr>
<tr><td>Drill sample recovery</td><td>

- Recovery recorded during core logging was generally 100%, with minor losses in broken / sheared and faulted ground.
- At times, triple tube drilling from surface has been used to maximise core recovery but this is not common.
- RQD (rock quality designation) data was logged and recorded in the database to measure the degree of jointing or fractures or core loss in the sample.
- Shearing and broken ground zones are located at the edges of the mineralisation zone and are not associated with locations of good grade interceptions. There is no relationship between core loss and mineralisation or grade - no sample bias has occurred due to core loss within broken/sheared ground.
</td></tr>
<tr><td>Logging</td><td>

- All core samples including RC pre-collars have been geologically logged (lithology, stratigraphy, weathering, alteration, geotechnical characteristics) to a level that can support the Mineral Resources estimation.
- The logging captures both qualitative (e.g. rock type, alteration) and quantitative (e.g. mineral percentages) characteristics. Core photographs are available for most drillholes. All drillholes post-2008 have been photographed (wet and dry).
- A representative sample of mineralised core is stored at -4°C in refrigerated containers to minimise oxidation for metallurgical testing. Non-mineralised core is stored on pallets in the yard.
- Currently, all drillholes are logged using laptop computers directly into the drillhole database. Logging has occurred in the past onto paper log-sheets and was then transcribed into the database.
</td></tr>
<tr><td>Sub-sampling techniques and sample preparation</td><td>

- Prior to 2007 various sub-sample techniques and sample preparation techniques were used for DD drilling including whole, ¾ (generally restricted to metallurgical samples) and ½ and ¼ (for general samples) core, where sample length is generally 1 metre. Since 2007 DD core was halved using a circular diamond saw, density tested before being sent to analytical testing. Sample lengths were cut as close to 1m as possible while respecting geological contacts. From 2016 whole core is sent for analysis for any in-fill drilling campaigns.
- For DD, the standard sampling length is 1m with a minimum of 0.2m and a maximum 1.5m within the mineralised zone was determined by lithology and visible mineralisation (i.e., samples were taken up to but not across
</td></tr>
</table>

Assessment Criteria	Commentary

	lithological contacts, and obvious high grade zones were sampled separately from lower grade intervals).
	• The sample collection protocol for RC grade control drillholes has typically been as follows;
	  ○ RC samples are collected from a cyclone at 2m intervals from pre–collar surface drilling.
	  ○ If the sample was dry the sample was passed through a riffle splitter and collected into a pre-numbered calico bag. Residual material was sampled and sieved for chip trays and the remainder returned to the larger poly-weave bag. The splitter was cleaned using compressed air or a clean brush and tapped using a rubber mallet. If the sample was wet then the sample was dried before being split according to the procedure above (for dry samples).
	  ○ Historical RC programs were designed to test the 'un-mineralised' hanging wall material in DD pre-collars. 2m bulk composites stored at the drill site were sampled using the spear method.
	• The vast majority of drillhole intersections are orthogonal to the mineralisation and as such are representative.
	• The sample preparation of RC chips and DD core adheres to industry good practice. Samples are bagged, numbered and dispatched to the ALS Mount Isa laboratory. At the laboratory, each sample is weighed then crushed using a Boyd crusher to 70% nominal passing 2mm. The sample is rotary split (500g-800g) and pulverised to in a LM2 to 85% passing 75µm. All rejected material is collected and saved. Pulps are then sent to ALS Brisbane for analysis.
	• Prior to 2013, measures taken to ensure sampling is representative of the in-situ material collected included:
	  ○ Field duplicates (quarter core) were sampled at a rate of 1 per 20 samples (approximately 4 per drillhole).
	• The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Dugald River mineralisation (sediment/shear hosted base metal) by the Competent Person.

Quality of assay data and laboratory tests	• The assaying methods currently applied at Dugald River are ICP-MS with a 4-acid digest which is used for the analysis of Zn, Pb, Cu, Ag, Mn and As which are estimated in the Mineral Resource. Total carbon (TotC) is analysed by Leco furnace. All of these analyses are considered total. • These assaying techniques are considered suitable for the Dugald River Mineral Resources. • No geophysical tools, spectrometers or handheld XRF instruments have been used in the analysis of samples external to the ALS laboratory for the estimation of Mineral Resources. • Certified reference materials (CRM) and blanks (coarse) were each submitted at the rate of 1:20. The selection and location of standards and

Assessment Criteria	Commentary
	blanks in the batch sequence is decided by the geologist on the basis of the logged mineralisation. • Batches that fail quality control criteria (such as standards reporting outside set limits) are entirely re-analysed. • Prior to 2015 duplicate sampling was performed by selecting from returned coarse rejects and resubmitted to ALS for analysis. • Since 2015 duplicates are taken by the laboratory every 20th sample alternating between a duplicate taken at the primary crushing stage or the pulverisation stage. Batches that return standard values above three standard deviations (3SD) are failed and all or part of the batch is re-analysed by the Laboratory (ALS).
Verification of sampling and assaying	• Verification of assay results was visually verified against logging and core photos by alternative company personnel. • No twinning of drillholes have occurred at Dugald River. However, close-spaced and crossing holes give comparable grade and width results • Core logging data was recorded directly into a Database (Geobank) by experienced geologists (geological information such as lithology and mineralisation) and field technicians (geotechnical information such as recovery and RQD). Where data was deemed invalid or unverifiable it was excluded from the Mineral Resources estimation. • No adjustments to the assay data is performed during import into the Geobank Database.

Location of data points	• All drillhole collars have been surveyed by licensed surveyors. Surface collars were surveyed in MGA94 and then converted to local mine grid.
	• Underground drillholes are marked up by surveying a collar pin at the designed collar point location which is supplied by the Geologists.
	    o Currently the drillers obtain their azimuth for the hole by utilising an azimuth aligner which is calibrated weekly using a test bed that is has a fixed azimuth.
	    o Upon completion of the drill program the collars of each drillhole are surveyed in local grid and saved into the drillhole register spreadsheet for the Geologists.
	    o The equipment used underground to perform drillhole surveys is a Leica TS-15 total station.
	• For surface holes a collar point is marked out with a survey peg and then two pegs at the extremities of the drill pad are surveyed for the azimuth of the hole.
	    o The drill rig lines up with these two pegs to drill on correct azimuth.
	    o The drillers also use a true north azimuth tool to check the bearing.
	    o The equipment used on surface for drillholes is a Trimble R8 RTK GPS.
	• Down-hole surveying has been undertaken using various methods

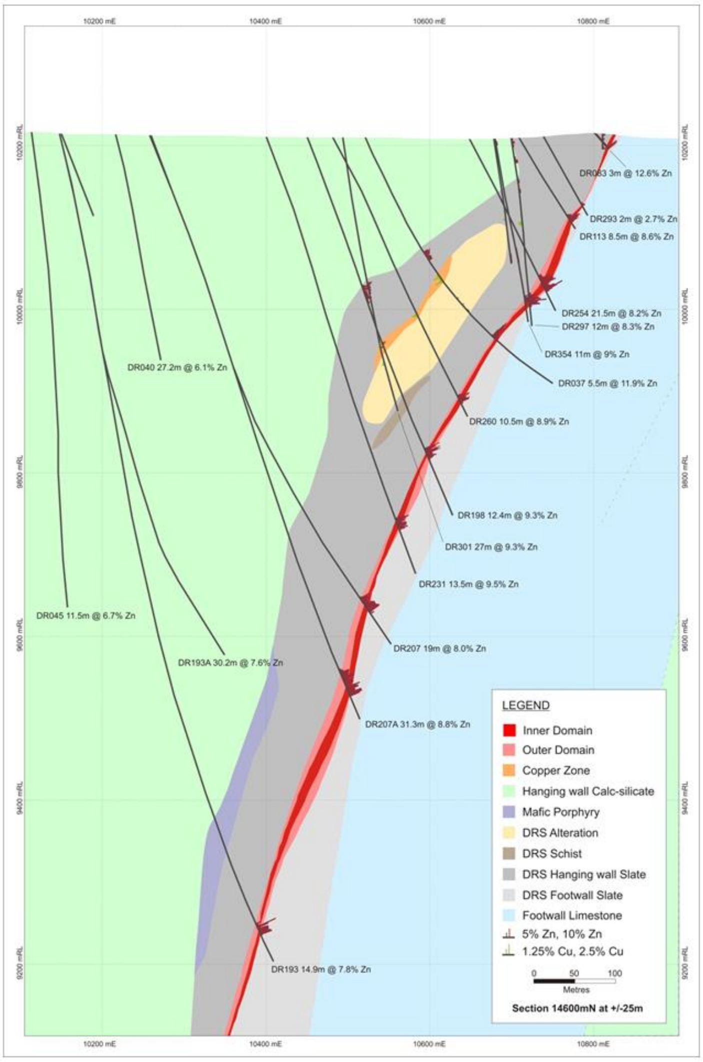
Assessment Criteria	Commentary
	including Eastman, Reflex and gyroscopic cameras. In general a spacing of 30m down-hole between survey readings was used. Interference due to magnetite and pyrrhotite has been an issue. o During the 2007/2008 drilling program routine survey checks were undertaken on the reflex survey camera using an aluminium calibration stand positioned in a known orientation. o Since 2008 all drillholes are gyroscopically surveyed. • The grid system used is MGA94, the conversion to local mine grid is rotated and scaled. The grid transformation is undertaken using a formula provided by the onsite surveyors. • A LIDAR survey flown in 2010 is used for topographic control on surface drilled drillholes. In the view of the Competent Person the LIDAR survey provides adequate topographic control.
Data spacing and distribution	• Drill spacing varies across the strike and dip of the mineralisation lode. The highest drill density in the orebody is 10m x 10m while the lowest drill density is greater than 100m x 100m spacing. • Locations drilled at 10m x 10m and up to 20m x 20m are adequate to establish both geological and grade continuity. Wider spaced drilling is adequate for definition of broader geological continuity but not sufficient for accurate grade continuity. • Underground mapping of faces is digitised and used in the interpretation and wireframing process. • Drillhole data is concentrated within the top 300m of the Mineral Resources with broader-spaced drilling at depth, due to the difficulty and cost involved in drilling deeper sections. • Drill spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resources estimation procedure and classification applied. • Samples are not composited prior to being sent to the laboratory for analysis.
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 sampling bias. Drillholes that have been drilled down dip and sub-parallel to the mineralisation have been excluded from the estimate.
Sample security	• Measures to provide sample security include: o Adequately trained and supervised sampling personnel. o Well maintained and ordered sampling sheds. o Cut core samples stored in numbered and tied calico sample bags.

Assessment Criteria	Commentary
	○ Calico sample bags transported by courier to assay laboratory. ○ Assay laboratory checks of sample dispatch numbers against submission documents. ○ Assay data is returned as a .sif file via email and processed via the MMG assay loading software.
Audit and reviews	• The Dugald River database has been housed in various SQL databases. iOGlobal managed the database until the end of 2009 when the database was transferred and migrated to an MMG managed database. ○ Internal audits and checks were performed at this time. Any suspicious data was investigated and rectified or flagged and excluded. • No external independent audits have been performed on the database. • No external independent audits have been performed on the sampling techniques or the database. • Both ALS Mount Isa and Brisbane laboratories are audited on an annual basis by MMG personnel. From the most recent audit there were no material recommendations made.

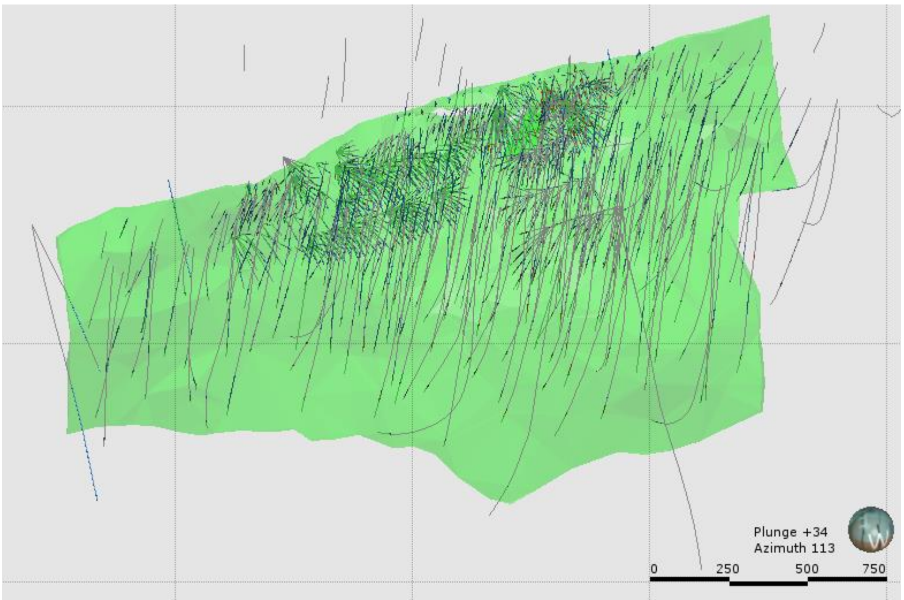
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	• The Dugald River Mining Leases are wholly owned by a subsidiary of MMG Limited. • MMG holds one exploration lease and one mineral development lease in addition to the mining leases on which the Dugald River Mineral Resources 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. • There are no known impediments to operating in the area.
Exploration done by other parties	• The History of the Dugald River zinc-lead deposit is summarised as follows: Discovered in 1881, the first drilling program in 1936 comprised three drillholes. The maiden Mineral Resources was reported in 1953 by Zinc Corporation. Drilling continued from 1970 through 1983 totalling 28 drillholes. CRA then re-estimated the Mineral Resources in 1987. Between 1989 and 1992 a further 200 drillholes were drilled, resulting from the discovery of the high-grade, north plunging shoot. Infrastructure, metallurgical and environmental studies were undertaken during this period. Between 1993 and 1996 irregular drilling was focused on the delineation of copper mineralisation in the hanging wall. In 1997 the project was transferred to Pasminco, which had entered a joint venture with CRA in 1990. Recompilation of the database, further delineation drilling, metallurgical test work, and the check assaying of old pulps was completed. Continued drilling between 2000 and 2009 with subsequent metallurgical studies culminated in a Feasibility Study. Structural analysis and a focused

	review on the northern copper zone in 2010 were completed. In 2011 the decline commenced which resulted in trial stoping. In 2014 some underground development was completed and drilling focused on confirming and extending continuity of the mineralisation within the Dugald River lode.
Geology	• The Dugald River style of mineralisation is a sedimentary and shear hosted base metal deposit. The main sulphides are sphalerite, pyrite, pyrrhotite and galena with minor arsenopyrite, chalcopyrite, tetrahedrite, pyrargyrite, marcasite and alabandite. • The deposit is located within a 3km-4km along strike north-south trending high strain domain named the Mount Roseby Corridor and is hosted by steeply dipping mid Proterozoic sediments of the Mary Kathleen Zone in the Eastern Succession of the Mount Isa Inlier. The host sequence is composed of the Knapdale Quartzite and the Mount Roseby Schist Group (which includes the Hangingwall calc-silicate unit, the Dugald River Slate and the Lady Clayre Dolomite). The sequence is an interbedded package of greenschist to amphibolite grade metamorphosed carbonate and siliclastic lithologies. • The main Dugald River lode is hosted within a major N-S 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, slatey breccia, and massive breccia. • The mineralogy of the Dugald River lode is typical of a shale-hosted base metal deposit. 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 mineralised zone extends approximately 2.4km in strike length and up to 1.4km down-dip.
Drillhole information	• 1,388 drillholes and associated data are held in the database. • No individual drillhole is material to the Mineral Resource estimates and hence this geological database is not supplied.
Data aggregation methods	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section. • No metal equivalents were used in the Mineral Resources estimation. However the Mineral Resources has been reported above an A$134 NSR calculated cut-off.
Relationship between mineralisation width and intercepts lengths	• Mineralisation true widths are captured by three-dimensionally modelled wireframes with drillhole intercept angles ranging from 90° to 40°. • The true thickness of the majority of the Mineral Resources is between 3m and 30m with the thickest zones occurring to the south.

Diagrams



Cross section looking north – showing true width of the mineralisation



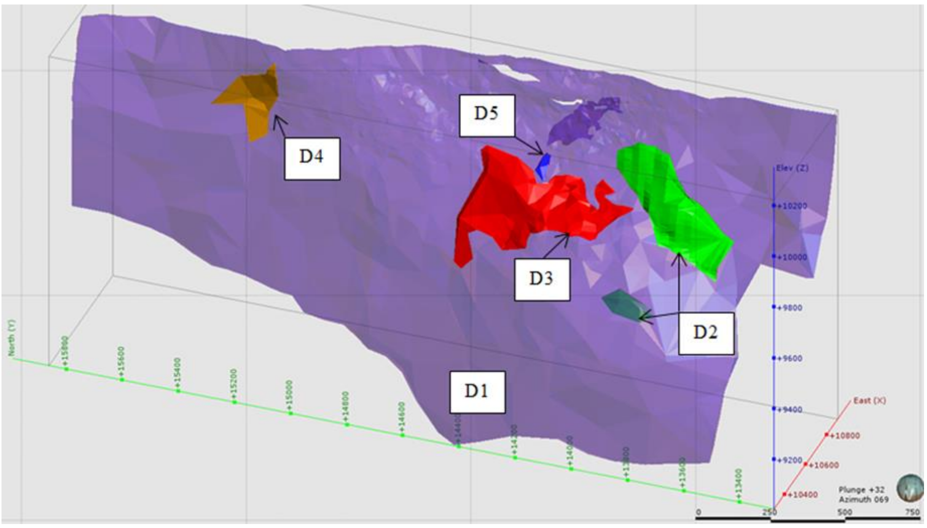
Perspective View – looking SE of main mineralised zone will drill density

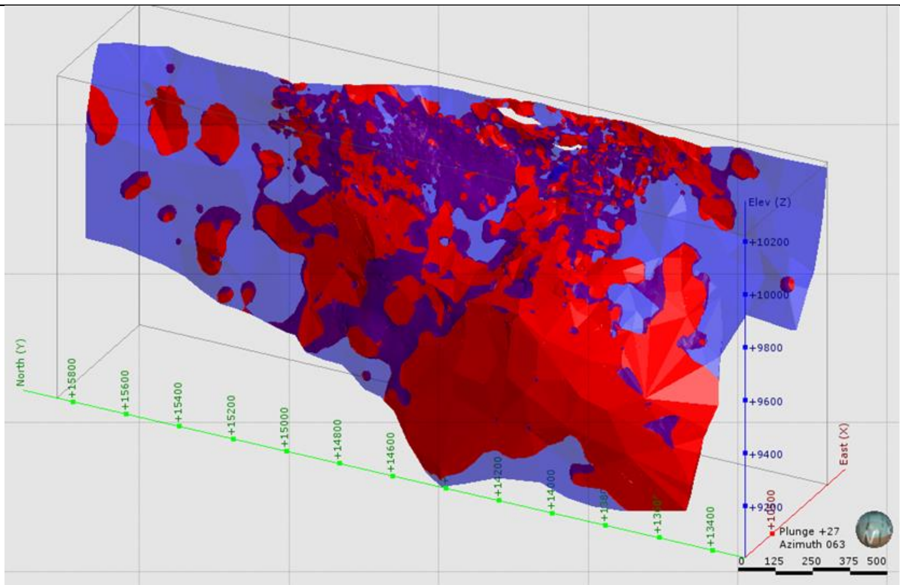
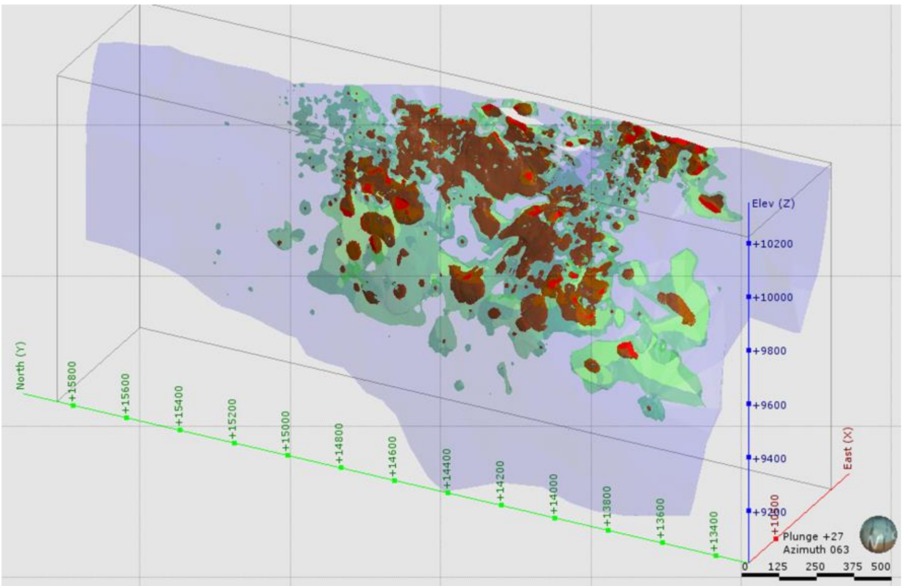
Balanced reporting	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section.
Other substantive exploration data	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	• MMG plans to continue to improve geological confidence and Mineral Resources classification through infill drilling programs ahead of the mining schedule.

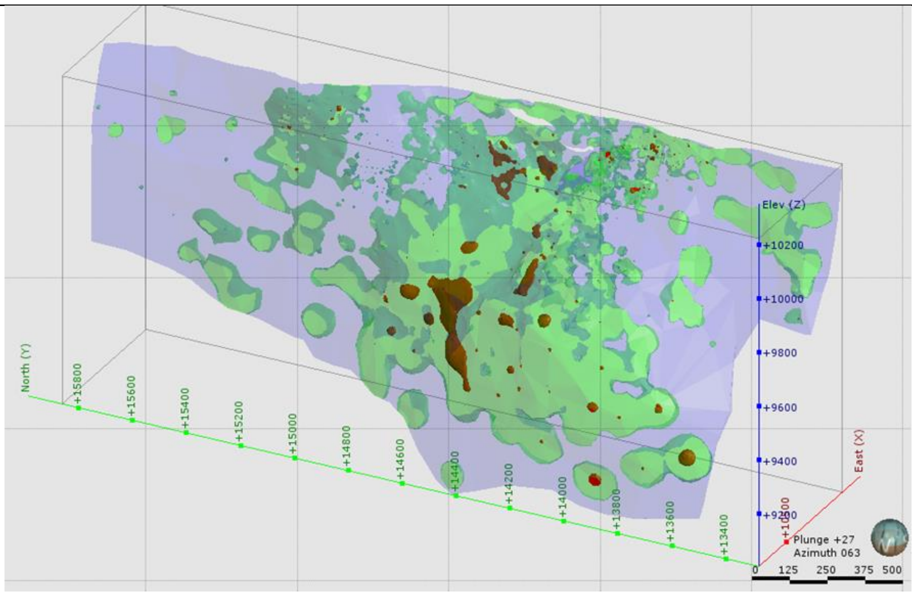
Section 3 Estimating and Reporting of Mineral Resources	

Database integrity	• The following measures are in place to ensure database integrity:
	  o All data is stored in an SQL database that is routinely backed up.
	  o All logging is digital and directly entered into the onsite Geobank database. Data integrity is managed by internal Geobank validation checks/routines that are administered by the database administrators and/or the site Geology Team.
	• The measures described above ensure that transcription or data entry errors are minimised.
	  o Data validation procedures include: Database validation procedures are built in the database system to manage accurate data entry during logging and collection of data.
	  o Prior to use in the Mineral Resources the data was checked externally by running Datamine macros on the drillhole file to check for end of hole depths, and sample overlaps.
	  o 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 2016 and 2017. Site visits included involvement with:
	  o Assist with wireframe interpretation and methodology as applied in the 2017 Mineral Resources work.
	  o Inspection of geological mapping plans.
	  o Inspection of underground workings.
	  o Inspection of drillholes and mineralisation interceptions.

Geological interpretation	• The mineralisation zone is modelled within a continuous corridor of zinc mineralisation. This zone is modelled based on zinc grade distribution and geological logging of mineralisation style. The mineralised envelope is determined by natural breaks in the grade distribution. There is good confidence on the geological continuity and interpretation of the deposit. o The mineralisation zone is further sub-divided into a high- and low-grade domain. o The "inner" high grade domain is the main Dugald River mineralisation lode, defined by high zinc grades associated with the massive sulphide assemblages. The high grade domain boundary was selected by looking at drillholes grade intercepts in Datamine; plotted as a grade histogram and selecting a boundary which is more representative of geology. o 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. o Where possible a low grade (internal dilution) domain has been identified and modelled within the high grade domain. o There are 4 other smaller, sub-parallel zinc domains which have been identified with more closely spaced drilling, which generally follow the main mineralised lode (Domain 1) or structures associated with it.  **Perspective View - Five Low Grade Zinc Domains (High Grade Zinc domains are within the low Grade Zinc Domains; D4 – is a Low Grade Zinc Domain only)** • Separate domains were modelled for Pb, Ag and Mn mineralisation, after exploratory data analysis (EDA), have shown these elements are possibly due to a secondary mineralisation event; and are contained within the "outer" lower grade zinc domain.

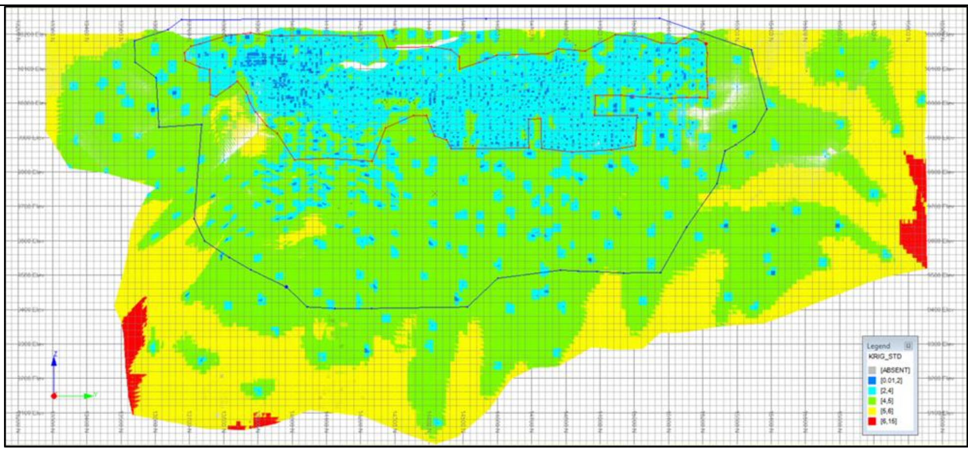
	 **Pb Domains – D1 (Blue – Low Grade / Red –High Grade)**  **Ag Domains – D1 (Blue – Low Grade/ Green – Medium Grade / Red – High Grade)**

	 **Mn Domains – D1 (Blue – Low Grade/ Green – Medium Grade / Red – High Grade)** • Selection of the low/high grade zinc domain was based on geological observations and natural breaks within the mineralisation. An approximate zinc grade of 2% to 7% Zn was used to define the "outer" low grade zinc domains and a grade of greater than 7% Zn was used to define the "inner" high grade zinc domain. • Underground mapping of development drives for both access and ore drives were also used in assisting with the geological interpretation. • Globally the Dugald River deposit follows a reasonably predictable lens/sheet of mineralisation but with short-range (10m to 20m-scale) variations associated with localised structures that are suitably defined by close-spaced drilling within the Measured Mineral Resources.
Dimensions	• The main Dugald River lode (Domain 1) is hosted within a major N-S 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. Dip varies between 85° and 40° to the west. • The true thickness of the majority of the Mineral Resources is between 3m and 30m with the thickest zones occurring to the south. • The mineralisation is open at depth. The deepest drill intersection of mineralised material is about 1,140m below the surface.
Estimation and modelling techniques	• Mineral Resources modelling was completed using both Isatis and Datamine software applying the following key assumptions and parameters: o Ordinary Kriging interpolation has been applied for the estimation of Zn, Pb, Ag, Mn, Fe, S, total carbon and Bulk Density. This is considered appropriate for the estimation of Mineral Resources at Dugald River. o Extreme grades were treated by grade capping and were applied after compositing, with values greater than the selected 'cut value' being set to the top cut value and used in the estimation. Grade

	cap values were selected using a combination of both histogram and cumulative log probability plots. o Grade estimation was performed using local anisotropy, which uses the dip and dip direction of 11 different estimation zones to align and optimise the search direction of the estimate. o Hard boundary contacts were used to select samples used to estimate blocks in each of the mineralised domains (high grade and low-grade) as well as into individual domains for Ag, Mn and Pb. o Variogram were modelled within each of the respective domain, these variogram ranges were then applied to the search parameters used in the estimation. o Orientation of the search ellipse was matched to the local anisotropy, that is dip and dip direction of the 11 estimation zones was used in the estimation of the model. o Drillhole compositing resulted in nominal 1m intervals with residual composite intervals absorbed evenly into the composites resulting in no loss of sample intervals. o Separate variography and estimation were performed for Zn, Pb, Ag, Mn, Fe, S, bulk density and total carbon, within each of their respective mineralised domains. o No assumptions have been made about the correlation between variables. All variables are comparably informed and independently estimated. o Interpolation was undertaken in two stages: – Stage1: Ordinary Kriging applying three passes with varying search ellipse dimensions. – First pass is equal to 80 - 100% of the variogram range. – Second pass is equal to 2 x variogram range. – Stage 2: assign blocks not estimated by the Ordinary Kriging 2 passes, the average grade of the respective domain. o A minimum number of 2 drillholes were used for all estimates. o Generally, the number of composite samples was restricted to a minimum of 6 and a maximum of 28 (Kriging Neighbourhood Analysis (KNA), was used to locally adjust on a mineralised domain basis). o Octant method was generally not applied to the Ordinary Kriging estimate, however from the KNA work some sectors (max of 2) were used to improve the estimate in some of the mineralised domains. o Block discretisation of 2 x 8 x 8 was applied. • At the end of 2015 ~400Kt of Dugald River ore (mined from trial stoping) was transported to the Century mine (also owned by MMG at the time) to be processed via the Century plant. A good correlation was found against the predicted grade control data. • Assumptions have been made regarding the recovery of all by-products in

	the Net Smelter Return (NSR) script. • Deleterious elements include manganese and carbon, which have been estimated in the block model. Ancillary elements estimated include Mn, Fe and S. • Parent block size was set at 2.5m x 12.5m x 12.5m with sub-cells of x=0.5m, y=0.5m, z=0.5m. Sub-cells were assigned parent block values. The parent block size assumes mining selectivity at the stope level. • No external dilution has been applied to block grades. However, parent block size has assumed mining selectivity at a stope level. No other selective mining unit size assumptions were made in the estimation process. • 2017 block model validation included the following steps: o Comparison against the previous 2014 block model including visual comparison of plans and cross-sections, tonnes grade curves, cumulative probability plots and trend plots. o Comparison against drillhole data using visual comparison of plans and cross-sections, statistics by domain, cumulative probability plots and swath plots.
Moisture	• Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	• The Mineral Resources is reported above an A$134/t NSR (net smelter return) cut-off. The selection of the A$134/t NSR cut-off defines mineralisation which is prospective for future economic extraction. The reporting cut-off grade is in line with MMG's policy on reporting of Mineral Resources which is prospective for future economic extraction.
Mining factors or assumptions	• Mining at Dugald River is planned to be underground via 2 methods (i) sub-level open stoping with paste fill and (ii) Avoca with rock fill. Currently the deposit is accessed by two declines. • No external dilution has been applied to block grades. However, parent block size has assumed mining selectivity at a stope level. • The Mineral Resources has been depleted to account for mining.
Metallurgical factors or assumptions	• The metallurgy process proposed for the Dugald River deposit involves crushing and grinding followed by floatation and filtration to produce separate zinc and lead concentrates for sale. • Deleterious elements include manganese and carbon, which have been estimated in the block model. • Manganese percentage in the zinc concentrate is calculated as a post-processing step to allow the generation of a value that can be used for the Ore Reserves. • Manganese percentage in the zinc concentrate is calculated by way of an algorithm contained within the NSR script.
Environmental factors or assumptions	• 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. • Non-acid forming (NAF) and Potentially Acid Forming (PAF) waste rock are distinguished by a north-south striking Limestone and Undifferentiated

	Black Slate lithological boundary respectively. Volumes are estimated and managed in accordance with site procedures. Waste rock storage space on surface is limited. The north mine area will allow for the return of waste rock as backfill and the south mine will be backfilled with paste fill generated from tailings, once the processing plant is operational. • PAF/NAF classification is based on the work by Environmental Geochemistry International (EGI) in 2010. Subsequent follow-up test work onsite confirms EGI's conclusions.
Bulk density	• Bulk density is determined using the weight in air and water method. Frequency of samples is approximately 1 determination per core tray and based on geological domains. • Dugald River rock is generally impermeable requiring no coatings for reliable measurements. • Bulk density in the model has been estimated using Ordinary Kriging. Density estimation is constrained within the defined mineralisation domains. • Un-estimated blocks were assigned a density value based on a stoichiometric formula which was used in the 2014 block model for bulk density calculations. • A density of 2.75g/cm^3 has been assumed for the waste host domain.
Classification	• 2017 Mineral Resources Classification incorporates a combination of Kriging variance (KV), Kriging efficiency (KE), Kriging slope of regression (RS), drilling density and location of underground development (presence of underground geological mapping). • Mineral Resource classifications are generally based on: o Measured: < 20m drill spacing, RS>0.85 plus grade control drilling. o Indicated: > 20m to <100m drill spacing, RS<0.6. o Inferred: > 100m drill spacing, within mineralised domain • The Competent Person reviewed the distribution of KV, KE and RS in long section view and generated three-dimensional 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 x 20m underground drilling. The use of wireframes ensured that contiguous areas of like classification were generated thus avoiding the "spotted dog" pattern of classified blocks. • The Mineral Resources classification reflects the Competent Persons view on the confidence and uncertainty of the Dugald River Mineral Resources. • Below is a long section looking east of the Dugald River mineralised lode showing blocks coloured by Kriging Standard Deviation (KSD) and the Measured, Indicated and Inferred wireframes used in selecting the Mineral Resources classification. o Measured = red o Indicated = grey o Inferred = other

<table>
<tr><td></td><td>

Long-section of the Dugald River Block Model, blocks coloured by KSD</td></tr>
<tr><td>Audits or reviews</td><td>No external independent audits have been performed in 2017. However, two external audits have been completed in recent years including Pennywise Ltd in 2015 and Lewis Mineral Resource Consultants Pty Ltd in 2013. All issues identified have been corrected in the current Mineral Resources model.
An internal MMG review has been carried on the current 2017 Mineral Resource estimates. This involved personnel from both Corporate and Dugald River sites. No material items to the Mineral Resources have been identified.</td></tr>
<tr><td>Discussion of relative accuracy / confidence</td><td>The Competent Person is of the opinion that the current block estimate provides a good estimate of tonnes and grades at a global scale. In locations where grade control drilling of approximately 10mN x 20mRL spacing has been completed the Competent Person has a high level of confidence in the local estimate of both tonnes and grades.
No change of support adjustments have been performed to the model.
There is no actual production data to compare Mineral Resources confidence against actual mined tonnes and grades of the deposit.
Tonnes and grade checks comparing the 2014 and 2017 Mineral Resources models to check for tonnes and grade variability and accuracy as a function of increase drilling density has been undertaken. The following is noted:
Drilling density of <20m spacing is required to estimate tonnes and grade accuracy with an acceptable level of confidence.</td></tr>
</table>

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

This Mineral Resources 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, Douglas Corley, 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 Geoscientists
- 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 Ltd. at the time of the 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 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 the Dugald River 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 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

18/10/2017

Douglas Corley MAIG R.P.Geo. (#1505)

Date:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.

Anna Lewin
Melbourne, VIC

Signature of Witness:

Witness Name and Residence: (e.g. town/suburb)

6.3 Ore Reserves – Dugald River

6.3.1 Results

The 2017 Dugald River Ore Reserves are summarised in Table 19.

Table 19 2017 Dugald River Ore Reserves tonnage and grade (as at 30 June 2017)

Dugald River Ore Reserves							
					Contained Metal		
	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	Zinc ('000)	Lead ('000)	Silver (Moz)
Primary Zinc[1]							
Proved	7.7	11.8	2.1	62	911	165	15
Inferred	24.9	11.9	2.2	39	2,961	546	31
Total	**32.6**	**11.9**	**2.2**	**44**	**3,872**	**711**	**47**
Stockpiles							
Proved	0.2	10.8	1.7	49	25	4	0.4
Total	**0.2**	**10.8**	**1.7**	**49**	**25**	**4**	**0.4**
Total	**32.8**	**11.9**	**2.2**	**44**	**3,897**	**715**	**47**

Cut-off grade is based on Net Smelter Return (NSR) after Royalties, expressed as a dollar value of A$134/t.

Contained metal does not imply recoverable metal.

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

6.3.2 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 20 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code.

Table 20 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Dugald River Ore Reserves 2017

Assessment Criteria	Commentary
Mineral Resource estimates for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Ore Reserves. • The Ore Reserves estimation used the MMG Feb 2017 Mineral Resources model. (dr0217m_rese.dm) • Risks associated with the model are related to orebody complexity seen underground, but not reflected in the Mineral Resources model due to the spacing of the drill holes that inform the model. • The 2017 Geotechnical model was used to estimate the hanging wall (HW) thickness, tonnes and grade of the unplanned dilution applied to the 2017 stope shapes.
Site visits	• The Competent Person for the Dugald River Ore Reserves frequently visited the site during 2016/2017.

Study status	• The Dugald River study has progressively been enhanced.
	  o 2008/09 Feasibility Study Report (FS09).
	  o 2010 Feasibility Report Update which incorporated work from both Ausenco and AMC.
	  o 2012 Board Submission – basis for conditional project approval.
	  o 2013 Mining Method Review – recommendation for alternate mining assumptions.
	  o 2013 Business Options Review – reassessment of business case for the revised mining assumptions (2013BOR).
	  o 2014 Dugald River DRAFT Ore Reserves Supporting Document_04 November 14 (2014 Ore Reserves).
	  o 2015 Dugald River Updated Development Plan (2015 Ore Reserves).
	  o 2016 Revise Business – Board approval to proceed with development at a reduced capital cost and improved project NPV.
	  o 2017 Life of Asset (LoA), economic evaluation report for Short, Medium and Long term that include a full redesign of underground development and stope shapes.
	• There has been a series of reviews undertaken on operating and capital costs, infrastructure optimisation, trial mining has taken place and metallurgical studies are in progress.
	• With physical access into the orebody occurring in 2012 it was recognised that the orebody was more complex than modelled from drilling results and that the Geotechnical conditions of the orebody hanging wall were more challenging for dilution control

Assessment Criteria	Commentary

	than assumed in the 2009 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 hanging wall zones that was implemented in the 2017 redesign of the stope shapes.
	• Detailed design work, including scheduling and cost modelling, was undertaken by MMG for a 20m to 25m development level spacing and various stope strike length base on the hydraulic radius set by the Group Technical Services. The 2014 trial mining was based on a 25m level spacing as the upper levels of the mine were already developed. Stopes were initially mined 15m along strike, with larger stopes mined based on the experience gained. The initial results have indicated that a 25m level spacing and 20m strike length are achievable and was the basis of the 2014 Ore Reserves.
	• The trial mining campaign, which was completed by the end of 2014, tested various mining parameters. As a result the chosen mining configuration to be applied to the area outside of trial mining has changed. The North Mine is planned to be mined using a modified Avoca benching method together with rock fill. The South Mine will be mined using a Sub-level Open Stope mining method at 25m level spacing but with stope strike lengths varying between 15m and 30m, dictated by hanging wall conditions. This was used as the basis for the 2017 Ore Reserves.
	• The 2017 Life of Asset (LoA) schedule and processing throughput rate applied is 1.5Mtpa for 2018 and 2019 then 1.7Mtpa forward.
	• The main differences between the Feasibility Study and Ore Reserves 2013 – Ore Reserves 2016 Revised Business Case versus the 2017 LoA are;
	  o Mining the core wide and high grade orebody in block B, C and D to depth before start the narrow and medium grade ore body in block A and E.
	  o The upfront capital required to commence production is high.
	  o Link North and South Mine fresh and return air at the bottom panel of each panel before starting production.
	  o The 2017 Ore Reserves has detailed revision of the capital and operating costs.
	  o The 2017 Ore Reserves undertaken shows that the Ore Reserves is technically achievable and economically viable. The material modifying factors have been considered.

Cut-off parameters	• The breakeven cut-off grade (BCOG) and Mineral Resources cut-off grade (RCOG) have been calculated using budget 2017 cost. • The operating costs, both fixed and variable, have been attributed

Assessment Criteria	Commentary
	on a per tonne basis using the planned mine production rate of 1.7Mtpa. • The Net Smelter Return (NSR) values are based on metal prices, exchange rates, treatment charges and refinery charges (TC's & RC's), government royalties and metallurgical recoveries. • The NSR value for both BCOG and RCOG is to the mine gate and includes sustaining capital for the 2017 Ore Reserves. • Infill diamond drilling has been included as part of the growth capital and not as sustaining capital.

2017 (Average 2020 - 2029)		
Category of cut-off	**Units**	**AU$/t processed**
BCOG	Au$/t	134
SCOG	Au$/t	120
DCOG	Au$/t	97
ICOG	Au$/t	75
MCOG	Au$/t	48
RCOG	Au$/t	134

Assessment Criteria	Commentary

Mining factors or assumptions	• A detailed design 2017 Ore Reserves was used to report Mineral Resources conversion to Ore Reserves.
	• The 2017 Geotechnical model was used to estimate the hanging wall thickness, tonnes and grade of the planned dilution applied to the 2017 stope shapes.
	• The orebody is split into a north and south mine, due to its 2km strike length and a low-grade zone in the outer side of the orebody.
	• 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.
	• Dugald River will be mined using sub-level open stopes (SLOS) in the South Mine and Avoca stoping in the North Mine. Results from the trial mining has indicated the level interval of 25m and variable stope strike length of 15m to 30m is possible, although further work is required to determine if the 25m level interval is suitable for the flatter dipping areas.
	• The stopes are broken into the following categories:
	  o Longitudinal SLOS, for any stopes less than 11m wide horizontally.
	  o Transverse SLOS, made up of 20m strike SLOS mined full width of the ore-body.
	  o Crown pillar SLOS, for the top level of each panel where

Assessment Criteria	Commentary
	stoping occurs directly below a previous mined area. o Avoca stopes for the North Mine. • The stopes were created by applying the Mineable Shape Optimiser (MSO) software in Deswik CAD to the 2017 Mineral Resources that was created in Datamine. • The parameters used to create the stope shapes were: o All Mineral Resources categories included. o 25m level interval. o Variable strike length. o Minimum mining width (MMW) of 2.5m. o Minimum dip of 45 degrees for footwall lode and 37 degrees for hanging wall. o Minimum waste pillar between parallel stopes of 5m. o A$134/t BCOG applied to create initial stope shapes (not final cut-off). • No additional dilution applied (as the unplanned dilution is applied later in the evaluation process). • Several aspects of dilution were considered, planned dilution, fill dilution, footwall dilution and hangingwall dilution. Planned dilution was included in the MSO stope shapes and covers localised variations in dip and strike as well as minimum mining width. Footwall dilution was included where ore development was wider than the stope width. No additional footwall dilution was applied as the initial stope shapes took into account minimum mining widths and dip. • The hanging wall dilution was calculated for each stope based on the Geotechnical conditions and thicknesses of the hanging wall materials. The site has compiled a detailed hanging wall dilution model as dilution varies across the ore-body according to the hanging wall conditions. • An allowance was included for fill dilution. The stope recovery allowance covers stope under-break, ore loss into fill at the base of the stope and ore left in the base of the stope. Fill Dilution and Stope Recovery Factors are: o Floor 0.15m and wall fill ranges from 0.3m to 0.5m dilution. o Recoveries Crown stopes 65%, Longitudinal and Transverse 95%. • Development, grades were diluted by the application of a grade factor of 90% to all ore development grades estimated from the corresponding stope grades. • No Inferred Mineral Resources are included in the Ore Reserves. A sensitivity was done inclusive of Inferred Mineral Resources

Assessment Criteria	Commentary
	increasing the scheduled mine life from 24 years to 35 years. • 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. • Currently, three raisebored ventilation shafts are in place: o The southern Fresh Air Raise (FAR) – at 3.5m diameter and 130m depth;. o The southern Return Air Raise (RAR) – at 5.0m diameter and 198m depth. o The northern FAR at 3.5m diameter and 172m depth (currently being used as RAR). • There is also a RAR longhole winze (LHW) system in the south mine – at 18.9m square and 116m long and a RAR longhole winze system in the north mine at 18.9m square and 129m long. • Two escape raises are in place: o South mine escape way at 1.8m diameter and 222m depth o North mine escape way at 1.8m diameter and 93m depth • Mining mobile fleet is planned to include 3 twin-boom jumbos, 3 cable bolting rigs, 6 loaders, 8 dump trucks, 2 long-hole drill rigs, 2 shotcrete rigs, 3 transmixers, 4 charge-up vehicles, 3 raisebore drill rigs, 3 integrated tool carriers and light vehicle fleet.
Metallurgical factors or assumptions	• The metallurgical process proposed for treatment of Dugald River ore involves crushing and grinding followed by selective flotation to produce separate lead and zinc concentrates. This process is conventional for this style of mineralisation and is used world-wide. MMG operated the Century Mine in Queensland using an almost identical process. • The flow sheet has been extensively tested at bench scale with over 200 tests being completed on a wide range of samples. The results of these tests (a combination of both open and locked cycle tests) have been used to establish the metallurgical factors used for Ore Reserves estimations. Key parameters are: o Lead recovery to a lead concentrate according to the equation: Pb rec (%) = MIN (85, [244.295 – (7.858 × S%) – (5.171 × Fe%) – (13.689 × Mn%) – (1.599 × SiO2%) + (0.395 × (S% × Fe%))] × 1.26). o Silver recovery to a lead concentrate according to the equation: Ag rec (%) = 10.2 × ln(Ag) – 11.65. o Zinc recovery to a zinc concentrate according to the equation: Zn rec (%) = 399.85 – 29.75 × Zn% – 10.98 × Fe% – 4.74 × SiO2%+ 0.99 × (Zn% × Fe%) + 0.47 × (Zn% × SiO2%). o Iron assay of zinc concentrate according to the equation:

Assessment Criteria	Commentary

% Fe in Zn concentrate = 3.53 + 2.16 × Pb% + 0.23 × Fe% + 0.89 × Mn% + 0.52 × TOEC% – 0.16 × Zn% × Pb%.

  - Manganese assay of zinc concentrate according to the equation: % Mn in Zn concentrate = -3.36 + 0.36 × Zn% - 0.065 × Fe% + 2.78 × Mn% + 1.39 × TOEC% – 0.116 × (Zn% × TOEC%).
  - Where Zn%, Pb%, Fe%, S%, Mn%, C%, TOEC%, SiO2% and Ag refer to the relevant assays of the ore

- A full check has been completed for possible deleterious elements and the only two that are material to economic value are Fe and Mn in the Zn concentrate. It is for this reason that the algorithms to predict these components have been developed.
- As required, it is expected that the feed for flotation will be blended to maintain Mn (and to a lesser extent Fe) within the contractual range for concentrate sales.

Locked cycle testing, as a way to assess the continuous run in a laboratory scale, was performed with five different samples. The results were favourable, and are summarised in the Table below.

Sample	Pb Conc, % Pb		Pb Rec, %		Zn Conc, % Zn		Zn Rec, %	
	OCT	LCT	OCT	LCT	OCT	LCT	OCT	LCT
2014 bulk sample	61.2	59.6	67.7	79.5	55.6	52.5	88.8	92.8
DU0209	56.0	53.8	68.6	81.7	52.1	52.2	87.6	88.7
DU0279	61.4	65.3	69.5	89.6	50.8	51.4	87.0	89.0
DU0275	54.6	47.0	51.1	72.8	50.0	52.1	86.8	87.8
2015 bulk sample	61.9	46.5	52.2	73.7	54.8	52.1	89.7	91.7
Average	59.0	54.4	61.8	79.5	52.6	52.1	88.0	90.0

- Two separate plant trials were conducted for processing of Dugald River ore, using the existing facilities at the Century processing plant. The 1st plant trial was too short (run for only 5 days) to assess the scale up reliably. Of significance is the 2nd plant trial, which was carried out from late November 2015 to early January 2016. The samples for the trial, totalling about 458,000 tonnes, were all ore samples stockpiled at the Dugald River site. Metallurgical performance for zinc significantly exceeded plan, with recovery averaging 76.8% to a concentrate assaying 51.5% Zn compared with the plan of 72% recovery. Performance continually improved throughout the trial and averaged 80.3% for the last 17 days. Concentrate quality remained within specification throughout.
- Metallurgical performance for lead was unfavourable due to known differences in ore characteristics between Dugald River and Century, a low lead head grade and equipment issues during the trial. For example, for the treatment of Dugald River ore at Century a number of process line changes were made to allow one of the zinc regrind mills to function as a lead regrind mill. In the event, these modifications proved problematic and the mill operated with low availability and poor grind size consistency. Issues with the lead regrind mill contributed to the unfavourable performance of

Assessment Criteria	Commentary
	the lead circuit. • A large number of valuable learnings emerged from the 2nd trial and these will assist with rapid ramp-up of the new Dugald River concentrator. • Reagent costs during the trial were favourable by a significant margin
Geotechnical	• Ground conditions at Dugald River are generally good to fair with isolated areas of poor ground associated with shear zones and faults. • Development ground support at this stage in the mine life of Dugald River focuses on static stability requirements with surface support matched to the ground conditions e.g. mesh for good ground/short design life and fibrecrete for poor ground and for long term design life (e.g. decline). • Stope stability is strongly influenced by the presence and proximity of hangingwall shear zones which are associated with very poor ground conditions. • The trial stoping conducted at Dugald River was used to calibrate the Geotechnical stope dilution model from which guidance on stope dimensions (strike/ hydraulic radius) and estimates of stope dilution could be made. • The life of mine, mining sequence is modelled using MAP3D (a linear-elastic method) to provide guidance on favourable sequences that will minimise the adverse effects of induced stress.
Environmental	• 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. • Non-Acid Forming (NAF) and Potentially Acid Forming (PAF) waste rock are distinguished by a north-south striking Limestone and Undifferentiated Black Slate lithological boundary respectively. Volumes are estimated and managed in accordance with site procedures. Waste rock storage space on surface is limited. • The north mine area will allow for the return of waste rock as backfill and the south mine will be backfilled with paste fill generated from tailings, once the processing plant is operational. • PAF/NAF classification is based on the work of the Environmental Geochemistry International (EGI) in 2010. Subsequent follow-up test work on-site confirms EGI's conclusions.
Infrastructure	• Currently the Dugald River project is operating using diesel generators and electricity grid if needed. Northwest Queensland is now connected to the state electricity grid, feed from Mica Creek gas fired power station on the southern outskirts of Mount Isa. • Gas will be supplied via the Carpentaria pipeline, which will require

Assessment Criteria	Commentary
	a compression station in Bellevue. Power will be transmitted to Chumvale using Ergon Energy's existing 220kV line, then to Dugald River via an MMG owned 220kV line. This line will be approximately 62km long and the route was selected after extensive community consultation. Power for the underground operation will be stepped down to 1000V for fixed plant and mining equipment. • The main source of raw water is from the Lake Julius. Raw water is supplied from the existing Lake Julius to Ernest Henry pipeline owned and operated by Sun Water, a Queensland-government-owned corporation. Two identical water treatment plants at the plant site and accommodation village. • Based on the current production schedule, Dugald River site manning numbers peak at 530 people in 2022. Cloncurry airport is used by commuter aircraft operating to Townsville, Cairns and Brisbane and serves as the fly-in–fly-out (FIFO) airport. • Existing surface infrastructure includes: o A 11km sealed access road from the Burke Developmental Road, which includes an emergency airstrip for medical and emergency evacuation use. o A construction camp. o A permanent camp. o Telstra communication tower. o A temporary contractors mobile equipment facility. o Ore and waste stockpile pads. o Contaminated run-off water storage dams. o Office facilities. o Office buildings, including emergency medical facilities. o A core shed. o A fuel farm and gensets for power generation. o Bore water fields; • Major infrastructure in progress that includes: a processing plant; a tailings storage facility; a permanent mobile equipment workshop; recreational facilities; power supply lines; and raw water supply pipe line.
Costs	• The estimation of capital cost for the Dugald River project was derived from first principles in the 2017 LoA schedule and in the process to be refined through further study work. • The mining operating costs were again estimated by MMG for the 2017 LoA Low, Mid and High case using first-principles. • Deleterious elements Mn (and to a lesser extent Fe) are to be controlled by metallurgical blending. It is expected that the feed

Assessment Criteria	Commentary
	for flotation will be blended to maintain Mn (and to a lesser extent Fe) within the contractual range for concentrate sales and thus not expected to attract additional costs. • The commodity price assumptions are aligned with the MMG corporate prices shown in section 2.1. The Dugald River Ore Reserves applied the January 2017 guidance as this applied to 2017 LoA schedule. • The long-term exchange rate used was 0.80 ($A/$US), which is the January 2017 Long Term (2019+) MMG guidance and assumptions. Short term (2017-2020) exchange rate assumptions based on May 2017 forecast supplied by the MMG Business Evaluation department. • The road freight and logistics for domestic and export sales have been updated using the costs from the 2017 LoA report. The additional costs for storing and ship loading of concentrate in Townsville are included. For the 2017 Ore Reserves the storage and ship loading cost has been added to the freight and logistics cost for export. The freight and logistic costs for the domestic sale of concentrate includes of the sea freight cost based on an agreement with Sun Metals. • Treatment and refining charges are based on MMG's estimate as contracts currently under review. • 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.
Revenue factors	• As part of the 2017 Ore Reserves process, the net smelter return (after royalty) (NSR) has been revised with the latest parameters and compared against the previous 2016 NSR calculation that was used for the 2017 Ore Reserves. • The NSR is used to convert the various zinc, lead and silver grades into a single number for cut-off estimation and determining if the rock is ore or waste. • Freight and logistic charges have increased by 30% due to the increase in road and rail freight costs from Dugald River to Townsville when compared to the 2016 Ore Reserves. • The commodity price assumptions are aligned with the MMG corporate prices shown in section 2.1.
Market assessment	• MMG considers that the outlook for zinc and lead prices is positive. While global demand is forecast to continue to grow, zinc and lead are structurally under supplied in the short to medium term. • Zinc mine production contracted significantly following the late 2015 closure of major mines such as Century and Lisheen due to resource exhaustion. There were a number of other mine

Assessment Criteria	Commentary
	production curtailments due to low prices, most notably Glencore's decision in October 2015 to place several mines on care and maintenance. Although the zinc price has improved during the course of 2017, most of the idled production currently remains shut. • The zinc market is also characterised by very few mine developments planned for the coming years while those projects that are being studied tend to have only modest capacity. There is a scarcity of high-grade and large-scale deposits driven by historical under-investment in exploration and the nature of zinc deposits. Future zinc supply will likely come from lower-grade, high-cost underground mines as current Ore Reserves are depleted. • The mine closures referred to above have resulted in the zinc market showing significant concentrate shortages in 2016 and 2017. This shortage of concentrate has constrained refined zinc production, resulting in a deficit in a refined zinc, leading to a reduction in the refined zinc stocks that overhang the market. • The lack of new mine capacity scheduled for development over the coming years, coupled with steadily rising demand should keep both the concentrate and refined zinc markets tight ensuring good demand for Dugald River zinc and lead concentrates. The current project pipeline cannot meet demand and this will place upward pressure on the zinc price going forward incentivising mine investment. • There is potential for elevated manganese levels to occur in some zinc concentrate batches. Despite this, there is good demand for Dugald River zinc concentrates. The impact of deleterious elements has been taken into account in the project economics and marketing plans. Risks are mitigated by some diversification of outlets.
Economic	• Economic modelling of the total mining inventory shows positive annual operating cash flows. Further cost analysis was completed in March 2016 on the mining, milling and site infrastructure. Applying the revised costs, metal prices and exchange rate (MMG January 2017 Long Term economic assumptions) return a positive NPV. MMG uses a discount rate in line with MMG's corporate economic assumption and is considered to be appropriate for the location, type and style of operation • All evaluations were done in real dollars.
Social	• The nearest major population centre for the Mine 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

Assessment Criteria	Commentary
	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. MMG have instituted the MMG/KCPL Liaison Committee, which meets at least twice yearly and addresses the Kalkadoon agreement obligations for both parties. An official 'Welcome to Country' ceremony was held for MMG in late March 2012. • The Mitakoodi and Mayi People filed a claim in October 1996 and covers an area that includes part of the power line corridor. Whilst the Mitakoodi have not yet been granted Native Title, MMG continue to liaise with them as a stakeholder due to the 'last claimant standing' legal tenement. • 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.
Other	• There is no identified material naturally occurring risks. • The legal agreements are in place. There are no outstanding material legal agreements. • No required mineral specifications have been identified for this deposit. The grade of the zinc concentrate is dependent on the Fe and Mn content of the sphalerite, but this dependence is taken into account by the algorithms presented earlier. Deleterious elements Mn (and to a lesser extent Fe) are to be controlled by metallurgical blending. It is expected that the feed for flotation will be blended to maintain Mn (and to a lesser extent Fe) within the contractual range for concentrate sales and thus not expected to attract additional costs. • The government agreements and approvals are in place. There are no material unresolved matters on which the extraction of the Ore Reserves is contingent.
Classification	• Ore Reserves are reported as Proved and Probable. • Only Measured (13%) and Indicated (43%) Mineral Resources have been used to inform the Proved and Probable Ore Reserves. No Inferred Mineral Resources are included in the Ore Reserves

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td>Audit or reviews</td><td>No external audits have been undertaken for the 2017 Ore Reserves. MMG personnel have been involved in reviewing the Ore Reserves process.
An Independent Peer Review was undertaken by Pennywise on the whole project as part of a mining method review study prior to the 2014 Ore Reserves estimate.</td></tr>
<tr><td>Discussion of relative accuracy / confidence</td><td>Risks that may materially change/affect the Ore Reserves:
Geological understanding of the grade continuity with respect to diamond drill spacing.
Geotechnical risk associated with hanging-wall instability and mining dilution.
Mining infrastructure analysis requires further work on underground trucking, ventilation and power constraints.
Metallurgical risks (recovery and concentrate grades) require additional testing to confirm scale up reliability, metallurgical performance and reagent consumption.
Economic risks involve the high upfront capital requirement, ongoing, detailed revision of the capital and operating costs, and the marginal basis for the current evaluation.
Close spaced drilling is applied to a locally defined tonnage and grade before mining selection. Ore Reserves are based on all available relevant information. Ongoing work as well as risk is detailed above. The Probable Ore Reserves are based on local and global scale.
Ore Reserves accuracy and confidence that may have a material change in modifying factors are discussed above.</td></tr>
</table>

6.3.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 21.

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 21 Contributing Experts – Dugald River Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Douglas Corley, Principal Geologist MMG Ltd (Melbourne)	Geological Mineral Resources
Scott Finlay, Senior Analyst Business Evaluation, MMG Ltd (Melbourne)	Economic assumptions
Steve Whitehead, General Manager Marketing, MMG Ltd (Melbourne)	Marketing, sea freight and TC/RC
Nigel Thiel, Principal Metallurgist, MMG Ltd (Melbourne)	Metallurgy
Tim Akroyd, Surface Ops Readiness Manager, MMG Ltd (Melbourne)	Project costs
George Zacharias, Superintendent Commercial, MMG Ltd (Australian Operations)	Mining capital and operating costs
Christian Holland, Principal Geotechnical Engineer, MMG Ltd (Melbourne)	Geotechnical parameters
Karel Steyn, Principal Mining Engineer, MMG Ltd (Melbourne)	Mining parameters, cut-off estimation, mine design and scheduling
Donna Noonan, Principal Closure Planning, MMG Ltd (Melbourne)	Environmental

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

This Ore Reserve 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, Karel Steyn, confirm that I am the Competent Person for the Dugald River Ore Reserve 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 more than 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 Reserve section of this Report, to which this Consent Statement applies.

I am a full time employee of MMG Ltd. at the time of the estimate.

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 Ore Reserve 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 the Dugald River Ore Reserve.

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 Reserve - I consent to the release of the 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

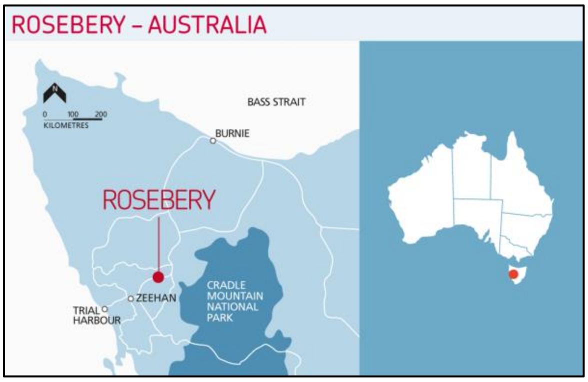
This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.	18/10/2017
Karel Steyn MAusIMM (#309192)	Date:
This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.	Yao Wu Melbourne, VIC
Signature of Witness:	Witness Name and Residence: (e.g. town/suburb)

7 ROSEBERY

7.1 Introduction and Setting

3 The Rosebery base and precious metals mining operation is held by MMG Limited and is located within Lease ML 28M/1993 (4,906ha), on the West Coast of Tasmania approximately 120km south of the port city of Burnie (Figure 7-1). The main access route to the Rosebery mine from Burnie is via the B18 and the Murchison Highway (A10).

Figure 7-1 Rosebery Mine location



The Rosebery operation consists of an underground mine and surface mineral processing plant. The mining method uses mechanised long-hole open-stoping with footwall ramp access. Mineral processing applies sequential flotation and filtration to produce separate concentrates for zinc, lead and copper. In addition, the operation produces gold/silver doré bullion. Rosebery mined and processed approximately 950kt of ore for the year ending 30 June 2017.

7.2 Mineral Resources – Rosebery

7.2.1 Results

The 2017 Rosebery Mineral Resources are summarised in Table 22. The Rosebery Mineral Resources is inclusive of the Ore Reserves.

Table 22 2017 Rosebery Mineral Resources tonnage and grade (as at 30 June 2017)

2017											
Rosebery Mineral Resources											
							Contained Metal				
Rosebery	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Copper (% Cu)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Lead ('000 t)**	**Copper ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Measured	6.0	9.3	3.3	0.26	118	1.4	554	197	15	22.7	0.3
Indicated	6.2	7.9	2.6	0.26	112	1.3	493	164	16	22.3	0.3
Inferred	6.5	7.4	2.7	0.30	90	1.4	477	175	19	18.6	0.3
Total	**18.6**	**8.2**	**2.9**	**0.27**	**106**	**1.4**	**1,525**	**536**	**51**	**63.7**	**0.8**
Stockpiles											
Measured	0.02	8.3	2.9	0.23	91	1.4	1	0.5	0.04	0.05	0.0
Total	**0.02**	**8.3**	**2.9**	**0.23**	**91**	**1.4**	**1**	**0.5**	**0.04**	**0.05**	**0.0**
Total Contained Metal							**1,526**	**537**	**51**	**63.7**	**0.8**

Cut-off grade is based on Net Smelter Return (NSR) after Royalties, expressed as a dollar value of A$166/t.
Contained metal does not imply recoverable metal.
The figures are rounded according to JORC Code guidelines and may show apparent addition errors.

7.2.2 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 23 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 23 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Rosebery Mineral Resources 2017

Assessment Criteria	Commentary
Section 1 Sampling Techniques and Data	
Sampling techniques	• Diamond drilling (DD) was used to obtain an average 1m sample that is half core split, crushed and pulverised to produce a pulp (>85% passing 75µm). • DD core is selected, marked and ID tagged for sampling by the logging geologist. Sample details and ID are stored in the SQL database for correlation with returned geochemical assay results. • Prior to May 2016, pulps were delivered to the ALS laboratory in Burnie, Tasmania for XRF analysis. Post May 2016 half core samples are delivered to the ALS laboratory in Burnie for sample preparation and XRF analysis. Analysis moved to the ALS Townsville laboratory in October, 2016. • There are no inherent sampling problems recognised. • Measures taken to ensure sample representivity include sizing analysis and duplication at the crush stage.
Drilling techniques	• The drilling type is diamond core drilling from underground using single or double tube coring techniques. As of January 2014, drill core is oriented on an ad hoc basis. • Drilling undertaken from 2012 is LTK48, LTK60, NQ, NQ2, NQTK, BQTK and BQ in size. Drilling in the reporting period is LTK60 and NQ2. • Historical (pre-2012) drillholes are a mixture of sizes from AQ, LTK (TT), BQ, NQ, HQ and PQ.
Drill sample recovery	• Diamond drill core recoveries average 96%. Drill crews mark and define lost core intervals with blocks. Sample recovery is measured and recorded in the drillhole database. If excess core loss occurs in a mineralised zone the hole is re-drilled. • Drilling process control by the drill crew and geological supervision provides a means for maximising sample recovery and ensures suitable core presentation. No other measures are taken to maximise core recovery. • There is no observable correlation between recovery and grade. • Preferential loss/gains of fine or coarse materials are not significant and do not result in sample bias as the nature of mineralisation is massive to semi-massive sulphide, diamond core sampling is applied, and recovery is very high.
Logging	• 100% of diamond drill core has been geologically and geotechnically logged to support Mineral Resources estimation, mining and metallurgy studies. • Geological and geotechnical logging is mostly qualitative, some variables are quantitative, using laptop computers which store data directly to the drillhole database.

Assessment Criteria	Commentary
	• All drill core is photographed, with photos labelled and stored on the Rosebery server.
Sub-sampling techniques and sample preparation	• All samples included in the Rosebery Mineral Resource estimates are from diamond drill core. Drill core is longitudinally sawn to give half-core samples within intervals directed by the logging geologist. The remaining half-core is kept and stored in the original sample tray. Un-sampled core is now stored; prior to 2014 the un-sampled core was discarded. The standard sampling length is one metre with a minimum length of 40cm and maximum of 1.5m. • From 2005 until 2010 geological samples have been processed in the following manner: Dried, crushed and pulverised to 85% passing 75µm. Sizing analysis is carried out on 1:20 pulps. • From 2010 until 2016 geological samples have been processed in the following manner at Rosebery site: Dried, crushed to 2mm; cone split to give primary and duplicate samples with the remainder rejected. Pulverised to 85% passing 75µm. Sizing analysis is carried out on 1:20 pulps. Samples despatched to ALS Burnie. • From 2016 geological samples have been processed in the following manner: Dried, crushed to 25mm, pulverised to 85% passing 75µm. Sizing analysis is carried out on 1:20 pulps. Samples are generally not split prior to pulverisation provided they weigh less than 3.5kg. The resulting pulps are bagged, labelled and boxed for despatch to ALS Brisbane and ALS Townsville. • Sample representivity is checked by sizing analysis and duplication at the crush stage. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Rosebery mineralisation by the Competent Person.
Quality of assay data and laboratory tests	• From 2005 until 2010 the assay methods undertaken by ALS Burnie for Rosebery were as follows: ○ 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. • From 2010 until 2016 the assay methods undertaken by ALS Burnie for Rosebery were as follows: ○ Dried, crushed to 2mm; cone split to give primary and duplicate samples with the remainder rejected. Pulverised to 80% passing 75µm. ○ Despatch to ALS Burnie. ○ Analysis of Pb, Zn, Cu and Fe by X-Ray Fluorescence (XRF) applying a lithium borate oxidative fusion (0.2g sub-sample charge). ○ Analysis of Ag by aqua regia digest and Atomic Absorption

Assessment Criteria	Commentary
	Spectrometry (AAS) (0.4g sub-sample charge). o Analysis of Au by fire assay, three acid digest and flame assay AAS (30g sub-sample charge). • From 2016 the assay methods undertaken by ALS Brisbane and Townsville Rosebery were as follows: o Analysis of Ag, Zn, Pb, Cu and Fe by four acid digest and ICPAES. (ALS Brisbane). o Analysis of Au by fire assay, three acid digest and flame assay AAS (30g sub-sample charge). (ALS Townsville). o Analysis of an additional 29 elements by ICP. • All of the above methods are considered effectively total and suitable for Mineral Resources estimation at Rosebery. • The employed assay techniques are considered suitable and representative; a comparison study using the Inductively Coupled Plasma (ICP) technique was completed to check the XRF accuracy in May 2013. Independent umpire laboratory ICP re-assay of 5% pulps took place in June 2015 and May 2016 using the Intertek laboratory in Perth. Pulps for analysis were randomly selected from a list of samples where (Pb + Zn)>5%. • No data from geophysical tools, spectrometers or handheld XRF instruments have been used for the estimation of Mineral Resources. • ALS laboratory Brisbane and Townsville releases its QAQC data to MMG for analysis of internal ALS standard performance. The performance of ALS internal standards appears to be satisfactory, with several standards used within the range of MMG submitted samples. • MMG routinely insert: o matrix-matched standards, dolerite blanks and duplicates at a ratio of 1:20 to normal assays. o Blanks are inserted to check contamination. o Duplicates were taken as quarter core, coarse crush and pulp repeats. o An umpire laboratory (Intertek Perth) is used to re-assay 5% of ore-grade pulps returned from the ALS laboratory. Analysis of routine sample results and control sample performance is reported monthly. • For the reporting period sizing analysis of pulverised samples shows that 99% of samples passed the 85% passing 75µm. • QA/QC analysis has shown that: o Matrix-matched standards: Have continued to indicate a long term trend of under-reporting of zinc and lead - since moving to the ALS Brisbane facility this has improved. Gold, silver and copper standards have indicated an improvement in laboratory accuracy since moving to ALS Brisbane and Townsville. None of which degrades the Mineral Resources estimation confidence to any significant extent.

Assessment Criteria	Commentary
	○ Blank performance for the reporting period has been good, with very low contamination rates. ○ Duplicates: Quarter core duplicate results show good correlation in base metals, and slightly more divergence in precious metals. Coarse crush duplicates show good correlation in base and precious metals. Pulp duplicates show good correlation particularly with the base metals. ○ Umpire laboratory samples: Umpire pulp duplicates have displayed reasonable correlations (with the exception of Au), with no apparent trends or excessive bias.
Verification of sampling and assaying	• All mineralised intersections, are viewed and verified by numerous company personnel by comparing assay results to core photos and logging. • Batches of sampling and assay data are entered by geologists; the performance of duplicates, blanks and standards is checked by the Mine Project Geologist after each assay batch is loaded to the database; batches with failed standards are flagged and pertinent samples are sent for re-assay. • Close twinning of mineralised intersections is not an intentional part of the delineation. However, the underground drill pattern often achieves a near-twinning or scissoring and this confirms individual intersections. • Re-assayed data, due to the failure of a standard, is reviewed to determine which batch is to be used for data export and Mineral Resources estimation. Batch status is recorded in the database for audit purposes. • Database validation algorithms are run to check data integrity before data is used for interpretation and Mineral Resources modelling. • Unreliable data is flagged and excluded from Mineral Resources estimation work. Data validation macros are used to identify data errors which are either rectified or excluded from the estimation process. • Since August 2014 all data below detection limit is replaced by half detection limit values. Prior to this date, the full detection limit was used. • No adjustments have been made to assay data – if there is any doubt about the data quality or location, the drillhole is excluded from the estimation process.
Location of data points	• All current diamond drillholes are downhole surveyed using a single-shot Reflex Ezi-shot tool at 30m intervals, with a full downhole Reflex gyro survey completed at end of hole by the drilling contractor. Where a gyro downhole survey is not practicable due to equipment limitations, then a multi-shot survey will be completed. • 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 drillholes have been downhole surveyed using a SPT north seeking gyro (parent holes only).

Assessment Criteria	Commentary
	• A downhole gyro measurement has been recorded from selective drillholes since March 2014 as an independent check of downhole survey accuracy. Initial analysis suggests the single shot surveys are accurate to 100m drillhole depth, and then diverge up to 4m at 400m depth. Given this outcome, gyro downhole surveys are now standard for all diamond holes. • Grid system used is the Cartesian Rosebery Mine Grid, offset from Magnetic North by 23°75′ with mine grid origin at AMG E= 378870.055, N= 5374181.69; mine grid relative level (RL) equals AHD + 1.490m + 3048.000m. • Topographic data derived from regular LIDAR overflights are carried out and correlated with surface field checks to confirm relativity to MGA and Rosebery Mine Grid.
Data spacing and distribution	• The Rosebery mineral deposit is drilled on variable spacing dependent on lens characteristics and access. Drill spacing ranges from 40m-60m to 10m-25m between sections and vertically. The final drill pattern varies somewhat due mostly to site access difficulties in some areas. Mineralisation has short scale structural variations observable in underground workings. Some of this variation is not discernible from drill data alone. Observations of mineralisation geometry are made by traditional geological mapping and more recently using photogrammetry images of mine development faces and backs. All ore drives and most non-ore development headings are covered. • The combination of drill and other data is sufficient to establish the degree of geological and grade continuity appropriate for Mineral Resources and Ore Reserves estimation and the classifications applied. • DD samples are not composited prior to being sent to the laboratory however the nominal sample length is generally 1m. Reverse circulation drill samples are not used for Mineral Resources estimation.
Orientation of data in relation to geological structure	• Drillhole orientation is planned orthogonal 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, thus minimising sampling bias related to orientation. Some drill intersections are at low angle to the dipping mineralisation due to access limitations. • Where drillholes from surface or older holes longer than 400m exist, attempts are made to confirm mineralised intercepts by repeat drilling from newly developed drives. Deep drill intersections are excluded from Mineral Resources modelling where duplicated by shorter underground drillholes. • Drilling orientation is not considered to have introduced sampling bias.
Sample security	• Measures to provide sample security include: o Samples are stored in a locked compound with restricted access during preparation. o Half-core samples for despatch to ALS Burnie are stored in sealed containers with security personnel at the MMG mine front gate for pick-up by ALS courier. o Receipt of samples acknowledged by ALS by email and checked

Assessment Criteria	Commentary
	against expected submission list. o Prior to 2014, assay data was returned in spreadsheet and PDF formats. Since 2014, assay data returned as a SIF file and loaded directly into the database.
Audit and reviews	• Coffey Mining Pty Ltd completed an audit of the core sample preparation area in April 2013. Key results are included in the 'Quality of assay data and laboratory tests' section above. • Two internal audits of the ALS Burnie facility during the reporting period. One was undertaken by the Competent Person. No material issues were identified. Historically, any issues identified have been rectified.

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 4,906 ha. • 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 $16.07km^2$. • The joint venture agreement was between the EZ Corp of Australia (now MMG Australia Limited) 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. • There are no known impediments to operating in the area.
Exploration done by other parties	• Tom Macdonald discovered the first indication of mineralisation in 1893 when he traced alluvial gold and zinc-lead sulphide boulders up Rosebery Creek. Twelve months later an expedition lead by Tom McDonald discovered the main lode through trenching operations, on what is now the 4 Level open cut (Easterbrook, 1962; Martin, 2002). • The Rosebery deposit was operated by several different operations until 1921 when the Electrolytic Zinc Company purchased both the Rosebery and

	Hercules Mines. • Drilling from surface and underground over time by current and previous owners has supported the discovery and delineation of Rosebery's mineralised lenses.
Geology	• The Rosebery volcanogenic massive sulphide (VMS) 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 massive to semi-massive base metal sulphide 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	• The Mineral Resources database consists of 5,417 diamond drillholes providing 209,135 samples. • No individual drillhole is material to the Mineral Resource estimates and hence this geological database is not supplied. • No exploration drilling took place in the 2016-2017 reporting period.
Data aggregation methods	• This is a Mineral Resources Statement and is not a report on exploration results hence no additional information is provided for this section. • No metal equivalents were used in the Mineral Resources estimation.
Relationship between mineralisation width and intercept lengths	• Mineralisation true widths are captured by interpreted mineralisation 3D wireframes. • Most drilling was at 50° to 60° angles in order to maximise true width intersections. • Geometry of mineralisation is interpreted as sub- vertical to vertical and as such current drilling allows true width of mineralisation to be determined.
Diagrams	• No individual drillhole is material to the Mineral Resource estimates and hence diagrams are not provided. • No exploration drilling took place in the 2016-2017 reporting period.
Balanced reporting	• This is a Mineral Resources 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 Resources Statement and is not a report on exploration results hence no additional information is provided for this section.
Further work	• Further underground near mine exploration drilling is being assessed.

Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	• The following measures are in place to ensure database integrity: o All Rosebery drillhole data is stored in an SQL database on the

	Rosebery server, which is backed up at regular intervals. o Geological logging is entered directly into laptop computers which are uploaded to the database. Prior to 1996 DD holes were logged using Lotus spread sheets or on paper. o Prior to 2014: assays were loaded into the database from spreadsheets provided by the laboratory. Since 2014: assays loaded directly into the database from SIF files provided by the laboratory. • A database upgrade and full data migration was undertaken in November 2014. Several rounds of data migration checks were undertaken before allowing the database to go live. • Data validation procedures include: o Validation routines in the new database check for overlapping sample, lithological and alteration intervals. o Random comparisons of raw data to recorded database data are undertaken prior to reporting for 5% of the additional drillholes added in any one year. No fatal flaws have been observed from this random data review. o Bulk data is imported into buffer tables and must be validated before being uploaded to the master database.
Site visits	• The 2017 Competent Person for Mineral Resources visits site on a regular basis.
Geological interpretation	• Economic Zn-Pb-Ag-Au mineralisation occurs as massive, semi massive and disseminated base metal sulphide lenses located within the Rosebery host sequence. Economic and near-economic mineralisation is easily visually identified in drill core and underground mine development. • Drill core is routinely sampled across zones of visible sulphide mineralisation. • The method used for defining mineralisation domains for the 2017 Mineral Resource estimates is described below: o Interpretation of mineralised lenses has been based on a statistical data clustering method (k-means clustering). Assayed Zn-Pb-Ag-Au-Cu-Fe values are used to assign each data point to k=30 groups, with the variance of each group minimised. The K means application of ioGAS statistical software package was used for the analysis. o The initial k=30 groups were further grouped on the basis of similar statistical properties. A final set of 5 domains, namely HIGH, MEDIUM, LOW and NON-MINERALISED, was defined using the grouped k-means clusters. Summary statistics show that the 3 mineralised domains are distinct in terms of Zn, Pb, Ag, Au, Cu and Fe grades. o Cross tabulation of the domain groups with logged mineralisation style data confirmed that the statistical clustering method was closely correlated to observed mineralisation style, namely massive, semi massive and disseminated sulphide mineralisation.

<table>
<tr><td></td><td>
On a section by section basis the results of the clustering groups was validated. Where clusters were mis-assigned these were corrected.
3D wireframe models of each mineralisation style were created using an Indicator interpolation similar to kriging, using Leapfrog Geo v3.0 software. Key data inputs included non-composited drill data converted to Indicators, 'pseudo drillholes' derived from underground Adamtech photgrammetry and mineralisation trend information derived from traditional mapping and high quality photo images of development faces and backs.
The interpolation uses a model representing the spatial variability of each variable and this was chosen on the basis of experimental variograms derived from the data. The variograms used are characterised by low nugget and ranges in the order of 60m-80m at low grade and 25m-30m at high grade. They are strongly anisotropic.
The resultant wireframe models were visually compared to mapped or recorded mineralisation contacts from traditional geological mapping and Adamtech photo images. A close correlation between the models and points of observation is noted in most areas where data are available.
The domain models are broadly consistent with logged and mapped observations of the main lithology units present at mine scale, namely black shale, porphyry and the hanging wall and footwall contacts with the host sequence.
</td></tr>
<tr><td>Dimensions</td><td>
The Rosebery mineral deposit extends from 400mE-1800mE, 2500mN to -1100mN, 3400mRL-1900mRL (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.
The minimum, maximum and average thickness of the lower mine mineralised lenses are as follows:
<table>
<tr><th>Lens</th><th>Minimum (m)</th><th>Maximum (m)</th><th>Mean (m)</th></tr>
<tr><td>K</td><td>0.2</td><td>36</td><td>6</td></tr>
<tr><td>N</td><td>0.3</td><td>16</td><td>4</td></tr>
<tr><td>P</td><td>0.2</td><td>12</td><td>3</td></tr>
<tr><td>WXY (grouped)</td><td>0.3</td><td>21</td><td>3</td></tr>
</table>
</td></tr>
<tr><td>Estimation and modelling techniques</td><td>
Grades estimation uses Ordinary Kriging (OK) as implemented in Maptek Vulcan version 9. The main inputs and parameters are described below:
Blocks and 1m composites flagged by domain and estimated individually.
Parent block size for estimation of 2mE x 7.5mN x 5mRL.
Block size approximates one half of drillhole spacing in northing and RL, and is consistent with the primary sampling interval in easting (1m).
Discretisation is 2x3x5 (X, Y, Z) for a total of 30 points per block.
Minimum sample search number is 6 and maximum number is 24.
</td></tr>
</table>

	○ Octant search methods were not used. ○ The block model covers the entire Lower Mine area. ○ No grade capping or other restrictions to high grade data were applied. ○ A second estimation pass was used to estimate blocks in sparsely sampled areas not estimated in the primary estimation. • All recoverable elements of economic interest to the Rosebery Operation (Zn, Pb, Cu, Ag, Au) and Fe have been estimated. • No other deleterious element or non-grade variables of economic significance have been identified – hence they are not estimated. • No dilution or recovery factors are taken into account during the estimation of Mineral Resources. These are addressed in the relevant Ore Reserves statement. • All metals are estimated individually, and no correlation between metals is assumed or used for estimation purposes. • Block model validation was conducted by: ○ Visual inspections for true fit with the high and low grade wireframes (to check for correct placement of blocks). ○ Visual comparison of block model grades against composite file grades. ○ Global statistical comparison of the estimated block model grades against the declustered composite statistics and raw length-weighted data. ○ Swath and drift plots were generated and checked for K, N, P, W, X and Y lenses. The plots confirm overall consistency between data and estimates with a reasonable degree of smoothing.
Moisture	• Tonnes have been calculated on a dry basis, consistent with laboratory grade determinations. • No moisture calculations or assumptions are made in the modelling process.
Cut-off parameters	• Net Smelter Return (NSR) has been calculated for all block model blocks, and accounts for MMG's long-term economic assumptions (metal price, exchange rate), metal grades, metallurgical recoveries, smelter terms and conditions and off-site costs. The NSR cut-off was updated in June 2017. • Rosebery Lower Mine Mineral Resources were reported above a $166/t NSR block grade cut-off. An example of minimum grades above $166/t NSR cut-off is as follows: 3.5% Zn, 1.9% Fe, 49 g/t Ag, 0.5 g/t Au, 0.1% Cu. • In the Upper Mine area the cut-off grade remains unchanged from the 2015 estimate and is $179/t NSR. There has been no change to the Mineral Resources model, NSR calculation or NSR cut-off for the upper mine areas. • The reporting cut-off grade is in line with MMG's policy on reporting of Mineral Resources which considers current and future mining and processing costs and satisfies the requirement for prospect for future economic extraction.
Mining factors	• Mineral Resources block models are used as the basis for detailed mine

or assumptions	design and scheduling and to calculate derived NPV for the life of asset (LoA). Full consideration is made of the reasonable prospect of eventual economic extraction in relation to current and future economic parameters. All important assumptions including minimum mining width and dilution are included in the mine design process. • Mined voids (stope and development drive shapes) are depleted from the final Mineral Resource estimates as at 30 June, 2017. • For Mineral Resources in the vicinity of past mining areas, remnant pillars and other unrecovered Mineral Resources were identified after removing actual mined voids with a lateral margin of approximately 5m across strike. The 5m margin removed near-void skins and pillars as these are considered not to have reasonable prospects for mining. • A remnant review of Mineral Resources near existing historical voids was undertaken in 2017. As a result of this review 6.2 Mt of previously reported Inferred Mineral Resources have been removed from the Mineral Resources.
Metallurgical factors or assumptions	• Metallurgical processing of ore at Rosebery involves crushing and grinding followed by flotation and filtration to produce saleable concentrates of copper, lead and zinc. Additionally, gold is partly recovered as doré following recovery from a gravity concentrator. • Metallurgical recovery parameters for all payable elements are included in the NSR calculation, which is used as the cut-off grade for the Mineral Resource estimates. The metallurgical recovery function is based on recorded recoveries from the Rosebery concentrator and monitored in monthly reconciliation reports.
Environmental factors or assumptions	• Environmental factors are considered in the Rosebery life of asset work, which is updated annually and includes provision for mine closure. • Potentially Acid Forming (PAF) studies have been completed for sulphide-rich waste at Rosebery Mine in 2014. Determination of surface treatable and untreatable waste is currently determined by visual assessment guided by geological modelling and is not estimated or included in the 2017 Mineral Resources block models.
Bulk density	• An empirical formula is used to determine the dry bulk density (DBD), based on Pb, Zn, Cu and Fe assays and assuming a fixed partition of the Fe species between chalcopyrite and pyrite. This formula is applied to the block model estimations after interpolation has been completed. The formula applied is: $DBD = 2.65 + 0.0560\ Pb\% + 0.0181\ Zn\% + 0.0005\ Cu\% + 0.0504\ Fe\%$ • A study conducted in August 1999 compared the estimated DBD against values determined using the weight in water, weight in air method and found the formula to be reliable. There has been no change to the formula in 2016-2017. • The Rosebery mineralisation does not contain significant voids or porosity. The DBD measurement does not attempt to account for any porosity.
Classification	• Mineral Resources classifications used criteria that required a certain minimum number of drillholes. The spatial distribution of more than one drillhole ensures that any interpolated block was informed by sufficient samples to establish grade continuity.

	• Drillhole spacing for classification were based on an internal Rosebery drillhole spacing study undertaken in 2017. Results from the study indicate:
	  ○ Measured Mineral Resources: 15m x 15m drillhole spacing.
	  ○ Indicated Mineral Resources: 30m x 30m drillhole spacing.
	  ○ Inferred Mineral Resources are generally defined as twice the spacing of Indicated Mineral Resources provided there is reasonable geological continuity.
	• Zinc estimated values were used for classification.
	• The Mineral Resources classification reflects the Competent Persons view on the confidence and uncertainty of the Rosebery Mineral Resources.

Audits or reviews	• Formal reviews by QG Consulting of the 2015 estimation methodology were conducted in the period August-December 2015 and recommended several changes to modelling methodology, specifically to incorporate additional geological and grade data into 3D models of mineralisation geometry as the basis for grades estimation. • The 2017 Mineral Resource estimates methodology reflects the recommendations of the 2015 reviews. • The 2017 Mineral Resource estimates was peer reviewed internally with no material issues identified.
Discussion of relative accuracy/ confidence	• There is high geological confidence that the spatial location, continuity and estimated grades of the modelled lenses within the Rosebery Mineral Resources. The sheet-like, lensoidal nature of mineralisation historically exhibited is expected to be present in the remaining Mineral Resources at least at global scale. • Minor local variations are observed at a sub-20m scale; it is recognised that the short scale variation cannot be accurately captured even at very close drill spacing, and additional mapping data is important. Short scale geometry variation appears to be related to the preferential strain around relatively competent units in the mine sequence; there is little evidence of brittle fault offsets. • Twelve month rolling reconciliation figures for the Mineral Resources model to Metallurgical Balance are within 10% for all metals on an annual basis, suggesting that the Rosebery Mineral Resources estimation process is sound. • Mining and development images (including traditional mapping and digital photographic images) shows good spatial correlation between modelled mineralised boundaries and actual geology. • The combination of Mineral Resources model, mapping, stope commentaries and face inspections provides reasonably accurate grade estimations for mill feed tracked on a rolling weekly basis, and in each end of month report. • Remnant mineralisation in close proximity to voids in the upper and lower levels has been removed from the reported Mineral Resources. • The accuracy and confidence of this Mineral Resource estimates is considered suitable for public reporting by the Competent Person.

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

This Mineral Resources 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, Anna Lewin, 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 and Metallurgy
- I have reviewed the relevant Rosebery Mineral Resources section of this Report to which this Consent Statement applies.

I am a contracted employee of MMG Ltd. at the time of the 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 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 Rosebery 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 2017 Mineral Resources and Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised	18/10/2017
Anna Lewin MAusIMM) (#992405)	Date:
This signature was scanned for the exclusive use in this document – the *MMG Mineral Resources and Ore Reserves Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.	Rex Berthelsen Melbourne, VIC
Signature of Witness:	Witness Name and Residence: (e.g. town/suburb)

7.3 Ore Reserves – Rosebery

7.3.1 Results

The 2017 Rosebery Ore Reserves are summarised in Table 24.

Table 24 2017 Rosebery Ore Reserves tonnage and grade (as at 30 June 2017)

Rosebery Ore Reserves											
							Contained Metal				
Rosebery	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Copper (% Cu)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Lead ('000 t)**	**Copper ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Proved	3.8	9.0	3.4	0.25	119	1.4	342	10	128	14.5	0.2
Probable	1.8	7.6	3.0	0.21	131	1.3	139	4	56	7.7	0.1
Total	**5.6**	**8.6**	**3.3**	**0.24**	**123**	**1.4**	**481**	**13**	**183**	**22.2**	**0.3**
Stockpile											
Proved	0.02	8.3	2.9	0.23	91	1.4	1	0	0	0.0	0.0
Total	**0.02**	**8.3**	**2.9**	**0.23**	**91**	**1.4**	**1**	**0**	**0**	**0.0**	**0.0**
Total Contained Metal							**482**	**13**	**184**	**22.2**	**0.3**

Cut-off grade is based on Net Smelter Return (NSR) after Royalties, expressed as a dollar value of A$166/t.
Contained metal does not imply recoverable metal.
The figures are rounded according to JORC Code guidelines and may show apparent addition errors.

7.3.2 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 25 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code.

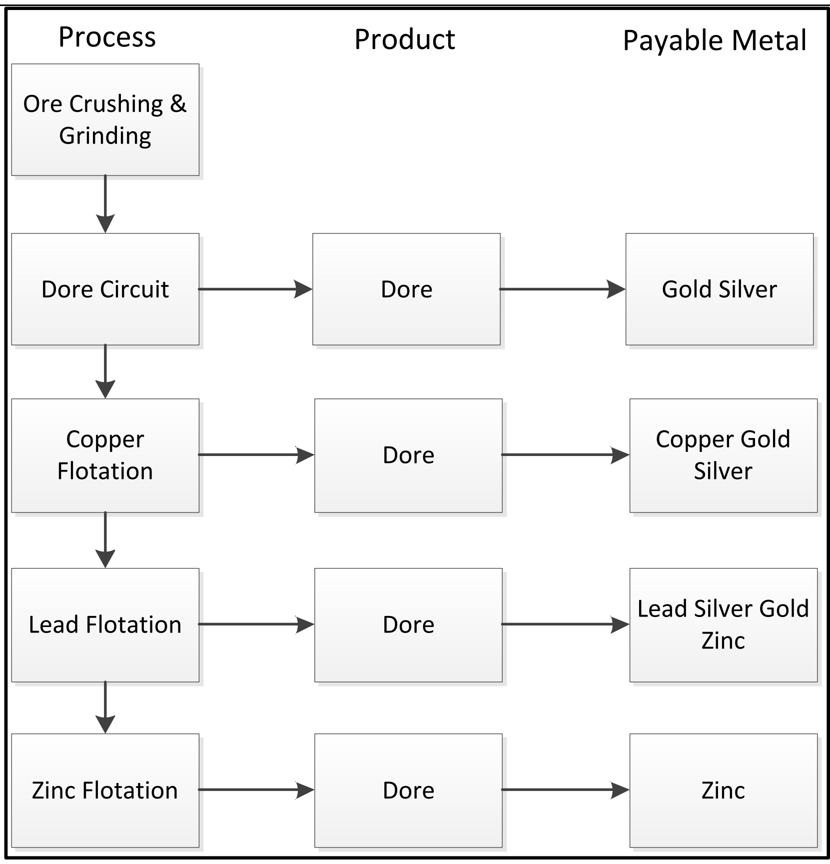
Table 25 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Rosebery Ore Reserves 2017

Assessment Criteria	Commentary
Mineral Resource estimates for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Ore Reserves. • The Ore Reserves estimation used the MMG March 2017 Mineral Resources model. (ros_gmr_1703_rese.dm) • There are high geological confidence that the spatial location, continuity and estimated grades of the modelled lenses within the Rosebery Mineral Resources. The sheetlike, lensoidal nature of mineralisation historically exhibited is expected to be present in the remaining Mineral Resources at least at global scale. • The 2017 Geotechnical model was used to estimate the hanging wall thickness, tonnes and grade of the unplanned dilution applied to the 2017 stope shapes.
Site visits	• The Competent Person for the Rosebery Ore Reserves frequently visited the site during 2016/2017.
Study status	• The mine is an operating site with on-going detailed Life of Mine planning.
Cut-off parameters	• The 2017 Mineral Resources and Ore Reserves have cut-off grades calculated, based on corporate guidance on metal prices and exchange rates. The site capital and operating costs, production profile, royalties and selling costs are based on the 2017 Budget. Processing recoveries are based on historical performance. • The operating costs, both fixed and variable, have been attributed on a per tonne basis using the planned mine production rate of 1.0Mtpa • The Net Smelter Return (NSR) values are based on metal prices, exchange rates, treatment charges and refinery charges (TC's & RC's), government royalties and metallurgical recoveries. • The NSR value for both BCOG and RCOG is to the mine gate and includes sustaining capital for the 2017 Ore Reserves. Infill diamond drilling has been included as part of the growth capital and not as sustaining capital • With respect to the Ore Reserves and Life of Mine (LoA) the break even cut-off grade (BCOG) has been used. The SCOG has been used to create the stope shapes only. The SCOG is the BCOG less development activity. The next best ore is treated as a critical business decision and has a high level of technical support. Each stope that is considered for next best ore must pass a business test. The business test considers many salient features which are used to determine if a stope will be cash positive. Only cash positive stopes can be considered for Next Best Ore. The Competent Person will deem the

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>stope Ore Reserve if the economic evaluation is satisfactory.
•
<table><tr><th colspan="3">BUD2017</th></tr><tr><th>Category of cut-off</th><th>Units</th><th>AU$/t processed</th></tr><tr><td>BCOG</td><td>Au$/t</td><td>166</td></tr><tr><td>SCOG</td><td>Au$/t</td><td>149</td></tr><tr><td>DCOG</td><td>Au$/t</td><td>81</td></tr><tr><td>ICOG</td><td>Au$/t</td><td>100</td></tr><tr><td>MCOG</td><td>Au$/t</td><td>80</td></tr><tr><td>RCOG</td><td>Au$/t</td><td>166</td></tr></table></td></tr>
<tr><td>Mining factors or assumptions</td><td>• Mining production carried out by long-hole open stoping. The majority is a longitudinal retreat while some limited areas are by wider transverse stopes.
• The 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. Stoping panels contain between 3 and 5 sub-levels with crown pillars left in-situ between the backs of up-hole stopes and the lowest sill drive of the panel above.
• Backfilling of stope voids is carried out using two methods; Cemented Rock Fill (CRF) and Rock Fill (RF). Up-hole retreat stopes are left as an open void due to lack of access for fill placement. CRF and RF are filling methods adopted in the K, N, P, W (upper) and X, and already developed W (lower) and Y levels. Within large areas of CRF a local pillar was left every 60m for stability purposes.
• Stope design is carried out using the Mineable Shape Optimiser (MSO) process within the Deswik Software with the stope cut-off grade of AU$149/t, allowing for a 1.5m hanging wall dilution within the designed shape. The length of each block used in MSO was set at five metres with each Stope is 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-25m (floor to floor) and the minimum mining width of 4.5m. This was adjusted to 4.65m in horisontal width to allow for the minor dip of the ore body and to achieve the 4.5m true width.
• A mining recovery factor of 90%-95%, depending on area been applied to mined tonnes mined.
• Access to the orebody is through a decline of 5.5mH x 5.5mW at a 1:7 gradient. The standoff distance from orebody, stoping footwall, major infrastructure, stockpiles, vent rises, escape-ways, declines and ancillary development are 65m.
• Inferred material is not included in the mine design process for Ore Reserves Reporting.
• Production of ore is in Measured Mineral Resources only with grade control drilling programs scheduled to convert Indicated Mineral</td></tr>
</table>

Assessment Criteria	Commentary
	Resources prior to development or stoping activities. Development is strictly under Survey control. Geology development control is currently not implemented at Rosebery. • The current primary ventilation system supplies approximately 630m^3/s (measured at depth) of air to the underground mine, which allow extraction from the multiple ore lenses designed.
Geotechnical	• Rosebery is one of the deepest mines in Australia with challenging ground conditions that result in the closure (squeezing) of development drives close to stoping areas and mining induced seismicity. • Just in time development, preferential drive orientations and high displacement support combined with multiple stages of rehabilitation are used to maintain serviceability of development. • Seismic monitoring, seismic re-entry exclusion periods following large stope firings and high displacement ground support are used to control risk to personnel from seismic events. • Dilution is estimated based on the average overbreak recorded in reconciliations from adjacent stopes in the different panels/stoping areas of the mine which are then used to inform dilution for the Ore Reserves. • Numerical modelling using both Linear eleastic (MAP3D) and non-linear (Abacus) is conducted by MMG personnel and Beck Engineering to assess the overall mine sequence to minimise potential seismicity and drive closure.
Metallurgical factors or assumptions	• Rosebery is a poly-metallic underground mine with all ore processed through an on-site mill and concentrator. Underground ore production is sourced from multiple ore lenses. • The table below outlines the key production physicals for 2017. These are based off actual data to September 2017 and forecast for the remainder of the year. The processing plant has a nameplate capacity of 1.0Mtpa. The site is currently mine constrained and mining and processing physicals are the same rate. Minimal stockpiles are maintained for the mill. • From the mill there are four saleable products generated: o Gold doré. o Copper concentrate. o Zinc concentrate. o Lead concentrate. • The flow chart below outlines the block flowsheet, products and payable metals.

Assessment Criteria	Commentary



- Based on the budget grades for 2017 (table above), the following recoveries are calculated from the regression analysis. These have been determined by inputting the grades into the NSRAR calculator spreadsheet to determine the relevant recoveries. These are summarised in the below table.

Product	Copper (%)	Zinc (%)	Lead (%)	Silver (g/t)	Gold (g/t)
Zinc Concentrate		87%			
Lead Concentrate		7%	80%	38%	12%
Copper Concentrate	67%			44%	37%
Gold Dore				0%	25%

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 Lake Pieman.
- The ETP hydraulic capacity is approximately 600l/sec and the plant is capable to receive 335l/sec of site mine water with remaining limited

Assessment Criteria	Commentary
	spare capacity of approximately 265l/sec to treat the site surface rain or storm water. • Environmental legacy sites - There are 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 area has a large impact on the land area along with major water issues. This area is the most significant "legacy site" for Rosebery management. Smaller historic legacy sites include the Zeehan Smelter site, South Hercules and historic mines numbering at least ten known sites, such as Jupiter's, along with a number of small historic workings. • Waste rock - Waste rock characterised as either Potentially Acid Forming (PAF) or Non-Acid Forming (NAF) or High PAF for disposal. The majority of waste rock produced is retained underground and used for stope filling, either as an RF or as CRF. Any surplus waste rock is trucked to the surface and unloaded at the current waste rock dump, referred to as the 3 Level WRD. Only NAF or PAF rock can be disposed at this facility, this rock is treated by adding a layer of lime on top and below every layer of waste rock. Work is performed to validate that enough buffering capacity has been added by validating the acid producing potential of the waste rock layer. High PAF material, preferably disposed back underground to be used as cemented rock fill.
Infrastructure	• 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. • Electric power to the site feeds in through No. 1 Substation located adjacent to the Mill Car Park on Arthur Street. There is a contract for the supply to the site with the electrical sSupply authority for the region. The electrical 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). This also provides the electrical supply authority the ability to manage a potential increase in supply requirement by the site. Further, works are currently underway to provide an upgrade to the substation infrastructure, the result of which will provide a significant increase in the security of the supply to the site. • Fresh water for the site is currently sourced from Lake Pieman and the Stitt River, with allotments of 5,500ML and 1,647ML respectively. • In total, the Rosebery Mine operation has 258 MMG employees (excluding contractors), covering all aspects of the operation. Within the mining area there are 172 MMG employees and a further 163 contractors. • Primary surface communication from the Rosebery Mine site is by phone along with mobile phone coverage, provided by Telstra.

Assessment Criteria	Commentary

	Phones are available throughout the main office building along with the mill and other surface buildings. There is also an extension of the phone system underground. Along with the phone system there is connected to email and internet services. This is available through the office area through a wireless system.
	• The main system for communication underground is through radio via a leaky feeder system. The radio system operates on multiple channels with general, extended discussion and emergency channels.
	• 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 to becoming the Haulage route, ore was hoisted up the No. 2 shaft, extending from 17L through to discharge on 7 Levels.
	• While there are multiple paths from the certain points underground, only one main route is used to access to the upper mid area of K Lens. From this point access splits between two declines to the K/N/P Lens area and the W/X/Y Lenses. Other declines are used to direct return air. The ore is hauled out of the mine in a fleet of 55-60 tonne trucks.
	• The Rosebery mine primary ventilation circuit is essentially a series circuit where airflow accumulates airborne contaminants and heat as it progresses deeper into the mine, at the 46K Level fresh air is introduced into the circuit via the NDC shaft diluting contaminated air, and finally reporting to the return airways and exhausting to surface. The current primary ventilation system supplies approximately 550m^3/s of fresh air used in the lower part of the underground mine (Y Lens Exhaust, 61K Exhaust, 58W Exhaust, U/G Fuel bay, U/G Compressor and U/G Magazine). The system comprises of three primary fan installations on the surface (PSF1, PSF2 and PSF3) and two booster fan installations underground (19B Booster Fans and 33P Booster Fans). The specifications of these fan installations are detailed below:
	  o PSF1 (New NUC)) are 2 x 1800kW Howden centrifugal fans. Design Duty is 400 m^3/s.
	  o PSF 2 (Old NUC) is a single 550kW centrifugal fan. The duty is 100 m^3/s.
	  o PSF 3 (SUC) are 2 x 550 kW Korfmann KGL 2600 mm axial fans in parallel. The duty is 161m^3/s.
	  o The 33P Booster fan is a Flaktwoods 550kW 2600 VP axial fan.
	  o 19B Booster fans are 1 x Twin 90kW & 1 x Twin 110kW CC1400 MK4 secondary fans mounted in parallel.
	• The main intake airways of the mine are the decline portal, No.2 Shaft and the NDC shaft.
	• A crib room and workshop facility at the 46K Level.

Assessment Criteria	Commentary
	• Concentrate is transported using the Emu Bay Railway, which is a freight only line that connects the West Coast area to the port in Burnie. • Tailings from the ore processing plant are placed in a Tailings Storage Facility (TSF) located to the north of Rosebery at Bobadil. The TSF has a capacity at current rates through mid 2018. • Other Rosebery site infrastructure includes mineral processing facilities (mill, concentrator, filtration and rail load-out), and buildings (offices, workshops, change-house).
Costs	• Costs used for calculating the the cut-off grade used for the Ore Reserve estimation were based on the 2017 Budget. Costs are inclusive of Operating and Sustaining Capital expanses. • The commodity price and exchange rate assumptions are prepared by MMG GroupFinance and approved by the MMG board. • All applicable inflation rates, exchange rates, transportation charges, smelting & refining costs, penalties for failure to meet specification and royalties are included as part of the NSR calculations evaluated against the block model to estimate projected value. • No deleterious elements of economic significance occur in the concentrates.
Revenue factors	• Commodity prices and the exchange rate assumptions, treatment, refining, royalties and transportation costs for different commodities approved by the MMG Board were supplied by MMG Group Finance and have been included in the NSR calculation. • The formulas and assumptions used in the NSR calculation are based on the historical data provided by the Rosebery Metallurgy Department. • The economic evaluation was carried out to verify whether the stope NSR value is above the cut-off grade including the development cost for it. All profitable and marginal stopes were included in the Ore Reserves.
Market assessment	• MMG considers that the outlook for zinc and lead prices is positive. While global demand is forecast to continue to grow, zinc and lead are structurally under supplied in the short to medium term. Zinc mine production contracted significantly following the late 2015 closure of major mines such as Century and Lisheen due to Resources exhaustion. There were a number of other mine production curtailments due to low prices, most notably Glencore's decision in October 2015 to place several mines on care and maintenance. Although the zinc price has improved during the course of 2017, most of the idled production currently remains shut. • The zinc market is also characterised by very few mine developments planned for the coming years while those projects that are being studied tend to have only modest capacity. There is a scarcity of high-grade and large-scale deposits driven by historical under-

Assessment Criteria	Commentary
	investment in exploration and the nature of zinc deposits. Future zinc supply will likely come from lower-grade, high-cost underground mines as current Ore Reserves are depleted. • The mine closures referred to above have resulted in the zinc market showing significant concentrate shortages in 2016 and 2017. This shortage of concentrate has constrained refined zinc production, resulting in a deficit in a refined zinc, leading to a reduction in the refined zinc stocks that overhang the market. • The lack of new mine capacity scheduled for development over the coming years, coupled with steadily rising demand should keep both the concentrate and refined zinc markets tight ensuring good demand for Rosebery zinc and lead concentrates. The current project pipeline cannot meet demand and this will place upward pressure on the zinc price going forward incentivising mine investment. • Rosebery has life of mine agreements in place with Nyrstar covering 100% of zinc and lead concentrate production, which is sold to them on international terms. There is also good demand for Rosebery's small production of copper concentrates and this is covered by two long term contracts.
Economic	• Rosebery is an established operating mine. Costs detailed used in the NPV calculation are based on historical data. Revenues are calculated based on historical and contracted realisation costs and Board approved long-term metal prices. • The life of Asset (LoA) financial model demonstrates the mine has a positive NPV calculated. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit.
Social	• 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. 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 within StakeTracker (the community engagement tool), where corrective actions are initiated and monitored. • During the 2016/2017 reporting period, a total of four complaints was received. One from vibration blasting of the TSF project, two from daytime grouting and drilling noise of TSF project and one for noise of ventilation exhaust fan. All complaints were investigated and resolved in consultation with the complainant.

<table>
<tr><th>Assessment Criteria</th><th>Commentary</th></tr>
<tr><td>Other</td><td>

- Approvals were received with construction of Stage 1 of 2/5 TSF and are currently under project. An estimated completion date for Stage 1 of the tails storage facility (Dam 2/5) is mid-November 2017. The table below outlines the expected tails storage capacities at the start of 2016 for Bobadil and at the completion of the stage lifts for Dam 2/5.

<table>
<tr><th>Location</th><th>Tailings Capacity
(Tonnes)</th><th>Comment</th></tr>
<tr><td>Bobadil</td><td>1,260,000</td><td>Forecast 2018 completion</td></tr>
<tr><td>Bobadil</td><td>TBD</td><td rowspan="5">Expect additional 1Mt from tails plan refinement</td></tr>
<tr><td>Dam 2/5 – Stage 1</td><td>3,000,000</td></tr>
<tr><td>Dam 2/5 – Stage 2</td><td>2,000,000</td></tr>
<tr><td>Dam 2/5 – Stage 3</td><td>TBD</td></tr>
<tr><td>Bobadil Future</td><td>TBD</td></tr>
</table>

- All projects have been approved in a timely manner with close communication maintained by the Environmental Protection Agency (EPA).
- The table below details the current surface waste stockpiles, and lists those that could be used for backfill activities prior to closure. Some waste rock dumps are located under existing infrastructure, and could not be recovered prior to closure on the mine.

<table>
<tr><th>Location</th><th>Closure Estimate
(Tonnes)</th><th>Assume Available*
(Tonnes)</th></tr>
<tr><td>WRD</td><td>330,000</td><td>330,000</td></tr>
<tr><td>WRD</td><td>220,000</td><td>-</td></tr>
<tr><td>WRD</td><td>570,000</td><td>570,000</td></tr>
<tr><td>WRD next</td><td>540,000</td><td>-</td></tr>
<tr><td>WRD</td><td>60,000</td><td>60,000</td></tr>
<tr><td>WRD next</td><td>130,000</td><td>130,000</td></tr>
<tr><td>WRD next</td><td>60,000</td><td>-</td></tr>
<tr><td>4L Waste</td><td>500,000</td><td>500,000</td></tr>
<tr><td>TOTAL</td><td>2,410,000</td><td>1,590,000</td></tr>
</table>

* *Assumes available for backfill activities prior to closure, WRD location not impacting the required infrastructure*

</td></tr>
<tr><td>Classification</td><td>

- Ore Reserves classification follows the Mineral Resources classification where Proved Ore Reserves are only derived from Measured Mineral Resources and Probable Ore Reserves are only derived from Indicated Mineral Resources. No Inferred Mineral Resources have been included in the Ore Reserves.
- The Competent Person deems this approach as being in accordance with the JORC code and is appropriate for the classification of the Rosebery Ore Reserves.
- Where stopes contain more than one Mineral Resources category, then the individual classification components have been reported as outlined above.

</td></tr>
</table>

Assessment Criteria	Commentary
Audit or Reviews	• The Competent Persons responsible for geology and metallurgy have reviewed the NSR script to ensure correct operation for each model. Detail has been added to the script and a background document created for record keeping. • Mineral Resources block models were validated 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	• The key risks that could materially change or affect the Ore Reserves estimate for Rosebery include:
	  ○ Seismicity: The Rosebery mine has had several seismic events in the past. Potential exists for future seismic events to occur that may potentially impact on the overall recovery of the Ore Reserves.
	  ○ Drop in zinc grade and metal price.
	  ○ Close-spaced drilling is applied to locally define tonnage and grade before mining. Ore Reserves are based on all available relevant information.
	  ○ The Proved Ore Reserves is based on a local scale and is suitable as a local estimate.
	  ○ The Probable Ore Reserves is based on local and global scale information.
	  ○ Ore Reserves accuracy and confidence that may have a material change in modifying factors are discussed above.
	  ○ This Ore Reserves is based on the results of an operating mine. The confidence in the estimate is compared with actual production data.

7.3.3 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 26.

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 26 Contributing Experts – Rosebery Ore Reserves

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Anna Lewin, Senior Resource Geologist MMG Ltd (Melbourne)	Geological Mineral Resources
Scott Finlay, Senior Analyst, Business Evaluation, MMG Ltd (Melbourne)	Economic assumptions
Steve Whitehead, General Manager Marketing, MMG Ltd (Melbourne)	Marketing, sea freight and TC/RC
Kevin Rees, Principal Metallurgist, MMG Ltd (Melbourne)	Metallurgy
George Zacharias, Superintendent Commercial, MMG Ltd (Australian Operations)	Mining capital and operating costs
Christian Holland, Principal Geotechnical Engineer, MMG Ltd (Melbourne)	Geotechnical parameters
Karel Steyn, Principal Mining Engineer, MMG Ltd (Melbourne)	Mining parameters, cut-off estimation, mine design and scheduling
Donna Noonan, Principal Closure Planning, MMG Ltd (Melbourne)	Environmental and Mine Closure

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

This Ore Reserve 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, Karel Steyn, confirm that I am the Competent Person for the Rosebery Ore Reserve section of this Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting 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 Reserve section of this Report, to which this Consent Statement applies.

I am a full time employee of MMG Ltd. at the time of the 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 Rosebery Ore Reserve 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 the Rosebery Ore Reserve.

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 2017 Ore Reserves Statement as at 30 June 2017 Executive Summary and Technical Appendix Report and this Consent Statement by the directors of MMG Limited:

This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.	18/10/2017
Karel Steyn MAusIMM (#309192)	Date:
This signature was scanned for the exclusive use in this document – the *MMG Mineral Resource and Ore Reserve Statement as at 30 June 2017* – with the author's approval. Any other use is not authorised.	Yao Wu Melbourne, VIC
Signature of Witness:	Witness Name and Residence: (e.g. town/suburb)

8 HIGH LAKE

This Mineral Resources remains unchanged since reporting in 2013. This information can be found in the 2013 MMG Mineral Resources and Ore Reserves Statement as at 30 June 2013 Technical Appendix.

9 IZOK LAKE

This Mineral Resources remains unchanged since reporting in 2013. This information can be found in the 2013 MMG Mineral Resources and Ore Reserves Statement as at 30 June 2013 Technical Appendix.

10 EXTERNAL REFERENCES

Easterbrook, B. E., 1962. History and development of the Read-Rosebery mining district: *Mining and Chemical Engineering Review, v. 54, pp. 36-40.*

Martin, N., 2004. Genesis of the Rosebery Massive Sulphide Deposit, Western Tasmania, Australia. *PhD. University of Tasmania.*