ASX Release

Independence Group

25 October 2013

MINERAL RESOURCES AND ORE RESERVES ESTIMATES AS AT 30 JUNE 2013

Independence Group NL ("Company") (ASX: IGO) is pleased to announce Mineral Resources and Ore Reserves estimates as at 30 June 2013 at its Long nickel operation, Jaguar zinc-copper operation, Stockman copper-zinc project, Karlawinda gold project (Mineral Resources only) in accordance with the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC (2012)], set out in the attached Tables 1 to 4 and 7 to 9, and Appendices A to E.

In addition, the Company provides Mineral Resources and Ore Reserves estimates as at 30 June 2013 for the Tropicana gold project (IGO Share 30%), that are unchanged from previously reported, in accordance with the 2004 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC (2004)], set out in the attached Tables 5 and 6.

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Table 1: Long Nickel Operation – June 2013 Resources (and 2012 comparison)

		Mineral Resource - 30 June 2012			Mineral Resource - 30 June 2013		
		Tonnes	**Ni %**	**Ni Tonnes**	**Tonnes**	**Ni %**	**Ni Tonnes**
Long	Measured	47,000	3.7	1,700	61,000	5.4	3,300
	Indicated	220,000	5.1	11,200	213,000	5.2	11,100
	Inferred	167,000	5.1	8,600	116,000	5.1	5,900
	Sub-Total	**434,000**	**5.0**	**21,500**	**390,000**	**5.2**	**20,300**
Victor South	Measured	-	-	-	-	-	-
	Indicated	53,000	7.3	3,900	212,000	2.4	5,000
	Inferred	34,000	1.5	500	28,000	1.4	400
	Sub-Total	**87,000**	**5.1**	**4,400**	**240,000**	**2.3**	**5,400**
McLeay	Measured	49,000	7.2	3,600	79,000	6.7	5,300
	Indicated	145,000	5.5	7,900	164,000	5.7	9,300
	Inferred	79,000	4.2	3,300	75,000	4.5	3,400
	Sub-Total	**273,000**	**5.4**	**14,800**	**318,000**	**5.6**	**18,000**
Moran	Measured	-	-	-	181,000	6.7	12,200
	Indicated	498,000	7.1	35,300	241,000	7.4	17,700
	Inferred	11,000	5.3	600	11,000	4.5	500
	Sub-Total	**509,000**	**7.0**	**35,900**	**433,000**	**7.0**	**30,400**
GRAND TOTAL		**1,303,000**	**5.9**	**76,600**	**1,381,000**	**5.4**	**74,100**

Notes:
1. Mineral Resources are reported using a 1% Ni Cut-off grade as at 30 June.
2. Excludes Victor South disseminated mineralisation of 175,000t @ 1.3% Ni using a cut-off grade of 0.6% Ni.
3. Mining depletion as at 30 June 2013 has been removed from the 2013 resource estimate.
4. Resources are inclusive of Reserves.
5. Ore tonnes have been rounded to the nearest thousand tonnes and nickel tonnes have been rounded to the nearest hundred tonnes.
6. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
7. JORC (2012) Table 1 Parameters are in Appendix A of this report.

Table 2: Long Nickel Operation – June 2013 Reserves (and 2012 comparison)

		Ore Reserve - 30 June 2012			Ore Reserve - 30 June 2013		
		Tonnes	**Ni %**	**Ni Tonnes**	**Tonnes**	**Ni %**	**Ni Tonnes**
Long	Proven	5,000	3.0	100	45,000	3.1	1,400
	Probable	91,000	2.6	2,400	66,000	2.9	1,900
	Sub-Total	**96,000**	**2.6**	**2,500**	**111,000**	**3.0**	**3,300**
Victor South	Proven	-	-	-	-	-	-
	Probable	55,000	4.2	2,300	20,000	3.9	800
	Sub-Total	**55,000**	**4.2**	**2,300**	**20,000**	**3.9**	**800**
McLeay	Proven	63,000	2.4	1,500	46,000	3.0	1,400
	Probable	139,000	2.8	3,900	70,000	3.6	2,500
	Sub-Total	**202,000**	**2.7**	**5,400**	**116,000**	**3.3**	**3,900**
Moran	Proven	-	-	-	229,000	4.5	10,300
	Probable	768,000	4.1	31,700	405,000	3.9	15,600
	Sub-Total	**768,000**	**4.1**	**31,700**	**634,000**	**4.1**	**25,900**
GRAND TOTAL		**1,121,000**	**3.7**	**41,900**	**881,000**	**3.8**	**33,900**

Notes:
1. Ore Reserves are reported above an economic Ni Cut-off value as at 30 June.
2. A Net Smelter Return (NSR) value of $239 per ore tonne has been used in the evaluation of the 2013 reserve.
3. Mining depletion as at 30 June 2013 has been removed from the 2013 reserve estimate.
4. Ore tonnes have been rounded to the nearest thousand tonnes and nickel tonnes have been rounded to the nearest hundred tonnes.
5. Revenue factor inputs (US$): Ni $18,087/t, Cu $7,694/t. Exchange rate AU$1.00 : US$1.01.
6. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
7. JORC (2012) Table 1 Parameters are in Appendix A of this report.

Table 3: Jaguar Operation – June 2013 Resources (and 2012 comparison)

		Mineral Resource - 30 June 2012					Mineral Resource - 30 June 2013				
		Tonnes	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**	**Tonnes**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**
Jaguar	Measured	429,000	2.5	4.4	61	-	264,000	2.4	3.4	47	-
	Indicated	129,000	1.8	2.6	32	-	181,000	1.8	2.0	28	-
	Inferred	31,000	2.6	2.7	43	-	30,000	2.6	2.7	42	-
	Stockpiles	6,000	1.9	3.7	54	-	-	-	-	-	-
	Sub-Total	**595,000**	**2.3**	**3.9**	**54**	-	**475,000**	**2.2**	**2.8**	**39**	-
Bentley	Measured	-	-	-	-	-	453,000	1.6	17.1	212	1.0
	Indicated	2,118,000	1.7	10.5	125	0.7	1,442,000	1.7	7.9	103	0.6
	Inferred	795,000	2.5	9.6	160	0.9	849,000	2.4	8.4	161	1.0
	Stockpiles	1,000	0.8	6.5	66	0.3	27,000	1.3	11.0	135	0.4
	Sub-Total	**2,914,000**	**1.9**	**10.2**	**134**	**0.7**	**2,771,000**	**1.9**	**9.6**	**139**	**0.8**
Teutonic Bore		Mineral Resource - August 2009					Mineral Resource - August 2009				
	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	946,000	1.7	3.6	65	-	946,000	1.7	3.6	65	-
	Inferred	608,000	1.4	0.7	25	-	608,000	1.4	0.7	25	-
	Sub-Total	**1,554,000**	**1.6**	**2.5**	**49**	-	**1,554,000**	**1.6**	**2.5**	**49**	-
GRAND TOTAL		**5,063,000**	**1.9**	**7.1**	**99**	**-**	**4,800,000**	**1.8**	**6.6**	**100**	**-**

Notes:
1. Teutonic Bore Mineral Resource estimate is as at August 2009 and was previously reported in accordance with the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. It has not been updated since to comply with the JORC Code 2012 on the basis that the information has not materially changed since it was last reported.
2. Mineral Resources include massive sulphide and stringer sulphide mineralisation. Massive sulphide resources are geologically defined, stringer sulphide resources are reported above cut-off grades of 0.5% Cu for Bentley and Jaguar, 0.7% Cu for Teutonic Bore.
3. Block modelling used ordinary kriging grade interpolation methods within wireframes for all elements and density.
4. Mining depletion as at 30 June 2013 has been removed from the 2013 resource estimates for Jaguar and Bentley.
5. Resources are inclusive of Reserves.
6. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
7. JORC (2012) Table 1 Parameters are in Appendices B and C of this report.

Table 4: Jaguar Operation – June 2013 Reserves (and 2012 comparison)

		Ore Reserve - 30 June 2012					Ore Reserve - 30 June 2013				
		Tonnes	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**	**Tonnes**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**
Jaguar	Proved	73,000	1.9	0.5	15	-	20,000	1.7	0.4	15	-
	Probable	6,000	1.5	0.4	10	-	3,000	1.8	0.3	11	-
	Sub-Total	**79,000**	**1.8**	**0.4**	**14**	-	**23,000**	**1.7**	**0.4**	**14**	**-**
Bentley	Proved	-	-	-	-	-	431,000	1.3	13.4	163	0.8
	Probable	2,373,000	1.3	8.5	100	0.5	830,000	1.8	7.7	107	0.6
	Sub-Total	**2,373,000**	**1.3**	**8.5**	**100**	**0.5**	**1,261,000**	**1.6**	**9.6**	**126**	**0.7**
GRAND TOTAL		**2,452,000**	**1.3**	**8.2**	**98**	**-**	**1,284,000**	**1.6**	**9.4**	**124**	**-**

Notes:
1. Cut-off values were based on NSR values of $180 per ore tonne for direct mill feed and $120 per ore tonne for HMS feed.
2. Revenue factor inputs (US$): Cu $7,694/T, Zn $2,270/t, Ag $33/troy oz, Au $1,740/troy oz. Exchange rate AU$1.00 : US$1.01.
3. Metallurgical recoveries – 82% Cu, 53% Ag, 43% Au in Cu concentrate; 83% Zn and 22% Ag in Zn concentrate.
4. Longitudinal sub-level long hole stoping will be used at Bentley and Jaguar.
5. All Measured Resource and associated dilution was classified as Proved Reserve. All Indicated Resource and associated dilution was classified as Probable Reserve.
6. Mining depletion as at 30 June 2013 has been removed from the 2013 reserve estimate.
7. The Bentley underground reserves have decreased by 1.1 million ore tonnes as a result of depletion 313,000 ore tonnes, changes in realised (AUD) metal prices within the net smelter return cut off valuation process and increases in the site cut-off values (2013 $180/t direct feed and $120/t marginal feed versus 2012 $160/t direct feed and $100/t marginal feed) have impacted mainly in the Arnage stringer material resulting in a reduction of 590,000 ore tonnes, changes in resource interpretation along boundaries on ore surfaces, and minor changes in mining method in the Brooklands surface of 16,000 ore tonnes.

8. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
9. JORC (2012) Table 1 Parameters are in Appendices B and C of this report.

Table 5: Tropicana Gold Project – 100% Project (IGO Share 30%) - December 2012 Resources (unchanged at 30 June 2013)

		Mineral Resource – 3 December 2012		
	Classification	**Tonnes (Mt)**	**Au g/t**	**Contained Au (M Oz)**
Open Pit	Measured	29.8	2.12	2.03
	Indicated	74.0	1.90	4.51
	Inferred	5.8	2.57	0.48
	Sub-Total	**109.6**	**1.99**	**7.02**
Underground	Measured	-	-	-
	Indicated	2.4	3.58	0.27
	Inferred	6.1	3.07	0.60
	Sub-Total	**8.5**	**3.21**	**0.87**
Total Tropicana	Measured	29.8	2.12	2.03
	Indicated	76.4	1.95	4.78
	Inferred	11.9	2.83	1.08
GRAND TOTAL		**118.0**	**2.08**	**7.89**

Notes:
1. For the Open Pit Mineral Resource estimate, mineralisation in the Havana, Havana South, Tropicana and Boston Shaker areas was calculated within a US$1,550/oz pit optimisation at an AUD:USD exchange rate of 1.03 (A$1,500/oz).
2. The Open Pit Mineral Resources have been estimated using the geostatistical technique of Uniform Conditioning, using cut-off grades of 0.3g/t Au for Transported and Saprolite material, 0.4g/t Au for Transitional and Fresh material.
3. The Havana Deeps Mineral Resource estimate has been reported outside the US$1,550/oz pit optimisation at a cut-off grade of 1.73g/t Au, which was calculated using a gold price of US$2,000/oz (AUD:USD 1.05) (A$1,896/oz).
4. The Havana Deeps Underground Mineral Resource was estimated using the geostatistical technique of Ordinary Kriging using average drill hole intercepts.
5. Resources are inclusive of Reserves.
6. JV partner and TGP Manager AngloGold Ashanti Limited reports Mineral Resources and Ore Reserves by calendar year. An updated Mineral Resource estimate is expected at the end of calendar 2013. JV partner AngloGold Ashanti Limited has advised that, as of 30 June 2013, there has been no material change to the Mineral Resource as reported in 2012.
7. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
8. JORC (2004) Table 1 Parameters were released in IGO's ASX Release of 4 December 2012.

Table 6: Tropicana Gold Project – 100% Project (IGO Share 30%) - December 2012 Reserves (and June 2011 comparison)

		Ore Reserve – June 2011			Ore Reserve – December 2012		
	Classification	**Tonnes (Mt)**	**Au g/t**	**Contained Au (M Oz)**	**Tonnes (Mt)**	**Au g/t**	**Contained Au (M Oz)**
Open Pit	Proved	25.8	2.30	1.90	25.9	2.28	1.90
	Probable	30.6	2.04	2.01	31.2	1.99	2.00
GRAND TOTAL		**56.4**	**2.16**	**3.91**	**57.1**	**2.12**	**3.90**

Notes:
1. The Proved and Probable Ore Reserve (December 2012) is reported above economic break-even gold cut-off grades of 0.4 g/t for Transported/Upper Saprolite material, 0.5 g/t for Lower Saprolite material, 0.6g/t for Sap-Rock (Transitional) material and 0.7g/t for Fresh material at nominated gold price US$1,300/oz, oil price US$86/barrel and exchange rate 1.02 AUD:USD (equivalent to A$1,278/oz Au).
2. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.

Table 7: Stockman Project – June 2013 Resources (and 2012 comparison)

		Mineral Resource - 30 June 2012					Mineral Resource - 30 June 2013				
Currawong		**Tonnes**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**	**Tonnes**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**
	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	9,548,000	2.0	4.2	42	1.2	9,548,000	2.0	4.2	42	1.2
	Inferred	781,000	1.4	2.2	23	0.5	781,000	1.4	2.2	23	0.5
	Sub-Total	**10,329,000**	**2.0**	**4.0**	**40**	**1.1**	**10,329,000**	**2.0**	**4.0**	**40**	**1.1**
Wilga											
	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	2,987,000	2.0	4.8	31	0.5	2,987,000	2.0	4.8	31	0.5
	Inferred	670,000	3.7	5.5	34	0.4	670,000	3.7	5.5	34	0.4
	Sub-Total	**3,657,000**	**2.3**	**4.9**	**32**	**0.5**[3]	**3,657,000**	**2.3**	**4.9**	**32**	**0.5**[3]
GRAND TOTAL		**13,986,000**	**2.1**	**4.3**	**38**	**1.0**[3]	**13,986,000**	**2.1**	**4.3**	**38**	**1.0**[3]

Notes:
1. All Resources tonnes have been rounded to the nearest one thousand tonnes and grade to the nearest 1/10th percentage/gram per tonne.
2. Mineral Resources include massive sulphide and stringer sulphide mineralisation. Massive sulphide resources are geologically defined, stringer sulphide resources are reported above cut-off grades of 0.5% Cu.
3. Au grades for Wilga are all inferred due to paucity of Au data in historic drilling.
4. Block modelling used ordinary kriging grade interpolation methods within wireframes for all elements and density.
5. Mining depletion as at end of historic mine life (1996) has been removed from the Resource estimate for Wilga.
6. Resources are inclusive of Reserves. The Resource estimate is unchanged since 2012.
7. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
8. JORC (2012) Table 1 Parameters are in Appendix D of this report.

Table 8: Stockman Project – June 2013 Reserves (and 2012 comparison)

		Ore Reserve - 30 June 2012					Ore Reserve - 30 June 2013				
Currawong		**Tonnes (Mt)**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**	**Tonnes (Mt)**	**Cu %**	**Zn %**	**Ag g/t**	**Au g/t**
	Proved	-	-	-	-	-	-	-	-	-	-
	Probable	-	-	-	-	-	7.3	2.2	4.1	40	1.2
	Sub-Total	-	-	-	-	-	**7.3**	**2.2**	**4.1**	**40**	**1.2**
Wilga											
	Proved	-	-	-	-	-	-	-	-	-	-
	Probable	-	-	-	-	-	1.1	2.5	5.3	30	0.5[3]
	Sub-Total	-	-	-	-	-	**1.1**	**2.5**	**5.3**	**30**	**0.5**[3]
GRAND TOTAL		-	-	-	-	-	**8.4**	**2.3**	**4.3**	**39**	**1.1**[3]

Notes:
1. All Reserves tonnes have been rounded to the nearest one hundred thousand tonnes and grade to the nearest 1/10th percentage/gram per tonne.
2. No Ore Reserves were reported in 2012.
3. Gold (Au) grades are Inferred at Wilga due to a paucity of gold assays in historic drilling. Revenue from gold in the Wilga ore was included in the estimation of the Ore Reserve. The contribution to Revenue of this gold was estimated to be $3.84 per gram of gold *in situ*. This inclusion was not material to the value of the mining envelopes considered and did not warrant downgrading of any portion of the Ore Reserve attributable to Wilga. The contribution from Wilga represents 13% of the Total Ore Reserve.
4. Historic mining depletion for Wilga has been removed from the 2013 reserve estimate.
5. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
6. JORC (2012) Table 1 Parameters are in Appendix D of this report.

Table 9: Karlawinda Gold Project – Bibra Prospect - June 2013 Resources (and 2012 comparison)

	Mineral Resource - 30 June 2012			Mineral Resource - 30 June 2013		
Classification	**Tonnes (Mt)**	**Au g/t**	**Contained Au (Oz)**	**Tonnes (Mt)**	**Au g/t**	**Contained Au (Oz)**
Measured	-	-	-	-	-	-
Indicated	-	-	-	-	-	-
Inferred	18.5	1.1	674,300	18.0	1.1	650,800
GRAND TOTAL	**18.5**	**1.1**	**674,300**	**18.0**	**1.1**	**650,800**

Notes:
1. The Mineral Resource estimate was estimated within a conceptual A$1,600/oz Au pit optimisation completed in 2012 and for the area of drill coverage at 100m x 50m spacing or less. Contained gold (oz) figures have been rounded to the nearest one hundred ounces.
2. Mostly RC drilling with 1m cone split samples analysed for Au by 50g fire assay.
3. Mineralisation was wireframed at a cut-off grade of 0.3g/t Au and Mineral Resources were reported above a cut-off grade of 0.5g/t Au.
4. The 2013 Mineral Resource estimate has reduced slightly from the 2012 estimate due to the closer spaced drilling in certain areas allowing refinement of the wireframes and grade interpolation search distances.
5. Block modelling used ordinary kriging grade interpolation methods for composites that were top-cut to 10g/t Au in the supergene zone and 16g/t Au for the remaining mineralisation. Top-cuts are not severe, trimming no greater than 0.5% of the samples.
6. There are no Ore Reserves for Karlawinda.
7. The Competent Persons statement is incorporated in the JORC Code and Forward-Looking Statements section of this report.
8. JORC (2012) Table 1 Parameters are in Appendix E of this report.

JORC Code and Forward looking Statements

General

The information in this report that relates to Exploration Results is based on information compiled by Mr Tim Kennedy. Mr Kennedy is a full-time employee of the Company and is a member of the Australasian Institute of Mining and Metallurgy. Mr Kennedy has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code) and consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Long Resources and Reserve

The information in this report that relates to the Long Nickel Mine's Mineral Resources is based on information compiled by Ms Somealy Sheppard. The information in this report that relates to the Long Nickel Mine's Ore Reserves is based on information compiled by Mr Brett Hartmann. Ms Sheppard is a full-time employee of the Company and is a member of the Australian Institute of Geoscientists. Mr Hartmann is a full-time employee of the Company and is a member of the Australasian Institute of Mining and Metallurgy. Ms Sheppard and Mr Hartmann 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 JORC Code, and consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

Jaguar / Bentley / Teutonic Bore Resources and Reserve

The information in this report that relates to the Jaguar Mineral Resources is based on information compiled by Mr Graham Sweetman. The information in this report that relates to the Bentley Mineral Resources is based on information compiled by Ms Michelle Wild. The information in this report that relates to the Jaguar and Bentley Ore Reserves is based on information compiled by Mr Brett Hartmann. Mr Sweetman, Ms Wild and Mr Hartmann are full-time employees of the Company and are members of the Australasian Institute of Mining and Metallurgy. Mr Sweetman, Ms Wild and Mr Hartmann have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they have undertaken to qualify as Competent Persons as defined in the 2012 edition of the JORC Code. Mr Sweetman, Ms Wild and Mr Hartmann consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

The information in this report that relates to the Teutonic Bore Mineral Resources is based on information compiled by Mr Graham Sweetman. Mr Sweetman is a full-time employee of the Company and a member of the Australasian Institute of Mining and Metallurgy. Mr Sweetman has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he has undertaken to qualify as a Competent Person as defined in the 2004 edition of the JORC Code. Mr Sweetman consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

The Teutonic Bore Mineral Resource estimate is as at August 2009 and was previously reported in accordance with the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. It has not been updated since to comply with the JORC Code 2012 on the basis that the information has not materially changed since it was last reported.

Tropicana Gold Project Resources and Reserve

The information in this report that relates to Tropicana Gold Project Mineral Resources and Ore Reserves was included in the 2012 AngloGold Ashanti Limited Annual Reports, available from the AngloGold Ashanti website (www.anglogoldashanti.com). The information that relates to Mineral Resources was based on information compiled by Mr Mark Kent, a full-time employee of AngloGold Ashanti Australia Limited, who is a member of the Australasian Institute of Mining and Metallurgy. Mr Kent has sufficient experience relevant to the type and style of mineral deposits under consideration, and to the activity which has been undertaken, to qualify as a Competent Person as defined in the 2004 edition of the JORC Code. Mr Kent consented to the release of the Mineral Resource estimate, based on the information in the form and context in which it appeared in the 2012 AngloGold Ashanti Annual Reports. The information that relates to Ore Reserves was based on information compiled by Dr Salih Ramazan, a full-time employee of AngloGold Ashanti Australia Limited, who is a member of the Australasian Institute of Mining and Metallurgy. Dr Ramazan has sufficient experience relevant to the type and style of mineral deposit under consideration, and to the activity which has been undertaken, to qualify as a Competent Person as defined in the 2004 edition of the JORC Code. Dr Ramazan consented to the release of the Ore Reserve in the 2012 AngloGold Ashanti Annual Reports, based on his information, in the form and context in which it appeared in the reports.

Joint Venture partner and Tropicana Gold Project Manager AngloGold Ashanti Limited reports Mineral Resources and Ore Reserves by calendar year. A new Mineral Resource position is expected at the end of calendar 2013. Since the Mineral Resource estimate of 3 December 2012 there has been no further update. The Tropicana Gold Project Mineral Resource estimate and Ore Reserve were previously reported in accordance with the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. They have not been updated since to comply with the JORC Code 2012 on the basis that the information has not materially changed since it was last reported.

Currawong and Wilga Stockman Resources and Reserve

The information in this report that relates to the Stockman Mineral Resources is based on information compiled by Mr Bruce Kendall who is a member of the Australian Institute of Geoscientists and is a full-time employee of the Company. Mr Kendall has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration, and the activity which he is undertaking, to qualify as a Competent Person as defined in the 2012 edition of the JORC Code. Mr Kendall consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

The information in this report that relates to the Stockman Ore Reserve is based on information compiled by Mr Geoff Davidson who is a member of the Australasian Institute of Mining and Metallurgy. Mr Davidson is a consultant working for Mining and Cost Engineering Pty Ltd. Mr Davidson has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration, and the activity which he is undertaking, to qualify as a Competent Person as defined in the 2012 edition of the JORC Code. Mr Davidson consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Karlawinda Resource

Bibra Prospect: The information in this report that relates to the Bibra Prospect Mineral Resources is based on information compiled by Ms Michelle Wild who is a member of The Australasian Institute of Mining and Metallurgy and is a full-time employee of the Company. Ms Wild has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which she is undertaking to qualify as a Competent Person as defined in the 2012 edition of the JORC Code. Ms Wild consents to the inclusion in the report of the matters based on her information in the form and context in which it appears.

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Conventional Diamond drilling is used to test Long, Victor South, McLeay and Moran ore bodies. Recent diamond drill core consisted of four different sizes. HQ, NQ2, LTK-60 and BQTK. Downhole EM and in-drive EM geophysical surveys have been undertaken to assist in targeting of massive sulphide horizons. Sampling was undertaken by half coring to logged geological intervals using an automatic core saw. Maximum sample length is 1.1m and minimum sample length was 0.1m for all core sizes. Sample lengths did not cross geological intervals. Core was cut into half core to give sample weight of approximately 3.2kg. All geological contacts between the footwall basalt and hanging wall ultramafics, with or without the presence of sulphides, were sampled. Sample intervals extend at least 5m beyond the sulphide zone (greater than 1% nickel grade) within the footwall and hanging wall geological contact positions. Samples were crushed and pulverised (total prep) to produce sub-samples of 400mg for analysis by mixed four acid digest, followed by ICP-OES analysis. Densities were determined using Archimedes water immersion technique.
Drilling techniques	Historical surface drill holes were drilled with percussion RC pre-collars and NQ diamond tails. Recent diamond drill core consisted of four different sizes. HQ (core diameter 63.5mm) holes are drilled where bad ground is expected, and the hole is often completed with a smaller NQ2 core diameter core (core diameter 50.6mm). Drilling also consisted of LTK-60 (core diameter 43.9mm) and BQTK core sizes (core diameter 40.7mm).
Drill sample recovery	Diamond core was logged and recorded in the database. Intervals of core loss are logged as geological units with a code of 'CLOSS'. Intervals of partial core recovery are rare, but are noted in comments for both the sample and geology logs. Overall recoveries are >95% and there are no core loss issues or significant sample recovery problems. Intervals of core loss were not included in the sample intervals. All recent drilling is completed using underground diamond drill holes with high (>95%) core recovery. Diamond core was reconstructed into continuous runs where possible and each interval identified on the core and the depths checked against the depth given on the core blocks. Rod counts are marked on additional core blocks routinely completed by the drill crew. Core losses are marked on additional core blocks marking the start of core loss and end of core loss intervals, by the drill crew. HQ drill core was used in areas of bad ground to assist in core recovery. No relationship between sample recovery and grade has been established for the Long, Victor South, McLeay and Moran mineralisation. The mineralisation is located in very competent fresh material so any loss of fine material would be negligible.
Logging	Geotechnical logging was captured on all recent diamond drill holes for recovery, RQD, and number of fractures (per interval). The information is captured in the main database. Logging of drill samples recorded lithology, mineralogy, mineralisation, veins, alteration minerals, contact type. Recent core samples were photographed wet and the images stored in the main database. The drill samples were logged qualitatively in full for all samples.
Sub-sampling techniques and sample preparation	All samples were cut in half using an automatic core saw cutter. All core samples were collected from the same side of the core. Extremely broken core is sampled by visually picking a representative sample consisting of half of the rock fragments. It is unknown how historical RC samples were collected. No RC samples were collected in recent drilling data and no RC data were used for grade interpolation. The core samples were totally crushed in a jaw crusher to a nominal particle size of 6mm then fine crushed in a Boyd crusher to a nominal size of 2mm. A sub-sample of approximately 750g is split out via a rotary divider (the rotary divider is adjustable so that consistent-sized splits can be taken for pulverising, regardless of original sample weights). The sample is then pulverised in a ring mill. A sub-sample of 100g is taken from the pulverised, homogenised sub-sample; this sub-sample is retained as the 'pulp'. An assay sample of 400mg is taken from the pulp for mixed four acid digest and then ICP-AES analysis. Sample preparation checks for grain size were carried out by the contract laboratories as part of its internal checks to ensure the grind size of 90% passing 75 microns. Greater than 90% of all sizing tests met acceptable limits. Field QC is through the use of certified reference material as assay standards inserted at irregular intervals and blank core samples inserted after massive sulphide mineralisation and at irregular intervals. The insertion rate is 1 in 10 blank samples and 1 in 20 standard samples.

Criteria	Commentary
	The performance of the blank results was of concern due to 49% of blank samples returning results above the detection limit of 100ppm nickel. A new contract laboratory was engaged to undertake the sample analysis and has returned 82% results within acceptable limits. Work is ongoing with the current laboratory to reduce contamination through the crushing and pulverising stages. Results of standards and blanks from each batch are scrutinised at the time they are reported, and compared with expected values. Variation outside two standard deviations of the expected result is reported to the lab for checking, and re-assaying if required. In-house QAQC reports are produced quarterly and yearly to examine variability in standards and blanks performance and reliability. Diamond quarter core samples are taken for field duplicates and submitted to the laboratory as separate batches. Results were compared to check for repeatability. Fourteen out of 51 samples returned values outside 20% precision limits. The half core, sampled at 0.1m to 1.1m sample intervals was considered to be appropriate to correctly represent the sulphide mineralisation based on the style of dominantly massive and matrix sulphides, the thickness and consistency of the intersections, the sample methodology and percent value assay range for the primary elements.
Quality of assay data and laboratory tests	The analytical techniques used a 400mg sub sample digested in mixed 4 acid digest (Nitric Acid, Perchloric Acid, Hydrochloric Acid and Hydrofluoric Acid). The digest commences with the samples at room temperature and after thirty minutes the beakers are transferred to a hotplate which heats the digest solution to 200°C. The digest solution is reduced until the solution is reduced to a dry, solid, state. This process takes approximately four hours. The dry, powdery, material which remains is soluble in Hydrochloric Acid and is ready for the next stage. The beaker is then removed from the hot plate and Hydrochloric Acid is added. The beaker is then returned to a hotplate, this time operating at 100°C. This “leach back” stage ensures all solids are dissolved back into solution. The beaker is then removed from the hotplate and allowed to cool. De-iodised water is then added to the beaker to bring the volume up of the solution up to a standard 18ml and the solution is then transferred to a test tube, where the volume is checked again and if necessary adjusted. This solution is vigorously agitated, so that solution is fully homogenised. This “Primary Digest Liquor solution” is diluted on a 1:1 basis. Included in the diluent are two rare elements, which are used as “internal standards” - Yttrium (Y) and Ytterbium (Yb). The ICP-OES analysis is run for either four (production drilling) or nine elements (exploration drilling). The four element suite with detection limits is: Ni (10ppm), Cu (10ppm), As (10ppm), S (100ppm). The nine element suite is: As (10ppm), Co (10ppm), Cr (20ppm plus the possibility of incomplete digestion), S (100ppm), Cu (5ppm), Fe (100ppm), Mg (100ppm), Ni (10ppm), Zn (10ppm). No geophysical tool was used to determine element concentrations used in the resource estimate or reporting of exploration results. Sample preparation checks for grain size were carried out by the contract laboratories as part of its internal checks to ensure the grind size of 90% passing 75 microns. Greater than 90% of all sizing tests met acceptable limits. The performance of the blank samples submitted to the previous contract laboratory was of concern due to 38 out of 77 blanks returning values above the detection limit of 100ppm nickel. A new contract laboratory has returned 125 out of 153 results within acceptable limits. Work is ongoing with the current laboratory to reduce contamination through the crushing and pulverising stages. Samples from three holes were duplicated and processed. The core sample was quarter cored and submitted to the contract laboratory as a separate batch. Greater than 95% of the duplicate samples met acceptable limits. No umpire labs were used. No precision checks have been implemented.
Verification of sampling and assaying	Due to the high visibility of mineralisation, significant intersections in diamond core were visually verified following lithological logging of core samples and after laboratory analysis, by IGO geologists. Core photos and visual checks from remaining half core samples were randomly checked. No drill holes were twinned. Primary data were collected using an Excel template on laptop computers using look-up codes. The information was transferred into Maxwell Geoservices Access Database “DataShed” front end with SQL2000 database server backend. There was no adjustment to assay data. Assay results are submitted from the laboratory via email in CSV and PDF files. Original Assay files are archived digitally in the company computer network. CSV files are imported into the DataShed database through a database importing protocol.
Location of data points	The planned drill collar for underground diamond drill holes are laid out by marking the back-sight and fore-sight pins drilled in the walls of the mine development by the Company Surveyor using a Viva TS15 Total Station Theodolite considered to be accurate to 0.002m. The collar position is later picked up locating the exact position of the drill hole. The collar coordinates are stored in a database. Historical downhole surveys were completed using Eastman and Reflex camera and recent down hole surveys were taken using an Electronic Reflex Ez-Trac down hole survey tool by the Diamond drilling contractors. Holes were down hole surveyed with multi-shot surveys (6m intervals) at the completion of the hole. Single-shot surveys were progressively taken as the hole was drilled to maintain planned drill direction at 15m, and 30m intervals. Stated accuracy of the Electronic Reflex Ez-Trac down hole survey tool is 0.35 degrees on azimuth and 0.25 degrees on Dip. All down hole surveys were stored in the database and de-surveyed as curvilinear projections down the drill hole

Criteria	Commentary
	trace. No gyroscopic validation of down hole surveys was undertaken in the drilling from July 2012 to December 2012, but validation of the surveys with the SMART TEM geophysical probe was completed. No significant survey problems were identified. Recent underground drill holes are within mine development with established survey wall stations located a minimum of 10m to a maximum of 30m intervals along the mine development. The grid system is MGA_GDA94, Zone52. The resource is calculated in Local Grid (KNO-Grid). It is a non-linear projection of MGA co-ordinates. All collars are captured in Local Grid. North-South Local Grid is -1 degrees off Magnetic North declination. MGA co-ordinates are generated by automated scripts within the database. The deposits are located at least 300m below surface. No topographic data are used in the resource estimation.
Data spacing and distribution	Diamond drill spacing at Long, Victor South and McLeay deposits is on a nominal 20m northing with 10m easting drill spacing with 5m by 5m closer-spaced drilling. Moran is on a nominal 40m northing with 10m easting drill spacing with some up to 20m by 10m closer-spaced drilling. The data spacing and distribution is considered to be sufficient to establish the degree of geological and grade continuity to support the Mineral Resource classification applied under the 2012 JORC Code. Sample compositing has not been applied to the drill core.
Orientation of data in relation to geological structure	Drill holes are generally angled near perpendicular to the Long, Victor South McLeay and Moran ore bodies. Hole collars are fanned off sections but kept to near true width as possible. Grade control holes (holes drilled within the ore bodies and within the ore drives) which were drilled up dip or down dip of the ore bodies were utilised to determine footwall or hangingwall geometry only. Some holes were drilled up dip or down dip of the ore bodies due to unfavourable geometries of the drill rig location and the ore bodies. These drill holes were utilised to determine footwall or hangingwall geometry only and the assay results were not used for later estimation of grade.
Sample security	Core samples are stored on site and delivered by IGO personnel to ALS in Kalgoorlie. Whilst in storage the samples are kept in a fenced and locked yard on site. ALS has a batch tracking system that allows IGO staff to track progress of batches of samples from delivery to submission of results. Half core is kept for reference is stored in a fenced and locked yard on site. The location and photographs of the core samples are stored on a regular basis in the database.
Audits or reviews	The sampling techniques and data are collected and managed by IGO staff geologists familiar with the local rock-types and data collection process established over 13 years, with IGO and previously through WMC Resources The major rock-types of the area are visually distinct from each other in drill core, there are no major inconsistencies or errors in the logging of lithology or mineralised zones. The database is audited annually by IGO staff and is considered to be of sufficient quality to carry out resource estimation.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	All resources lie within mining tenements own by Independence Group NL, except for M15/1515 which forms a part of a Joint Venture Agreement with St Ives Gold Mining Co. Pty Ltd (SIGM). The agreement allows Independence Group NL (IGO) to mine and explore for nickel on the leases. SIGM is paid a royalty based on Ore Recovered under a “Ore Tolling and Concentrate Agreement” between IGO and BHP Billiton. Listed below are tenement numbers and expiry dates. M15/1761 – 05/10/2025 M15/1762 – 05/10/2025 M15/1763 – 05/10/2025 M15/1515 – 23/12/2025 Location 48 - Non Crown Lease There are no Native Title Claims registered over the lease and no other known impediments.
Exploration done by other parties	Exploration was initially undertaken by WMC and eventually commissioned the Long Shaft and Victor decline mine development. This data is of high quality with most of the historic work is concentrated in areas that have been mined out.
Geology	The Long, McLeay, Moran and Victor South deposits are typical Kambalda-style nickel deposits, consisting of narrow, steeply dipping, shallowly south-plunging, ribbon-like accumulations of massive and semi-massive (with minor disseminated) sulphides. The mineralisation is located at the base of Archaean komatiitic ultramafic flows at the contact with an underlying tholeiitic basalt unit. The massive sulphide is overlain by matrix then disseminated mineralisation, with the bulk of the nickel mineralisation being massive and matrix in nature. The host rocks and associated contacts have been subjected to lower amphibolite facies metamorphism, structural modification, and

Criteria	Commentary
	intrusion by multiple felsic to intermediate igneous dykes and sills.
Drill hole Information	Drillhole data have been collected from this area since 1978 and total over 2000 drill holes. Reproduction of this number of drillholes, the majority of which has been mined out, would not assist in understanding of this report on resource estimation.
Data aggregation methods	Exploration results are calculated as the length and density weighted average to a 1% nickel cut-off. Maximum internal waste of 2m may be included however the total nickel composite average grade must be >1% nickel. Intercepts are length-density weighted across the entire width of the mineralised unit. No metal equivalents have been used.
Relationship between mineralisation widths and intercept lengths	All mineralisation intervals are reported as down hole lengths as well as true widths. The plunge and dip of the mineralisation is generally well understood so estimated likely true widths are calculated and reported.
Diagrams	See Long section.
Balanced reporting	Long sections shows the down hole widths and average grade for all drill holes recently drilled.
Other substantive exploration data	Geophysical plates generated from down hole EM or in-drive EM surveys are used for targeting additional drilling. EM targets are generated as 3D surfaces in a geological modelling program to target exploration testing. EM targets are displayed as rectangular shapes on plans to identify the proximal location of potential nickel mineralisation targets.
Further work	Further drilling is to be targeted in the Long (M07C surface) to the north of current mine development to test continuation of nickel mineralisation and porphyry dykes that cut through the mineralisation. Additional down plunge extensions of the Moran and McLeay mineralisation will also be tested. See Long section.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Primary data were collected by IGO geological staff using Excel templates on laptop computers using look-up codes. The information was transferred into Maxwell Geoservices Access Database “DataShed” with SQL2000 database server backend. All validation is completed by company geologists on site. Lab assay results are printed out and results for site blanks and standards are visually checked for acceptable values before the assay data are loaded from the digital laboratory files directly into the primary database. Drill hole collar coordinates, geology and assay data are visually checked by printing out a drill log with the combined information. The drill hole geology and assay results are also validated using a 3D geological modelling package. Core photos and visual checks from remaining half core samples were randomly checked.
Site visits	The competent person employed by IGO is site based. Reviews on QAQC and sampling procedures are undertaken quarterly. Competent persons from external consultants have not visited site for over 12 months but have reviewed the extracted database and estimation process.
Geological interpretation	Geological interpretation has a high to moderate confidence as up/down dip and plunge edges are well established. Barren porphyry dykes are irregularly spaced and orientated so geological interpretation is considered to be of moderate confidence. Data used for geological interpretation consists of diamond drill holes, lithology logging, assay grades and underground mapping of mineralisation and lithology units. Unmineralised porphyry dykes cutting across the ore bodies are mapped, logged and modelled into 3D wireframes. These are stamped in the block model as being waste. No alternative interpretations were investigated. Lithological control is used to determine the footwall and hanging wall contacts of the ore bodies and the unmineralised porphyry intrusions. The ore bodies are off-set by porphyry intrusions and faults. The mineralised komatiite volcanic flows continue past the fault off-sets.
Dimensions	Long deposit consists of 15 mineralised surfaces and is approximately 2.2km down plunge, 3m thick and 500m down dip in extent. The surfaces are narrow and ribbon-like accumulations of massive and semi-massive sulphides and starts from approximately 300 metres below surface topography. McLeay deposit consists of 7 mineralised surfaces and is approximately 600m down plunge, 3m thick and 160m down dip in extent and starts from approximately 700 metres below surface topography. Victor South deposit consists of 3 mineralised surfaces and is approximately 180m down plunge, 4m thick and 130m down dip in extent and starts from approximately 700 metres below surface topography.

Criteria	Commentary
	Moran deposit consists of 2 mineralised surfaces and is approximately 650m down plunge, 5m thick and 120m down dip in extent and starts from approximately 900 metres below surface topography.
Estimation and modeling techniques	Surpac v6.1 modelling software was used for the variography and block modelling. Ordinary kriging was used for grade interpolation, based on the variography and validation of the search orientations in Surpac. The minimum number of samples used in the estimate was 6 and the maximum was 24. The search ellipse radius used was 200m. All of the mineralised surfaces (except for Victor South Surface 1 and 4) were estimated using a 2D projection method with block centroids and grades converted to 3D and imported into a "real world' block model using nearest neighbour assignment. The orientation, block size and sub-celling regime of the real world block model was designed to provide sufficient volume resolution for accurate surface geometry representation, mine design, depletion and porphyry flagging. Comparisons with previous estimates show that the grade estimation is robust and does not vary significantly with new drilling data or depletion. Reconciliations show a positive result with more metal being produced when compared to the mineral resource block models and attributed to conservative resource estimation. Reconciliation is completed monthly using 3D wireframes of surveyed mine development. The wireframes are imported to the original resource model and volume and metal calculated using the Surpac modelling package. The volume and metal are compiled in an Excel spreadsheet and compared with reconciled volume and metal produced. No assumptions have been made regarding the recovery of by-products. No deleterious elements are estimated. The Long block model parent cell size was 10x4x8m sub-celling to 1.25x0.25x0.5m. The McLeay, Victor South and Moran block models parent cell size was 10x4x4m sub-celling to 5x0.5x0.5m. Diamond drill spacing at Long, Victor South and McLeay deposits are on a nominal 20m northing with 10m easting drill spacing with 5m by 5m closer-spaced drilling. Moran is on a nominal 40m northing with 10m easting drill spacing with some up to 10m by 10m closer-spaced drilling. Due to the narrow nature of most of the lodes, selectivity of ore is not possible and the entire ore surface will be mined with no internal selectivity. Most samples had measured densities determined using the Archimedes water immersion technique. Historical samples without measured densities were assigned calculated densities using the regression curve formula from measured data. Recent samples from ownership by IGO (13 years) all have measured bulk density values. Block cells had been coded with the wireframe name and only composite samples from that zone were used to interpolate grades into the zone. All grade interpolation was constrained to within geological contacts and to 1% nickel cut-off grade. Victor South disseminated zone cut-off grade was 0.6% nickel. Top-cutting was not applied to the nickel assays as the data did not present any apparent outliers. Densities were checked against density vs grade regression curves and density outliers were replaced with calculated densities. Block model validation was undertaken by comparing the volume of the modelled ore and the block model volume for each ore body. Comparisons were also undertaken on the average grade and density of the block model and the drill data for each model estimated using the 3D estimation technique or the accumulated metal variables where the 2D accumulation estimation technique has been used. The comparisons were undertaken for each ore body in total and also based on reporting in sections along the length of each ore body.
Moisture	The natural moisture of nickel sulphides is typically very low (<1%) due to the deposit being in fresh rock. Moisture is not factored into the estimation process.
Cut-off parameters	All grade interpolation was constrained within geological contacts and to 1% nickel cut-off grade. Victor South disseminated zone cut-off grade was 0.6% nickel. This is based on a natural grade boundary that exists for the two areas and also relates to an economic grade boundary.
Mining factors or assumptions	The mining method used will be underground mechanised cut and fill, long hole stoping and airleg stoping. Minimum mining width is in the order of 1.2m to greater than 4.5 metres depending on the ore body and mining method used in extraction of the ore. Long hole stopes range from 5m to 15m high stopes. No internal mining dilution assumptions have been made.
Metallurgical factors or assumptions	All intersections are below oxidation. Recoveries are calculated from contractual agreement where the ore treatment processes are undertaken using the BHP Billition nickel concentrator located within 5km of the mine. This process plant has been in use for over 30 years and is appropriate for ore sourced from this area.
Environmental factors or assumptions	Waste is trucked to the surface or used for backfill into old stopes. See Section 4.
Bulk density	All recent samples have measured densities determined using the Archimedes water immersion technique. Historical samples without measured densities were assigned calculated densities using the regression curve formula. This water immersion technique is undertaken on drill core before crushing and pulverising. As the drill core material is fresh the impact of contained moisture is very low. The core is sampled based on lithology and different

Criteria	Commentary
	mineralisation zones so bulk density values will not cross different rock types and mineralisation style interval. Bulk density is estimated into the block model using ordinary kriging. Porphyry intrusions are assigned a bulk density value of 2.7 g/cm^3 as this represents the average value of porphyry bulk density measurements.
Classification	Mineralisation classification is based primarily on drill data density and mine development proximity, in conjunction with a review of the understanding of footwall geology and fault controls on the mineralisation. Classification of Measured material is only used where ore drives are developed at the top and base of the ore block. Classification of Indicated material is generally because of closely spaced drilling and a production history, as well as good confidence in the geological model. Close-spaced drilling is on a 20m x 10m grid for all Long, Victor South and McLeay deposits and 40m x10m for Moran. Mineralisation modelled with a drilling density sparser than that defined above is classified as Inferred Resource. The classification scheme takes into account all of the relevant factors when assigning the resource classification. This result appropriately reflects the Competent Person's view of the deposit.
Audits or reviews	A review of the resource estimate was conducted by Cube Consultants in 2013. Variography used in the estimation for all Long deposits was validated by Cube prior to use in estimations.
Discussion of relative accuracy/confidence	The block model has block sizes set at approximately half the drill hole spacing to enable robust volume and grade estimation that is not overly smoothed. The parameters chosen for the estimate are selected to best represent the drilling data taking into account any declustering effect. The block model estimate is a global resource estimate. Comparisons of the block model depletion with the production figures show generally 20% to 25% positive correlation between the resource estimate block model and reconciled production data.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	All Ore Reserves estimated for the Long Victor mining complex are a sub-set of the Long Victor Mineral Resources. No reserves exist outside of the Mineral Resource base. Mineral Resources are inclusive of Ore Reserves.
Site visits	A site visit was undertaken by John Farr and Brett Hartmann of IGO on the 10 Apr 2013. In this visit all working areas were inspected. Philip Bremner of Mining One Pty Ltd conducted a site visit on the 23rd May 2013 as part of the review process.
Study status	The Long Victor Operations has a history of being mined by Lightning Nickel since October 2002. The mine reserves have been designed based off the current operational practices of the mine. All Ore Reserves are estimated by constructing three dimensional mine designs and reported against updated Mineral Resource block models. After modifying factors are applied, all physicals (tonnes, grade, metal, development and stoping requirements etc.) are input to a Reserve Evaluation model for an economical evaluation on a stope-by-stope basis. Previous mine performance has demonstrated that the current mining methods are technically achievable and economically viable. Material Modifying Factors have been considered and compared well to reconciled performance.
Cut-off parameters	Cut off values are calculated on the basis of the NSR (Net Smelter Return) calculation. The resource model is evaluated against the NSR cut off value and mining areas (stopes and development) are designed for those areas above the NSR cut off value. Once designed, the entire mining area / stope is evaluated again, against the NSR cut off value.
Mining factors or assumptions	Three dimensional mine designs are designed based on known information about orebodies physical characteristics and the geotechnical environment. Modifying factors such as unplanned dilution (25% for Long hole stoping and 5% for all other methods) and reserve recovery (90% for Jumbo stoping and 95% for all other methods) are applied based on the chosen mining method. In some cases geotechnical losses are applied for particularly adverse geotechnical conditions. In certain cases where a mined stope contained both Indicated and Inferred Mineral Resources, the stope was only designed around the Indicated Resources, however in some case a small quantity of Inferred Resources have been captured. Only in cases where the Inferred mineralisation had to be mined to access Measured or Indicated Resources has it been included in the reserve calculation. Any Inferred material mined was converted to a Probable Reserve and had no bearing on the economic outcome of the reserves. No additional infrastructure was required for the mining of the current reserves.
Metallurgical factors or assumptions	Lightning Nickel is contractually required to supply all ore to the BHP_B Ni_West Kambalda Concentrator. All metallurgical recoveries are well defined within this contract and are built into the above mentioned Reserve Evaluation model.
Environmental	Lightning Nickel operates under an environmental management plan, which meets or exceeds all environmental legislative requirements. Lighting Nickel's license to operate is in good standing. Environmental rehabilitation plans are constructed and acted upon when the timing is appropriate. The costing of the rehabilitation works is accounted for in the operations Life of Mine model.
Infrastructure	The current infrastructure at Lightning Nickel is adequate for the Ore Reserves statement. Maintenance costs for current equipment were included in the reserve economic model.
Costs	Capital costs for decline development were included where applicable. An allowance per ore tonne is also made for ongoing exploration costs.

Criteria	Commentary
	Operating costs were updated against the previous twelve months actual costs. Nil allowances were made for the content of deleterious elements as none have been previously encountered. A fixed processing charge from BHP_B Ni_West Kambalda Concentrator was applied to all ore tonnes mined; this includes all transport and shipping from their facilities. All unit costs are updated from the most recent financial year's actual costs.
Revenue factors	The assumptions made for commodity prices are: Nickel price US$18,087, Copper price US$ 7,694 per tonne and Foreign exchange rate of US$1.01.
	These values were selected after reviewing a number of industry recognised price forecasting leaders, which included Bloomberg and Brook Hunt.
	Metal prices were assumed fixed for the life of the project.
Market assessment	Longer term market assessments are provided by a number of independent companies such as Brook Hunt and Bloomberg. Market conditions are considered in part of the long term cost evaluation.
Economic	NPV was not taken into account in the economic review. The estimated life of mine is currently under five years and so fixed cost and prices were used.
	Sensitivity analysis work has been undertaken on variables such as head grade, tonnages, foreign exchange rate and metal price.
	The project is highly sensitivity to the foreign exchange rate (AUD:USD) and nickel metal price.
Social	Tenement status is currently in good standing.
Other	The Long Operation is a historically seismically active mine. This risk is managed as far as practically possible through the operations Ground Control Management Plan and the allocation of appropriate resources.
	There are no other foreseeable risks associated with the Long Operation on a sociological or political assessment.
Classification	Ore Reserves are based on geological and mining confidence and categorised as either Proved or Probable.
	This result appropriately reflects the Competent Person's view of the deposit.
	The proportion of Proved Ore Reserves is a subset of the Measured Mineral Resources. All Probable Ore Reserves have been derived from Indicated or in a small proportion Inferred Mineral Resources.
Audits or reviews	An independent audit is undertaken annually on both the Mineral Resource and Reserve process. No unexpected results have come from this review.
Discussion of relative accuracy/confidence	As part of the Ore Reserve estimation process a comparison is undertaken reviewing actual reconciled extraction versus previous years Ore Estimation and Resource Estimation.
	A review of last year's performance by the Competent Person found that both the Resource and Reserve estimation processes are conservative estimates.

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Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Almost all sampling has been core sampling, with the surface resource drilling programs being mostly ½ NQ core or ¼ HQ core. In these drill programs, the minimum sample length was set at 0.3m, while the maximum sample length was 1.5m. In the underground drilling, NQ2 core samples were minimum length 0.3m and maximum length 1m. Core was cut with an automated core cutter after orientation and mark-up.
	Zinc and copper mineralisation is visible and zones containing sphalerite and chalcopyrite, whether in massive sulphide or stringer form, are sampled, along with a 5m buffer zone either side of the mineralised interval.
	Core was cut with an automated core saw after orientation, mark-up, logging and photography. The same side of the core is always selected for sampling.
Drilling techniques	Principally diamond drilling with the exception of several RC precollars. Surface holes were drilled by Titeline Drilling Pty Ltd and Boart Longyear Pty Ltd. The surface diamond drilling is a mixture of HQ and NQ core sizes. Core was oriented using an Ace tool or spear. Underground drilling from 2011 was by Sanderson Drilling, Kalgoorlie and holes were NQ2 core size. Core was oriented using a Reflex ACT II tool and the orientation line was drawn on core prior to mark-up for cutting and sampling.
Drill sample recovery	Core recovery was good to excellent, being consistently >90%. Measured core lengths and core losses are compared with driller's blocks and recorded in the database. The measured lengths are compared with expected lengths to calculate recovery.
	Core was cut with an automated core saw after orientation, mark-up, logging and photography. The same side of the core is always selected for sampling.
	Most core is competent and cuts well with minimal loss of fines. No sample bias is suspected.
Logging	Core was photographed both dry and wet and copies of the digital images stored on the Jaguar minesite server. All core holes are logged. Geological logging included rocktype, deformation, structure, alteration, mineralisation, veining and RQD measurements. Logging of underground core occurs digitally straight into acQuire data entry objects and is loaded into the acQuire database. Surface drilled holes were logged on paper and subsequently data entered and loaded into the acQuire database. Underground faces and backs are also mapped and used with the drilling data to guide geological interpretation. Geological logging is adequate for resource estimation.
	Logging is qualitative and semi-quantitative in nature.
	All mineralised zones are logged in detail and the remainder of the hole is logged in slightly less detail (at distances >20m from economic ore zones, detailed structural alpha and beta angles are not collected).
Sub-sampling techniques and sample preparation	Core was cut with an automated core cutter after orientation and mark-up. Core sample sizes are discussed in the Sampling Techniques section.
	Samples were sent to Genalysis (now Intertek) in Maddington, WA. The sample preparation method was to dry the core in ovens for at least 2 hrs (105°C), then jaw crush the samples to a nominal minus 10mm size then Boyd crushed to a nominal minus 2mm. After crushing, the surface drillhole samples were pulverised in a mixer mill in a single stage mix and grind process (SSMG) to a nominal 85% passing 75 micron. Any samples that exceeded the 3kg mill limit were rotary split prior to the pulverising stage. For the underground holes, total sample pulverisation has occurred. These techniques are appropriate for base metals samples.
	Coarse crush washes at the crusher stage and quartz washes at the pulverising stage have been implemented between every sample to combat sample carryover (contamination) during the sample preparation process. Sieve tests on 10% of the samples have been implemented to measure the fraction of pulp passing the 75 micron threshold.
	Field duplicates in the form of second half core sampling are inserted at a rate of 2 per 100 samples in the underground drilling. The sampling is representative of the material drilled. 85% of the field duplicate samples in the 2012-2013 drilling were within +/-20% relative difference for Zn, 79% for Ag and 77% for Cu.
	Sample sizes are appropriate for the material sampled.
Quality of assay data and laboratory tests	At the exploration stage, assaying for Cu, Pb, Zn, Ag and Fe was by four-acid digest involving hydrofluoric, nitric, perchloric and hydrochloric acids and analysis by Flame Atomic Absorption Spectrometry (AAS), while Au was analysed by fire assay with AAS finish. Assay techniques in the resource definition program consisted of four-acid digest with AAS finish for base metals to 0.01% detection limits, while Ag used four-acid digest with an MS finish to 0.2-1ppm detection limit. Au was analysed by 50g fire assay to 0.01ppm detection limit. For the underground drill samples similar methods were used but in Feb 2012 the 50g fire assay was reduced to a 25g charge due to high sulphide content samples. In Nov 2012 the underground samples were changed to a finish by ICP-OES method for Cu, Zn, Pb, Ag and Fe, so that As, Sb and S could also be analysed. Gold analysis remained as AAS. The assay techniques used are considered appropriate for this type of mineralisation, both are total extraction methods.
	No geophysical or XRF results are used in the resource estimate.
	Quality control procedures included the insertion of standards, blanks, field duplicates, cross-lab checks and same laboratory checks. Check-assay samples identified poor precision for Au, most likely due to the need to reduce the catchweight in high sulphide samples, sometimes to 5g (from 25g), which impacts the repeatability of the Au assays. The Ag, Cu, Zn and Pb analyses were shown to be reasonably accurate and precise and only low bias (<5%) was observed for these elements in the 2012 drilling. IGO is satisfied that the base metal analyses are suitable for resource estimation. IGO will continue to work with the laboratory to further reduce bias.
Verification of sampling and	Significant intersections are checked by company personnel to see they meet the known geological and

Criteria	Commentary
assaying	mineralisation models.
	Twin holes were drilled as wedge holes in surface drilling for resource delineation, in 2009. Holes are fan drilled in the underground and twinned holes are not drilled.
	Primary data are collected in Excel spreadsheets or using off-line acQuire data entry objects on Toughbooks. Data are imported directly to the database with importers and have built in validation rules. Assay data are imported directly from digital assay files and are merged in the database with sample information. All holes have a hard copy summary plotted for review with geological and assay information.
	From time to time assays will be repeated if they fail company QAQC protocols, however no adjustments are made to assay data once accepted into the database.
Location of data points	Surface holes were collar surveyed by independent surveyors and later drill holes by on-site surveyors. Drillhole collar positions were surveyed using RTK GPS equipment. Dip and Azimuth readings – good quality surveys using downhole camera shots at about 30m intervals for the initial exploration program, whilst a gyro survey tool was used for the follow-up resource definition programs (surface drilling). Underground drilling used a DeviFlex 8377 non-magnetic multi-shot tool (referencing gyro) with surveys at 4m intervals, accuracy to +/-0.01° Azimuth (per station) and +/-0.2° Dip. Mine workings are surveyed by the on-site surveyors using a Leica 1205 instrument to an accuracy of +/- 2mm. A CMS (Cavity Monitoring System) tool is used for surveying stope voids. Collar and downhole surveys are considered accurate, which is supported by location of mine workings into the modelled mineralisation.
	All resource work has been conducted on the local mine grid.
	All mineralisation is mined by underground methods so no surface topographic control is required.
Data spacing and distribution	Surface diamond hole drill coverage at Bentley is on a nominal 50x50m pattern with fan drilled patterns from underground to intersect the mineralisation at a nominal spacing of 20m (northing) x 20m (RL). Minimum hole spacing of ~10m where wedge holes have been drilled, while the maximum hole spacing does not exceed 70m (Inferred Resource).
	The data spacing and distribution are sufficient to establish the geological and grade continuity for the classifications applied. The wide spaced drilling (surface holes) below 3950mRL has resulted in a resource classification of Inferred, until greater confidence through drilling can be demonstrated.
	Samples were composited to 1m downhole composites with length and density weighting, for grade estimation.
Orientation of data in relation to geological structure	Surface drilling intersects the massive sulphide lenses almost perpendicular to the lens orientation at Bentley, and at a mean angle of 45-50° to the sulphide veins in the stringer sulphide domain. 09BTDD015, 09BTDD017, 10BTDD017 and 10BTDD018 were drilled down dip and along strike of mineralisation to test for dolerite bodies and faults that might not have been intersected by drilling perpendicular to the orebody. Underground fan drilling is drilled from the footwall through to the hangingwall, orientation is good, not always optimal but is considered adequate for resource estimation given the limited choices for drill access underground. Drilling where intersection angles would be strongly oblique to mineralisation is postponed to later programs with deeper access.
	No orientation biased sampling is suspected or has been identified in the data.
Sample security	All samples are securely contained and sealed during transport to and from the laboratory in Perth and site. All transportation is direct with corresponding sample submission forms and consignment notes travelling with the samples which are also recorded at site. The laboratory receives samples and checks them against dispatch documents. IGO staff are advised of any missing or additional samples. All storage is secure on site, at the laboratory, and when the samples return to site after assay.
Audits or reviews	Sampling techniques and data collection processes are reviewed regularly by IGO staff. No external review has been conducted.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The Bentley deposit is within mining lease M37/1290 held 100% by Jabiru Metals Ltd (JML), a wholly owned subsidiary of Independence Group NL (IGO). There is no native title claim over the area.
	The tenure is secure and no known impediments exist. The Bentley mine has been operating since 2011.
Exploration done by other parties	The Bentley mineralisation was discovered by JML in 2008. No exploration is being conducted by other parties in or around the Bentley mine.
Geology	Bentley is a V(H)MS style deposit, occurring as polymetallic (pyrite-sphalerite-chalcopyrite-galena) massive sulphide mineralisation within a volcano-sedimentary succession. Intrusion by tholeiitic dolerite has led to disruption of the original massive sulphide lenses into four or more discrete lenses (Arnage, Mulsanne, Brooklands and Comet). The footwall to the Arnage massive sulphide lens consists typically of stringer and disseminated sulphide mineralisation comprising pyrite, chalcopyrite and minor sphalerite in a rhyolitic unit. The mineralisation dips steeply (75-80°) to the west (local grid). The largest lens, (the Arnage lens) has a strong southerly plunge.
Drill hole Information	Holes drilled into the Bentley deposit are described in Section 1. Current drilling is from underground and involves infill drilling known mineralised zones within the resource envelope to a nominal 20m x 20m hole spacing.
	A summary of drillholes is not considered applicable in this instance as drilling is considered to be development drilling within and below the mine, and exploration results are not being reported.
Data aggregation methods	There are no exploration results reported for the immediate Bentley mine area.
Relationship between	There are no exploration results reported for the immediate Bentley mine area. Orientation of mineralisation with

Criteria	Commentary
mineralisation widths and intercept lengths	drilling angles has been covered in Section 1.
Diagrams	There are no exploration results reported for the immediate Bentley mine area.
Balanced reporting	There are no exploration results reported for the immediate Bentley mine area.
Other substantive exploration data	There are no exploration results reported for the immediate Bentley mine area.
Further work	Infill drilling of the Inferred mineralisation at depth (below 3950mRL) will commence this year, so that the Inferred Mineral Resource, once upgraded to Indicated, can be used for mine planning purposes and become part of the Ore Reserve.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	The parent database for all collar, survey, geology and assay data is a SQL database with the acQuire software as the front end. This acQuire database has a number of built in fields and reports to ensure data are entered correctly and obey certain validation rules. Assay data are imported directly from laboratory files and merged with sampling data. Most other data are captured digitally and imported directly to the database with few opportunities for keying errors. All data with the parent Jaguar or OP-Bentley project code are exported to a Microsoft Access database which is frozen in time as a permanent record of the database used for that resource estimate.
	Data are visually and graphically checked to ensure that there are no outlying errors. Any errors noted are corrected on an ongoing basis. The database is again checked and corrected for errors and missing data prior to resource estimation work.
Site visits	The Competent Person, Michelle Wild, is the Principal Resource Geologist for IGO and is based in Perth. She regularly visits site to review procedures and recommend improvements to processes. Two site visits were conducted in 2012-2013 (October 2012 and April 2013). In addition, regular monitoring of data collection, quality and QAQC results is undertaken.
Geological interpretation	Confidence in the geological interpretation for Bentley is high, with the mineralisation and geological setting being simple and well understood, and the drilling and face mapping confirming the interpretation.
	Good geological cross-sectional interpretations were available to guide modelling of the mineralisation. Face and backs mapping, face sampling, as well as geological interpretation on section from drilling information, were used to refine the interpretation for this estimate.
	Face and backs mapping in the underground workings has confirmed the interpretation originally based on drilling data. There is no alternative interpretation.
	The mineralisation was domained into massive and stringer domains. Geology was used to define the massive sulphide domain whereas both geology and cut-off grades were used to define the stringer domain. Grades for each domain were interpolated independently.
	The main factors controlling continuity at Bentley are a series of post-mineralisation dolerite intrusives which are interpreted to be disrupting the lenses, and a minor east-west fault displacing the Arnage and Mulsanne lenses by 8m to the east.
Dimensions	The Arnage massive sulphide lens, which is the largest of the mineralised lenses at the Bentley deposit, has an average length of 250m along strike (north-south) with a southerly down plunge length of 600m and a maximum thickness of 30m. It sits 160m below the surface and extends a vertical depth of 500m. Mulsanne is about 200m long, 150m vertical extent, and approximately 3m thick. Brooklands is about 150m long, 200m vertical extent, and approximately 5m thick. The Comet lens is about 175m long, 150m vertical extent and approximately 2m thick.
Estimation and modelling techniques	Ordinary Kriging was used for grade estimation utilising Surpac v6.3.2 software. Kriging and search parameters were derived from variogram models for each element and Kriging Neighbourhood Analysis (KNA). Grade estimation was constrained to each of the massive sulphide and stringer sulphide lens wireframes. Top-cut grades were determined from a review of the composite sample data statistics, histograms and log-probability plots. A top-cut of 20% was applied to Cu within the massive sulphide domain, while top-cuts were applied to Zn (13%), Cu (12%), Pb (1.2%), Ag (250ppm) and Au (2.2ppm) within the stringer mineralisation domain.
	This estimate compares well with previous estimates. Reconciliation with mine production shows the model tends to underestimate ore tonnes (-21%) and overestimate Zn grade (15%), whilst Cu and Ag grades are within 4%. The tonnage difference is due to identification and mining of mineralisation outside the model. The Mineral Resource estimate is undiluted, so some of the additional tonnage will be mining dilution.
	No assumptions have been made regarding the recovery of by-products.
	Economic minerals estimated are Zn, Cu, Ag and Au. Fe, Pb and density are also estimated.
	Drill intercept spacing is 20x20m in the developed portion of the mine and nominally 50x50m in deeper portions of the mineralised envelope (below current development). The parent block size was set at 15m Y x 7.5m X x 7.5mZ and grades were interpolated into these blocks. Parent block grades are assigned to sub-blocks within the parent block and the constraining wireframe.
	No modelling of selective mining units has taken place.
	No correlation between variables has been assumed in the grade interpolation stage. Each variable has been interpolated with its own kriging parameters based on variography.
	The block model cells were coded according to which style of mineralisation and which wireframe they were within. Corresponding composite sample files were used for grade interpolation into cells inside each of the wireframe domains.
	Top-cuts were described in the section on Estimation and Modelling Techniques.

Criteria	Commentary
	The block model is checked visually first, in Surpac graphics, and compared with drilling data, then checked on a section and lens basis by comparing composite sample grades with block model grades in swath plots. Life of mine reconciliation is completed each time the resource estimate is updated.
Moisture	No samples were tested for moisture content. All sampled core was from well below the oxidised rock profile. The samples were considered impermeable and moisture content is expected to be well below 1%. On this basis the tonnage estimate is considered to have been estimated with natural moisture.
Cut-off parameters	No cut-off grades have been applied to define the massive sulphide domain. A lower assay cut-off of 0.3% Cu or 4% Zn was applied to define the stringer mineralisation domain. A block cut-off grade of 0.5% Cu was applied to the stringer zone for resource estimation and was based on marginal mining and processing costs and recoveries for the Jaguar Operation, plus some allowance for changes in metal price and NSR assumptions in the Ore Reserve estimation stages.
Mining factors or assumptions	No mining method, minimum mining width, dilution or other mining factors have been assumed in the Mineral Resource estimate. The mine has been in production for over 2 years.
Metallurgical factors or assumptions	No metallurgical factors or assumptions have been made; the mill on site is a flotation plant which generates two concentrate types, and has treated the ore proficiently and successfully for over 2 years.
Environmental factors or assumptions	No environmental factors or assumptions have been made; the waste dump and tailings storage facilities are well established with approval from the Department of Mines and Petroleum (DMP).
Bulk density	JML/IGO performed density testwork on almost all core samples that were submitted to the laboratory for assay. All density measurements have been determined using the simple water immersion technique, on uncoated core and for the entire sample interval. Core was uncoated because it was impervious. The assays for Cu, Pb, Zn and Fe were combined and compared with the measured densities and regression curves determined for massive sulphide and stringer domains. Outliers (outside +/-10% from the regression curves) were removed from the dataset. A calculated density, using the appropriate regression formula, was assigned to those samples without their own correct density measurement. Density was interpolated into the block model using Ordinary Kriging. Density was also used to weight each of the sample composites used in grade estimation.
Classification	The average drill hole spacing in the upper portion of the resource is approximately 20m along strike and 20m vertically. The average drill hole spacing in the central portion of the resource is approximately 50m along strike and variable between 30m and 50m down dip. The spacing and confidence in the geological interpretation is considered adequate to allow classification of the resource as Measured Resource in the area with mine development and drilling with spacing 20m or less, and as Indicated Mineral Resource where drill spacing is <50m. Where the drill spacing is greater than this an Inferred classification has been assigned. Minor zones with poor continuity or limited sample points were not classified as Resource.
	Input data are of excellent quality and there is high confidence in the geological and mineralisation interpretations where drillholes are <50m apart. Confidence in grade and tonnage estimates where the drilling is at a greater spacing is less, and is reflected in the Inferred classification.
	The classification of the Mineral Resource reflects the Competent Person's view of the confidence in the estimate.
Audits or reviews	The Bentley resource estimate for 2012 was reviewed by Optiro Pty Ltd in October 2012. No significant flaws were identified, and most of the recommendations have been incorporated into the 2013 estimate.
Discussion of relative accuracy/confidence	Confidence in the Mineral Resource estimate is high in the areas with mine development and/or drilling with a 20m x 20m pattern. Confidence is moderate to poor in the deeper portions of the model which are the target for more closely spaced drilling over the next year. Factors considered in classifying the resource estimate were drill spacing, confidence in defining mineralisation boundaries along strike and down plunge, sufficient (or not) numbers of drillholes and samples for good grade estimation. Sample quality was excellent and did not factor into the classification. The main factor that could affect the accuracy of the estimate is the drill spacing.
	The estimate is a global estimate.
	Reconciliation with production data takes place each time a new Mineral Resource estimate is completed. Reconciliation for the life of mine to 30 June 2013 was carried out in July 2013 and commented on in the section on Estimation and Modelling Techniques, above.
Resource Model number	BT_RSC_2013_03

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	All Ore Reserves estimated for the Jaguar Operation are a sub-set of the Jaguar and Bentley Mineral Resources. No reserves exist outside of the Mineral Resource base.
	Mineral Resources are inclusive of Ore Reserves.
Site visits	A site visit was undertaken by Brett Hartmann of IGO on the 17th – 19th of April 2013. In this visit all working areas were inspected. Philip Bremner of Mining One Pty Ltd conducted a site visit on the 20th -21st May 2013 as part of the review process.
Study status	The Jaguar Operation has a history of being mined by Jabiru Metals since October 2007. The mine reserves have been designed based off the current operational practices of the mine. All Ore Reserves are estimated by constructing three dimensional mine designs and reported against updated Mineral Resource block models. After modifying factors are applied, all physicals (tonnes, grade, metal, development and stoping requirements etc.) are input to a Reserve Evaluation model for an economical evaluation on a stope-by-stope basis.
	Previous mine performance has demonstrated that the current mining methods are technically achievable and economically viable. Material Modifying Factors have been considered and compared well to reconciled performance.

Criteria	Commentary
Cut-off parameters	Cut off values are calculated on the basis of the NSR (Net Smelter Return) calculation. The resource model is evaluated against the NSR cut off value and mining areas (stopes and development) are designed for those areas above the NSR cut off value. Once designed the entire mining area is evaluated on a stope-by-stope basis again, against the NSR cut off value. Two cut off values are utilised, a higher value ($180.00 per ore tonne) for direct mill feed and a lower value ($120.00 per ore tonne) for marginal costs.
Mining factors or assumptions	Three dimensional mine designs are designed based on known information about the orebodies physical characteristics and the geotechnical environment. Modifying factors such as unplanned dilution (20% for Long hole stoping and 5% for development) and reserve recovery (95% for stoping) are applied based on the chosen mining method. In some cases geotechnical losses are applied or *in situ* pillars are left, reducing the overall recovery factor.
	In certain cases where a mined stope contained both Indicated and Inferred Mineral Resources, the stope was only designed around the Indicated resources.
	No additional infrastructure was required for the mining of the current reserves.
Metallurgical factors or assumptions	Both Jaguar and Bentley underground reserves are processed at the Jaguar processing facilities. The process and recovery of both ore sources is well understood and reasonably consistent in performance. Conservative recovery factors has been used: o 82.0% Cu recovery into Cu concentrate o 53.0% Ag recovery into Cu concentrate o 43.0% Au recovery into Cu concentrate o 83.0% Zn recovery into Zn concentrate o 22.0% Ag recovery into the Zn concentrate
Environmental	Jabiru Metals operates under an environmental management plan, which meets or exceeds all environmental legislative requirements. Jabiru Metals' license to operate is in good standing. Environmental rehabilitation plans are constructed and activated upon when the timing is appropriate. The costing of the rehabilitation works is accounted for in the operations Life of Mine model.
Infrastructure	The current infrastructure at the Jaguar Operation is adequate for the extraction of both the Jaguar and Bentley underground reserves. Maintenance costs for current equipment were included in the reserve economic model.
Costs	Capital costs for decline development were included in the financial evaluation. An allowance per ore tonne is also made for ongoing exploration costs. Operating costs were updated against the previous twelve months actual costs. Concentrate payables, which includes accounting for any deleterious elements, has been calculated and used within the NSR evaluation process. All unit costs are updated from the most recent financial year's actual costs.
Revenue factors	The assumptions made for commodity prices are: Copper price US$ 7,694 per tonne, Zinc price US$2,270 per tonne, Silver price US$33.00 per troy ounce, Gold price US$1,740 per troy ounce and Foreign exchange rate of US$1.01.
	These values were selected after reviewing a number of industry recognised price forecasting leaders, which included Bloomberg and Brook Hunt.
	During the calculation of reserves the Metal prices were assumed fixed for the life of the project.
Market assessment	Longer term market assessments are provided by a number of independent companies such as Brook Hunt and Bloomberg. Market conditions are considered in part of the long term cost evaluation.
Economic	NPV was not taken into account in the economic review. The estimated life of mine is currently under five years and so fixed cost and prices were used.
	Sensitivity analysis work has been undertaken on variables such as head grade, tonnages, foreign exchange rate and metal price.
	The project is highly sensitivity to the foreign exchange rate (AUD:USD) and copper and zinc metal prices.
Social	Tenement status is currently in good standing.
Other	There are no other foreseeable risks associated with the Jaguar Operation on a sociological or political assessment.
Classification	Ore Reserves are based on geological and mining confidence and categorised as either Proved or Probable.
	This result appropriately reflects the Competent Person's view of the deposit.
	The proportion of Proved Ore Reserves is a subset of the Measured Mineral Resources. All Probable Ore Reserves have been derived from Indicated or in a small proportion Inferred Mineral Resources.
Audits or reviews	An independent audit is undertaken annually on both the Mineral Resource and Reserve process. No unexpected results have come from this review.
Discussion of relative accuracy/confidence	As part of the Ore Reserve estimation process a comparison is undertaken reviewing actual reconciled extraction versus previous years Ore Estimation and Resource Estimation.
	A review of last year's performance by the Competent Person found that both the Resource and Reserve estimation processes are conservative estimates.

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	The Jaguar Project is drilled by HQ or NQ2 diameter diamond drill holes (DD) on a 20m (easting) x 20m (northing) grid spacing for underground holes and 50m (easting) x 50m (northing) grid spacing for surface holes. Samples are taken through visible mineralisation and for 5m buffer zones around the visible mineralisation. Core is measured and marked up with metre marks and sampling intervals, prior to cutting. All massive sulphide intercepts have been sampled. Samples throughout the deposit are good quality core samples.
	Sampling was carried out under IGO protocols and QAQC procedures at industry standard or better.
	The core was sampled to a nominal length of 1m, however, sample lengths varied between 0.3m up to 1.5m in the massive sulphide and stringer sulphide domains, with intercepts adjusted to geological boundaries to ensure representivity. Samples were crushed, dried and pulverised to produce a sub-sample for digestion using a 4 acid digest and analysis with ICP/OES, ICP/MS, or AAS.
Drilling techniques	Diamond drilling accounts for 100% of drilling at Jaguar. The surface diamond drilling is a mixture of HQ and NQ2 core sizes. The underground holes at Jaguar are all NQ2 core size. Core was oriented using a Reflex EZ-mark tool. Underground face sampling is used to define resource boundaries where appropriate, however, they are not used for resource estimation. The method of face sampling used channel chip sampling with a rock hammer, 1m above the floor of the drive.
Drill sample recovery	Diamond core recoveries are logged and recorded in the database by comparing core length measured with core length expected. Overall recoveries are >90% and there are no core loss issues or significant recovery problems.
	Diamond core is reconstructed into continuous runs on an angle iron cradle for orientation marking and metre marks. Depths are checked against the depth given on the core blocks, and rod counts are routinely carried out by the drillers. Core is usually competent and good quality.
	The mineralisation is defined by diamond core drilling, which has high recoveries and is of good quality. There are no issues with preferential losses or gains in the core samples.
Logging	Diamond drill hole logging recorded lithology, mineralogy (determined via hand lens), mineralised zones, structural, weathering, colour, alteration, veining and other features of the core. All surface holes were photographed wet and dry, and all underground holes post March 2011 were also photographed wet and dry. Geotechnical logging was carried out on all diamond drill holes for recovery, RQD and number of defects (per interval) information on structure type, dip, dip direction, alpha angle, beta angle, shape, roughness and fill material are stored in the geotech and structure tables of the database. All drill holes were logged in full for their entire length.
Sub-sampling techniques and sample preparation	Core is cut half (NQ2) and quarter core (HQ) on site using an automated Almonte core cutter. To ensure repeatability the core is cut 1cm off of the orientation line to ensure that the orientation mark and other marking on the core is retained. To ensure repeatability the same side of the core is sampled each time.
	No RC samples have been used at Jaguar.
	The sample preparation for diamond core follows industry standard in sample preparation. This involves oven drying for 2 hours, coarse crushing of the half core sample down to 2mm followed by pulverisation of the entire sample by Essa LM5 grinding mills to a grind size of 85% passing 75 micron.
	The laboratory duplicates approximately 5% of the samples in a batch using a separate pulp sub-sample from the same pulp packet. These are checked against the original assays in the IGO QAQC reporting per batch. In November 2011, pulp sizing checks were introduced such that 10% of samples were tested for percentage of pulp passing 75 micron.
	Field duplicate results (second half (NQ2) or second quarter (HQ)) performed from the 2009/2010 drilling program onwards show that half core sampling is representative of the interval drilled.
	The sample sizes are considered appropriate to correctly represent the sulphide mineralisation at Jaguar. This is based on the massive and stringer mineralisation, the thickness and consistency of the intercepts, the sampling method and percentage value assay ranges for the primary elements.
Quality of assay data and laboratory tests	The analytical technique has varied over the sampling history. The digest is a 4 acid digest with an ICP/OES or ICP/MS finish for recent samples (2013), (with 25 gram AE/AAS for gold), previously a 4 acid digest multi element AAS finish has been used. The acids for this digestion are hydrofluoric, nitric, perchloric and hydrochloric acids, suitable for silica based samples. The method approaches total dissolution for most minerals.
	No geophysical tools were used to determine any element concentrations used in this resource estimate.
	QC procedures involve the use of certified reference material as assay standards, along with blanks, and duplicates. The insertion rate for standards and blanks was 1:20 and duplicates 1:50, all being within mineralised zones. In 2011 quartz washes were implemented between each sample in the mills, and in 2012 blue metal flushes were carried out between each sample in the crushing stage, both methods to combat contamination seen in the blanks. Fineness tests are carried out by the laboratory to ensure that 85% passing 75 microns was attained (insertion rate 1:10). Laboratory QAQC also requires the use of internal lab standards using certified reference material, blanks, splits and replicates. Cross-lab checks are now performed on a regular basis. Results highlight that assay values are accurate, precision is good and bias is minimised.
Verification of sampling and assaying	Both the Competent Person and senior geologists from both the production and exploration departments have verified significant intersections of diamond core from Jaguar.
	No twin holes have been drilled at Jaguar.

Criteria	Commentary
	Primary data was collected using offline versions of the acQuire database on Toughbook's. Collar surveys, down hole surveys and assay results were loaded into the online acQuire database using importing routines. All holes have a summary plotted for review in hard copy with geological and assay information, also assay results arrive in electronic and hard copy format for electronic and physical storage.
	No adjustments or calibrations were made to assay data used in this estimate.
Location of data points	Drill hole collar surveys were carried out by the on-site surveyors using a Leica 1205 instrument to an accuracy of +/-2mm; the same surveyors used the same tool for the pick-up of drives and massive sulphide mark-ups, with a CMS (Cavity Monitoring System) tool being used for surveying stope voids. Down hole surveys were carried out in the underground holes at various intervals using a Reflex-EZ multi shot tool (30m intervals, changing to 6m in January 2009) accurate to +/-0.5° Azimuth and +/-0.2° Dip, Reflex Gyro (north-seeking, 3m intervals) accurate to +/-0.5° Azimuth and +/-0.2° Dip and more recently down hole DeviFlex tool (referencing gyro, 3m intervals) accurate to +/-0.01° Azimuth (per station) and +/-0.2° Dip. Surface holes were down hole surveyed at 50m intervals using a single shot Eastman camera. Surface holes have been superseded by more accurate underground drill holes in the resource estimate. Data point location, quality and accuracy is excellent.
	Surface drill holes used the MGA94 grid, later converted to the local Jaguar Mine Grid, whereas the underground holes, coupled with the workings used the local Jaguar Mine Grid. Elevations are in AHD RL and a value of +4,000m was added to the AHD RL by IGO for local coordinate use.
	Surface holes were collar surveyed by independent surveyors and later surface drill holes by on-site surveyors. All mineralisation is mined by underground methods.
Data spacing and distribution	The nominal spacing is 20m (northing) x 20m (easting) for underground drilling. The original drill out of the deposit was on a 50m x 50m drill spacing.
	The data spacing and distribution is more than sufficient to establish geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure and classification applied.
	1m down hole composites for drill hole samples were used with length and density weighting.
Orientation of data in relation to geological structure	Drilling from underground has largely been located from within the footwall of the deposit which has enabled generally good orientation of massive sulphide intercepts. Drilling of the Far Side lodes was all completed from the hangingwall which also provided good intersection angles. Surface holes provide a good intersection angle for the shallow holes; however, for the deeper holes the angle is closer to the mineralisation dip. These holes have mostly been superseded by underground drill holes.
	No orientation biased sampling has been identified in the data.
Sample security	All samples are securely contained and sealed during transport to and from the lab in Perth and site. All transportation is direct with corresponding sample submission forms and consignment notes travelling with the samples which are also recorded at site. All storage is secure both at the lab and when the samples return to site after assay.
Audits or reviews	No formal audits or reviews have been completed on the sampling techniques and data, however, Michelle Wild (Principal Resource Geologist, IGO, Perth) has outlined a company-wide sampling and QAQC protocol that was implemented in 2011. In-house reviews of procedures on site are conducted on a regular basis.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The Jaguar deposit is located within M37/1153, a granted mining lease held 100% by Jabiru Metals Limited (JML), a wholly owned subsidiary of IGO. There are no Native Title Claims registered over the lease.
Exploration done by other parties	Exploration at the Jaguar deposit was initially carried out by Inmet Mining (Australia Pty Ltd) as part of a joint venture with Jabiru Metals. The deposit was discovered in an Inmet program in 2002, with Jabiru Metals acquiring 100% of the deposit on the 31st March 2003. All exploration from surface and underground was subsequently completed by Jabiru Metals.
Geology	Jaguar is a V(H)MS style deposit, occurring as a polymetallic (pyrite-sphalerite-chalcopyrite) massive sulphide lens with stringer feeder zones within a volcano-sedimentary succession.
Drill hole Information	Holes drilled from underground are as described in Section 1, this includes holes drilled to extend known mineralisation and to explore underground for the Jaguar deposit.
Data aggregation methods	There are no exploration results reported for the immediate Jaguar mine area.
Relationship between mineralisation widths and intercept lengths	There are no exploration results reported for the immediate Jaguar mine area. Orientation of mineralisation with drilling angles has been covered in Section 1.
Diagrams	There are no exploration results reported for the immediate Jaguar mine area.
Balanced reporting	There are no exploration results reported for the immediate Jaguar mine area.
Other substantive exploration data	There are no exploration results reported for the immediate Jaguar mine area.
Further work	No further work is planned as the mine is near completion.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	The parent database for all collar, survey, geology and assay data is a SQL database with the acQuire software as the front end. This acQuire database has a number of built in fields and reports to ensure data are entered correctly and obey certain validation rules. Assay data are imported directly from laboratory files and merged with sampling data. Most other data are captured digitally and imported directly to the database with few opportunities for keying errors. All data with the parent Jaguar or OP-Jaguar project code are exported to a Microsoft Access database which is frozen in time as a permanent record of the database used for that resource estimate.
	Data are visually and graphically checked to ensure that there are no outlying errors. Any errors noted are corrected on an ongoing basis.
Site visits	The competent person, Graham Sweetman, is the Geology Manager at Jaguar operations and is based on-site on a 9 and 5 FIFO roster. He regularly checks procedures and processes used to collect data used for resource estimation.
Geological interpretation	Confidence in the geological interpretation for the Jaguar deposit is very high. The deposit has been mined for 6 years and the geological model has been robust over this time.
	The mineralisation and geological setting have been confirmed by significant underground development, drilling and mapping.
	There have been no alternative interpretations of the mineralisation model with the Jaguar deposit being well regarded as a V(H)MS style orebody.
	The geology controls the domaining of the Jaguar deposit into massive sulphide and stringer styles and the various lodes. These domains are used to constrain grade interpolation and to code mineralisation styles in the block model.
	Continuity of grade is excellent within the massive sulphide. Disruptions occur as dolerite sills intrude and separate the Main Lode from the Main Lode Split as well as offsetting the Far Side Lode stringer zone further into the footwall.
Dimensions	Jaguar (Main Lens) is 400m long, 420m wide (down-dip), up to 16m thick and located 320m below the surface. Mineralisation is continuous with low variability.
Estimation and modeling techniques	Ordinary kriging was used for grade estimation in the massive sulphide and stringer lenses. Inverse distance squared was used to interpolate the internal waste portion of the Main Lode lens. Both interpolation techniques are deemed appropriate for this type of deposit. Grade estimation was constrained to the massive sulphide lens wireframes for the Main Lode, Main Lode Split and Bubble Lode. For stringer zones, a 0.5%Cu cut-off was utilised for wireframe boundaries and grade estimation was constrained to within the stringer wireframes. Statistical analysis was carried out using Geoaccess software, with data constrained to the Main Lode for massive sulphide mineralisation and the Footwall Lode, Far Side Main and Far Side Split lodes for stringer mineralisation. Declustering prior to statistical analysis was not deemed necessary due to the regular drill pattern spacing throughout. The statistics showed a good coefficient of variation (below 1) for all elements in the Main Lode lens, and for the majority of elements in the stringer zones, however, Pb, Zn and Ag were top-cut to 0.35% Pb, 3% Zn and 50ppm Ag respectively due to higher coefficients of variation. There was no evidence to sub-domain any zones as mineralisation is fairly continuous in grade within the domains. Estimation and variography utilised Surpac V6.2 (2012 estimate) and Surpac V6.3 (2013 estimate) software. For both the massive sulphide and stringer mineralisation the search ellipse orientations were updated for the other lenses to fit the geometry of each of them, with the minor direction search radius being increased in pass 2 to allow for flexing of the wireframes The nugget for each of the elements was obtained using a downhole variogram. All major elements (Cu, Zn, Pb, Ag, Fe) encountered in the Jaguar deposit have been estimated. The block model interpolation uses a parent cell size of 10m Northing x 5m Easting x 10m RL. This cell size is half the drill spacing (20m x 20m) for the majority of the deposit, as well as the average thickness of the lenses. Subcell sizes are 1.25 Northing x 0.625 Easting x 1.25 RL. No assumptions have been made regarding the recovery of by-products. No assumptions have been made regarding selective mining units as the mining technique is well established, there is little variation in the orientation of the deposit and the current parent cell size with sub-celling is adequate. No assumption has been made on the correlation of variables and all variables have been interpolated into the model independently. The block model was validated using swath plots comparing the interpolated block grade for Cu, Zn and Ag with the composited sample data for the Main Lode, Footwall Lode and (combined) Far Side Lodes by northing and elevation to check if any model bias had been introduced. The results were very close with very little bias noted. Block grades were also compared graphically by comparing block element grades with drillhole assays. Both methods confirmed the block model was acceptable for use in mine planning. The resource estimates have reconciled well with production over the life of the mine. The Jaguar resource estimate is updated annually and compared with production. The tonnes and grade of the mined portion of the resource have also been compared to the reconciled life of mine production, with results showing a good comparison with the mine and mill production figures.
Moisture	Tonnages have been estimated using densities that contained natural moisture. The natural moisture of the Jaguar massive sulphides and volcanic rocks is assumed to be very low (<1%) but has not been measured. All rock types are fresh and impermeable.
Cut-off parameters	No cut-off grades have been applied to the massive sulphide mineralisation. Stringer mineralisation has been defined by a 0.5% copper lower cut-off grade as this allows for continuity of zones that could be amenable to mining. It also allows some flexibility for changes in metal price and NSR assumptions in the Ore Reserve estimation stages.
Mining factors or assumptions	No mining factors or assumptions have been made; the mine has been operating for 6 years.
Metallurgical factors or assumptions	No metallurgical factors or assumptions have been made; the mill on site has treated the ore sufficiently and successfully for 6 years.
Environmental factors or	No environmental factors or assumptions have been made; the waste dump is well established with approval from

Criteria	Commentary
assumptions	the Department of Mines and Petroleum (DMP).
Bulk density	The density has been determined for the majority of samples submitted for analysis. All underground half-core samples have measured densities using the water immersion technique. Those measurements taken prior to 1st June, 2010 have not been used in the estimate as the method applied was incorrect. The assays for Cu, Pb, Zn and Fe were compared with the measured densities for both massive and stringer mineralisation, and second power regression curves developed for each mineralisation style. These formulae were used to calculate densities for those samples without measured densities or with spurious measured density data. Densities and length were used for weighting in the sample compositing. The measured and estimated density data were interpolated into the associated mineralisation blocks in a similar manner as the grades. A background density of $2.77 g/cm^3$ was applied to the block model which was determined as the average density of the waste zones.
	Tonnages have been estimated using densities that contained natural moisture. The natural moisture of the Jaguar massive sulphides and volcanic rocks is assumed to be very low (<1%) but has not been measured. All rock types are fresh and impermeable.
Classification	The Measured resource was defined where drill hole spacing was nominally 20m by 20m, or mineralisation was within 20m of known development. Resources were defined as Indicated where the drill hole spacing was greater than 20m but still displayed good continuity. A number of 'main lode rafts' have been defined by a single drill hole or are not continuous across more than one section, and these zones have been classified as Inferred.
	Previous resource estimates, as well as mining reconciliation and mapped geology have been taken into account to give satisfactory confidence in tonnage/grade estimations and models.
	Confidence in the resource estimate is high and backed by excellent mining reconciliation.
Audits or reviews	No formal audits or reviews have been completed since 2011 but resource estimates are similar across several years and methods. The 2011 external review showed no major flaws in that estimate. The Jaguar deposit is near the end of its mine life so a review for the 2013 resource estimate is not warranted.
Discussion of relative accuracy/confidence	Confidence in the geological interpretation for the Jaguar deposit is very high, with the mineralisation and geological setting confirmed by significant underground development, drilling and mapping. Confidence in the resource estimate is also high with accuracy confirmed by the mined part of the resource (1,774,000 @ 2.8% Cu, 7.5% Zn 0.5% Pb and 90ppm Ag) reconciling well with the reconciled mine production (1,767,000 @ 2.7% Cu, 7.8% Zn, 1.4% Pb and 87ppm Ag) and reconciled mill production (1,703,000@ 2.8% Cu, 7.7% Zn, 0.5% Pb and 87ppm Ag).
Resource Model Number	JG_RSC_2013_06

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	All Ore Reserves estimated for the Jaguar Operation are a sub-set of the Jaguar and Bentley Mineral Resources. No reserves exist outside of the Mineral Resource base.
	Mineral Resources are inclusive of Ore Reserves.
Site visits	A site visit was undertaken by Brett Hartmann of IGO on the 17^{th} – 19^{th} of April 2013. In this visit all working areas were inspected. Philip Bremner of Mining One Pty Ltd conducted a site visit on the 20^{th} -21^{st} May 2013 as part of the review process.
Study status	The Jaguar Operation has a history of being mined by Jabiru Metals since October 2007. The mine reserves have been designed based off the current operational practices of the mine. All Ore Reserves are estimated by constructing three dimensional mine designs and reported against updated Mineral Resource block models. After modifying factors are applied, all physicals (tonnes, grade, metal, development and stoping requirements etc.) are input to a Reserve Evaluation model for an economical evaluation on a stope-by-stope basis.
	Previous mine performance has demonstrated that the current mining methods are technically achievable and economically viable. Material Modifying Factors have been considered and compared well to reconciled performance.
Cut-off parameters	Cut off values are calculated on the basis of the NSR (Net Smelter Return) calculation. The resource model is evaluated against the NSR cut off value and mining areas (stopes and development) are designed for those areas above the NSR cut off value. Once designed the entire mining area is evaluated on a stope-by-stope basis again, against the NSR cut off value. Two cut off values are utilised, a higher value ($180.00 per ore tonne) for direct mill feed and a lower value ($120.00 per ore tonne) for marginal costs.
Mining factors or assumptions	Three dimensional mine designs are designed based on known information about the orebodies physical characteristics and the geotechnical environment. Modifying factors such as unplanned dilution (20% for Long hole stoping and 5% for development) and reserve recovery (95% for stoping) are applied based on the chosen mining method. In some cases geotechnical losses are applied or *in situ* pillars are left, reducing the overall recovery factor.
	In certain cases where a mined stope contained both Indicated and Inferred Mineral Resources, the stope was only designed around the Indicated resources.
	No additional infrastructure was required for the mining of the current reserves.
Metallurgical factors or assumptions	Both Jaguar and Bentley underground reserves are processed at the Jaguar processing facilities. The process and recovery of both ore sources is well understood and reasonably consistent in performance. Conservative recovery factors has been used: o 82.0% Cu recovery into Cu concentrate o 53.0% Ag recovery into Cu concentrate o 43.0% Au recovery into Cu concentrate o 83.0% Zn recovery into Zn concentrate

Criteria	Commentary
	o 22.0% Ag recovery into the Zn concentrate
Environmental	Jabiru Metals operates under an environmental management plan, which meets or exceeds all environmental legislative requirements. Jabiru Metals' license to operate is in good standing. Environmental rehabilitation plans are constructed and activated upon when the timing is appropriate. The costing of the rehabilitation works is accounted for in the operations Life of Mine model.
Infrastructure	The current infrastructure at the Jaguar Operation is adequate for the extraction of both the Jaguar and Bentley underground reserves. Maintenance costs for current equipment were included in the reserve economic model.
Costs	Capital costs for decline development were included in the financial evaluation. An allowance per ore tonne is also made for ongoing exploration costs. Operating costs were updated against the previous twelve months actual costs. Concentrate payables, which includes accounting for any deleterious elements, has been calculated and used within the NSR evaluation process. All unit costs are updated from the most recent financial year's actual costs.
Revenue factors	The assumptions made for commodity prices are: Copper price US$ 7,694 per tonne, Zinc price US$2,270 per tonne, Silver price US$33.00 per troy ounce, Gold price US$1,740 per troy ounce and Foreign exchange rate of US$1.01.
	These values were selected after reviewing a number of industry recognised price forecasting leaders, which included Bloomberg and Brook Hunt.
	During the calculation of reserves the Metal prices were assumed fixed for the life of the project.
Market assessment	Longer term market assessments are provided by a number of independent companies such as Brook Hunt and Bloomberg. Market conditions are considered in part of the long term cost evaluation.
Economic	NPV was not taken into account in the economic review. The estimated life of mine is currently under five years and so fixed cost and prices were used.
	Sensitivity analysis work has been undertaken on variables such as head grade, tonnages, foreign exchange rate and metal price.
	The project is highly sensitivity to the foreign exchange rate (AUD:USD) and copper and zinc metal prices.
Social	Tenement status is currently in good standing.
Other	There are no other foreseeable risks associated with the Jaguar Operation on a sociological or political assessment.
Classification	Ore Reserves are based on geological and mining confidence and categorised as either Proved or Probable.
	This result appropriately reflects the Competent Person's view of the deposit.
	The proportion of Proved Ore Reserves is a subset of the Measured Mineral Resources. All Probable Ore Reserves have been derived from Indicated or in a small proportion Inferred Mineral Resources.
Audits or reviews	An independent audit is undertaken annually on both the Mineral Resource and Reserve process. No unexpected results have come from this review.
Discussion of relative accuracy/confidence	As part of the Ore Reserve estimation process a comparison is undertaken reviewing actual reconciled extraction versus previous years Ore Estimation and Resource Estimation.
	A review of last year's performance by the Competent Person found that both the Resource and Reserve estimation processes are conservative estimates.

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 37 additional diamond drill holes were drilled at Currawong since the previous resource estimate in 2011. An additional 34 diamond drill holes were drilled into Wilga, from both surface and underground. A total of 13,803.8m of additional drilling has been completed at Currawong and Wilga as part of the 2011/2012 drilling program. All new holes at Wilga were infill holes. New holes at Currawong were a mixture of infill and extension drilling with a maximum drill spacing of 25m X 25m. Only diamond drilling has been utilised for resource estimation at Stockman. Sampling of mineralised zones was predominantly half core with a nominal 1m sample length.
	Diamond drilling is solely used at Stockman to ensure a high quality of sampling. All sampling and check sampling is conducted to industry best practice in accordance with IGO QAQC protocols.
	The 2010-2012 drilling campaigns included a combination of sawn half-core NQ or quarter-core HQ, with a typical sample length of 1m. A minimum sample length of 0.15m and maximum sample length 1.5m in mineralised domains were adjusted to geological boundaries. All massive sulphide intercepts have been sampled and sampling generally extends 10m into waste rock. All drill core to be sampled from the Jabiru Metals Ltd (Jabiru) and Independence Group NL (IGO) holes, was marked up by the geologist. The sampling book was filled out detailing the from and to depths for each sample, the corresponding sample numbers as well as which standard to insert at which point and where to insert blank samples. Field technicians cut the core using an Almonte automated core cutting machine. The core was systematically cut 1cm off the orientation line to allow the orientation line to remain in the core tray. JML and IGO samples were cut, dried and pulverised for analysis by 4 acid digest, ICP/OES (Cu, Pb, Zn, Ag, Fe, S) and fire assay FA/AAS (Au) at and independent laboratory. Historic sampling involved crushing, with a sub-sample pulverised, followed by three or four acid digest with AAS or ICP determination. All samples apart from the WMC samples were prepared and analysed at independent laboratories. WMC and Denehurst did not routinely analyse for Au.
Drilling techniques	All JML and IGO holes were diamond drilled for the entire hole using a combination of HQ and NQ core sizes. Historical holes were principally diamond drilling with the exception of several RC precollars drilled by Denehurst and Austminex. None of the RC samples have been used in the resource estimates. The surface diamond drilling is a mixture of HQ, NQ and BQ core sizes, with BQ occurring only in the older WMC holes. The historic underground holes at Wilga were drilled LTK46 (Ø = 35.6mm).
Drill sample recovery	Drill sample recovery is logged and recorded by field technicians and subsequently entered into the acQuire database. Core sample recovery was good to excellent. Some lost core intervals have been recorded, particularly where structures such as faults or underground workings (Wilga) were intersected by the drilling. These intervals do not affect the resource estimate.
	The diamond drill core is reconstructed in the core yard as part of the orientation process and metre marks are checked against driller's depth blocks.
	One small area of poor sample recovery at Wilga has been identified and isolated. This area corresponds with the presence of chalcocite and its classification has been downgraded to Inferred. Recent core recoveries are reviewed annually to ensure there are no new areas of poor sample recovery. There is no evidence of bias or preferential loss or gain of material in samples except for the chalcocite zone mentioned above.
Logging	Entire holes were logged and photographed by the various companies completing the drilling programs. Geological and geotechnical logging is very thorough and more than adequate for resource estimation. Logging has previously been on paper logs, which were data entered and then loaded into the Acquire database. Paper logs were scanned and stored on the IGO Perth server. Starting In 2011, drillholes have been logged straight into a digital format via Acquire data entry objects which were then uploaded directly into the database. Acquire data entry objects have built-in rules that allow for validation of data as it is logged.
	Detailed logging routinely consisted of lithology, alteration, mineralisation, veining, structure, deformation and oxidation state and was recorded using the JML logging codes. JML/IGO core has been photographed both wet and dry. Historical geological codes were converted to JML codes in 2008.
	All drill holes were logged in full.
Sub-sampling techniques and sample preparation	Mostly cut half-core samples of NQ, BQ and LTK46, or quarter-core samples of HQ varying in length up to 1.3m in the massive sulphide and adjusted to geological boundaries. Some quarter-core NQ samples by Austminex where core was needed for metallurgical testwork. The JML/IGO drilling campaigns included a combination of cut half-core NQ or quarter-core HQ, with a typical sample length of 1m. A minimum sample length of 0.3m and maximum sample length 1.5m in mineralised domains were adjusted to geological boundaries. All massive sulphide intercepts have been sampled and JML/IGO sampling generally extends 10m into waste rock. The samples were routinely taken from the same side of the core in relation to the orientation lines.
	No non-core samples were taken in the 2010-2012 JML/IGO drilling.
	Samples from the 2010-2012 JML/IGO diamond drillholes were sent to Genalysis Adelaide for sample preparation and analysis. Sample preparation consisted of drying the core for 8 hours at 121°C then jaw crushing to a nominal minus 10mm size. Pulverising then occurred in a LM5 pulverising machine for 5 minutes to 85% passing 75 microns. The entire sample undergoes pulverising in the LM5 machines, resulting in no coarse rejects, only bulk pulp rejects. The sample preparation technique is normal industry practice and is considered suitable for Stockman samples.
	Quality control procedures during JML/IGO sampling included the insertion of certified reference standards and blanks (1 in 20 samples) as well as the inclusion of barren quartz washes between every sample.

Criteria	Commentary
	Historic drilling contained very little QAQC work.
	Apart from 62 duplicate samples collected by Macquarie Resources there were no field duplicates collected prior to the JML/IGO programs. JML/IGO field duplicates were taken during the 2010-2012 drilling campaigns. In addition, pulp repeats, bulk pulp repeats and cross lab pulp checks were completed on ~5% of the samples. All these quality control measures confirmed that sampling and sub-sampling techniques used were appropriate for the style of mineralisation and that samples were representative of the in situ material.
	The sample size is considered appropriate for massive sulphide mineralisation.
Quality of assay data and laboratory tests	All samples were crushed and a sub-sample pulverised followed by three or four acid digest with AAS or ICP determination. All samples apart from the WMC samples were prepared and analysed at independent laboratories. The assay techniques by JML/IGO are for total digestion of the sulphides and are considered appropriate for this type of mineralisation. For the JML/IGO drill programs, all samples were assayed at Genalysis Adelaide Laboratory using a 4 acid ore grade digest with an ICP-OES finish. Au was assayed using a fire assay 50g charge. Lower detection limits were to 50ppm for Cu, Pb, Zn, 1ppm/5ppm for Ag and 0.005ppm for Au.
	No geophysical or handheld XRF instrument data were used in this resource estimate.
	In comparison with modern requirements, minimal quality control procedures were adopted by companies completing the drilling programs before JML (eg. inclusion of only 17 field standards, 62 duplicates, 84 external laboratory checks in total). This shortfall was recognised by JML and more rigorous check sampling programs were implemented. For the JML/IGO drill programs, comprehensive QAQC programs were completed following company QAQC guidelines, which include the insertion of standards, blanks, duplicates and cross-lab checks. Results indicate that sample contamination is kept at a minimum and that assay values are within acceptable accuracy. In 2011, IGO also implemented particle sizing checks to be completed at the laboratory on 10% of the samples submitted for assay. These tests were to determine the pulverising quality of the samples.
Verification of sampling and assaying	All significant intersections were verified by alternative company personnel. No independent personnel verified any intersections.
	A total of 10 holes were drilled as twin holes by JML/IGO (4 at Wilga and 6 at Currawong). These showed that there was no bias between the twin and original holes but they did indicate that the degree of sulphide development is quite variable even over short distances. Consequently metal grades are quite variable also.
	An acQuire database was used by JML/IGO which includes all drilling information. Data are entered into the database mainly through acQuire data entry objects which have the required filters and validation rules built in. Data entry objects with built in validation tables are used to capture collar information, survey information (single shot), sampling information, geotech, and all geological logging information. Excel spreadsheets are used to capture downhole survey (multi-shot) data, surveyed collars and density information. All data entry objects and Excel spreadsheets were sent to the Database Administrator in Perth for uploading into acQuire. Assays received from laboratories were imported by the Database Administrator using customised acQuire importers thus alleviating any data entry mistakes.
	No adjustments were made to any assay data used in this estimate.
Location of data points	Most historic drillhole collar positions were surveyed by licensed or company surveyors. The JML/IGO (2008-2012) drillhole collar positions were located using RTK GPS equipment with a horizontal accuracy of +/-10mm and a vertical accuracy of +/-20mm. Historical drilling includes generally good quality surveys using downhole camera shots at about 30m intervals. Initial JML/IGO downhole surveys were taken by the drillers every 30m using the ORI-Shot digital camera. The results from the downhole camera were checked at the end of every hole and prior to uploading into the acQuire database. In addition, at the end of hole, a multi-shot survey was taken which recorded a reading every 6m. These multi-shot surveys were transferred to the site geologists digitally at the end of every hole, and then uploaded into the acQuire database.
	Since 2008, all drilling information has been converted into Stockman Regional Grid (SRG). This grid was created by JML in 2008 and extends over the Currawong and Wilga deposits. All holes were collar surveyed in MGA94 grid and transformed to SRG in MapInfo using transformation ties.
	Topographic surface is a DTM created from height measurements collected during an aeromagnetic survey during 2008. All historical drillhole collars were surveyed by a surveyor and all JML/IGO drillhole collars were surveyed up with an RTK GPS with a nominal height accuracy of +/-20mm.
Data spacing and distribution	No exploration results are included in this report.
	Diamond drill coverage in the massive sulphide at Wilga and Currawong is on a nominal 25x25m pattern. In the stringer sulphide lenses of both deposits, drillhole spacing ranges from 25x25m to 50x50m. Minimum hole spacing ~10m and maximum hole spacing ~70m. In general, drillhole spacing of less than 50x50m is classed as Indicated whereas drillhole spacing greater than this is classed as Inferred. No part of the resource at Currawong is classified Measured due to the nominal required drillhole spacing of 25x25m in the massive sulphide, as well as existence of multiple generations of drilling. The data spacing and distribution is more than sufficient to establish geological and grade continuity appropriate for the Mineral Resource estimation procedure and classification applied.
	Drillholes were composited to 1m downhole with length and density weighting. Face sampling at Wilga and recent probe drillholes at Wilga were not used for grade interpolation nor were the down plunge holes at Currawong.
Orientation of data in relation to geological structure	Surface drilling intersects the massive sulphide lenses almost perpendicular to the lens orientation at both Currawong and Wilga. The underground fan drilling at Wilga has some intercepts that are almost dip parallel. Some sample bias will occur in the Wilga deposit due to this fan drilling orientation but most of the affected area has already been mined and is excluded from the resource estimate. Two down-plunge or down-dip holes were drilled at Currawong however these were excluded from the estimate. They were drilled to detect offsetting faults, cross-cutting intrusions and test the grade continuity along strike. In the resource estimate they were used solely for geometry purposes. No down-plunge or down-dip holes were drilled at Wilga.
	Three of the 2012 stringer drillholes at Wilga were drilled at low angles to the mineralisation due to the lack of more appropriate drilling locations. These holes also do not represent a large volume of the resource estimate and are not

Criteria	Commentary
	considered material.
Sample security	Drill core was transported from the drilling site to the Stockman core yard by JML/IGO personnel on a daily basis. All samples are stored in the Stockman core yard which is either manned or locked at all times. They are then transported to the assay laboratory in Adelaide using Toll IPEC. All deliveries are tracked using consignment numbers. Once they are received at the laboratory, the samples are reconciled against the sample despatch.
Audits or reviews	The Stockman database was rigorously checked during a data compilation and validation stage in 2008. Since then, routine validation of the database has been conducted in-house. M Wild (IGO Principal Resource Geologist) completed an onsite review of drilling and sampling techniques in February, 2012. The procedures in place were considered to be of a suitable standard for the drilling data to be included in this resource estimate. In addition, laboratory audits were completed for the Genalysis Adelaide Laboratory by B Kendall (IGO Principal Geologist – Advanced Projects) in February 2010, and by M Wild (IGO Principal Resource Geologist) and K Kitchen (Stockman Senior Project Geologist) on the 29th February, 2012. No major issues were identified during these visits.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The Currawong and Wilga deposits are both within MIN5523 held by Stockman Project Pty Ltd, a wholly owned subsidiary of IGO. There are no native title claims registered over the lease, but an agreement is in place with a previous claimant group that makes provision for both the previous claimants and/or other indigenous groups who may assert an interest in the future. The tenement is located on crown land administered by the Department of Sustainability & Environment. The area is rugged and heavily forested with no significant heritage sites identified.
	The tenure was secure at the time of this report. No significant impediments are believed to exist.
Exploration done by other parties	Exploration at the Stockman Project was initially carried out by WMC in the early 1970s, WMC discovered both Currawong and Wilga deposits. Subsequent exploration has been completed by Macquarie Resources, Denehurst, Austminex, JML and IGO.
Geology	Currawong and Wilga are V(H)MS style deposits, occurring as polymetallic (pyrite-sphalerite-chalcopyrite) massive sulphide lenses with stringer feeder zones within a volcano-sedimentary succession. Wilga is a single stratabound lens whereas Currawong comprises multiple stratabound lenses with a series of faults offsetting and stacking the lenses.
Drill hole Information	There are no exploration results reported for the immediate Currawong and Wilga areas.
Data aggregation methods	There are no exploration results reported for the immediate Currawong and Wilga areas.
Relationship between mineralisation widths and intercept lengths	There are no exploration results reported for the immediate Currawong and Wilga areas.
Diagrams	There are no exploration results reported for the immediate Currawong and Wilga areas.
Balanced reporting	There are no exploration results reported for the immediate Currawong and Wilga areas.
Other substantive exploration data	There are no exploration results reported for the immediate Currawong and Wilga areas.
Further work	No further work is planned.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	An acQuire database is used by IGO which includes all drilling information. Data are entered into the database mainly through acQuire data entry objects which have the required filters and validation rules built in. Data entry objects are used to capture collar information, survey information (single shot), sampling information, geotech, and all geological logging information. Excel spreadsheets are used to capture downhole survey (multi-shot) data, surveyed collars and density information. All data entry objects and Excel spreadsheets were sent to the JML/IGO Database Administrator in Perth for uploading into acQuire. Assays received from laboratories were imported by the Database Administrator using customised acQuire importers thus alleviating any data entry mistakes. The acQuire database for the Stockman Project is exported to an Access database for resource estimation work.
	Ongoing data validation checks include visual checks in Surpac of collar, downhole surveys as well as checks between logging and assays received. Most of the data validations occur during the importing process and are built in to the acQuire database.
Site visits	The competent person for this report, Bruce Kendall (IGO Principal Geologist - Advanced Projects) was employed at the Stockman Project as Project Manager until January 2012. Whilst at the Stockman Project, he was closely involved in the planning and management of the drilling programs. Another site visit by Bruce Kendall was conducted in August 2012, where the resource estimate was reviewed. There has been no further drilling since that date.
Geological interpretation	Confidence in the geological interpretation for Wilga is high, with the mineralisation and geological setting being simple and the availability of underground drilling, mapping and plans confirming the interpretation. Currawong is more structurally complex and whilst confidence in the geological interpretation is good, additional drilling and further data review may result in modifications to the detail of the geological model, but this is unlikely to have an impact on the estimate.
	Thorough geological logging of all drill holes formed the basis of the geological interpretations. East-West sections were used to create mineralisation wireframes of both deposits. Several of the mineralisation wireframes were also constrained by shear planes, particularly at Currawong. At Wilga, historic backs mapping of development drives has been used to confirm mineralisation boundaries.
	The confidence in the geological interpretation, in particular of the mineralisation domain, is high. All infill drilling completed has supported the current geological interpretation. It is thought that any alternative interpretations will not have an impact on this estimate.
	Both deposits have been modelled using the massive sulphide as the main geological constraint. The main factors controlling continuity at Currawong are a series of post-mineralisation faults which are interpreted as disrupting the lenses. Controls on stringer mineralisation are essentially independent of the host sequences and lithology was not used to constrain the resource estimation for the stringer mineralisation.
	At Wilga, minor structures within the massive sulphide have been mapped which affect the distribution of the high grade copper massive sulphide, otherwise this is a continuous lens. At Currawong, faulting has controlled the geometry of the Currawong mineralisation. Some stacking may be the result of early growth faults during the formation of the massive sulphide lenses. Observed D2 shearing has dislocated many lenses and appears to be responsible for the termination of some. The extent of the D3 faulting is less certain but is thought to terminate the down dip portion of some lenses.
Dimensions	Currawong (Main Lens) is approximately 300m long, 240m wide (down-dip), up to 35m thick and located 100-300m below surface. Wilga is about 400m long, 220m wide (down-dip), up to 35m thick and located 50-150m below surface.
Estimation and modelling techniques	Ordinary kriging was used for grade estimation utilising Surpac software (v6.2) for Cu%, Pb%, Zn%, Fe%, Ag ppm, Au ppm and As ppm. Bulk density values were interpolated as for the other elements. Search parameters were based on variogram models for each element and density (variography also completed using Surpac v6.2 software). The various mineralisation wireframes were intersected with the drillholes in the database and the resulting intervals were written to tables in the Access database. Density weighted 1m composites were created using the lens coding as the control with a minimum passing of 50%. Grade estimation was constrained to the massive sulphide lens and stringer sulphide lens wireframes. At Wilga and Currawong, additional, internal subdomains of high grade Cu and Zn (Cu>1.2%, Zn>3%) were included in the massive sulphide lenses. No dilution was included in the resource models for Wilga or Currawong. Grade estimation for Au at Wilga may not be reliable due to a paucity of Au assays in the historic sample data and so Au is classified as Inferred at Wilga. Mild top-cut grades have been used for some elements where required. For Currawong, variography was performed on the M Lens massive sulphide (the largest of the massive sulphide lenses) and the kriging parameters obtained from M Lens variography were applied to the other massive sulphide and all the subordinate sulphide lenses. This approach was used as all the massive sulphide lenses are interpreted to be originally part of the same massive sulphide horizon which has subsequently been structurally disturbed into the different lenses. Variography for the stringer domain was conducted on the main stringer zone. For Wilga, variography was conducted on the main massive sulphide lens (most massive sulphide mineralisation is within the one lens) and the main stringer zone. Variography was conducted on Cu, Pb, Zn, Ag, Au, As and Fe as well as density.
	There is a 10% increase in global tonnage compared with the previous 2011 estimate due to additional drilling at both deposits. The grades remained consistent with the previous estimate.
	No assumptions were made regarding the recovery of by-products.
	As part of this estimate, the deleterious element As was estimated along with the economic elements. Fe was also estimated as it is important from a metallurgical perspective.
	Currawong 10mX, 10mY, 10mZ parent cell size as this is approximately ½ the average drill hole spacing. At Wilga

Criteria	Commentary
	10mX, 10mY, 5mZ parent cell size was used as this is approximately ½ the average drill hole spacing. For both deposits, subcelling to 1.25m in all directions was used to ensure adequate delineation of mineralisation boundaries. The size of the search ellipses was determined from the variography for each element.
	No selective mining units were assumed in this estimate.
	Correlation matrices were produced for each separate mineralisation domain. In general, As, Ag, Au and Pb display good positive correlations in all mineralisation styles. Grades were interpolated independently into the block model, assuming no correlation with each other, and based on variography for each element.
	The individual massive sulphide and stringer sulphide wireframes were used to code the block model with a unique identifier. The composite files for each domain were then used to estimate only the blocks which were attributed the same zone coding.
	No cut-off grades have been applied to the massive sulphide outer boundary but cut-off grades were applied to help delineate the high grade Cu mineralisation (1.2%Cu) and the high grade Zn mineralisation (3%) within the massive sulphide zones for both deposits. Cut-off grades were also used to delineate the stringer mineralisation at both Wilga and Currawong. These cut-off grades were 0.5% Cu or 2% Zn. Mild top-cut grades have been used for elements where the Co-efficient of Variation was > 1.0. The top-cut grades were determined from disintegration points on log probability plots. (Currawong massive sulphide 8% Pb, 10g/t Au, no top-cut for Zn, Ag or Cu; Currawong stringer sulphide 3% Pb, 13.9% Zn, 106g/t Ag, 10g/t Au, no top-cut for Cu; Wilga massive sulphide 26% Cu, 4% Pb, 31% Zn, 110g/t Ag, 2.6g/t Au; Wilga stringer sulphide 15% Cu, 1% Pb, 11% Zn, 120g/t Ag, 0.95g/t Au). A geological constraint (the massive sulphide zone) has been used as it is stable and will not vary over time, unlike cut-off grades. Mineralisation within both the massive sulphide and stringer lenses has been reported.
	Initial visual validation was completed by comparing drillhole assays with modelled values. A comparison was also completed to ensure the volumes of the wireframes closely resembled the block modelled volumes. The interpolated block grades were compared to the composited sample data and the declustered sample data (obtained via a nearest neighbour model created in Surpac) for each of the lenses by easting and by elevation to check if any model bias has been introduced.
Moisture	Tonnages have been estimated using densities some of which were dry (those analysed at external laboratories) and others that contained natural moisture. The natural moisture of the Stockman massive sulphides is typically low (<0.5%).
Cut-off parameters	No cut-off grades have been applied to the massive sulphide outer boundary but cut-off grades were applied to help delineate the high grade Cu mineralisation (1.2% Cu) and the high grade Zn mineralisation (3% Zn) within the massive sulphide zones for both deposits. Cut-off grades were also used to delineate the stringer mineralisation at both Wilga and Currawong. These cut-off grades were 0.5% Cu or 2% Zn. Mild top-cut grades have been used for elements where the Co-efficient of Variation was > 1.0. The top-cut grades were determined from disintegration points on log probability plots. (Currawong massive sulphide 8% Pb, 10g/t Au, no top-cut for Zn, Ag or Cu; Currawong stringer sulphide 3% Pb, 13.9% Zn, 106g/t Ag, 10g/t Au, no top-cut for Cu; Wilga massive sulphide 26% Cu, 4% Pb, 31% Zn, 110g/t Ag, 2.6g/t Au; Wilga stringer sulphide 15% Cu, 1% Pb, 11% Zn, 120g/t Ag, 0.95g/t Au). A geological constraint (the massive sulphide zone) has been used as it is stable and will not vary over time, unlike cut-off grades. Mineralisation within both the massive sulphide and stringer lenses has been reported.
Mining factors or assumptions	Mining of the Currawong and Wilga deposits is planned to occur using underground mechanised mining techniques. No assumptions regarding minimum mining width or dilution have been made. The resource estimate is undiluted.
Metallurgical factors or assumptions	A detailed metallurgical testwork program has been completed using samples from drill holes drilled during the period 2008-2011. Results indicate all styles of mineralisation are amenable to being recovered by flotation with no issues apparent due to deleterious elements.
Environmental factors or assumptions	Investigations are ongoing into suitable waste and tailings disposal options for the Stockman Project. A preferred option for both tailings and waste was selected as part of the Feasibility Study. Although these are yet to be approved by the regulating authorities, they have been fully informed of the preferred option.
Bulk density	Many samples had measured densities using either water immersion or air pycnometer techniques. All JML/IGO samples were measured for density using water immersion techniques. For those samples with no density measurement, a calculated density was applied to the sample. The assays for Cu, Pb, Zn and Fe were compared with the measured densities and a second power regression curve developed for each deposit and for each mineralisation style. Densities were used in the sample compositing. Tonnages have been estimated using densities some of which were dry (those analysed at external laboratories) and others that contained natural moisture, expected to be <1%.
	No samples were sealed prior to bulk density determination due to low porosity in the mineralised zones.
	Density was kriged into the block model in a similar method as was used for all other elements. However, a density regression formula was required in order to assign densities to historical samples which did not already have a density measurement. This was achieved in excel by ascertaining a multi element regression formula based on the existing assays and their corresponding measured densities.
Classification	Classification was based on sample density and confidence in the geometry of the lenses. All of the major massive sulphide lenses in both deposits were classified as Indicated. Stringer sulphide was classified as Indicated or Inferred or sometimes left as Unclassified if there is limited repeatability across sections. Generally, where the sample density was 50x50m or less the resource was classified as Indicated, where the spacing was greater than 50x50m the resource was classified as Inferred. The Au grades at Wilga are considered Inferred due to a paucity of gold assays in the historic drilling data.
	The classification has taken into account the quality, quantity and distribution of the input data. In addition, the high confidence in the geological interpretation and modelling parameters were taken into account.
	The Mineral Resource estimate reflects the Competent Person's view of the Currawong and Wilga deposits.

Criteria	Commentary
Audits or reviews	No audits or reviews have been completed on this particular Mineral Resource Estimate. The previous estimate (2011) was reviewed by Cube Consulting Pty Ltd and several recommendations were implemented in this update. No significant issues were identified.
Discussion of relative accuracy/confidence	The 2009 and 2011 resource estimates were independently reviewed and the classification and resource estimation method of Ordinary Kriging were deemed to be appropriate. The same estimation methods including recommendations made during previous reviews were implemented in this resource estimation. The 2012 Mineral Resource estimate correlates well with previous resource estimates. There are no known significant factors that might impact the accuracy and confidence of the estimate. Mineralisation has been classified as Indicated and Inferred. No mineralisation has been classified as Measured.
	The statement relates to global estimates of tonnes and grade.
	No production data are available for Currawong as it has not been mined previously. There is a slight discrepancy between the historic total reported tonnes mined at Wilga (956kt) and the calculated tonnes mined using the volumes of underground void models (802kt), with the reported tonnes being greater. During 2012, Wilga was re-opened and all voids above the current water table checked to see if the wireframes were accurate. Below the water table several holes were drilled to test for the presence of voids which were not indicated by the void wireframes, in areas of high grade. Although some discrepancies were identified they do not entirely account for the difference. The difference, since revision of the void model after probe drilling and access to the underground workings down to the water table, is 154kt, only 4.2% of the Wilga resource tonnage. This Mineral Resource estimate assumes the void model as being correct and the resource model was depleted accordingly.
Resource Model Numbers	CU_RSC_2012_07 and WG_RSC_2012_07

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	Mineral Resource estimates were created using Ordinary Kriging. Variography was completed on Cu, Pb, Zn, Fe, Ag, As, Au and density. The Mineral Resource estimate was completed in June 2012 and covers both Wilga and Currawong deposits. This Ore Reserve was derived from resource block model currawong_2012.mdl and resource block model wilga_2012.mdl
	The Mineral Resources reported are inclusive of the Ore Reserves.
Site visits	The site was visited by Mr Geoff Davidson in August 2008. During the site visit diamond drill core for Currawong was inspected, visits were made to the existing TSF of Lake St Barbara , the old plant site at Waxslip spur and both Wilga and Currawong portal sites.
Study status	This Ore Reserve was based on designs and estimates consistent with a detailed Feasibility Study. The costs were derived from Vendor estimates specific to the project and are considered to be within +/- 15% order of accuracy.
	A detailed mine plan was developed from which a practical mining schedule was determined. Standard modifying factors associated with the selected mining method have been applied. The mining method will use long hole open stoping techniques to recover economic mineralisation. Amongst others, the study included geotechnical analysis of the mine openings and detailed analysis and design of the paste backfill and its application in the mining method.
Cut-off parameters	The Net Smelter Return (NSR) method was used to determine the economic cut-off for the mineralisation. The NSR values were calculated on a 'mine gate' sale basis and incorporate metal pricing current at the time. The NSR value was adjusted for transport costs, port handling charges and TC/RC on all payable metals. Payable metals are copper, zinc, gold and silver.
	Gold (Au) grades are classified as Inferred at Wilga due to a paucity of gold assays in historic drilling. Revenue from gold in the Wilga ore was included in the estimation of the Ore Reserve. The contribution to revenue of this gold was estimated to be \$3.84 per gram of gold *in situ*. This inclusion was not material to the value of the mining envelopes considered and did not warrant downgrading of any portion of the Ore Reserve attributable to Wilga. The tonnage contribution from Wilga represents 13% of the total Ore Reserve tonnage.
	The cut-off NSR value was determined from the site operating costs including mining, processing and site administration and overhead costs. The cut-off value was estimated to be between \$92 and \$104 per tonne processed. An incremental cut-off of \$60 per tonne was also estimated as a subset of these costs and represented the minimum value of material economic to process once delivered to the surface stockpile.
Mining factors or assumptions	The Ore Reserve was determined by digitising practical stope wireframes around contiguous blocks of Indicated material above the cut-off value. The wireframes were expanded by 0.5 m to include unplanned mining dilution from over break. An additional 2% allowance was included in the dilution of certain stopes where there would be significant exposures of paste backfill during mining. A nominal 5% ore loss was applied to account for losses such as under-break, unrecovered bridges and toe, ore lost due to excessive dilution from fall dirt, misclassified ore, localised variations in the ore outline which cannot be efficiently removed during mining and reduction in the stoping boundaries as a consequence of updated resource information. In addition, any development outside the stope wireframes which reported an average value above an incremental NSR cut-off of \$60 per tonne was also included in the Reserve.
	The Ore Reserves for both Currawong and Wilga were determined on the basis of long hole open stoping using paste backfill. This mining method and associated parameters used to estimate the Ore Reserve were deemed to be appropriate for the nature and geometry of the deposits at Currawong and Wilga.
	Stope spans and other ground support requirements were determined from analysis conducted by geotechnical consultants Mining One Pty Ltd. Grade control methods would entail methods used by IGO at their existing operations in WA and will include stope definition diamond drilling, face and stockpile sampling.

Criteria	Commentary
	The Mineral Resource estimate was prepared and reported by IGO geologists in accordance with the JORC Code (2004), and was recently updated to comply with the JORC Code (2012) reporting requirements. Ordinary Kriging was used to estimate the grade of key elements such as Cu, Zn, Au, Ag, Pb, Fe and As.
	Sufficient detailed analysis was carried out to provide confidence in key assumptions such as stability of stope spans and mining rate. Testwork of the paste backfill has demonstrated degradation over time (260 days) which has been compensated for through the increased addition of binder which has maintained strength values well above the minimum threshold. Test work is ongoing to examine the long term performance of the paste.
	The method used to apply dilution estimated the addition of material to be approximately 10%. The mining method requires total extraction within the stoping envelope; therefore, no losses will occur from ore tied up in pillars. A nominal 5% ore loss was applied for reasons as described above. A minimum mining width of 2 m (true width) was used to digitise stope wireframes.
	Capital infrastructure for the proposed mining methods at Currawong and Wilga will require conventional decline access and primary ventilation shafts and tunnels as well as services infrastructure such as electrical distribution, air and water reticulation, dewatering facilities, communications, and refuge chambers. In addition, a backfill paste plant will need to be constructed and paste reticulated throughout the stoping areas of the Currawong mine. Paste will be trucked to Wilga using agitator trucks and discharged into a borehole and reticulated throughout the mine.
	Inferred Mineral Resources were not included in the Ore Reserve.
	Whilst the Wilga Ore Reserve only includes blocks classified as Indicated due to confidence in copper and zinc grades, gold grades within these blocks were considered to be Inferred due to a paucity of gold assays in historic drilling. Revenue from gold in the Wilga ore was included in the estimation of the Ore Reserve. The contribution to Revenue of this gold was estimated to be \$3.84 per gram of gold *in situ*. This inclusion had no material impact to the value of the mining envelopes considered and did not warrant downgrading of any portion of the Ore Reserve attributable to Wilga. The tonnage contribution from Wilga represents 13% of the total Ore Reserve tonnage.
Metallurgical factors or assumptions	The metallurgical process will use differential floatation to produce separate concentrates of copper and zinc.
	The processing method is commonly used throughout the world for the style of mineralisation that exists at Stockman and is currently being used at IGO's Jaguar operations.
	Numerous composite samples have undergone batch kinetic testing. The samples tested were selected from different geological domains from both Currawong and Wilga deposits. Geo-metallurgical algorithms were developed for the mineralisation at Stockman. The recoveries therefore vary depending on the combination of minerals present in the feed at any increment in time. The average recoveries across the life of mine (including ramp-up) were estimated from monthly schedules and are as follows: o Life of mine recovery of Copper to copper concentrate= 80% o Life of mine recovery of Gold to copper concentrate = 17% o Life of mine recovery of Silver to copper concentrate = 42% o Life of mine recovery of Zinc to zinc concentrate= 76% o Life of mine recovery of Silver to zinc concentrate = 17%
	Metallurgical test work has demonstrated that marketable concentrates of both copper and zinc can be produced from both deposits. Marketable electrolytic grade zinc concentrates are produced from both deposits when treating lower lead grade feeds (<1% Pb). Arsenic is low (<0.25%), iron is acceptable (8-10%), lead relatively low (<2%) and silica is also acceptable (<1.5%).
	The penalty element assays were generally low, but where slightly elevated, remained in the negotiable range for settlement. No provision was made in the NSR estimate for penalty elements due to limited likelihood of breaching threshold values. Deductions for penalty elements were however applied in the cash flow model in the periods where threshold values were exceeded. The life of mine estimated cost of penalties represents < 1% of the project operating cost.
	Locked cycle tests, which are designed to simulate a continuous and stable condition of the proposed flotation process, were conducted on a range of composited samples considered to be representative of the various types of mineralisation, including a blend representative of the first 5 years production. Locked cycle test results by previous owners observed similar results to those conducted by IGO.
	No bulk samples or pilot scale testing has been carried out.
	Economic concentrations of minerals were defined by their intrinsic value derived through beneficiation to produce concentrates within marketable specifications. The commercial value was determined through the application of an economic cut off, as described above. No other mineralogical specifications were applied in determining the Ore Reserve however charges were applied to the concentrate product where the estimated level of penalty elements exceeded threshold levels. These penalty elements included Zn and Pb for Copper concentrate and Fe for Zinc concentrate.
Environmental	No permanent waste rock landforms will be created during operations. All material determined to be potential acid forming (PAF) or containing soluble metals will be either returned underground as backfill for workings or disposed of sub-aqueously in the tailings storage facility.
	Tailings produced from on-site processing will be either returned to the underground workings as backfill or disposed of in an approved tailings storage facility (TSF). The existing decommissioned TSF will be reinstated to accept the tailings from operations. The proposed TSF has been designed in accordance with the Australian National Committee on Large Dams (ANCOLD) guidelines.
	Water produced from dewatering the underground workings will be treated and recycled for use in the mining or processing operations. Surplus treated water will be discharged into the TSF.
	An Environment Effects Statement (EES) was compiled as a requirement of the Victorian state government project approval process. The EES prepared for the Stockman base metals project provides a comprehensive and integrated assessment of the potential environmental, social and economic impacts of project implementation. Technical studies conducted for the project provided confidence that the project can be implemented in a way that is consistent with relevant Victorian and Commonwealth government environmental and social policy objectives.

Criteria	Commentary
	Project licensing and approval, including permitting of the TSF is subject to a favourable assessment by the Victorian Minister for Planning of the EES and approval by the federal Environment Minister under the EPBC Act. Both these processes are still in progress.
	There are no known impediments to the approval process; however, the review and approval process is still in progress and project development will be subject to the conditions placed on the project by respective regulators.
Infrastructure	The project will be supported by limited existing infrastructure. The project is currently serviced by an existing access road which will be upgraded to accommodate increased traffic and concentrate transport. Telecommunications are available within the wider area; however, repeater stations will be required to bring these services to the site. Power will be generated on site using compressed natural gas sourced from a proposed compressor station at Bairnsdale. Supplemental process water will be sourced via a borefield located adjacent to the Benambra township and piped to site. The availability of labour is limited in the immediate area and an accommodation village will be constructed to house a drive-in drive-out workforce with most personnel expected to commute from regional population centres.
	IGO currently holds the mining lease on which the Stockman project is located (MIN5523). Land access to other support infrastructure is the subject of various draft agreements and Memoranda of Understanding (MOU's) with respective land holders. The site for the TSF is currently located within an Exploration Exemption area and application to have this lifted will be subject to approval of the proposed facility by state and federal regulators. Any new tenement covering the current Exploration Exemption area, if granted, would be as a separate tenement to MIN5523. There are no known impediments to the granting of this license.
Costs	Capital costs for the project were based on budget quotations provided by potential vendors based on a design and scope specific to the project. Where vendor quotations were not available, cost estimates were provided by consultants with expertise in their specific field or were built-up from first principals based on IGO operational experience.
	Mining capital and operating costs were estimated from first principals using vendor quotations for materials and equipment running costs. Productivities were based on internal industry experience.
	Labour costs were based on an assessment of independent surveys of the Australian mining industry.
	Provision was made within the cash flow analysis for the penalties applied to deleterious elements in excess of the limits proposed by independent metal traders.
	Road transport costs and port handling charges were based on vendor quotations specifically for the project scope of work. Sea freight charges were based on market assessment by logistics consultants with expertise in this industry.
	Treatment charges, refining costs and element penalties were based on budget quotations provided by recognised metal traders. Consideration was also given to existing contracts in place at the IGO's Jaguar operations.
	Victorian government standard state royalties were applied to Copper, Zinc and Silver. No royalty was applied to Gold. Under Part 2, Section 7 of the Mineral Resources Development Regulations 2002, state royalties do not apply to gold.
	No third party royalties are applicable to this project.
Revenue factors	The project head grade was determined on a month by month basis from a detailed schedule of mining of the Ore Reserve. The schedule incorporated a logical development and extraction sequence of the Ore Reserve and utilised productivity rates commensurate with industry standards.
	Provision was made within the cash flow analysis for the penalties applied to deleterious elements in excess of the limits proposed by independent metal traders.
	Transport costs, port handling charges and sea freight charges have been discussed in the section on Costs.
	Smelter recoveries, treatment charges, refining costs and element penalties were based on budget quotations provided by recognised metal traders and were in line with standard contracts for copper and zinc concentrates. Consideration was also given to existing contracts in place at the IGO's Jaguar operations.
	The commodity prices and exchange rates used for the cash flow model were applied as flat forward real pricing and were based on spot prices current as at January 21st 2013.
	Metal prices and foreign exchange rate used in the cash flow model were as follows: o Copper $US 8068.84 per tonne of copper metal o Zinc $US 2,072.32 per tonne of zinc metal o Gold $US 1,687.00 per ounce troy of gold o Silver $US 32.32 per ounce troy of silver o Exchange rate of $1.05 $AU per $US
	The prices used for the cash flow model were applied as flat forward real pricing and were based on spot prices current as at January 21st 2013. The cash flow was modelled in real terms and no price or cost escalation was applied.
Market assessment	In its June 2012 Long Term Outlook, Wood Mackenzie forecast the average growth rate in copper demand to be around 3.5% for the next 15 years.
	In its June 2012 Long Term Outlook, Wood Mackenzie estimated the global consumption of zinc to be circa 13 Mt and the forecast average growth rate to average 3.7% over the next 15 years.
	Wood Mackenzie conducted an analysis of the copper supply side of the market. This included the contribution made by scrap metal from refinery and smelter processes. These contributions are forecast to be relatively constant over the next 15 year period, supplying around 20% of the global demand.
	Wood Mackenzie conducted an analysis of the zinc supply side of the market. The general outlook for mine production was for short-term growth to 2014, followed by a decline in global production due to numerous closures as mining reserves are depleted. By 2020, global output will be back at current levels following the 2014 peak and continue to fall to around 20% below current levels by 2025.
Economic	A detailed cash flow model was created using the design case commodity pricing described above. The cash flow included detailed schedule of Capital and Operating cost expenditures for each of the project cost centres. Revenue from product sales were modelled by shipment with 90% payable in the month of loading and the balance paid the following month. Typical off take contracts were incorporated in the cash flow and were based on input parameters

Criteria	Commentary
	provided by recognised industry metal traders. The cash flow was modelled in real terms, hence no price or cost escalation was applied. A discount rate of 8.6% was applied using the weight averaged cost of capital (WACC) method to determine a Net Present Value (NPV) from the project cash flow.
	The cash flow analysis demonstrated a positive return for the project with a pre-tax internal rate of return of 10%.
	Input costs were considered to be accurate to within +/- 15%. Costs were taken either directly from vendor quotes or consultant estimates for specific scopes of work. Mining costs were developed from first principals on an owner operator basis.
	Various sensitivity analyses were carried out on the cash flow model. Key parameters were varied by +/- 15%. These parameters included metal prices, foreign exchange rate, treatment charges, capital and operating costs. The results were evaluated on the basis of pre-tax operating cash flow less capital. With the exception of the foreign exchange rate, all parameters tested returned a positive result. Foreign exchange was considered to be a manageable risk through the implementation of currency hedging.
Social	There are currently no Native Title claims or determinations over the Stockman project area. A license for the mining lease has been granted.
	A program of community engagement has been undertaken and will continue through the life cycle of the project. This has included the establishment of a “shop front” to facilitate two-way communications with the public. No material objections to the project have been received throughout the community engagement process and the general consensus is one of positive economic benefit to the local community.
	An MOU has been executed between IGO and the East Gippsland Shire Council. The MOU commits both parties to working in collaboration to identify and progress opportunities that will deliver social and economic development benefits for the region whilst, through endeavouring to maximize the efficiency and robustness of the project’s operations, not compromising or placing an unnecessary financial burden on IGO as a company with obligations to its shareholders.
Other	IGO currently holds the mining lease on which the Stockman project is located. Land access to other support infrastructure is the subject of various draft agreements and MOU’s with respective land holders. The site for the proposed TSF is currently located in an Exploration Exemption area. An application for an Infrastructure Mining Licence will be made following project approval by IGO. There are no known impediments to the granting of this license.
	The project is located in state forest which is prone to bushfires. Analysis of the risk has been undertaken by independent consultants WSP and mitigation measures recommended including the establishment of fire protection zones and fire-resistant construction materials. In addition, procedures and training for bushfire events will be implemented as part of the project Work Plan and procedures.
	Land access agreements and MOU’s for required external infrastructure have been tabled in draft form to the following stakeholders: o Local pastoralist for land access to the Stockman Village site and borefield site. o East Gippsland Shire Council for land (road verge) access for the borefield pipeline and high voltage underground power cable. o Commercial land owner to access land for the Bairnsdale CNG compressor station. o GeelongPort for land access for the storage of half height sea containers used for concentrate transport.
	The project will require vegetation offsets for ground required to be disturbed for construction and mining. These offsets have largely been identified and secured in part or subject to a draft heads of agreement with existing land holders. There are no known impediments to securing the final calculated offsets required for the project.
	State and federal approval of the project will be subject to acceptance of the Environmental Effect Statement (EES). The project viability will be reassessed based on the conditions imposed on these approvals. No material changes to the project are anticipated as a result of any foreseeable conditions that may be imposed by the state and federal ministers.
	There are currently no unresolved matters with third parties.
Classification	The Ore Reserve was classified in accordance with the JORC (2012) code. Standard modifying factors and conversions were applied as described above. No known issues existed at the time which required the levels of confidence of the Ore Reserve to be downgraded; hence all Indicated Mineral Resource within the mining envelope was converted to Probable Ore Reserve. The Mineral Resource does not contain any material classified as Measured.
	Gold (Au) grades are Inferred at Wilga due to a paucity of gold assays in historic drilling. Revenue from gold in the Wilga ore was included in the estimation of the Ore Reserve. The contribution to Revenue of this gold was estimated to be $3.84 per gram of gold *in situ*. This inclusion was not material to the value of the mining envelopes considered and did not warrant downgrading of any portion of the Ore Reserve attributable to Wilga. The tonnage contribution from Wilga represents 13% of the total Ore Reserve tonnage.
	The classification methods used are considered by the Competent Person to be appropriate for the style and nature of the deposit.
Audits or reviews	The Ore Reserve estimate has been subject to internal peer review.
Discussion of relative accuracy/confidence	The Ore Reserve is a global estimate derived from the global Stockman Mineral Resource.
	The Stockman Ore Reserve was classified as Probable only and includes only Mineral Resources classified as Indicated. No downgrading was applied to economic material within the mining envelope. The accuracy of the Ore Reserve is reflected in the classification of the Ore Reserve and the classification of the underlying Mineral Resources upon which it is based.
	Discrepancy exists between the historical tonnes reported as mined at Wilga (circa 956 kt) and those accounted for in current digital wireframe of the workings (circa 802 kt).The reason for the discrepancy remains unclear and reconciliation between the digital model and the actual mined areas is ongoing, subject to further drilling or access to areas of the workings that are currently flooded. This discrepancy represents <2% of the Ore Reserve and was not

Criteria	Commentary
	considered material to the viability of the project.
	Gold (Au) grades are classified as Inferred at Wilga due to a paucity of gold assays in historic drilling. Revenue from gold in the Wilga ore was included in the estimation of the Ore Reserve. The contribution to Revenue of this gold was estimated to be $3.84 per gram of gold *in situ*. This inclusion had no material impact to the value of the mining envelopes considered and did not warrant downgrading of any portion of the Ore Reserve attributable to Wilga. The tonnage contribution from Wilga represents 13% of the total Ore Reserve tonnage.
	Vendor quotation used in the cost estimates were requested on the basis of +/-10% to 15% accuracy.
	Revenue assumptions were based on flat forward pricing. In Australian dollar terms the flat forward copper pricing is between 1% and 11% above the June 2012 forecast copper pricing by Wood Mackenzie. Similarly, the flat forward zinc pricing is between 29% and 59% below the June 2012 forecast zinc pricing by Wood Mackenzie.

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Prior to 2011, RC samples were collected at the rig using a cone splitter that split the 1m cuttings into 87½% & 12½% splits. RC samples were originally composited to 2m by taking scoops from each of the 1m interval 87½% portions, and submitted to Genalysis for sample preparation and analysis. Samples that returned values >0.5g/t Au were submitted as 1m samples to Genalysis (the 12½% splits from the cone splitter). In 2011, RC samples were not composited and 1m interval samples were sent directly to Genalysis. A rig-mounted cone splitter was used to split the samples into 87½% & 12½% splits. NQ2 core was half-core sampled and PQ and PQ3 core was quarter-core sampled using a manual core-cutting diamond saw without water in the oxide zone. The dry cutting was to prevent loss of clays for the metallurgical samples. Sample quality is considered to be good and all RC drilling within the resource area was dry. In 2012, RC samples were collected for 1m intervals using a rig-mounted cone splitter that was not hydraulically adjustable. Samples were meant to be 12½% from each of the two sample chutes and 75% collection of the remainder in plastic bags. A system for measuring weights of bags to prove sample representivity commenced with the program, and showed that the splitter and collection system was not optimal for much of the RC drilling. Issues such as undersize and oversize samples were common, and bias between the paired samples was seen, particularly in the regolith as well as in the fresh rock where the collection system had not been cleaned. These issues are discussed in the section on Drill Sample Recovery. Wet samples were grab sampled and recorded as such in the database, few were within mineralised zones. NQ core was half-core sampled and HQ/HQ3 core was initially quarter-core sampled. Issues with quarter-coring in the regolith with complete disintegration of the sample and loss of material were identified, and reverted to half-core sampling with less water for better sample quality. Standards, blanks and field duplicates were inserted into each batch of samples submitted to the laboratory.
Drilling techniques	In 2009-2010, principally Reverse Circulation (RC) drillholes using face sampling bits (Ranger Drilling Services, Boart Longyear Pty Ltd or Profile Drilling Services) with 3 diamond holes that have RC precollars (precollars drilled by Ranger Drilling Services (70-202m downhole depth) and NQ2 diamond tails drilled by Boart Longyear Pty Ltd) and 2 other diamond holes (PQ3 sized core by Drill West for metallurgical testing purposes). Three core holes (KBD026-028) were oriented using an Ace orientation tool. In 2011, 78 RC drillholes for 14,103m were drilled by Profile Drilling Services using a Schramm RC rig and 11 diamond holes (two with RC precollars, precollars drilled by Profile Drilling Services) drilled by Drill West using a Boart Longyear LF90D skid mounted rig. Core diameter was PQ3 and PQ to provide samples for metallurgical testwork and to also twin RC drillholes. Core was oriented (where possible) using a Reflex ACE orientation instrument. In 2012, 60 RC drillholes for 8409m and RC precollars for 534.8m were drilled by Blue Spec Mining using a KLBS900 Multipurpose rig with 4inch drill rods and face sampling 5inch bits. Two HQ3/NQ diamond holes were drilled by Blue Spec for 305.3m using the Multipurpose rig and 24 HQ/HQ3 diamond holes were drilled by Foraco for 3158.6m using a UDR1000 truck-mounted rig. Core from the Foraco drilling was oriented using an Ezymark orientation tool. Numerous aircore holes have been drilled into the project but these were not used in the resource estimate.
Drill sample recovery	Core recovery was generally good. RC sample recovery prior to 2012 has been logged as good with samples kept dry during drilling. In 2012 RC sample recovery was variable, particularly in the regolith. Sample quality was recorded during logging (wet/dry samples) and qualitative recovery codes (C=contaminated, G=good, M=moderate, O=oversize, P=poor, U=undersize) were assigned to each sample. Sample weights were measured for each component of RC hole cuttings in mineralised zones, with results showing that regolith samples were generally poor quality (both under and over-weight samples) and quality was moderate in the other zones. Quantitative sample recoveries for RC samples can be calculated from the total recovered weights, and will be taken into consideration prior to any future change from an Inferred classification. Core was reassembled for mark-up and was measured, with metre marks and down-hole depths placed on the core. Depths were checked against driller's core blocks and discrepancies corrected after discussion with drillers. Core loss was recorded in the geological log.
	RC sample weights in 2012 drilling were used as a check on blockages and bias in the sample collection system. The rig was regularly stopped and the sample collection system cleaned when blockages occurred and when biased sample weights were noted. Core sampling in 2012 involved an automated core saw, which, in competent rock, should remove sampling bias. The same side of the core was taken during sampling.
	There is no obvious relationship between sample recovery and grade. The poor precision in Bibra assays hinders this analysis to some degree, however the review was completed and no clear relationship observed. Sample bias was observed as measured by the sample weights program. Every attempt was made to minimise this by cleaning the collection system and re-levelling the splitter regularly. There is no evidence for a preferential loss or gain of fine/coarse material in the RC drilling. There may have been loss of clays in regolith core before the cutting procedure was changed to half core only rather than quarter core in the HQ/HQ3 core, and minimal addition of water during sawing.

Criteria	Commentary
Logging	Geological logging of core and RC chips used standard logging digital data entry objects and the IGO coding system. Data on rocktype, deformation, colour, structure, alteration, veining, mineralisation and oxidation state were recorded. RQD, magnetic susceptibility and core recoveries were recorded in spreadsheets. For RC chips sample quality and weights were also recorded, including whether wet or dry. All data were imported to the acQuire database in Perth. Logging is adequate and sufficient detail has been gathered for resource estimation, mining and metallurgical studies.
	Logging is both qualitative and quantitative or semi-quantitative in nature. Core was photographed both dry and wet and copies of the digital images stored on the IGO Perth server.
	Each hole is logged and sampled in full.
Sub-sampling techniques and sample preparation	All core has been cut into half or quarter core for sampling. For early drillholes KBRC005-010, RC composite samples (2m) were submitted to Genalysis where they were sorted, dried and the total sample pulverised in a single stage mix and grind if the sample mass was <3kg. Samples >3kg mass were riffle split using a 50:50 splitter and one half pulverised. Samples were analysed for Au using an aqua regia digestion (AR10/OM) of a 10g pulp sample with ICP-MS determination. Samples that returned values >0.5g/t were submitted to Genalysis as 1m resplit samples and prepared in a similar manner as the composites. For drillholes from KBRC011 onwards (2009-2012), no compositing took place, 1m split RC samples and core samples were submitted to Genalysis for fire assay. Samples were oven dried at 105°C then jaw crushed to -10mm followed by a Boyd crush to a nominal -2mm. Samples were rotary split to 2.5kg (2012 drilling). Samples were then pulverised in LM5 mills to 85% passing 75µm. All the samples were analysed for Au using the FA50/AAS technique which is a 50g lead collection fire assay with analysis by Flame Atomic Absorption Spectrometry. The fire assay method is considered a suitable assaying method for total Au determination. The aqua regia digestion results (used for samples that were <0.5g/t Au) may not allow for total Au determination in the transition and fresh rock zones. These aqua regia samples are only present for 5 holes and therefore represent only a very small percentage of the samples.
	For core and RC samples the sample preparation technique is appropriate and is standard industry practice for a gold deposit.
	Quality control for maximising representivity of samples included sample weights measuring, insertion of field duplicates and laboratory duplicates. IGO has been aware for some time that 50g fire assay is not giving adequate assay repeatability due to the nuggety gold found at Bibra, even though it is a generally low grade deposit. Testwork during 2012 and 2013 involved assessing the cost and effectiveness of using multiple fire assays (up to 4, averaging the results) to simulate a larger sample mass, as well as 1kg LeachWell tests with fire assay of the tail, and screen fire assays. All methods would improve precision but at significant cost. Testwork on grind time to see if finer particles would improve precision showed that any increase in grind time over 5mins resulted in rolling and plating of the gold particles and did not reduce their size, whereas the gangue minerals were substantially reduced in size. The inability to comminute the nuggety gold particles is part of the poor precision problem when using 50g fire assay charges. Field duplicates were inserted but review of results is hampered by the assay repeatability problem when using the 50g fire assay method. Field duplicate and primary sample pairs, whether assayed by screen fire assay or LeachWell assay (with tail assay), and which used much larger sample mass (1kg) for each of those methods, showed much better precision in comparison. Laboratory duplicates (50g fire assay) showed the effects of the nuggety gold at Bibra also, with poor precision seen in paired data plots. Screen fire assay data has shown that the sieved fraction below 75µm shows dramatically improved precision and that the fraction with the +75µm particles is causing the repeatability issue. IGO is investigating cost effective analysis methods using a larger sample size.
Quality of assay data and laboratory tests	The 50g fire assay is a total extraction method and under normal circumstances would be a suitable method. At Bibra, the nuggety gold grains are problematic in that 50g fire assay does not always provide repeatable results, on an individual sample basis. Overall drillhole intercepts, and within block model blocks where numerous samples are used for grade interpolation, the poor assay repeatability becomes much less of an issue. Twin holes from the 2011 drilling showed that over an intercept, the grades and lengths of mineralisation compared well, whereas at the individual assay level the results are highly variable. IGO is investigating cost effective methods to improve repeatability of assays. This is in preparation for more selective mining assessment and for grade control purposes in future, as well as greater confidence in results from review of check assay programs.
	No geophysical or XRF results are used in the resource estimate.
	Quality control procedures included insertion of certified standards (1 in 20), blanks (1 in 50 or two blanks after visible gold) and field duplicates (2 in 100) in batches of samples submitted to the laboratory. In addition, 5% of pulps are submitted back to the primary laboratory (renumbered) as well as to an umpire laboratory for cross-checking. Batches were re-assayed if they failed the accuracy checks or showed consistent bias. Control charts show the accuracy has been reasonable (some low bias to -4%) and contamination minimal. Bias will need to be monitored more closely in future. No significant contamination was noted. Precision is poor as has been described previously.
Verification of sampling and assaying	From 2011, qualitative verification of mineralised zones has been through field panning. Significant intersections are checked by staff to see they meet the known geological and mineralisation models. Significant intersections are also checked by senior company personnel.

Criteria	Commentary
	Analysis of the RC/diamond hole twinning up to the end of 2011, showed that mineralised intervals above a cut-off grade of 0.3g/t Au were similar in length and moderately well correlated in grade. This suggests there has not been any significant downhole smearing in the RC drilling and sampling. It also shows that averaging of numerous assays over an interval gives repeatable results compared with poor repeatability at the individual assay level, as described above. No twin holes were drilled in 2012.
	Primary data are collected in Excel spreadsheets, Field Marshall files or using off-line acQuire data entry objects on electronic Notebooks. Data are imported directly to the database with importers and have built in validation rules. Assay data are imported directly from digital assay files and are merged in the database with sample information. Data are uploaded to a master SQL database stored in Perth, which is backed up daily. Data are reviewed – missing data, incorrect data, as part of the resource estimation process, on an annual basis.
	From time to time assays will be repeated if they fail company QAQC protocols, however no adjustments are made to assay data once accepted into the database.
Location of data points	2009 - 2012 drillhole collar positions were surveyed by licensed surveyors MHR Surveyors of Cottesloe, WA after drilling was completed. The instrument used was a Trimble R8 GNSS RTK GPS (differential) system. Expected relative accuracies from the GPS base station were ±2cm in the horizontal and ±5cm in the vertical direction. Co-ordinates were surveyed in the MGA94 grid system. Downhole surveys in 2009 & 2010 were carried out by the drillers at about 50m intervals using a Reflex EZ shot digital downhole camera. Readings were taken in a non-magnetic stainless steel rod near the bottom of the drill string. The depth, dip, azimuth and magnetic field were recorded at each survey point. In 2009 gyro surveys were attempted however most holes had collapsed and the gyro survey was successful to end of hole in only one drillhole. The top parts of other holes were surveyed using the gyro instrument (Downhole Surveys Australia, readings at 5m intervals) and given priority over Reflex surveys in the database. The gyro survey was not continued in 2010 due to the limited success of the 2009 program. Downhole survey readings have been checked by extracting the drillholes and displaying them in graphics in the Surpac software program, with spurious readings removed by assigning them a lesser priority in the database. The lesser priority surveys were not used during the resource estimation. Drillholes KBRC101-105;107-123;125-129;131-134 had only one survey downhole (near the bottom of the hole) due to their short lengths (<112m long). In 2011 the frequency of downhole surveys in new drillholes was increased to about 30m intervals. Surveys were carried out by the RC drillers using a Camteq Proshot electronic camera and by diamond drillers using a Pathfinder Electronic Single Shot camera. In 2012, both RC and diamond drilling used a Reflex EZ-Trac tool. Surveys were carried out every 30m downhole. Camera calibration certificates prior to the commencement of drilling have been collected since 2011 as a check on camera accuracy. The downhole surveys are considered to be of adequate quality for resource estimation work.
	Drillhole location data were initially captured in the MGA94 grid system and have been converted to a local grid for resource estimation work. The MGA94 ties to local grid were surveyed by independent surveyors MHR Surveyors. An elevation adjustment of +2000m was also conducted on the local grid co-ordinates.
	The natural surface topography was modelled using a DTM generated from the 2012 airborne LiDAR survey conducted in November 2012 by AAM Pty Limited. The DTM was rotated in-house to the local grid co-ordinate system. Horizontal point accuracy is expected to be <0.33m and vertical accuracy to 0.15m. Ground control was established using RTK GPS and ALTM3100 Static GPS. The reference datum was GDA94 and the projection was MGA Zone 50, with the data supplied as 50cm and 1m contours in MGA Zone 51. Topographic control is of good quality and is considered adequate for resource estimation.
Data spacing and distribution	No exploration results have been reported.
	Data spacing and distribution has been taken into account in the classification of Inferred Mineral Resource.
	Samples were composited to 1m lengths.
Orientation of data in relation to geological structure	Drilling is mostly oriented local grid east at an average dip of 60°. A number of holes were drilled vertical in 2011 when delineating higher grade shoots. These holes were drilled perpendicular to the continuity direction. The orientation of the drilling is suitable for the mineralisation style and orientation encountered to date.
	No sampling bias has occurred due to orientation of the drillholes.
Sample security	Samples are sealed in calico bags, which are in turn placed in large poly-weave bags and cable-tied. A certain number of filled poly-weave bags are stacked in a cage secured on a wooden crate and transported directly via road freight to the laboratory with a corresponding submission form and consignment note. Genalysis checks the samples received against the submission form and notifies IGO of any missing or additional samples. Once Genalysis has completed the assaying, the pulp packets, pulp residues and coarse rejects are held in their secure warehouse. On request, these are returned to the IGO warehouse on secure pallets where they are documented for long term storage and retrieval. In addition, a sample tracking register is kept where samples dispatched to the laboratory are tracked until return of the assays to IGO.
Audits or reviews	A review of practices documented in the IGO technical report supplied to Optiro Pty Ltd in 2012 as part of the resource estimate review did not highlight any significant issues. The reviews by the Competent Person from site visits highlighted some issues which were addressed.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The Bibra mineralisation is within the granted E52/1711 exploration tenement in the Pilbara region of Western Australia. E52/1711 was acquired from BHP_B in 2008. BHP_B retain a 2% NSR and a claw-back provision whereby BHP_B can elect to acquire a 70% equity in the project only if JORC compliant reported resources of 5,000,000 ounces of gold and/or 120,000 tonnes of contained nickel have been delineated. The Nyiyaparli group are Native Title claimants covering an area including E52/1711. There are no known heritage or environmental impediments over the lease. A mining lease sufficient in size to cover the Bibra resource area and potential associated infrastructure for a future mining operation has been applied for, and IGO is currently in negotiation with the Nyiyaparli group over this application.
	The tenure was secure at the time of resource estimation and reporting. No known impediments exist to operate in the area.
Exploration done by other parties	The Bibra mineralisation was discovered by IGO after a geochemical anomaly was defined along strike and underneath surface cover with broad spaced aircore drilling in 2009. The area surrounding the Bibra deposit had previously been unexplored until WMC discovered gold mineralisation at the Francopan Prospect 5km south-east of the Bibra deposit in 2004 and IGO acquired the project in 2008.
Geology	The Bibra deposit is hosted in an Archaean greenstone belt in the Pilbara region of Western Australia. The host rocks are an amphibolite hangingwall and chlorite-biotite-garnet-feldspar schist footwall. Gold mineralisation has been intersected over a wide area at Bibra with at least 4 sub-parallel lodes identified. The lodes strike NE-SW (MGA94) and plunge shallowly to the NW in typically wide, low grade zones. A series of shallowly NW plunging rod-like higher grade shoots have been identified within the more continuous lower grade halo. Primary gold mineralisation in fresh rock is marked by 3-10% sulphides, subhedral magnetite grains, quartz vein/veinlets and fine grained gold. Mineralisation in fresh rock continues to near surface in the oxide zone and includes a laterally extensive supergene horizon that is hosted within a laterite.
Drill hole Information	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Data aggregation methods	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Relationship between mineralisation widths and intercept lengths	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Diagrams	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Balanced reporting	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Other substantive exploration data	There are no exploration results reported for the immediate Bibra area that have not been reported previously.
Further work	Further work involves large scale (regional) step-out drilling for determination of additional mineralisation.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Data are collected by the geologists and field staff in either Excel spreadsheets or acQuire data entry objects on laptops for RC and diamond drilling. Previously, Field Marshall has been used to capture data from drilling programs. Once the geologists are confident that the data are correct and complete, the files are emailed to the Database Administrator in Perth or copied to a designated data folder on the server. These data files are then loaded into a Master Data Management (MDM) SQL database using acQuire software as the front end. If any errors occur during the loading and validation of the data, a note is made and emailed to the relevant geologist to advise and correct the data. Assays are received from the various independent laboratories in electronic ASCII files of varying format, and are merged with sampling data already present in the database. Assays received from laboratories were imported by the Database Administrator using customised acQuire importers thus alleviating any data entry mistakes. The acQuire database for the Bibra Prospect is exported to an Access database and reviewed for errors prior to resource estimation work.
	The acQuire drilling database had been compiled and validated by IGO on an ongoing basis through to the Access database dump in June 2013 for resource work.
Site visits	Site visits by the Competent Person, Michelle Wild, were conducted on 7 November and 20 November 2012. Recommendations were made regarding RC sampling, core sampling in the regolith zones, core density measurements, general QAQC, and equipment and cleanliness.
Geological interpretation	Confidence in the geological interpretation is moderate, given the wide-spaced drilling. Stratigraphy seems consistent in that it can be correlated between holes and along strike. It is expected that refinements to the geological model will be made with increased density of drilling.
	Drillholes are wide-spaced and as such the interpretation has been kept simple. Geological logging and structural measurements from drillholes has been used to construct the geological model and northern fault.

Criteria	Commentary
	Sections were interpreted, digitised and a 3D wireframe model constructed. Geological continuity has been assumed along strike and down-dip.
	The interpretation will evolve as drilling spacing decreases and more information becomes available for modelling, however the overall impact on Mineral Resources is expected to be low. It is unlikely that an alternative interpretation will develop. There is currently sufficient drilling to broadly map the stratigraphic units and the supergene zone.
	The geological model has been used to guide mineralisation envelopes and subsequent mineralisation wireframe modelling. The interpreted fault zone in the north end has disrupted the stratigraphy and the mineralisation model was built to conform with the geological model. Changes in this area of the interpretation are expected when additional drilling is completed.
	Geological continuity has been assumed along strike and down-dip based on reasonably wide-spaced drilling data. Factors that might affect continuity are that with closer-spaced drilling the geological model could become more complex if new faults are discovered that are currently undetectable. In general, continuity both geologically and grade-wise is good. Grades and thickness are more consistent down-dip than along strike.
Dimensions	The Bibra mineralisation wireframes have been extended large distances down-dip based on very wide drilling intercepts, however this extrapolation has been removed from the resource estimate by limiting the reported tonnes and grade to within a conceptual optimal pit shell ($1600/oz Au). The supergene zone modelled was 900m along strike and 230m wide in the NE widening to 560m in the southern half. It ranges from 1.7m to 14m in vertical thickness. The primary mineralisation extends below the supergene zone for a further vertical depth of 270m. The transition/fresh rock boundary is about 60m below surface. The primary mineralisation has 4 main sub-parallel zones and several smaller zones. The main zone is 740m long (N-S) and 970m wide (horizontal width) at its widest part in the north, tapering to 300m wide (horizontal width) at the southern end. Note that only a portion of this mineralisation has been classified as resource (i.e. that portion within the region defined by the 100m x 50m spaced drilling or closer, and within the conceptual optimal pit shell). The thickness of the main primary mineralisation zone ranges from 1.7m vertical thickness to 30m in the thickest part.
Estimation and modelling techniques	Higher grade wireframe domains were built for mineralisation above 1.0g/t Au in the supergene zone and 1.5g/t Au in the main zones in order to constrain the higher grade portions of the mineralisation. Ordinary Kriging was used for grade estimation utilising Surpac software v6.4.1. Search and kriging parameters were derived from variogram models for Au. The block dimensions were 20mY, 10mX and 10mZ for parent cells, sub-blocked to 10mY, 5mX and 1.25mZ. Grade estimation was constrained to blocks within each of the mineralisation wireframes. The major direction search distance in the supergene mineralisation was 150m. In the primary mineralisation the major search distance was 80m for pass 1 and 160m for pass 2. The search bearing for the main zone was 180° with plunge of -8° and dip of -15°. Search ellipse alterations were made for changes in wireframe geometry and in the lesser mineralised zones. The maximum number of samples used for grade interpolation was 50 (min 25, reducing to 6 in smaller zones) and 5 maximum per drillhole. This estimation technique is suitable for a global model. A local uniform conditioned model was also generated to test the effect of modelling at the selective mining unit (SMU) scale. Results are preliminary and require closer spaced drilling for better modelling, however they suggest that the effect of mining SMU sized blocks could be lower tonnes at a slightly higher grade, for cut-off grades up to 1.0g/t Au on the grade tonnage curve.
	This estimate is similar to that of 2012. Grades are similar and tonnes have reduced slightly due to refinements in the wireframes based on infill drilling. No mining has occurred at Bibra.
	No assumptions have been made regarding by-products.
	No deleterious elements are known or expected.
	Block size was based on Kriging Neighbourhood Analysis. The block size is reasonable for drilling at a 50x50m pattern. Anisotropic searches were employed and were based on variography. The block size is reasonable for the search distance used for Pass 1.
	Modelling of selective mining units has taken place in preliminary studies but is not part of this resource estimate. Closer spaced drilling is required for better modelling at SMU scale.
	Only Au has been modelled.
	The geological interpretation was used to control mineralisation modelling and to assign densities to rock-types.
	Top-cuts were established after a study of statistics, histograms and log-probability plots for the main domains. A top-cut of 10g/t Au was determined for the supergene zone (Co-efficient of Variation (CV) = 1.35, 4 samples top-cut) and 16g/t Au for the main mineralisation (CV = 1.76, 21 samples top-cut). The percentiles for cutting are less than 0.5% of the samples. There is only a small number of high grade samples at Bibra.
	The block model is checked visually in Surpac graphics by comparing drillhole assays with block grades. Blocks with no interpolated grades are checked and corrections made to the model. Swath plots are generated to compare block grades with sample composite grades on a sectional and plan slice basis.
Moisture	Tonnages have been estimated on a dry basis. Core samples in the oxide zone have been measured for density after drying and coating at an independent laboratory. Transition and fresh rock samples have been tested uncoated on site after sun-drying, and added to the database of samples tested by the independent laboratory. New measurements in 2012 confirmed earlier density measurements for rocktype and oxidation state.

Criteria	Commentary
Cut-off parameters	The mineralisation has been wireframe modelled using a 0.3g/t Au assay cut-off grade. The resource estimate has been reported above a block grade of 0.5g/t Au. Cut-off grades will be refined as the mining and metallurgical processes become better defined.
Mining factors or assumptions	Currently a large open-pit mining option is the basis for the cut-off grade. The shallow dip precludes using large bench heights without incurring significant dilution. Ore and waste would be paddock blast on 5m benches and subsequently excavated as 2.5m flitches utilising a conventional excavator and truck mining fleet to facilitate moderate ore excavation selectivity. Internal dilution to 2m has been included but no external dilution has been applied to the estimate.
Metallurgical factors or assumptions	Systematic metallurgical testwork programs over 2012/13 on master and variability composites from diamond core identifies mineralisation as free milling and amenable to cyanidation. Adoption of a conventional gravity and carbon-in-leach process circuit design is likely to yield gold recoveries in the low to mid 90's (%) for fresh and oxide material respectively. Physical testwork indicates bond work indices of 10kWhr/t to 18KWhr/t and moderate to high abrasion indices.
Environmental factors or assumptions	Waste rock from open pit operations would be placed in a waste rock landform adjacent to open pit operations, progressively contoured and revegetated throughout mine life. Process plant residue would be disposed of in a surface tailings storage facility to the immediate south. Adoption of an upstream, central decant design would utilise mine waste material for dam wall construction and facilitate water recovery to supplement process water requirements. Geochemical testwork on mineralised and non-mineralised waste regolith and bedrock samples indicates the material to be non-acid forming.
Bulk density	Densities were based on measured densities sorted by rock type and oxidation state. Outliers were removed and remaining measurements were averaged for each rock type and oxidation state domain. In the 2012 core drilling program, all samples sent for analysis from the transition or fresh rock zones were density measured. Initially, 10cm pieces of core were selected from each sample and density measured, but this was changed to entire samples after the site visit by the Competent Person in November 2012. Density determination by IGO was by the water immersion method. The independent laboratory used two methods – both involved oven drying and wax coating the samples and water immersion. The density database has a total of 1585 measurements for Bibra.
	Densities measured at the independent laboratory accounted for void spaces and moisture. Densities measured by IGO were in competent core that was sun-dried but uncoated. Natural moisture in the competent core is expected to be low. On-site testing in future will use improved methods and equipment. As noted above, rock type and oxidation state were the main divisors for density measurements and application to the block model.
	No assumptions have been made for bulk density estimates. Bulk densities assigned to the block model are based on measured data.
Classification	A classification wireframe was built for an area defining the 100 x 50m drill spacing. The wireframe was applied as a cookie-cut to the block model to code all blocks within the wireframe and the oxide, transition and fresh zones as Inferred. All other mineralisation has been left unclassified due to the wide-spaced drilling pattern. Potential exists to upgrade the classification with infill drilling. The inferred mineralisation was further constrained to a $1600/oz AUD conceptual optimal pit shell. The remainder of the modelled mineralisation does not form part of the current resource estimate. The conceptual optimal pit shell has a pit base at 230m below surface.
	The Inferred classification reflects the relative confidence in the estimate, the wide-spaced drilling input data, the assay repeatability and the assumed continuity of the mineralisation.
	The classification as Inferred reflects the Competent Person's view of the deposit.
Audits or reviews	The resource estimate and technical documentation for 2012 was reviewed by Optiro Pty Ltd in October 2012 and recommendations from that review have been included in the resource estimate for 2013.
Discussion of relative accuracy/confidence	The confidence level is reflected in the Inferred classification of the estimate. Mineralisation modelled but outside the criteria used for classification as Inferred has been excluded from the estimate. Potential for upgrading the classification exists if closer spaced holes are drilled, continuity is proven, and RC sampling issues and assay repeatability are addressed.
	The Mineral Resource estimate is an undiluted global estimate.
	There is no production data to compare the resource estimate with, as Bibra has not been mined.
Resource Model Number	BI_RSC_2013_06