MMG LIMITED

五礦資源有限公司

(Incorporated in Hong Kong with limited liability)

(STOCK CODE: 1208)

MINERAL RESOURCES AND ORE RESERVES STATEMENT AS AT 30 JUNE 2015

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 2015 (Mineral Resources and Ore Reserves Statement).

The highlights of the Mineral Resources and Ore Reserves Statement as at 30 June 2015 include:

- Las Bambas Mineral Resource and Ore Reserves are included officially for the first time. The Las Bambas project is held by a joint venture company, of which 62.5% is owned by MMG.
- The Group's Mineral Resources (contained metal) has increased for copper (304%), silver (65%) and gold (29%); decreased for lead (18%) and zinc (7%) and remains unchanged for nickel. Molybdenum is being reported for the first time this year. Assuming Las Bambas was included in the Group's Mineral Resources in 2014, the Group's Mineral Resources (contained metal) has increased for molybdenum (10%) and copper (8%); decreased for gold (21%), lead (18%) and zinc (7%) and remains unchanged for silver and nickel.[1]
- The Group's Ore Reserves (contained metal) has increased for copper (596%), gold (443%) and silver (149%); decreased for lead (12%) and zinc (8%) and reporting molybdenum for the first time. Assuming Las Bambas was included in the Group's Ore Reserves in 2014, the Group's Ore Reserves (contained metal) has increased for molybdenum (14%), gold (7%) and copper (2%); decreased for lead (12%), zinc (8%) and silver (3%).[2]
- Mineral Resources and Ore Reserve Tonnes at Las Bambas[1] increased by 226Mt and 127Mt respectively.

All data reported here is on a 100% asset basis, with the MMG's attributable interest shown against each asset within Table 1.

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[1] The Mineral Resources for Las Bambas used for this comparison purpose are those disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to the Las Bambas acquisition.

[2] The Ore Reserves for Las Bambas used for this comparison purpose are those disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to the Las Bambas acquisition.

MINERAL RESOURCE AND ORE RESERVES STATEMENT

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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 2015 published on 8 December 2015 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
Andrew Gordon Michelmore
CEO and Executive Director

Hong Kong, 8 December 2015

As at the date of this announcement, the Board comprises nine directors, of which three are executive directors, namely Mr Andrew Gordon Michelmore, Mr David Mark Lamont and Mr Xu Jiqing; two are non-executive directors, namely Mr Jiao Jian (Chairman), and Mr Gao Xiaoyu; 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 RESOURCE AND ORE RESERVES STATEMENT

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EXECUTIVE SUMMARY

Mineral Resources and Ore Reserves for MMG have been estimated as at 30 June 2015, 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 Rules Governing the Listing of Securities of The Stock Exchange of Hong Kong Limited (Listing Rules). Mineral Resource and Ore Reserve tables are provided on pages 4-9, which include the 30 June 2015 and 2014 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 Resource and Ore Reserve 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 10.

MMG has established processes and structures for the governance of Mineral Resource and Ore Reserve estimation and reporting. MMG has a Mineral Resource and Ore Reserve Committee that regularly convenes for the regulation of estimation and reporting matters, which reports to the MMG Governance and Nomination Committee and the Board of Directors.

Key changes to the Mineral Resources (contained metal) since the 30 June 2014 estimate include increases in copper, gold, silver and molybdenum due to the inclusion of the Las Bambas Mineral Resources for the first time. The contained metal at Las Bambas contributes 78% of the total Group's Mineral Resources for copper, 44% for gold, 44% for silver and 100% for molybdenum. Decreases in the Group's Mineral Resources for lead and zinc are due to depletion at Century, Golden Grove and Rosebery, removal of mineralised remnants at Rosebery and the results of technical investigations across all sites. Sepon Copper and Gold Mineral Resources have decreased as a result of technical investigations removing lower grade materials and mill depletion.

The MMG Ore Reserves (contained metal) have increased since the 30 June 2014 statement for copper, gold, silver and molybdenum due to the inclusion of Las Bambas Ore Reserve. The contained metal at Las Bambas contributes 87% of the total MMG Ore Reserve for copper, 79% for gold, 60% for silver and 100% for molybdenum.

Compared to the Mineral Resources and Ore Reserves disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to the Las Bambas acquisition, Mineral Resources (contained metal) at Las Bambas increased for copper (15%), silver (11%) and molybdenum (10%) but decreased for gold (11%).

Ore Reserves (contained metal) as Las Bambas increased for molybdenum (14%), gold (6%) and copper (5%) but decreased for silver (4%).

Tonnes of Mineral Resources and Ore Reserve at Las Bambas have also increased by 226Mt and 127Mt respectively.

Page 11 provides further discussion of the Mineral Resource and Ore Reserve changes.

MINERAL RESOURCES[3]

All data reported here is on a 100% asset basis, with the MMG's attributable interest shown against each asset within brackets and in Table 1.

	2015							2014						
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[4] (62.5%)														
Ferrobamba Oxide Copper														
Indicated	21	1.9						55	0.9					
Inferred	6	1.7						10	0.9					
Total	**27**	**1.8**						**65**	**0.9**					
Ferrobamba Primary Copper														
Measured	388	0.8			3.7	0.07	204	405	0.7			3.3	0.07	200
Indicated	490	0.6			2.9	0.05	209	365	0.7			4.0	0.08	200
Inferred	452	0.6			2.2	0.03	148	310	0.5			2.1	0.07	200
Total	**1,330**	**0.7**			**2.9**	**0.05**	**187**	**1,080**	**0.6**			**3.2**	**0.07**	**200**
Ferrobamba Total	**1,357**							**1,145**						
Chalcobamba Oxide Copper														
Indicated	5.9	1.4						35	0.6					
Inferred	0.5	1.5						1	0.3					
Total	**6.4**	**1.4**						**36**	**0.6**					
Chalcobamba Primary Copper														
Measured	96	0.4			1.3	0.02	151	85	0.4			1.4	0.02	140
Indicated	190	0.6			2.3	0.03	138	250	0.6			2.3	0.03	130
Inferred	41	0.5			1.5	0.02	122	45	0.3			1.1	0.02	120
Total	**327**	**0.5**			**1.9**	**0.02**	**140**	**380**	**0.5**			**2.0**	**0.03**	**131**
Chalcobamba Total	**334**							**416**						
Sulfobamba Oxide Copper														
Inferred	0.02	2.8												
Total	**0.02**	**2.8**												
Sulfobamba Primary Copper														
Indicated	102	0.6			4.4	0.02	164	105	0.6			4.6	0.02	200
Inferred	214	0.5			4.2	0.02	117	115	0.4			3.8	0.01	100
Total	**315**	**0.5**			**4.3**	**0.02**	**132**	**220**	**0.5**			**4.2**	**0.01**	**148**
Sulfobamba Total	**315**							**220**						
Las Bambas Total	**2,007**							**1,781**						

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

[4] 2014 Las Bambas Mineral Resource has been taken from the Competent Person's report prepared for the Circular released on 30 June 2014.

MINERAL RESOURCES

	2015							2014						
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%)														
Stockpiles														
Measured	6.4	2.3						5.3	2.7					
Total	**6.4**	**2.3**						**5.3**	**2.7**					
Oxide Copper														
Measured	3.7	4.5						7.0	3.8					
Indicated	11.9	3.4						12.2	3.2					
Inferred	4.2	3.3						0.5	2.9					
Total	**19.8**	**3.6**						**19.7**	**3.4**					
Primary Copper														
Measured	1.6	3.2												
Indicated	10.9	2.2												
Inferred	14.6	2.4						24.6	2.5					
Total	**27.1**	**2.3**						**24.6**	**2.5**					
Kinsevere Total	**53.3**							**49.6**						
Sepon (90%)														
Oxide Gold														
Measured								0.8				8	2.9	
Indicated	1.1					3.0		3.1				4	1.5	
Inferred	0.2					2.1		1.4				3	1.2	
Total	**1.2**					**2.9**		**5.3**				**4**	**1.6**	
Partial Oxide Gold														
Measured								0.9				13	3.5	
Indicated	0.6					5.4		1.6				6	2.3	
Inferred	0.01					4.1		1.0				5	1.2	
Total	**0.6**					**5.4**		**3.5**				**7**	**2.2**	
Primary Gold														
Indicated	7.5					3.4		11.2				10	3.2	
Inferred	0.3					2.5		5.7				8	3.3	
Total	**7.8**					**3.4**		**16.9**				**9**	**3.2**	
Gold Stockpiles														
Measured								0.7					1.5	
Total								**0.7**					**1.5**	
Supergene Copper														
Indicated	13.4	3.3						30.8	2.2					
Inferred	1.0	2.5						11.5	1.4					
Total	**14.4**	**3.2**						**42.2**	**2.0**					
Primary Copper														
Indicated	7.6	1.0						7.7	0.9			6		
Inferred	3.8	1.5						2.4	1.3			5		
Total	**11.4**	**1.1**						**10.1**	**1.0**			**6**		
Copper Stockpiles														
Measured	5.9	2.1						8.5	1.5					
Total	**5.9**	**2.1**						**8.5**	**1.5**					
Sepon Total	**41.4**							**87.3**						

MMG Limited

MINERAL RESOURCE AND ORE RESERVES STATEMENT

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MINERAL RESOURCES

	2015							2014						
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														
Measured	5.7		14.5	2.0	63			5.6		14.7	2.0	64		
Indicated	25.9		13.3	2.2	51			25.2		13.5	2.3	52		
Inferred	25.7		12.7	1.8	13			24.4		13.1	1.9	14		
Total	**57.3**		**13.2**	**2.0**	**35**			**55.2**		**13.4**	**2.1**	**36**		
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**	
Dugald River Total	**61.7**							**59.6**						
Golden Grove (100%)														
Oxide Gold														
Indicated	0.6				89	3.2		0.8				52	3.6	
Inferred	0.04				55	2.8		0.3				25	2.1	
Total	**0.6**				**87**	**3.2**		**1.1**				**45**	**3.2**	
Partial Oxide Gold														
Indicated	0.1				130	2.6		0.1				177	2.9	
Inferred	0.01				71	2.0		0.1				74	2.1	
Total	**0.1**				**123**	**2.5**		**0.2**				**149**	**2.7**	
Primary Gold														
Indicated	0.1				54	2.2		0.1				39	1.8	
Inferred	0.01				49	2.1		0.04				28	1.5	
Total	**0.1**				**53**	**2.2**		**0.1**				**35**	**1.7**	
Primary Zinc														
Measured	2.7	0.5	11.3	1.3	89	1.7		1.5	0.3	13.2	1.6	111	1.4	
Indicated	2.0	0.3	11.0	1.5	108	1.5		1.8	0.4	14.4	1.6	103	3.1	
Inferred	3.7	0.5	13.7	0.5	40	0.6		5.5	0.4	12.7	0.9	56	0.8	
Total	**8.4**	**0.5**	**12.3**	**1.0**	**72**	**1.1**		**8.9**	**0.4**	**13.2**	**1.1**	**75**	**1.4**	
Oxide Copper														
Measured								0.2	3.3					
Indicated								0.4	2.0				0.1	
Inferred								0.01	1.7				0.02	
Total								**0.6**	**2.4**				**0.1**	
Partial Oxide Copper														
Indicated	0.3	2.2						0.6	3.6					
Inferred	0.004	2.1						0.01	3.5					
Total	**0.3**	**2.2**						**0.6**	**3.6**					
Primary Copper														
Measured	6.2	2.9	2.6	0.3	33	1.3		6.1	2.7	0.5	0.1	19	0.5	
Indicated	2.0	2.8	2.0	0.2	29	1.2		2.6	2.8	1.2	0.2	26	1.0	
Inferred	8.4	3.3	0.7	0.0	26	0.2		11.5	2.9	0.4	0.0	23	0.3	
Total	**16.7**	**3.1**	**1.6**	**0.2**	**29**	**0.7**		**20.2**	**2.8**	**0.6**	**0.1**	**22**	**0.4**	
Golden Grove Total	**26.2**							**31.6**						

MINERAL RESOURCES

	2015							2014						
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)**
Rosebery (100%)														
Rosebery														
Measured	9.0	0.3	8.6	2.8	96	1.2		7.7	0.4	12.6	3.9	127	1.6	
Indicated	6.4	0.3	7.3	2.5	103	1.1		4.3	0.3	10.0	3.5	125	1.5	
Inferred	7.0	0.3	7.4	2.8	96	1.4		5.2	0.6	10.3	3.4	115	2.2	
Total	**22.4**	**0.3**	**7.9**	**2.7**	**98**	**1.2**		**17.2**	**0.4**	**11.3**	**3.6**	**123**	**1.7**	
South Hercules														
Measured	0.1	0.1	4.6	2.5	151	3.8		0.6	0.1	4.0	2.1	164	3.1	
Indicated	0.02	0.1	3.7	1.8	161	4.3		0.1	0.1	2.7	1.3	168	3.0	
Total	**0.2**	**0.1**	**4.5**	**2.4**	**152**	**3.9**		**0.7**	**0.1**	**3.8**	**2.0**	**165**	**3.1**	
Rosebery Total	**22.6**							**17.9**						
Century (100%)														
Century Pit														
Indicated	0.7		9.7	1.4	36			7.9		9.3	1.7	41		
Inferred								0.5		9.1	1.5	38		
Total	**0.7**		**9.7**	**1.4**	**36**			**8.4**		**9.3**	**1.7**	**41**		
Eastern Fault Block														
Indicated								0.5		11.6	1.1	48		
Total								**0.5**		**11.6**	**1.1**	**48**		
Stockpiles														
Measured	1.9		6.1	1.7	42			1.1		5.7	2.3	51		
Total	**1.9**		**6.1**	**1.7**	**42**			**1.1**		**5.7**	**2.3**	**51**		
Silver King														
Inferred								2.7		6.9	12.5	121		
Total								**2.7**		**6.9**	**12.5**	**121**		
Century Total	**2.6**							**12.8**						
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**	
High Lake Total	**14.0**							**14.0**						
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**	
Izok Lake Total	**14.6**							**14.6**						

	2015		2014	
Deposit	**Tonnes (Mt)**	**Ni (%)**	**Tonnes (Mt)**	**Ni (%)**
Avebury (100%)				
Measured	3.8	1.1	3.8	1.1
Indicated	4.9	0.9	4.9	0.9
Inferred	20.7	0.8	20.7	0.8
Total	**29.3**	**0.86**	**29.3**	**0.86**
Avebury Total	**29.3**		**29.3**	

ORE RESERVES

All data reported here is on a 100% asset basis, with the MMG's attributable interest shown against each asset within brackets and in Table 1.

	2015							2014						
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[5] (62.5%)														
Ferrobamba Primary Copper														
Proved	424	0.7			3.4	0.08	187	386	0.7			3.4	0.07	180
Probable	360	0.6			2.8	0.06	187	271	0.8			4.5	0.09	210
Total	**784**	**0.7**			**3.2**	**0.07**	**187**	**657**	**0.7**			**3.8**	**0.08**	**190**
Chalcobamba Primary Copper														
Proved	77	0.5			1.5	0.02	155	63	0.5			1.5	0.02	140
Probable	150	0.7			2.6	0.03	137	172	0.7			2.8	0.03	130
Total	**227**	**0.6**			**2.2**	**0.03**	**143**	**235**	**0.7**			**2.4**	**0.03**	**140**
Sulfobamba Primary Copper														
Proved														
Probable	68	0.8			5.5	0.03	176	60	0.9			6.6	0.02	140
Total	**68**	**0.8**			**5.5**	**0.03**	**176**	**60**	**0.9**			**6.6**	**0.02**	**140**
Las Bambas Total	**1,079**							**952**						
Kinsevere (100%)														
Stockpiles														
Proved	1.4	3.7						1.6	4.6					
Probable	3.4	1.4						2.7	1.5					
Total	**4.8**	**2.1**						**4.3**	**2.6**					
Oxide Copper														
Proved	2.9	4.7						5.2	4.2					
Probable	6.6	3.9						6.8	3.6					
Total	**9.4**	**4.1**						**12.0**	**3.8**					
Kinsevere Total	**14.3**							**16.4**						
Sepon (90%)														
Supergene Copper														
Probable	8.3	3.6						8.8	4.3					
Total	**8.3**	**3.6**						**8.8**	**4.3**					
Primary Copper														
Probable	2.9	1.1												
Total	**2.9**	**1.1**												
Copper Stockpiles														
Proved	5.7	2.1						5.1	1.8					
Total	**5.7**	**2.1**						**5.1**	**1.8**					
Sepon Total	**16.9**							**14.0**						

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[5] 2014 Las Bambas Ore Reserve has been taken from the Competent Person's report prepared for the Circular released on 30 June 2014.

ORE RESERVES

	2015							2014						
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	0.5		15.5	1.4	38									
Probable	22.1		12.3	2.0	50			21.2		12.6	2.2	49		
Total	**22.5**		**12.3**	**2.0**	**50**			**21.2**		**12.6**	**2.2**	**49**		
Dugald River Total	**22.5**							**21.2**		**12.6**	**2.2**	**49**		
Golden Grove (100%)														
Primary Zinc														
Proved	1.1	0.5	12.0	1.6	103	3.2		0.9	0.5	12.3	1.7	138	1.7	
Probable	0.9	0.3	11.1	1.9	148	1.4		1.0	0.7	12.4	1.5	81	4.0	
Total	**2.0**	**0.4**	**11.6**	**1.7**	**123**	**2.4**		**1.9**	**0.6**	**12.3**	**1.6**	**107**	**2.9**	
Oxide Copper														
Proved								0.2	3.3					
Probable														
Total								**0.2**	**3.3**					
Partial Oxide Copper														
Proved	0.1	2.8												
Probable	0.2	2.1						0.4	3.7					
Total	**0.3**	**2.3**						**0.4**	**3.7**					
Primary Copper														
Proved	1.8	3.1	2.0	0.2	24	1.3		2.1	2.9	0.4	0.0	17	0.5	
Probable	1.0	2.7	3.4	0.4	31	2.2		1.0	3.0	2.9	0.3	30	1.8	
Total	**2.7**	**2.9**	**2.5**	**0.3**	**27**	**1.6**		**3.1**	**2.9**	**1.2**	**0.1**	**21**	**1.0**	
Golden Grove Total	**5.1**							**5.5**						
Rosebery (100%)														
Proved	4.8	0.2	8.2	2.6	85	1.0		3.2	0.3	10.7	3.4	111	1.4	
Probable	2.6	0.2	6.0	2.4	100	1.0		2.3	0.3	8.2	3.3	121	1.3	
Total	**7.4**	**0.2**	**7.4**	**2.6**	**91**	**1.0**		**5.4**	**0.3**	**9.7**	**3.4**	**115**	**1.4**	
Rosebery Total	**7.4**							**5.4**						
Century (100%)														
Century Pit														
Proved	1.9		6.1	1.7	42			0.8		6.8	2.6	69		
Probable	0.7		8.7	1.1	34			7.2		8.3	1.5	37		
Total	**2.7**		**6.8**	**1.5**	**40**			**8.0**		**8.2**	**1.6**	**40**		
Century Total	**2.7**							**8.0**						

COMPETENT PERSONS

Deposit	Accountability	Competent Person	Professional Membership	Employer
MMG Mineral Resources and Ore Reserves Committee	Mineral Resources	Jared Broome	FAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Ore Reserves	Richard Butcher	FAusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Metallurgy: Mineral Resources / Ore Reserves	Geoffrey Senior	MAusIMM	MMG
Las Bambas	Mineral Resources	Rex Berthelsen	FAusIMM(CP)	MMG
Las Bambas	Ore Reserves	Richard Butcher	FAusIMM(CP)	MMG
Sepon	Mineral Resources	Chevaun Gellie	MAusIMM	MMG
Sepon	Ore Reserves	Dean Basile	MAusIMM(CP)	Mining One Pty Ltd.
Kinsevere	Mineral Resources	Douglas Corley	MAIG R.P.Geo.	MMG
Kinsevere	Ore Reserves	Dean Basile	MAusIMM(CP)	Mining One Pty Ltd.
Rosebery	Mineral Resources	Jared Broome	FAusIMM(CP)	MMG
Rosebery	Ore Reserves	Karel Steyn	MAusIMM	MMG
Golden Grove (Underground & Open Pit)	Mineral Resources	Paul Boamah	MAusIMM	MMG
Golden Grove - Underground	Ore Reserves	Wayne Ghavalas	MAusIMM	MMG
Golden Grove - Open Pit	Ore Reserves	Chris Lee	MAusIMM	MMG
Century	Mineral Resources	Claudio Coimbra	MAusIMM	MMG
Century	Ore Reserves	Claudio Coimbra	MAusIMM	MMG
Dugald River	Mineral Resources	Douglas Corley	MAIG R.P.Geo.	MMG Ltd.
Dugald River	Ore Reserves	Karel Steyn	MAusIMM	MMG
High Lake, Izok Lake	Mineral Resources	Allan Armitage	MAPEG[1] (P.Geo)	Formerly by MMG
Avebury	Mineral Resources	Peter Carolan	MAusIMM	Formerly by 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] Member of the Association of Professional Engineers and Geoscientists of British Columbia.

SUMMARY OF SIGNIFICANT CHANGES

MINERAL RESOURCES

The MMG 30 June 2015 Mineral Resources have changed since the 30 June 2014 estimate for a number of reasons with the most significant changes outlined in this section.

The Group's Mineral Resources (contained metal) have increased for copper (304%), silver (65%), gold (29%), and molybdenum is being reported first time as a direct result of the inclusion of Las Bambas. Contained metal has decreased for lead (18%) and zinc (7%) due to mill depletion and changes due to technical investigations. Nickel remains unchanged.

Assuming Las Bambas was included in the Group's Mineral Resources in 2014[6], the Group's Mineral Resources (contained metal) has increased for molybdenum (10%), copper (8%); decreased for gold (21%), lead (18%) and zinc (7%) and remains unchanged for silver and nickel.

However, on an individual site by site basis there are both increases and decreases to the Mineral Resources (contained metal) the significant changes are discussed below.

Increases:

Increases to the Mineral Resources (contained metal) for copper, silver and molybdenum at Las Bambas are related to positive drilling results and re-estimation as a result of changes to modelling techniques.

Reductions:
Technical investigations and studies have resulted in significant reductions in Mineral Resources for:

- Sepon (copper and gold) through the removal of lower grade materials.

Milling depletion at all MMG Operations has reduced Mineral Resources, with the largest impacts on:

- Century (zinc, lead and silver) as a result of mine closure (where all in-situ Mineral Resources not within the 2015 mine schedule have been removed) and mining depletion;
- Sepon (copper); and
- Rosebery (zinc and copper) as a result of mill depletion and removal of mineralised remnants.

No changes have been made to the Mineral Resources at High Lake, Izok Lake and Avebury.

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[6] For the purpose of comparison, the Mineral Resources for Las Bambas used are those disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to the Las Bambas acquisition

ORE RESERVES

The MMG 30 June 2015 Ore Reserves increased for contained metal compared to the 2014 Ore Reserves for copper (596%), gold (443%) and silver (149%) and decreased for lead (12%) and zinc (8%). The most significant change is due to the inclusion of the Las Bambas Ore Reserves for the first time.

Assuming Las Bambas was included in the Group's Mineral Resources in 2014[7], the Group's Ore Reserves (contained metal) has increased for molybdenum (14%), gold (7%) and copper (2%); decreased for lead (12%), zinc (8%) and silver (3%), compared to the 2014 Ore Reserves for the Group inclusive of Las Bambas.

Ore Reserves (contained metal) at Las Bambas increased for molybdenum (14%), gold (6%) and copper (5%) but decreased for silver (4%) compared to the Ore Reserves disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to Las Bambas acquisition. Ore Reserve tonnes at Las Bambas have increased by 127Mt.

At all other sites Ore Reserve tonnage increases have almost offset mill depletion.

The Ore Reserve (contained metal) increases are due to:

- Increases in Mineral Resources at:
    - Las Bambas.
    - Golden Grove – zinc.
- Inclusion of new mineralisation zones into the Ore Reserves:
    - Sepon – inclusion of primary copper.
    - Rosebery – inclusion of X lens.
- Technical Investigations:
    - Las Bambas – Tailings Storage Facility (TSF) Prefeasibility study and metallurgical test work on the Sulfobamba mineralisation.

Contained metal decreases are primarily attributed to milling depletion:

- Century – accounts for the largest reduction, due to the completion of mining with only stockpiles remaining. Ore Reserves will be reconciled after the completion of processing.
- Golden Grove – copper.
- Kinsevere.

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[7] For the purpose of comparison, the Ore Reserves for Las Bambas used are those disclosed in the Competent Person's report prepared for the Circular released by the Company on 30 June 2014 in relation to the Las Bambas acquisition

KEY ASSUMPTIONS

ATTRIBUTABLE INTEREST

The following table details the attributable interest MMG has in all Mineral Resource and Ore Reserves stated within this report.

Table 1 : MMG's attributable interest for all projects.

Deposit	Attributable Interest
Las Bambas	62.5%
Kinsevere	100%
Sepon	90%
Dugald River	100%
Golden Grove	100%
Rosebery	100%
Century	100%
High Lake	100%
Izok Lake	100%
Avebury	100%

PRICES AND EXCHANGE RATES

The following price and foreign exchange assumptions, set according to the relevant MMG Standard as at January 2015, have been applied to all Mineral Resource and Ore Reserve estimates.

Table 2 : Price (real) and foreign exchange assumptions

	Ore Reserve	Mineral Resource
Cu (US$/lb)	2.95	3.50
Zn (US$/lb)	1.20 (1.18 if < 3 yrs)	1.45
Pb (US$/lb)	1.12	1.35
Au US$/oz	1010	1212
Ag US$/oz	21.10	25.50
Mo (US$/lb)	11.1	15.0
AUD:USD	0.82 (0.85 if <3 yrs)	As per Ore Reserves
CAD:USD	1.09	

CUT-OFF GRADES

Mineral Resource and Ore Reserve cut-off values are shown in Table 3 and Table 4 respectively.

Table 3 : Mineral Resources cut-off grades

Site	Mineralisation	Likely Mining Method[a]	Cut-Off Value	Comments
Las Bambas	Oxide Copper	OP	1% Cu	*In-situ* Copper Mineral Resources constrained within US$3.5/lb Cu pit shell.
	Primary Copper	OP	0.2% Cu	
Sepon	Oxide Gold & Stockpiles	OP	1.2-1.3 g/t Au	*In-situ* Gold Mineral Resources constrained within US$1212/oz Au pit shell. Cut-off values are dependent on processing costs, haul distance and recovery. No UG gold Mineral Resources have been considered.
	Partial Oxide	OP	3.3-4.5 g/t Au	
	Primary Gold	OP	1.7-2.3 g/t Au	
	Supergene Copper - Carbonate	OP	1.2% Cu	*In-situ* Copper Mineral Resources constrained within US$3.5/lb Cu pit shell
	Supergene Copper - Chalcocite	OP	1.1% Cu	
	Primary Copper	OP	0.5% Cu	
Kinsevere	Oxide Copper & Stockpiles	OP	0.6% ASCu[b]	*In-situ* Copper Mineral Resources constrained within a US$3.5/lb Cu pit shell
	Primary Copper	OP	0.8% TCu[c]	
Rosebery	Rosebery (Zn, Cu, Pb, Au, Ag)	UG	A$179/t NSRAR[d]	
	South Hercules (Zn, Cu, Pb, Au, Ag)	UG	A$179/t NSRAR[d]	
Golden Grove	Primary Zinc & Primary Copper (Zn, Cu, Pb, Au, Ag)	UG	A$145/t NSRAR[d]	
	Oxide & Partial Oxide & Stockpiles-Gossan Hill	OP	1.0% Cu	*In-situ* Mineral Resources constrained within the current mine design based on US$3.33/lb pit-shell above the 10255mRL.
	Oxide, Partial Oxide & Primary Gold – Gossan Hill	OP	1.1 g/t Au	Above 10240m RL reported
	Primary Copper – Gossan Hill	OP	1.0% Cu	*In-situ* Mineral Resources constrained within the current mine design based on US$3.33/lb pit-shell above the 10255mRL.
	Primary Zinc – Gossan Hill	OP	3% Zn	Above 10240m RL reported
Century	Century Pit, Eastern Fault Block & Stockpiles (Zn, Pb, Ag)	OP	3.5% ZnEq[e]	ZnEq[e] = Zn + 1.19*Pb based on price and metallurgical recovery constrained within the Century final pit shell
Dugald River	Primary Zinc (Zn, Pb, Ag)	UG	A$134/t NSRAR[d]	
	Primary Copper	UG	1% Cu	
Avebury	Ni	UG	0.4% Ni	
High Lake	Cu, Zn, Pb, Ag, Au	OP	2.0% CuEq[f]	CuEq[f] = 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%
		UG	4.0% CuEq[f]	
Izok Lake	Cu, Zn, Pb, Ag, Au	OP	4.0% ZnEq[e]	ZnEq = Zn + (Cu×3.31) + (Pb×1.09) + (Au×1.87) + (Ag×0.033); prices and metal recoveries as per High Lake

[a] : OP = Open Pit, UG = Underground, ASCu[b] = Acid Soluble Copper, TCu[c] = Total Copper,
NSRAR[d] = Net Smelter Return After Royalty, ZnEq[e] = Zinc Equivalent, CuEq[f] = Copper Equivalent, RL = Relative Level

Table 4 : Ore Reserves cut-off grades

Site	Mineralisation	Mining Method	Cut-Off Value	Comments
Las Bambas	Primary Copper Ferrobamba	OP	0.16-0.20%Cu	Range based on rock type recovery.
	Primary Copper Chalcobamba		0.18-0.24%Cu	
	Primary Copper Sulfobamba		0.22-0.43% Cu	
Sepon	Copper - LAC[a] sulphide material	OP	1.1% to 1.5% Cu	For non-primary materials, cut-off values are dependent upon pit haul distance to crusher and its estimated GAC[c] value.
	Copper – HAC[b] sulphide material		1.2% to 5.3%Cu	
	Copper – LAC[a] carbonate material		1.4% to 1.5%Cu	
	Copper – HAC[b] carbonate material		1.4% to 5.3% Cu	
	Primary		0.5% Cu	
Kinsevere	Copper Oxide	OP	0.8% to 1.2% ASCu[d]	Cut-off grade is 1.2% AsCu under current operating conditions and 0.8% at the cessation of mining activities.
Rosebery	(Zn, Cu, Pb, Au, Ag)	UG	A$179/t	NSRAR[e] Stopes with access already available applied a A$165/t cut-off grade
Golden Grove	Gossan Hill - Primary Zinc and Primary Copper (Zn, Cu, Pb, Au, Ag)	UG	A$145/t	NSRAR[e]
	Scuddles - Primary Zinc and Primary Copper (Zn, Cu, Pb, Au, Ag)	UG	A$140/t	
	Oxide Copper	OP	1.76% Cu	
Century	Zinc	OP	4.2% ZnEq[f]	ZnEq[f] = Zn + (1.19*Pb).
Dugald River	Primary Zinc	UG	A$134/t	

LAC[a] = Low Acid Consuming; HAC[b] = High Acid Consuming, GAC[c] = Gangue Acid Consuming, ASCu[d] = Acid Soluble Copper, NSRAR[e] = Net Smelter Return After Royalty[8], ZnEq[f] = Zinc Equivalent

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[8] Net Smelter Return is a measure of in-ground value of a metal grade or set of metal grades after all the realisation costs down-stream of the mill have been accounted for and effectively represents the dollar value at the mine gate of the in-ground minerals. NSRAR (NSR after Royalties) is similar to NSR but includes the cost effects of Royalties payable. See the following paper for a detailed explanation: Goldie, R. and Tredger, P., 1991. Net Smelter Return Models and Their Use in the Exploration, Evaluation and Exploitation of Polymetallic Deposits, *Geoscience Canada*, Vol 18, No. 4, pp 159-171

PROCESSING RECOVERIES

Output average processing recoveries are shown in Table 5. More detailed processing recovery relationships are provided in the Technical Appendix.

Table 5: Processing Recoveries

Site	Product	Recovery						Concentrate Moisture Assumptions
		Copper	Zinc	Lead	Silver	Gold	Mo	
Las Bambas	Copper Concentrate	82%	-	-	64%	60%		10%
	Molybdenum Concentrate						55%	5%
Century	Zinc Concentrate	–	79%	–	56%	–		-
	Lead Concentrate	–	–	68%	10%	–		-
Golden Grove - Underground	Zinc Concentrate		90%	–	-	–		9.5%
	Lead Concentrate	–	–	71%	59%	56%		9.5%
	Copper Concentrate	90%	–	–	59%	50%		9.5%
Golden Grove – Open Cut	Oxide Copper Concentrate	55%	–	–	–	–		16%
	Transition Copper Concentrate	55%	–	–	51%	64%		16%
Rosebery	Zinc Concentrate	-	87%	-	-	-		8%
	Lead Concentrate	-	6%	76%	40%	16%		6%
	Copper Concentrate	64%	-	-	42%	36%		7%
	Gold Doré				0.1%	22%		
Dugald River	Zinc Concentrate	–	87%		30%	–		10%
	Lead Concentrate	–		64%	22%	–		12%
Sepon	Copper Cathode	86%	–	–	–	–		–
Kinsevere	Copper Cathode	85% (96% ASCu)	–	–	–	–		–

a: Silver for Rosebery Gold Doré is calculated as a constituent ratio to gold in the Doré. Silver is set to 0.17 against gold being 20.7.

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

MMG Mineral Resources and Ore Reserves Statement

as at 30 June 2015

Technical Appendix

8 December 2015

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APPROVALS PAGE

	Richard Butcher	GM Technical Services	24/11/15
Signature	**Name**	**Position**	**Date**
	Jared Broome	Group Manager Geology	24/11/15
Signature	**Name**	**Position**	**Date**
	Geoffrey Senior	Group Manager Metallurgy	
Signature	**Name**	**Position**	**Date**

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

1. INTRODUCTION

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

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

This Technical Appendix provides these supporting details.

Under the JORC Code, reporting in compliance with the guidelines of JORC Code 2012 Edition 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).

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[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 Reserve 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 2015 Mineral Resource and Ore Reserves estimation at the date at which work commenced on the Mineral Resources and Ore Reserves are as shown in Table 1.

Table 1 Price (real) and foreign exchange assumptions

	Ore Reserve	Mineral Resource
Cu (US$/lb)	2.95	3.50
Zn (US$/lb)	1.20 (1.18 if < 3 yrs)	1.45
Pb (US$/lb)	1.12	1.35
Au US$/oz	1010	1212
Ag US$/oz	21.10	25.50
Mo (US$/lb)	11.1	15.0
AUD:USD	0.82 (0.85 if <3 yrs)	As per Ore Reserves
CAD:USD	1.09	
USD:PEN	2.90	

The key assumptions for cut-off grade calculation are those for critical commodities, inflation and interest rates from the October 31, 2014 corporate economic assumptions and metal strike prices approved by the CEO and CFO for 2015 Mineral Resources and Ore Reserves.

2.2 Metal Market Analysis – Basis for Pricing Assumptions

MMG's corporate economic assumptions for metal prices are derived from a combination of broker consensus and internal strategy evaluations.

2.2.1 Market Assessment – The Global Demand for Metals

Although commodity prices have adjusted downwards during the first half of 2015, consumption of metals over the longer term are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. The global economy grew at an estimated 2.6% in 2014, and the World Bank is forecasting global growth of 2.8% in 2015, rising to 3.2% in 2016-17. Economic recovery in higher-income countries is forecast to outpace growth in developing countries. Growth in China is forecast to slow only modestly, from 7.4% in 2014 to 7.1% in 2015 and 7.0% the following year[3].

2.2.2 Copper Demand and Supply

MMG has a long-term positive view of copper market fundamentals with future supply likely to be constrained as declining grades, increasing costs, slow future mine production and investment. Demand for copper is expected to increase as China and developing economies continue urbanisation and investment in infrastructure.

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[3] World Bank Group. 2015. Global Economic Prospects, June 2015: The Global Economy in Transition.

Copper Supply

MMG's long-term view centres on future copper supply contracting as current reserves are depleted. Supply is expected to be replaced by lower grade and higher cost operations in locations that are more remote and have higher geopolitical risk.

Declining grades are a significant issue among existing producers as mining companies process increased ore to maintain levels of production. This is having an impact on cost inflation and the incentivised price of copper for mining companies to invest in future projects.

Copper Demand

The global demand for copper is dominated by China which now accounts for approximately 46% of global consumption. Although China is reducing its reliance on infrastructure investment for economic growth in favour of consumer demand, the nation's demand for copper is still forecast to grow at above the global average rate. Key drivers of China copper demand include power generation, transportation and consumer goods. In addition to this, economic growth in other developing nations, especially in Asia, coupled with improved growth in the world's advanced economies will contribute to global demand growth going forward.

2.2.3 Zinc Demand and Supply

Zinc supply is expected to contract in the future as the market forecasts a supply deficit, given planned mine closures and a lack of major new development projects. Demand for zinc will be driven by its end use as a cost-effective anti-corrosive coating, improving the longevity of steel. Continued growth in the construction, transportation and infrastructure sectors especially in the developing economies, will support solid demand for zinc in the medium to long term.

Zinc Supply

The closure of the Century and Lisheen mines during Q4 this year will remove significant zinc concentrate supply from the market.

There are limited committed greenfield or brownfield zinc developments expected to commence operations in the short term. This is due to the 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, higher-cost underground mines as current reserves are depleted.

Zinc Demand

Demand for zinc is driven by its use in the galvanising of steel which is used mainly in building and construction, transport (including automotive) and consumer goods and appliances.

The end use of zinc is essential for the continuing industrialisation of the developing world.

The long-term outlook for zinc will be determined by the ability of miners to offset the impact of scheduled mine closures and growing demand.

2.3 Competent Persons

Deposit	Accountability	Competent Person	Professional Membership	Employer
MMG Mineral Resources and Ore Reserves Committee	Mineral Resources	Jared Broome	F.AusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Ore Reserves	Richard Butcher	F.AusIMM(CP)	MMG
MMG Mineral Resources and Ore Reserves Committee	Metallurgy: Mineral Resources/ Ore Reserves	Geoffrey Senior	MAusIMM	MMG
Las Bambas	Mineral Resources	Rex Berthelsen	F.AusIMM(CP)	MMG
Las Bambas	Ore Reserves	Richard Butcher	F.AusIMM(CP)	MMG
Sepon	Mineral Resources	Chevaun Gellie	MAusIMM	MMG
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Kinsevere	Mineral Resources	Douglas Corley	MAIG R.P.Geo.	MMG
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Golden Grove (Underground & Open Pit)	Mineral Resources	Paul Boamah	MAusIMM	MMG
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Avebury	Mineral Resources	Peter Carolan	MAusIMM	Formerly by 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] Member of the Association of Professional Engineers and Geoscientists of British Columbia

The Competent Person Consent and Compliance Statements have been signed by each Competent Persons. These Consent Statements are kept on file by MMG.

3 LAS BAMBAS OPERATION

3.1 Introduction and Setting

The Project is a world class copper gold (Cu-Au) mine located in the Andes of southern Peru approximately 75 km south-southwest of Cusco, approximately 300 km north-northwest of Arequipa, and approximately 150 km northeast of Espinar (also named Yauri). The Project 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.

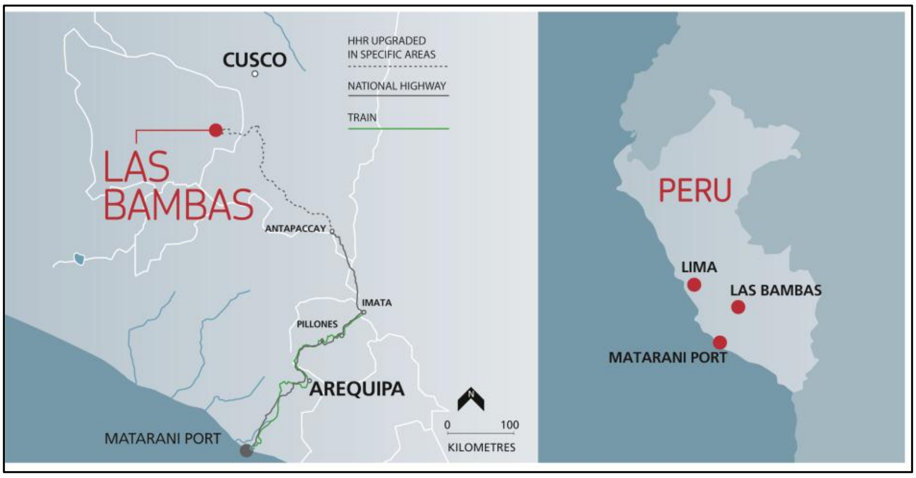


Figure 3-1: Las Bambas Mine location

Las Bambas is a truck and excavator mining operation with a conventional copper concentrator. As at 30 June 2015, overall project construction was on plan at 95% complete with concentrate-related construction at 90% completion. First production of concentrates is expected during the March quarter 2016. The commissioning of the 130 kilometre 220kV power transmission line to site and mine power loop occurred in June quarter. This allowed the site to ramp up pre-stripping activities, with approximately 23 million tonnes of material moved to date. As well, MMG entered into contracts for the transportation of copper concentrates by road and rail from Las Bambas to the Port of Matarani.

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 Resource and Ore Reserve have been completely re-estimated for the June 2015 release. The 2015 Mineral Resource estimation includes all recent drilling undertaken during 2014-2015, updated geological interpretation, block model parameters and extents.

3.2 Mineral Resources – Las Bambas

3.2.1 Results

The 2015 Las Bambas Mineral Resource is summarized in Table 2. The Las Bambas Mineral Resource is inclusive of the Ore Reserve.

Table 2 2015 Las Bambas Mineral Resource tonnage and grade (as at 30 June 2015)

Las Bambas Mineral Resource									
						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	21	1.9				399			
Inferred	6	1.7				99			
Total	**27**	**1.8**				**498**			
Ferrobamba Primary Copper[2]									
Measured	388	0.8	3.7	0.07	204	2,953	46	0.9	79
Indicated	490	0.6	2.9	0.05	209	3,172	45	0.8	102
Inferred	452	0.6	2.2	0.03	148	2,555	32	0.5	67
Total	**1,330**	**0.7**	**2.9**	**0.05**	**187**	**8,680**	**123**	**2.2**	**248**
Ferrobamba Total						**9,178**	**131**	**2.4**	**252**
Chalcobamba Oxide Copper[1]									
Indicated	5.9	1.4				84			
Inferred	0.5	1.5				8			
Total	**6.4**	**1.4**				**92**			
Chalcobamba Primary Copper[2]									
Measured	96	0.41	1.3	0.02	151	391	4	0.0	15
Indicated	190	0.61	2.3	0.03	138	1,154	14	0.2	26
Inferred	41	0.46	1.5	0.02	122	188	2	0.0	5
Total	**327**	**0.53**	**1.9**	**0.02**	**140**	**1,733**	**20**	**0.2**	**46**
Chalcobamba Total						**1,824**	**21**	**0.3**	**46**
Sulfobamba Oxide Copper[1]									
Inferred	0.02	2.8				0.4			
Total	**0.02**	**2.8**				**0.4**			
Sulfobamba Primary Copper[2]									
Indicated	102	0.63	4.4	0.02	164	639	14	0.1	17
Inferred	214	0.45	4.2	0.02	117	964	29	0.1	25
Total	**315**	**0.51**	**4.3**	**0.02**	**132**	**1,603**	**43**	**0.2**	**42**
Sulfobamba Total						**1,603**	**43**	**0.2**	**42**
Total Contained Metal						**12,606**	**195**	**2.8**	**340**

1 1% Cu Cut-off grade contained within a US$3.5/lb pit shell for oxide material.

2 0.2% Cu Cut-off grade contained within a US$3.5/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.

4.1 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 Resource 2015

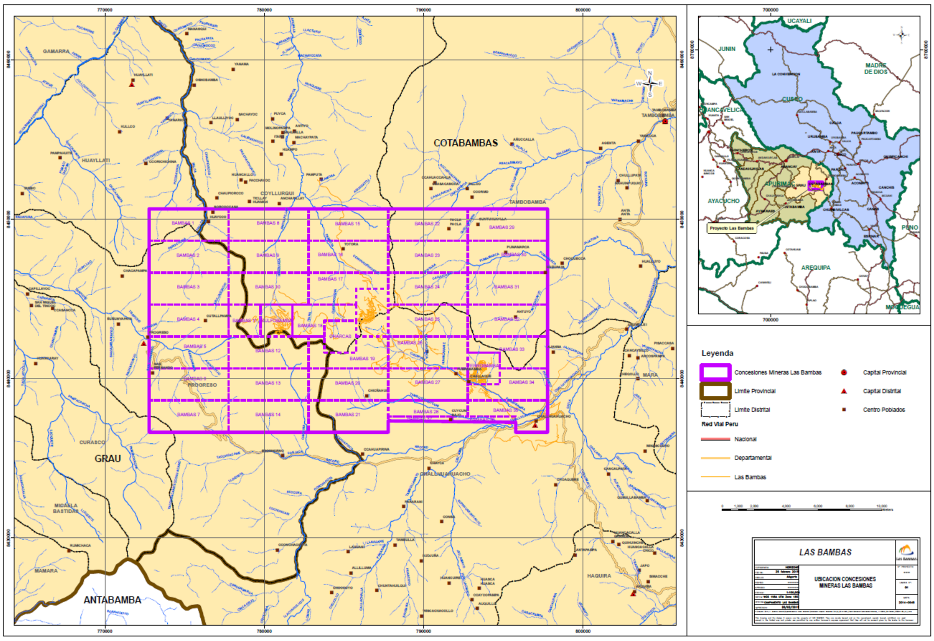
Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• Diamond drilling (DD) was used to obtain an average 2 m sample that is half core split, crushed and pulverised to produce a pulp (95% passing 105 µm). Diamond core is selected, marked and numbered for sampling by the logging 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 present) for half core splitting and sample preparation. • 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 drill holes used in the Mineral Resource estimate 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 333,279 m of the total 363,864 m of diamond drilling in the data used for Mineral Resource estimation for the Sulfobamba, Chalcobamba and Ferrobamba deposit. Diamond drill recovery average is about 96% for all deposits (97% for Sulfobamba, 96% for Chalcobamba and Ferrobamba deposits). • 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 estimate has been geologically and geotechnically logged to support Mineral Resource estimation, mining and metallurgy studies. • Geological logging is qualitative and geotechnical logging is quantitative. All drill core is photographed.
Sub-sampling techniques and sample	• All samples included in the Mineral Resource estimate 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

preparation	the original sample tray. The standard sampling length is 1.20 m for HQ and a maximum of 2.20 m. 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 105 µm. 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 105 µm. Sizing analysis is 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 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 assay methods undertaken by Inspectorate (Lima) for Las Bambas were as follows: ○ Cu, Ag, Pb, Zn, Mo - 0.5 g of sample. Digestion by 4-Acids. Reading by Atomic Absorption Spectrometry (AAS). ○ Acid soluble - 0.2 g sample. Leaching by a 15% solution of H_2SO_4 at 73°C for 5 min. Reading by AAS. ○ Acid soluble - 0.2 g of sample. Digestion by a citric acid solution at 65°C for 15 min. Reading by AAS. ○ Au - Cupellation at 950°C. Reading by AAS. Above detection limit analysis by gravimetry. ○ 35 elements - Digestion by aqua regia and reading by ICP. • From 2010 to present routine assay methods undertaken by Certimin (Lima) for Las Bambas are as follows: ○ Cu, Ag, Pb, Zn, Mo - 0.5 g of sample. Digestion by 4-Acids. Reading by Atomic Absorption Spectrometry (AAS). ○ Acid soluble copper – 0.2 g sample. Leaching by a 15% solution of H_2SO_4 at 73°C for 5 min. Reading by AAS. ○ Acid Soluble copper - 0.2 g of sample. Digestion by a citric acid solution at 65°C for 15 min. 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). Since 2014 independent umpire laboratory checks have not occurred, work is underway to re-commence routine independent umpire laboratory checks. • Inspectorate and Certimin 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: o Primary coarse duplicates: Inserted 1 in every 25 samples (2005-2007), every 50 samples (2008), and every 40 samples (2010- present). o Coarse blank samples: Inserted after a high-grade sample (coarse blank samples currently make up about 4.2% of all samples analysed). o Pulp duplicates samples: Inserted 1 in every 25 samples (2005-2007), every 50 samples (2008), and every 40 samples (2010-present). o 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). o Certified Reference Material (CRM) samples: Inserted 1 in every 50 samples (2005-2006), every 40 samples (2007) and every 20 samples (2008 to present). • QAQC analysis has shown that: o Blanks: a minimum level of sample contamination by Cu 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>0.90). These results were also repeated in the external ALS check samples. o Standard Reference Material: acceptable levels of accuracy and precision have been established. • Sizing tests 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. The personnel completing the above listed checks are not necessarily the Competent Person. • Apart from 20 metallurgical drillholes drilled in 2007 twinning Mineral Resource Ferrobamba drill holes, no twinned drill holes have been completed. To date site

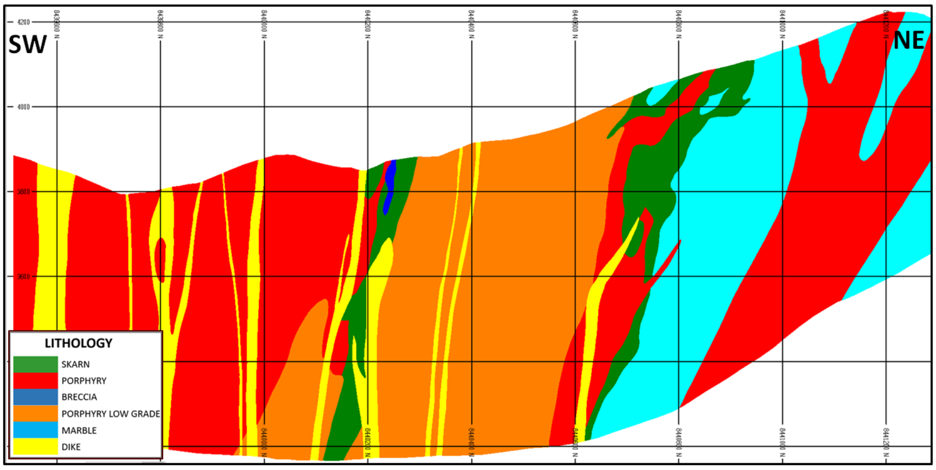
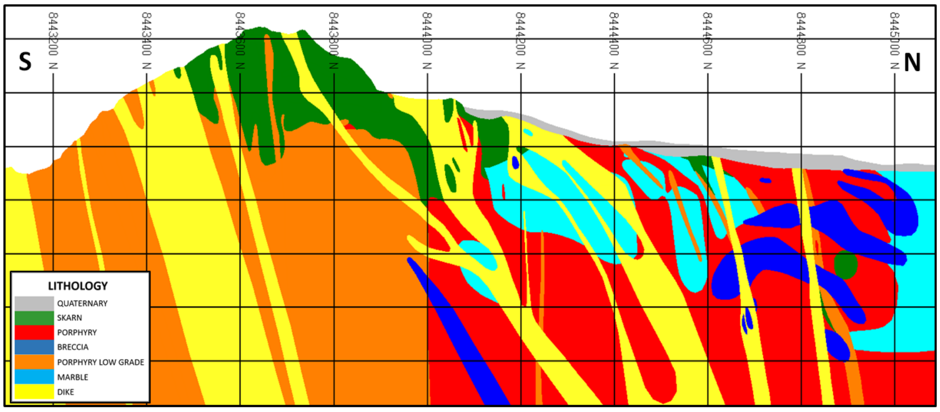
	geologists have elected not to assess the short-range variability of the deposit. • All drill holes are logged using laptop computers directly into the drill hole database (acQuire). Prior to November, 2014 diamond drill holes were logged on paper and transcribed into the database. Assay results are 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 to be stored. Additional manual data validation occurs in Vulcan software before data is used for interpretation and Mineral Resource modelling. Unreliable data is flagged and excluded from Mineral Resource estimation work. • No adjustments have been made to assay data – if there is any doubt about the data quality or location, the drill hole is excluded from the estimation process.
Location of data points	• In 2005 collar positions of surface drill holes 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, drill holes are set out using UTM co-ordinates with a hand held DGPS and are accurate to within 1 m. 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 0.5 m. • 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 undertake independent checking of any Sulfobamba drillholes. The collar locations are considered accurate for Mineral Resource estimation work. • In 2005 the drilling contractor conducted down-hole survey’s using the AccuShot method for non-vertical drill-holes. Vertical holes were not surveyed. If the AccuShot arrangement was not working, the acid test (inclination only) was used. Since 2006, all drill holes are surveyed using Reflex Maxibor II equipment units which take measurements every 3 m. The down-hole surveys are considered accurate for Mineral Resource 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 10 m 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 Resource and Ore Reserve modelling purposes. The topography, however will be updated in the near future.
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 100 m x 100 m to 25 m x 25 m and is considered sufficient to establish the degree of geological and grade continuity appropriate for Mineral Resource estimation processes and Mineral Resource classifications applied. • Drill hole spacing of approximately 25 m x 25 m within skarn hosted material and 50 m x 50 m within porphyry hosted is considered sufficient for long term Mineral

	Resource estimation purposes based on a drill hole spacing study undertaken in 2015. While the 25 m spacing is suitable for Mineral Resource estimation, the Las Bambas deposits tend to have short scale 5 m - 10 m variations within the skarn that are not captured by the infill drilling at this spacing. This localised geological variability is captured by mapping and drill hole logging. • Diamond drill hole samples are not composited prior to routine chemical analysis; however the nominal sample length is generally 2 m. All sequential copper analysis is undertaken on pulps that are composited most commonly to 8 m but sample lengths as small as 1 m are contained in the database.
Orientation of data in relation to geological structure	• Overall drill hole orientation is planned at 90 degrees to the strike of the mineralised zone for each deposit. Drill hole 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 drill holes 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: ○ Adequately trained and supervised sampling personnel. ○ Samples are stored in a locked compound with restricted access during preparation. ○ Dispatch to various laboratories via contract transport provider in sealed containers. ○ Receipt of samples acknowledged by receiving analytical laboratory by email and checked against expected submission list. ○ 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 Certimin laboratory in Lima.
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.

	 • Processing of ores requires the approval of a Beneficiation Concession. Las Bambas is working towards receiving the grant of the Beneficiation Concession before the end of 2015. • Tenure over the 41 Concessions is in good standing. There are no known 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 over 343 km 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 resource 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	1966	Chalcobamba	Exploration	DDH	6	Unknown	906.44
Cyprus	1996	Chalcobamba	Exploration	DDH	9		1,367.31
Phelps Dodge	1997	Ferrobamba	Exploration	DDH	4	Unknown	737.80
		Chalcobamba			4		653.40
BHP	1997	Ferrobamba	Exploration	DDH	3	Unknown	365.80
		Chalcobamba			4		658.55
Teck	1998	Chalcobamba	Exploration	DDH	4	Unknown	875.10
Pro Invest	2003	Ferrobamba	Exploration	DDH	4	HQ	738.00
		Chalcobamba			7		1,590.00
Xstrata	2005	Ferrobamba	Resource Evaluation	DDH	109	HQ	26,839.90
		Chalcobamba			66		14,754.10
		Sulfobamba			60		13,943.00
	2006	Ferrobamba	Resource Evaluation	DDH	125	HQ	51,004.15
		Chalcobamba			95		27,982.90
		Sulfobamba			60		16,971.45
		Charcas			8		2,614.05
		Azuljaja			4		1,968.85
	2007	Ferrobamba	Resource Evaluation	DDH	131	HQ	46,710.35
		Chalcobamba			134		36,617.55
		Sulfobamba			22		4,996.60
	2008	Ferrobamba	Resource Evaluation	DDH	111	HQ	43,564.10
		Chalcobamba			90		22,096.60
	2010	Ferrobamba	Resource Evaluation	DDH	92	HQ	28,399.85
MMG	2014	Ferrobamba	Resource Evaluation	DDH	23	HQ	12,609.70
	2015	Ferrobamba	Resource Evaluation	DDH	36	HQ	17,373.55
			Total				376,339.10

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. 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.
Drill hole information	• No exploration diamond drill holes have been completed in the 2015 reporting period. Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate. • Drill hole data is not provided in this report as this report is for the Las Bambas Mineral Resource which use all available data and no single hole is material the Mineral Resource estimate.
Data aggregation methods	• Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate. • No metal equivalents were used in the Mineral Resource estimation.
Relationship between mineralisation width and intercepts	• No exploration diamond drill holes have been completed in the 2015 reporting period. Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate. • Mineralisation true widths are captured by interpreted mineralisation 3D wireframes. Drill holes are drilled to achieve intersections as close to orthogonal

lengths	as possible.
Diagrams	- No exploration diamond drill holes have been completed in the 2015 reporting period. Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate.  **Schematic Section Through Ferrobamba**  **Schematic Section Through Chalcobamba**
Balanced reporting	- No exploration diamond drill holes have been completed in the 2015 reporting period. Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate.
Other substantive exploration data	- No exploration diamond drill holes have been completed in the 2015 reporting period. Historical exploration drill holes (prior to 2005) have been excluded from the Mineral Resource estimate. - In the past year several orebody knowledge studies have been carried out including skarn zonation, vein densities and a large age dating programme. Results are pending for many of these studies.
Further work	- A program of exploration assessment and targeting is currently underway to identify exploration options for the Las Bambas leases. - Ongoing drill programs will be planned to increase deposit confidence as the need arises.
Section 3 Estimating and Reporting of Mineral Resources	
Database Integrity	- The following measures are in place to ensure database integrity: o All Las Bambas drill hole data is stored in an SQL database (acQuire) on

	the Las Bambas site 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 November 2014, diamond drill holes were logged on paper logging forms and transcribed into the database. From November 2015 logging is entered directly into a customised 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, Rex Berthelsen has undertaken three site visits to Las Bambas in the past year. In the view of the Competent Person there are no material risks to the Mineral Resource 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 current practice of "double blind" sample randomisation at the laboratory is commendable as it essentially guarantees the secrecy of the results from the operating laboratory. However, in the view of the Competent Person, this level of complication is no longer necessary and has the potential to irreversibly compromise the integrity of the results if errors occur during the process. It is recommended this practice cease.
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 original geological interpretation as undertaken by site geologists in 2010 was used as the basis for the 2015 geological model. • The 2015 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 25 m at Ferrobamba to 50 m at Sulfobamba. The diamond drilling geological logging, assay results 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 2015 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. • Mineralisation and alteration domains were produced for the Ferrobamba and Chalcobamba models. Mineralisation domains were based on logged mineral species and acid soluble copper to total copper assay ratios. Alteration domains were based on logged observations. • 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 Group Office geologists. • 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 Resource refers to three distinct deposits; each have been defined by drilling and estimated: o Ferrobamba Mineral Resource occupies a footprint which is 2500 m N-S and 1800 m E-W and over 900 m vertically. o Chalcobamba Mineral Resource occupies a footprint of 2300 m N-S and 1300 m E-W and 800 m vertically o Sulfobamba Mineral Resource is 1800 m along strike in a NE direction and 850 m across strike in a NW direction and 450 m deep.
Estimation and modelling techniques	• Mineral Resource 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 and Sulfobamba block models utilised sub-blocking, while the Chalcobamba block model was estimated directly into a regular block model with no sub-blocking. Estimates of each domain were combined post-estimate by density weighted averaging. 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 boundary between the low and the high-grade skarn were treated as hard boundaries. o Data compositing for estimation was set to 4 m, which matches two times the majority of drill hole sample lengths (2 m), provides good definition across interpreted domains, and adequately accounts for bench height.

	CuAs and CuCn was composited to 8 m 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. All variables are comparably informed and independently estimated. o 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 Resource estimations after additions due to drilling and re-modelling or the site. • As production has not yet commenced no reconciliation data is available to compare. • Assumptions about the recovery of by-products is accounted in the net-smelter return after royalty (NRSAR) 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 20 m x 20 m x 15 m with sub-blocks of 5m x 5m x 5m. Chalcobamba block size was set to 30 m x 30 m x 15 m. The block size at Sulfobamba was set to 50 m x 50 m x 15 m (sub-blocked to 25 m x 25 m x 7.5 m which roughly equates to the drill spacing. The search anisotropy employed was based on both the ranges of the variograms and the drill spacing. • 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: o Visual inspections for true fit for all wireframes (to check for correct placement of blocks and sub-blocks). o Visual comparison of block model grades against composite sample grades. o Global statistical comparison of the estimated block model grades against the declustered composite statistics. o Change of support analysis on major lithological domains. o 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 Resource is reported above 0.2% Cu cut-off grade for hypogene material. Oxide material has been reported above a 1% Cu cut-off grade. The

parameters.	reported Mineral Resources have also been constrained within a US$3.50/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 t trucks and 100 t face shovels will be used for material movement. The minimum mining width is expected to be 20 m which is in line with the horizontal dimensions of the block size of the model. • During block regularisation, internal dilution is included to produce full block estimates. • No other mining factors have been applied to the Mineral Resource.
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 Resource 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 Resource which is expected to be converted to Ore Reserves and treated in the onsite concentrator facilities. • The calc-silicate lithologies (including marble) containing estimated copper above the cut-off grade have been reported in 2015. • No other metallurgical factors have been applied to the Mineral Resource.
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 between 30% to 40% of waste rock from Sulfobamba would be potentially acid forming (PAF). • Tailings generated from processing of Ferrobamba and Chalcobamba were determined to be NAF. Geochemical characterisation of tailings generated from processing of Sulfobamba ores have not been conducted however for environmental assessment purposes it was assumed to have PAF behaviour. Processing of Sulfobamba ores is expected to be completed after those of Ferrobamba and Chalcobamba which would leave tailings generated from Sulfobamba effectively encapsulated in the TSF and therefore not requiring any additional mitigation activities.
Bulk Density	• Bulk density is determined using the Archimedes' principle (weight in air and weight in water method). Samples of 20 cm 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 Resource 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. • Density data from the recent drilling campaign (2014/2015) was not imported into the database in time for Mineral Resource modelling. As such this information was not used in the 2015 Mineral Resource estimate. This information represents 59 drill holes out of a total 627 drill holes used in the estimate.
Classification	• Mineral Resource classifications used criteria that required a certain minimum number of drill holes. The spatial distribution of more than one drillhole ensures that any interpolated block was informed by sufficient samples to establish grade continuity. As well, drill hole spacing specific to rock type (skarn vs. porphyry) were used to classify each Mineral Resource category. • Drill hole spacing for classification were based on an internal Ferrobamba drill hole spacing study undertaken in 2015. Results from the study indicate: o Measured Mineral Resource: 25 m x 25 m drill hole spacing in the skarn, 50 m x 50 m drill hole spacing for the porphyry. o Indicated Mineral Resource: 50 m x 50 m drill hole spacing in the skarn, 100 m x 100 m drill hole spacing for the porphyry. o Inferred Mineral Resources are generally defined as twice the spacing of Indicated 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	• MMG has an internal 'Mineral Resource and Ore Reserve Policy' that requires at a minimum external reviews every three years, and internal company review every interim year by MMG representatives reporting findings to a sub-set of the MMG Mineral Resource and Ore Reserve Committee. • 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 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 Resource update. • A self-assessment of all 2015 Mineral Resource modelling was completed by the Competent Person and MMG Senior Resource Geologist in August 2015 using a standardised MMG template. • No fatal flaws were detected in the review. Areas identified for improvement included: o Mineral Resource classification for the Ferrobamba block model would benefit from a wireframe shape to constrain the final Mineral Resource

	category this is to avoid spuriously coded blocks. ○ Review how acid soluble and total copper values are collected and how they can be estimated in the Mineral Resource.
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 Resource. Minor, local variations are expected to occur on a sub-25 m 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. In most cases differences were significantly less than 10%. 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 estimate is considered suitable for Ore Reserves and mine design purposes. The Mineral Resource 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 Resource model should be a reasonable predictor of tonnes and grade selected during mining. • While no production has occurred as at the effective date of this Report, pre-stripping commenced in 2014. Initial ore production is forecast to commence in late 2015 and full production planned in 2016. As such no reconciliation data is available. • The accuracy and confidence of this Mineral Resource estimate is considered suitable for public reporting by the Competent Person.

4.2 Ore Reserves – Las Bambas

4.2.1 Results

The 2015 Las Bambas Ore Reserve are summarised in Table 4.

Table 4 2015 Las Bambas Mineral Resource tonnage and grade (as at 30 June 2015)

Las Bambas Ore Reserve									
						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	424	0.7	3.4	0.08	187	3,014	47	1.1	79
Probable	360	0.6	2.8	0.06	187	2,288	33	0.7	67
Total	**784**	**0.7**	**3.2**	**0.07**	**187**	**5,302**	**79**	**1.8**	**147**
Chalcobamba Primary Copper[2]									
Proved	77	0.5	1.5	0.02	155	357	4	0.0	12
Probable	150	0.7	2.6	0.03	137	1,042	13	0.2	21
Total	**227**	**0.6**	**2.2**	**0.03**	**143**	**1,398**	**16**	**0.2**	**32**
Sulfobamba Primary Copper[3]									
Probable	68	0.8	5.5	0.03	176	519	12	0.1	12
Total	**68**	**0.8**	**5.5**	**0.03**	**176**	**519**	**12**	**0.1**	**12**
Total Contained Metal						**7,219**	**108**	**2.0**	**191**

1 0.16% to 0.2% Cu Cut-off grade based on rock type and recovery

2 0.18% to 0.24% Cu Cut-off grade based on rock type and recovery

3 0.22% to 0.43% 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.

4.3 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 2015

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td>**Mineral Resource estimate for conversion to Ore Reserves**</td><td>• Mineral Resource Block Models have been provided by the Ore Control & Modelling group and Corporate Technical Services Group (MMG) in different memos. The block models contain descriptions for Lithology, Category, mineralization, GMU, and other variables described in each memo mentioned lines below. These block models were used for the pit optimization purpose using reasonable assumptions for cost and metal prices, MineSight 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>Elaborated by</td><td>Rex Berthelsen & Modelling Team at LB</td><td>Rex Berthelsen & Modelling Team at LB</td><td>Anna Lewin & Modelling Team at LB</td></tr>
<tr><td>Memorandum date</td><td>August 17, 2015</td><td>June 07, 2015</td><td>June 23, 2015</td></tr>
<tr><td>ASCII file</td><td>fe20150817_reg.rar (1)</td><td>cha_est_07_06_2015.asc</td><td>sulfo_2015_v1.3R_252515.bmf</td></tr>
<tr><td>Bloque Size</td><td>20 x 20 x 15</td><td>30 x 30 x 15</td><td>25 x 25 x 15</td></tr>
<tr><td>Model Rotation</td><td>35º</td><td>0º</td><td>0º</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, Richard Butcher, has visited the Las Bambas site no less than 4 times in the past year. Outcomes from the visits have included action plans to investigate matters related to Ore Reserve modifying factors including: Geotechnical studies, pit slope angles, tailings and waste storage, metallurgical recoveries and testing and open pit design criteria. All matters have been investigated and included in the Las Bambas Ore Reserve estimation.
• Site visits were also carried out by Nan Wang, Jared Brome and Rex Berthelsen from GTS – MMG.</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:
o Bechtel Feasibility Study 2010
o Tailings Storage Pre-Feasibility Study, 2015
• Additional work/studies include:
o Glencore Mineral Resources and Ore Reserves Report 2013
o Audit of Las Bambas Ore Reserves 2013, elaborated by Mintec, Inc in October 2013.</td></tr>
</table>

Assessment	Commentary
	○ MMG Competent Person Report prepared by Runge Pincock Minarco (RPM) ○ MMG Las Bambas Cut-Off Grade Report 2015 ○ Geotechnical Review, Itasca, 2015 ○ Sulfobamba Metallurgy Testing, 2015

Cut-off parameters

- Metal prices for the Cut-Off calculation have been provided by MMG Group Technical Services in accordance with the MMG MROR policy.

	Units	2015
ORE RESERVES		
Metal Commodity Prices		
Copper	US$/lb	2.95
Molybdenum	US$/lb	11.10
Silver	US$/oz	21.10
Gold	US$/oz	1010.00
Exchange Rates		
Peruvian Sole	USD/PEN	2.90
MINERAL RESOURCES		
Metal Commodity Prices		
Copper	US$/lb	3.50
Molybdenum	US$/lb	15.00
Silver	US$/oz	25.50
Gold	US$/oz	1212.00
Exchange Rates		
Peruvian Sole	USD/PEN	2.90

- Costs were estimated based on information provided by the Las Bambas Finance group.
- The breakeven Cut-off (BCoG) 2015 has been calculated with updated metal prices and costs, and is applied to the CuT%. (Source: 2015 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 GMU				
	FSSL	FSSM	FPSL	FPSM	FBRE
$\text{BCOG}_{\text{inpit}}$	0.16%	0.18%	0.16%	0.19%	0.20%

Cut-Off grades by ore-type for Chalcobamba:

COG Component	Chalcobamba by GMU						
	CSSL	CSSM	CSML	CSMM	CPSL	CPSM	CBRE
$\text{BCOG}_{\text{inpit}}$	0.18%	0.22%	0.18%	0.21%	0.18%	0.22%	0.24%

Cut-Off grades by ore-type for Sulfobamba:

COG Component	Sulfobamba by GMU		
	SSKR	SPOR	SBRE
$\text{BCOG}_{\text{inpit}}$	0.22%	0.23%	0.43%

Mining factors or assumptions

- The method of Ore Reserve 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

Assessment	Commentary

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 tuck and shovel operation. This method and selected mining equipment is appropriate for these deposits.

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

Assessment	Commentary
	• To improve the reliability and safety of the pit design at Ferrobamba, decoupling berms have been included on all sectors, where previously some sectors did not have them. • The 2015 Mineral Resource models for each of Ferrobamba, Chalcobamba and Sulfobamba have been used for the updated 2015 Ore Reserve. • The pit optimization was developed for the three open pits based on the 2015 Mineral resource block models, the strategy for the final pit selection was based on the NPV by pit shell at different revenue factors (RF), for Ferrobamba the final pit was selected at RF=0.78, and for Chalcobamba and Sulfobamba, the final pit shell was selected at RF=1.0. These selections provide a good balance of NPV and global Ore Reserve. Final pit designs that incorporate more practical mining considerations were carried out using those shells. • Dilution and recovery have been accounted for in the regularised block model used for the Ore Reserve estimate. Hence, the Ore Reserve estimate has applied no further factoring. Additional studies for dilution and recovery will be undertaken when reconciliation data is available. • In the pit, the minimum mining width is 70m; the minimal SMU has been set at 10m x 10m x 15m. • Inferred Mineral Resource material has not been included in the pit optimization and/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 infrastructures for Chalcobamba pit have been identified, with 50% of the north waste dump not on within the property boundary. • The main Sulfobamba infrastructure has been identified within the Las Bambas mining concession, however they are not located within the property boundary.

Assessment	Commentary
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 mineralization. • 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. Production will begin in the first quarter of 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 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 and for Sulfobamba, the data analyzed were those from testing at G&T in 2015 and documented in the report by He (S. He, 2015). • The ore contains no deleterious elements. • The recovery equations have been provided by Metallurgical Group at Las Bambas in coordination with MMG Group Technical Services. • The equations were generated based on metallurgical test work, based on information sourced from diamond drilling in the different pits. It should be noted that the copper recovery is a function to the ratio of acid soluble copper (CuSAc) to total copper (CuT), which is a determining factor for the recovery.The copper recovery is determined by the following equations. ***Ferrobamba:*** $Cu=(101.03-98.9*CuSAc/CuT)*0.95$ ***Chalcobamba:*** $Cu=94.9-90*(CuSAc/CuT)$ ***Sulfobamba:*** $Cu=(91.92-82.9*CuSAc/CuT)*0.97$ • The recovery of Mo, Ag, Au, has been provided by Metallurgical Group at Las Bambas.

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>
<table>
<tr><th>Metal</th><th>Ferrobamba</th><th>Chalcobamba</th><th>Sulfobamba</th></tr>
<tr><td>Mo %</td><td>55.50</td><td>55.50</td><td>55.50</td></tr>
<tr><td>Ag %</td><td>72.70</td><td>54.30</td><td>54.30</td></tr>
<tr><td>Au %</td><td>65.00</td><td>56.60</td><td>56.60</td></tr>
</table>

Benchmarking of data from four other mines in South America has also been completed and the recovery algorithms that describe performance at these mines compared with those proposed for the three Las Bambas deposits. The four other mines were Tintaya, Toquepala, Cerro Verde and Antamina.

</td></tr>
<tr><td>Environmental</td><td>

The Environmental Impact Study for the Las Bambas Project was approved on the 07th of 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 on May 31st, 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 on June 11, 2013 through Directorial Resolution N°187-2013-MEM-AAM
A first amendment to the Environmental Impact Study was approved on the 14th of August 2013, whereby amendments to the Capacity of the Chuspiri water reservoir and changes to the environmental monitoring program were approved.
On 17th September 2013 the relocation of the molybdenum plan, filter plant and concentrate storage shed from the Antapaccay region to the Las Bambas project area was approved by the environmental regulator through Directorial Resoultion Nº319-2013-MEM-AAM, after assessment of the technical report showed that the environmental impacts of the proposed changes were not significant.
On the 6th of November 2013, the General Directorate of Mining approved the amendment of the construction permit for the processing facilities to allow the construction of the molybdenum plant, the filter plant and the concentrate storage shed at the Las Bambas project area.
On the 17th November 2014, the second modification of the Study of Environmental Impact was approved with resolution Nº 559-2014-EM/DGAAM whereby changes to the water management infrastructure were approved.
Minor changes to project layout were also approved on February 13th, 2014 through Directorial Resolution N°078-2014-MEM-DGAAM and on 26 February 2015, through Directorail Resolution N°113-2015-MEM-DGAAM
As per Peruvian regulatory requirements an update of the Mine Closure Plan is being prepared and will be submitted in 2016; similarly an update of the

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

Assessment	Commentary
	project´s Environmental Impact Study will be submitted in 2017. • Geochemical characterization studies on waste rock samples were conducted in 2009/2010 as part of the Environmental Impact Study, 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. 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. • Further geochemical characterization studies on waste rock and tailings samples are currently underway to support the Environmental Impact Study Update. • The operation of the Ferrobamba waste rock dump was approved on 29th September 2015 by the General Directorate of Mining through Directorial Resolution N°1780-2015-MEM/DGM.
Infrastructure	• Las Bambas has the following infrastructure established on the site. o Planned Concentrator currently in the commissioning phase. o 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 784 Mt of tailings from processing 800 Mt. Three studies have been conducted looking at increasing Tailings storage capacity at Las Bambas: –Tailings characterisation 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. o Camp accommodation for staff o Water supply is sufficient for site and processing, sourced from the following: Challhuahuacho River, rainfall and runoff into Chuspiri dam, groundwater wells, contact waters, recirculating water in the process plant. o Transport of the concentrate will be performed by trucks to the Imata Village, then from that point it will be transported by train up to Matarani Port in Arequipa region. o There are principal Access roads that connect Las Bambas and national routes, Cotabambas. Cusco & Cotabambas-Arequipa o The source of energy comes from the national grid 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 in Lima and Melbourne Group office personnel.

Assessment	Commentary
	○ Chalcobamba pit operation requires additional purchase of land to the North side of the lease (Waste dump 2). ○ Sulfobamba pit operation requires additional purchase of land for the pit and other infrastructure.
Costs	• Construction of the Project is currently about 98% complete; accordingly, projected capital costs are soundly established. The costs are actuals based on tenders and local conditions. • The operating costs used for Ore Reserves estimation are based on the LOM2015 projections (Financial Model LOM2015 v3.7) based on tendered prices for consumables and estimated quantities of consumables, labour, and services. Las Bambas considers that these costs are representative of the best estimate of future costs for the operation. • 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 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 are based on the usual charges commonly seen in the last five years. • Royalties payable are based on information provided by the Finance and Commercial Group at Las Bambas. • Sustaining capital costs have been included in the pit optimization as mining cost as well as processing costs, 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 the guidelines of Corporate (MMG) according the objective of each capital expenditure in the operation.
Revenue factors	• All mining input parameters are based on the Ore Reserve estimate LOM production schedule. All cost inputs are based on tenders and estimates from contracts in place as with net smelter returns (NSR) and freight charges. These costs are comparable with the regional averages. MMG has based its metal prices on long-term bank consensus forecast of US $2.95 Cu, Molybdenum price: $11.10/lb; Silver price: $21.10/oz; Gold price: $1010/oz • The Gold and Silver revenue is via a credit at the refinery. • The Treatment charges and Refining costs have been included in the revenue calculation for the project.
Market assessment	• It is proposed that the majority of the product will be sold to Chinese customers. • MMG has a long term positive view of copper market fundamentals with future supply likely to be constrained by declining grades, increasing costs, slowing future mine production and investment. Demand for copper is expected to increase as China and developing countries' economies continue rural urbanization and investment in infrastructure.

Assessment	Commentary
Economics	• The costs are based on the LOM2015 projections (Financial Model LOM2015 v3.7) based on tendered prices for consumables and estimated quantities of consumables, labour, and services. • The Financial model of the Mine Plan (version LOM 2015 v3.7) shows that the mine has a substantially positive NPV. MMG uses an appropriate discount rate for the size and nature of the organization and deposit. • No sensitivity analyses were undertaken for the Ore Reserves work.
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 re-settlement of the rural community of Fuerabamba. The township of Nuevo Ferrobamba has been constructed and almost all families re-settled. • During the general extraordinary meeting in January 2010, Fuerabamba 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 the 13 thematic areas. • Las Bambas Project provides important support to the community in the areas of agriculture, livestock and other social projects, which is well received. • Regional unrest near the Las Bambas project in late 2015 was the result of protests brought about by highly organised groups, from outside the direct area of influence of Las Bambas. The protestors claimed a lack of consultation on modifications to the approved Environmental Impact Assessment. However, these modifications were fully approved by all responsible government departments in November 2014. The situation was also driven by current regional politics. The Las Bambas team has maintained positive dialogue with communities in the region of Apurimac for close to ten years of project development. MMG will continue to do so. The situation has currently stabilised. MMG is confident that significant social benefits have been, and will continue to be, provided, while protecting the environment and heritage of the region.
Other	• Las Bambas owns 7,718 Ha of land within the mining project. • The mining concession totals an area of 35,000 hectares, which includes the area containing the 3 open pits and their corresponding infrastructures. • Only 10% of the concession of Las Bambas has been explored year to date. • According to DR N° 187-2013-MEM-AAM, dated June 11th, 2013, the Mine Closure Plan is approved for the Las Bambas Project.

Assessment	Commentary
	• According to DR N°187-2013-MEM-DGM/V, dated May 02nd, 2013, the Pit Mining Plan and the waste dump is approved for the Las Bambas project. • The Concentrator Plant is currently in the construction phase, which began in June 2012, after obtaining permission to construct, approved by D.R. N°178-2012-MEM-DGM-DTM/PB, on May 31st 2012 and modified through D.R. N° 419-2013-MEM-DGM/V on November 6th 2013. Due to the decision to relocate the filtration and molybdenum plant from Antapaccay area to Las Bambas, was necessary to modify the permit of construction of the Project, which was authorized by D.R. N° 419-2013-MEM/DGM/V.
Classification	• The classification of Ore Reserves is based on the requirements JORC code 2012, based on the classification of Mineral Resources and the Cut-Off grade. The material classified as Measured and Indicated Mineral Resource and is above the BCoG CuT% grade is classified as Proved and Probable Ore Reserve respectively. • The Competent Person considers this appropriate for the Las Bambas Ore Reserve estimate. • No Probable Ore Reserves have been derived from Measured Mineral Resources.
Audit or Reviews	• The 2014 Ore Reserve was reviewed by Runge Pinock Minarco for the MMG Competent Person's Report. • The 2015 Ore Reserves estimates has been reviewed and validated by Carlos Gutierrez, Las Bambas Mine Planning Superintendent; in coordination with Corporate group GTS-MMG. • No external reviews have been conducted for the 2015 Ore Reserve estimate.
Discussion of relative accuracy/ confidence	• The principal factors that can affect the confidence on the Ore Reserves are: o Geotechnical risk related to slope stability. o Metallurgical recovery model uncertainty as they were developed at pilot study level. In the best case scenario, this would represent variability of +/- 2% and in the worst scenario +/- 5% to metal recovery. o Increases in the cost of consumables resulting 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 delays for TSF needs. o Mining dilution and recovery adjustments as limited ore mining to date at Las Bambas is not available to verify these factors. o Ore treatment throughput models are based on testing but remain unverified until processing commences.

4.3.1 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 Berthlesen, Principal Resource Geologist - MMG Ltd (Melbourne) Anna Lewin, Senior Resource Geologist - MMG Ltd (Melbourne) Helber Holguino, Modelling Geologist – MMG Ltd (Las Bambas) Cicino Sernaque, Modelling Geologist – MMG Ltd (Las Bambas)	Mineral Resource model
Geoff Senior, Group Manager Metallurgy - MMG Ltd (Melbourne) Helbert Zinanyuca, Metallurgy Superintendent – MMG Ltd (Las Bambas)	Updated processing parameters and production record
Luis Tejada, Geotechnics & Hydrogeology Superintendent - MMG Ltd (Las Bambas) Gian Ticona, Geotechnics Supervisor – MMG Ltd (Las Bambas)	Geotechnical parameters
Carlos Gutierrez, Mine Planning Superintendent – MMG Ltd (Las Bambas) Mitchel Neyra, Short Term Planning Supervisor – MMG Ltd (Las Bambas) Edgard Mendoza, Short Term Planning Supervisor – MMG Ltd (Las Bambas)	Cut-Off Grade calculations Whittle/MineSight optimisation and pit designs
Luis Ticona, Technical Services Director – MMG Ltd (Las Bambas) Carlos Gutierrez, Mine Planning Superintendent – MMG Ltd (Las Bambas)	Production reconciliation
ATC Williams Pty Ltd Alejandro De Bary, Senior Water Management Specialist, MMG Ltd (Melbourne)	Tailings dam design
Alvaro Ossio, Commercial Vice President – MMG Ltd (Las Bambas)	Economic Assumptions

5 KINSEVERE OPERATION

5.1 Introduction and setting

Kinsevere is located in the Katanga Province, in the southeast of the Democratic Republic of Congo (DRC). It is situated approximately 27 kilometres north of the provincial capital, Lubumbashi (Figure 5-1), at latitude S 11° 21′ 30″ and longitude E 27° 34′ 00″.

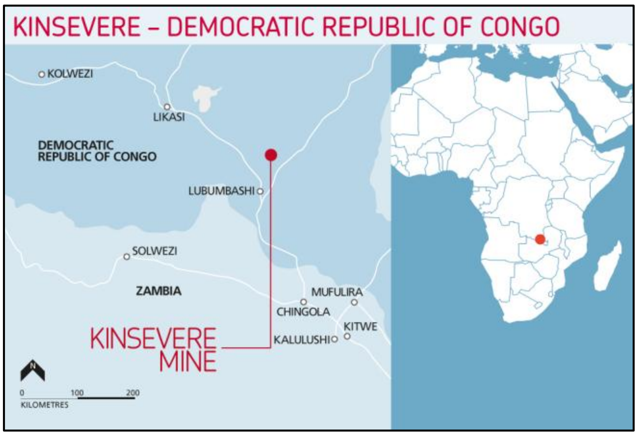


Figure 5-1 Kinsevere Mine location

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% copper. The HMS was decommissioned in June 2011 when the Stage II SXEW plant was commissioned. The Stage II plant was designed to process up to 1.6 Mtpa of ore and produces approximately 65,000 tonnes of copper cathodes. These design rates are now being exceeded. Molybdenum

5.2 Mineral Resources - Kinsevere

5.2.1 Results

The 2015 Kinsevere Mineral Resource are summarised in Table 7. The Kinsevere oxide Mineral Resource is inclusive of the Ore Reserve.

Table 7 2015 Kinsevere Mineral Resource tonnage and grade (as at 30 June 2015)

Kinsevere Mineral Resource					
				Contained Metal	
Oxide Copper[2]	Tonnes (Mt)	Copper (% TCu[1])	Copper (% AsCu[1])	**Copper TCu ('000 t)**	**Copper AsCu ('000 t)**
Measured	3.7	4.5	3.9	164	144
Indicated	11.9	3.4	3.1	403	366
Inferred	4.2	3.3	2.9	139	123
Total	**19.8**	**3.6**	**3.2**	**706**	**633**
Oxide Copper Stockpiles					
Measured	6.4	2.3	1.6	149	104
Total	**6.4**	**2.3**	**1.6**	**149**	**104**
Total Oxide Copper	**26.2**	**3.3**	**2.8**	**854**	**737**
Primary Copper[3]					
Measured	1.6	3.2	0.9	51	15
Indicated	10.9	2.2	0.4	238	45
Inferred	14.6	2.4	0.2	343	28
Total Primary Copper	**27.1**	**2.3**	**0.3**	**632**	**88**
Total Contained Metal				**1,486**	**825**

1 TCu stands for Total Copper, AsCu stands for Acid Soluble Copper.
2 0.6% Acid soluble Cu cut-off grade contained in US$3.50/lb oxide pit shell.
3 0.8% Total Cu cut-off grade contained in US$3.50/lb sulphide pit shell.
Contained metal does not imply recoverable metal.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

5.3 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.

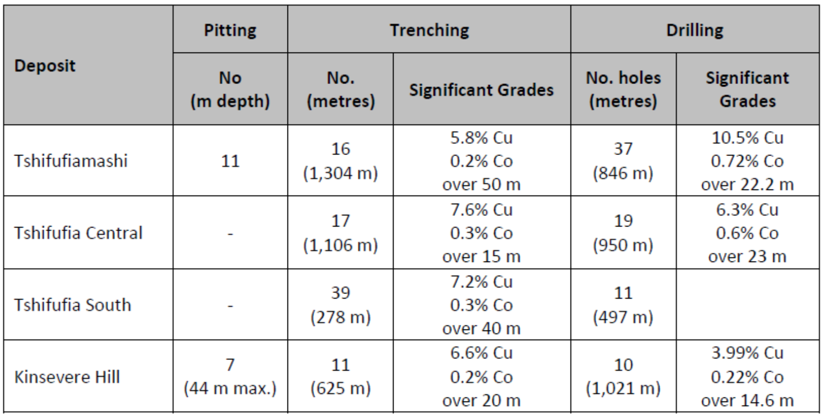
Table 8 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Kinsevere Mineral Resource 2015

Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• The Mineral Resource uses both grade control, reverse circulation (RC) drilling and exploration/resource delineation diamond drilling (DD). GC samples are obtained by RC drilling and composited into 2 m samples. • Resource delineation and exploration drilling samples are obtained by DD. DD core is sampled in 1 m intervals while samples in un-mineralised zones are over 4 m lengths. Sampling is performed by cutting half core, with half retained on site for future reference. • Each sample was crushed and pulverised to produce a pulp (>85% passing 75 µm) prior to analysis at the site SGS laboratory. • 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. • 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 2 m composited RC chip samples. 184,126 m or 77% of the sample data used in the Mineral Resource were from RC samples (5.5" hammer). Of the RC drilling 130,860 m (55%) was from Grade Control drilling. • DD recovered PQ and HQ size DD core was used to obtain nominal 1 m sample lengths. DD core was not routinely oriented. 55,069 m or 23% of the sample data used in the Mineral Resource were from DD samples (PQ and HQ size). • In the view of the Competent Person sampling is of a reasonable quality to estimate the Mineral Resource.
Drill sample recovery	• Recovery recorded during RC drilling is considered high, with minor losses in broken ground. However no measurements of recovery are recorded for RC drilling. • DD core recovery recorded was generally 100%, with minor losses in broken ground. DD uses triple tube core barrels 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. • There is no relationship between core loss and mineralisation or grade - no preferential bias has occurred due to any core loss.

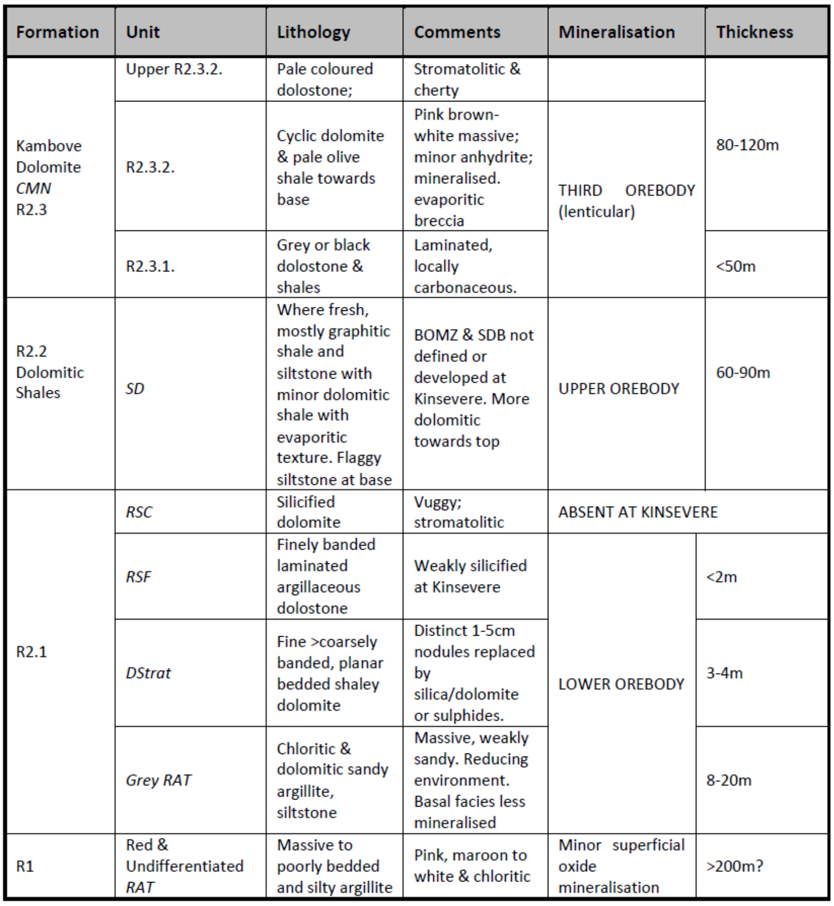
Logging	• RC chips are logged by geologists directly into an Excel logging template with general geological information logged – lithology, stratigraphy, weathering, oxidation, colour, texture, grain size, mineralogy and alteration. This data is then imported into the database. DD core samples have geological and geotechnical information (lithology, stratigraphy, mineralisation, weathering, alteration, geotechnical parameters: strength, RQD, structure measurement, roughness and infill material) recorded. • All RC chip and DD core samples (100%) have been geologically logged to a level that can support appropriate Mineral Resource estimation. • Logging captured 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.
Sub-sampling techniques and sample preparation	• DD core was split in half or quartered using a diamond saw. Sample lengths were cut as close to 1 m as possible while respecting geological contacts. Samples were generally 2 kg to 3 kg 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 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 in the laboratory oven before being split according to the procedure above (for dry samples). • Samples from individual drill holes 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 60 to 80°C, crushed to 85% passing -2 mm using a jaw crusher and milled to 85% passing 75 µm using one of three single sample LM2 vibratory pulverising mills. • The RC and DD sample preparation techniques are considered to be of high-quality and appropriate as sample preparation

	techniques for the style of the material being sampled.
Quality of assay data and laboratory tests	• RC samples are currently assayed at the site SGS Laboratory. It is unknown where the RC samples were previously assayed. ○ Following preparation, 50 g pulp samples were routinely analysed for total and acid soluble copper, cobalt and manganese. ○ A 3 acid digest with AAS finish was used to analyse for total values. ○ 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: ○ ALS Chemex Laboratory, Johannesburg ○ McPhar Laboratory, Philippines ○ ACTLabs Laboratory, Perth ○ Samples were analysed for total copper and acid soluble copper with some having a full suite of elements analysed with a four acid digest and ICP-OES analysis. • From 2011, prepared samples were submitted to the ISO 17025 accredited SGS Laboratory in Johannesburg with the following assay scheme: ○ ICP-OES method with a 4-acid digest analysing 32 elements including copper from 0.5ppm to 1%. ○ ICP-OES method using alkali fusion is applied to over-range copper results. ○ ICP-AES with a 4-acid digest was used for calcium and sulphur analysis ○ XRF was used for uranium analysis. ○ Acid soluble copper using a sulphuric acid digest and AAS 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 in Certified Reference Material (CRM) materials for every batch of 25 samples; blanks, field duplicates, coarse duplicates and pulp duplicates for every batch of 50 samples; and umpire laboratory checks for every batch of 20 samples are analysed in order 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	• Verification by independent or alternative company personnel was

sampling and assaying	not undertaken at the time of drilling. • Twinned pre-collars are present in the database. These were used to confirm and check geological intervals and/or assay intervals. Twin drill holes are not used in the Mineral Resource. • 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 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 drill hole collars were located using a hand held GPS. Accuracy of GPS is +/- 5 m for x and y coordinates and 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 was 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: -8000000 m in northing and -22.3 m in elevation. • A LIDAR survey is used to generate a topographic surface. This surface was also used to better define the elevation of drill hole 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 5 m x 15 m, which is sufficient to adequately define lithology and mineralisation domain contacts and transition zones. • The overall DD pattern spacing is between 25 m to 100 m, which is sufficient to establish the required degree of geological and grade continuity that is appropriate for the Mineral Resource. • DD samples are not composited prior to being sent to the laboratory however the nominal sample length is generally 1 m. RC samples are 2 m intervals but compositing up to 4 m has occurred in the past.
Orientation of data in relation to geological structure	• The mineralisation strikes north-south, and north-west, south east for Kinsevere Hill. All drill holes are oriented such that drill holes 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 drill holes 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.

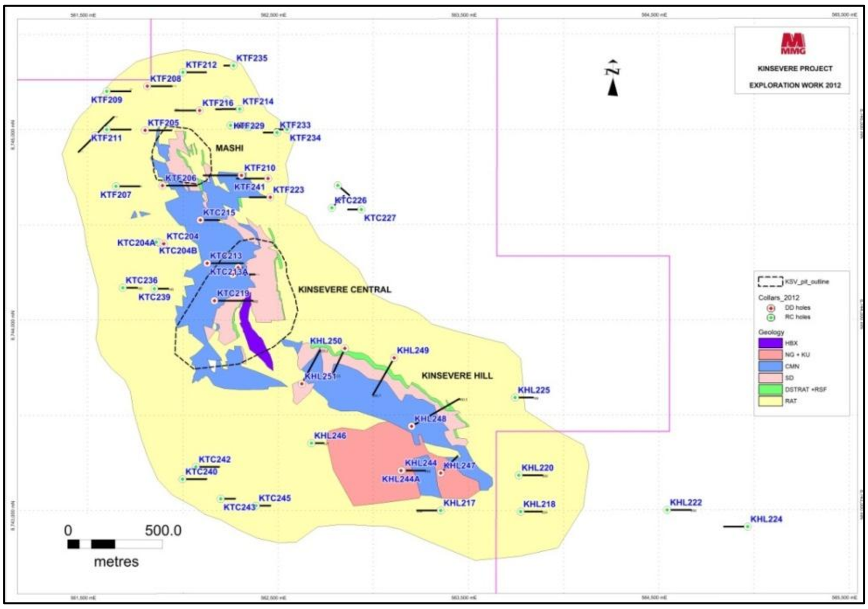
<table>
<tr><td>Sample security</td><td>Measures to provide sample security include:
Adequately trained and supervised sampling personnel.
Sea containers where samples are stored are locked with keys given to security department.
Assay laboratory checks of sample dispatch numbers against submission documents.</td></tr>
<tr><td>Audit and reviews</td><td>An independent audit of the Mineral Resource model was completed in June 2014, by the MSA Group Pty Ltd and was commissioned by MMG Limited (J2851 Kinsevere Mineral Resource Audit June 2014). No material errors were found and recommendations and suggestions have been incorporated into the current Mineral Resource Estimate.
Internal visits by the Competent Person and MMG Group Office geologists to the site laboratory; sample preparation area and drill locations did not identify any material risks.</td></tr>
<tr><td colspan="2">Section 2 Reporting of Exploration Results</td></tr>
<tr><td>Mineral tenement and land tenure status</td><td>The Kinsevere Mining Licence (PE 528) is located approximately 27 km north of Lubumbashi, the provincial capital of the Katanga Province, in the southeast of the Democratic Republic of the Congo (DRC).
Anvil Mining sold the Kinsevere project to MMG in 2012.
A royalty of 2.5% of gross revenue were adopted in January 2009 to reflect revised royalty payments after the Government reviewed all the mining contracts in 2008 where the terms of the Lease Agreement were amended.
There are no known impediments to operating in the area.</td></tr>
<tr><td>Exploration done by other parties</td><td>.
Summary of Previous Exploration Work by Gecamines and EXACO

<table>
<tr><th rowspan="2">Deposit</th><th>Pitting</th><th colspan="2">Trenching</th><th colspan="2">Drilling</th></tr>
<tr><th>No
(m depth)</th><th>No.
(metres)</th><th>Significant Grades</th><th>No. holes
(metres)</th><th>Significant Grades</th></tr>
<tr><td>Tshifufiamashi</td><td>11</td><td>16
(1,304 m)</td><td>5.8% Cu
0.2% Co
over 50 m</td><td>37
(846 m)</td><td>10.5% Cu
0.72% Co
over 22.2 m</td></tr>
<tr><td>Tshifufia Central</td><td>-</td><td>17
(1,106 m)</td><td>7.6% Cu
0.3% Co
over 15 m</td><td>19
(950 m)</td><td>6.3% Cu
0.6% Co
over 23 m</td></tr>
<tr><td>Tshifufia South</td><td>-</td><td>39
(278 m)</td><td>7.2% Cu
0.3% Co
over 40 m</td><td>11
(497 m)</td><td></td></tr>
<tr><td>Kinsevere Hill</td><td>7
(44 m max.)</td><td>11
(625 m)</td><td>6.6% Cu
0.2% Co
over 20 m</td><td>10
(1,021 m)</td><td>3.99% Cu
0.22% Co
over 14.6 m</td></tr>
</table>
In 2004 Anvil Mining carried intense exploration to define the deposits in Kinsevere.
In 2012 MMG conducted exploration to identify additional mineralisation beyond the Anvil Mining Mineral Resource.
In 2013/2014 MMG Exploration have been conducting works around the Mine Lease within a 50 km radius of the known</td></tr>
</table>

	deposit to explore for additional high-grade oxide material. • 2015 MMG conducted a Scoping Study on the potential to process copper sulphide ore at Kinsevere, beneath the current oxide Ore Reserves. In August 2015, drilling is expected to commence to increase confidence in the copper sulphide Mineral Resource.
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 and oxide mineralisation in the Kinsevere copper deposit are either disseminated in recrystallised layers or infilling bedding plans, reactivated bedding, fractures and joints. The sulphides include: pyrite, chalcopyrite, bornite and chalcocite. Although in the supergene zone, sulphides are partially or completely replaced by malachite and other copper oxide minerals. **Kinsevere Mine Series Stratigraphy**

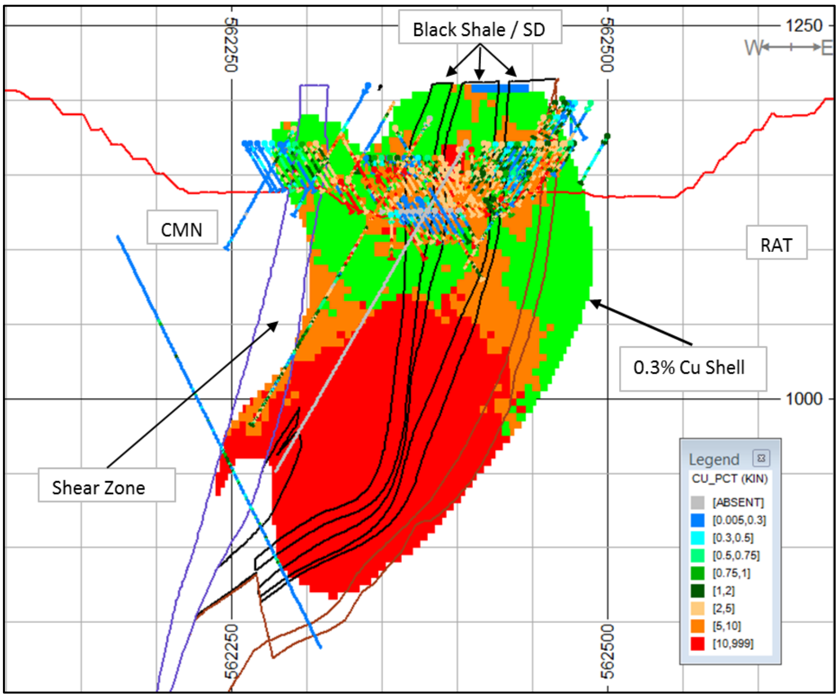


Formation	Unit	Lithology	Comments	Mineralisation	Thickness
Kambove Dolomite *CMN* R2.3	Upper R2.3.2.	Pale coloured dolostone;	Stromatolitic & cherty		80-120m
	R2.3.2.	Cyclic dolomite & pale olive shale towards base	Pink brown-white massive; minor anhydrite; mineralised. evaporitic breccia	THIRD OREBODY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

Drill hole information	• Within the database used, there are 773 Exploration drill holes (195 DD, 31 RC with DD tail and 547 RC) and 4882 grade control drill holes (all RC). • No individual drill hole 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 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.

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


Plan view of the Kinsevere deposit



0.3% Cu Shell coded by Oxidation state (Green – Oxide / Orange – Mixed / Red – Primary)

Typical cross section through Tshifufia (Central) pit showing drilling interceptions (Cu%)
</td>
</tr>
<tr>
<td>Balanced reporting</td>
<td>

- This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

</td>
</tr>
<tr>
<td>Other substantive exploration data</td>
<td>

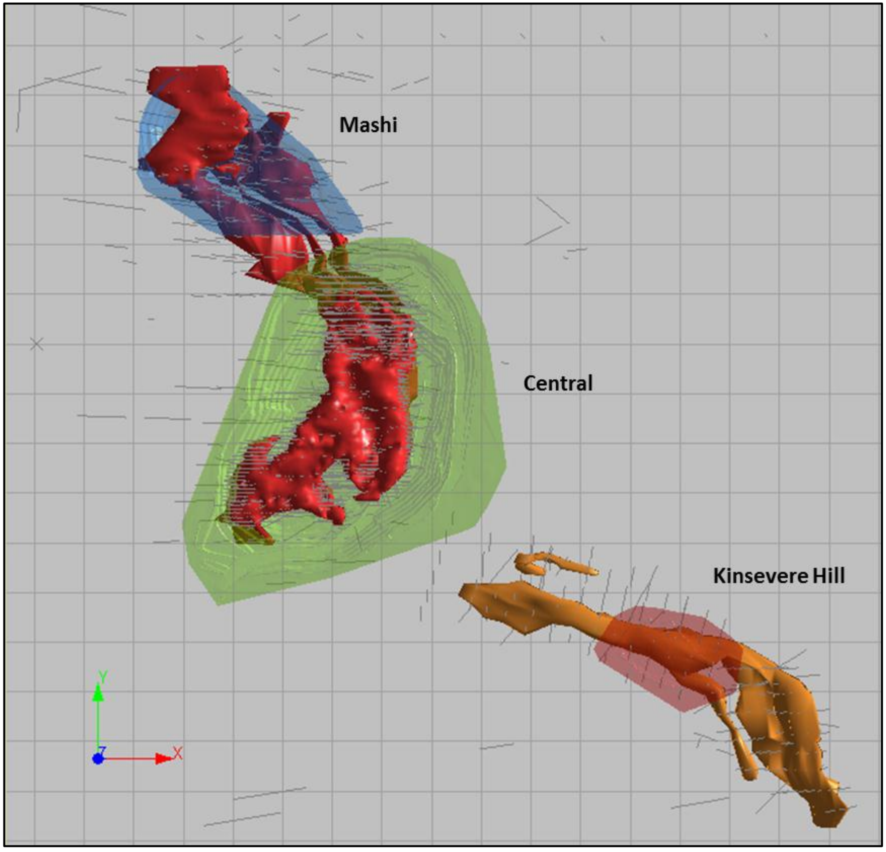
- This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.

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

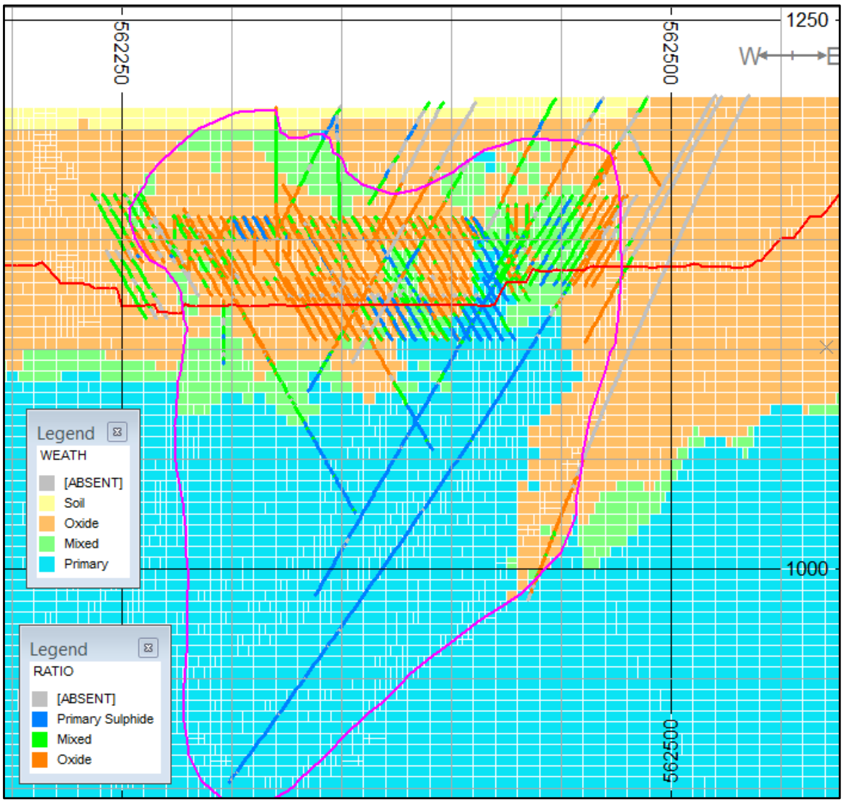
Further work	• The exploration focus will be within the Mine Lease within a 50 km radius of the known deposit to explore for additional high-grade oxide material. • Further drilling is required to increase confidence and upgrade Inferred to Indicated or Measured categories.
Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	• The following measures are in place to ensure database integrity: o The complete drill hole database (RC grade control and DD) data is stored in two SQL databases using the DataShed and GBIS front end management systems: o The grade control data (RC) is stored in DataShed and is managed by the onsite Geology team. o The exploration/resource (DD) data is stored in a GBIS database. Management of this database is performed by the Group Technical Services database team in Melbourne. o 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. o 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: o Internal database validation systems and checks. o Visual checks of exported drill holes in section and plan view, checking for accuracy of collar location against topography, and downhole trace de-surveying. o External checks in Datamine 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 GBIS/Datashed. Data used in the Mineral Resource has passed a number of validation checks both visual and software related prior to use in the Mineral Resource.
Site visits	• The Competent Person visited site twice occasions during 2015 (February and June). Site visit work included: o Visits to the ROM stockpiles, open pit mine, core yard, sample preparation and on-site assay laboratory. o 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 a lower level of uncertainty in the geological interpretation in the lower portions of the Mineral Resource, due to the sparse drilling. • 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 Resource. • No alternative interpretations of the Mineral Resource have been used. However, an Indicator Kriging approach was used to determine weathering domains (within the mineralised zone), and for Mg high grade domains (using a 6% Mg cut-off). • Wireframe solids and surfaces were created for the mineralisation (using a 0.3% Cu cut-off), and high grade Ca (using a 9% Ca cut-off) domains. String envelopes were digitised along drill sections were used to generate the wireframe surfaces. • Geological logging was used to determine the lithological domains : o Dolomite Stratifiee (DStrat), the Roche Siliceuse Feuilletee and Roche Argilo-Talqueuse (RAT). o Shale Dolomitiques (SD) and the incorporated Black Shales (BLKSH). o Calcaire a Mineraux Noirs (CMN) and shear zone. • The weathering domains were further subdivided into a soil (based on wireframe shapes used in the 2014 Mineral Resource), oxide, mixed and primary sulphide zones. Wireframe shapes from the 2014 Mineral Resource were used to code the oxide, mixed and primary sulphide zones, outside of the mineralised zone. • 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, mixed and primary sulphide zones. The following ratios have been used to delineate the respective zones: o Oxide > 0.8 o Transition between 0.15 and 0.8 o Primary < 0.15 • The resulting weathering, lithology, mineralisation domains were combined to code the drill hole data and the block model used for estimation. • On a local scale grade continuity is affected by "clasts" or 'pods' of un-mineralised RAT that have been incorporated within the main deposit. This internal waste is better defined during the grade control drilling, and mined accordingly.

- Structural features provide an important control on the mineralisation and grade continuity.



Plan View of Kinsevere Mineralised Domain

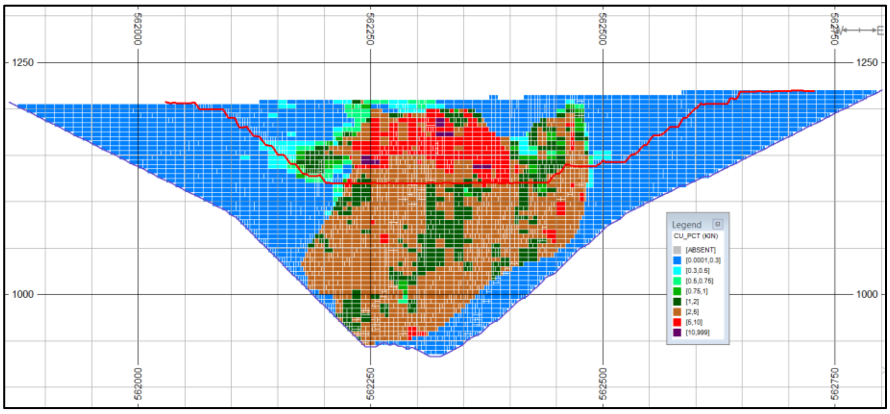


Cross section view looking north showing the oxide, mixed and primary sulphide domains with drill holes

Dimensions	• The mineralisation strike length is approximately 1.3 km for the Tshifufia and Tshifufiamashi deposits while Kinsevere Hill has a 1k m strike length. The mineralisation dips sub-vertically. Mineralisation extends to 400 m at depth and it can be up to 300 m in width. • The mineralisation outcrops on Kinsevere Hill, and at the Tshifufiamashi deposit.
Estimation and modelling techniques	• Mineral Resource modelling and variogram generation was completed in Isatis (Geovariance) software and compiled within Datamine software for reporting, applying the following key assumptions and parameters: ○ Cu, AsCu, AsCu/Cu (RATIO), Co, Ca, Fe, Mg, Mn and S were estimated using Ordinary Kriging (OK). Ordinary Kriging is considered an appropriate technique for estimating the Kinsevere Mineral Resource. 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, based on the AsCu/Cu ratio, and for determining the high grade Mg domain. ○ RATIO was only estimated where the sample had a Cu > 0.1% 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. Variables were domained by WEATH and AREA coding for top cutting. ○ Wireframes and surfaces of the topography; soil; high grade Ca domains; mineralised domains, together with IK domain are used to tag the drill holes 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 variography locally within the mineralised domain. 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 2 m was used within the defined domains. Any residual intervals less than half the composite interval were appended to the previous sample

	interval. ○ Separate variography was performed for Cu, AsCu, RATIO, Co, Ca, Fe, Mg, Mn and S; within the mineralised and waste zones segregated by respective WEATH domain or where high grade domains were established. ○ 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 400 m x 400 m x 400 m and using the AREA codes, U grades higher than 250 ppm were distance limited to 20 m. RATIO within the waste area was calculated post-estimation (AsCu est. / Cu est.). ○ 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 2 times the variogram range. ○ Minimum of 5 to 14 and a maximum of 12 to 30 samples (depending on element and /or domain) required for an estimate. ○ The search neighbourhood was also limited to a maximum of 4 to 7 samples per drill hole depending on the domain to be estimated. ○ Octant searches were used in some of the estimations, based on QKNA studies, and is documented in the report (maximum of 2 sectors was used). ○ Discretisation was set to 4 x 8 x 2 (X,Y,Z). ○ Kriging variance (KV), kriging efficiency (KE) and kriging regression slope (RS) of the Cu estimate were calculated during the estimation. • Reconciliation has been conducted between the Mineral Resource and grade control models. The comparisons are for total copper and acid soluble copper and only for the material constrained within the volume of the grade control model. ○ The Mineral Resource model has less tonnes at higher grades. Tonnes were approximately 6% less and grade was approximately 8% higher than the grade control model. The metal was approximately 4% higher than the grade control model; this is believed due to the fact that the bulk density is 4% higher and needs to be investigated further before the next Mineral Resource model update. ○ The Comparison between the Mineral Resource and the mill feed grade is complicated by the operational strategy

	of treating high-grade ore and stockpiling lower-grade ore for later treatment.
	• 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.
	• Ancillary element: U ppm was estimated by ID2.
	• Parent block size of the Kinsevere block model is 10 m x 20 m x 5 m with sub-blocking down to 2.5 m. Estimation was into the parent block. The size of the blocks is appropriate to the spacing of drill holes.
	• No further assumptions have been made regarding modelling of selective mining units.
	• The block model and estimate has been validated in the following ways:
	  ○ Visual checks in section and plan view against the drill holes.
	  ○ Grade trend plots comparing the model against the drill holes.
	  ○ Reconciliation with grade control model.

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 grade of 0.6%. This has changed from the 2014 Mineral Resource which was 0.75% AsCu. • The primary sulphide Mineral Resource has been reported above a total copper cut-off grade of 0.8%. There is no Ore Reserve for the primary copper. This cut-off has changed from the 2013 Mineral Resource which was 0.75% Cu. • The reported oxide and primary copper Mineral Resources have also been constrained within a US$3.50/lb pit shell. 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.  **Copper Mineral Resource Contained in the US$3.50lb pit shell (cross**

	section)
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. - No mining factors have been applied to the Mineral Resource.
Metallurgical factors or assumptions	- The metallurgical process applied at the Kinsevere Operation is acid leaching coupled with solvent extraction electro-winning (SXEW) technology to produce copper cathode plates for sale. - No metallurgical factors have been applied to the Mineral Resource. - Currently primary mineralisation is not processed at Kinsevere, an ongoing PFS is being carried out to consider the treatment of primary and mixed mineralisation.
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. - Acid rock drainage (ARD) properties of the waste rock and black shales are unknown. Further work is planned to understand the properties of the rock and the required stockpile management policies. - The property is not subject to any environmental liabilities.
Bulk density	- In-situ dry bulk density values are determined from 1,696 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 measurements on drill core did not use a sealed method and may be subject to pore-space variations. - 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. - Bulk density values for the primary sulphide domains have been revised and set to 2.7 g/cm^3 for the 2014 estimate. The increase needs to better understood, and the bulk density measurements needs to be spatially compared with the new weathering domains. Changes to the current bulk density measurements used are not considered to be material, but will improve local estimation.
Classification	- Wireframes used for Mineral Resource classification are based on a combination of confidence in assayed grade, geological continuity and kriging metrics (kriging variance, efficiency and slope of regression).

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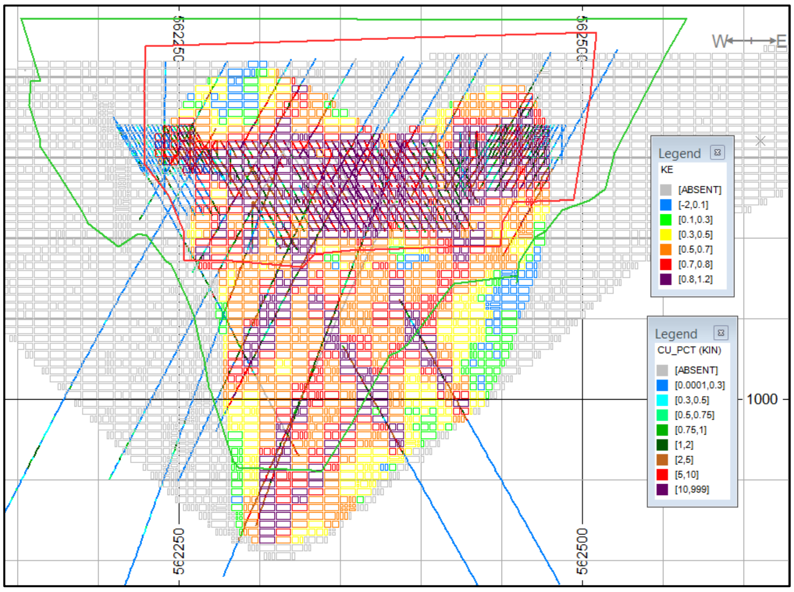
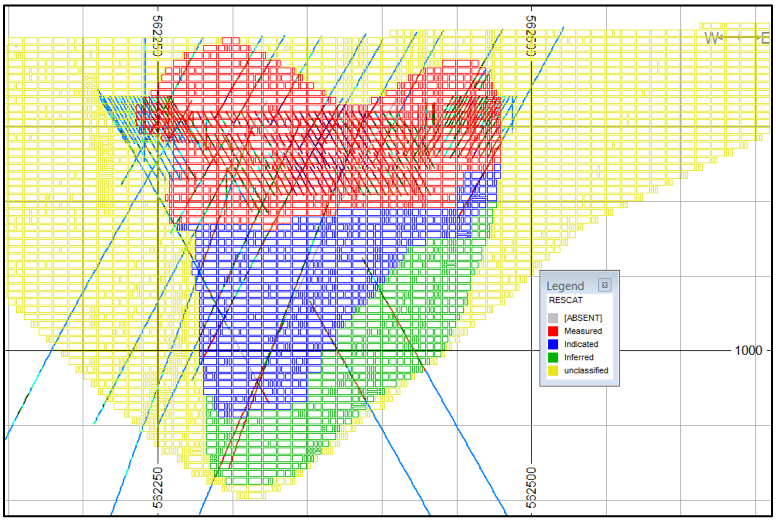
- The Mineral Resource classification reflects the Competent Persons view on the confidence and uncertainty of the Kinsevere Mineral Resource.



Cross section showing Kinsevere Model coloured by kriging efficiency, drilling density and Measured, Indicated selection polygons.



Cross section showing Kinsevere Mineral Resource classification and drilling density

</td>
</tr>
<tr>
<td>Audits or reviews</td>
<td>

- MMG has an internal ‘Mineral Resource and Ore Reserve Policy’ that requires at a minimum external reviews every three years, and internal company review every interim year by MMG representatives reporting findings to a sub-set of the MMG Mineral Resource and Ore Reserve Committee. 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.

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

	- An external Mineral Resource 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 Resource estimated.
Discussion of relative accuracy / confidence	- Close-spaced grade control drilling within the Measured Mineral Resource 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. This level of uncertainty is captured by the Indicated Inferred Mineral Resource category. - In-situ dry bulk density values used needs to be re-examined, and spatially linked to the new weathering domain (IK generated). The bulk density values used were from the 2014 Mineral resource and are not considered to have any material impact on the reported tonnages.

5.4 Ore Reserves - Kinsevere

5.4.1 Results

The 2015 Kinsevere Ore Reserve are summarised in Table 9.

Table 9 2015 Kinsevere Ore Reserve tonnage and grade (as at 30 June 2015)

Kinsevere Ore Reserve					
				Contained Metal	
	Tonnes (Mt)	Copper (% TCu[1])	Copper (% AsCu[1])	**TCu ('000 t)**	**AsCu ('000 t)**
Oxide Copper					
Proved	2.9	4.7	4.2	135	120
Probable	6.6	3.9	3.6	255	235
Total	**9.4**	**4.1**	**3.8**	**390**	**355**
Oxide Copper Stockpiles					
Proved	1.4	3.7	3.1	53	44
Probable	3.4	1.4	1.1	49	37
Total	**4.8**	**2.1**	**1.7**	**102**	**81**
Total Contained Metal				**492**	**436**

1 TCu stands for Total Copper, AsCu stands for Acid Soluble Copper.
Cut-off grade is 1.2% AsCu under current operating conditions and 0.8% AsCu at the cessation of mining activities.

Contained metal does not imply recoverable metal.

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

The major differences from the 2014 Ore Reserves are:

(i) A new Mineral Resource model that has resulted in a slight increase in both tonnes and grade.

5.5 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.

Table 10 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Kinsevere Ore Reserve 2015

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td>Mineral Resource estimate for conversion to Ore Reserves</td><td>

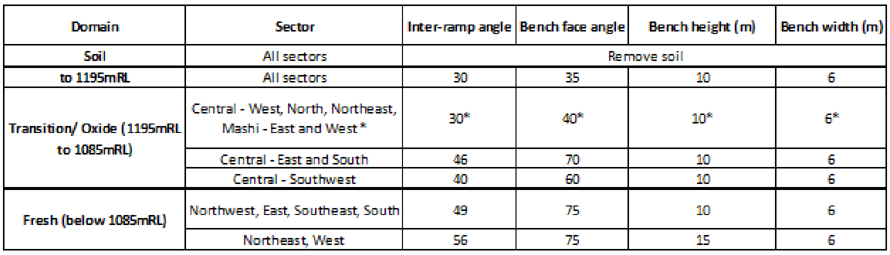
The Mineral Resources are reported inclusive of the Mineral Resources used to define the Ore Reserves.
The normal sub-celled Mineral Resource block model named “ksjun15m.dm” dated 05-06-2015 was used for the optimisation purposes.
</td></tr>
<tr><td>Site visits</td><td>

The Competent Person is Dean Basile AusIMM(CP) who visited the site in late July 2015.
Richard Butcher (previous Competent Person) undertook visits to site.
</td></tr>
<tr><td>Study status</td><td>

The current mine and processing plant configuration have been in operation since September 2011. Ore Reserves inputs are based on actual historical performance data. Although full production ramp-up has been achieved for a number of months, further work is underway to ensure consistency of results and full realisation of the installed capacity.
</td></tr>
<tr><td>Cut-off parameters</td><td>

Estimated breakeven cut-off grade calculated as per historical practices is 1.2% Acid Soluble Copper (AsCu) at a US$2.95/lb copper price under current operating conditions and 0.8% under post mining conditions.
For the price assumptions please see section “Costs” below.
</td></tr>
<tr><td>Mining factors or assumptions</td><td>

Kinsevere mine is an open pit operation that is mining and processing oxide 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.
Pit slope geotechnical parameters:


<table>
<tr><th>Domain</th><th>Sector</th><th>Inter-ramp angle</th><th>Bench face angle</th><th>Bench height (m)</th><th>Bench width (m)</th></tr>
<tr><td>Soil</td><td>All sectors</td><td colspan="4">Remove soil</td></tr>
<tr><td>to 1195mRL</td><td>All sectors</td><td>30</td><td>35</td><td>10</td><td>6</td></tr>
<tr><td rowspan="3">Transition/ Oxide (1195mRL to 1085mRL)</td><td>Central - West, North, Northeast, Mashi - East and West *</td><td>30*</td><td>40*</td><td>10*</td><td>6*</td></tr>
<tr><td>Central - East and South</td><td>46</td><td>70</td><td>10</td><td>6</td></tr>
<tr><td>Central - Southwest</td><td>40</td><td>60</td><td>10</td><td>6</td></tr>
<tr><td rowspan="2">Fresh (below 1085mRL)</td><td>Northwest, East, Southeast, South</td><td>49</td><td>75</td><td>10</td><td>6</td></tr>
<tr><td>Northeast, West</td><td>56</td><td>75</td><td>15</td><td>6</td></tr>
</table>
<p>* these sectors differ from pre-2014 work (inter-ramp angle previously = 40-46° versus 30° currently)</p>

Assumed mining dilution 5%. Preliminary dilution modelling suggests that dilution maybe slightly higher in Mashi and Central pits. Further work is planned to be carried out in this area.
The assumed mining recovery 95%. Preliminary dilution modelling
</td></tr>
</table>

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>suggests that mining recovery maybe slightly lower in all pits. Further work is planned to be carried out in this area.
• Minimum mining width (bench size) is 50 m.
• No Inferred 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 that is considered to be within the capacity of the contract mining fleet.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>• The Kinsevere mine is an operating entity. The metallurgical process is a hydrometallurgical process involving grinding, tank leaching, counter-current decantation (CCD) washing, solvent extraction and electrowinning. The process has been operating successfully since start up in September 2011.
• Copper recovery is determined by the equation:
$\text{Cu recovery (\%)} = 0.96*\text{AsCu}/\text{Cu}$
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.
• The reconciliation between expected and actual recovery is checked each month. The following table summarises the outcomes for the last 4 half yearly periods. In all periods, actual recovery has exceeded predicted recovery.
<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>H1 2013</td><td>96.0</td><td>96.6</td></tr>
<tr><td>H2 2013</td><td>96.0</td><td>99.0</td></tr>
<tr><td>H1 2014</td><td>96.0</td><td>99.0</td></tr>
<tr><td>H2 2014</td><td>96.0</td><td>97.8</td></tr>
</table>
• The main deleterious components of the ore are carbonaceous (black) shale which increases 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 acid consumption is estimated in the optimisation for oxide ore using the formula:
$AC = 27\times Ca\%+10\times Mn\%+5$
• The inclusion of a term for Mn in the gangue acid equations</td></tr>
</table>

Assessment	Commentary
	reflects a small, but significant consumption of acid by manganese wad. • For Ore Reserves, a processing rate of 1.8 Mtpa of ore and 72 ktpa of copper cathode has been assumed. Both production rates have been demonstrated as sustainable over the last 6 months. • Kinsevere does not produce any by-products.
Environmental	• The property is not subject to any environmental liabilities, apart from the standard rehabilitation requirements associated with Closure. • Following submission of the EIA (Consultants, July 2007), DRC Government approval of the Kinsevere Copper Project, Stages I and II, was issued by CAMI on 15th October 2007. Approval of a variance to the design and operation of the Stage II tailings storage facility was issued by the DPEM on 28th October 2008 (DPEM, Oct 2008). • An Environmental and Social Impact Assessment (ESIA) was prepared by KP (October, 2009) as a condition of the then proposed Project debt financing. In relation to increased groundwater discharge arising from an expansion to the mine dewatering capacity, a Revision No. 1 to the 2007 EIA was prepared by Knight Piésold (Piésold, Dec 2010) and was submitted to the DPEM in December 2010, and finally approved in March 2011. The next update to the EIA is due in 2017, however an minor amendment is currently being prepared for some operational changes associated with the mine plan and new process water infrastructure. • ARD properties of the waste rock and black shales have been characterised through preliminary studies, with more detailed studies ongoing. However it is obvious on site that sulphides in the black shale stockpiles is oxidising and generates heat. A mineral waste management plan has been developed for the site, taking into account these results and documenting design controls. A Conceptual Closure Plan has also been developed, with a supporting cost estimate. • To comply with the DRC Mining Regulations, it is necessary to manage surface water runoff in such a way that contaminated runoff is contained and sediment loadings (from disturbed catchments) are maintained at acceptable levels. In order to achieve this, a number of strategically placed Sediment Control Ponds (SCPs) and diversion channels will need to be implemented and completed.
Infrastructure	• The Kinsevere mine site is well established with the infrastructure in place: ○ The plant is operational ○ There is an existing accommodation camp ○ There is sufficient water for the processing, the ground

Assessment	Commentary
	water is discharged into the river ○ The transportation of the copper cathode is done using trucks ○ Site has an access road that is partially sealed ○ There is a power supply from the national grid and from onsite generators ○ Labour is mostly sourced from the Lubumbashi and surrounding villages with some expats on site ○ There is no need for additional land for any expansions • The power situation rates this aspect as a high risk, with current mitigation by expensive on site diesel based power generation. The future grid power availability is assumed to be restored and thus the costs have been assumed to be lower. • Timely dewatering of the mining areas continues to be important since this might affect pit slope stability.
Costs	• Kinsevere is an operating mine and has historical costs that have been used in Ore Reserves estimation. • The future costs will have to be lowered in order to process low grade stockpiled material economically. • Transportation charges used in evaluation are based on the actual invoice costs that MMG are charged by the commodity trading company per the agreement. • The processing costs include variable gangue acid consumption (GAC) that is dependent on calcium grade. • US dollars have been used thus no exchange rates have been applied. • To date there was minimum in-pit blasting. There is a risk that in the future the amount of blasted material might increase which would increase mining costs. • Since the final product is LME grade A copper cathode there are no applicable treatment or refining or any other similar charges • Sustaining capital costs have been included in the pit optimisation. • Allowances have been made for the royalties. • The impact of any future mining tax is unknown.
Revenue factors	• For cost assumptions see section above – "Costs" • The assumed copper price is US$2.95/lbs. The copper price assumption is based on the MMG corporate finance assumption.
Market assessment	• There is an off-take agreement with the trading company in place for all of the copper cathodes produced on site from oxide ore. As a result the product from this operation is not subject to market supply and demand conditions. • No customer and competitor analysis has been completed due to

Assessment	Commentary
	the current offtake agreement. • Price forecast is based on MMG corporate finance assumption while the product demand is irrelevant because of the off-take agreement between MMG and commodity trader.
Economic	• The costs are based on historical actuals. Revenues are based on historical and contracted realised costs and a realistic long-term metal price. • The life of mine (LOM) financial model demonstrates the mine has a substantially positive NPV. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit. • The value associated with the treatment of low grade stockpiles at the end of the project life is negative without substantial fixed costs reduction; however these reductions are not considered to be unrealistic. • No sensitivity analyses were undertaken for the Ore Reserves work.
Social	• Kinsevere site provides significant support to community with farming and other social projects financed by the site. It has strong support from the local community. • Lubumbashi is a regional capital of the Katanga region. It has a population of approximately 1.6 M people (2012). Lubumbashi has a university that has some mining, geology and processing programs that prepare professionals. • Personnel are recruited from the local villages. The majority of these people are unskilled and require training. Skilled artisans and professional people are recruited from Lubumbashi. • Several hundred artisanal miners were previously active at Kinsevere before the Project commenced. There is little evidence that artisanal miners are active in the area.
Other	• MMG has a Contrat d'Amodiation (Lease Agreement) with Gécamines to mine and process ore from the Kinsevere Project until 2024, followed by an automatic fifteen year extension. • 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
Classification	• The Ore Reserves classification is based on the JORC 2012 requirements. The basis for the classification was the Mineral Resource classification and cut-off grade. Due to the uncertainty of the future site costs all stockpiled material (Measured and Indicated) that is above 0.8% below 1.2% AsCu was classified as Probable. The material that is classified as Measured and Indicated and has a grade higher than 1.2% AsCu was classified as Proved and Probable respectively. • The reason for the downgrade of Measured Mineral Resource stockpile to Probable Ore Reserves classification is due to higher

Assessment	Commentary
	uncertainty in the processing economics of lower grade stockpiles in the future.
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 continuous to be valid considering no significant changes in Ore Reserve estimation methodology has occurred between the 2014 to 2015 Ore Reserve Estimate.
Discussion of relative accuracy/ confidence	• The most significant factors affecting confidence in the Ore Reserves are: ○ Reliability of the grid power supply. Although the project NPV might be impacted by ongoing issues of reliable power supply and the costs of that supply, it is not expected much impact on the Ore Reserves; The Risk is Low. ○ There is a moderate risk of removing up to 3 Mt of low grade stockpile Ore Reserves due to not reducing end of life treatment costs. ○ Further work is required to investigate the possible impact of the change in percentage of the processable black shales; the risk is High, impact is unknown. ○ Estimates of gangue acid consumption that rely on calcium grade estimation; Risk is low, impact is Low. ○ Ore dilution needs further investigation; Potential risk is High; due to the high grade of the deposit possible impact is Low.

5.5.1 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 Resource model
Ryan Whyte, Processing Manager, Kinsevere	Updated processing parameters and production record
Christian Holland, Geotechnical Engineering Specialist, MMG Ltd (Melbourne)	Geotechnical parameters
Dean Basile, Principal Mining Engineer, Mining One Consultants (Melbourne)	Whittle optimisation and pit designs
Kinsevere Geology department	Production reconciliation
Knight Piésold	Tailings dam design
Jarod Esam, Senior Analyst, Business Evaluation MMG Ltd (Melbourne)	Economic Assumptions

6 SEPON – COPPER AND GOLD OPERATIONS

6.1 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 6-1).

Figure 6-1 **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. The Sepon Operation is located east of Savannakhet via National Route 9 then northward from Ban Nabo along National Route 28A.

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

Technical investigations and studies carried out in 2014-2015 have resulted in significant increases in cut-off grades and corresponding reductions in Mineral Resources for both copper and gold through the removal of lower grade materials. These changes have not impacted the Ore Reserve.

6.2 Mineral Resources - Sepon

6.2.1 Results

The 2015 Sepon Mineral Resource are summarised in Table 12. The Sepon Mineral Resource is inclusive of the Ore Reserve.

Table 12 2015 Sepon Mineral Resource tonnage and grade (as at 30 June 2015)

Sepon Mineral Resources

				Contained Metal	
Supergene Copper[1]	Tonnes (Mt)	Copper (%)	Gold (g/t)	**Copper ('000 t)**	**Gold (Moz)**
Measured					
Indicated	13.4	3.3		439	
Inferred	1.0	2.5		25	
Total	**14.4**	**3.2**		**465**	
Copper Stockpiles					
Measured	5.9	2.1		122	
Total	**5.9**	**2.1**		**122**	
Primary Copper[1]					
Measured					
Indicated	7.6	1.0		75	
Inferred	3.8	1.5		56	
Total	**11.4**	**1.1**		**130**	
Oxide Gold[2]					
Measured					
Indicated	1.1		3.0		0.1
Inferred	0.2		2.1		0.0
Total	**1.2**		**2.9**		**0.1**
Partial Oxide Gold[2]					
Measured					
Indicated	0.6		5.4		0.1
Inferred	0.0		4.1		0.0
Total	**0.6**		**5.4**		**0.1**
Primary Gold[2]					
Measured					
Indicated	7.5		3.4		0.8
Inferred	0.3		2.5		0.0
Total	**7.8**		**3.4**		**0.9**
Total Contained Metal				**717**	**1.1**

1 0.5%-1.2% Cu cut-off grade contained in US$3.5/lb pit shells based on, material type and Ore Reserve economics but with factored price assumptions and assumptions for future processing routes.
2 1.2g/t-4.5g/t Au cut-off grade contained in US$1,212/oz pit shells. Various cut-offs based on material type and proximity to the Processing Plant.
Contained metal does not imply recoverable metal.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

All Mineral Resources quoted in this report were estimated from 3 dimensional block models created with Vulcan software. Mineral Resources are modelled using solid wireframes of geological boundaries and/or a nominal 0.1% Cu or 0.3 to 0.5 g/t Au grade boundaries which approximate the natural breaks between copper and gold mineralisation and background grades.

Grade estimation was done using an Ordinary Kriging algorithm. Variogram and estimation parameters were defined using Vulcan and Snowden Supervisor Software. Estimates were modelled on geological domains and density assigned or interpolated where sufficient data exists based on Archimedes method bulk density tests.

The Mineral Resource was estimated and compiled for all Sepon deposits. Some estimates remain unchanged from those reported in June 2014 while others were subjected to minor changes due to pit shells generated using new economic parameters, some Mineral Resource estimates have been removed due to failure to meet the reasonable prospects test whilst significant changes have occurred at the Khanong and Thengkham South and Thengkham North copper deposits where the new modelling methods have been adapted and new drilling incorporated in the estimates up to 30th June 2015. A new copper Mineral Resource has been included for the first time in this estimation to include mineralisation in the Songkham West region.

6.3 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 Resource 2015

Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• Grade control sample data is not used in the Mineral Resource. • Reverse circulation (RC) drilling was sampled at 1 m (3 kg-5 kg) intervals for sub-sampling and analysis. Diamond drilling (DD) was sampled at nominal 1 m lengths modified (+/-0.5 m) to geological boundaries as appropriate. • Samples are crushed and pulverised to produce a pulp (>85% passing 85 µm). • 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 to estimate the Mineral Resource.
Drilling techniques	• RC and DD was used in the Sepon Mineral Resource estimate. • Since 2006 all RC holes were stopped and converted to DD when dry sample could not be maintained. The exception to this was in RC pre-collars drilled as a twin to a pre-existing hole that could not be re-entered. • All DD drilling used triple tube core barrels, largely HQ3 but with PQ3 common in the clayey near surface zones. • All DD core was orientated in fresh rocks where good orientation marks allow.
Drill sample recovery	• Sample recoveries tend to be better in DD (90%) than RC (80% - calculated) with minor differences between mineralised zones and waste. • DD sample recoveries were recorded as the length of core recovered per 1 m of drilling and stored in the Sepon database. RC sample recoveries are recorded as sample weight in the database and a recovery calculated based on expected weight given a particular density. • 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 in core and RC samples is better and generally of lower grade in 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 of the competent person.
Logging	• All RC and DD core was logged on paper log sheets and entered manually into the Sepon database. 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. Core is photographed and stored digitally. - All drill cores are stored at the Sepon core shed. - A total of 1,263501.13 m of drilled data is contained in the database, of this 99% is geologically logged, and 98% of drilled data contains gold and copper assays.

Sub-sampling techniques and sample preparation	- DD core was orientated along the apical trace of the reference plane (usually offset 1 cm from structural orientation mark when available), and then half-core samples were taken using a diamond core saw for competent core or sampling by hand using a spatula or blade for clay-rich or rubbly material.
	- RC samples were collected from a cyclone and, if dry, put through a three stage riffle splitter for a 12.5% sub-sample. A 3 kg-5 kg (1 m) 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.
	- The RC and DD sample preparation techniques are considered to be of high-quality and appropriate as sample preparation techniques.
	- All samples were bagged with a waterproof sample number into numbered calico bags and weighed. The samples were stacked and wrapped on a pallet before being transported by truck to the laboratory.
	- Sample processing and gold fire assaying takes place at ALS laboratory Vientiane. In the past a small proportion of the data used has been assayed at the site assay laboratory using the same methods.
	- 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 2 mm. The sample was rotary split to 3 kg if required, then pulverised using an LM5 to 85% passing 85 µm.
	- 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 15 m for RC drilling. DD field duplicate samples were taken as quarter core every 15 m (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

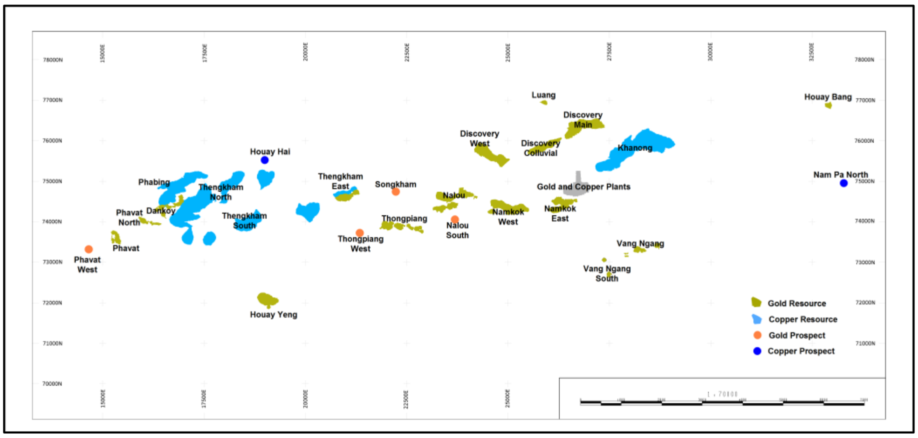
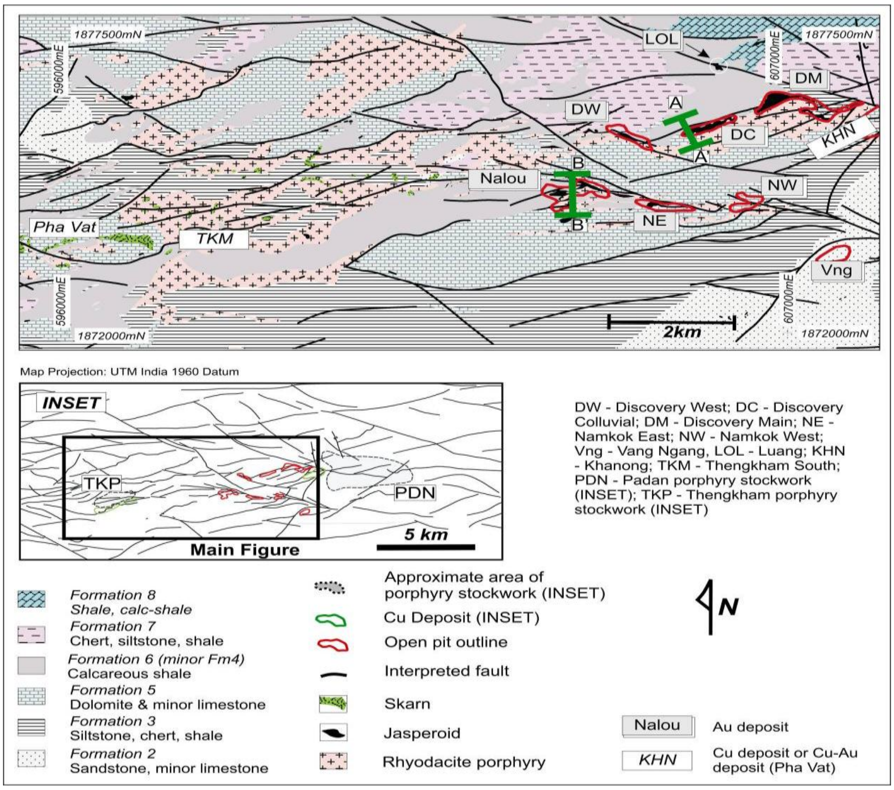
	considered appropriate for the style of the Sepon mineralisation (sediment hosted base metal) by the Competent Person.
Quality of assay data and laboratory tests	• Following sample preparation a 110 g pulp aliquot for Au Fire Assay and 20 g pulp aliquot for ICP multi element was taken. The 20 g pulp 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. • In 2014 a subset of copper samples above 0.2% Cu were sent for sequential copper analysis. Going forward this method will be used on all copper samples to aid in speciation determination. • If Cu > 0.5% Cu, the sample was re-assayed using an Ore Grade technique (either AAS or diluted ICP). These methods are considered total methods. The use of partial copper analysis would assist in copper species identification and is being investigated. • For some samples total sulphur, sulphide sulphur, sulphate sulphur, total carbon, carbonate carbon and organic carbon were analysed by Leco Furnace following appropriate digestion. • 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 drill hole: o grade and matrix matched certified standard material (CRM) o coarse and pulp blanks o field duplicates o pulp repeats. • At a minimum, every drill hole contains at least one coarse blank, pulp blank and 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 is currently being re-instated.
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 drill hole logging and photos on an ad-hoc basis.

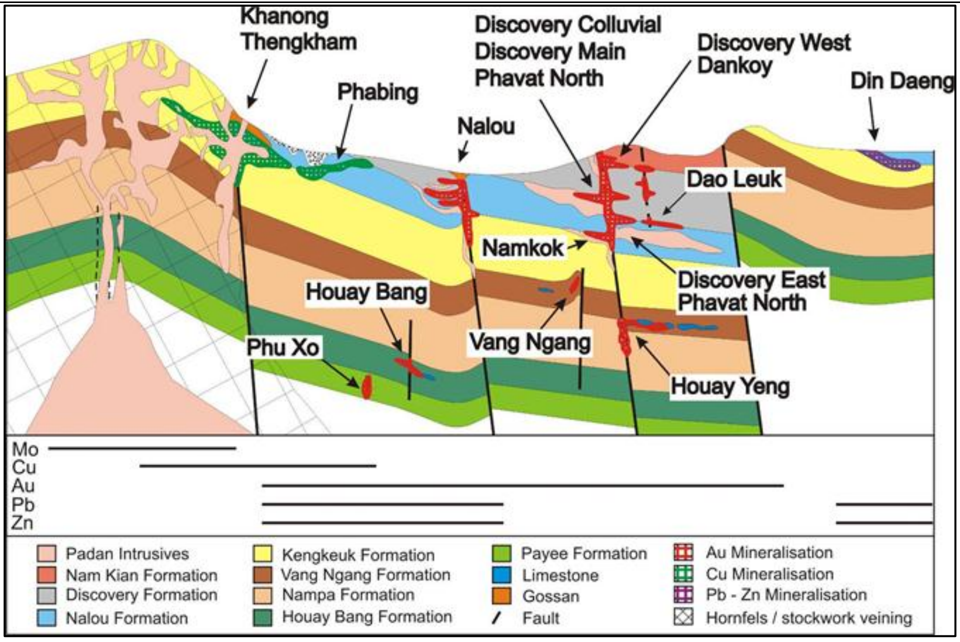
	• Monthly internal reviews are carried out for all assay batches returned. Any samples that exceed 3SD will have its batch returned for re-analysis. Within the reporting period, a total of 17,965 samples were sent to ALS Vientiane and the Sepon onsite laboratory (sterilisation and limestone drill holes only) however no batch were rejected. The frequency of technical operator mistakes caused by laboratory personnel or resource geology geotechnicians varies through the period. Sample swaps mislabelling and incorrect control sample insertion cases have been identified. There are no obvious deficiencies in the assay data quality from ALS and Sepon laboratories that could possibly affect the resource classification, The repeats show a much higher variation than would be expected, however the limitations on the sampling make representative samples difficult to achieve. • For Mineral Resources containing large proportions of RC drilling, twinned drill holes are periodically drilled as part of drill quality analysis. The general conclusion is that wet and moist samples (RC) have demonstrated smearing and a positive grade bias. Areas where wet samples influence the estimate are given a lower level of confidence when Mineral Resources are classified. Current practice is to use DD when wet conditions are experienced. Drill holes suspected of smearing are removed from the dataset prior to 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. • Where data was deemed invalid or unverifiable it was excluded from the Mineral Resource estimation.
Location of data points	• Drill hole 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 12 m, 30 m and then every 30 m to the bottom of hole. • All drill hole 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 accurate topographic surface. Drill hole 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.
Data spacing and distribution	• Drill hole spacing generally ranges from 100 m to 25 m. • 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 the Mineral Resource estimation and classification methods used at Sepon. • DD samples are not composited prior to being sent to the laboratory. RC samples are 1 m intervals but compositing up to 4 m has occurred in the past.

Orientation of data in relation to geological structure	• Geological mapping and interpretation show that mineralisation generally strikes about 070^0 - 090^0 (deposit dependent); hence drilling is conducted on north -south directions so as to intersect the mineralised zone at a high angle. Most drill holes are drilled with dips of -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 TKM and TKN model areas, drill holes were drilled at -60^0 along 090^0 or 270^0 from 50 m 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.
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. ○ Assay data returned separately in both spreadsheet and PDF formats.
Audit and reviews	• REFLEX Geochemistry completed a QC review on data from 1 January 2011 – 31 May, 2014. The conclusions reached 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. No material issues have been identified at the laboratory. • 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 drill hole 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.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure	• 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

status	extraction of any Mineral Resources discovered. • The Sepon Mineral Exploration and Production Agreement (Sepon MEPA) is a direct agreement with the Laos Government. 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 5212 km^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 1247 km^2. • A royalty is payable to the Government of Laos, representing 2.5 % of the FOB value of minerals received by LXML, less all selling, transport, smelting, refining and other attributable cost to the minerals sold. • The operating period which follows is for 20 years, to begin at the commencement of a mining operation. • 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 ten (10) years. • There are no known impediments to operating in the area.
Exploration done by other parties	• CRA Exploration (CRAE, later RTZ) first identified the Sepon Mineral District as an area of interest in 1990 and formed Lane Xang Minerals Limited (LXML) as holder of the MEPA. • Between 1995 and 1999 RTZ (RTZ was formed from the merger of CRA and Rio Tinto in 1997) discovered and defined several gold only Mineral Resources and copper and gold Mineral Resources at the Khanong prospect. • Oxiana became manager of the Sepon Project in 2000 by buying 80% of LXML before later buying the remaining 20% interest from RTZ. The Laos Government exercised its option to acquire a 10% interest in LXML in 2006. • In 2008 Oxiana merged with Zinifex Ltd to form OZ Minerals. • In 2009 MMG acquired LXML from OZ Minerals. • The Sepon exploration and resource geology groups remained unchanged throughout the OZ Minerals and MMG takeovers; hence the exploration management and methods have effectively remained constant since Oxiana acquired the project in 2000.
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 eluvial gold in karst fill deposits. • All primary deposits are hydrothermal and, at least spatially, related to the RDP intrusive. 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 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.
Drill hole information	• No individual drill hole 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° angles in order to maximise true width 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	 **Location of Sepon gold and copper deposits**  **Summary geological map of the Sepon Mineral District (Smith, S.G. et.al, 2005)**

	 **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 2015/2016 drill season's primary focus is to: o Increase 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 The Sepon geological database system consists of three components. A manual Field Logging System, a Data Entry Database (DEDB), and a Master Database (LaoDB). Each digital component is configured to run in SQL Server with user access and permissions. The DEDB works as a quarantine and compilation system. o The supervising database geologist reviews all new data against original paper logs, with corrections made prior to loading into LaoDB which is done via SQL Server stored procedures to detect and hold any errors on import. • The GBIS database and logging system was introduced in 2006 and populated from the pre-existing aQuire© database. Ongoing analytical data is uploaded directly from laboratory SIF files. • The measures described above ensure that transcription or data entry errors were minimised.

	- Data validation procedures include: o Validation routines by database personnel check for overlapping sample, 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 both visual and software related prior to use in the Mineral Resource.
Site visits	- The Competent Person has undertaken numerous (more than 10) visits to Sepon since 2013 in the course of providing Mineral Resource 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.
Geological interpretation	- Prior to Mineral Resource 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 is interpreted using logged drill hole data, core photos and assay data. The base of supergene is important for the carbonate oxide copper mineralisation as high grade copper is known to collect within depressions. - A boundary cut-off for most copper deposits is 0.1% Cu, with a 1% Cu domain used in the Khanong copper carbonate material. 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 identifying population breaks in the sample data geostatistically. As well, visual investigations ensuring that these cut-offs displayed reasonable continuity in three-dimensional space taking into account the local geology. - 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. - 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 Resource classification with supergene copper mineralisation in the Thengkham area generally requiring closer spaced drilling to achieve the same Mineral Resource category compared to the Khanong deposit. This largely reflects reduced ore mineralogy continuity rather than ore grade continuity.

Dimensions	• Sepon hosts a number deposits, the dimension of each respective deposit included in this Mineral Resource is listed below. Various block models cover the extents, where block model extents may overlap, to prevent the double reporting of Mineral Resources wireframes are created to limit various blocks to only within the wireframe and also a combine model created for TKM ridge with area code within the block model for TKN, TKS, TKE and SKW. o TKM(combine): 16400mE-19900mE, 73000mN-75640mN, 0mRL-600mRL o PHB: 15950mE-17750mE, 74250mN-75470mN, 0mRL-500mRL o KHN: 26600mE-29125mE, 74748mN-76524mN, -50mRL-650mRL o DSW: 23875mE-25555mE, 75250mN-75970mN, 0mRL-300mRL o DSM: 25500mE-28200mE, 75250mN-77110mN, 150mRL-450mRL o NLU: 22700mE-24500mE, 73730mN-75350mN, 150mRL-325mRL o NKW: 24500mE-26060mE, 74000mN-75320mN, 0mRL-300mRL o PVN_DKY: 15300mE-17460mE, 73700mN-75200mN, -0mRL-650mRL o PVW: 15300mE-17280mE, 73450mN-75010mN, -325mRL-655mRL o VNS: 27100mE-27820mE, 72400mN-73300mN, -0mRL-500mRL o HYN: 18700mE-19975mE, 71700mN-72420mN, -50mRL-400mRL o LOL: 25500mE-26550mE, 76600mN-7720mN, -400mRL-350mRL o PVT: 15000mE-15768mE, 73100mN-73895mN, 75mRL-300mRL
Estimation and modelling techniques	• Mineral Resource estimation was 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, Au, Ag, Ca, Mg, Mn, Fe, TotS (total sulphur), +/-Mo, +/-SCIS (sulphide sulphur), +/-S_SO4 (sulphate sulphur), +/-TotC (total carbon), +/-CCO3 (carbonate carbon) and +/-CAI (organic carbon)). Inverse distance to the power of two interpolation has been applied where there was insufficient data. This is considered appropriate for the estimation of Mineral Resources at Sepon. o Extreme grade values were managed by upper grade capping. The typical upper-cap used is the 99th percentile to contain outliers however this may vary depending on the results of geostatistical analysis. For some domains however, high-yield restrictions was applied to contain outliers, these values were generally not less than in the 99th percentile of the data, however this may vary depending on the results of geostatistical analysis. o In the copper block models geostatistical domains comprised various combinations of copper grade (nominal 0.1% Cu grade shell), lithology, oxidation (supergene / hypogene), sulphur grade (0.5% S grade shell in supergene – "TKS") and orientation wireframes based on the nature of the copper mineralisation deposition. In the gold block models geostatistical domains comprised gold grade domains (0.3g/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 was undertaken using hard domain boundaries and a series of elliptical search passes orientated in the plane of mineralisation. These search orientations and sizes were supported by variography analysis. ○ Visual assessment of the relationship between grade distribution and underlying geology supports the use of grade-based domains for constraining the Mineral Resource estimation. This is confirmed by statistical analysis which shows a clear grade increase across mineralisation boundaries. ○ A composite length of 1 m downhole was chosen for 2015 copper models, and the compositing process checked and validated. ○ 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 TKS, TKE and SKW ○ Interpolation was undertaken in multiple passes. ○ The minimum and maximum number of composites allowable to interpolate a block was typically set at 4-10 and 30-32 respectively and dependent on EDA results. ○ Octant searches were used for the first two passes. • Check estimates on the KHN, TKM and NLU models was undertaken by H&SC in 2014 on the models. The copper check estimates were unconstrained and employed independent variography and search strategies. These check models were within 10% of the contained metal of the MMG models. The NLU check model indicates that the MMG model underestimated total metal by up to 20% at a 0.5g/t Au cut-off. At 1g/t Au cut-off metal between the check model and the MMG model are comparable. All recommendations from the independent review have been implemented or are still part of ongoing investigation. In 2015 check estimate were done by Group Technical Services and Site Technical Services teams. • No assumptions about the recovery of by-products have been made. • Manganese is considered a deleterious element and has been estimated. Calcium, manganese and magnesium are estimated to assist in determining gangue acid consumption in the plant, and to assist in the identification of acid neutralising material. The various forms of sulphur and carbon are estimated to assist with copper speciation and in the determination of NAF (non-acid forming), PAF (potentially acid forming) material. Only total carbon and total sulphur have been estimated in the TKS_TKE_SKW estimations due to lack of data in other deposit areas. • In the copper block models parent block sizes of 25 m x 24 m x 5 m [East (X); North (Y); Elevation (Z)] were used. The parent block size took into consideration: the data spacing, likely mining methods and copper variogram models (QKNA). Sub-blocks honouring topography, copper grade domains and the base of supergene were used. The parent block size adequately delineates the ore zones within the block model, without

	compromising the localised calculated block variances. Using similar methods, parent blocks of 15 m x 6 m x 2.5 m [East (X); North (Y); Elevation (Z)] were used in the gold block models. Search distances in general are 1/3 of the variogram structures in pass 1 and doubled in subsequent passes additional larger passes were used to interpolate less well informed blocks. However this varies from deposit to deposit. • 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 Block model to wireframe volume differences are checked. 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) swathplots through the deposits are undertaken. o Validation block models to determine the impact of each variable change. o Reconciliation with grade control block models (where available) were undertaken. Results indicate good global reconciliation, but at times significant monthly variances. This led to a review of the copper estimation methods resulting in the new methods implemented in the KHN, TKN and TKM models.
Moisture	• Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	• Copper Mineral Resources have been reported at a cut-off grade of 1.1%Cu for type 1 (T1) chalcocite ore, 1.2% Cu for type 2 (T2) carbonate ore and 0.5% Cu for primary ore within a US$3.50 pit shell. The opportunity (through 2014-2015 studies) to process low grade supergene material is currently considered to be marginal and has therefore been removed from the Mineral Resource. Ongoing investigations of low grade supergene material are continuing and in the future some of this material may enter into the Mineral Resource. • Gold Mineral Resources have been reported at variable cut-off grades for open pit oxide/partial/primary gold as a result of an in depth cut-off grade review and investigation. Potential underground primary gold was reported at 5.9g/t Au cut-off. This cut-off grade was different from previous years due to different cost revenue, new metallurgical formulas and different cut-off grade method applied. Cut-off's for all deposits reported in this Mineral Resource are listed below.

Pit	**Classification**	**Oxide Cut-off**	**Partial Oxide Gold Cut-off**	**Primary Gold Cut-off**
DSM	Indicated	1.3	3.3	1.7
	Inferred	1.3	3.3	1.7

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<table>
<tr><td>DSW</td><td>Indicated</td><td>1.3</td><td>3.3</td><td>1.7</td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td>3.3</td><td>1.7</td></tr>
<tr><td>HYB</td><td>Indicated</td><td>1.3</td><td></td><td></td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td></td><td></td></tr>
<tr><td>LOL</td><td>Indicated</td><td>1.3</td><td></td><td>1.7</td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td></td><td>1.7</td></tr>
<tr><td>NKW</td><td>Indicated</td><td>1.3</td><td>4.5</td><td>2.3</td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td>4.5</td><td>2.3</td></tr>
<tr><td>NLU</td><td>Indicated</td><td>1.2</td><td></td><td>1.7</td></tr>
<tr><td></td><td>Inferred</td><td>1.2</td><td></td><td>1.7</td></tr>
<tr><td>PVN_DKY</td><td>Indicated</td><td>1.3</td><td>3.4</td><td>1.8</td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td>3.4</td><td>1.8</td></tr>
<tr><td>VNS</td><td>Indicated</td><td>1.3</td><td></td><td>1.7</td></tr>
<tr><td></td><td>Inferred</td><td>1.3</td><td></td><td>1.7</td></tr>
</table>

These Mineral Resource cut-off grade represents material that has reasonable prospects for eventual economic extraction.

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<tr>
<td>Mining factors or assumptions</td>
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The Mineral Resources are further constrained within pit shells optimised using MMG's pricing assumptions. For copper pit-shells pricing used was US$3.50 and for gold pricing used was US$1,212.

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

That the current copper processing plant will continue to operate treating chalcocite and copper carbonate ores.
That options for treating low grade copper ores (>1.1% Cu), including chalcocite (pre-concentration by flotation), copper carbonate (blended with existing feed) and primary ores (flotation) will prove technically and economically successful as per the current PFS and FS studies. That the existing oxide gold plant could be re-started to treat copper ore material, this is currently being investigated within an FS .
That the reconfiguration of the gold plant to treat copper is reversible and does not remove the possibility of processing gold at Sepon in the future.

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<tr>
<td>Environmental factors or assumptions</td>
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Environmental permitting in the Lao PDR is currently completed through the environmental and social impact assessment (ESIA) process.

The ESIA process is similar in nature to the process followed around the world. In February 2010, the Lao PDR issued the Decree on Environmental Impact Assessment. This decree replaces previous legislation.
In November 2011, the Environmental Impact Assessment Guidelines were released. This document was prepared in partnership with the Environmental Management Support Programme (a joint venture of the Lao and Finnish governments) and the Ministry for Foreign Affairs of the Government of Finland. The EIA Guidelines document provides an interpretation of the Decree and outlines the government expectations for what will be in an ESIA and how the ESIA should be conducted.

These two documents outline the process for environmental assessments in the Lao PDR. The Decree and associated guidelines - change the process for future environmental certificates required sustaining business growth.

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The acid rock drainage (ARD) characteristics were assigned to the model item ARD using the lithology domain and sulphur grade as described in table below. The values used were based on a study undertaken by Environmental Geochemistry International.

<table>
<tr><th colspan="4">NAF and PAF values assigned in the modelling process</th></tr>
<tr><th>Lithology domain</th><th>S1/2/3/4</th><th>ARD 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 = not acid forming
</td></tr>
<tr><td>Bulk density</td><td>

Samples for bulk density determination are taken from diamond drill core every 10 m using wax coated core immersion method.
The bulk density determinations were estimated into the TKM block model by inverse distance squared weighting within lithological domains where adequate data exists.
In the other deposits where density data is sparser, 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:

Geological: mineralisation continuity including spatial configuration and spatial continuity.
Sample quality: areas of wet RC drilling are downgraded.
Statistical: kriging efficiency and kriging slope of regression.
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.

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<tr><td>Audits or reviews</td><td>

MMG has an internal ‘Mineral Resource and Ore Reserve Policy’ that requires at a minimum external reviews every three years, and internal company review every interim year by MMG representatives reporting findings to a sub-set of the MMG Mineral Resource and Ore Reserve Committee. 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.
Independent technical review on the copper and gold Mineral Resource was undertaken in 2010 by AMC Consultants.
In 2014 the updated TKM and KHN Copper Mineral Resource estimates were audited by H&S Consultants. Numerous recommendations were made, the most significant relating to the treatment of high grades (considered too harsh by H&SC) and the distinction between supergene and primary copper mineralisation. These were addressed in 2015 within relevant Mineral Resource estimates.
Reconciliation to grade control models is used to assess the suitability of Mineral Resource models.

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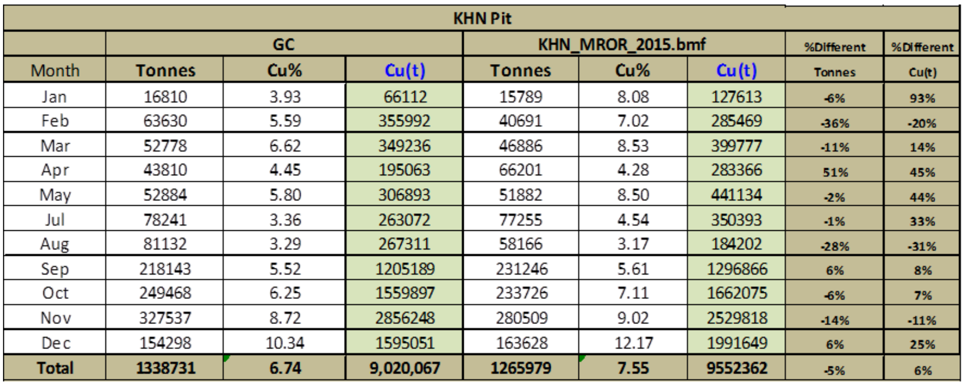
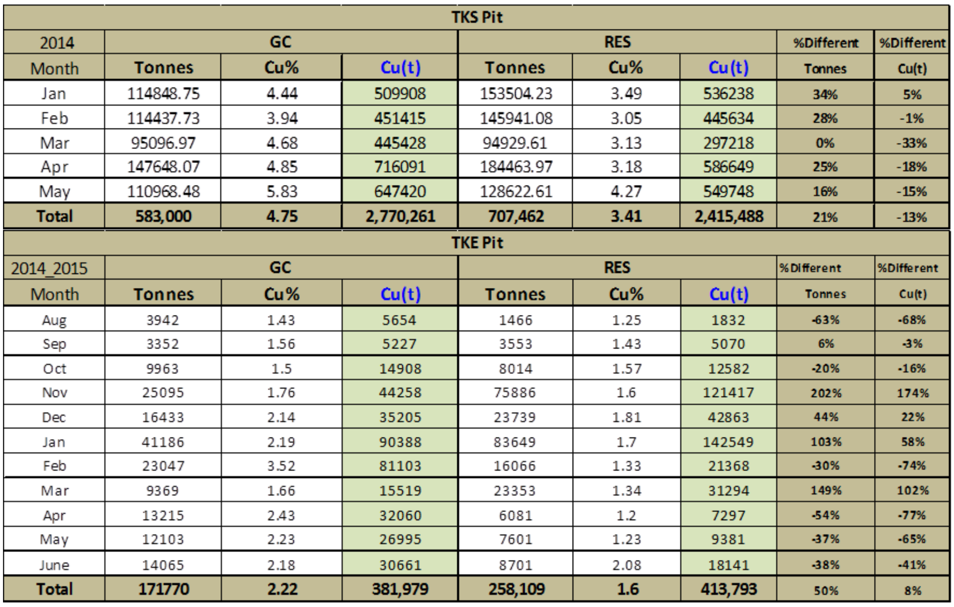
Discussion of relative accuracy/ confidence

- Block model estimation provides a global estimate of tonnes and grade without adjustment for change of support.
- Available reconciliation data (below) shows some level of inconsistency on predicted copper grade and tonnage by the Mineral Resource models on a pit by pit basis, however inconsistency falls within the acceptable limits for an Indicated Mineral Resource.



KHN Pit								
	GC			KHN_MROR_2015.bmf			%Different	%Different
Month	Tonnes	Cu%	Cu(t)	Tonnes	Cu%	Cu(t)	Tonnes	Cu(t)
Jan	16810	3.93	66112	15789	8.08	127613	-6%	93%
Feb	63630	5.59	355992	40691	7.02	285469	-36%	-20%
Mar	52778	6.62	349236	46886	8.53	399777	-11%	14%
Apr	43810	4.45	195063	66201	4.28	283366	51%	45%
May	52884	5.80	306893	51882	8.50	441134	-2%	44%
Jul	78241	3.36	263072	77255	4.54	350393	-1%	33%
Aug	81132	3.29	267311	58166	3.17	184202	-28%	-31%
Sep	218143	5.52	1205189	231246	5.61	1296866	6%	8%
Oct	249468	6.25	1559897	233726	7.11	1662075	-6%	7%
Nov	327537	8.72	2856248	280509	9.02	2529818	-14%	-11%
Dec	154298	10.34	1595051	163628	12.17	1991649	6%	25%
Total	**1338731**	**6.74**	**9,020,067**	**1265979**	**7.55**	**9552362**	-5%	6%



TKS Pit								
2014	GC			RES			%Different	%Different
Month	Tonnes	Cu%	Cu(t)	Tonnes	Cu%	Cu(t)	Tonnes	Cu(t)
Jan	114848.75	4.44	509908	153504.23	3.49	536238	34%	5%
Feb	114437.73	3.94	451415	145941.08	3.05	445634	28%	-1%
Mar	95096.97	4.68	445428	94929.61	3.13	297218	0%	-33%
Apr	147648.07	4.85	716091	184463.97	3.18	586649	25%	-18%
May	110968.48	5.83	647420	128622.61	4.27	549748	16%	-15%
Total	**583,000**	**4.75**	**2,770,261**	**707,462**	**3.41**	**2,415,488**	21%	-13%

TKE Pit								
2014_2015	GC			RES			%Different	%Different
Month	Tonnes	Cu%	Cu(t)	Tonnes	Cu%	Cu(t)	Tonnes	Cu(t)
Aug	3942	1.43	5654	1466	1.25	1832	-63%	-68%
Sep	3352	1.56	5227	3553	1.43	5070	6%	-3%
Oct	9963	1.5	14908	8014	1.57	12582	-20%	-16%
Nov	25095	1.76	44258	75886	1.6	121417	202%	174%
Dec	16433	2.14	35205	23739	1.81	42863	44%	22%
Jan	41186	2.19	90388	83649	1.7	142549	103%	58%
Feb	23047	3.52	81103	16066	1.33	21368	-30%	-74%
Mar	9369	1.66	15519	23353	1.34	31294	149%	102%
Apr	13215	2.43	32060	6081	1.2	7297	-54%	-77%
May	12103	2.23	26995	7601	1.23	9381	-37%	-65%
June	14065	2.18	30661	8701	2.08	18141	-38%	-41%
Total	**171770**	**2.22**	**381,979**	**258,109**	**1.6**	**413,793**	50%	8%

- There is some variability in the monthly reconciliation of copper tonnes and grade however there is no clearly defined trend for the Mineral Resource model under/over-estimating tonnage and grade.
- Prior to the oxide gold plant shutting down in late 2013 reconciliations showed some level of inconsistency on predicted grade and tonnage by the Mineral Resource models on a pit by pit basis. The primary gold material has never been mined and as a result, no reconciliation can be undertaken.

6.4 Ore Reserves – Sepon

6.4.1 Results

The 2015 Sepon Ore Reserve are summarised in Table 14.

Table 14 2015 Sepon Ore Reserve tonnage and grade (as at 30 June 2015)

Sepon Ore Reserve			
			Contained Metal
Supergene Copper[1]	Tonnes (Mt)	Copper (% Cu)	**Copper ('000 t)**
Proved			
Probable	8.3	3.6	300
Total	**8.3**	**3.6**	**300**
Copper Stockpiles[1]			
Proved	5.7	2.1	120
Total	**5.7**	**2.1**	**120**
Primary Copper[2]			
Probable	2.9	1.1	31
Total	**2.9**	**1.1**	**31**
Total Contained Metal			**450**

1 Cut-off grade is based on 1.1% Cu cut-off.

2 Cut-off grade is based on 0.5% Cu cut-off.

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

For low acid consuming sulphide material cut-off ranges from 1.1% to 1.5% Cu dependent upon pit haul distance to the crusher and its estimated Gangue Acid Consumption (GAC) value. For high acid consuming sulphide material the cut-off ranges from 1.2% to 5.3% dependent upon haul distance from the crusher and its estimated GAC value. For low acid consuming carbonate material the cut-off ranges from 1.4% to 1.5% Cu dependent on its GAC value. For high acid consuming carbonate material the cut-off ranges from 1.4% to 5.3% Cu dependent upon pit haul distance to the crusher and average gangue acid consumption. For primary material the cut-off is 0.5% Cu. Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

6.5 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 Reserve 2015

Assessment Criteria	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Ore Reserves. • MMG updated the Sepon Mineral Resource in June 2015 in accordance with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code) 2012 edition. The Mineral Resources were estimated and compiled for 18 deposits. Mineral Resources are modelled using solid wireframes of geological boundaries and/or a minimum 0.5% Cu cut-off boundary which approximates the natural break between copper and gold mineralisation and background grades. • The Ore Reserves includes ore on stockpiles. • Mineral Resource classification does not take into account the calcium and manganese grade. Classification is only based on the confidence of the copper or gold estimation. Calcium and manganese forms the basis for the gangue acid consumption (GAC) estimate which is a large proportion of the processing costs. As a result, there is a risk to the economics used in the cut-off grade (COG) calculation. Some risk has been removed though the averaging of the GAC.
Site visits	• The Competent Person, Dean Basile, visited the Sepon site in September 2015. He is currently contracted by MMG as the mining lead for a Prefeasibility Study. The 2014 Audit concluded that the operational practices were sound and comparable to industry standards. All recommendations from the audit have been addressed in the 2015 Ore Reserves. There have been no significant changes to the Ore Reserve Estimation methodology, hence the 2014 Audit continues to be valid.
Study status	• The mine is an operating entity. The Ore Reserves are based on actual operating data.
Cut-off parameters	• Break even cut-off grades were calculated for both the sulphide and carbonate ores. The COG estimates included all relevant costs incurred post mining and as "from the pit edge". The calculated COG's are approximately 1.1% Cu for supergene and vary depending upon pit haul distance to crusher and average GAC per area. The calculated GOG for primary material is 0.5% Cu. • Improvement has been made in the method used to estimate copper cut-off grades, from previous years.
Mining factors or assumptions	• Pit optimisations and designs adhered to recommended geotechnical parameters. The inter-ramp slopes used range between 18° – 40° for the clay material and 35° – 50° for the more competent material. The direct parameters that influence the

Assessment Criteria	Commentary
	inter-ramp slope angle using the current slope design parameters at Sepon are: ○ Rock type; ○ Hardness of the material ○ Geotechnical Sector; and ○ Groundwater conditions (Wet or Dry). - The mine is an operating entity. The Ore Reserves are based on actual operating data and projected forecasts. Additional pits that are yet to be developed are similar in nature to the current mining environment. - Mining dilution was estimated to be 5% and mining recovery used was 95%. Historical performance and high level dilution modelling were used to support these assumptions. Further work is required to better quantify and define mining dilution and recovery, particularly for primary ore. A test pit is proposed for primary ore mining during the second half of 2015. - The minimum mining width used for optimisations and design consideration was 20 m, based on the size of current mining fleet. - No Inferred Mineral Resource has been considered in any part of the derivation of the Ore Reserves. - Current mining infrastructure is sufficient to realise the Ore Reserves. - At the time of preparing the 2015 Ore Reserve further investigations were ongoing to refine the geotechnical parameters. The outcome from this analysis has the potential to influence overall NPV its potential impact on Ore Reserves is low.
Metallurgical factors or assumptions	- The Sepon mine is an operating entity. 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. - Copper recovery is determined by the equation: Cu recovery (%) = (Cu Grade – Tails Grade (0.32%) / Cu Feed Grade) – Soluble Loss (3.66%) - 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 5 years actual recovery has averaged over 90%. - The main deleterious component in the ore is dolomite which increases acid consumption in the leaching process. Gangue acid consumption is estimated for supergene 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 if Ca/Mg < 2.5 AND Mg>Ca 30.5 + 43.8 x % Mg x 0.1 + 16.5 x % Mn

Assessment Criteria	Commentary
	Net acid consumption (NAC) is then calculated using the formula: NAC (kg/t) = GAC (kg/t) + Acid Lost To Tails (kg/t) – Acid Generated From POX (kg/t) • 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 sulphide ore, which does not contain much dolomite a gangue acid consumption of 15 kg/t is assumed which is the historical average. • As required, blending of carbonate and sulphide ore is used to control the acid requirement based on their respective calculated GAC values. • A further GAC control measure is currently in the commissioning phase. It essentially consists of a trammel screen (scrubber) that is designed to screen out competent dolomite material, effectively reducing the average GAC of the material entering the plant. • 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 2.1 Mtpa of supergene ore and 0.6 Mtpa of primary ore to produce a maximum of 90 ktpa of copper cathode has been assumed. Production rates for the supergene and the cathode have been demonstrated as sustainable over the last 12 months. Primary production rates have been estimated as part of the Sepon Sustain Expansion study – the technical level of the study related to the engineering portion of the proposed primary processing rate is considered to be at a Prefeasibility level of study confidence.
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). • Current management of all mineral wastes at Sepon is governed by the MMG Waste Rock Management Sustainability Standard and Waste Rock Management Code of Practice. • Waste rock acid generation is currently based on the sulphur grade. Potential acid forming (PAF) material is defined as material with a sulphur grade above 0.3%. Waste is separated based on its potential for acid generation, with acid generating material being encapsulated within the waste dump. • The tailings dam is currently at the 300 mRL, to provide capacity for the Ore Reserves it needs to be increased to the 303.5 mRL, this cost has been included in the cash flow model.
Infrastructure	• No significant additional site infrastructure is required to realise the open pit Ore Reserves. The Primary processing facilities

Assessment Criteria	**Commentary**
	proposed consist of existing decommissioned gold processing facilities, only minor work is required to tie these facilities into the existing plant.
Costs	• The site operating costs used in the determination of Ore Reserves were provided by the site Commercial department. Information was sourced from the 2015 cut-off grade report, these costs are in line with historical performance. • The mining costs have been based on the historical; contractor rates (rehandle and reclaim) and forecasted costs. All relevant costs have been considered in the derivation of the Ore Reserves. Mining costs have been included in the optimisations and cash flow model. Relevant costs approximate to a mining cost of US$2.80/t mined, this cost varies with mine area, elevation and the number activities applied to a given mining block. • The processing costs, including pant maintenance used in the determination of Ore Reserves were based on historical and forecasted site cost models. Approximate processing costs of US$60/t of ore and US$450/t copper produced are estimated once all relevant fixed and variable costs have been applied. The specific cost will vary by ore type and GAC. • The Ore Reserves used information supplied by MMG Corporate in regards to metal prices and economic assumptions. The long term copper price assumption of US$2.95/lb was used for evaluating all pits. Metal prices are derived from a combination of broker consensus and internal strategy evaluations • No exchange rate is used in the Ore Reserves estimate as all expenditure and revenue is reported in US dollars. • Copper cathode is produced on site limiting the selling costs to US$91/tonne copper metal. The selling cost includes all onward costs such as transportation and marketing. • A cash flow model was produced based on the detailed schedule. This model includes the aforementioned costs as well as all sustaining capital that is needed to realise the Ore Reserves. • A royalty of 4.5% to be paid to the Government of the Lao P.D.R has been used. • Current general and administration site costs and the mining costs have been used to forecast future costs. All relevant costs have been included in the Ore Reserves estimation, however a greater understanding and separation of fixed and variable costs would refine the cash flow model. Potential changes are not expected to have a material effect on the Ore Reserve estimate.
Revenue factors	• The Ore Reserves used information supplied by MMG Corporate in regards to metal prices and economic assumptions. The long copper price assumption of US$2.95/lb was used for evaluating all pits. Metal prices are derived from a combination of broker consensus and internal strategy evaluations.

Assessment Criteria	**Commentary**
	• Copper cathode is produced on site limiting the selling costs to US$91/tonne copper metal. The selling cost includes all onward costs such as transportation and marketing. • An LME premium of US$118/tonne copper metal is received on Grade A cathode that is produced. It is assumed that 90% of produced copper will receive this premium.
Market assessment	• MMG has a long-term positive view of copper market fundamentals with future supply likely to be constrained as declining grades, increasing costs, slow future mine production and investment. MMG's view centres on the future copper supply contracting as current reserves are depleted and continued global demand copper is forecast to grow at or above the global average rate.
Economic	• The inputs that inform the economic analysis include all foreseeable operating and capital costs, resulting in a positive NPV for the Ore Reserve. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit. • The assumptions outside of MMG's direct control i.e. metal price were varied +/-20% with the results indicating that the Ore Reserve is robust. • At the cut-off grades used for the Ore Reserves the Sepon operations have robust economics.
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. • Where community access agreements to land for mining have not been settled, no Ore Reserves have been declared.
Other	• No naturally occurring risks have been identified. • All necessary legal and marketing arrangements are in place to realise the Ore Reserves. • All government agreements and approvals required to realise the Ore Reserves are current and will be in place until the end of mine life.
Classification	• The Ore Reserves estimate is based on the Mineral Resource estimates classified as "Measured" and "Indicated" after consideration of all mining, metallurgical, social, environmental and financial aspects of the operations. • All Proved Ore Reserves have been derived from the Measured Mineral Resource only where grade control drilling has been carried out on a 5 m x 3 m pattern and material mined and

Assessment Criteria	**Commentary**
	stockpiled. • All Probable Ore Reserves have been derived from Indicated Mineral Resources based on the supporting data. Indicated Mineral Resources exist where grade control has not been conducted with drilling generally based on a 25 m to 50 m spacing. • The Ore Reserves do not include any Inferred Mineral Resources in any of the Ore Reserves classifications.
Audit or reviews	• An Ore Reserve Method Audit was completed by Mining One Pty Ltd in April 2014.
Discussion of relative accuracy/ confidence	• Whilst there are a number of parameters for which there is low confidence, the impact of this uncertainty on the remaining Ore Reserves is such that the likelihood of destroying the robust economics of the remaining Ore Reserves is low. • The Ore Reserves are based on a global estimate with the exception of material that has been grade controlled on a 5m x 3m pattern, mined and stockpiled. • This Ore Reserve is based on the results of an operating mine. The confidence in the estimate is compared with actual production data. This data is subject to continual review.

6.5.1 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 Reserve

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
EXPERT PERSON / COMPANY	**AREA OF EXPERTISE**
Chevaun Gellie, Senior Specialist Resource Geologist, MMG Ltd (Melbourne)	Geological Mineral Resources
Phonesavanh Khamvongsa, Mining Manager – Sepon MMG Ltd (Sepon)	Mining
Timothy Cribb, Operation Manager – Sepon MMG Ltd (Sepon)	Infrastructure and Capital Projects
Richard Horton, Senior Mining Engineer MMG Ltd (Sepon)	Mining Engineering
Leonardo Paliza, Principal Metallurgist MMG Ltd (Sepon)	Metallurgy
Latdavanh Nhotmanhkhong, Manager - Commercial MMG Ltd (Sepon)	Economic Assumptions

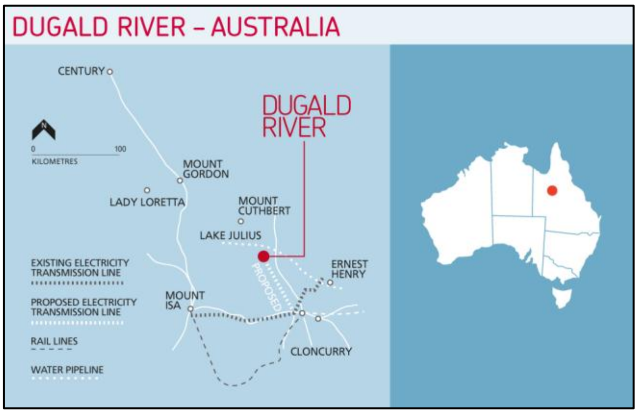
7 DUGALD RIVER PROJECT

7.1 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 7-1). 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 a Mineral Resource of 62Mt at 13% Zn, 2% Pb, 35g/t Ag and is wholly owned by a subsidiary of MMG Limited.

Figure 7-1 Dugald River project location



7.2 Mineral Resources – Dugald River

7.2.1 Results

The 2015 Dugald River Mineral Resources are summarised in Table 17. The Mineral Resource has been depleted to account for mining of ore by way of underground development of ore drives and stope production. The 2015 Mineral Resource has been reported above an A$134/t NSRAR *(net smelter return after royalty*) cut-off, which is comparable to a grade of 6.8% Zn, 0.82% Pb and 10.17 g/t Ag.

Table 17 2015 Dugald River Mineral Resource tonnage and grade (as at 30 June 2015)

Dugald River Mineral Resources							**Contained Metal**				
Zinc[1]	Tonnes (Mt)	Zinc (% Zn)	Copper (% Cu)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Measured	5.3	14.4	0.0	2.0	66		759	0	107	11.1	
Indicated	25.9	13.3	0.0	2.2	51		3,439	0	579	42.5	
Inferred	25.7	12.7	0.0	1.8	13		3,273	0	464	11.1	
Total	**56.9**	**13.1**	**0.0**	**2.0**	**35**		**7,471**	**0**	**1,150**	**64.8**	
Zinc Stockpile[1]											
Measured	0.5	15.5	0.0	1.4	38		72	0	6	0.6	
Total	**0.5**	**15.5**	**0.0**	**1.4**	**38.4**		**72**	**0**	**6.3**	**0.6**	
Primary Copper[2]											
Measured											
Indicated											
Inferred	4.4	0.0	1.8	0.0	0.0	0.2		79	0	0.0	0.03
Total	**4.4**	**0.0**	**1.8**	**0.0**	**0.0**	**0.2**		**79**	**0**	**0.0**	**0.03**
Total Contained Metal							**7,542**	**79**	**1,156**	**65.3**	**0.03**

1 Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value A$134 /t. NSRAR A$134 equates to approximately 6.8% Zn + 0.82% Pb + 10.17 g/t Ag. NSRAR = (1770 x Zn%) + (1680 x Pb%) + (0.24 x Ag g/t).

2 1% Cu cut-off grade.

Contained metal does not imply recoverable metal.

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

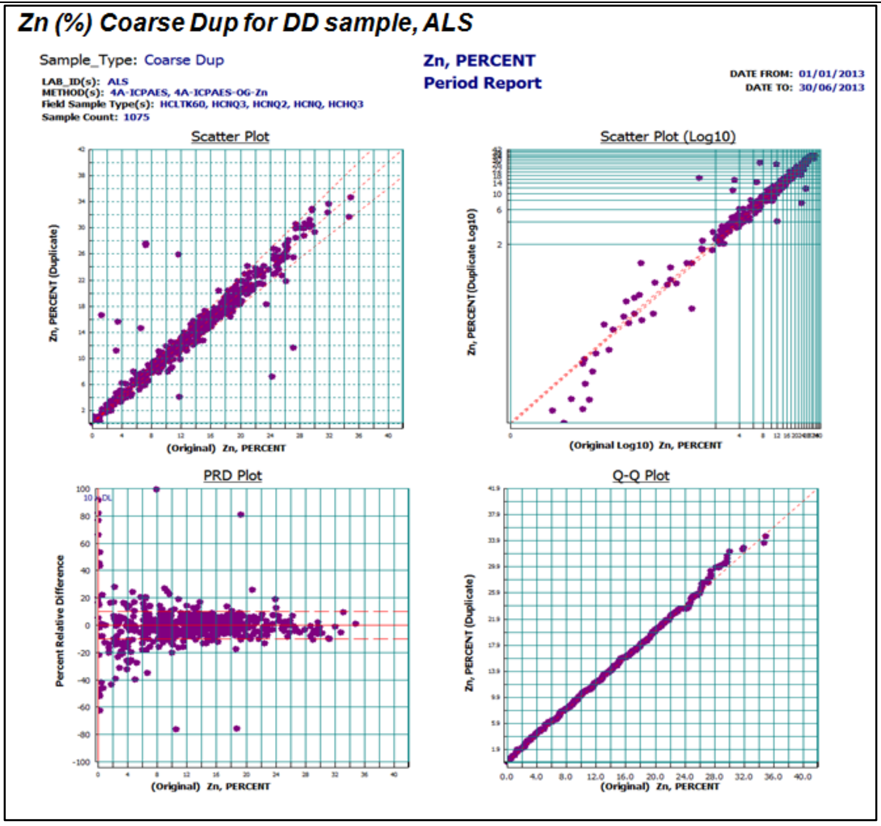
7.3 Mineral Resources JORC 2012 Assessment and Reporting Criteria

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 Resource 2015

Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• Diamond drilling (DD) was used to obtain an average 1 m sample length while still respecting geological contacts. DD core was sampled either whole, ¾, ½, ¼, or sliver for the PQ core. Once samples were selected by a geologist the samples were marked and sample ID's stored in the database. • Samples were transported to ALS Mount Isa laboratory where the sample was crushed and pulverised to produce a pulp (>85% passing 75 µm). The pulps were transferred and analysed at ALS Brisbane. • 3% of the dataset was sampled using reverse circulation (RC) drilling techniques. • There are no inherent sampling problems recognised. • 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.
Drilling techniques	• A number of drilling techniques were used; with 97% of samples used in the Mineral Resource from DD samples (HQ, HQ2, HQ3, NQ2, NQ3, PQ, LTK60 and unknown core size), the remaining 3% of samples used in the Mineral Resource were from RC samples. 59% of pre 2007 surface diamond drilling does not have an entry in the database that allows separation of the data by drill hole diameter. • Core sizes of drill holes post 2007 have been captured in the database. • Post 2007 data has correctly been captured in the database. • 2014 underground drilling data is predominantly NQ2 with some LTK60.
Drill sample recovery	• 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 core. • 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.
Logging	• All core samples including RC pre-collars have been geologically logged (lithology, stratigraphy, weathering, alteration, geotechnical characteristics) to a level that can support appropriate Mineral Resource estimation.

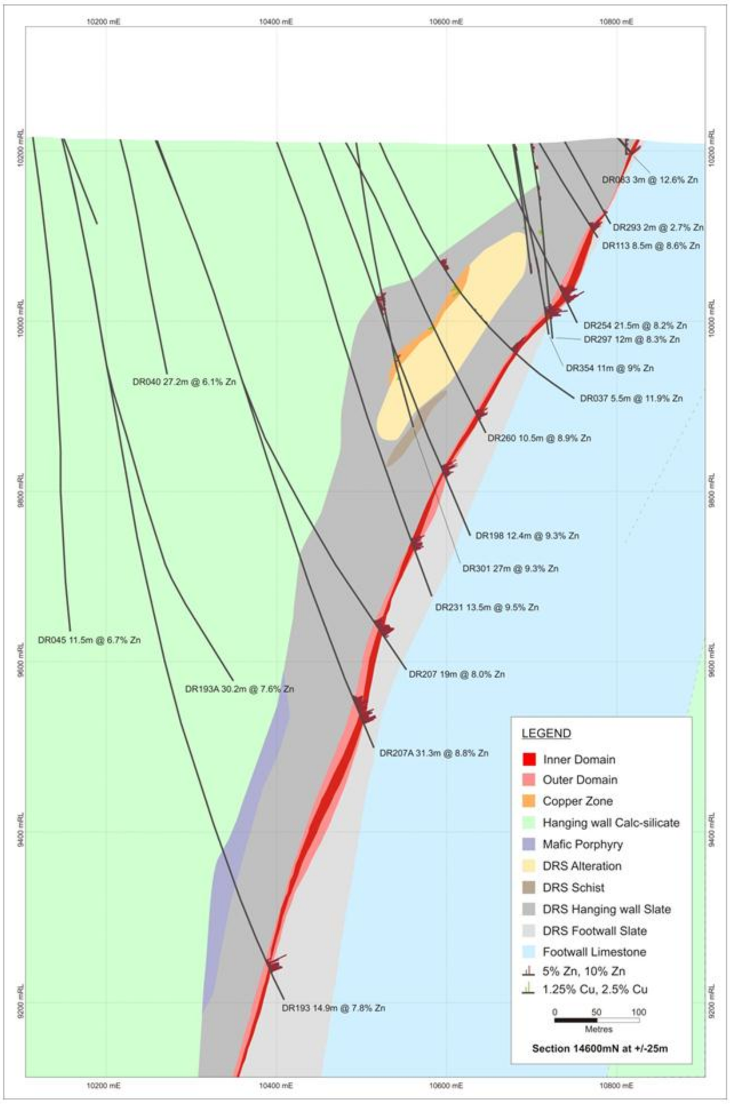
	• The logging captures both qualitative (e.g. rock type, alteration) and quantitative (e.g. mineral percentages) characteristic. Core photographs are available for most drill holes. All drill holes post-2008 have been photographed (wet and dry). • Mineralised core is stored at -4°C in refrigerated containers to minimise oxidation for metallurgical testing. Non-mineralised core is stored on pallets in the yard. • Currently, all drill holes are logged using laptop computers directly into the drill hole database. Logging has occurred in the past onto paper log-sheets and was then transcribed into the database. • A total of 256,167.1 m of drilled data is contained in the database, of this 88% is geologically logged, and 20% of drilled data contains assays.
Sub-sampling techniques and sample preparation	• Core was cut by diamond saw. Half of the core was retained onsite for future reference, the other half was sampled. • Post 2010 all drilling is DD. DD allows collection of representative samples of the mineralisation 'in situ'. The vast majority of drill hole intersections cross cut the mineralisation and as such are representative. • The standard sampling length is about 1 m with a minimum of 0.7 m and a maximum 1.2 m. Sample intervals do not cross geological boundaries. • Historical RC programmes were designed to test the 'unmineralised' hanging wall material in DD pre-collars. 2 m bulk composites stored at the drill site were sampled using the spear method. • The sample preparation of RC chips and DD core adheres to industry best practice. Samples are bagged, numbered and dispatched to the ALS Mount Isa laboratory. At the laboratory, each sample is weighed, the The sample is jaw crushed, 50% split and then further crushed using a Boyd crusher 70% nominal passing 2 mm. The sample is rotary split with 500 g - 800 g retained and pulverised to 85% passing 75 µm. All rejected material is collected and saved. Pulps are then sent to ALS Brisbane for analysis. • Quality control procedures for all sub-sampling stages to maximise representivity include: 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 are inserted at a rate of 1 per 20 samples (approximately 4 per drill hole). Replication of the field duplicates is considered satisfactory. A total of 1075 coarse duplicates were selected and submitted to ALS for analysis. The performance of the duplicates is charted below showing good repeatability of expected zinc values. The scatter between the duplicates and original data sits roughly along the one to one line for the majority of the data set indicating reasonable accuracy and precision.

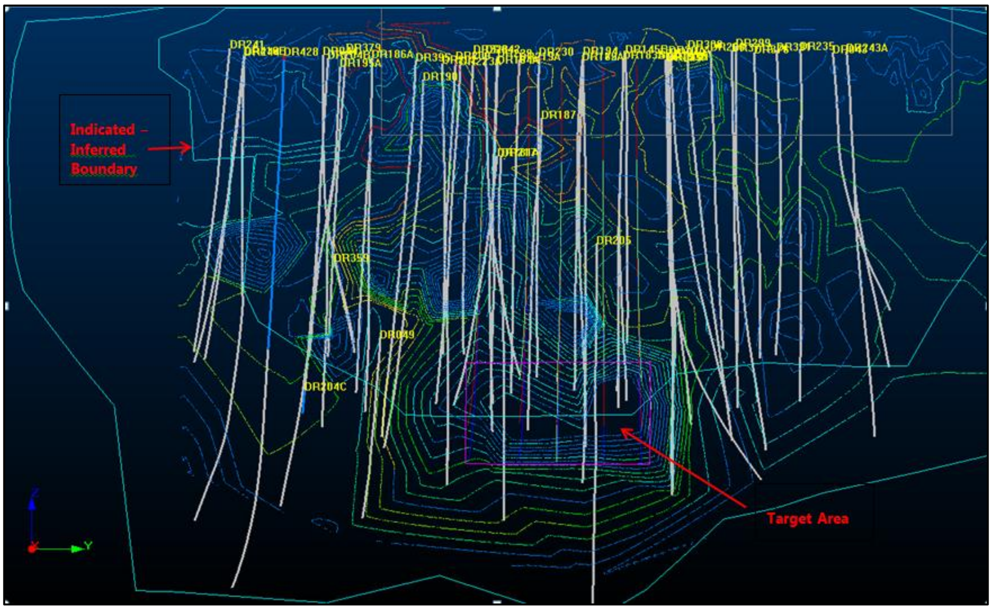
	 **Coarse Duplicates from Diamond Drill Samples at ALS (Zn%)** • Repeats of coarse rejects are undertaken periodically • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Dugald River mineralisation (sediment hosted base metal) by the Competent 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 Resource. • 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) are each submitted at the rate of 1:20. The selection and location of standards and 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. • The performance of the CRM's indicates a scatter of returned values within 3 standard of deviation to the expected mean value. This indicates poor precision and repeatability. • Duplicates are selected from returned coarse rejects and submitted to an

	independent assay laboratory (Genalysis). The performance of the duplicates indicates good repeatability of expected values and no bias. • Repeats of coarse rejects are analysed periodically.
Verification of sampling and assaying	• Verification of assay results was visually verified against logging and core photos by alternative company personnel. • No twinning of drill holes have occurred at Dugald River. • Core logging data was recorded directly into a Database (GBIS) by experienced geologists (geological information such as lithology and mineralisation) and field technicians (geotechnical information such as recovery and RQD). Where data was deemed invalid or unverifiable it was excluded from the Mineral Resource estimation. • No adjustments to the assay data is performed during import into the GBIS Database. Conversion of negative (below detection limit) data is later performed in Datamine by adjusting negative values to half the detection limit.
Location of data points	• All drill hole collars have been surveyed by licensed surveyors. Surface collars were surveyed in MGA94 and then converted to local mine grid. Underground collars were surveyed in local mine grid using total station. • Strong local magnetic fields associated with pyrrhotite mineralisation within the deposit reduce the effectiveness of conventional downhole survey tools. Hence all underground diamond drill holes and 181 surface drill holes have been gyroscopically surveyed. All other drill holes have been surveyed by downhole single-shot camera surveys. • 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 drill holes. 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 10 m x 10 m while the lowest drill density is greater than 100 m x 100 m spacing. • Locations drilled at 10 m x 10 m and up to 20 m x 20 m 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. • Drill hole data is concentrated within the top 300 m of the Mineral Resource with broader-spaced drilling at depth, due to the difficulty and cost involved in drilling deeper sections. • DD samples are not composited prior to being sent to the laboratory for analysis however the nominal sample length is generally 1 m.
Orientation of data in relation to geological	• 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

structure	and west-east directions to intersect mineralisation across-strike. • Drilling orientation is not considered to have introduced sampling bias. Drill holes that have been drilled down dip and semi-parallel to the mineralisation have been excluded from the estimate.
Sample security	• Measures to provide sample security 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. o Calico sample bags transported by courier to assay laboratory. o Assay laboratory checks of sample dispatch numbers against submission documents. o Assay data is returned separately as a text file and a pdf file.
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 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 Resource is located. EPM12163 consists of 6 sub-blocks and covers an area of 20 sqkm 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 in the following points: o Discovered in 1881 with the first drilling programme commencing in 1936 comprising three drill holes. The maiden Mineral Resource was reported in 1953 by Zinc Corporation. Drilling continued from 1970 through 1983 totaling 28 drill holes. CRA re-estimated the Mineral Resource in 1987. Between 1989 and 1992 a further 200 drill holes 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 and 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 studies and a focus on the northern copper zone in 2010 were completed. In 2011 the decline commences which resulted in trial stoping. In 2014 underground development and drilling focused on confirming and extending mineralisation continuity of the Dugald River load.
Geology	• The Dugald River style of mineralisation is a sedimentary 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 3 km-4 km wide 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 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 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. • Mineralised widths vary from 3 m to 30 m. The mineralised zone extends approximately 2.4 km in strike length and up to 1.35 km down dip.
Drill hole information	• 1,115 DD holes and associated data are held in the database. No individual hole is material to the Mineral Resource estimate and hence this geological database is not supplied.
Data aggregation methods	• 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. However the Mineral Resource has been reported above an A$134 NSRAR cut-off.
Relationship between mineralisation width and intercepts lengths	• Mineralisation true widths are captured by three-dimensionally modelled wireframes with drill hole intercept angles ranging from 90° to 45°. • The true thickness of the majority of the Mineral Resource is between 3 m and 30 m with the thickest zones occurring to the south.

Diagrams	 **Cross section looking north – showing true width of the mineralisation**
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	- MMG plans to continue to improve geological confidence and Mineral Resource classification through infill drilling programmes over the next few years. Two planned drilling programmes are shown below and include: o 12,500 m of down-dip infill drilling from surface targeting the thick, high-grade mineralisation in the south of the deposit is planned for 2015. This drilling is designed to convert Inferred Mineral Resources to Indicated Mineral Resources. o An on-going program of more extensive underground drilling throughout the deposit is expected to convert Inferred and Indicated Mineral Resources to Measured.

	○ Future exploration work will focus on identifying extensions of the Dugald River load.  **Long Section indicating proposed Indicated – Inferred drilling target area**

Section 3 Estimating and Reporting of Mineral Resources

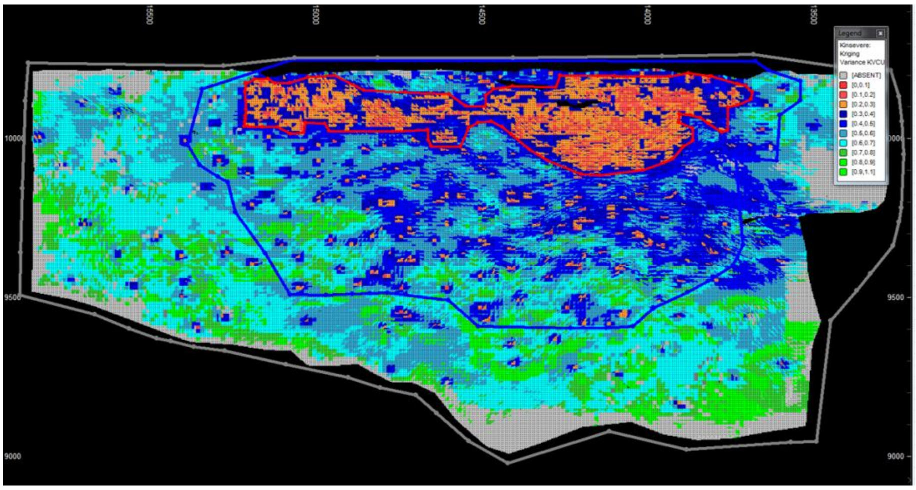
Database integrity	• The following measures are in place to ensure database integrity: ○ All data is stored in an SQL database that is routinely backed up. ○ All logging is digital and directly entered into the onsite GBIS database via a wireless connection (at the core shed).Data integrity is managed by internal GBIS validation checks/routines that are administered by the Melbourne Database Group and/or the site Geology Team. • The measures described above ensure that transcription or data entry errors are minimised. ○ Data validation procedures include: Database validation procedures are built in the database system to manage accurate data entry during logging and collection of data. ○ Prior to use in the Mineral Resource the data was checked externally by running Datamine macros on the drill hole file to check for end of hole depths, and sample overlaps. ○ Manual checks were carried out by reviewing the drill hole data in plan and section views.
Site visits	• The Competent Person visited site on various occasions through 2014 and 2015. Site visits included involvement with: ○ Assist with wireframe interpretation and methodology as applied in the 2014 Mineral Resource work. ○ Inspection of geological mapping plans.

	o Inspection of underground workings. o Inspection of drill holes 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. There is no defined zinc cut-off boundary rather 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 the drill holes 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. • Alternative geological interpretations were not considered. • Selection of the low/high grade domain was based on geological observations and assay results. Zinc grade histograms in combination with geological logging were used to assist in selecting this contact. • Underground mapping of development drives for both access and ore drives were also used in assisting with the geological interpretation. • Globally the Dugald River deposit follows a reasonably predictable sheet of mineralisation but with short-range (10 m to 20 m-scale) variations associated with localised structures that are suitably defined by close-spaced drilling as within the Measured Mineral Resources.
Dimensions	• The main Dugald 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. • The strike length of mineralisation is approximately 2,400 m. Dip varies between 85° and 45° to the west. • The true thickness of the majority of the Mineral Resource is between 3 m and 30 m with the thickest zones occurring to the south. • The mineralisation is open at depth. The deepest drill intersection of mineralised material is about 1,140 m below the surface.
Estimation and modelling techniques	• Mineral Resource modelling was completed using 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 and total carbon. This is considered appropriate for the estimation of Mineral Resources at Dugald

	River. ○ 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. Capping was applied to domains that showed a coefficient of variation (CV) > 1.2. A CV > 3 that indicated a high degree of population mixing and required a revision of domains used in the estimate. Grade cap values were selected using a combination of both histogram and cumulative log probability plots (using cell declustering). ○ Grade estimation was performed using dynamic anisotropy, which uses the dip and dip direction of the wireframes to align and optimise the search direction of the estimate. ○ 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 lithological domains. ○ An unfolding process was used on the drilling data prior to variogram analysis being performed. These variogram ranges were then applied to the search parameters used in the estimation. ○ Orientation of the search ellipse was optimised using the Datamine Dynamic Anisotropy method, that is dip and dip direction of the wireframes was used in the estimation of the model. ○ Drill hole compositing resulted in nominal 1 m intervals with residual composite intervals absorbed evenly into the composites resulting in no loss of sample intervals. ○ Separate variography and estimation were performed for Zn, Pb, Ag, Mn, Fe, S and total carbon. ○ No assumptions have been made about the correlation between variables. All variables are comparably informed and independently estimated. ○ The mineralisation boundary assumes that the zinc, lead and silver populations are spatially correlated. However, there is evidence to suggest that the lead and silver exhibit population mixing within the zinc domain. ○ Interpolation was undertaken in two stages: ▪ Stage1: Ordinary Kriging applying three passes with varying search ellipse dimensions ▪ First pass is equal to 80% of the variogram range ▪ Second pass is equal to the variogram range ▪ Third pass is equal to 1.5 x variogram range ▪ Stage 2: Inverse distance squared technique used to estimate blocks not estimated by the Ordinary Kriging stage. ○ A minimum number of 2 drill holes were used for all estimates. ○ Number of composite samples was restricted to a minimum of 8 and a maximum of 20.

	- Octant method was applied to the Ordinary Kriging estimate requiring a minimum of 2 octants to be filled. Minimum and maximum samples per octant are 2 and 6 respectively. - Block discretisation of 2 x 4 x 4 was applied. • There has been no significant mineral processing for the Dugald River material. Reconciliation from production cannot be undertaken. Reconciliation work is restricted to comparisons between models for volumes mined as the bulk of mined ore is yet to be processed. Comparisons between the 2013 to 2014 model in the stoped areas (represents ~1% of the deposit) shows comparable tonnes but with high variability in grade (+ 27% Zn, - 16% Pb and -48% Ag). The difference is due to additional close-spaced (10 m by 10 m) drilling available for the 2014 model. • Assumptions have been made regarding the recovery of all by-products in the NSRAR. • Deleterious elements include manganese and carbon, which have been estimated in the block model. Ancillary elements estimated include Mn, Fe, S and total carbon. • Parent block size was set at 2.5 m x 12.5 m x 12.5 m with sub-cells of x=0.5 m, y=0.5 m, z=0.5 m. Sub-cells were assigned parent block values. The parent block size assumes mining selectivity at the stope level. • No external dilatation 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. • 2014 block model validation included the following steps: - Comparison against the 2013 block model including visual comparison of plans and cross-sections, tonnes grade curves, cumulative probability plots and trend plots. - Comparison against drill hole data using visual comparison of plans and cross-sections, statistics by domain, cumulative probability plots and trend plots.
Moisture	• Tonnes in the model have been estimated on a dry basis.
Cut-off parameters	• The Mineral Resource is reported above an A$134/t NSRAR (net smelter return after royalties) cut-off, which is approximately 70% of the Dugald River Ore Reserve cut-off. NSRAR A$134 equates to approximately 6.8% zinc, 0.82% lead, 0.17 g/t Silver. The selection of the A$134/t NSRAR 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 with the long-hole open stoping method favoured. Currently the deposit is accessed by two declines and trial stoping is being undertaken to determine the optimal stoping method. • No external dilatation has been applied to block grades. However, parent block size has assumed mining selectivity at a stope level.

	• The Mineral Resource has been depleted to account for trial stope 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 Reserve. • Manganese percentage in the zinc concentrate is calculated by way of an algorithm contained within the NSRAR 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. • All material brought to the surface is stockpiled in designated areas, based on their classification of either potentially acid forming (PAF) or non-potentially acid forming (NAF) material. Waste PAF/NAF determined based on the buffering potential of limestone material such that : o Limestone material with less than 3%-5% sulphide is considered NAF o Limestone material with greater than 3%-5% sulphide is considered PAF o All other material is considered PAF • 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 inverse distance squared. 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 2013 block model for bulk density calculations. • A density of 2.75 g/cm3 has been assumed for the waste host domain.
Classification	• 2015 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). This method has remained unchanged from 2013. • Mineral Resource categories are generally based on: o Measured: < 20 m drill spacing, KV<0.3, KE>40%, RS>0.7 plus

	presence of underground development, geological mapping and grade control drilling. o Indicated: > 20 m to <100 m drill spacing, KV>0.3<0.6, KE < 40%, RS<0.4. o Inferred: > 100 m drill spacing, KV>0.6, KE<40%, RS < 0.3. • 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 20 m x 20 m 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 Resource classification reflects the Competent Persons view on the confidence and uncertainty of the Dugald River Mineral Resource. • Below is a long section looking east of the Dugald River mineralisation lode showing blocks coloured by KV and the Measured, Indicated and Inferred wireframes used in selecting the Mineral Resource classification. o Measured = red o Indicated = blue o Inferred = grey  **Long-section of the Dugald River Block Model, blocks coloured by KV**
Audits or reviews	• No external independent audits have been performed in 2015. However, several external audits have been completed in recent years including Pennywise Ltd and Lewis Mineral Resource Consultants Pty Ltd in 2013. • The Lewis Mineral Resource Consultants Pty Ltd report found that the estimation process is sound, however was concerned about the method of selection for the inner domain boundary as it was not considered "natural". This concern has been addressed in the 2014 Dugald River Mineral Resource with selection of the grade boundary completed by looking at the drill holes grade interceptions in Datamine; plotted as a grade histogram and selecting a boundary which is more representative of geology. • An internal MMG review has been carried on the current 2014 Mineral Resource estimate. This involved personnel from both Corporate and

	Dugald River sites. No material items to the Mineral Resource have been identified.
Discussion of relative accuracy / confidence	• 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 10 m x 10 m drilling density, the Competent Person is of the opinion that confidence is increased 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 Resource confidence against actual mined tonnes and grades of the deposit. • Tonnes and grade checks comparing the 2013 and 2014 Mineral Resource and grade control 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 <20 m is required for both tonnes and grade accuracy and confidence. • Drilling at 20 m in locations that also have underground ore drive development and geological mapping of the deposit provides good confidence in the geological continuity and confidence in the tonnes. • Drilling at >20 m provides less confidence in both tonnes and grades accuracy if no underground development and geological mapping of the deposit is present.

8.1 Ore Reserves – Dugald River

8.1.1 Results

The 2015 Dugald River Ore Reserve are summarised in Table 19.

Table 19 2015 Dugald River Ore Reserve tonnage and grade (as at 30 June 2015)

Dugald River Ore Reserve							
					Contained Metal		
	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
Primary Zinc							
Proved							
Probable	22.1	12.3	2.0	50	2,703	435	36
Total	**22.1**	**12.3**	**2.0**	**50**	**2,703**	**435**	**36**
Primary Zinc Stockpiles							
Proved	0.5	15.5	1.4	38	72	6	0
Total	**0.5**	**15.5**	**1.4**	**38**	**72**	**6**	**0**
Total Contained Metal					**2,774**	**441**	**36**

Cut-off grade based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value A$134/t.
Contained metal does not imply recoverable metal.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.

8.2 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 Reserve 2015

Assessment Criteria	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Mineral Resources used to define the Ore Reserves. • The Mineral Resource model used was the MMG June 2014 Mineral Resource model. • Risks associated with the model are related to orebody complexity seen underground but not reflected in the Mineral Resource model due to the spacing of the drill holes that inform the model. • The 2014 geotechnical model was used to estimate the hangingwall (HW) thickness, tonnes and grade of the unplanned dilution applied to the 2015 stope shapes.
Site visits	• Karel Steyn is the Competent Person for the Dugald River Ore Reserve base in Melbourne Office and frequently visited the site during 2015.
Study status	• The Dugald River study has progressively been enhanced. ○ 2008/09 Feasibility Study Report (FS09) ○ 2010 Feasibility Report Update which incorporated work from both Ausenco and AMC ○ 2012 Board Submission – basis for conditional project approval ○ 2013 Mining Method Review – recommendation for alternate mining assumptions ○ 2013 Business Options Review – reassessment of business case for the revised mining assumptions (2013BOR) ○ 2014 Dugald River DRAFT Ore Reserve Supporting Document_04 November 14 (2014OR) ○ 2015 Dugald River Updated Development Plan (2015OR) • The initial mine design was detailed in a Feasibility Study undertaken in 2008 and released in January 2009 (FS09). • There has been a series for reviews done on operating and capital costs, infrastructure optimisation, trial mining has taken place, and metallurgical studies are in progress. The enhancements and risks to FS09 have been stated below. • 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 HW were more challenging for dilution control than assumed in the 2009 Feasibility Study. • In November 2012, a major geotechnical study was commenced

Assessment Criteria	Commentary
	involving re-examination and re-logging of all diamond drill core and re-analysis of the geotechnical parameters of the ore-zone and HW zones. • Detailed design work, including scheduling and cost modelling, was undertaken by AMC Consultants Pty Ltd for a 20 m development level spacing x 15 m stope strike length for the 2013 Ore Reserve process. The 2014 trial mining was based on a 25 m level spacing as the upper levels of the mine were already developed. Stopes were initially mined 15 m along strike, with larger stopes mined based on the experience gained. The initial results have indicated that a 25 m level spacing and 20 m strike length are achievable and was the basis of the 2014 Ore Reserve. • 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 benching method together with rock fill. The South Mine will be mined using a Sub-level Open Stope mining method at 25 m level spacing but with stope strike lengths varying between 15 m and 30m, dictated by hangingwall conditions. This was used as the basis for the 2015 Ore Reserves. • Further studies are underway regarding the treatment of the Dugald River (DR) ore that may modify the results of the Ore Reserve. • 2013 Mining Method Review – recommendation for alternate mining assumptions was done and used for Ore Reserve 30 June 2013 • Business Options Review 2013 (2013BOR). The 2014 Ore Reserve process completed on 4 November 2014 incorporates the knowledge of previous studies and the mining experience gained from the 2014 trial mining campaign. • Business Options Review (2014OR) of 4 November 2014 incorporates these enhancements into earlier studies. 2014OR is the study used for 30 June 2014 Ore Reserve and Mineral Resource. • The most up-to-date study is the 2015 Updated Development Plan (31 March 2015). The processing throughput rate applied in this study is 1.5Mtpa. • The main differences between FS09, 2013OR, 2014OR and the 2015OR are; ○ Analysis of the trial mining has proved up the detailed geotechnical and underground design. The results have further enhanced the confidence in the 2015OR schedule. ○ Testing of bulk samples of ore from trial mining of the deposit is currently in progress to confirm that the design metallurgical performance can be achieved in continuous

Assessment Criteria	Commentary
	operation. ○ A plant trail of ore through the Century concentrator has also been completed, but the duration was too short to assess scale-up reliably. The primary purpose was to produce zinc concentrate for market acceptance evaluation. ○ In addition to scale-up of metallurgical performance the other risk is higher than expected reagent additions due to a recycling of ultrafine carbon in the return water. A program to mitigate this effect is currently in progress. ○ The upfront capital required to commence production is high. ○ The 2015OR has detailed revision of the capital and operating costs. ○ The 2015OR undertaken shows that the Ore Reserve is technically achievable and economically viable. The material modifying factors have been considered.
Cut-off parameters	• A$134/t NSRAR cut-off has been used for the 2015 Ore Reserves. • The commodity price and exchange rate assumptions are supplied by MMG Finance department. The majority of 2015 Ore Reserve is assessed as a long-term Ore Reserve scenario. The January 2015 guidance was used. • The cut-off was used in selecting Ore Reserves shapes.
Mining factors or assumptions	• A detailed design 2015OR was used to report Mineral Resource conversion to an Ore Reserve. • The Mineral Resource model used was the MMG June 2014 Mineral Resource model (2014 MRM). • Risks associated with the model are related to orebody complexity seen underground but not reflected in the Mineral Resource model due to the spacing of the drill holes that inform the model. • The 2015 geotechnical model was used to estimate the HW thickness, tonnes and grade of the unplanned dilution applied to the 2015 stope shapes. • The orebody is split into a north and south mine, due to its 2 km strike length and a low-grade zone in the centre of the orebody. • The north mine is narrow (average ~5 m 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 200 m from surface) and lower zone (~below 700 m from surface). The central zone is flatter and thicker than the upper and lower zones. • DR will be mined using sub-level open stopes (SLOS) in the South Mine and Bench Stoping in the North Mine. Results from the trial

Assessment Criteria	Commentary
	mining has indicated the level interval of 25 m and variable stope strike length of 15 m to 30 m is possible, although further work is required to determine if the 25 m 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 11 m wide horisontally. o Transverse SLOS, made up of 20 m strike SLOS mined full width of the ore-body. o Crown pillar SLOS, for the top level of each panel where stoping occurs directly below a previous mined area. o Bench stopes for the North Mine • The stopes were created by applying the Mineable Shape Optimiser (MSO) software in Deswik CAD to the 2014MRM (drmd0614_min.dm) that was created in Datamine. • The parameters used to create the stope shapes were: o All Mineral Resource categories included o 25 m level interval o Variable strike length o Minimum mining width (MMW) of 2.5 m o Minimum dip of 45 degrees o Minimum waste pillar between parallel stopes of 5m o A$134/t NSRAR cut-off 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). • Completed ore production from DR consists of ore development and nineteen trial stopes. A total of 461 kt of ore has been mined from DR. This consists of 322 kt of stoping and the remainder from ore development. In October 2013, approximately 93 kt ore from the trial mining at DR was batch fed through the Century mill. • Several aspects of dilution were considered, planned dilution, fill dilution, foot wall (FW) dilution and HW dilution. Planned dilution was included in the MSO stope shapes and covers localised variations in dip and strike as well as minimum mining width. FW dilution was included where ore development was wider than the stope width. No additional FW dilution was applied as the initial stope shapes took into account minimum mining widths and dip. • The HW dilution was calculated for each stope based on the geotechnical conditions and thicknesses of the HW materials. The site has compiled a detailed HW dilution model as dilution varies across the ore-body according to the HW conditions.

Assessment Criteria	**Commentary**
	• 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: ○ Floor 0.15 m and wall fill ranges from 0.3 m to 0.5 m dilution. ○ 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 Reserve. A sensitivity was done inclusive of Inferred Mineral Resources increasing the scheduled mine life from 22 years to 29 years. • The underground (UG) 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. As at 30 June 15 there is 3,475 m of decline in place. In addition there is 14,165 m of lateral development in place. • Currently three raisebored ventilation shafts are in place: ○ the southern Fresh Air Raise (FAR) – at 3.5 m diameter and 130 m depth; ○ the southern Return Air Raise (RAR) – at 5.0 m diameter and 198 m depth; ○ the northern FAR at 3.5 m diameter and 172 m depth (currently being used as RAR). • There is also a RAR longhole winze (LHW) system in the south mine – at 18.9 m square and 116 m long and a RAR longhole winze system in the north mine at 18.9 m square and 129 m long. • Two escape raises are in place: ○ south mine escape way at 1.8 m diameter and 222 m depth ○ north mine escape way at 1.8 m diameter and 93 m depth • The expected total underground development for the Ore Reserve only is summarised in the table below:

	Life-of-Mine		**Ore Reserve ONLY**	
Description	**Length**	**Tonnes/Material**	**Length**	**Tonnes/ Material**
Decline	10 km	0.9 Mt of waste	7 km	0.7 Mt of waste
Access horizontal development	39 km	3.8 Mt of waste	25 km	2.4 Mt of waste

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<table>
<tr><td>Vertical development</td><td>12 km</td><td>0.5 Mt of waste</td><td>8 km</td><td>0.3 Mt of waste</td></tr>
<tr><td>Footwall drives</td><td>30 km</td><td>2.6 Mt of waste</td><td>17 km</td><td>1.5 Mt of waste</td></tr>
<tr><td>Cross-cuts</td><td>32 km</td><td>1.9 Mt of waste</td><td>25 km</td><td>1.4 Mt of waste</td></tr>
<tr><td>Ore development</td><td>80 km</td><td>5.4 Mt of ore</td><td>45 km</td><td>2.9 Mt of ore</td></tr>
</table>

- 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.

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

- The metallurgical process proposed for treatment of Dugald 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 currently operates the Century Mine in Queensland using an almost identical process.
- The flowsheet has been extensively tested at bench scale with over 200 tests being completed on a wide range of samples. The results of these tests have been used to establish the metallurgical factors used for Ore Reserve calculations. Key parameters are:
    - Lead recovery to a lead concentrate (con) assaying 72.2% Pb according to the equation: *Pb rec* = 244.295 - (7.858 x S%) - (5.171 x Fe%) - (13.689 x Mn%) - (1.599 x SiO2%) + (0.395 x (S% x Fe%)) R^2 = 0.39
    - Silver recovery of 22.1% to a lead concentrate assaying 970 g/t Ag which is the average grade for all tests completed. Determined from results from Phase 3 samples analysed for silver, rather than from the results of the 52 samples
    - Zinc recovery of 87.2% to a concentrate assaying 51.6% Zn (rounded from the average of results for 52 variability samples using the selected flowsheet) *Zn rec* = 399.85 - 29.75 × Zn - 10.98 × Fe - 4.74 × SiO_2 + 0.99 × (Zn × Fe) + 0.47 × (Zn × SiO_2)
    - Iron assay of zinc concentrate according to the equation: *%Fe in Zn con* = 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 Con* = -3.36 + 0.36 × Zn - 0.065 × Fe + 2.780 × Mn + 1.39 × TOEC - 0.116 × (Zn × TOEC) R^2 = 0.85

where Zn%, Pb%, Fe%, S%, Mn% and C% refer to the relevant assays of the

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A full check has been completed of possible deleterious elements and the only two that are material to economic value are Fe and Mn in the Zn con. 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.
Testing of bulk samples of ore from trial mining of the deposit is currently in progress to confirm that the design metallurgical performance can be achieved in continuous operation. The results so far are given in Table 21 which also includes the expected result for comparison.

Table 21 Pilot plant results
<table>
<tr><th></th><th colspan="2">Pb Con</th><th colspan="2">Zn Con</th></tr>
<tr><th></th><th>Pb Rec</th><th>Pb Grade</th><th>Zn Rec</th><th>Zn Grade</th></tr>
<tr><td>Expected performance</td><td>69.2</td><td>70.0</td><td>87.0</td><td>51.5</td></tr>
<tr><td>Best pilot plant run</td><td>64.1</td><td>53.4</td><td>85.1</td><td>46.9</td></tr>
</table>

The best available evidence is that instabilities in circulating loads impacted performance. Changes to the recycle strategy have been developed to overcome such issues and will shortly be tested.
A plant trial of ore through the Century concentrator has also been completed, but the duration was too short to assess scale-up reliably. An objective of the trial was also to produce zinc concentrate for market acceptance evaluation. The average grade of concentrate produced was 50.3% Zn, 8.3% Fe, 3.0% SiO_2 and 1.3% Mn.
In addition to scale-up of metallurgical performance the other risk is higher than expected reagent additions due to a recycling of ultrafine carbon in the return water. A program to mitigate this effect is currently in progress.
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.
</td></tr>
<tr><td>Environmental</td><td>
Dominant vegetation comprises remnant woodlands and there are no major watercourses on the site however there are several minor ephemeral tributaries.
The Environmental Impact Statement (EIS) was submitted to Queensland Environmental Protection Agency (QEPA, now Department of Environment and Heritage Protection) in November 2010 with final Environmental Assessment (EA) approval issued in August 2011. The Plan of Operations is valid until June 2016.
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</table>

Assessment Criteria	Commentary
	• Dugald River has both a Construction Environmental Management Plan and an Environmental Management Plan; the former is still being used as the operation is still in a trial mining phase. • Environmental license conditions require DR to deposit all waste rock underground as this reduces waste onsite and decreases impact to the environment. The waste rock will be used as stope backfill.
Infrastructure	• Currently the DR project is operating using diesel generators. Northwest Queensland is not connected to the state electricity grid. Plans are to connect to the Mica Creek gas fired power station on the southern outskirts of Mount Isa. • Gas will be supplied via the Carpentaria pipeline which will require a compression station at Bellevue. Power will be transmitted to Chumvale using Ergon Energy's existing 220 kV line then to DR via an MMG owned 220 kV line. This line will be approximately 62 km 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 will be Lake Julius. Raw water will be 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, DR 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: ○ a 11 km sealed access road from the Burke Developmental Road which includes an emergency airstrip for medical and emergency evacuation use; ○ a construction camp; ○ a permanent camp; ○ Telstra communication tower ○ a temporary contractors mobile equipment facility; ○ ore and waste stockpile pads; ○ contaminated run-off water storage dams; ○ Office facilities; ○ office buildings including emergency medical facilities; ○ a core shed; ○ a fuel farm and gensets for power generation; ○ bore water fields;

Assessment Criteria	**Commentary**
	• Major infrastructure yet to be built includes: a processing plant; a tailings storage facility; a permanent mobile equipment workshop; recreational facilities; power supply lines; and raw water supply pipe line. The land for the infrastructure yet to be built has been identified and is available.
Costs	• The estimation of Capital cost for the Dugald River project were derived from first principles in the Feasibility Study and since been refined through further study work. In 2015 the costs were again estimated from first principles for the 2015 Updated Development Plan. • The mining operating costs were again estimated by AMC for the 2015 Updated Development Plan using first-principles and include a 2.5% contingency. • 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 commodity price assumptions are supplied by MMG Finance department. The Dugald River Ore Reserve applied the January 2015 guidance as this applied to study updates. • The exchange rate used was 0.82 ($A/$US), which is the January 2015 Long Term (+2018) MMG guidance and assumptions. • The road freight and logistics for domestic and export sales have been updated using the costs from the 2013 BOR report with a 3% escalation in costs. The additional costs for storing and ship loading of concentrate in Townsville are included. For the 2014 Ore Reserve 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 50% of the sea freight cost based on an agreement with Sun Metals. • Treatment and refining charges are based on MMG's estimate as no contracts are currently in place. • 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	• **Realised Revenue Factors** (Net Smelter Return after Royalty) • As part of the 2015 Ore Reserve process the net smelter return (after royalty) (NSRAR) has been revised with the latest parameters and compared against the previous 2014 NSR calculation that was used for the 2014 Ore Reserve. • The NSRAR 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.

Assessment Criteria	**Commentary**
	• Freight and logistic charges have increased by 42% due to the increase in road and rail freight costs from Dugald to Townsville when compared to the 2014OR. • Assumptions of commodity prices and exchange rates are provided by the MMG Finance department.
Market assessment	• MMG's long-term view on global consumption of metals are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. • The long-term outlook for zinc will be determined by the ability of miners to offset the impact of scheduled mine closures and growing demand. Future zinc supply will likely come from lower-grade, higher-cost underground mines as current reserves are depleted. Continued growth in the construction, transportation and infrastructure sectors especially in the developing economies, will support solid demand for zinc in the medium to long term. • Dugald River does not market any industrial minerals. • There is potential for elevated manganese levels to occur in some zinc concentrate batches. The impact of deleterious elements has been taken into account in the project economics and marketing plans.
Economic	• Economic modelling of the total mining inventory shows positive annual operating cash flows. Further cost analysis was completed in March 2015 on the mining, milling and site infrastructure. Applying the revised costs, metal prices and exchange rate (MMG January 2015 Long Term economic assumptions) returns a positive NPV. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit. • All evaluations were done on real dollars.
Social	• The nearest major population centre to the project is Cloncurry with a population of approximately 4,000 and the largest employers are mining, mining related services and grazing. • In terms of Native Title, the Kalkadoon # 4 People filed a claim in December 2005 covering an area which includes the project area, water pipeline corridor and part of the power line corridor. This claim over 40,000 square kilometres of land was granted in 2011. • MMG has concluded a project agreement with the Kalkadoon People dated 6 April 2009. Under this agreement, the claimant group for the Kalkadoon are contractually required to enter into an s31 Native Title Agreement pursuant to the Native Title Act 1993. The agreement sets out the compensation payments and MMG's obligations for training, employment and business development opportunities if/when the project is commissioned. MMG has developed an excellent working relationship with the Kalkadoon claimant group. MMG have instituted the MMG/KCPL Liaison Committee which meets at least twice yearly and addresses the

Assessment Criteria	**Commentary**
	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 are 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 Reserve is contingent.
Classification	• Ore Reserves are all reported as Probable. Due to uncertainties with a number of the modifying factors, no Proved Ore Reserves have been declared. • Only Measured (16%) and Indicated (84%) Mineral Resources have been used to inform the Probable Ore Reserves. No Inferred Mineral Resources are included in the Ore Reserves.
Audit or reviews	• No external audits have been undertaken for the 2014 Ore Reserve. MMG personnel have been involved in reviewing the Ore Reserve 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 Reserve estimate.
Discussion of relative accuracy/ confidence	• Risks that may materially change/affect the Ore Reserve. **Identified Risk**

Assessment Criteria	**Commentary**
	1) Geological understanding of the grade continuity with respect to diamond drill spacing. 2) Geotechnical risk associated with hanging-wall instability and mining dilution. 3) Mining infrastructure analysis requires further work on underground trucking, ventilation and power constraints. 4) Metallurgical risks (recovery and concentrate grades) require additional testing to confirm scale up reliability, metallurgical performance and reagent consumption. 5) 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 Reserve is based on local and global scale. • Ore Reserves accuracy and confidence that may have a material change in modifying factors are discussed above. • This Ore Reserve is based on the results of a Feasibility Study and continuous enhancements. The confidence in the estimate is based on all the information available. This data is subject to continual review and update. Data analysis continues to verify or mitigate risks that are detailed above. It is currently being discussed to upgrade the Ore Reserve – further trials and optimise the cost basis to strengthen the project combat the above factors.

8.2.1 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 22.

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 22 Contributing Experts – Dugald River Ore Reserve

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Douglas Corley, Principal Geologist MMG Ltd (Melbourne)	Geological Mineral Resources
Claire Beresford, Project Development Analyst, MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing, Sea freight and TC/RC
Geoffrey Senior, Group Metallurgy Manager, MMG Ltd (Melbourne)	Metallurgy
Mario Car, General Manager Project Delivery, MMG Ltd (Melbourne)	Project costs
Daniel Kahler, Principal Mining Engineer, AMC Consultants Pty Ltd (Brisbane)	Mining capital and operating Costs
Max Lee, Geotechnical Specialist, MMG Ltd (Melbourne)	Geotechnical
Riek Muller, Technical Services Superintendent, MMG Ltd (Dugald River)	Mining Parameters, Cut-off estimation
Anthony Allman, Director/ Mining Engineer Antcia Consulting Pty Ltd (Brisbane)	Mine Design, scheduling

8 GOLDEN GROVE UNDERGROUND OPERATIONS

8.1 Introduction and Setting

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

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

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

Figure 8-1 Golden Grove Mine location

The underground mines are operated by MMG employees and the open pit is operated by a mining contractor. Gossan Hill ore is trucked to surface and crushed at the Gossan Hill ROM pad, before being transported approximately 3 km overland by conveyor to the treatment plant at Scuddles (refer Figure 8-2). Scuddles ore undergoes primary crushing underground before being hoisted to surface.

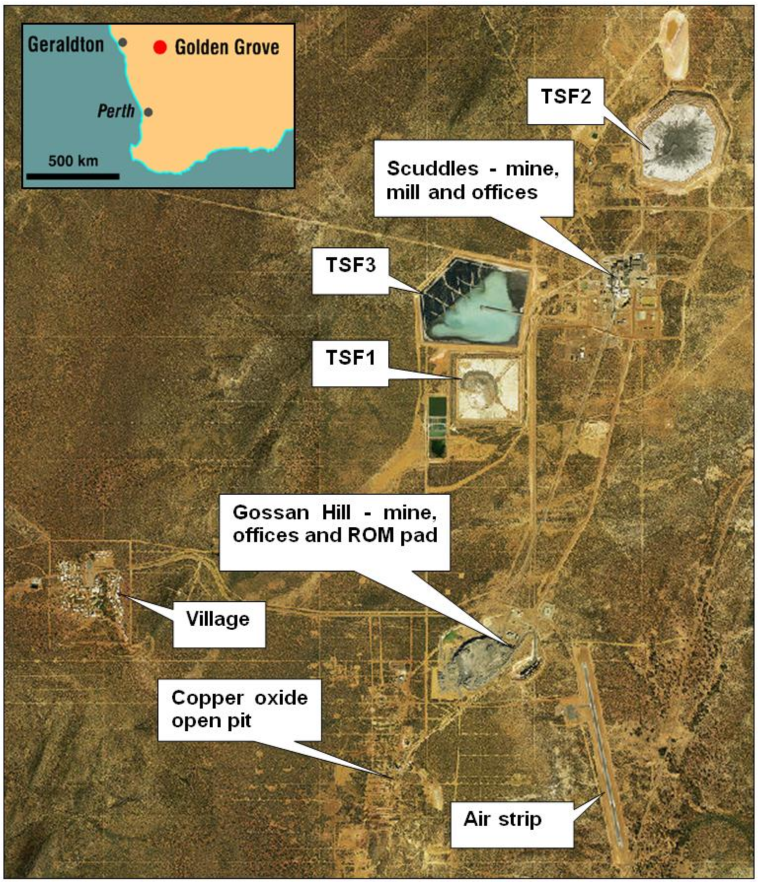
The open pit operations are an adjunct to the main underground operations of Golden Grove. The open pit operations area is located on the northern flank of Gossan Hill, directly over the current underground mining operations at Gossan Hill.

Mining is carried out using conventional truck and shovel methods using a mining contractor under the supervision of the MMG Golden Grove Open Pit department. When completed, the pit will extend over an area of approximately 15.7 ha and reach a maximum vertical depth of approximately 120 m. The mined ore will comprise oxide, transitional and primary sulphide material.

Copper oxide ore currently being mined is stockpiled on the existing run-of-mine (ROM) pad adjacent to the pit and conveyed to the Scuddles processing plant for treatment on a campaign basis, supplementing sulphide ores from Gossan Hill and Scuddles underground operations. Sulphide ores will be mined in the later parts of the operation and will also be stockpiled separately.

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

Figure 8-2 Aerial view of Golden Grove Operations showing location of key surface infrastructure



8.2 Mineral Resources – Golden Grove Underground

8.2.1 Results

The 2015 Golden Grove Mineral Resource are summarised in Table 23. The Golden Grove Mineral Resource is inclusive of the Ore Reserve.

Table 23 2015 Golden Grove Underground Mineral Resource tonnage and grade (as at 30 June 2015)

Golden Grove Underground Mineral Resource											
							Contained Metal				
	Tonnes (Mt)	Copper (%)	Zinc (%)	Lead (%)	Silver (g/t)	Gold (g/t)	**Copper ('000)**	**Zinc ('000)**	**Lead ('000)**	**Silver (Moz)**	**Gold (Moz)**
Gossan Hill											
Primary Copper											
Measured	3.2	2.7	4.4	0.6	46	2.0	87	140	19	4.7	0.2
Indicated	1.0	2.6	3.7	0.4	43	2.1	26	36	4	1.3	0.1
Inferred	6.3	3.5	0.7	0.0	27	0.1	222	47	2	5.4	0.0
Total	**10.5**	**3.2**	**2.1**	**0.2**	**34**	**0.9**	**335**	**224**	**25**	**11.5**	**0.3**
Primary Zinc											
Measured	1.8	0.6	9.9	1.3	82	2.0	12	182	24	4.8	0.1
Indicated	1.5	0.3	11.0	1.6	111	1.6	5	167	24	5.4	0.1
Inferred	1.6	0.7	13.2	0.7	71	0.6	12	205	10	3.6	0.0
Total	**4.9**	**0.6**	**11.3**	**1.2**	**87**	**1.4**	**28**	**554**	**58**	**13.8**	**0.2**
Gossan Hill Total							**363**	**777**	**83**	**25.2**	**0.5**
Scuddles											
Primary Copper											
Measured	3.0	3.0	0.7	0.1	20	0.6	89	22	2	1.9	0.1
Indicated	0.8	3.2	0.4	0.0	14	0.3	27	3	0	0.4	0.0
Inferred	0.8	2.5	0.3	0.0	19	0.3	21	2	0	0.5	0.0
Total	**4.6**	**3.0**	**0.6**	**0.0**	**19**	**0.5**	**136**	**27**	**2**	**2.8**	**0.1**
Primary Zinc											
Measured	0.8	0.3	14.4	1.2	104	0.9	3	119	10	2.8	0.0
Indicated	0.1	0.2	11.7	1.1	86	1.0	0	14	1	0.3	0.0
Inferred	0.5	0.7	13.9	0.9	41	0.9	4	71	5	0.7	0.0
Total	**1.5**	**0.5**	**14.0**	**1.1**	**80**	**0.9**	**7**	**205**	**16**	**3.8**	**0.0**
Scuddles Total							**143**	**232**	**18**	**6.6**	**0.1**
Gossan Valley											
Primary Copper											
Measured											
Indicated											
Inferred	1.3	2.9	0.6	0.0	24	0.6	36	8	0	1.0	0.02
Total	**1.3**	**2.9**	**0.6**	**0.0**	**24**	**0.6**	**36**	**8**	**0**	**1.0**	**0.02**
Primary Zinc											
Measured											
Indicated											
Inferred	1.6	0.1	13.9	0.2	8	0.4	2	221	3	0.4	0.02
Total	**1.6**	**0.1**	**13.9**	**0.2**	**8**	**0.4**	**2**	**221**	**3**	**0.4**	**0.02**
Gossan Valley Total							**38**	**229**	**3**	**1.4**	**0.04**
Underground Total							**544**	**1238**	**104**	**31.8**	**0.6**
Surface Stockpiles											
Primary Copper											
Measured	0.1	3.0	0.1	0.1	0.1	30	1.8	0.1	0.1	0.06	0.1
Total	**0.1**	**3.0**	**0.1**	**0.1**	**0.1**	**30**	**1.8**	**0.1**	**0.1**	**0.06**	**0.1**
Primary Zinc											
Measured	0.01	0.5	1.9	9.9	2	120	0.0	0.2	1.0	0.04	0.0
Total	**0.01**	**0.5**	**1.9**	**9.9**	**1.5**	**120**	**0.0**	**0.2**	**1.0**	**0.04**	**0.0**
Surface Stockpile Total							**2**	**0**	**1.1**	**0.10**	**0.1**
Total Contained Metal							**546**	**1,238**	**105**	**33.2**	**0.8**

Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value A$145/t (Gossan Hill), A$145/t (Scuddles).

Contained metal does not imply recoverable metal.

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

8.3 Mineral Resources – Golden Grove Open Pit

8.3.1 Results

The 2015 Golden Grove Open Pit Mineral Resource are summarised in Table 24. The Golden Grove Mineral Resource is inclusive of the Ore Reserves.

Table 24 2015 Golden Grove Open Pit Mineral Resource tonnage and grade (as at 30 June 2015) (Gossan Hill)

Golden Grove Open Pit Mineral Resource (Gossan Hill)											
							Contained Metal				
	Tonnes (Mt)	Copper (%)	Zinc (%)	Lead (%)	Silver (g/t)	Gold (g/t)	**Copper ('000)**	**Zinc ('000)**	**Lead ('000)**	**Silver (Moz)**	**Gold (Moz)**
Gossan Hill Copper Pit[1]											
Partial Oxide Copper											
Measured											
Indicated	0.2	2.1					4				
Inferred	0.0	2.1					0.1				
Total	**0.2**	**2.1**					**4**				
Primary Copper											
Measured											
Indicated	0.1	1.9					1				
Inferred	0.0	2.2					0.1				
Total	**0.1**	**1.9**					**1**				
Gossan Hill Gold Pit											
Oxide Gold[2]											
Measured											
Indicated	0.6				89	3.2				1.7	0.1
Inferred	0.0				55	2.8				0.1	0.0
Total	**0.6**				**87**	**3.2**				**1.8**	**0.07**
Partial Oxide Gold[2]											
Measured											
Indicated	0.1				130	2.6				0.3	0.01
Inferred	0.01				71	2.0				0.0	0.0
Total	**0.1**				**123**	**2.5**				**0.3**	**0.01**
Primary Zinc[3]											
Measured											
Indicated	0.3	0.3	10.5	1.0	105	1.3	1	35	4	1.1	0.0
Inferred	0.1	0.1	9.9	0.4	83	1.9	0	6	0	0.2	0.0
Total	**0.4**	**0.3**	**10.4**	**0.9**	**102**	**1.4**	**1**	**42**	**4**	**1.3**	**0.02**
Primary Gold[2]											
Measured											
Indicated	0.1				54	2.2				0.1	0.0
Inferred	0.0				49	2.1				0.0	0.0
Total	**0.1**				**53**	**2.2**				**0.1**	**0.0**
Total Contained Metal							**7**	**42**	**4**	**4**	**0**

1 1% Cu cut-off grade contained in US$3.3/lb pit-shell.

2 1.1g/t Au cut-off grade.

3 3% Zn cut-off grade.

Contained metal does not imply recoverable metal.

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

8.4 Mineral Resources 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 1-3" of the Code.

Table 25 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Golden Grove Underground and Open Pit Mineral Resource 2015

Criteria	Status
	Section 1 Sampling Techniques and Data
Sampling techniques	• Samples have been collected by reverse circulation (RC) and diamond drilling (DD), both from surface and underground. Sample length ranges from 0.5 m to 1.0 m of half core. Sample intervals do not cross geological boundaries; this ensures samples were representative of the lithological unit without mixing of grade at lithological boundaries. There is no current database limit for shortest sample interval; however Geologists are advised to not sample intervals shorter than 0.5 m. • Current sampling practice is to collect half-core samples from NQ2 or LTK60 diamond drilling, which is crushed and the entire sample pulverised to 85% passing 75 µm. • Historical underground sampling practices are comparable with the current practice, the only difference being primary core diameter for the underground drilling. • During surface Aircore and RC drilling before 1994, samples were captured in a bag attached to the cyclone. These samples were then split using a 40 mm or 50mm PVC pipe spear. • Post 1994 surface RC samples were captured in a bag attached to the cyclone and subsequently split using a triple stage riffle splitter. • Current grade control RC drilling involves taking a 2 m sample, captured in the cyclone with subsequent cone splitting. • Measures taken to ensure sample representivity include the collection, and analysis of field and coarse crush duplicates.
Drilling techniques	• DD core and minor RC data was used in the Mineral Resource estimation for Gossan Hill, Scuddles and Gossan Valley. ○ 7,584 drill holes used in the Gossan Hill Mineral Resource model. ○ 3,546 drill holes used in the Scuddles Mineral Resource model. ○ 361 drill holes used in the Gossan Valley Mineral Resource model. ○ 1,645 drill holes were used in the Open Pit Mineral Resources (comprised of 77 Aircore, 162 DD Core and 930 RC holes). • The SQL database contains 25,174 drill holes, totaling 2,601,455 m, consisting of 45% diamond core, 43% RAB drill holes, 11% RC holes, minor air core holes and water bores. • The Reflex Act II™ tool is used for core orientation marks on selected DD holes.

Drill sample recovery	• Surface and underground recoveries of DD core are recorded as percentages calculated from measured core versus drilled metres. The intervals are logged and recorded in the database. Average core recovery was greater than 99.5%. • Drilling process was controlled by the drill crew and geological supervision provides a means for maximising sample recovery and ensures suitable core presentation. Drilled core is reconstructed into a continuous run on an angled iron cradle for orientation marking. Depth is checked against depth provided on core blocks. No other measures are taken to maximise core recovery. • No RC drill holes drilled before 2000 have recovery data except for the 1994 RC program. Recovery data is not used in the Mineral Resource estimation. • Preferential loss/gains of fine or coarse materials are not considered significant. There is no known relationship bias between recovery and grades.
Logging	• All drill core and chips are logged geologically using codes set up for direct computer input into the Micromine Geobank™ database software package. All DD cores are geotechnically logged to record recovery, RQD, roughness, fill material. Structural logging is recorded for all oriented core. • Logging is both qualitative and quantitative (percentage of sulphide minerals present). • All drill holes are logged in full detail from start to finish using laptop computers directly into the drill hole database. 1,441,032.3 m of logged drillcore was used in the Mineral Resource, of which 691,729 m was sampled.

Sub-sampling techniques and sample preparation	• All DD core is half-cut onsite using an automatic core saw with samples always taken from the same side. Half core is used for routine sampling and quarter core for field duplicates. Current sample length ranges between 0.5 m and 1 m (historically this can have been from 0.2 m to 1.5 m) and is adjusted to geological boundaries. Historic DD core has been sampled using whole, half, quarter and third core.
	• RC drilled samples have been cone split and dry sampled. Wet sampling only conducted when drill holes intersected the water table. All routine and duplicate samples were 2 m composites. Historical RAB, AC and RC drilling has been sampled using spear, grab, riffle and other unknown methods but none of these were used in the Mineral Resource estimation.
	• The sample preparation of RC chips and DD core adheres to industry best practice. A commercial laboratory is used which involves:
	  ○ Weighing
	  ○ Oven drying at 90° C
	  ○ Coarse crushing to 6 mm
	  ○ Samples > 3 kg crushed to 2 mm and split using a rotary splitter (this represent < 0.01% of total sample used for

	Mineral Resource estimation). - Pulverising in an LM5 to a grind size of 85% passing 75 µm. - Collection of 400 g pulp from each sample; rejects kept or discarded depending on drilling programme. • Quality control procedures for all sub-sampling stages to maximise representivity include: - Duplicates are taken after coarse crushing and pulverisation at a rate of 1:20 alternating between the two. These are subject to the same assay process as the routine samples. - Sizing tests ensure the grind size of 85% passing 75 µm is achieved. • Measures taken to ensure sampling is representative of the insitu material collected include: - RC field duplicate sampling is carried out at a rate of 1:50 taken directly from the on-board cone splitter at the same time as the routine sample. These are subject to the same assay process as the routine samples and the laboratory is unaware of such submissions. - Historically duplicate DD were taken from core at a rate of 1:50 and the half core was cut into quarter core. This practiced ceased in July 2014, and was replaced by taking duplicates after the coarse crushing stage. - Although field duplicates showed good reproducibility across the grade range for Cu, Zn and Au, their use was ceased in 2014 after consultation with the Principle Resource Geologist and Technical Services Manager regarding their collection method and application as a true duplicate. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Golden Grove mineralisation (volcanic hosted massive sulphide) by the Competent Person.
Quality of assay data and laboratory tests	• A four acid "near-total" digestion is used to determine concentrations for silver, copper, iron, lead, sulphur and zinc. This method underwent a change in October 2014 after extensive test work was conducted. Previously it used a 0.4 g sample in a HF-HNO3-HClO4 digestion, with HCl leach and finished using ICP-AES. Since October 2014 the sample charge weight has been changed to 0.2 g with no other changes to the method. This is an ore grade method suitable for use in VHMS deposits. • For gold, prior to October 2014 a 30 g fire assay with AAS finish was used to determine the gold concentration for RC chips and DD core samples. As the precision of AAS was limited to 20 times the detection limit which coincided with the value at which gold was deemed significant determination since October 2014 is by ICP-AES. Grades above 10g/t are determined using AAS. Historic analysis includes fire assay, aqua regia, four acid digest and AAS or ICP. • No geophysical tools, spectrometers or handheld XRF instruments

	have been used in the analysis of samples external to the laboratory for the estimation of Mineral Resources. • Quality control procedures adopted include: o Matrix matched Certified Reference Materials (sourced from Golden Grove and prepared by Ore Research Pty. Ltd.) with a wide range of values are inserted at a rate of 1:25 into every RC and DD batch to assess laboratory accuracy. o A certified blank is inserted at a rate of 1:50 to identify any contamination issues. o Duplicates are taken after coarse crushing and pulverisation at a rate of 1:20 alternating between the two to assess laboratory precision. Control samples are not identified to the laboratory. • QAQC data returned are checked against pass/fail limits once the results have been loaded into the database. QAQC data is reported monthly and demonstrates sufficient levels of accuracy and precision. • The laboratory performs internal QC including standards, blanks, repeats and checks. • Oxide grade control analysis: o Certified Reference Materials have been used in most programs. o Base metals assay method: four acid digest followed by ICP MA-ICPOES for the first program with XRF applied for subsequent programs. Checks showed no bias between analysis methods. o Gold and silver assay method: fire assay, AAS FA-AAS.
Verification of sampling and assaying	• Significant intersections are reviewed by a senior geologist and other site geologists. Where there is a significant intersection, in the oxide zones specifically, holes have either been twinned or scissored. • No specific twinned holes have been drilled at Golden Grove for the underground sulphide deposit. However nearby and scissor drill holes show compatible geology and results. A program of twinned holes was drilled for the Gossan Hill Copper Oxide deposit to check correlation with historic data. Good correlation was established. A full report of these twinned holes was written. • All underground logging, sample and geotechnical data is captured using Panasonic Toughbook™ computers and entered directly into the secure Geobank database via wireless network. The database has inbuilt validation functions plus additional triggers to prevent incorrect data capture and importation. Exploration DD are graphically logged on paper before entry into the database. All paper logs are scanned to pdf and hardcopies kept in labelled folders. Periodic review is undertaken to ensure data has been correctly transcribed. • Assay data is retained in text files (.SIF) and stored once loaded into the database. All re-assayed data will replace original results that failed QAQC; both results are retained in the database, with the results that failed QC being excluded from general use and export.

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During the 1990's the database was in Explorer III, a Microsoft Access™ based application. In 2008 the data was migrated to a Micromine Geobank™ database. Validation of data has been undertaken during each migration and is periodically reviewed against hardcopy records
All assay data remains in its original state and has not been adjusted.

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<td>Location of data points</td>
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All underground drill hole collars are picked up by MMG surveyors using a Leica TS-15 (total station) with an expected accuracy of 10 mm. Surface exploration drill hole collars are picked up by company surveyor using a Trimble RTK R8 GPS with an expected accuracy of 40 mm. All drill holes are down hole surveyed gyroscopically by the drilling companies (currently DDH1 and Swick Mining Services) once each drill hole is completed. Eastman single shot camera surveys are taken every 30 m downhole during DD. The accuracy and quality of historic surveys is generally unknown.
A local grid system (GGMINE) is used. It is rotated 52.4 degrees west of MGA94 zone 50. The two point conversion is as follows:

<p>Mine Grid to MGA94 Two-Point Convertion</p>
<table>
<tr><th>Point</th><th>GGMINE East</th><th>GGMINE North</th><th>MGA East</th><th>MGA North</th></tr>
<tr><td>1</td><td>3644.47</td><td>10108.13</td><td>502093.5</td><td>6810260.7</td></tr>
<tr><td>2</td><td>9343.2</td><td>29162.02</td><td>490480.1</td><td>6826394.2</td></tr>
</table>

All historic data is in the GGMINE local mine grid.
Topographic measurement on Mining Leases is by GPS using surface control points with an accuracy of 10 mm. Topographic control on most of the Exploration Leases is by 1 m contours generated from aerial photography.

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<tr>
<td>Data spacing and distribution</td>
<td>

Drill data spacing ranges from less than 10 m x 10 m in the active mining areas to greater 80 m x 80 m in exploration areas.
Data spacing is sufficient to establish geological and grade continuity for the appropriate estimation and classification of Mineral Resources and Ore Reserves. Underground drive mapping below the surface deposits supports understanding of geological structure and strike continuity
DD samples are not composited prior to being sent to the laboratory. Sample lengths range from 0.5 m to 1.2 m.
Current gold pit RC grade control drilling is sampled on 1 m intervals. Current copper pit RC grade control drilling is sampled on 2 m intervals. Past RC samples (gold and copper) up to 5 m has occurred.

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<tr>
<td>Orientation of data in relation to geological structure</td>
<td>

Drilling has mostly been oriented on sections that are orthogonal to the strike of mineralisation. Drill holes frequently overlap and are scissored as drilling is oriented from both footwall and hanging-wall directions.
No significant sampling bias has been recognised due to orientation of

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</table>

	the drilling in regards to mineralised structures.
Sample security	• Measures to provide sample security included: o Adequately trained and supervised sampling personnel. o Half-core samples placed in a numbered and tied calico sample bags. o Bag and sample numbers are entered into Geobank database. o Samples are couriered to assay laboratory via truck in plastic bulker containers. o Assay laboratory checks off sample dispatch numbers against submission documents and reports any inconsistencies. o Remaining DD core is stored within the Golden Grove core yard.
Audits or reviews	• Regular auditing of the external lab has been performed in the last 12 months. Regular laboratory audits have been completed by the Geological Database Administrator with support from Resource, Senior Mine, and Mine Geologists. No material issues have been identified. The most recent laboratory audit was conducted on 2nd June, 2015 while the previous one was conducted on 28th July 2014. • An internal review of RC and DD core sampling procedures were completed in 2014. The sampling procedures were found to meet industry standards. • In 2012 Optiro completed a review of the Gossan Hill Gold Oxide data. The review found there was no historic QAQC data (1990 to 2000) around Gossan Hill. This has now been rectified.

Section 2 Reporting of Exploration Results

Mineral tenement and land tenure status	The mineral tenement and land tenure status of the Golden Grove operations are listed in the below table.

Mineral tenement and land tenure status for Golden Grove operations

Tenement No.	Prospect Name	Date Expires	Term (Years)	Date Granted
M59/03	Scuddles	08/12/2025	21	28/01/2005*
M59/88	Chellews	18/05/2030	21	20/04/2009*
M59/89	Coorinja	18/05/2030	21	20/04/2009*
M59/90	Cattle Well	18/05/2030	21	20/04/2009*
M59/91	Cullens	18/05/2030	21	20/04/2009*
M59/92	Felix	18/05/2030	21	20/04/2009*
M59/93	Flying Hi	18/05/2030	21	20/04/2009*
M59/94	Bassendean	18/05/2030	21	20/04/2009*

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<tr><td></td><td>
<table>
<tr><td>M59/95</td><td>Thundelarra</td><td>18/05/2030</td><td>21</td><td>20/04/2009*</td></tr>
<tr><td>M59/143</td><td>Bassendean</td><td>09/05/2031</td><td>21</td><td>21/04/2009*</td></tr>
<tr><td>M59/195</td><td>Gossan Hill</td><td>17/05/2032</td><td>21</td><td>17/06/2011*</td></tr>
<tr><td>M59/227</td><td>Crescent</td><td>07/05/2033</td><td>21</td><td>08/05/2012*</td></tr>
<tr><td>M59/361</td><td>Badja</td><td>01/03/2016</td><td>21</td><td>02/03/1995</td></tr>
<tr><td>M59/362</td><td>Badja</td><td>01/03/2016</td><td>21</td><td>02/03/1995</td></tr>
<tr><td>M59/363</td><td>Badja</td><td>01/03/2016</td><td>21</td><td>02/03/1995</td></tr>
<tr><td>M59/543</td><td>Walgardy</td><td>04/02/2023</td><td>21</td><td>05/02/2002</td></tr>
<tr><td>M59/480</td><td>Marloo</td><td>01/07/2029</td><td>21</td><td>02/07/2008</td></tr>
</table>
* Renewal date
• There are no known impediments to operating in the area but the operation is subjected to environmental conditions pertaining to land and water management, as well as adherence to cultural sensitivity pertaining to the local indigenous people.
• All tenements are 100% owned by MMG-Golden Grove.
</td></tr>
<tr><td>Exploration done by other parties</td><td>
• Original definition and exploration drilling was performed by Joshua Pitt, of Aztec Exploration in 1971.
• From 1971 until 1992 multiple joint ventures continued the definition of the Mineral Resource, with highlights being the Scuddles, A Panel Zn, B Panel Zn, C Panel Zn and Cu discoveries. Parties involved included Amax Exploration, Esso Exploration, Australian Consolidated Minerals and Exxon.
• Newmont, Normandy, Oxiana, OZ Minerals and MMG have all been involved with the drilling and exploration of the Golden Grove leases since 1991.
• The exploration and resource geology groups remained unchanged throughout the OZ Minerals and MMG takeovers; hence the exploration management and methods have effectively remained constant since Oxiana acquired the project in 2005.
</td></tr>
<tr><td>Geology</td><td>
• The mineralisation style is volcanogenic hosted massive sulphide (VHMS) which occurs as sub-vertical lenses within layered sediments and volcanics.
• The Golden Grove deposits are located in the Murchison Province in the North-Western part of the Achaean Yilgarn Craton in Western Australia within the Yalgoo Greenstone Belt. Mineralisation occurs at the base of the Warriedar Fold Belt (“WFB”) within a sequence of felsic to intermediate volcaniclastic sediments, lavas and associated autoclastic breccias.
• The Golden Grove Domain that hosts the Gossan Hill and Scuddles deposits lies along the northeast flank of the WFB. The Mougooderra Fault (west), recrystallised monzogranite (east) and post folding granites (north and south) bound the domain. The
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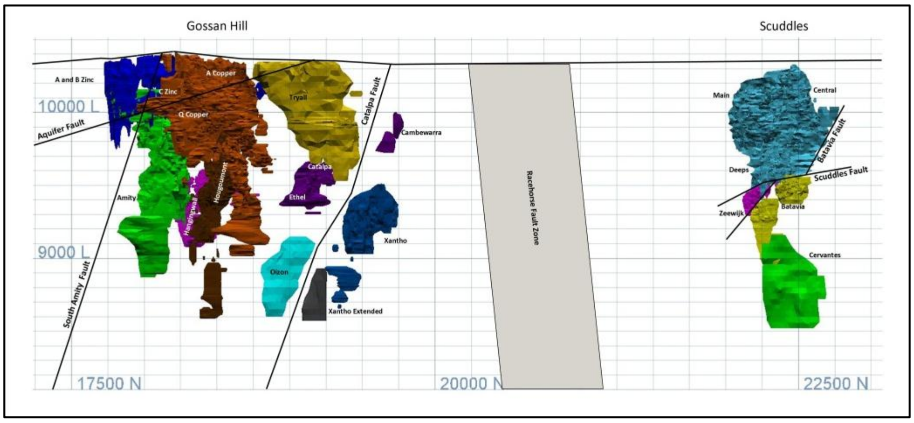
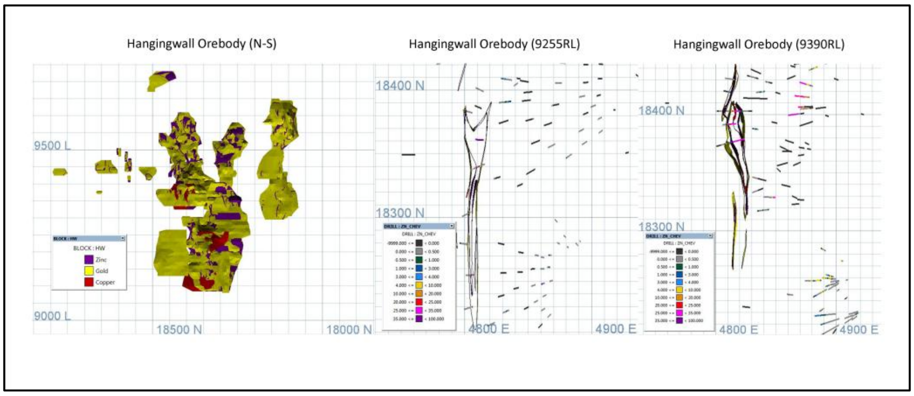
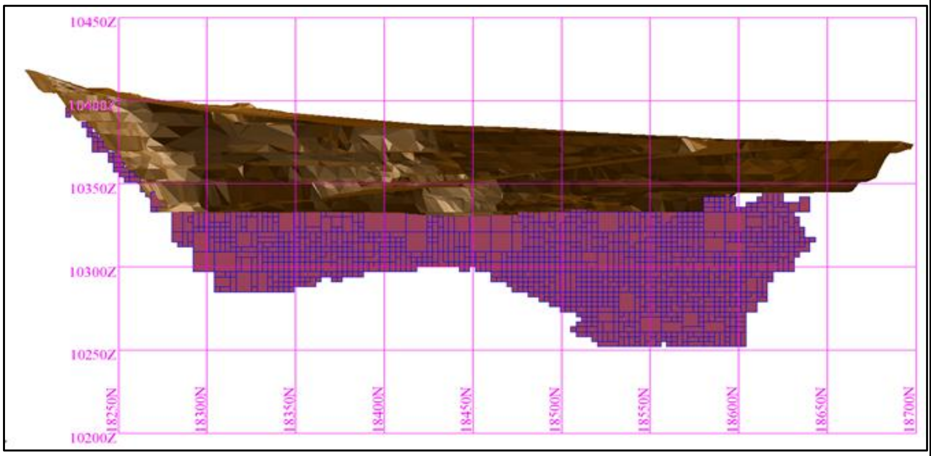
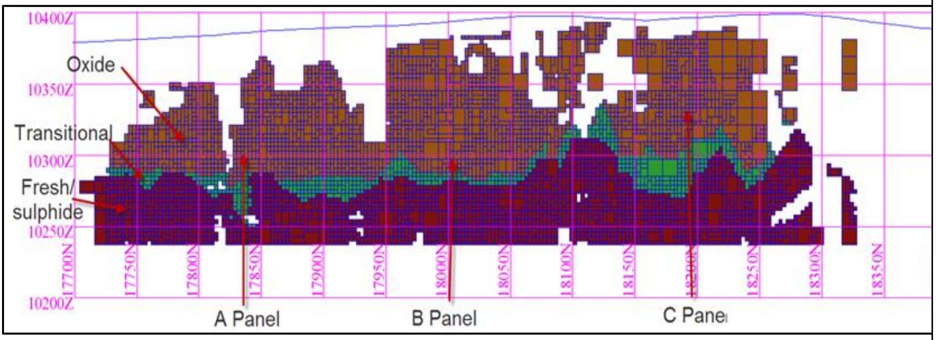
</table>

	current interpretation of the structure places the Golden Grove Domain on the eastern limb of a syncline. The stratigraphy has a westerly younging direction and dips steeply west.
Drill hole information	• Over 25,174 drill holes and associated data are held in the database. No individual drill hole 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 widths and intercept lengths	• Exploration drill holes have been excluded from the Mineral Resource estimate. The drilling results are excluded from this report as they are not considered Material by the Competent Person. • Mineralisation true widths are captured by interpreted mineralisation 3D wireframes. Drill holes are drilled to achieve intersections as close to orthogonal as possible.
Diagrams	 **Long-section of the Golden Grove mining area: Gossan Hill and Scuddles**  **Long section and representative cross-sections of Hangingwall orebody**
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	• This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for

exploration data	this section.
Further work	- Exploration and delineation drilling will continue underground and the results will be modelled and reported in subsequent Mineral Resource estimates. - Surface exploration activities including RC and DD drilling will continue on the Exploration and Mining leases.
Section 3 Estimating and Reporting of Mineral Resources	
Database integrity	- The following measures are in place to ensure database integrity: o Golden Grove uses an SQL database system. o Data are logged directly into Micromine Geobank™ (front-end software) using wireless transfer protocols on Panasonic Toughbook™ portable computers. A limited number of primary tables have read/write privileges to the geologist and geotechnicians. User profiles restrict the data that any individual can access and alter. o The database is backed up each night with hourly log backups during the day. Data backups from the previous seven days are stored on the database server. Data older than seven days is backed up onto tape and stored securely. o Assays are imported electronically from files (sif) received from the laboratory. o Drill holes are checked and locked from users modifying data whenever assays are received. The measures described above ensure transcription or data entry errors are minimised. - Data validation procedures include: o Data is validated on-entry using library of codes and key fields which ensure intervals cannot duplicate or overlap. o Collar co-ordinates and drilling direction (azimuth and dip) are validated via comparison of planned data to surveyed data. o Deviations of more than 1 degree over 30 m of drill hole depth are flagged and evaluated for re-drilling. All data attributed to a given drill hole undergoes final validation and sign-off procedure. Any errors found are rectified prior to releasing the data for Mineral Resource estimation.
Site visits	- The Competent Person is a full time employee of the Golden Grove Mining Operation and as such is regularly present at the site, is fully conversant with the geology and has a detailed understanding of the sample collection process, modelling process and mining methods employed at Golden Grove.

Geological interpretation	• Geological interpretation of the mineral deposits and associated lithologies is considered to be moderate. • Data used for the geological interpretation included geological mapping of development drives, assay results and geological logging of all DD holes. • Alternate structural and geological interpretations are routinely considered and tested with diamond drilling. • Geological interpretations have been modelled as three-dimensional wireframes of mineralisation and other lithologies, which have been used to construct block models and to control grade estimation as hard boundaries. • Primary sulphide interpretation: o Zinc-rich mineralisation occurs as massive to semi-massive sulphide lenses. These lenses also contain moderate copper, lead, silver and gold mineralisation. o Copper-rich mineralised lenses are composed of zones of chalcopyrite-rich stringers within quartz-rich domains. These domains can have moderate grades of gold and silver but are weakly mineralised with zinc and lead. o Zinc and copper lenses are each surrounded by low-grade mineralisation haloes. Low-grade domains have been constructed for some of the deposits. o Intrusive rocks and faults have been interpreted that cut-across and displace mineralisation and stratigraphy. o Lithological codes obtained from the logging of drill holes aids in establishing continuity of geology. In modelling the mineralised domains this way, it is assumed that there is geological continuity and stationarity within each respective domain. o Wireframes have been constructed from interpreted polygons snapping to drill hole intersections on 10 m spaced plan sections, though sections are shortened or lengthened appropriately with clustering of data. Interpretations account for all available geological information. • Oxide copper interpretation: o Oxide copper mineralisation generally occurs as mineralised zones that cut across stratigraphy. Interpretation was undertaken using all drilling and mapping on east-west sections. Transitional and fresh (primary sulphide) ore is constrained by lithology however; mineralised lenses can tend to bifurcate in places. o The stratigraphy dips steeply to the west (mine grid). o A copper domain boundary of 0.2% Cu was selected as the by observing the distribution of sample data in 3D and

	consideration of geology. The 0.2% Cu shape maintains a consistent mineralisation shape after considering the geology and assay data. o Wireframes have been constructed from interpreted polygons on 10 m spaced plan sections. Interpretations account for all available geological information. • Oxide gold, silver and zinc interpretation: o Mineralisation occurs as steep westerly dipping stratabound lenses that have been modelled separately based on the following boundary grades: ▪ Gold: 0.3g/t Au ▪ Silver: 15g/t Ag ▪ Zinc: 0.1% Zn o The basis for each of the above domain boundaries were selected by observing the distribution of sample data in 3D and consideration of geology. These domains maintain a consistent mineralisation shape after considering the geology and assay data. o Wireframes have been constructed from interpreted polygons on 20 m spaced plan sections. Interpretations account for all available geological information.
Dimensions	• The primary sulphide mineralisation at Gossan Hill and Scuddles comprises multiple steeply dipping zones. Each zone varies from 200 m to 400 m along strike, 200 m to 700 m down-dip and 3 m to 20 m in thickness. The current Mineral Resource is located from 200 m to 1,500 m below surface. • The oxide copper mineralisation consists of two lenses with a strike length of 100 m, joined by a 100 m long low grade zone. Depth is constrained by a large flat dolerite and underground production stoping beneath the pit.  **Gossan Hill copper oxide pit, elevation looking west** • Oxide gold mineralisation is approximately 600 m long and was reported above the 10200 mRL.

	 **Gossan Hill Ag, Au, Zn oxide resource, elevation looking west**
Estimation and modelling techniques	• Mineral Resource estimation for the primary sulphide Mineral Resource was 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, Zn, Pb, Ag, Au, Fe and density. This is considered appropriate for the estimation of Mineral Resources at Golden Grove. 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 Non-sampled intervals in drill holes have been flagged with values of -99 in the primary database, which are then assigned detection limit values for grade interpolation in waste areas. This is undertaken to ensure that any sampled and mineralised grades in these domains are not over-represented in the estimate. o Copper and zinc domains were modelled as separate three-dimensional wireframes and used as hard boundaries in the estimate. Cross-cutting intrusive dykes are barren and have been modelled as such. o Data compositing for estimation was set to 1 m, which matches the majority of drill hole sample lengths underground and provides good definition across interpreted domains. o Variogram analysis was reviewed and updated for new interpretations and for existing domains materially affected by new drill data. Variogram analysis was undertaken in Supervisor (Snowden) software. o No assumptions have been made about the correlation between variables. All variables are comparably informed and independently estimated. o Interpolation was undertaken in four passes. o Discretisation was set to 4 x 4 x 4. • Check estimates using Discrete Gaussian Modelling have been performed on all models. Block model results are comparable with

previous Mineral Resource estimations after depletion and additions due to drilling and re-modelling.

- Reconciliation of block model against mill production for stoped volumes in 2014/2015 shows tonnes have been underestimated (16% and 17% for copper and zinc stope blocks respectively) as has grade (9% and 1% for copper and zinc stoping blocks respectively). Further work is planned to improve reconciliation at Golden Grove.
- Assumptions about the recovery of by-products is accounted for in the net-smelter return after royalty (NRSAR) calculation which includes the recovery of Cu, Zn, Pb, Ag and Au along with the standard payable terms.
- Iron has been estimated as it is related to the recovery of payable elements. Sulphur is not estimated in the underground Mineral Resources. However, underground waste material is used to back fill mined stopes or treated as potential acid forming (PAF) material when moved to the surface.
- The block size ranges from 20 m (x) x 50 m (y) x 50 m (z) in the waste domains down to 2 m (x) x 10 m (y) x 10 m (z) (with 1 m (x) x 5 m (y) x 5 m (z) sub-cells) in well drilled areas where drilling has been undertaken on a 10 m x 10 m pattern with samples taken on 1 m intervals.
- The selective mining unit size is based on the smallest regular stope shape of 3 m x 10 m x 15 m. No other selective mining unit size assumptions were made in the estimation process.
- The block models and estimate has been validated in the following ways:
  - Visual checking of block model interpolated grades against the input drilling data on a section by section basis.
  - Comparison of block model and sample statistics.
  - Drift plots comparing block model grades against input samples by easting, northing and RL.
  - Reconciliation data as described above.
- Block modelling for the oxide Mineral Resource has been undertaken in Surpac Gemcom software with the following key assumptions and parameters:
  - Ordinary Kriging interpolation has been applied for the estimation of Cu, Zn, Pb, Ag and Au. This is considered appropriate for the estimation of Mineral Resources at Golden Grove.
  - 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.
  - Copper and zinc domains were modelled as separate three-dimensional wireframes and used as hard boundaries in the estimate.

  - Data compositing for estimation was set to 2 m for the Gossan Copper Pit and 1 m for the gold pit. These match the majority of drill hole sample lengths and provide good definition across interpreted domains.
  - Variogram analysis was reviewed and updated for new interpretations and for existing domains materially affected by new drill data. Variogram analysis was undertaken in Supervisor (Snowden) software.
  - No assumptions have been made about the correlation between variables. All variables are comparably informed and independently estimated.
  - Interpolation was undertaken in three passes.
  - Discritisation was set to 3 x 3 x 3.
- Alternate check estimates have not been undertaken. However results are comparable with previous Mineral Resource estimations for the site.
- The gold oxide Mineral Resource has not been reconciled. Reconciliation of the copper oxide block model against mill production for mined bench volumes in 2014 shows tonnes have been underestimated (2% for copper oxide) as has grade (5% for copper oxide).
- There have been no assumptions made regarding the recovery of by-products.
- For the gold oxide material, copper and silver has been identified as deleterious for Carbon in Pulp (CIP) gold extraction. Material with more than 0.2% Cu and 50g/t Ag is separately stockpiled. For copper oxide material, chlorine has been identified to be the deleterious and is estimated in the block models. Iron has been estimated as it is related to the recovery of payable elements. Sulphur is modelled within a 0.2% sulphur domain for environmental considerations. No other deleterious or ancillary elements have been modelled.
- The parent block size for the gold oxide model block size is 6 m x 10 m x 10 m with 1.5 m x 3 m x 3 m sub-cells. The parent block size for the copper oxide is 6 m x 10 m x 10 m with 1.5 m x 2.5 m x 2.5 m sub-cells.
- The current mining fleet is composed of a Komatsu PC1250 digger and 100t trucks. Mining is carried out on 3 m high benches. The selective mining unit size is based on the smallest regular digger bucket dig of 2.5 m x 2.5 m x 3 m. No other selective mining unit size assumptions were made in the estimation process.
- The mineralisation domains do not cut across major stratigraphic units i.e. mineralisation domains do not cut-over from GG6 to GG5. Within the oxide zone, mineralisation domains has been modelled into intrusive units i.e. dolerites and dacites. This relationship has been validated in the field.
- The block models and estimate has been validated in the following ways:

	○ Visual checking of block model estimated grades against the input drilling data. ○ Comparison of block model statistics against sample statistics. ○ Swath plots comparing average block model estimated grades against input samples by easting, northing and RL.
Moisture	• All tonnages have been estimated on a dry basis.
Cut-off parameters	• Primary sulphide Mineral Resources were reported above a cut-off Net Smelter Return after Royalties (NSRAR) dollar value. The Golden Grove primary sulphide Mineral Resources were reported above an A$145/t block grade cut-off. • A minimum width of mineralisation of 3 m is applied to ensure narrow mineralised zones which have very low potential of eventual economic extraction have been excluded from the report. • Oxide copper Mineral Resources were reported at a cut-off grade of 1% Cu for all copper open pit Mineral Resources • Oxide gold Mineral Resources were reported at a cut-off grade of 1.1g/t Au for all gold open pit Mineral Resources. • The Mineral Resources are further constrained to within pit shells optimised using optimistic pricing. ○ The Gossan Hill Mineral Resource was reported within: ▪ Copper: the current mine design based on US$3.33/lb pit-shell • Gold: was not reported within a pit-shell, but instead reported above the 10240 mRL. • The reporting cut-off grades are in line with MMG's policy on reporting of Mineral Resources which is prospective for future economic extraction.
Mining factors or assumptions	• Underground mining at Golden Grove comprises long-hole open stoping and ore is hauled or hoisted to the surface. The minimum mining width is 3 m, which is based on the minimum spacing for a dice five drill hole pattern. • Surface mining is applied to the oxide and partial oxide copper and the oxide gold mineralisation and involves the open pit mining method. • Future mining factors and assumptions have been based on current mining practices using 100t trucks and 15t diggers.
Metallurgical factors or assumptions	• Metallurgical processing of ore at Golden Grove has been in operation since 1990 and involves crushing, grinding, sequential froth flotation followed by filtration before being transported to market as concentrates of copper, zinc and lead (including high-precious metals). • Primary sulphide material: ○ Metallurgical factors are incorporated into model block

	values via the calculation of the NSR value. o Recovery of payable minerals is dependent on iron ratios. Lower iron mineralisation is more amenable to copper and zinc recovery. o Higher grade zinc mineralisation is amenable to better precious metal recoveries. • Copper oxide material: o Pyrite within the transition material frequently has a thin layer of copper sulphide, which results in lower grade concentrates with high iron and sulphur concentration. o Au, Ag and Zn oxide material: o The gold and silver in the oxide material is recovered through a carbon in pulp (CIP) circuit. In the CIP process, copper is considered to be a deleterious element. Currently the model only contains ore grade assays for copper, no geochemical or cyanide soluble assays have been performed. o The transitional material can cause issues as it contains a mixture of oxides and secondary sulphides. Composition changes throughout the deposit. At best a mixed concentrate will be produced. o Lead oxide will be produced from the transition ore. o Fresh “copper oxide material” will behave in the plant in a similar manner to underground material.
Environmental factors or assumptions	• No environmental assumptions have been used in the classification of the Golden Grove Mineral Resources. • Material from underground and the open pit is sent to a designated stockpile based on material classification of either potentially acid forming (PAF) or non-potentially acid forming (NAF) material. Waste material with less than 0.2% sulfur is classified NAF while material with 0.2% sulphur or more is classified PAF. PAF/NAF classification is based on recommendation from Coffey Environment after their test work on-site.
Bulk density	• All core samples are measured for bulk density in the on-site core processing facility. The bulk density method used is the Archimedes' principle (weight in air and weight in water). The core is air dried and generally has low permeability and so the results are considered suitable for Golden Grove. • No wax coating or sealing of core is applied as the rocks rarely exhibit natural void spaces. • Density values in the Mineral Resource models are estimated using Ordinary Kriging within the mineralised domain shapes.
Classification	• Primary sulphide Mineral Resources: o A multidisciplinary approach to Mineral Resource classification, involving geology, geostatistics and mining,

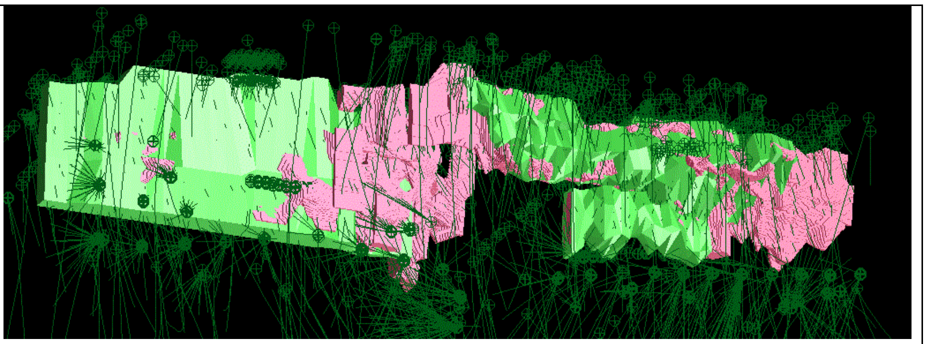
was undertaken. This was used in conjunction with an overriding consideration in grade confidence and geological continuity.

  - Drill hole density, estimation run, number of samples and drill holes used in estimation for each given block also influenced the classification. A summary of the guidelines are presented in the following table.

Quantitative Mineral Resource Classification Criteria

Classification	Drill hole spacing	Estimation run filled	Number of drill holes	Number of samples
Measured	10mx10m to 15mx15m	1-2	>=5	10-15
Indicated	20mx30m to 30mx30m	2-3	2-5	5-15
Inferred	Wider spacing	3-4	<=2	1-15

  - As well as the quantitative approach taken to the Mineral Resource classification, subjective qualification was also used. This included decisions to include, or not, areas that did not strictly meet the quantitative criteria.
- Oxide Mineral Resources:
  - Classification of the Mineral Resource was primarily based on confidence in the assayed grade, geological continuity.
  - Geological confidence is supported by nearby underground exposures including geological mapping and drill hole data, which in turn reinforces drill hole sample results and domain volumes. Confidence in the Kriged estimate is associated with drill hole coverage, analytical data integrity.
  - Indicated Mineral Resources was considered appropriate with a drill hole grid spacing of 10 m to 40 m.
  - Inferred Mineral Resource was considered appropriate with a drill hole grid spacing greater than 40 m and within the mineralisation domain.
  - The Gossan Hill oxide gold deposit has good potential for upgrading a large portion of the Indicated Mineral Resource into a Measured Mineral Resource. Requirements for this rely upon improvement/assurance of sample quality (recoveries) and QAQC, together with supporting studies for defining a cut-off grade and the eventual economic extraction of this mineralisation.
  - Long section view (looking east) of the estimated gold deposit block model showing Mineral Resource classification categories (green = Indicated, pink = Inferred)

	 **Long section view (looking west) of the estimated gold deposit block model, with classification categories (green = Indicated, pink = Inferred)** • Copper oxide Mineral Resources (specific only to the Cu Ox): o No material was classified as Measured due to insufficient bulk density information and the highly variable nature of the ore body's bulk density. Bulk density has been sampled on a 20 m x 20 m pattern. o Indicated is classified within the copper ore wireframe and covered by 10 m x 10 m grade control pattern. Inferred is classified to be inside the copper ore wireframe but outside of the grade control area. • The Competent Person is satisfied that the stated Mineral Resource classification reflects the geological domains interpreted and the estimation constraints of the deposits.
Audits or reviews	• MMG has an internal 'Mineral Resource and Ore Reserve Policy' that requires at a minimum external reviews every three years, and internal company review every interim year by MMG representatives reporting findings to a sub-set of the MMG Mineral Resource and Ore Reserve Committee. 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. • Internal audits were conducted in 2014 and 2015 which included MMG group office and site personnel. No material issues with the Mineral Resource estimates were identified.
Discussion of relative accuracy/ confidence	• The Mineral Resource data collection, data analysis and estimation techniques used for the Golden Grove deposits are consistent with the currently mining areas both underground and open cut and there has not been any known major discrepancies between the mined grades and the milled grades. • The Competent Person is satisfied with the accuracy and the confidence of the Mineral Resource estimates. At this time confidence limits of grade and tonnage have not been calculated.

8.5 Ore Reserves – Golden Grove Underground

8.5.1 Results

The 2015 Golden Grove Underground Ore Reserve are summarised in Table 26.

Table 26 2015 Golden Grove Underground Ore Reserve tonnage and grade (as at 30 June 2015)

Golden Grove Underground Ore Reserve

							Contained Metal				
	Tonnes (Mt)	Copper (% Cu)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	Gold (g/t Au)	**Copper ('000 t)**	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Gossan Hill[1]											
Primary Copper											
Proved	0.7	3.0	4.5	0.5	40	2.5	22	33	4	0.9	0.1
Probable	0.5	2.6	6.1	0.7	49	3.9	14	32	4	0.8	0.1
Total	**1.3**	**2.8**	**5.2**	**0.6**	**44**	**3.1**	**36**	**65**	**7**	**1.8**	**0.1**
Primary Zinc											
Proved	0.8	0.6	12.3	1.8	109	4.0	5	103	15	2.9	0.1
Probable	0.9	0.3	11.2	2.0	150	1.5	2	95	17	4.1	0.0
Total	**1.7**	**0.4**	**11.7**	**1.9**	**130**	**2.7**	**7**	**198**	**32**	**7.0**	**0.1**
Surface Stockpiles											
Primary Copper											
Proved	0.05	2.7					1				
Primary Zinc											
Proved	0.02	0.4	9.4	1.7	106	1.3	0.1	1.9	0.3	0.1	0.0
Gossan Hill Total							**44**	**265**	**39**	**8.9**	**0.3**
Scuddles[2]											
Primary Copper											
Proved	1.0	3.1	0.3	0.0	13	0.4	31	3	0	0.4	0.01
Probable	0.4	2.9	0.2	0.0	11	0.2	11	1	0	0.1	0.00
Total	**1.4**	**3.1**	**0.3**	**0.0**	**13**	**0.4**	**43**	**4**	**0**	**0.6**	**0.02**
Primary Zinc[1]											
Proved	0.3	0.3	11.1	1.0	84	0.9	1	30	3	0.7	0.01
Probable	0.0	0.1	10.5	1.0	82	0.9	0	3	0	0.1	0.00
Total	**0.3**	**0.3**	**11.0**	**1.0**	**84**	**0.9**	**1**	**34**	**3**	**0.8**	**0.01**
Scuddles Total							**44**	**38**	**3**	**1.4**	**0.0**
Total Contained Metal							**88**	**302**	**43**	**10.3**	**0.3**

Reserve numbers are inclusive of surface stockpiles.

1 Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value A$145/t.

2 Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value A$140/t.

Contained metal does not imply recoverable metal.

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

8.6 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 27 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code.

Table 27 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Golden Grove Underground Ore Reserve 2015

Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the sub-set of the Mineral Resources used to define the Ore Reserves. The Ore Reserves are derived from Mineral Resources using the geological database current as at 1 January 2015. The Ore Reserves are based on detailed development and stoping designs and have been economically evaluated according to current metal price forecasts, recent operational costs, and mining and metallurgical recoveries. • An average of 43% of the Measured and Indicated current Mineral Resources estimate was converted into Ore Reserves for both Gossan Hill and Scuddles mine.
Site visits	• The Competent Person (Wayne Ghavalas) is based on site in his capacity as underground mining manager.
Study status	• Gossan Hill and Scuddles mines are operating mines, Mineral Resources to Ore Reserve conversion was carried out based on the latest Mineral Resource model using the geological database as of 1 January 2015. • Ore Reserves inputs parameters have been estimated based on historical performance data.
Cut-off parameters	• Due to the polymetallic nature of Golden Grove, all the factors relating to the value of the ore (representative of cash costs to mine gate) are combined into a net smelter return after royalities (NSRAR) value. Based on the economic assumptions and cost review, the NSRAR cut-off is A$145/t for Gossan Hill mine and A$140/t for Scuddles mine. The NSRAR calculation includes metallurgical recovery, milling cost, financial assumptions which include metal price and exchange rate, concentrate road and sea transportation costs (both dollar value and concentrate loss), royalties payable, treatment and refining charges. • The NSRAR cut-off was used as a guide to generate the Ore Reserves shapes; however each stope is assessed individually for the development, haulage distance, backfilling requirements and any other additional costs to ensure that it is profitable to mine. These costs are considered on an individual stope basis and stopes that make a loss have not been included in the Ore Reserves.
Mining factors or assumptions	• Gossan Hill and Scuddles mine are operating mines, therefore all of the mining factor assumptions are using the historical performance data. • Geotechnical parameters applied in the design process are based on each stoping areas given an estimate of stable Hydraulic Radius (HR). This is based on Q' values, experience with similar mine areas and numerical modelling. The table below shows base values of

Assessment	Commentary
	allowable hydraulic radius for different orebodies.

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

- Mining dilution and recovery factors used are different for each orebody, based on the historical performance data. See the Mining dilution and recovery factors for each orebody in the table below.

Orebody	**Recovery**	**Dilution**			
		Zn Dil	**Grade***	**Cu Dil**	**Grade***
ABCZinc	95%	11%	2%	11%	10%
ACopper	93%	6%	2%	6%	10%
Amity	94%	9%	2%	9%	10%
Camberwarra	90%	10%	2%	10%	10%
Catalpa	90%	10%	2%	10%	10%
CCopper	97%	7%	2%	7%	10%
DZinc	90%	10%	2%	10%	10%
Ethel	86%	7%	2%	7%	10%
Hougomount	94%	11%	2%	11%	10%
Hougomount HW	94%	11%	2%	11%	10%
Oizon	90%	20%	2%	20%	10%
QCopper	95%	10%	2%	10%	10%
Scuddles	95%	15%	2%	15%	10%
Tryall	90%	10%	2%	10%	10%
Xantho	95%	18%	2%	18%	10%
XanthoExtended	90%	20%	2%	20%	10%

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

- Based on historical performance data and current equipment capability, the minimum mining width used for the design is 4 m with level interval generally 30 m.
- Inferred Mineral Resource is not included in the Ore Reserve estimate.
- The infrastructures required have been established for both Gossan Hill and Scuddles mines.

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td>Metallurgical factors or assumptions</td><td>

Metallurgy processing of ore at Golden Grove has been in operation since 1990 and involves crushing, grinding, sequential froth flotation followed by filtration before being transported to market as concentrates of copper, zinc and lead (including high-precious metals).
The current Golden Grove processing operation caters for either a single concentrate production in the case of copper sulphide (CuS) or copper oxide (CuO) ore, or a 2-stage sequential concentrate production for lead (Pb)/zinc (Zn) ore.
Parameters and assumptions for the processing plant are determined annually by reviewing historical data, the Gossan Hill/Scuddles Mining Plan (advising ore types and masses to be made available for processing in the forthcoming year) and anticipated metal prices for Cu, Zn and Pb.
The table below shows the average YTD actual metallurgical recovery since 2010.

<table>
<tr><th>Concentrate</th><th>Mineral</th><th>Recovery</th></tr>
<tr><td>Zinc</td><td>Zn</td><td>89%</td></tr>
<tr><td rowspan="2">Copper</td><td>Cu</td><td>89%</td></tr>
<tr><td>Ag</td><td>56%</td></tr>
<tr><td rowspan="3">HPM</td><td>Pb</td><td>70%</td></tr>
<tr><td>Ag</td><td>64%</td></tr>
<tr><td>Au</td><td>67%</td></tr>
</table>

Deleterious elements such as high level of talc could impact the recovery. Depending on the severity of the talc, the flotation circuit configurations may require alterations to counter the problem. The most common method of reducing the impact of talc in feed is to prefloat a talc rich concentrate and discard to tailings. Talc and Magnetite are not estimated or modelled extensively in the Mineral Resource. High iron content could also impact the recovery.
The Ore Reserve estimation is based on the appropriate mineralogy.

</td></tr>
<tr><td>Environmental</td><td>

Waste rock from underground shall preferentially remain underground where it is used to as a source of backfill. The waste rock that is transported to surface is either returned to underground as road base, or is encapsulated in the dedicated ROM Pad Potential Acid Forming Encapsulation Facility.
Tailing is directed to a Tailings Storage Facility, some of the mill tailings are returned underground as Cemented Hydraulic Fill (CHF) for backfilling. Water from this facility is decanted after the tailings has settled and is returned to the processing plant for recycling in the process. Golden Grove currently has two tailings storage facilities.

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

Assessment	Commentary
Infrastructure	• Being an operating site, the infrastructure required to mine and process ore from both Gossan Hill and Scuddles mines is well established. • The site airstrip was sealed in 2007 and is serviced by flights from both Perth and Geraldton. The current site facilities and short commute (1 hour flight time from Perth and 45 minutes from Geraldton). • Golden Grove operates with a work force of approximately 650 full time equivalent employees drawing labour from both the Perth labour market and from Mid-West country centres such as Geraldton and towns close to the mine such as Yalgoo. This includes both people employed directly through MMG and contractors providing both contracting services and/or labour hire. • Accommodation village located 5 km to the south-southwest of the mine offices and accessed via a sealed road. • Electricity is supplied from the WA grid through a southern distribution centre at Three Springs. Power consumption is typically around 14 MW. Three of 1.15 MW power generators are installed to enable essential services and underground fans to operate and to prevent bogging of tanks and thickeners. • Water supply for the operations is secure with sufficient groundwater supply, The majority of the groundwater is supplied through dewatering of both the Gossan Hill and Scuddles underground mines. The site equipped by two backup potable water bores to ensure the sites potable demand can be met. • Transportation of bulk commodities access to Golden Grove minesite is via sealed roads from Perth to Paynes Find and from Geraldton to Yalgoo. The Yalgoo to Paynes Find road is sealed between Yalgoo and Golden Grove while the remainder between Golden Grove and Paynes Find is a formed gravel road that can be closed to traffic during periods of wet weather.
Costs	• The operating and capital costs were determined using the historical cost data. • Deleterious element such as high iron was included in the NSRAR calculation. High iron content level could impact the copper and zinc recoveries. Talc and magnetite are not estimated in the Mineral Resource and the NSRAR calculation. • The commodity price and exchange rate assumptions are supplied by Melbourne office. The majority of 2015 Ore Reserve is assessed as a medium-term Ore Reserve scenario. The ore scheduled to be mined after 2018 is assessed by long-term Ore Reserve scenario. • Treatment, refining and transportation costs for different commodities were supplied by Melbourne office and have been included in the NSRAR calculation. • The royalty value varies based on commodity type were supplied by Melbourne office. A 5% royalty has been applied to base metals, and 2.5% to precious metals. The royalties have been

Assessment	Commentary
	included in the NSRAR calculation
Revenue factors	• The commodity prices and exchange rate assumptions, treatment, refining, royalties and transportation costs for different commodities were supplied by Melbourne office and have been included in the NSRAR calculation. • The formulas and assumptions used in the NSRAR calculation are based on the historical data provided by Metallurgy Department. • The economic evaluation was carried out to verify whether the stope designed using the NSRAR cut-off generate economic outcome. The mining physicals required to access and mine individual stopes were determined during the mine design process. The cost assumptions were applied to the mining physicals and the revenue was calculated by multiplying the recovered ore tonnes by the applicable NSRAR value. The profitable and marginal stopes were included in the Ore Reserve.
Market assessment	• MMG's long-term view on global consumption of metals are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. • MMG has a long-term positive view of copper market fundamentals with future supply likely to be constrained as declining grades, increasing costs, slow future mine production and investment. MMG's view centres on the future copper supply contracting as current reserves are depleted and continued global demand copper is forecast to grow at or above the global average rate. • The long-term outlook for zinc will be determined by the ability of miners to offset the impact of scheduled mine closures and growing demand. Future zinc supply will likely come from lower-grade, higher-cost underground mines as current reserves are depleted. Continued growth in the construction, transportation and infrastructure sectors especially in the developing economies, will support solid demand for zinc in the medium to long term.
Economic	• Golden Grove 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 a realistic medium to long-term metal prices. • The life of mine (LOM) financial model demonstrates the mine has a positive NPV. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit.
Social	• MMG Golden Grove is located within the Shire of Yalgoo in the Murchison Region of Western Australia. The nearest community to Golden Grove is the Yalgoo Township, which is situated approximately 56 km to the north of the site. The key stakeholders include the local government and community, pastoralists, employees and the Geraldton Port Authority. • Golden Grove owns the Muralgarra Pastoral Station leasehold land

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>which was purchased in 2007. Golden Grove is currently developing a strategic and diversified management plan with a focus on carbon sequestration and biodiversity offset project opportunities that were implemented from 2012.
• Stakeholder consultation was initiated when mining was first proposed for the Golden Grove operations. Consultation has evolved since mining commenced and occurs during additional approval work and through routine and ad-hoc engagement.
• Golden Grove continues to fulfil its statutory heritage responsibilities in line with the Aboriginal Heritage Act (1972) and the Aboriginal and Torres Strait Islander Heritage Protection Act (ATSHIP 1984).
• Golden Grove has committed to a range of Indigenous Relations investment initiatives including; the Bayalgu Indigenous Pre-Employment Training program; the Yalgoo Centacare Indigenous Children's Program (ICP); a Cross Cultural Awareness (CCA) program delivered by a local service provider; and an Indigenous Employment Implementation Plan (IEIP).</td></tr>
<tr><td>Other</td><td>• Gossan Hill and Scuddles mines tenement and land tenure status are listed in the table below.
<table>
<tr><th>Tenement No.</th><th>Prospect Name</th><th>Date Expires</th><th>Term Years</th><th>Date Granted</th></tr>
<tr><td>M59/03</td><td>Scuddles</td><td>08/12/2025</td><td>21</td><td>28/01/2005</td></tr>
<tr><td>M59/195</td><td>Gossan Hill</td><td>17/05/2032</td><td>21</td><td>17/06/2011</td></tr>
</table>
• The Gossan Hill and Scuddles mines operate under license L8593/2011/2 issued by the Western Australian Department of Environment and Regulation (DER) as required by the Environmental Protection Act 1986. This license was issued 11 September 2014 and expires on 15 September 2019.</td></tr>
<tr><td>Classification</td><td>• Ore Reserve is classified as Proved and Probable. Proved Ore Reserve category is determined when Mineral Resource confidence level is "Measured" and financially satisfied which is either "Profitable" or "Marginal". Probable Ore Reserve category is determined when Mineral Resource confidence level is "Indicated" and financially level is either "Profitable" or "Marginal".
• An average of 43% of the Measured and Indicated current Mineral Resources estimate was converted into Ore Reserves for both Gossan Hill and Scuddles mine (it should be noted that the Mineral Resources NSRAR cut-off are A$145/t for Gossan Hill and A$140/t for Scuddles using specific financial assumptions for Mineral Resource supplied by Melbourne office; commodity prices, exchange rate, treatment, refining, royalties and transportation).</td></tr>
<tr><td>Audit or reviews</td><td>• No external audits were undertaken to the Ore Reserve estimate.</td></tr>
</table>

Assessment	Commentary
Discussion of relative accuracy/ confidence	• A qualitative risk assessment of each discussed item is included with each individual item in the second column of this table. Details of various risks are discussed in each relevant section. • The Ore Reserve estimate is compared with the production data.

8.6.1 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 28.

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 28 Contributing Experts Golden Grove Underground Ore Reserve

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Kiki Kosmara, Senior Mining Engineer, MMG Ltd (Golden Grove)	Underground Mining
Wayne Ghavalas, Underground Mine Manager MMG Ltd (Golden Grove)	Underground Mining
Paul Boamah , Senior Mine Geologist – Resource MMG Ltd (Golden Grove)	Geology, (2015 Mineral Resource Estimation – Gossan Hill)
Lauren Stienstra, Senior Mine Geologist MMG Ltd (Golden Grove)	Geology, (2015 Mineral Resource Estimation – Scuddles)
Andrew Dowling, Senior Metallurgist MMG Ltd (Golden Grove)	Metallurgy and general processing
Stephen Ross, Commercial Manager MMG Ltd (Golden Grove)	Operating costs
Ben Ryan, Environment Superintendent MMG Ltd (Golden Grove)	Environment
Chris Pennisi, Geotechnical Engineer MMG Ltd (Golden Grove)	Geotechnical
Danae Sheldrick, Community Relation Specialist MMG Ltd (Golden Grove)	Social and Community Relation Agreements

8.7 Ore Reserves – Golden Grove Open Pit

8.7.1 Results

The 2015 Golden Grove Open Pit Ore Reserve are summarised in Table 29.

Table 29 2015 Golden Grove Open Pit Ore Reserve tonnage and grade (as at 30 June 2015)

Golden Grove Open Pit Ore Reserve

							Contained Metal				
Partial Oxide Copper[1]	Tonnes (Mt)	Zinc (%)	Copper (%)	Lead (%)	Silver (g/t)	Gold (g/t)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Proved	0.1		2.8					3	-	-	-
Probable	0.2		2.1					4	-	-	-
Total	**0.3**		**2.3**					**7**	**-**	**-**	**-**
Primary Copper[1]											
Proved									-	-	-
Probable	0.1		1.9					1			
Total	**0.1**		**1.9**					**1**	**-**	**-**	**-**
Total Contained Metal								**8**			

1.4% Cu cut-off grade.

Contained metal does not imply recoverable metal.

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

Proved Ore Reserves are on stockpiles waiting processing. Probable Ore Reserves remain to be mined.

8.8 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 35 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code.

Table 30 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Golden Grove Open Pit Ore Reserve 2015

Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of the Mineral Resources used to define the Ore Reserves. • The Ore Reserves estimate has been generated by applying the metallurgical, social, environmental and financial aspects of the operations (the modifying factors) on the portion of the Mineral Resource Estimate, classified as "Measured" and "Indicated".
Site visits	• The Competent Person is Christopher Lee AusIMM (CP) who is based on site in a permanent role as the Open Pit Manager.
Study status	• The Golden Grove open pit is 98% mined by volume consequently the ore body is well understood.
Cut-off parameters	• A cut-off grade of 1.8% Cu was used in determining the Ore Reserve. • Break even cut-off grades (COG) were calculated for oxide, transition, and primary ores. The COG estimates included all relevant costs incurred post mining. The calculated COG is 1.8% for transition and primary material. • Revenue factors used in the cut-off grade calculation include forecast copper price, moisture content of concentrate, freight and shipping costs, insurance, royalty payments, treatment and refining charges. • Fixed costs to process the ore and revenue factors are well understood.
Mining factors or assumptions	• Mining is carried out using a 120T class backhoe excavator and 100T rigid dump trucks. All material is drilled and blasted prior to mining occurring. The orebody is surveyed prior to mining and all ore is mined on dayshift under direct geological supervision. • The pit over its life has experienced very minor geotechnical issues with no significant failures. The pit walls are now moving into fresh rock. Wall angles are determined by an independent geotechnical consultant who models defects and predicts likely failure modes carrying out a range of analyses. Batter angles vary from 55 degrees to 70 degrees with an 18 m vertical interval. Overall slope angles are between 48 degrees and 52 degrees. Actual geotechnical performance against design is reviewed by the external consultant on an annual or greater frequency if required. • The Ore Reserve is based on grade control drilling results; grade control is carried out using reverse circulation drilling on a 10 m by 10 m staggered pattern. • The mining dilution factor is zero and the mining recovery factor is 100%. This is consistent with the approach used in the Feasibility

Assessment	Commentary
	Study. The impact of the modelling process is assumed to compensate for any additional impact from either dilution or ore loss. The orebody is wide; typically 60 m to 90 m, has clear ore/waste boundaries and no internal waste. There is no reconciliation data to support using alternative numbers. • Inferred Mineral Resources are excluded from the Ore Reserve. • No minimum mining widths are applied as the orebody dimensions are much larger than the physical size of the excavator bucket. • The infrastructure is established for the selected mining method.
Metallurgical factors or assumptions	• The Golden Grove mine is an operating entity. The metallurgical process consists of crushing, grinding, and floatation to produce a copper concentrate. • The remaining ore to be mined is largely primary ore. In determining what recovery to use for Ore Reserves the actual mill data from the last two mill campaigns has been analysed. From this data a formula has been developed that closely matches actual plant performance on this ore and can be expressed as follows. Cu rec (%) = 95 - 75/feed % Cu
Environmental	• Progressive rehabilitation of the waste dump is continuing with excellent results. During the year potentially acid forming waste (PAF) was encountered and this was delivered to the PAF storage facility adjacent to the Gossan Hill ROM Pad as planned. Areas of PAF waste are identified from grade control drilling and marked up in the field for mining; a sulphur grade of 0.2% is used to define PAF material. • During the last year a perimeter fence around the open pit and waste dump was completed to control feral goat grazing pressure on the waste dump. A second sediment trap was installed to collect any runoff from the waste dump. Extensive re-growth is evident on the dump and monitoring stations have been established to provide landform data.
Infrastructure	• Being an operating site, the infrastructure required to mine and process ore from the Gossan Hill open pit mine is well established. • Access to Golden Grove Operation is via sealed roads from Perth to Paynes Find and from Geraldton to Yalgoo. The Yalgoo to Paynes Find road is sealed between Yalgoo and Golden Grove while the remainder between Golden Grove and Paynes Find is a formed gravel road that can be closed to traffic during periods of wet weather. • The site airstrip was sealed in 2007 and is serviced by flights from both Perth and Geraldton. The current site facilities and short commute (1 hour flight time from Perth and 45 minutes from Geraldton) has seen Golden Grove maintain a steady workforce with current employee turnover less than 10%, a solid result in a fly-in-fly-out environment.

Assessment	Commentary
	• Drawing labour from both the Perth labour market and from MidWest country centres such as Geraldton and towns close to the mine such as Yalgoo, Golden Grove operates with a work force of approximately 650 full time equivalent employees. This includes both people employed directly through MMG and contractors providing both contracting services and/or labour hire. • Accommodation village located 5 km to the south-southwest of the mine offices and accessed via a sealed road. • Electricity is supplied from the WA grid through a southern distribution centre at Three Springs. Three of 1.15MW power generators are installed to enable essential services and underground fans to operate and to prevent bogging of tanks and thickeners. • Water supply for the operations is secure with sufficient groundwater supply; the majority of the groundwater is supplied through dewatering of both the Gossan Hill and Scuddles underground mines. The site equipped two backup potable water bores to ensure the sites potable demand can be met. • Additional infrastructure includes waste dumps (including potentially acid forming materials), ore pad/stockpile, workshop facility, bulk fuel storage, explosive magazines, ammonium nitrate storage facility, and offices for technical staff, metallurgical processing plant and tailings storage facility. • All required infrastructure are in place to realise the mining and processing of the open pit Ore Reserve.
Costs	• The open pit is being mined by a contractor using a schedule of rates. The schedule of rates allows unit costs to be determined bench by bench with a high degree of certainty. For the Ore Reserve a bench by bench financial model has been prepared. • Milling, administration, and other costs are based on historical site and budget forecast costs. • The Ore Reserves used information supplied by MMG Corporate in regards to metal prices and economic assumptions. • An exchange rate of $0.82 was used A$ to US$ supplied by MMG Corporate. • Freight, shipping and insurance, charges are based on historical site and budget forecast costs and have been included in the net smelter return after royalties (NSRAR) calculation. • The source of treatment and refining charges are based on historical site and budget forecast costs, and have been included in the NSRAR calculation. Penalties for failure to meet specification are not included as they are thought to be non-material and have not been included in the NSRAR calculation. • Applicable Royalties payable and have been included in the NSRAR calculation. • There is no additional capital expenditure required to mine and

Assessment	Commentary
	process the Ore Reserve.
Revenue factors	- Revenue factors were based on MMG Corporate assumptions. The remaining Ore Reserve has a high profit margin and short life insulating the pit from fluctuations in revenue factors.
Market assessment	- MMG's long-term view on global consumption of metals are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. - MMG has a long-term positive view of copper market fundamentals with future supply likely to be constrained as declining grades, increasing costs, slow future mine production and investment. MMG's view centres on the future copper supply contracting as current reserves are depleted and continued global demand copper is forecast to grow at or above the global average rate.
Economic	- The open pit is mined by a contractor on a schedule of rates basis and these rates combined with owner's cost have been used to prepare a bench by bench financial model that indicates the remaining ore to be mined from the pit has a positive cash flow. - A cash flow model was produced based on a bench by bench analysis of physical mining quantities, all costs and revenue factors to generate a cumulative cash flow bench by bench. There is no additional capital expenditure required to mine and process the Ore Reserve. Revenues are calculated based on historical and contracted realisation costs and metal prices from MMG Corporate.The life of mine (LOM) financial model demonstrates the mine has a positive NPV. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit. - As the pit is 98% complete and the remaining volume to be mined is relatively small no sensitivity analyses were undertaken for the Ore Reserves work.
Social	- There are no known issues that could potentially affect the realisation of the Ore Reserve. - Golden Grove is located within the Shire of Yalgoo in the Murchison Region of Western Australia. The nearest community to Golden Grove is the Yalgoo Township, which is situated approximately 56 km to the north of the site, with a population of approximately 100. The key stakeholders include the local government and community, pastoralists, employees and the Geraldton Port Authority. - Golden Grove owns the Muralgarra Pastoral Station leasehold land which was purchased in 2007. Golden Grove is currently developing a strategic and diversified management plan with a focus on carbon sequestration and biodiversity offset project opportunities that were implemented from 2012. - Stakeholder consultation was initiated when mining was first

Assessment	Commentary
	proposed for the Golden Grove operations. Consultation has evolved since mining commenced and occurs during additional approval work and through routine and ad-hoc engagement. • Golden Grove continues to fulfil its statutory heritage responsibilities in line with the Aboriginal Heritage Act (1972) and the Aboriginal and Torres Strait Islander Heritage Protection Act (ATSHIP 1984). • Golden Grove has committed to a range of Indigenous Relations investment initiatives including; the Bayalgu Indigenous PreEmployment Training program; the Yalgoo Centacare Indigenous Children's Program (ICP); a Cross Cultural Awareness (CCA) program delivered by a local service provider; and an Indigenous Employment Implementation Plan (IEIP). • Golden Grove is located in an area that is under claim by two Indigenous Native Title claimant groups. • All existing government approvals are in place for the continued mining of the Ore Reserve. The two key approvals are Operating Licence L5175/1988/9 and Mining Proposal ID 29469. • All mining activities occur on tenement M59/195 expiry date May 2032.
Other	• The Golden Grove Mine has been established since 1980 all necessary regulatory approvals exist and there is no known risk that could jeopardise the realisation of the remaining Ore Reserve. • There are no material naturally occurring risks identified. • There are no outstanding material legal agreements. Marketing agreements are in place for the life of the open pit asset. • All mining activities occur on tenement M59/195 expiry date May 2032.
Classification	• The in-pit Ore Reserves are classed as Probable only, in line with Mineral Resources classification of Indicated. • Stockpiled ore has been classified as Proved, in line with Mineral Resources classification of Measured. • Inferred Mineral Resources are not included in the Ore Reserves.
Audit or reviews	• An Internal Post Investment Review was carried out on the Copper Open Pit Project. The review concluded that the investment in the open pit at Golden Grove added value to MMG, however this value added has not been at the level outlined in the board approval. There are a variety of contributing factors to this with the dominant one being the lower than anticipated recovery on oxide and transitional material.
Discussion of relative accuracy/ confidence	• The open pit is largely exhausted with a small amount of primary ore remaining to be mined. Uncertainty on the remaining Ore Reserves is such that the likelihood of destroying the robust economics of the remaining Ore Reserves is extremely low. • Monitoring of actual mill performance will be important

Assessment	Commentary
	particularly recovery in order to optimise the return from remaining Ore Reserve as we move into primary material. Mining and milling of the open pit Ore Reserve will be completed during 2016 with the exception of the stockpiled low grade ore.

8.8.1 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 31.

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 31 Contributing Experts Golden Grove Open Pit Ore Reserve

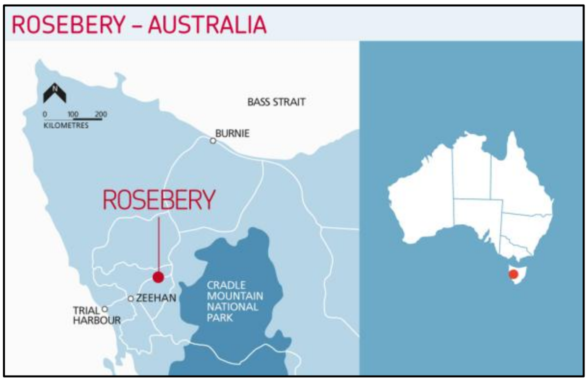
EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Chris Lee, Open Pit Manager, MMG Ltd (Golden Grove)	Open Pit Mining
Rob Oakley, Senior Open Pit Geologist MMG Ltd (Golden Grove)	Geology
Toby Fields, Manager Processing MMG Ltd (Golden Grove)	Metallurgy and general processing
Stephen Ross, Commercial Manager MMG Ltd (Golden Grove)	Operating costs
Ben Ryan, Environment Superintendent MMG Ltd (Golden Grove)	Environment
Peter O'Bryan, Geotechnical Engineer Peter O'Bryan & Associates	Geotechnical
Danae Sheldrick, Community Relation Specialist MMG Ltd (Golden Grove)	Social and Community Relation Agreements

9 ROSEBERY

9.1 Introduction and Setting

MMG Limited holds the 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. The Mining Lease is located on the West Coast of Tasmania approximately 120km south of the port city of Burnie. The main access route to Rosebery Township from Burnie is via the B18 and the Murchison Highway (A10), which connect 8km east of Waratah. The mining lease encompasses the township of Rosebery.

Figure 9-1 **Rosebery Mine location**



Rosebery is a mechanised long-hole open-stope underground operation with footwall ramp access. The mine currently employs a benching mining method, but has historically also used a cut and fill stoping method. The mine has historically produced approximately 800,000 tonnes of ore per year with plans to increase this going forward. The ore is processed into separate concentrates for zinc, lead and copper. The mine also produces gold/silver doré bullion.

9.2 Mineral Resources – Rosebery

9.2.1 Results

The 2014 Rosebery Mineral Resource are summarised in Table 32. The Rosebery Mineral Resource is inclusive of the Ore Reserve.

Table 32 2015 Rosebery Mineral Resource tonnage and grade (as at 30 June 2015)

Rosebery Mineral Resource

							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	9.0	8.6	2.8	0.3	96	1.2	773	253	23	27.8	0.3
Indicated	6.4	7.3	2.5	0.3	103	1.1	470	159	16	21.3	0.2
Inferred	7.0	7.4	2.8	0.3	96	1.4	523	194	20	21.6	0.3
Total	**22.4**	**7.9**	**2.7**	**0.3**	**98**	**1.2**	**1,766**	**606**	**59**	**70.7**	**0.9**
South Hercules											
Measured	0.1	4.6	2.5	0.1	151	3.8	7	4	0.2	0.7	0.02
Indicated	0.02	3.7	1.8	0.1	161	4.3	1	0	0.0	0.1	0.00
Inferred											
Total	**0.2**	**4.5**	**2.4**	**0.1**	**152**	**3.9**	**7**	**4**	**0.2**	**0.8**	**0.02**
Total Contained Metal							**1,774**	**610**	**59**	**71.5**	**0.9**

Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value of A$179/t.
Figures are rounded according to JORC Code guidelines and may show apparent addition errors.
Contained metal does not imply recoverable metal.

9.3 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 33 complies with the 2012 JORC Code requirements specified by "Table-1 Section 1-3" of the Code.

Table 33 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Rosebery Mineral Resource 2015

Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• Diamond drilling (DD) was used to obtain an average 1 m sample that is half core split, crushed and pulverised to produce a pulp (>80% passing 75 µm). • DD core is selected, marked and ID tagged for sampling by the logging geologist. Sample details and ID are stored in the database for correlation with returned geochemical assay results. • Pulps are delivered to the ALS laboratory in Burnie, Tasmania for chemical analysis. • There are no inherent sampling problems recognised.
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. • Drilling undertaken from 2012 is LTK48, LTK60, NQ, NQTK, BQTK and BQ in size. • Historical (pre-2012) drill holes are a mixture of sizes from AQ, LTK (TT), BQ, NQ, HQ and PQ.
Drill sample recovery	• DD core recoveries average 96.9% based on 42,063 measured intervals since 2012. Drill crews mark and define lost core intervals with blocks. Sample recovery is measured and recorded in the drill hole database. • 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 known 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 considered high.
Logging	• 100% of diamond drill core has been geologically and geotechnically logged to support Mineral Resource estimation, mining and metallurgy studies. • Geological logging is qualitative and geotechnical logging is quantitative. • All drill core is photographed. • All drill holes are logged using laptop computers into the drill hole database. • Prior to 1996 diamond drill holes were logged using Lotus spread sheets or on paper.
Sub-sampling techniques and sample preparation	• Geological samples are prepared according to the Rosebery Work Instruction - DD Core Sample Preparation. • All samples included in the Mineral Resource estimate are from DD 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 1 m with a minimum length of 40 cm and maximum of 1.5 m. Sample intervals do not extend over domain boundaries (checked by database algorithm). • From 2005 geological samples have been processed in the following manner: ○ Dried, crushed and pulverised to 80% passing 75 µm, ○ 3-Acid Partial Digest (considered suitable for base metal sulphides), analysis of Pb, Zn, Cu, Ag, Fe by Atomic Absorption Spectrometry (AAS). ○ Au values are determined by fire assay ○ Sizing analysis is carried out on 1:20 pulps • From 2010 geological samples have been processed in the following manner: ○ Dried, crushed to 2 mm; cone split to give primary and duplicate samples with the remainder rejected. Pulverised to 80% passing 75 µm onsite, dispatch to ALS Burnie. ○ Analysis of 0.2 g Pb, Zn, Cu and Fe by lithium borate oxidative fusion, followed by X-Ray Fluorescence (XRF). ○ Analysis of 0.4 g Ag by aqua regia digest and Atomic Absorption Spectrometry (AAS). ○ Analysis of 30 g Au by fire assay, 3 acid digest and flame assay AAS. ○ Sizing analysis is carried out on 1:20 pulps • Representivity of samples is checked by duplication at the crush stage. • Current crush and pulverise procedures liberate representative sample grain sizes for analytical purposes. • Twelve month rolling QAQC analysis of sample preparation techniques indicate the process is appropriate for high sulphide samples; for Ag and Au the main sources of sample error occur as gravity settling in pulps during transport and storage. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Rosebery mineralisation (sediment hosted base metal) by the Competent Person.
Quality of assay data and laboratory tests	• The assaying methods currently applied for the Rosebery samples include; X-ray Fluorescence (XRF) for Pb, Zn and Cu, Atomic Absorption Spectrometry (AAS) for Ag and fire assay with AAS finish for Au, which is considered suitable for Mineral Resource estimation at Rosebery. All of these analyses are considered total digest. • Advertised ALS detection limits are as follows: Pb 0.01% Zn 0.01% Cu 0.01% Ag 1ppm

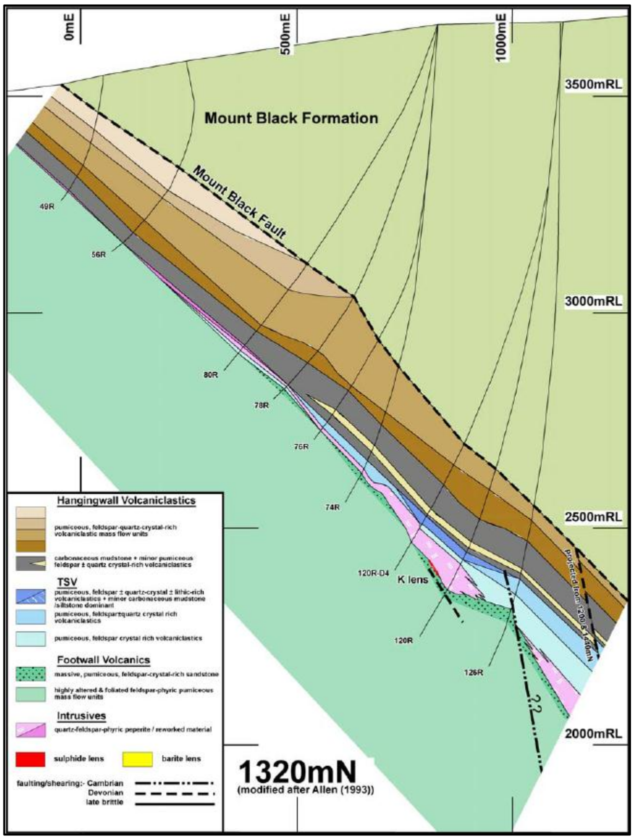
Au 0.01ppm
Fe 0.01%

- Assay techniques are considered suitable and representative; a comparison study using ICP techniques was completed to check the XRF accuracy in May 2013. Independent umpire laboratory re-assay of 5% pulps took place in October 2014 using the ALS Brisbane laboratory and ICP analysis, and again by ICP in June 2015 using the Intertek laboratory in Perth. Pulps for analysis were randomly selected from a list of samples where (Pb + Zn)>5%.
- 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.
- ALS laboratory Burnie releases its QAQC data to MMG for analysis of internal ALS certified reference material (CRM) performance. The performance of ALS internal CRM's appears to be satisfactory, with several CRM's used within the range of MMG submitted samples.
- MMG routinely insert matrix-matched CRM's, dolerite blanks and duplicates at a ratio of 1:25 to normal assays.
    - Blanks are inserted to check crush and pulverisation performance.
    - Duplicates are taken from a cone splitter after crush.
- Independent audit of the ALS Burnie laboratory and MMG Rosebery sample preparation area was undertaken in April 2013 by Coffey Mining Pty Ltd. Outcomes from the audit included:
    - Installation of a Boyd crusher cone splitter and instigation of crush duplicates and minor procedural sampling improvements at the MMG site.
    - Confirmation that the ALS analysis methods are sound except for a minor bias (up to 3% relative below expected) in the zinc, lead and copper assays for the MMG submitted certified reference material (CRM).
    - Sizing analysis shows that 96% of samples have at least 80% passing 75µm.
- During 2013 the sample preparation process was improved with the Boyd crusher output size being reduced from 3 to 2mm.
- The most recent sizing analysis of pulverised samples since August 2014 shows that 100% have at least 80% passing 75µm.
- QA/QC and analysis history:
    - 1996: Commenced use of three locally-sourced internal reference standards.
    - 2008: Commenced use of certified matrix-matched reference materials, duplicates and blanks.
    - 2010: Change to XRF analysis for Zn, Pb and Cu following a review suggesting results would be more accurate (lower bias) than 3-acid digest and AAS method previously applied.
    - 2013: Commenced internal reviews and reporting of monthly QAQC. Instigation of umpire laboratory pulp re-assays. Installation of Boyd crusher cone splitter.
    - 2014: Review and upgrading of QAQC and sample preparation procedures.
    - 2015: Updated database software allowing review of batch QAQC

	upon import. • Analyses of the control samples have established acceptable levels of accuracy and precision.
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 CRM's, and significant intersections are checked against core log entries and core photos. Errors are rectified before data is entered into the database. The personnel completing the above listed checks are not necessarily the Competent Person. • While formal drill hole twinning has not been undertaken, where drill holes from surface or older holes longer than >350 m exist, attempts are made to confirm mineralised intercepts by delineation/infill drilling programs collared from underground in newly developed drives. • Database validation algorithms are run to check data integrity before data is used for interpretation and Mineral Resource modelling. • Unreliable data is flagged and excluded from Mineral Resource estimation work. Datamine macros are also used to ensure unreliable data is not included in the modelling and estimation process. • No adjustments have been made to assay data – if there is any doubt about the data quality or location, the drill hole is excluded from the estimation process. As of August 2014, all data below detection limit is replaced by half detection limit values. Prior to this date, the full detection limit was used.
Location of data points	• All current diamond drill holes are down-hole surveyed using a single-shot Reflex Ezi-shot tool at 30m intervals, with a full down-hole Reflex gyro survey completed at end of hole by the drilling contractor. • Collar positions of underground drill holes are picked up by Rosebery mine surveyors using a Leica TPS 1200. • Collar positions of surface drill holes are picked up by contract surveyors using differential GPS. • Selected surface exploration holes have been downhole surveyed using a SPT north seeking gyro (parent holes only). Multi-shot downhole surveys were completed to check single-shot survey performance on selected drill holes. A total of 37 drill holes had multi-shot data recorded and compared to single-shot in 3D space. • A downhole gyro measurement has been recorded from selective drill holes since March 2014 as an independent check of downhole survey accuracy. Initial analysis suggests the single shot surveys are accurate to 100 m downhole depth, and then diverge up to 4 m at 400 m depth. Given this outcome, gyro downhole surveys are now standard for all DD holes. • The grid system used is the Cartesian Rosebery Mine Grid, offset from Magnetic North by 24°42' with mine grid origin at AMG E= 378870.055, N= 5374181.69; mine grid relative level (RL) equals AHD + 1.490m + 3048.000m. • Topographic control updated by five yearly LIDAR overflights carried out and correlated with surface survey datum. The LIDAR survey is considered to be of high quality and accuracy by the Competent Person for topographic

	control.
Data spacing and distribution	• The Rosebery mineral deposit is drilled on variable spacing dependent on lens characteristics. Drill spacing typically ranges from 80 m x 80 m (Inferred) to 40 m x 40 m (Indicated) to 20 m x 20 m (Measured) and is considered sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserves (MROR) estimation processes and classifications applied. • Drill hole spacing at 20 m x 20 m is considered sufficient for long term Mineral Resource estimation purposes based on variogram achievement of 70% of the sill variance at 20 m lag for all metals. Whilst the 20 m spacing is suitable for resource estimation, the Rosebery deposit has short scale 5 m-10 m structural variations that are not captured by the resource infill drill spacing. This localised geological variability is captured by Adamtech™ photographic mapping and transferred to digital grade control models for short term mine planning. • DD samples are not composited prior to being sent to the laboratory however the nominal sample length is generally 1 m. There has been no reverse circulation drilling at Rosebery.
Orientation of data in relation to geological structure	• Drill hole orientation is planned at 90 degrees to lens strike (north – south) 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. • Drilling orientation is not considered to have introduced sampling bias.
Sample security	• Measures to provide sample security include: ○ Adequately trained and supervised sampling personnel. ○ Samples are stored in a locked compound with restricted access during preparation. ○ Dispatch to ALS Burnie via contract transport provider in sealed containers. ○ Receipt of samples acknowledged by ALS by email and checked against expected submission list. • Assay data returned separately in both spreadsheet and PDF formats.
Audit and reviews	• No independent audits of the Rosebery database have been undertaken. • Coffey Mining Pty Ltd completed an audit of the core sample preparation area in April 2013. Key results of this audit are outlined in the 'Quality of assay data and laboratory tests' section above.
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 16.07 km^2. - The joint venture agreement was between the EZ Corp of Australia (now MMG Rosebery Mine) and Shell Company of Australia Limited (now AngloGold Australia Metals Pty. Ltd., formerly Acacia Resources (formerly Billiton)). A Heads of Agreement was signed on 16th May 1988 with initial participating interest of 50% for each party. Other partners in the joint venture are Little River Resources Ltd. and Norgold Ltd. They have a combined net smelter return royalty of 2.3695%, payable on production from the Rosebery Extension Joint Venture area. AngloGold withdrew from the joint venture on the 31st December 2001. - 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. - Inadequate metallurgical technology prevented fulfilment of the mine's potential until 1936 when the commissioning of a flotation plant by the Electrolytic Zinc Company enabled successful separation of the fine grained sulphide minerals. - Overall core drilling has steadily increased over time, with the peak of 73,651m during Pasminco's tenure in 1998 when a major surface drilling campaign was underway to delineate K and P lenses. - Surface drilling has been sporadic since 2000 and is currently in hiatus pending target generation. - Underground drilling has been consistently increasing and under MMG's tenure to between 35,000m and 40,000m. - Discovery of the Hercules and South Hercules ore deposits is attributed to A.E. Conliffe in 1891, two years before the Rosebery deposit was discovered. - Hercules mining operations ceased in 1986, having produced over 3.12Mt at 5.7%Pb, 17.8%Zn, 0.42%Cu, 180g/t Ag and 2.94g/t Au. - The South Hercules deposit has had minor early production and is currently in the feasibility stage (Easterbrook, 1962). - No exploration drilling carried out by other parties in the 2015 reporting period.
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.
Drill hole information	- Three exploration diamond drill holes were completed in the 2015 reporting period for a total of 1,599 m (two underground holes R10388 – 400 m and R10185 – 758 m, plus one surface hole 479R – 440.6 m).
Data aggregation methods	- All drilling data was treated exactly the same as Resource Infill drilling described in Section 1 of Table 4. The drilling results are excluded from this report as they are not considered Material by the Competent Person. - No metal equivalents were used in the Mineral Resource estimation. However the Mineral Resource has been reported above an A$179 NSRAR cut-off.
Relationship between mineralisation width and intercept lengths	- No significant mineralisation was intercepted. The drilling results are excluded from this report as they are not considered Material by the Competent Person. - Drill holes are drilled to achieve intersections as close to orthogonal as possible.
Diagrams	- No significant mineralisation was intercepted. The drilling results are excluded from this report as they are not considered Material by the Competent Person. 

	Cross-section of local geology at 1320mN, view to north (Martin, 2004).
Balanced reporting	- No significant mineralisation was intercepted. The drilling results are excluded from this report as they are not considered Material by the Competent Person.
Other substantive exploration data	- No significant mineralisation was intercepted. The drilling results are excluded from this report as they are not considered Material by the Competent Person.
Further work	- No further surface exploration work has been planned at this stage. - Ongoing underground drill programs will be planned to increase deposit confidence as the need arises. - 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:
	  ○ All Rosebery drill hole data is stored in an SQL database on the Rosebery server, which is backed up at regular intervals.
	  ○ 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.
	  ○ Assays are loaded into the database from spreadsheets provided from the assay laboratory.
	- The measures described above ensure that transcription or data entry errors are minimised.
	- 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:
	  ○ Validation routines in the database check for overlapping sample, lithological and alteration intervals as well as general validation requirements.
	- Bulk data is imported into buffer tables and must be validated before being uploaded to the master database.

Site visits	- The 2015 Competent Person has visited the Rosebery site several times in the past year.
Geological interpretation	- Geological interpretation is based on massive to semi-massive base metal sulphide lenses located within the Rosebery host sequence, which are readily identified in drill core logs and mapping of underground mine development. - The occurrence of massive to semi-massive sulphide in drill core is further verified by assay results of >5% Pb+Zn. - Mineralised lenses are independently interpreted as they are physically

<table>
<tr><td></td><td>separate and differ in mineralogy, chemical and metallurgical characteristics.
• The broad stratiform nature and continuity of the lenses is confirmed by underground mapping of the mining and development exposures.
• Geological interpretations are modelled as wireframe solids and are peer reviewed within the Rosebery Mine Geology department.
• No alternative interpretations have been generated for the Rosebery mineralisation and geology, however, recent 3D structural analysis by consultant Ian Neilson (Model Earth) in 2014 has enhanced the understanding of the ore body kinematics.
• Interpretive geological models are updated and cross checked with digitised underground photogrammetry of newly developed faces, walls and backs. In general, the photography of development drives confirms major geological boundary placement within 1 m-2 m of the interpreted location.
• The Competent Person has reviewed the Mineral Resource estimation work and concluded that improvements in the geological domaining are required to further improve the local estimation of the Mineral Resource models, which will be undertaken for the 2016 Mineral Resource.</td></tr>
<tr><td>Dimensions</td><td>• The Rosebery mineral deposit extends from 400E-1800E, 2500N to -1100N, 3400RL-1900RL (Rosebery Mine grid co-ordinates) and is currently open to the north, south and at depth. Individual lenses vary in size from a few hundred metres up to 1000 m along strike and/or down-dip.
• The minimum, maximum and average thickness of the mineralised lenses are as follows:
<table>
<tr><td>Lens</td><td>Minimum (m)</td><td>Maximum (m)</td><td>Mean (m)</td></tr>
<tr><td>K</td><td>0.2</td><td>36.1</td><td>6.2</td></tr>
<tr><td>N</td><td>0.3</td><td>16.4</td><td>3.8</td></tr>
<tr><td>P</td><td>0.2</td><td>11.8</td><td>2.7</td></tr>
<tr><td>WXY</td><td>0.3</td><td>20.5</td><td>3.3</td></tr>
</table></td></tr>
<tr><td>Estimation and modelling techniques</td><td>• Block modelling was done with Multiple Indicator Kriging (MIK) as the primary method, with Ordinary Kriging (OK) as a secondary process where MIK was unable to provide an estimate. Datamine was the software product used for estimation with the following key parameters:
o Domains coded by wireframes based on geological interpretation.
o Parent block size of 5mE x 5mN x 5mRL with no sub-cells allowed. Block size approximates one third of drill hole spacing in northing and RL, and approximates development drive width in easting.
o Domains and variables are written to parent blocks then independently estimated by MIK.
o Discretisation is 2x2x2 (XYZ) for a total of 8 points per block.
o Minimum sample search number is 12; maximum number is 24.
o Octant search methods were not used.
o Assays were composited as close to 1 m lengths as possible by Datamine using command ‘compdh’ mode=1 (all samples included).
o Separate block models were created for K and P lenses, with an</td></tr>
</table>

	amalgamated block model for NWX and Y lenses. o Grade capping was not applied. o Two MIK estimation passes were run with varying search ellipse sizes calculated from variogram range analysis. The first pass search ellipse is restricted to prevent the smearing of localised high grade samples within the individual lenses – ellipse dimensions of 25 m x 25 m x 5 m were used. The second search ellipse has a radius factor varying from 0 to 3 based on variogram ranges to infill the blocks that failed to estimate in the first pass. Unestimated blocks after MIK pass 2 were estimated by two pass OK. Finally, the remaining unestimated blocks are allocated default values and are flagged as such within the block model. • All recoverable elements of economic interest to the Rosebery Operation (Zn, Pb, Cu, Ag, Au) have been estimated. The metallurgical recovery of these variables is accounted for in the net smelter return after royalties (NSRAR) calculation, which is included in the block model. • The NSRAR calculation incorporates current and future mining, commercial and financial assumptions which provides a standardised relative estimate of resource value for mining decisions. • No deleterious element or non-grade variables of economic significance have been identified – hence they are not estimated. • Block modelling with Datamine software has been used to estimate Mineral Resources for Rosebery since 1999. • Various estimation techniques have been used historically at Rosebery including polygonal, nearest neighbour, inverse distance and ordinary kriging. Current models for K, N, P, W, X, and Y lenses are estimated using ordinary kriging; historical models for R/S, V and were also estimated by ordinary kriging; historical models for M/Q U and T used the inverse distance method. • No dilution or recovery factors are taken into account during the estimation of Mineral Resources. • Selective Mining Units (SMU) are not defined or applied to the Rosebery Mineral Resource estimate. • All metals are estimated individually, and no correlation between metals is assumed or used for estimation purposes. • Block model validation was conducted by the following processes: o Visual inspections for true fit with the high and low grade wireframes (to check for correct placement of blocks). o Block model to wireframe volume differences are checked. o Visual comparison of block model grades against composite file grades. o Global statistical comparison of the estimated block model grades against the declustered composite statistics and raw length-weighted data. o Swath plots were generated and checked for K, P, and NWXY lenses. • Monthly reconciliation reports compare block model performance against mill concentrate output. In general, these figures are within 10% of each other throughout the year. Final model kriged estimates for annual depleted volumes are checked against back-calculated tonnes and grade results from the previous 24 months of mill output. On average, for 2014 the model estimates were within 108% of the mill claim with 100% tonnage; for 2015 the model estimates were within 99% of the mill claim with 103% tonnage.
Moisture	• Tonnes have been calculated on a dry basis. • No moisture calculations or assumptions are made in the modelling process. • The NSRAR calculation uses a moisture content of 8% to convert dry

	concentrate to wet concentrate tonnes in order to calculate freight costs from mill to port.
Cut-off parameters	• Net Smelter Return After Royalties (NSRAR) 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. • Rosebery and South Hercules Mineral Resources were reported above an A$179/t NSRAR block cut-off grade. • The reporting cut-off grades are in line with MMG's policy on reporting of Mineral Resources which is prospective for future economic extraction.
Mining factors or assumptions	• Mined voids (stope shapes and development drive as-builts) are depleted from the final Mineral Resource estimate as at 30 June 2015. • 2.5 m – 5 m skins in mined out areas have been applied with respect to remnant pillars and unrecovered Mineral Resource; such material remains in the Mineral Resource model but has not been reported. • Mineral Resource block models are used as the basis for physical mining schedules and to calculate derived NPV for the life of asset (LoA). As such, consideration is made of the reasonable prospect of eventual economic extraction in relation to current and future economic parameters.
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 NSRAR calculation, which is used as the cut-off grade for the Mineral Resource estimate. 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. • Net Acid Generating (NAG) 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 2015 Resource block models. Prospects to include a NAG estimate are being considered for future models.
Bulk density	• Rosebery uses an empirical formula 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 kriging has been completed. The formula applied is: ○ SG =2.65+0.0560Pb%+0.0181Zn%+0.0005Cu%+0.0504Fe% • A study was 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. • The Rosebery mineralisation does not contain significant voids or porosity

	and the DBD measurement does not attempt to account for any porosity. • No assumptions apart from the use of the empirical formula are applied to the different materials in the Rosebery Mineral Resource estimate.
Classification	o Mineral Resource classifications were based on sample spacing determined by Datamine macro and then subsequent Mineral Resource classification wireframes were configured to assure the following criteria were met: o **Measured Mineral Resources:** The sample spacing is no more than 22.5 m. At this spacing for Rosebery the knowledge of the geology and grade continuity is confirmed, and distribution of the mineralisation is sufficiently known to allow detailed drive and stope planning. o **Indicated Mineral Resources:** The sample spacing is between 22.5 m and 47.25 m. Historically, drill hole evidence at this spacing at Rosebery is enough to assume geological and grade continuity. o **Inferred Mineral Resources:** The sample spacing is between 47.25 m and 81 m. At Rosebery, implied continuity can be demonstrated 50% of the time at this sample spacing. o The Mineral Resource classification reflects the Competent Persons view on the confidence and uncertainty of the Rosebery Mineral Resource.
Audits or reviews	• A self-assessment of all 2015 Resource modelling was completed by D. Evans and M. Aheimer in July 2015 using a standardised MMG template. • An in-house peer review of modelling and estimation techniques was held in the MMG Melbourne office between 6-8th May 2015. • No fatal errors were detected in the review or the self-assessment. Suggestions were made to further improve procedures or to assess suitability of alternate modelling approaches. • An internal peer review was carried out in September 2015. It was recommended that an independent external review be undertaken prior to work on the 2016 estimate commencing. This was recommended due to the significant change in estimation methodology compared to previous years. • An external audit of the Rosebery 2015 Resource will be carried out before the 2016 Mineral Resource estimate commences. • The Competent Person has reviewed the Mineral Resource estimation work with the finding that the global estimated Mineral Resources are reasonable for Rosebery. This review concluded that improvements in the geological domaining will further improve the local estimation of the Mineral Resource models.
Discussion of relative accuracy/ confidence	• There is high geological confidence that the spatial location, continuity and estimated grades of the modelled lenses within the Mineral Resource are representative. The sheet-like, lensoidal nature of mineralisation historically exhibited is expected to be present in the remaining Mineral Resource at a global scale. Minor local variations are expected to occur on a sub-20 m scale that are not detectable by the current drill spacing. • Twelve month rolling reconciliation figures for the Ore Reserves model to Metallurgical Balance are within 10% for all metals on an annual basis, suggesting that the Rosebery Mineral Resource estimation process is sound.

	The 2015 Mineral Resource Model has also been reconciled against the 2013-2014 global mined tonnes and grade and found to be within 10% of mill output figures. - Mining and development mapping by mine geologists shows good spatial correlation between interpreted mineralised boundaries and actual geology. - The combination of Mineral Resource 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. - The issue of whether remnant mineralisation in the upper and lower levels should still remain within the stated Resource has been addressed by removal from the stated Mineral Resource. - The accuracy and confidence of this Mineral Resource estimate is considered suitable for public reporting by the Competent Person.

9.4 Ore Reserves – Rosebery

9.4.1 Results

The 2015 Rosebery Ore Reserve are summarised in Table 34.

Table 34 2015 Rosebery Ore Reserve tonnage and grade (as at 30 June 2015)

Rosebery Ore Reserve											
							Contained Metal				
	Tonnes (Mt)	Zinc (%)	Copper (%)	Lead (%)	Silver (g/t)	Gold (g/t)	**Zinc ('000 t)**	**Copper ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**	**Gold (Moz)**
Proved	4.8	8.3	0.2	2.6	85	1.0	395	12	126	13	0.2
Probable	2.6	6.0	0.2	2.4	100	1.0	158	5	63	8	0.1
Total	**7.4**	**7.4**	**0.2**	**2.6**	**91**	**1.0**	**553**	**17**	**190**	**22**	**0.2**
Total Contained Metal							**553**	**17**	**190**	**22**	**0.2**

Cut-off grade is based on Net Smelter Return after Royalties (NSRAR), expressed as a dollar value of A$179/t.

Contained metal does not imply recoverable metal.

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

9.5 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 35 complies with the 2012 JORC Code requirements specified by "Table-1 Section 4" of the Code.

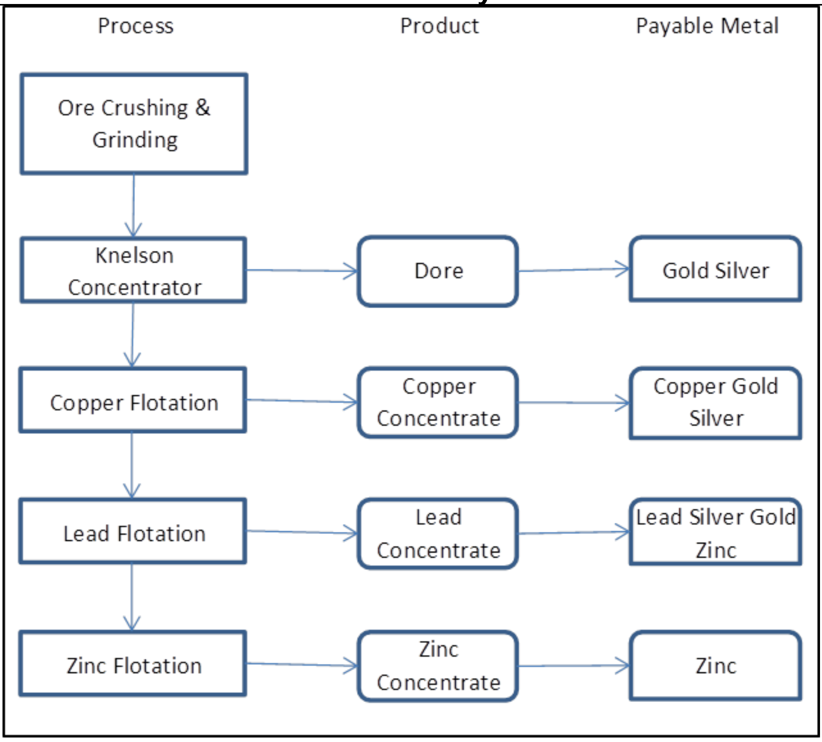
Table 35 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Rosebery Ore Reserve 2015

Assessment	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of those Mineral Resources modified to produce the Ore Reserves. • The Ore Reserves estimate has been generated by applying the metallurgical, social, environmental and financial aspects of the operations (the modifying factors) on that portion of the Mineral Resource Estimate, classified as "Measured" and "Indicated".
Site visits	• The Competent Person, Karel Steyn is a full time employee based in Melbourne Group Offices and has visited the site regularly during this year.
Study status	• The mine is an operating site with on-going detailed Life-of-Mine planning.
Cut-off parameters	• The 2015 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 2015 Budget. Processing recoveries are based on historical performance. The calculation and application of the cut-off grades and the Net Smelter Return After Royalities (NSRAR) are described in this report • Costs used in assisting with the setting of the cut-off value used for the Ore Reserve estimation were based on the Budget 2015. This was considered to be a more accurate representation of the costs. **Break-even Cut-Off grade Estimate (BCOG)** The generalised formula utilised for the determination of the BCOG is as follows: $\text{BCOG} = \frac{\text{Operating Costs (Mine + Mill + G\&A) + Sustain Capital}}{\text{Production Tonnes}}$ The BCOG will be utilised to identify economic stopes available for inclusion in the Ore Reserves and ultimately production. BCOG includes all fixed and variable cost mining, processing, administration (G&A) and sustaining capital. The cut-off or NSRAR is to the Mine gate. The NSRAR also includes the royalty and metallurgical recoveries. The BCOG Operating cost of the gate is AU$179/t. **Stope Cut-Off grade (SCOG)** 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 stope Ore Reserve if he is satisfied with the economic evaluation.

Assessment	Commentary
	The SCOG Operating cost of the gate is AU$166/t. **Mineral Resource Cut-Off grade Estimates (RCOG)** The generalised formula utilised for the determination of the RCOG is as follows: $\text{RCOG} = \frac{\text{Operating Costs (Mine + Mill + G\&A) + Sustain Capital}}{\text{Production Tonnes}}$ The RCOG is utilised in determining the Mineral Resource available; its estimation is similar to the estimation of the BCOG except it utilises a different set of metal prices and exchange rates. The RCOG operating cost of the gate is AU$179/t.
Mining factors or assumptions	• Designs are generated around the Mineral Resource and evaluated against cut-off grade to convert the Mineral Resource to an Ore Reserve. The following assumptions are used to generate this design: ○ 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. The nature of this mining sequence causes fluctuations in the grade profile of the short term schedules. 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, WU & X, and already developed WL & Y levels. Within large areas of CRF a local pillar was left every 60 m for stability purposes. ○ Stope design is carried out using the Mineable Shape Optimiser (MSO) process within the CAE Mining Software (Studio 5DP) with stope cut-off factor of AU$165/t NSRAR, allowing for a 7.8% 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 15 m or 20 m. Stope strike lengths of 15 m were used in W and X Lens while the others lenses used 20 m. The height was set to 20 m (floor to floor) and the minimum mining width to 3.5 m. This was adjusted to 4.65 m for horisontal width to allow for the low dip of the ore body and to achieve the 3.5 m true width. ○ A Mining Recovery factor of 95% and Unplanned Dilution of 105% was also applied to Ore Tonnes mined. ○ Access to the orebody is through a decline 5.5 mH x 5.5 mW at a 1:7 gradient. The standoff distance from orebody

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>and stoping footwall and major infrastructure; stockpiles, vent rises, escape-ways, declines and ancillary development are 50 m.
○ Inferred material is not included in the mine design process for Ore Reserve Reporting.
○ Production of ore is in Measured Mineral Resource only with grade control drilling programs scheduled to convert Indicated Mineral Resource prior to development or stoping activities. Development is strictly under Survey control. Geological development control is currently not implemented at Rosebery.
○ The current primary ventilation system supplies approximately 686 m3/s of air to the underground mine, which allow extraction from the multiple ore lenses designed.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>• 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 the 2015 Budget. As the site is currently mine constrained, mining and processing physicals are the same rate. Minimal stockpiles are maintained for the mill and the 2015 Budget (830 kt) is less than 2013 productions (897 kt). Additionally, the processing plant has a nameplate capacity of 1.0 Mtpa.
<table><tr><th>Tonnes (t)</th><th>Zinc (%)</th><th>Lead (%)</th><th>Copper (%)</th><th>Gold (g/t)</th><th>Silver (g/t)</th><th>Fe (%)</th></tr><tr><td>830,189</td><td>11.6</td><td>3.6</td><td>0.44</td><td>1.5</td><td>107.8</td><td>8.1</td></tr></table>
• From the mill there are four saleable products generated being:
○ Gold Doré
○ Copper Concentrate
○ Zinc Concentrate
○ Lead Concentrate
• The flow chart below outlines the basic process, products and payable metals.</td></tr>
</table>

Assessment	Commentary



- In 2014 there was a shift in the recovery estimation away from linear estimation to the use of regression analysis. The regression analysis was based on mill data from November 2013 and May 2014 and used for the 2014 Ore Reserve. The same regression analysis has been used in 2015, with an update expected later in the year, ready for the 2016 process.
- Based on the budget grades for 2015, as indicated in the above table, the following recoveries are predicted. These have been calculated by inputting the grades into the NSRAR calculator spreadsheet to determine the relevant recoveries. These are summarised in the below table.

Product	Zinc (%)	Lead (%)	Copper (%)	Gold (g/t)	Silver (g/t)
Dore				**20.7**	**0.2**
Copper Concentrate			**67.6**	**42.6**	**37.8**
Lead Concentrate	**4.4**	**78.2**		**14.3**	**44.1**
Zinc Concentrate	**89.7**				

Environmental

- Waste water - The waste water management at Rosebery involves collecting all potentially contaminated water including storm water, mine water and mill tailings at the Effluent Treatment Plant (ETP), where lime is added prior to pumping the whole volume of treated water to the Bobadil TSF via the Flume (an open concrete channel flowing under gravity to the TSF). After the final polishing stage, water is subsequently discharged to the Pieman River.
- The ETP hydraulic capacity is approximately 600 l/sec and the plant is capable to receive 335 l/sec of site mine water with remaining

Assessment	Commentary
	limited spare capacity of approximately 265 l/sec to treat the site surface rain or storm water. • Environmental legacy sites - There is a range of environmental legacy sites that are indirectly related to Rosebery that are being managed by Group Office. While these are not directly related to the current operations, they are located either on the mine lease or are in the local region. • The historic Hercules and South Hercules area has a large impact on the land area along with major water issues. This area is the most significant "legacy site" for Rosebery management. Smaller historic legacy sites include the Zeehan Smelter site and historic mines numbering at least ten known sites, such as Jupiter's, along with a number of small historic workings. • Waste rock - Waste rock is not currently characterised differently for disposal. Work has commenced to collate information in relation to acid forming potential which is ongoing. The majority of waste rock produced is retained underground and used for stope filling, either as RF or as CRF. Any surplus waste rock is trucked to the surface and unloaded at the current waste rock dump, referred to as Assay Creek and 4 Level. • Approval for a new waste rock dump (EPN 8815/1) is currently awaiting for approval from regulatory authorities. The proposed dump will be located on 3 Levels within the existing open pit. Aspects of the approval for this waste dump have implications in regards to the management of potentially acid forming (PAF) waste rock, hence the work being undertaken to understand the acid forming potential of waste rock. • A further area of work being investigated which will impact on waste rock management is the utilisation of paste fill. This will impact on the amount of waste rock required for filling and the amount brought to the surface.
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 Supply Authority for the region. This is managed by the Commercial Department and all responsibilities (such as notification to change in supply by either party) are detailed in this contract. 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 to 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 5,500 ML and 1,647 ML

Assessment	Commentary
	respectively. As part of the asset hand back to Cradle Mountain Water, the Stitt River allocation is due to be handed back 2015. This will leave Lake Pieman as the sole source of fresh water. • In total, the Rosebery Mine operation employs 258 MMG people, covering all aspects of the operation. Within the mining area there are 172 MMG employees and a further 163 contractors. • Primary communication from the Rosebery Mine site is by phone along with surface mobile phone coverage, provided by Telstra. 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 Level. • 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. Ore is hauled out of the mine in a fleet of 55-60 tonne trucks. • The Rosebery 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 540 m3/s of air usefully used in the lower part of the underground mine. The system comprises of three primary fan installations on the surface and two booster fan installations underground. The specifications of these fan installations are detailed below: ○ PSF1 (New NUC)) are 2 x 1800 kW Howden centrifugal fans. Duty is 360 m3/s. ○ PSF 2 (Old NUC) is a single centrifugal fan. Duty is 90 m3/s. ○ PSF 3 (SUC) are 2 x 500 kW Korfmann KGL 2600 mm axial fans in parallel. Duty is 130 m3/s. ○ The 33P Booster fan is a Flaktwoods 550kW 2600 VP axial fan. ○ 19B Booster fans are 1 x Twin 90 kw & 1 x Twin 110 kW CC1454 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. This moved the facilities from the 33P and 35 Level to provide these facilities closer

Assessment	Commentary
	to the current and ongoing working areas. • 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 treatment are placed in a Tailings Storage Facility (TSF) located to the north of Rosebery at Bobadil. This facility has a capacity at current rates through to late 2017. Beyond 2017 there are two options for tailings storage. • Other Rosebery site infrastructure includes mineral processing facilities (mill, concentrator, filtration and rail load-out), and buildings (offices, workshops, change-house).
Costs	• Costs used in assisting with the setting of the cut-off value used for the Ore Reserves estimation were based on the 2015 Budget. Costs were included of Operating and Sustaining Capital. • The commodity price and exchange rate assumptions are supplied by MMG Group Finance. These price assumptions, then apply to the period to which the ore is scheduled to be produced to determine the extracted NSRAR. • 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 NSRAR 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 were supplied by MMG Group Finance and have been included in the NSRAR calculation. • The formulas and assumptions used in the NSRAR calculation are based on the historical data provided by the Rosebery Metallurgy Department. • The economic evaluation was carried out to verify whether the stope designed using the NSRAR cut-off generate economic revenue. The mining physicals required to access and mine individual stopes were determined during the mine design process. The cost assumptions were applied to the mining physicals and the revenue was calculated by multiplying the recovered ore tonnes by the applicable NSRAR value. The profitable and marginal stopes were included in the Ore Reserve.
Market assessment	• MMG's long-term view on global consumption of metals are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. • The long-term outlook for zinc will be determined by the ability of miners to offset the impact of scheduled mine closures and growing demand. Future zinc supply will likely come from lower-grade, higher-cost underground mines as current reserves are depleted. Continued growth in the construction, transportation and infrastructure sectors especially in the developing economies,

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>will support solid demand for zinc in the medium to long term.</td></tr>
<tr><td>Economic</td><td>

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 realistic long-term metal prices.

The mine is profitable and life-of-asset economic modelling shows that the Ore Reserves are economic. The life of Mine (LOM) 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.

</td></tr>
<tr><td>Social</td><td>

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 2014/2015 reporting period, a total of ten complaints was received. Only two complaints were related to noise issues. All complaints were fully investigated, resolved and closed out.

</td></tr>
<tr><td>Other</td><td>

Rosebery is currently going through an approval process for an expansion of the tails storage facility (Dam 2/5). The permit is being sought for stages 1 and 2, however there is potential for a stage 3, should the Rosebery Mineral Resources be further expanded. The table below outlines the expected tails storage capacities at the start of 2016 for Bobadil and at 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>1,000,000</td><td>Forecast 2017 Q1 completion</td></tr>
<tr><td>Bobadil</td><td>TBD</td><td>Expect additional 1 Mt from tails plan refinement</td></tr>
<tr><td>Dam 2/5 – Stage 1</td><td>3,000,000</td><td></td></tr>
<tr><td>Dam 2/5 – Stage 2</td><td>2,000,000</td><td></td></tr>
<tr><td>Dam 2/5 – Stage 3</td><td>TBD</td><td></td></tr>
<tr><td>Bobadil Future</td><td>TBD</td><td></td></tr>
</table>

The status of approvals and communication with the government is provided in the table below. All projects have been approved in a

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

<table>
<tr><th>Assessment</th><th>Commentary</th></tr>
<tr><td></td><td>
timely manner with close communication maintained by the Environmental Protection Agency (EPA).
<table>
<tr><th>NATURE OF CHANGE</th><th>STATUS</th><th>DETAILS</th></tr>
<tr><td>Stage 9b embankment lift</td><td>Planned</td><td>Has been approved by the EPA and Assessment Committee for Dams Construction (ACDC) through Environmental Protection Notice 9139/1 and is scheduled to commence in October 2015.,</td></tr>
<tr><td>New tailings storage facility to replace Bobadil</td><td>Planned</td><td>MMG is currently preparing a Development Proposal and Environmental Management Plan (DPEMP) which will be submitted to the EPA for approval prior to construction of the facility.</td></tr>
</table>
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 Assay</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 along</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>
* <i>Assumes available for backfill activities prior to closure, WRD location not impacting required infrastructure</i>
</td></tr>
<tr><td>Classification</td><td>
Ore Reserve classification follows the Mineral Resource 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 Reserve.
Where stopes contain more than one Mineral Resource category, then the individual classification components have been reported as outlined above.
</td></tr>
<tr><td>Audit or reviews</td><td>
The Geology Department at Rosebery reviewed the NSRAR script to ensure correct operation for each model. Detail has been added to the script and a background document to track when and who has made changes.
Mineral Resource block models were validated during the design
</td></tr>
</table>

Assessment	Commentary
	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 Reserve estimate for Rosebery include:
	  ○ Tailing storage plans to be finalised to support tailings storage beyond 2017. Studies are well advanced on this project.
	  ○ Potentially acid forming (PAF) waste rock characterisation to support a proposed waste rock dump.
	  ○ 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.
	• 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 Reserve is based on a local scale and is suitable as a local estimate.
	  ○ The Probable Ore Reserve 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 Reserve is based on the results of an operating mine. The confidence in the estimate is compared with actual production data.

9.5.1 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 36.

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 36 Contributing Experts – Rosebery Ore Reserves

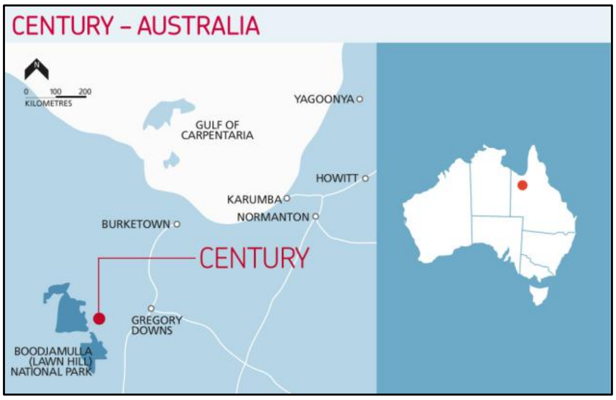
EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Mark Aheimer MMG	Mineral Resource Model
Elmar van Breda MMG	Senior Surveyor
Johannes Holtzhausen MMG	Senior Engineer, Ventilation
Robert Whitworth AMC	Senior Mining Engineer
Mark Slater MSC	Consultant Mining Engineer
Claire Beresford MMG	Senior Business Analyst

10 CENTURY OPERATION

10.1 Introduction and Setting

The Century zinc-lead-silver mine is located in the remote lower Gulf region of north-west Queensland, approximately 250 km north-west of Mount Isa. The mine is 100% owned by MMG Limited and has been in operation since 1999. Century operations comprises of two sites: the mine at Lawn Hill, and associated concentrate dewatering and ship-loading facilities at Karumba on the Gulf of Carpentaria connected by a 300 km slurry pipeline. Century's regional location is shown in Figure 10-1.

Figure 10-1 Century Mine location



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

As at November 30, 2015 all Century Ore has been processed and all Mineral Resources and Ore Reserves were depleted. This change will be reflected in the 2016 MMG Mineral Resource and Ore Reserve statement.

10.2 Mineral Resources - Century

10.2.1 Results

The Century Mineral Resource estimate was estimated by Quantitative Group (QG) in 2014 utilising geological interpretations and data provided by MMG geologists. The 2015 Mineral Resource has used depleted 2014 Mineral Resource block model.

The Mineral Resource at the Century Mine is summarised in Table 37. The Century Mine Open Pit Mineral Resource is reported within the current final pit shell design. The Century Mineral Resource is inclusive of the Ore Reserve.

Table 37 2015 Century Mineral Resource tonnage and grade (as at 30 June 2015)

Century Mineral Resources							
					Contained Metal		
Century Pit	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000)**	**Lead ('000)**	**Silver (Moz)**
Measured	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Indicated	0.7	9.7	1.4	36	68	10	0.8
Inferred	0.0	0.0	0.0	0	0	0	0.0
Total	**0.7**	**9.7**	**1.4**	**36**	**68**	**10**	**0.8**
Stockpiles							
Measured	1.9	6.1	1.7	42	118	32	2.7
Total	**1.9**	**6.1**	**1.7**	**42**	**118**	**32**	**2.7**
Total Contained Metal					**186**	**42.0**	**3.5**

Cut-off grade is based on a 3.5% ZnEq (zinc equivalent cut-off). Where ZnEq equals Zn+Pb * 1.19 contained in the Century final pit shell.

Contained metal does not imply recoverable metal.

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

10.3 Mineral Resources JORC 2012 Assessment and Reporting Criteria

The following information contained in Table 38 is provided to comply with the "Table-1 Section 1 to 3" of the Code.

Table 38 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Century Mineral Resource 2015

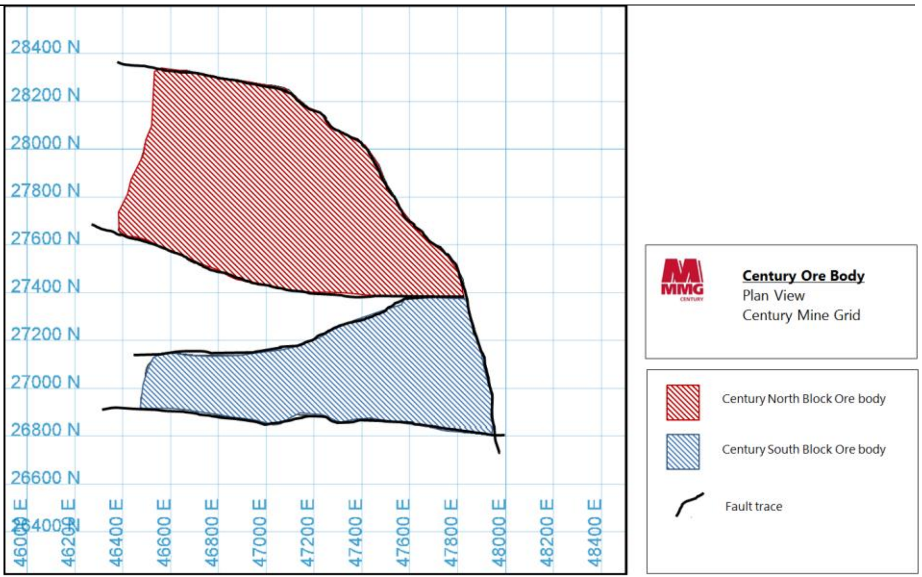
Criteria	Status
Section 1 Sampling Techniques and Data	
Sampling techniques	• The majority of assay results for the Mineral Resource are from diamond drill (DD) core HQ and NQ size drill core. A total of 84 reverse circulation (RC) and percussion drilling drill holes intersected the deposit and were used for their geological information, however assay samples from RC or PD drilling were not considered robust and therefore were not used in grade estimation. Samples from RC and PD drilling are typically 1 m or 2 m in length and are not able to achieve the resolution required in the mineralised sequence to avoid smearing of grade across Unit boundaries. • Channel samples were excluded from the Mineral Resource estimate due to concerns about their accuracy and repeatability. The concern about using channel samples was derived from the low repeatability displayed by channel sample field duplicate data. • Measures taken to ensure sample representivity include collection, and analysis of field duplicates.
Drilling techniques	• The geological interpretation was based on 534 DD holes (NQ and HQ size core), of which 436 contained valid intersections and were used in grade estimation.
Drill sample recovery	• Recovery was recorded for all DD holes. The difference between the length of the recorded drilling interval and the recovered length of the physical core was defined as core loss. Where core loss = 0, core recovery = 100%. Drill core recovery within the mineralised sequence is approximately 100%. • Drilling process was 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. • No bias was identified between sample recovery and grade as recovery approximated 100%.
Logging	• All DD drill core has been geologically logged by experienced geologists with the following data recorded: recovery, RQD, breaks per metre (BPM), stratigraphy, lithology, structure, colour, weathering, mineral proportion estimate and sample intervals. The stratigraphy was logged as developed by Solid Geology (2002). Logs were then uploaded into the GBIS database. The logging has been undertaken to an appropriate level of detail to support Mineral Resource estimation, mining studies and metallurgical studies. • Logging captured both qualitative descriptions such as geological details (e.g. stratigraphy) with some quantitative data (e.g. mineral proportions). Drill core was photographed and catalogued in both wet and dry states as a record of the drill hole. • In total 91,724 m of drill core was logged and used in the 2013 Mineral Resource model update (and subsequent 2015 report).

Sub-sampling techniques and sample preparation	• Half-core and quarter-core samples were taken from DD core using a diamond core saw. • No RC samples were used in the Mineral Resource. • Sampling for the initial drill holes up to LH083 was completed on 1 m intervals through the entire mineralised sequence. From LH084 onwards sampling was typically completed within the known geological Unit boundaries. Between lithological boundaries the nominal sample length was 1 m. All samples were dried, crushed and pulverised to appropriate specifications. • A limited number of duplicates for each sub-sample were plotted on Thompson-Howarth plots and analysed for precision, with no suggestion of any material bias being present in the process. • No grind size checks were carried out throughout the process. The final grind size of the Century Ultra fine circuit is P80 of 6 µm, far exceeding the standard grind size of any commercial laboratory. Grind size was not considered material to the analytical results and the downstream application. • The sample types, nature, quality and sample preparation techniques are considered appropriate for the style of the Century mineralisation (sediment hosted base metal) by the Competent Person.
Quality of assay data and laboratory tests	• Samples in all drilling programs were analysed at high quality commercial laboratories which included AMDEL, Analabs, Genalysis and ALS. Samples were analysed at the Century Mine Lab between 1999 and 2013. Analytical methods include Atomic Absorption Spectrometry (AAS), Induced Coupled Plasma Optical Emission Spectroscopy (ICP-OES) and Leco furnace methods. X-ray Fluorescence was also used during the latter assaying programs. All analyses completed are total or near total digest methods. • No geophysical tools, spectrometers or handheld XRF instruments have been used in the analysis of samples external to the commercial laboratories for the estimation of Mineral Resources. • The QAQC controls for all sets of drilling campaigns included: o The insertion of laboratory certified standard reference materials matrix matched to the Century mineralisation, o Duplicate samples of quarter core, with the exception of the 2013 campaign, o Duplicate samples of 5 mm splits (Century laboratory only), o Duplicate samples of pulverised splits were taken to assess variability of the tertiary sub sample, o Submission of pulps to off-site "umpire" laboratory, o Repeats of assayed pulps. • Analysis of the above quality controls suggests that no material bias exists in the assay database, sampling or sample preparation procedures.
Verification of sampling and assaying	• Verification by independent or alternative company personnel was not undertaken at the time of drilling. • No twinning of drill holes was undertaken due to the low variability across

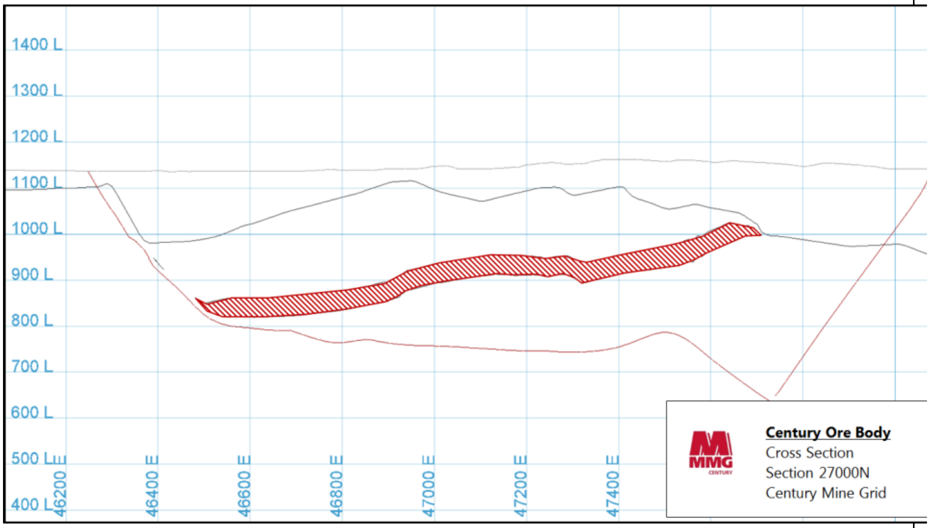
	the deposit. • Geologists and field assistants worked alongside the drill rig, ensuring drill compliance to the MMG QA/QC procedure with regards to sample collection. Core logging data was recorded in Excel spread sheets (pre-made templates with drop down option lists) by site geologists. • All assay results were verified against logging. Drill holes were also viewed in 3D modelling software to confirm no gross errors. All data was reviewed by site geologists or the Group Exploration GISDB (Geographic Information Systems and Database) team before data was entered into the Micromine GBIS system. All required fields were completed prior to import. All data entries and edits are fully auditable. All QAQC data sets from the Century site Lab and ALS Brisbane were reviewed by site geologists who confirmed suitability of data for use in Mineral Resource modelling. • Sample or assay data has not been adjusted in any way. • Where data was deemed invalid or unverifiable it was excluded from the Mineral Resource estimation.
Location of data points	• Collar co-ordinates of all drill holes were determined to an accuracy of 0.1 m in all directions by a licensed surveyor and stored in Century Mine Grid. Downhole surveys were taken at 30 m intervals for all inclined drill holes and 30% to 40% of vertical holes using single-shot Eastman camera equipment. • Century Mine Grid was originally an Exploration grid based on an interpolated position from 1:100,000 map sheet 6660 Lawn Hill and a compass orientation, i.e. truncated AMG. Subsequent formal survey determined that an exact truncated grid required a shift of about 18 m west and 152 m north and a swing of 0°20'. Too many drill holes had been referred to this grid so it was retained and adopted as the master grid for the project. Levels are (AHD+1000). • Annual aerial surveys are carried out at the mine. Topographic surfaces are updated using point data derived from DGPS, and then converted into Century Mine Grid. Topographic control is considered to be of a high standard.
Data spacing and distribution	• In-fill drilling was carried out in 2013 reducing the drill hole spacing in the remaining Mineral Resource to between 30 m and 40 m. The objective of this programme was to increase the confidence in the local estimate. Century mine does not routinely collect grade control data. • Drill hole spacing in the historic Mineral Resource is 50 m – 70 m and is deemed appropriate for a global Mineral Resource and Ore Reserve estimates. Intra-hole variability, particularly in unit thickness and continuity mean tonnage reconciliations may be poor locally – however grade estimates remain robust based on the low variability within the deposit overall. The EFB drill hole spacing is approximately 20 m x 20 m. Due to the EFB's provenance as part of the Century deposit, grade distribution is assumed to be relatively homogenous across units. The increased resolution of the drilling down to 20 m spacing has aided the improved definition of unit thickness and extent in this area. • DD samples are not composited prior to being sent to the laboratory,

	however the nominal sample length is generally 1 m.
Orientation of data in relation to geological structure	• The Century mineralised sequence dips at between 5 and 25 degrees over most of the deposit area, with dips up to 70 degrees around the margins. The EFB mineralisation dips approximately 65 degrees toward the north-north-west. The majority of drill holes are vertical with inclined drill holes targeted at the more steeply dipping zones. • Drilling orientation is not considered to have introduced any sampling bias.
Sample security	• Measures to provide sample security include: o Sample intervals were logged and recorded by geologists, and sample numbers assigned to each interval. DD samples were cut by field assistants and placed into calico bags, clearly marked with the relevant sample number in water proof ink. o The individual calico bags were placed into poly-woven sacks which were tied with either metal wire ties or plastic cable ties. o Samples were transported by commercial carriers to off-site laboratories. Sample sheets were entered into the Geological database and a corresponding sample inventory was attached to the freight. o Upon receipt, the laboratory staff completed a sample receipt report, noting any missing or damaged samples. o No further measures were undertaken.
Audits or reviews	• In 2002 and 2003 Snowden completed reviews on the data quality and QAQC procedures for geology sample data from 1999 to 2003. • In 1996 Mining and Resource Technologies (MRT) completed data validation and review of the initial drilling completed by CZL from 1990 to 1995. • Data identified in the above reviews that did not pass validation requirements was not flagged as 'Century Resource' within the database, and thus not used in the modelling and estimation process.
Section 2 Reporting of Exploration Results	
Mineral tenement and land tenure status	• The Century Mine Lease is ML 90045/90058. o Tenure is held by MMG Century for 40 years from 19th September, 1997. o Lease expiry date is 19th September, 2037. o The Gulf Communities Agreement (GCA) was negotiated between Pasminco Century Mine Limited, the Queensland Government and three native title groups - the Waanyi, Mingginda, and Gkuthaarn and Kikatj - under the right to negotiate provisions of the Native Title Act 1993 (Cth). This agreement, which was signed in May 1997, came into effect in September 1997 when Pasminco purchased the Century Mine project from Rio Tinto. o The GCA specifies particular benefits and obligations on each party, which exist throughout the life of the mining project. In negotiating the GCA, Traditional Owners intended for the mine to contribute to the social and economic development of the Gulf while protecting

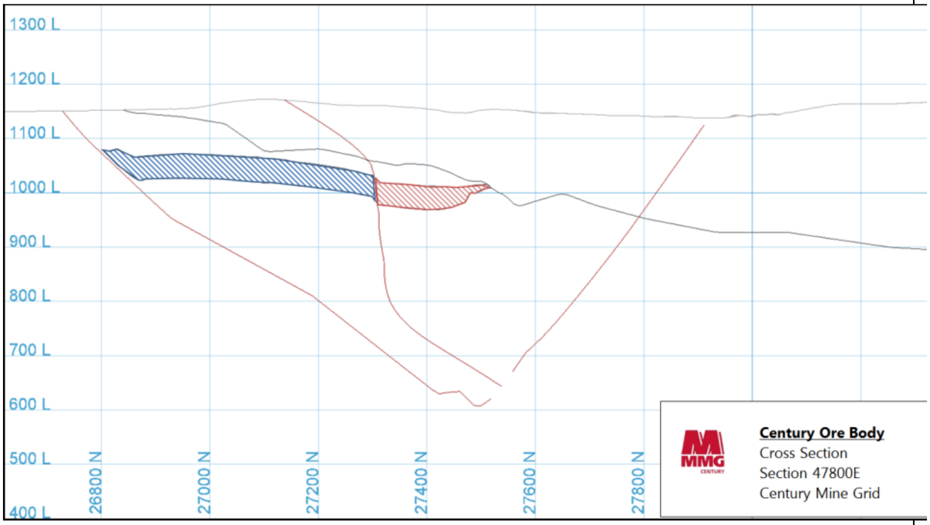
	and promoting cultural heritage. o All activities undertaken are subject to the conditions of the Environmental Authority EPML00888813, issued by the Queensland Department of Environment and Heritage Protection. • There are no known impediments to operating in the area.
Exploration done by other parties	• Significant exploration has been completed by various companies and individuals in the known Burketown mineral field over 100 years since the initial discoveries of lead and silver mineralisation.
Geology	• The deposit is hosted within the Lawn Hill Formation, a Middle Proterozoic sequence of shale, siltstone and sandstone overlain by younger Cambrian limestone. • Structurally, the deposit is located within the Page Creek syncline and is terminated to the east by Cambrian limestone and faults associated with the Termite Range Fault. Magazine Hill Fault and Nikki's Fault define the southern and northern boundaries respectively. • The western boundary is truncated by Cambrian limestone and by the present day surface at the Discovery Hill gossan. • The mineralisation is divided into northern and southern blocks by the north dipping normal Pandora's Fault. • Mineralisation in the deposit is strata bound and confined by stratigraphy, which is well understood and well defined throughout the deposit.
Drill hole information	• No individual drill hole is material to the Mineral Resource estimate 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.
Relationship between mineralisation widths and intercept lengths	• No significant mineralisation from exploration drilling was intercepted. The drilling results are excluded from this report as they are not considered Material by the Competent Person. • Most drilling was drilled vertical with inclined drill holes targeted at the more steeply dipping zones in order to achieve intersections as close to orthogonal as possible. 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. • Mineralisation true widths are captured by interpreted mineralisation 3D wireframes.
Diagrams	• The following diagrams illustrate the geometry of the Century Mineralisation



Century Deposit - Plan View



Section 27000N(CMG) looking North



Section 47800E(CMG) looking West

<table>
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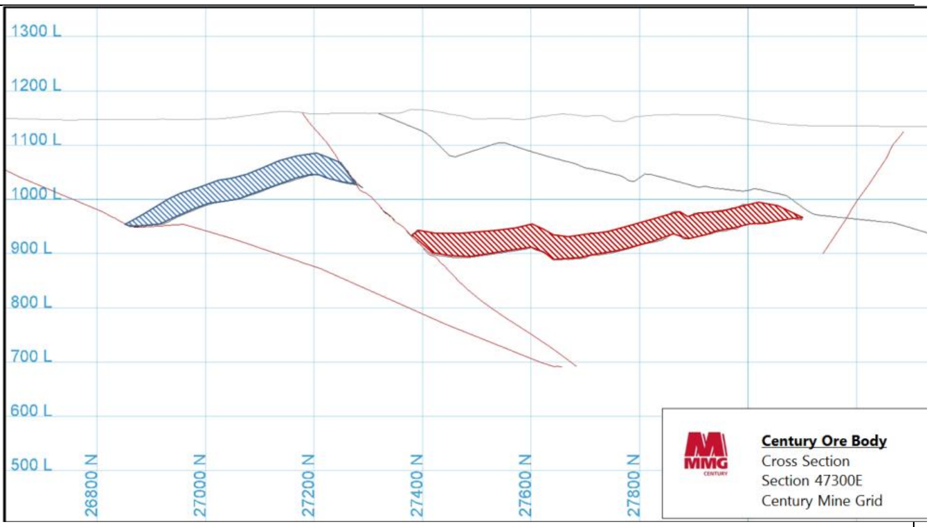
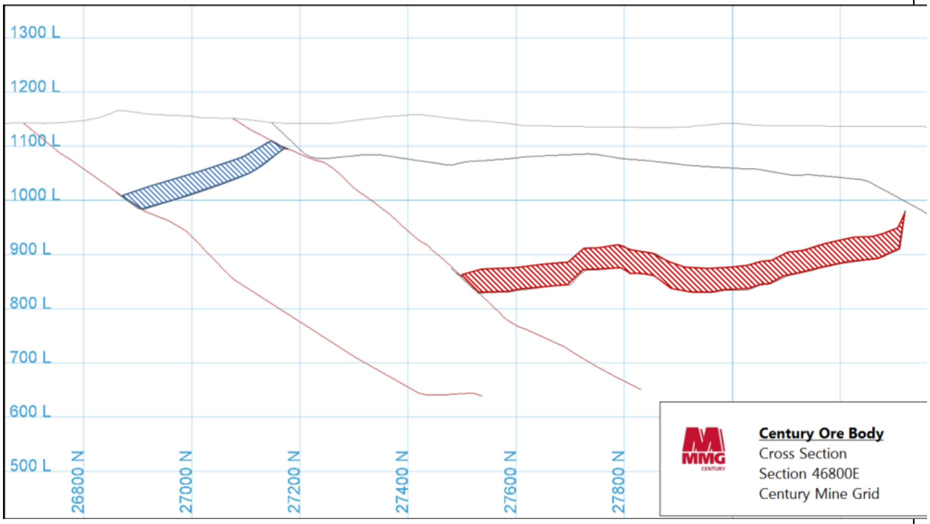


Section 47300E(CMG) looking West



Section 46800E(CMG) looking West
</td></tr>
<tr><td>Balanced reporting</td><td>

- This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
</td></tr>
<tr><td>Other substantive exploration data</td><td>

- This is a Mineral Resource Statement and is not a report on exploration results hence no additional information is provided for this section.
</td></tr>
<tr><td>Further work</td><td>

- Exploration has ceased on the Century Mine Lease.
</td></tr>
<tr><td colspan="2">Section 3 Estimating and Reporting of Mineral Resources</td></tr>
<tr><td>Database integrity</td><td>

- All drilling, sampling, assay, density and geological data previously stored in Microsoft Access databases was migrated to a central GBIS database in 2011, this was the source for all drilling data used in the 2014 Mineral Resource.
- The geology database was validated and audited by independent parties; Mining and Resource Technologies (MRT) in 1996 and Snowden in 2002 and 2003 prior to the migration.
- All data was entered manually into Excel spread-sheets with look up tables, and then uploaded into GBIS.
</td></tr>
</table>

	Post 2011 data – - Qualitative logging was carried out into standardised Microsoft Excel logging spread-sheets with drop down logging codes for each variable. The logging geologist then completes a commentary for the relevant section which should correspond with the logging codes for the interval. Where there is inconsistency the commentary information is prioritised. - Assay data transcriptions errors are eliminated by the use of an automated results receipt process which imports data directory from the laboratory to the database. All changes and updates within the GBIS database are fully auditable, and traceable. - Partial data may not be imported to the database. Mandatory fields must be completed to allow import to occur. All results must correspond to identical sample dispatch numbers to be accepted. - Data used in the Mineral Resource has passed a number of validation checks both visual and software related prior to use in the Mineral Resource.
Site visits	- The Competent Person is a full time employee of the Century Mining Operation and as such is regularly present at the site, is fully conversant with the geology and has a detailed understanding of the sample collection process, modelling process and mining methods employed at Century Mine.
Geological interpretation	- The confidence in the global geological interpretation is high. Confidence at a local scale is reduced due to the large number of fault offsets and the distance of interpolation between drill holes. - The interpretation of the deposit geology was based on all available drilling information at the time of model generation and down-hole dip-metre information was used to verify structural interpretations and geophysical probing of blast holes in the mineralised sequence. This information was used in the geological interpretation along with mapping data collected during mining. - No alternate interpretations have been considered. The Century deposit is well understood and is in the final year of production. - The geological interpretation is constrained by several faults (Termite Range Fault, Magazine Hill Fault and Nikki's Fault) described in the Geology section above. In addition the eastern margin of the orebody is bound by the overlying Cambrian Limestones. - There are also several modelled internal structures that displace the ore by various amounts including: o Geckos Fault o Rayners Fault o Homers Fault - The faults mentioned above have an effect the continuity of both grade and geology have been used to bound the grade estimates through the use of block coding. - Due to the stratiform nature of the deposit all estimates are confined to their relative stratigraphic units through matching composite and block codes. The wireframe horizons for the units define hard boundaries for

block generation and estimation.

- Interpretation and construction of stratigraphic surfaces bounding the pre-defined Upper Ore Zone, Lower Ore Zone, the Interburden Waste unit and the 'Marginal' 165, 155 and 145 units was used for the estimation.

Century Simplified Stratigraphic Column with Units

Major unit	Stratigraphic Unit Detailed	Typical thickness (m)	Description
	100		WASTE - Interbedded siltstone and shale
	140	2.5	ORE/WASTE - Interbedded shale and siltstone
	145	4-9	WASTE - Interbedded siltstone and shale
Marginal Ore Zone (~10)	150	1	ORE/WASTE - Thinly bedded shale
	155	2	WASTE - Medium bedded siltstone
	160	0.7	ORE - Laminated black shale
	165	2	WASTE - Thickly bedded brown siltstone
	170	0.5	ORE - Laminated black shale
	175	1	WASTE - Grey medium bedded siltstone.
	180	0.7	ORE - Shale band
	185	0.5	WASTE - Interbedded grey and brown siltstone.
Upper Ore Zone (~9)	190	0.4	ORE - Small band of laminated black shale
	195	0.4	WASTE - Dark grey to pale brown medium bedded siltstone.
	200	4	ORE - Laminated shale
	311	3	ORE/WASTE -Thin bedded shale and siltstone.
Interburden (~5.5m)	312		ORE/WASTE -Thinly bedded siltstone
	320	4	WASTE - Thickly bedded brown siltstone
Lower Ore Zone (10m)	410	3.5	ORE -Thick bedded shale
	420	1.2	WASTE - Carbonaceous sideritic mudstone
	430	2.2	ORE -Thickly bedded shale
	440	0.5	WASTE - Thin brown siltstone
	450	6	ORE -Thickly bedded shale and mudstones

Dimensions

- The Century deposit is a synformal, tabular, stratiform, ore body with a thickness of approximately 40 m. It is fault bound to the north and south, and truncated by erosional unconformities in the east and west. The

	mineralised units extend 1500 m north to south, and 1500 m east to west. Mineralisation outcrops at surface, and extends to a maximum depth of 310 m below the natural topographic surface.

Estimation and modelling techniques	• The Century Mineral Resources was completed in Maptek Vulcan mining software. Key assumptions and parameters used in the estimation:
	  ○ Grades were estimated using Ordinary Kriging independently for Zn, Pb, Ag, Fe, Mn, S and C (total and organic) for each unit of "ore" and "waste". Ordinary Kriging is considered an appropriate technique for estimating the Century Mineral Resource.
	  ○ Each mineralised zone was Kriged into a two dimensional model using 20 m x 20 m block centres. Zone thickness was represented by varying block heights. The model was coded by "ore" and "waste" stratigraphic members. "Ore" and "Waste" members were composited and treated separately which results in co-located "ore" and "waste" intersections within each unit. The proportion of "ore" and "waste" is calculated for each unit.
	  ○ Variography was completed in two-dimensions by using only the easting and northing of the samples in Isatis software.
	  ○ Quantitative Kriging Neighborhood Analysis (QKNA) to determine estimation search parameters were completed.
	  ○ Densities and sulphide mineralogy were calculated stochiometrically from the estimated Pb, Zn, S and Fe estimates.
	  ○ Grades were estimated for the following elements: Zn, Pb, Ag, Fe, Mn, and S.
	  ○ A one pass search strategy was employed. All blocks were estimated in the first pass.
	  ○ All estimates were validated, then the 2D estimates were migrated to 3D block model.
	• The deposit has comprehensive historical production data to support the Mineral Resource estimate and assumptions.
	• No assumptions were made regarding recovery of by-products within the Mineral Resource Estimate.
	• Non-carbonate carbon is the only recognised deleterious element considered at Century Mine. This is managed through the blending of ore types at the crushing and milling stage. No other deleterious elements have been estimated.
	• A block size of 5 m x 5 m x stratigraphic unit thickness was used. This both honours the geometry of the deposit and allows for regularisation to the larger 10 m x 10 m x 3 m SMU for the purpose of Ore Reserve estimation.
	• The deposit has been mined by excavator at an average 3 m bench height since 2000 and has comprehensive historical production data to support the Mineral Resource estimate and assumptions.
	• No correlations between variables have been used in the Mineral Resource estimate.
	• The geological interpretation of the deposit has defined hard estimation

boundaries, using both stratigraphic horizons and also fault surfaces.

- A top-cut (capping) strategy was applied which effected a small proportion of the data. The value of the top-cuts (caps) was decided based on visual examination of histograms to identify the limits of coherent populations.

Top-cut values applied by domain

	Ore						Waste					
unit	Zn%	Agppm	Pb%	Fe%	Mn%	S%	Zn%	Agppm	Pb%	Fe%	Mn%	S%
100								25	0.7		2	
145	12	100	4				5	50	3			5
155	20	200	10					55	5			
165		550	15		1.5			55	3			
200		450	10				10	100	7.5			10
320		100	5				5	25	1.6			5
450		100					7	30	0.9			7
460								25	0.7			

- The block models and estimate has been validated in the following ways:
    - Block estimates were visually validated in Isatis and compared to the sample composites. This allowed for rapid assessment of domain selections, and whether any blocks have not been filled by the search applied. In addition, the search ellipsoid and samples selected were visualised.
    - Global sample statistics were compared to block estimates. This ensured that no global bias exists.
    - A semi-local estimation check was carried out by plotting the average grade of the inputs and outputs in moving window slices (swath plots).

Moisture

- Tonnes have been estimated on a dry basis.

Cut-off parameters

- The Mineral Resource is reported above a 3.5% Zinc Equivalent (ZnEq) grade, where: ZnEq = Zn + (Pb * 1.19) within the Century final pit shell.
- 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

- The deposit will be mined by excavator at an average 3 m bench height, and hauled by Komatsu 630 and 830 trucks respectively.
- No mining dilution has been applied to the Mineral Resource.
- The deposit has been mined by this method since 2000 and has comprehensive historical production data to support the Mineral Resource estimate and assumptions.

Metallurgical factors or assumptions

- Ore from the deposit has been processed since 2000 and has comprehensive historical production data to support the Mineral Resource estimate and assumptions.
- Various stockpiles waiting processing are at a high risk of combusting due to the sulphide content. The material within a combusting stockpile may result in lower metallurgical recoveries.
- No material changes are anticipated with regards to the metallurgical assumptions.

Environmental factors or assumptions	• All activities undertaken are subject to the conditions of the Environmental Authority EPML00888813, issued by the Queensland Department of Environment and Heritage Protection.
Bulk density	• Bulk density was estimated into the block model. The process of estimating bulk density involved calculating the stoichiometric density of composites, applying a correction factor for porosity based on grab sample results and then estimating the corrected stoichiometric density using Ordinary Kriging, which is summarised below: o Select samples that have assay results for all elements required in the stoichiometric equation. o Composite these samples for intervals that have been coded as having valid samples. o Calculate the stoichiometric density for these samples. o Apply correction factor derived from the grab sample bulk density to convert the stoichiometric density into bulk density. o Derive variograms for bulk density for each Unit. o Estimate bulk density into block model. • Due to the low data density across the EFB the average density by unit from the main deposit was assigned using a post script.
Classification	• The Mineral Resource has been classified according to the JORC Code (2012) guidelines and takes into account: drill hole spacing, estimation results, the internal and bounding structures of the deposit, and a considerable production history involving Mineral Resource conversion to Ore Reserves. • The classification of the 2015 Mineral Resource was based on the Competent Persons empirical observations of a large production dataset spanning 14 years from the mine. No statistical evaluation scheme was adopted • Due to the local variability within the model, and smaller available work areas toward the end of mine life no Measured Mineral Resources other than stockpiles were declared. • Within the area of remaining Mineral Resource, the distribution of drill holes analysed for zinc and lead are generally evenly spaced at ~40 m x 40 m. There are no known issues with the data quality, and the estimation quality as quantified by metrics such as slope of regression is high. The greatest risk within the classification lies in the accuracy of the volume model between drill hole intercepts. That said, the confidence in the volume model is sufficient to declare an Indicated Mineral Resource and is well supported by annual production data since 2000. • Confidence in the estimate proximal to Pandora's fault is reduced due to the accuracy of the fault model. Due to the large scale, and irregular nature of Pandora's fault, there is a risk that modelled mineralisation does not exist in some localised areas. There is also potential for mineralisation in this area to be displaced by the intrusive carbonate breccia material present in this area. As such, material within 15 m of Pandora's fault has been classified as Inferred. • The Mineral Resource classification reflects the Competent Person's view on the confidence and uncertainty of the Century Mineral Resource.

Audits or reviews	• MMG has an internal 'Mineral Resource and Ore Reserve Policy' that requires at a minimum external reviews every three years, and internal company review every interim year by MMG representatives reporting findings to a sub-set of the MMG Mineral Resource and Ore Reserve Committee. 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. • Quantitative Group (QG) carried out an independent review of the Century Mineral Resource model in 2012. Based on the recommendations of this review the modelling approach was altered in 2013. The new QG Mineral Resource model is the basis of this report. • The current Mineral Resource model has not been independently audited but has been internally peer reviewed by QG.
Discussion of relative accuracy/ confidence	• The confidence in the global geological interpretation is high. Confidence at a local scale is reduced due to wide drill hole spacing and the large number of fault offsets. Particularly, in the Southern Block of the ore body thickness variations such as thinning, thickening or doubling of the mineralised sequences due to these fault offsets and shear zones have resulted in mining more ore tonnes than estimated in the block model. • The estimation quality of the Mineral Resource as quantified by geostatistical metrics (such as Kriging variance, slope of regression etc.) throughout the deposit is high. • There is comprehensive historical production data to support the Mineral Resource estimate and assumptions. Confidence on both a global and local scale is considered high. • In the Competent Persons opinion, the estimation quality in the remaining Mineral Resource generally supports a classification of Indicated. This level of Mineral Resource classification is easily supported by estimates of the payable variables (zinc, lead and silver), which have high confidence. However, this statement presumes that the geometry of the principal ore units will be reliably identified during the grade control phase prior to mining.

10.4 Ore Reserves – Century

10.4.1 Results

The 2015 Century Ore Reserve are summarised in Table 39.

Table 39 2015 Century Ore Reserve tonnage and grade (as at 30 June 2015)

Century Ore Reserve					Contained Metal		
	Tonnes (Mt)	Zinc (% Zn)	Lead (% Pb)	Silver (g/t Ag)	**Zinc ('000 t)**	**Lead ('000 t)**	**Silver (Moz)**
Proved							
Probable	0.7	8.7	1.1	34	66	8	0.8
Total	**0.7**	**8.7**	**1.1**	**34**	**66**	**8**	**0.8**
Stockpiles							
Proved	1.9	6.1	1.7	42	118	32	2.7
Total	**1.9**	**6.1**	**1.7**	**42**	**118**	**32**	**2.7**
Total Contained Metal					**184**	**40**	**3.5**

Cut-off grade is based on a 4.2% ZnEq (zinc equivalent cut-off).

Contained metal does not imply recoverable metal.

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

10.5 Ore Reserves JORC 2012 Assessment and Reporting Criteria

The following information provided in Table 40 is provided to comply with the "Table-1 Section 4" of the Code.

Table 40 JORC 2012 Code Table 1 Assessment and Reporting Criteria for Century Ore Reserve 2015

Assessment Criteria	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resources are reported inclusive of those Mineral Resources modified to produce the Ore Reserves. • The Ore Reserves are based on the 2013 Mineral Resource model built by QG Pty Ltd utilising geological interpretations and data provided by MMG geologists. The block model was generated in Vulcan software, with grade estimation being carried out in Isatis using the ordinary kriging estimation method. The Mineral Resource estimate is based on significant drilling conducted between 1990 and 2013. • The Competent Person for the Mineral Resource estimate is Claudio Coimbra of MMG Century. • Risk exists with respect to structural complexity potentially causing off-sets of the ore zones. Drilling has however not shown any blank zones in the mineral package. • Further details are discussed in the Mineral Resources Section of this report.
Site visits	• The Competent Person for Ore Reserves works on site on a fly-in-fly-out roster for the last 2.5 years. This enabled direct communication with the contributing experts and clarification of the parameters used in the estimation of the Ore reserves. • The Competent Person was solely responsible for the definition and application of reasonable mining dilution and ore loss parameters, calculation of mining costs, pit optimisation and comparing shell results against design, and Ore Reserve reporting.
Study status	• The mine is an operating site with on-going detailed Life-of-Mine planning. Factors and costs used are based on predicted, current and recent historical values.
Cut-off parameters	• The cut-off grade used for the Ore Reserves estimate is break-even cut-off grade taking into account mining and metallurgical recovery, concentrate transport costs, concentrate treatment and refinery charges and royalties. • The zinc equivalent used to apply a cut-off grade is the equivalent grade of zinc only which is required to generate the same revenue as the combined zinc and lead grades present. • The cut-off grade used for the Ore Reserves estimate is 4.2% ZnEq a 0.9% decrease from the 5.1% ZnEq value used in the 2014 Ore Reserves estimate.
Mining factors or assumptions	• Century has established mining operations with well understood and managed mining risks and mining methods.

Assessment Criteria	Commentary
	• The 2015 Ore Reserves is based on revised pits designed in 2014. After assignment of dilution, any blocks not exceeding a 2015 Net Smelter Return After Royalty (NSRAR) of A$51/t material in-situ (equating to 4.2% ZnEq) were excluded from the Ore Reserves. • A planning ("Ore Reserves") block model was created using the 2013 regularised Mineral Resource model modified with mining dilution and ore loss assumptions. This model was then used in a Whittle software analysis to generate optimal Whittle shells for current parameters (costs, prices, geotechnical and surface topography). • Comparison of the optimal Whittle shell to the 2014 design showed no material difference in both ore and waste, hence re-design was not required. Comparison of the selected optimisation shell (1.0 Revenue Factor) to the current pit designs showed a difference in potential Ore Reserves of less than two per cent. • Geotechnical parameters are well understood from 14 years of mining and managed through the Geotechnical Management Plan, collection of monitoring data and external auditing. Monitoring of the Century open pit has improved with the implementation of slope movement monitoring radar, and the installation of tri-axial geophones in 2013. • The consulting firm Mining One Pty Ltd has been involved since the inception of the Century project and the continuity of the involvement of one of the senior partners has resulted in a high level of external expertise for the slope management. Regular audits of practices and data collected are undertaken on a quarterly basis. • Most of the slopes at the Century pit are in a stable configuration as a result of a combination of careful design, monitoring and sound mining techniques that have been adapted to suit the mining conditions. • The approach used for the 2014 Ore Reserves was generally continued for the 2015 Ore Reserves, and was validated against the site's detailed grade control model which now extends to the end of mining. • A regularised model suitable as an input to mine planning was created by regularising the 2013 Mineral Resource model to a fixed Selective Mining Unit (SMU) of 10x10x3. This SMU was based on the 2013 investigation into site's practical ability to achieve mining selectivity. • The regularised block model contains blocks with information for up to three stratigraphic units, as well as a proportion based "owner" of the block – the unit with the largest contained tonnage becomes the "owner". • This regularised block model was then further manipulated to apply reasonable estimates of internal and external dilution and

Assessment Criteria	**Commentary**
	mining recovery (loss). In general this process involved considering the relative proportions of potential ore and waste within boundary blocks (blocks between defined mineralised and waste sequences) and: o Excluding them from the Ore Reserves if the waste proportion of an "ore" owner block was too high; and o Including some waste "owner" blocks if the proportion of "ore" was substantial. • The final Ore Reserves are reported as the flagged Marginal, Upper and Lower Zones above cut-off, inclusive of dilution and mining loss. • No Inferred Mineral Resources were considered in the Ore Reserves or in the mine plan. • Infrastructure required for the mining of the Ore Reserve is entirely in place and no additional investment is expected. The mining infrastructure at the time of the Ore Reserve estimate included the open pit, in pit dump, ex-pit dumps, ROM pad and other site infrastructure as listed in the Infrastructure section. The mobile mining fleet included 18 Komatsu dump trucks (15 x 830E and 3 x 630E), 2 shovels, 4 excavators, 2 loaders, 4 production drill rigs (251 mm, 2 x 165 mm, 140 mm), 4 water trucks, 3 graders, 4 dozers and a light vehicle fleet. Additional equipment included 2 radar and 1 micro-seismic array for geotechnical monitoring.
Metallurgical factors or assumptions	• The Century mine is an operating entity. The metallurgical process is crushing and grinding followed by sequential flotation to produce separate lead and zinc concentrates. The mine commenced operations in 1997 and is expected to close in late 2015. • Metallurgical factors for Ore Reserves for the remaining mine life were derived from average plant operating data and are: o Lead recovery to Lead Concentrate = 56.7% o Lead assay of Lead Concentrate = 60% o Silver recovery to Lead Concentrate = 8.5% o Zinc recovery to zinc Concentrate = 66.9% o Zinc assay of zinc Concentrate = 56.5% • In general, zinc recoveries are lower than historical values, which reflect the expected impact of declining head grade. • The two main deleterious components of the ore are organic carbon which absorbs reagents readily and floats naturally, and silica which is finely intergrown with the sphalerite. • The strategy for dealing with organic carbon is to operate a pre-flotation stage such that downstream performance is maximised without excessive losses of values to the pre-float concentrate.

Assessment Criteria	**Commentary**
	• The strategy for dealing with silica is to operate an ultrafine grinding circuit which grinds the sphalerite to nominally 6 micron before flotation in multiple cleaning stages. In this way, the silica content of the final zinc concentrate is controlled to less than 5.2% SiO_2. • No required mineral specifications have been identified for this deposit. The grade of the zinc concentrate is dependent on the iron content of the sphalerite, but is typically about 55% Zn which is more than acceptable for downstream processing.
Environmental	• In pit and ex-pit water management is an ongoing issue across wet seasons. Increased dam and pumping capacities have worked toward mitigating this risk based on hydrological forecasts. • Waste rock from the mining operation has been arranged in three (3) ex-pit waste rock dumps. In addition, a large volume of mineralised waste has been stored in-pit. A waste rock management plan has been developed that describes the processes for siting and developing the final landforms. Potentially Acid Forming (PAF) and Non Acid Forming (NAF) waste has been preferentially handled to resist the ingress of water into the dumps and the development of acid mine drainage. Store and release cover systems have been trialed and selected as the means of final encapsulation. • The southern waste rock dump was capped in 2009 and approximately 90% of the encapsulation effort has been completed. Establishment of a vegetation cover is ongoing. Rehabilitation of the western waste dump has commenced. The northern waste dump is still in active use and is considered adequate for life of asset requirements. • Process residues are stored in the on-site Tailings Storage Facility (TSF). The TSF has been progressively constructed and developed in a series of lifts which are now complete, and the remaining capacity is well in excess of the tailings that will be produced from the remaining Ore Reserve. Following the completion of processing, an encapsulation system must be established to retard the migration of oxidation products from inside the mass to the wider environment. The current capping system design is at concept level.
Infrastructure	• The operational infrastructure consists of a single large open pit mine, crushing and concentrator, and a pipeline to the port facilities. Support infrastructure includes maintenance workshops, electricity supply, water, accommodation, communications and a site airport. All of the infrastructure is established and is considered sufficient to support the Ore Reserves. • The concentrate pipeline and concentrate storage shed was refurbished in 2012 and will be in operational state past Century's current LOA of 2016.

Assessment Criteria	**Commentary**
Costs	• The mining costs were derived from recent actual costs. The methodology involved determining months where the vast majority of mining occurred in a defined location (Stage 10 at an elevation) and comparing that with another month when the vast majority was at a lower elevation (Stage 8). This enabled the mining cost to be defined for a specific elevation, and a relationship for incremental mining cost to be established. • The processing and site administration cost estimates that were used were based on the 2014 Century Strategic Review. These estimates are not less than recent actual costs. • Assumptions of commodity prices and exchange rates are provided by the MMG Finance department. • Treatment and refining charges, and penalties for failure to meet specification, are negotiated between MMG Marketing and customer smelters. Negotiated terms are included in written contracts. • Transportation charges are based on recent actual costs. • Very limited capital cost was included due to the short remaining life of the mine. • Penalties for inclusion of deleterious elements in saleable concentrates (predominately silica) are included in the determination of the Ore Reserves as per written contracts. • Queensland State Government royalties payable are prescribed by the Minerals Resources Regulation and are based on a variable ad valorem rate between 2.5% to 5.0% depending on metal prices, advised quarterly and calculated on payable metal. They are published by the Queensland Government Department of Mines and Energy and can be found at the web-site of the "Office of State Revenue". For the prices used in the Ore Reserves estimation at the time of evaluation, the relevant rates were 3.2% for zinc, 4.70% for lead, and 5.00% for silver.
Revenue factors	• The realised revenue from the ore is expressed using a calculated Net Smelter Return After Royalty (NSRAR). The metal prices and exchange rates used for the 2014 Ore Reserves estimate are:
Market assessment	• LOM sales contracts were established at commencement of production and are still in place. In addition, a ready market exists for zinc and lead concentrates. • MMG's long-term view on global metal consumption is

	Ore Reserve
Zn (US$/lb)	1.18
PB (US$/lb)	1.12
Ag (US$/oz)	21.1
AUD:USD	0.85

Assessment Criteria	**Commentary**
	consumption of metals are expected to increase as developing economies undertake further industrialisation and economic growth prospects improve in advanced economies. • The long-term outlook for zinc will be determined by the ability of miners to offset the impact of scheduled mine closures, here at Century and Lisheen, and growing demand. Future zinc supply will likely come from lower-grade, higher-cost underground mines as current reserves are depleted. Continued growth in the construction, transportation and infrastructure sectors especially in the developing economies, will support solid demand for zinc in the medium to long term.
Economic	• The economics of the remaining Ore Reserves are robust, as costs are based on current and recent historical values, and revenues are based on ultra-short term forecasts. The financial plan is profitable and shows a healthy cash flow. The total operating cost of sales (before depreciation and amortisation) equates to approximately 64% of the total net sales value. MMG uses a discount rate appropriate to the size and nature of the organisation and deposit.
Social	• The project enjoys strong local support with significant continuing contributions to the local communities and economy. The main regional community of Lawn Hill supports mining developments.
Other	• There are no identified material naturally occurring risks other than those already discussed in this table such as wet season and related hydrological risks. • Century has contracts in place for the sale of concentrates and the provision of most materials and supplies for the operation. • Century operates under Mining Licenses 90045 and 90058 issued by the Minister for Mines and Energy for the State of Queensland, which are valid until 2037. These licenses cover the area incorporating the open pit, waste dumps, concentrator, tailings facility, offices, accommodation facility and airport.
Classification	• The in-pit Probable Ore Reserve estimate is based on the Indicated Mineral Resource estimate after consideration of all mining, metallurgical, social, environmental and financial aspects of the project. The Proved Ore Reserve is based on the estimate of tonnes and grades remaining on pre- and post-crusher stockpiles. • No Inferred Mineral Resource is included in the Ore Reserves. • The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources is 0% (there are no Measured Mineral Resources within the pit design). • The conversion of in-pit Indicated Mineral Resources to Probable Ore Reserves after application of reasonable mining dilution and ore loss assumptions is considered appropriate by the Competent Person.

Assessment Criteria	**Commentary**
Audit or reviews	• QG Pty Ltd (QG) carried out an independent review of the Century Mineral Resource model in 2012. Based on the recommendations of this review the modelling approach was altered in 2013. The new Mineral Resource model is the basis of the 2014 Ore Reserves. • QG subsequently carried out an independent review of the Century Ore Reserve process in 2013, resulting in the use of a regularised Mineral Resource model modified with dilution and ore loss factors for the 2013 Ore Reserve. This process was continued for the 2014 Ore Reserve, using updated dilution and ore loss assumptions.
Discussion of relative accuracy/ confidence	• The Century Ore Reserves are planned to be entirely mined out by late-2015. The relative accuracy and confidence in the Ore Reserve is high over this short time interval and there are no identified material risks to the total Ore Reserve. • The Ore Reserve estimates are at a global scale and no additional close-spaced grade control drilling is applied before mining. However, local spatial adjustments are made from additional monitor holes that are drilled.

10.5.1 Expert Input Table

A number of persons have contributed key inputs to the Ore Reserves determination. These are listed below in Table 41.

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 41 Contributing Experts - Century Ore Reserve

EXPERT PERSON / COMPANY	AREA OF EXPERTISE
Claudio Coimbra, Technical Services Superintendent, MMG Ltd (Century)	Competent person for Mineral Resources and Ore Reserves sign off, Management
Jedda Malone, Superintendent Mine Engineering MMG Ltd (Century)	Mine Engineering, waste dumps, truck haul cycles
Damian O'Donohue, Senior Geologist MMG Ltd (Century)	Geological Block Model, mine-to-mill reconciliation
Damian Sullivan, Senior Business Analyst MMG Ltd (Century)	Site Operating Costs
Roger Wynn, Senior Plant Metallurgist, MMG Ltd (Century)	Metallurgy
Erin Sweeney, Senior Geotechnical Engineer MMG Ltd (Century)	Geotechnical Parameters
Greg Ballarino, Asset & Reliability Strategic Growth Superintendent. MMG Ltd (Century)	Engineering Information
Matthew Lord, Mine Closure Superintendent MMG Ltd (Century)	Environmental
John Maconachie, Senior Surveyor MMG Ltd (Century)	Surveying
David Purdey, Principal Mining Engineer QG Pty Ltd	Cut-off grade optimisation, pit optimisation, mine planning, pit design,
Gavin Marre, Senior Business Analyst MMG Ltd (Melbourne)	Economic Assumptions
Simon Ashenbrenner, Concentrate Marketing Manager, MMG Ltd (Melbourne)	Marketing

11 HIGH LAKE

This Mineral Resource 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.

12 IZOK LAKE

This Mineral Resource 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.

13 AVEBURY

This Mineral Resource 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.

14 EXTERNAL REFERENCES

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