ASX RELEASE

14 October 2016

ANNUAL MINERAL RESOURCE AND ORE RESERVE STATEMENT

Independence Group NL (IGO or the Company) (ASX: IGO) is pleased to provide its annual Mineral Resource and Ore Reserves estimates, as at 30 June 2016.

Highlights

- Total Mineral Resource for the Tropicana Gold Mine (100% basis) is 7.5Moz and Ore Reserve is 2.4Moz, under the existing mining costs structure. An updated Mineral Resource and Ore Reserve is expected to be released in the December 2016 quarter as part of the Long Island study.
- Nova Project Mineral Resource and Ore Reserves remain unchanged although the resource is currently being de-risked with the current grade control program. Results to-date are encouraging.
- Successful completion of the Flying Spur and Arnage resource definition drilling at Bentley has resulted in a significant Ore Reserve increase with extension of the life of mine to greater than 3 years.
- IGO actively drilling both highly prospective brownfields and greenfields projects with the aim of delivering further resource and reserve growth in the future.

Commenting on the update IGO's Managing Director and CEO, Peter Bradford, said:

> *"We have significantly strengthened our growth platform during FY16 as we transform the business in FY17.*
>
> *A key element of the Tropicana Long Island study has been completed with development of the mineral resource model, which captures the resource drilling completed as part of this study. The model will form the framework for the completion of the prefeasibility study. The current Mineral Resource stands at 7.5Moz of gold using the current cost structure to constrain the resource, with the Ore Reserve at 2.4Moz of gold. A change in the mining costs delivered by the strip mining approach and optimisation of waste haulage being investigated as part of our Long Island Study will unlock the resource to reserve conversion. An updated Mineral Resource and Ore Reserve will be issued as part of the study expected in the December quarter"*
>
> *At Nova there has been no change to our Mineral Resource or Ore Reserve statement, however the resource continues to be de-risked with the current grade control drilling program. At present, we have three underground diamond drill rigs executing this program. Results to date, although only representative of a small proportion of the total resource have been encouraging with positive nickel equivalent metal reconciliations when compared to the resource model.*
>
> *IGO has consistently extended the Jaguar and Long Operations life of mines through brownfields exploration. We have again demonstrated this during FY16 with Ore Reserve growth at Jaguar from the drilling program targeting the Flying Spur and Arnage lenses at Bentley underground with zinc contained metal, net of depletion increasing approximately 56%. Exploration has re-commenced at Long with the aim of again extending the life of mine for FY17, as has been the historic trend of this operation.*

There are numerous exciting exploration programs planned for FY17 across our extensive and highly prospective brownfields and greenfields portfolio. It is these programs which will continue to deliver resource and reserve growth for IGO in the future."

An overview of each of the assets in the Company's portfolio is provided within this release.

Table 1: IGO Group – 100% basis - 30 June 2016 Total Mineral Resources

Project	Tonnes	Grade					Contained Metal				
	Ore (Mt)	Ni (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Ni (kt)	Cu (kt)	Zn (kt)	Ag (Moz)	Au (Moz)
Nova	14.3	2.3	0.9	-	-	-	325	134	-	-	-
Long	1.3	4.7	-	-	-	-	60	-	-	-	-
Tropicana	37.4	-	-	-	-	1.9	-	-	-	-	2.2
Jaguar	3.7	-	1.4	7.0	111	0.6	-	51	256	13.1	0.1
Stockman	14.0	-	2.1	4.3	38	1.0	-	294	598	17.1	0.4
Total	**70.6**										

Notes:

1. Detailed tables setting out each of the Measured, Indicated and Inferred Mineral Resource categories are set out on tables 3 to 12 on each of the relevant sections.
2. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
3. Tropicana Gold Mine - IGO share (30%) shown.
4. Long Operation, Jaguar Operation, Nova Project and Stockman Project are 100% IGO owned.
5. Metal quantities are contained metal.
6. Mineral Resources are inclusive of Ore Reserves.

Table 2: IGO Group – 100% basis - 30 June 2016 Total Ore Reserves

Project	Tonnes	Grade					Contained Metal				
	Mt	Ni (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	Ni (kt)	Cu (kt)	Zn (kt)	Ag (Moz)	Au (Moz)
Nova	13.6	2.0	0.8	-	-	-	275	112	-	-	-
Long	0.4	3.9	-	-	-	-	14	-	-	-	-
Tropicana	12.3	-	-	-	-	1.8	-	-	-	-	0.7
Jaguar	1.4	-	1.1	9.5	145	0.8	-	16	137	6.7	0.0
Stockman	9.0	-	2.1	4.5	39	1.1	-	189	408	11.3	0.3
Total	**36.7**										

Notes:

1. Detailed tables setting out each of the Proven and Probable Ore Reserves are set out on tables 3 to 12 on each of the relevant sections.
2. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
3. Metal quantities are contained metal
4. Tropicana Gold Mine - IGO (30%) share shown.
5. Long Operation, Jaguar Operation, Nova Project and Stockman Project are 100% IGO owned.

Nova Project

The world-class Nova Project is located 160km by road, east of Norseman. The Nova discovery hole was drilled in July 2012 and a maiden resource was released twelve months later, in May 2013. The Definitive Feasibility Study was completed in July 2014. As at 30 June 2016, the overall Project construction and development was 93% complete. The Project is on schedule and on budget to produce first nickel and copper concentrates by December 2016.

Geology

The Nova–Bollinger magmatic nickel-copper-sulphide deposits are located in the Fraser Zone of the ~1.3 billion year (Ga) old Albany–Fraser Orogenic belt, on the south-eastern margin of the Archaean aged Yilgarn Craton, in Western Australia. The mineralisation is hosted by meta-gabbro to meta-picrite cumulates that were originally emplaced as a series of sills in an extensional sedimentary basin during the late stages of continental breakup.

The deposit is located on the north-western side of an ovoid structural feature measuring approximately 3km by 1.5km, which is best seen in regional and ground magnetics. The deposits are analogous to many mafic-hosted nickel-copper deposits worldwide.

The Nova and Bollinger deposits are generally tabular in geometry, with clear boundaries that define the various mineralised domains. The Nova and Bollinger deposits are joined by a feeder zone.

Mineralisation at Nova and Bollinger occurs as massive, breccia, matrix or net-textured and disseminated sulphides. Massive sulphides within Nova typically grade 6.0 to 7.0% nickel, and disseminated sulphide mineralisation contains in the order of 0.5 to 1.5% nickel. The massive sulphide at Bollinger is lower grade, typically containing 4.5 to 5.5% nickel. The primary ore mineralogy is pyrrhotite (~80-85%), pentlandite (~15-10%) and chalcopyrite (5-10%), irrespective of the ore type.

Mineral Resource

The Mineral Resource estimate for the Nova Project as at 30 June 2016, which covers both the Nova and Bollinger deposits is estimated at **14.3Mt at 2.3% Ni, 0.9% Cu and 0.08% Co** with no change to the 30 June 2015 Mineral Resource.

An extensive grade control diamond drilling program commenced in May 2016. As of the end of September 2016 there were three underground diamond drill rigs executing the work program which is expected to take 14 months, to complete an estimate of 139km of drilling. The nominal drill spacing is 12.5m by 12.5m.

Grade control drilling and interpretation has been completed between the 2080mRL level and the 2030mRL level and updated into the Nova grade control model. The grade control model provides a more robust local estimate given the improved geological understanding and higher resolution of data. Reconciliation between the grade control model and resource model has been encouraging. To-date, although over a small volume compared to the total resource, the reconciliation is showing a slight decrease in tonnes and increase in nickel equivalent grades for an approximate 5% increase in contained nickel equivalent metal.

Ore Reserve

The Ore Reserve for the Nova Project was updated as part of the Nova Optimisation Study (ASX 14 December 2015 – Optimisation Study Significantly Enhances Nova Project Value). The Probable Ore Reserve estimate as at 30 June 2016 is **13.6Mt @ 2.0% Ni, 0.8% Cu and 0.07% Co** for contained metal of 275kt of nickel, 112kt of copper and 9kt of cobalt metal. This underpins an initial mine life of 10 years.

Exploration

IGO has approximately 3,200km^2 of tenements in the Fraser Range region with an additional 450km^2 of tenements added to the portfolio with the joint venture entered into with Buxton Resources. IGO's strategy is to pursue further consolidation to strengthen its brownfields exploration portfolio proximal to the Nova Project.

Some key exploration activities planned for FY17 include:

- Resource extension at Nova including the drill testing of the Conductor 5 from the Bollinger Decline;
- Establishing an underground Electromagnetic Loop (EM) with deeper framework drilling to be completed beneath the Nova and Bollinger orebodies as a geophysical platform;
- 3D seismic to provide the geological and structural architecture for the Nova-Bollinger deposits;

- Commencement of an embedded postdoctoral research program with the Commonwealth Scientific and Industrial Research Organisation (CSIRO) and University of Western Australian (UWA) on the lithostratigraphy, magmatic evolution and control on emplacement and mineralisation associated with the Nova-Bollinger orebodies;
- Systematic ground EM over key prospects;
- Extensive aircore, reverse-circulation (RC) and diamond drill testing of the highly ranked prospects.

Table 3: Nova Project – 30 June 2016 Mineral Resources (and 2015 comparison)

		Mineral Resources - June 2015							Mineral Resources - 30 June 2016						
		Tonnes	Grade			Contained Metal			Tonnes	Grade			Contained Metal		
Deposit	Classification	(Mt)	Ni (%)	Cu (%)	Co (%)	Ni (kt)	Cu (kt)	Co (kt)	(Mt)	Ni (%)	Cu (%)	Co (%)	Ni (kt)	Cu (kt)	Co (kt)
Nova	Measured	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Indicated	9.1	2.5	1.0	0.08	230	94	7.3	9.1	2.5	1.0	0.08	230	94	7.3
	Inferred	1.0	1.4	0.6	0.05	14	6	0.5	1.0	1.4	0.6	0.05	14	6	0.5
	Sub-total	10.1	2.4	1.0	0.08	244	100	7.7	10.1	2.4	1.0	0.08	244	100	7.7
Bollinger	Measured	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Indicated	2.4	2.7	1.1	0.11	64	26	2.6	2.4	2.7	1.1	0.11	64	26	2.6
	Inferred	1.8	1.0	0.4	0.04	17	8	0.7	1.8	1.0	0.4	0.04	17	8	0.7
	Sub-total	4.2	2.0	0.8	0.08	82	34	3.3	4.2	2.0	0.8	0.08	82	34	3.3
	Stockpile	-	-	-	-	-	-	-	-	-	-	-	-	-	-
GRAND TOTAL		14.3	2.3	0.9	0.08	325	134	11.0	14.3	2.3	0.9	0.08	325	134	11.0

Notes:

1. Mineral Resources are reported above a 0.6% nickel equivalent cut-off grade which is calculated as NiEq% = ((Cu % x 0.95) x ($7,655/$16,408)) + (Ni % x 0.89).
2. As at 30 June 2016 the resource broken stocks was not material to the Mineral Resource with an estimated 11.8kt at 0.88% Ni, 0.55% Cu and 0.03% Co stockpile.
3. There is no change to the Mineral Resources from June 2015 to June 2016, with no drilling completed nor changes to the understanding of the geological controls.
4. Mineral Resources are inclusive of Ore Reserves.
5. No depletion has occurred during the period.
6. Ore tonnes have been rounded to the nearest hundred thousand tonnes.
7. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
8. JORC Code (2012) Table 1 Parameters are contained within Appendix A of this release

Table 4: Nova Project – 30 June 2016 Ore Reserves (and 2015 comparison)

		Ore Reserves - December 2015							Ore Reserves - 30 June 2016						
		Tonnes	Grade			Contained Metal			Tonnes	Grade			Contained Metal		
Deposit	Classification	(Mt)	Ni (%)	Cu (%)	Co (%)	Ni (kt)	Cu (kt)	Co (kt)	(Mt)	Ni (%)	Cu (%)	Co (%)	Ni (kt)	Cu (kt)	Co (kt)
Bollinger	Proven														
	Probable	2.7	2.2	0.9	0.09	59	24	2	2.7	2.2	0.9	0.09	59	24	2
	Sub-Total	2.7	2.2	0.9	0.09	59	24	2	2.7	2.2	0.9	0.09	59	24	2
Nova	Proven														
	Probable	10.9	2.0	0.8	0.06	216	89	7	10.9	2.0	0.8	0.06	216	89	7
	Sub-Total	10.9	2.0	0.8	0.06	216	89	7	10.9	2.0	0.8	0.06	216	89	7
	Stockpile	-	-	-	-	-	-	-	-	-	-	-	-	-	-
GRAND TOTAL		13.6	2.0	0.8	0.07	275	112	9	13.6	2.0	0.8	0.07	275	112	9

Notes

1. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.

2. As at 30 June 2016 the Ore Reserves broken stocks was not material to the Ore Reserve with an estimated 9.3kt at 0.99% Ni, 0.62% Cu and 0.03% Co stockpile.
3. A Net Smelter Return (NSR) cut-off value of $64/t of stope ore has been used in the evaluation of the Ore Reserve, which includes mining and G&A operating costs. Processing costs are captured as a variable to the NSR block value.
4. There is no change to the December 2015 Ore Reserve as the project is still under construction and no new significant information is available as of 30 June 2016.
5. Minor Ore Reserves are now broken stocks on the ROM pad but as yet have not been reconciled through processing and sampling.
6. Sub-level open-stoping with paste backfill is the primary method of mining to be used at Nova.
7. The Ore Reserve has been estimated as part of the Optimisation Study completed by IGO in December 2015.
8. JORC Code (2012) Table 1 Parameters are contained within Appendix A of this release

Tropicana Gold Mine (IGO 30%, AngloGold Ashanti 70%)

The Tropicana Gold Mine, a joint venture between IGO and AngloGold Ashanti Australia (AGAA) is located 330km east-northeast of Kalgoorlie. Tropicana is a greenfields exploration discovery, with open pit mining commencing in 2012 and first gold production occurring during September 2013. The Tropicana Joint Venture partners hold a dominant land position covering the geological host sequence to the exploited mineralisation and consequently the Joint Venture partners have a competitive advantage in understanding its mineralised system. The land position covers approximately 2,900km^2 of tenements along a strike length of approximately 200km.

A significant framework drilling program was executed during FY16 as part of the Long Island study. The Long Island study is an internal name given to a work program designed to unlock the future potential of the Tropicana Gold Mine. The goals of the Long Island study are to provide an economically robust method of accessing the substantial mineralisation identified below the current Ore Reserve pit design via open pit mining methods. Current studies are quantifying alternative lower cost mining methods to enable mining of ore below the current planned pits, which would extend the life of mine and improve the mining cost structure.

Due to the unique geometry and continuity of mineralisation at Tropicana, the JV partners are considering a strip mining approach, which has the effect delivering a step change reduction in mining costs as a result of reducing haulage of waste which is predominantly hauled and dumped in pit. The current concept involves using the Tropicana pit as a void for waste dumping once it has been mined to the end of its current Ore Reserve limit. Once the Ore Reserves are depleted at the Tropicana pit, it will be backfilled with waste from the strip mining of the Havana, Havana South, and Boston Shaker ore zones. The approach reduces waste removal mining costs as a result of shorter haul distances which would, in turn, facilitate economically viable extensions to the life of mine.

The resource model which forms part of the Mineral Resource statement captures the framework drilling completed as part of the Long Island study. However, no changes to mining cost assumptions from the Long Island study have been captured to constrain the 2016 Mineral Resource. Completion of the Long Island study to pre-feasibility level is expected to be completed during the December 2016 quarter, which will also include an update to the Mineral Resource and Ore Reserve statements.

Geology

The Tropicana Gold Mine lies to the west of a major tectonic suture between the Yilgarn Craton and the Proterzoic Albany-Fraser Belt that stretches over 550km northeast-southwest. The regional geology is dominated by granitoid rocks, felsic to mafic paragneiss and orthogneiss, and felsic to ultramafic intrusive and volcano-sedimentary rocks.

The Tropicana, Havana, Havana South and Boston Shaker deposits define a north-east trending mineralised corridor approximately 1.2km wide and 5km long, that has been tested to a vertical depth of more than 1,200m below the Havana pit. Mineralisation is up to 50m thick, which is hosted predominantly in quartzo-feldspathic gneiss, bounded by a garnet-gneiss hangingwall sequence. The mineralisation is accompanied by pyrite (2% to 8%) with accessory pyrrhotite, chalcopyrite and other minor sulphides and tellurides.

Mineralisation remains open down-dip from the Havana, Havana South and Boston Shaker deposits.

Mineral Resource

The total Tropicana Mineral Resource (100% basis) for as at 30 June 2016 is estimated at **124.8Mt @ 1.86g/t Au for 7.48Moz** of contained gold compared to 115.7Mt at 1.89g/t Au for 7.04Moz.

The Mineral Resource as reported captures the Long Island exploration drilling through to the end of June 2016. The Mineral Resource has been reported internal to an optimised shell based on an assumed gold price of US$1,400/oz (A$1,817/oz), depleted through to 30 June 2016. The optimised pit shell is based on current mining costs and does not capture expected mining cost reductions as part of the Long Island study. In comparison to the Mineral Resource as at 30 June 2015, the break-down of the year-on-year changes to the Mineral Resource include:

- Depletion of ~-480koz of gold
- Stockpile adjustments of ~-10koz of gold
- Long Island exploration success internal to the open pit resource shell of ~300koz of gold
- Changes to the underground mining shapes of ~110koz of gold
- Long Island exploration success external to the open pit resource shell captured as an underground increase of 840koz of gold
- Changes to the gold price and cost modifying parameters resulting in a decrease of ~-320koz of gold

An update on the Mineral Resource and Long Island study based on expected changes to the mining cost structure is expected to be completed in the December 2016 quarter.

Ore Reserve

During the FY16, 7.3Mt of ore at 2.13g/t Au were mined from the Havana, Tropicana and Boston Shaker pits (>0.6g/t Au) on a 100% basis. IGO attributable gold production was 134koz of gold produced for this period.

The Total Tropicana Ore Reserves (100% basis) as at 30 June 2016 was **41Mt at 1.83g/t Au for 2.41Moz** of contained gold. The Ore Reserve has been restated based on mining depletion as at 30 June 2016 using the 30 June 2015 Mineral Resource model.

It is important to note that neither the exploration drilling nor the expected changes to the mining costs as part of the Long Island study have been captured as part of this Ore Reserves Statement. Release of an updated Ore Reserve for Tropicana is expected to be provided in the December quarter 2016. A change in the mining cost structure will likely deliver a favourable resource to reserve conversion.

Exploration

The framework drilling associated with the Long Island study was substantially completed in FY16. The drill hole spacing ranges from 50m by 50m down plunge of known higher-grade ore shoots to 100m by 100m or greater, elsewhere along strike. Current resource extensional exploration drilling programs will focus on improved definition of high-grade ore shoots on a 50m by 50m spacing at Havana South, Havana and Boston Shaker. Results to date have been encouraging, extending the high-grade ore shoots.

Regional exploration on the extensive ground position has been the priority at the completion of the current stage of the Long Island study. The joint venture has developed an improved understanding on the geological host sequence and structural controls which has resulted in enhanced targeting and prioritisation of prospects. The tenements host a portfolio of numerous exploration prospects including, Tumbleweed, Voodoo Child, Zebra, Beetlejuice, Sanpan, Monsoon, Panama and Angel Eyes. These prospects are planned to be systematically tested during FY17.

Table 5: Tropicana Gold Mine – 100% basis (IGO 30%) - 30 June 2016 Mineral Resources (and 2015 comparison)

		Mineral Resources - 30 June 2015			Mineral Resources - 30 June 2016		
		Tonnes	Grade	Contained Metal	Tonnes	Grade	Contained Metal
	Classification	(Mt)	Au (g/t)	Au (Moz)	(Mt)	Au (g/t)	Au (Moz)
Open Pit	Measured	12.8	2.09	0.86	10.9	1.91	0.67
	Indicated	75.3	1.85	4.47	78.3	1.71	4.32
	Inferred	5.8	2.54	0.48	4.4	2.23	0.32
	Sub-Total	93.9	1.92	5.80	93.7	1.76	5.30
Underground	Measured	-	-	-	-	-	-
	Indicated	2.4	3.58	0.27	5.4	3.36	0.59
	Inferred	5.8	3.14	0.59	12.1	3.13	1.22
	Sub-Total	8.2	3.26	0.86	17.6	3.20	1.81
Stockpiles	Measured	13.6	0.87	0.38	13.6	0.85	0.37
Total Tropicana	Measured	26.4	1.46	1.24	24.5	1.32	1.04
	Indicated	77.7	1.90	4.74	83.8	1.82	4.90
	Inferred	11.7	2.84	1.06	16.6	2.89	1.54
GRAND TOTAL		115.7	1.89	7.04	124.8	1.86	7.48

Notes:
1. The open pit Mineral Resource is reported at a 0.3g/t Au cut-off for oxide material and a 0.4g/t Au cut-off for transitional and fresh material, constrained within an a US$1,400/oz Au (A$1,817/oz Au) optimised pit shell based on actual mining and processing costs.
2. The underground Mineral Resource is reported outside the US$1,400/oz Au pit optimisation based on underground mineable shapes at a cut-off grade of 2.0g/t Au.
3. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
4. Mineral Resources are inclusive of Ore Reserves.
5. All Mineral Resources are completed in accordance with the 2012 JORC Code.
6. JORC Code (2012) Table 1 Parameters are contained within Appendix B of this release

Table 6: Tropicana Gold Mine – 100% basis (IGO 30%) - 30 June 2016 Ore Reserves (and 2015 comparison)

		Ore Reserves - 30 June 2015			Ore Reserves - 30 June 2016		
		Tonnes	Grade	Contained Metal	Tonnes	Grade	Contained Metal
	Classification	(Mt)	Au (g/t)	Au (Moz)	(Mt)	Au (g/t)	Au (Moz)
Open Pit	Proved	11.1	2.27	0.81	7.6	2.33	0.57
	Probable	29.0	2.05	1.91	24.2	2.01	1.56
	Sub-Total	40.1	2.11	2.72	31.8	2.07	2.12
Stockpiles	Proved	8.4	1.09	0.29	9.2	0.98	0.29
GRAND TOTAL		48.5	1.93	3.01	41.0	1.83	2.41

Notes
1. The Proven and Probable Ore Reserves is reported above economic break-even gold cut-off grade for each material type at nominated gold price of US$1,100/oz (A$1,436/oz).
2. The Ore Reserve estimate is update based on depletion as at 30th June 2016, using the Resource model from July 2015.
3. The cut-off grades reported were 0.6/g Au for oxide material and 0.7g/t Au for transitional and fresh.
4. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
5. JORC Code (2012) Table 1 Parameters are contained within Appendix B of this release

Long Operation

The Long Operation is located 60km south of Kalgoorlie and forms part of the world-class Kambalda Dome nickel sulphide mineralised complex. The Long Operation has been in continuous production since 1979, except for a short hiatus in 2002 between transition of ownership from BHP Billiton Nickel West Pty Ltd (formerly WMC Resources Ltd (WMC)) and IGO.

Since the acquisition, IGO has produced over 3.2Mt of nickel ore, containing approximately 125kt of nickel metal from the Long Operation. This is in addition to the 5.4Mt of nickel ore with contained nickel metal of 203kt previously produced from the operation. The Long Operation has a history of continued exploration success including, extensions to known mineralisation defined by WMC and discovery of new mineralisation by IGO at the Long, Victor, Gibb, Gibb South, Victor South, Long North, Moran, McLeay, Moran South and McLeay South orebodies. This discovery record has enabled the operation to maintain an Ore Reserve base to support a two to three year mine life over the previous 13 years of IGO operation. The current life of mine plan extends for the next 18 months based on current Ore Reserves.

Geology

The Long Operation occurs along the eastern flank of the Kambalda Dome in the South Central portion of the Archaean aged Norseman-Wiluna Greenstone belt of the Yilgarn Craton in Western Australia. The Kambalda Dome is a double-plunging antiform with a core of granodiorite flanked by meta-tholeiitic basalt that is overlain by komatiite and intercalated sediments. The host rocks and associated contacts have been subjected to lower amphibolite metamorphism, structural modification, and intrusion by multiple felsic to intermediate igneous dykes that are mostly barren.

The Kambalda style nickel-sulphide orebodies are associated with Archaean aged ultramafic lava channels where molten liquid nickel-sulphides pooled at topographic lows along the komatiite channel. Subsequent folding has rotated the channels from 30° to vertical dip to the east, 10° plunge to the south as well as resulting in the remobilisation of some of the original sulphides into new, structurally imposed positions. The nickel sulphide orebodies are narrow, ribbon-like accumulations of massive to semi-massive sulphides to disseminated sulphides up to 12m thick.

The Long Operation consists of four main deposits in two parallel lava channels: Long and Moran in one channel (the Long Channel), McLeay and Victor South in another (the Victor Channel). Victor South contains the only disseminated sulphides of economic value whereas the other three are massive to semi-massive or matrix ore nickel-sulphide deposits. The Victor Channel was also host to the Gibb, Gibb South and Victor deposits, which are now mined out.

Mineral Resource

The Mineral Resource estimate for FY16, as at 30 June 2016 has been reduced in both tonnes and grade from 30 June 2015, attributed dominantly by mining depletion of the Moran orebody. As at 30 June 2016 the Mineral Resource estimate was **1,259kt at 4.7% Ni for 59,700t of contained nickel metal**. This compares with 1,379kt at 4.8% Ni for 66,400t of contained nickel metal as at 30 June 2015.

Ore Reserve

During FY16, 215kt at an average grade of 3.9% Ni for 8,500t of contained nickel metal were mined, predominantly from the Moran deposit.

The Ore Reserve as at 30 June 2016 was **351kt at 3.9% Ni for 13,600t** of contained nickel metal. This compares with 608kt at 3.6% Ni for 22,000t of contained nickel metal as at 30 June 2015.

Exploration and Resource Extension

In response to low nickel prices, the Long business plan was reviewed in late 2015 and a new business plan implemented, designed to ensure the profitability and sustainability for the current operation at lower nickel prices. As part of the plan all exploration activities where suspended.

Exploration has recommenced in July 2016 with the prioritisation of testing prospective targets that could be easily accessed from the current mine development without significant capital investment. The exploration program planned for FY17 is designed to extend the Long Operation life of mine back to a two to three year window, as has been the case historically.

The first of the targets to be tested which has commenced in July 2016 is directly west of the Victor orebody which has the potential to host a Kambalda style flow channel. Historic drilling by WMC has intersected lower contact mineralisation (eg. KD6165 0.71m @ 5.11%Ni including 0.12m @ 19.20%Ni). The target is approximately 250m west of the current Victor decline mine development.

Table 7: Long Operation - 30 June 2016 Mineral Resources (and 2015 comparison)

		Mineral Resources - 30 June 2015			Mineral Resources - 30 June 2016		
		Tonnes	Grade	Contained Metal	Tonnes	Grade	Contained Metal
	Classification	(t)	Ni (%)	Ni (t)	(t)	Ni (%)	Ni (t)
Long	Measured	65,000	5.4	3,500	62,000	5.3	3,300
	Indicated	287,000	5.1	14,600	287,000	5.1	14,600
	Inferred	355,000	4.7	16,700	355,000	4.7	16,700
	Sub-Total	707,000	4.9	34,800	704,000	4.9	34,600
Victor South	Measured	-	-	-	-	-	-
	Indicated	147,000	2.1	3,100	147,000	2.1	3,100
	Inferred	33,000	1.5	500	33,000	1.5	500
	Sub-Total	180,000	2.0	3,600	180,000	2.0	3,600
McLeay	Measured	63,000	6.3	4,000	61,000	6.4	3,900
	Indicated	71,000	4.9	3,500	71,000	4.9	3,500
	Inferred	21,000	6.7	1,400	21,000	6.7	1,400
	Sub-Total	155,000	5.7	8,900	153,000	5.8	8,800
Moran	Measured	234,000	6.6	15,500	126,000	7.2	9,100
	Indicated	51,000	3.3	1,700	44,000	3.9	1,700
	Inferred	52,000	3.7	1,900	52,000	3.7	1,900
	Sub-Total	337,000	5.7	19,100	222,000	5.7	12,700
Stockpiles	Measured	-	-	-	-	-	-
GRAND TOTAL		1,379,000	4.8	66,400	1,259,000	4.7	59,700

Notes:
1. Mineral Resources are reported using a 1% Ni cut-off grade except for the Victor South disseminated Mineral Resource, which is reported using a cut-off grade of 0.6% Ni.
2. Block modelling used the ordinary-kriging grade-interpolation method on 1m composites within wireframes for all elements and density for the Victor South, McLeay and Moran deposits. For the Long mineralisation, ordinary-kriging was used to estimate metal accumulation and horizontal width variables for each drill hole intercept into a two-dimensional block model. The final block grades were back-calculated and the block model was converted to a conventional three-dimensional block model using nearest neighbour assignment.
3. Mining as at 30 June 2016 has been removed from the 2016 Mineral Resource estimate.
4. Mineral Resources are inclusive of Ore Reserves.
5. All figures are rounded to reflect appropriate levels of confidence. Apparent difference may occur due to rounding.
6. JORC Code (2012) Table 1 Parameters are contained within Appendix C of this release.

Table 8: Long Operation – June 2016 Ore Reserves (and 2015 comparison)

		Ore Reserves - 30 June 2015			Ore Reserves - 30 June 2016		
		Tonnes	Grad	Contained Metal	Tonnes	Grad	Contained Metal
	Classification	(t)	Ni (%)	Ni (t)	(t)	Ni (%)	Ni (t)
Long	Proved	28,000	3.6	1,000	23,000	3.5	800
	Probable	94,000	2.8	2,600	45,000	3.1	1,400
	Sub-Total	122,000	3.0	3,600	68,000	3.2	2,200
Victor South	Proved	7,000	3.0	200	4,000	5.0	200
	Probable	15,000	2.2	300	6,000	1.7	100
	Sub-Total	22,000	2.5	500	10,000	3.0	300
McLeay	Proved	22,000	3.5	800	18,000	3.9	700
	Probable	24,000	3.1	700	19,000	3.2	600
	Sub-Total	46,000	3.3	1,500	37,000	3.5	1,300
Moran	Proved	380,000	4.0	15,200	224,000	4.2	9,400
	Probable	38,000	3.0	1,200	12,000	3.3	400
	Sub-Total	418,000	3.9	16,400	236,000	4.2	9,800
Stockpiles	Proved	-	-	-	-	-	-
GRAND TOTAL		608,000	3.6	22,000	351,000	3.9	13,600

Notes:

1. Ore Reserves are reported above an economic Ni Cut-off value as at 30 June 2016.
2. A NSR value of $176/t has been used in the evaluation of the 2016 Ore Reserve.
3. Mining as at 30 June 2016 has been depleted from the 2016 Ore Reserves estimate.
4. All figures are rounded to reflect appropriate levels of confidence. Apparent difference may occur due to rounding.
5. Revenue factor inputs (US$): Ni $11,766/t, Cu $5,173/t. Exchange rate A$1.00 : US$0.74.
6. JORC Code (2012) Table 1 Parameters are contained within Appendix C of this release.

Jaguar Operation

The Jaguar Operation is located 300km north of Kalgoorlie and 60km north of Leonora. Ore is currently sourced from the Bentley underground mine and processed through the Jaguar concentrator to produce a high grade zinc and copper concentrates with silver and gold credits.

A total of 498kt of ore at 8.9% Zn, 1.8% Cu, 131g/t Ag and 0.77g/t Au was mined from predominantly the Arnage and Comet lenses at Bentley during FY16. Metal production was 39kt Zn, 7.4kt Cu, 1.6Moz Ag, and 4.9koz Au.

Geology

The Jaguar Operation is a cluster of Volcanic Massive Sulphides (VMS) deposits located within the Gindalbie Terrane, a part of the late Archaean Eastern Goldfields Superterrane of the Yilgarn Craton of Western Australia. The area is dominated by volcanic and lesser sedimentary and intrusive rocks that have undergone tilting to sub-vertical positions. Regional metamorphism is lower greenschist facies.

The three principal deposits forming the known VMS cluster are Bentley, Jaguar and Teutonic Bore. The deposits have formed by sub-seafloor replacement, principally of shales and volcanoclastic sediments with mineralisation located in a similar stratigraphic position near a transition from calc-alkaline to tholeiitic volcanism.

The Bentley mineralisation occurs at the contact of a thick basal rhyolitic sequence with an overlying andesite. The rhyolitic sequence is overlain by a sediment unit comprising carbonaceous mudstones and siltstones.

The Bentley massive sulphide mineralisation is banded and consists of pyrite, sphalerite, chalcopyrite, galena and minor pyrrhotite. The upper contact of the massive sulphide is typically sharp. The footwall to massive sulphide consists typically of stringer and disseminated sulphide mineralisation comprising pyrite, chalcopyrite and minor sphalerite.

A dolerite sill at low angle to the Bentley massive sulphide mineralisation and steeper in dip intrudes the orebody, cutting it into five main segments (Arnage, Mulsanne, Brooklands, Comet and Flying Spur lenses).

Mineral Resource

The Mineral Resource estimate for the Jaguar Project as at 30 June 2016 is **3.7Mt at 7.0% Zn, 1.4% Cu, 111g/t Ag and 0.6g/t Au** which is inclusive of both the Teutonic Bore remnant massive sulphide and stringer mineralisation (1.6Mt @ 2.5% Zn, 1.6% Cu, and 49 g/t Ag) and the Bentley Mineral Resource (2.1Mt @ 10.3% Zn, 1.2% Cu, 157g/t Ag and 1.0g/t Au).

Work programs at the Bentley mine completed during FY16 aimed at defining potential life of mine extensions. This entailed an extensive underground drilling program at Bentley from the hanging wall drill drive established in FY15. The drilling program was designed for the conversion of Inferred Mineral Resource to Indicated Mineral Resource category and to drill test mineralisation extensions at depth. As a result, the conversion of Arnage and Flying Spur lenses from Inferred Mineral Resources to Indicated Mineral Resource category extended from 3820mRL in FY15 to 3625mRL in FY16. In addition, the Arnage lens is extended 270m down dip from the FY15 resource boundary. This confirms the Arnage mineralisation is continuous to Bentley Deeps mineralisation identified from FY15 drilling to a depth of 1,000m below surface.

Ore Reserve

Jaguar Operation Ore Reserve as at 30 June 2016 was **1.4Mt at 9.5% Zn, 1.1% Cu, 145g/t Ag and 0.8g/t Au** compared to the 30 June 2015 Ore Reserve of 1.2Mt at 7.6% Zn, 1.7% Cu, 126g/t Ag and 0.7g/t Au.

The resource to reserve drilling programme executed during FY16 has resulted in a 155%, 143% and 142% increase in zinc, silver and gold metals. Copper metal has reduced by 20% inclusive of FY16 mining depletions. This has resulted in the extension of mine life to 3.5 years from June 30 2016.

Exploration and Resource Definition

Planned exploration and resource definition drill programs at the Jaguar Operation during FY17 will focus on:

- Down-dip extensions of the Arnage and Flying Spur lens at Bentley including testing several off hole geophysical electromagnetic conductors.

- Improved definition of the Triumph mineralised system, located 5 km north of the Jaguar processing facility will be completed based on conceptual studies supporting the economic viability of the high-grade core (Stag lens) as incremental ore feed to the Jaguar Operation. Drilling to-date has intersected zinc-copper-silver-gold mineralisation from a vertical depth of 200m extending over a strike length of approximately 400m, with a steep-dip and a shallow southerly-plunge. Mineralisation is in a broad, low to moderate grade envelope around a linear high-grade core of variable thickness (2m to 30m) and dip extent (40m to 100m). Triumph mineralisation has not been included in the Mineral Resource statement as at 30 June 2016.

- Continuation of systematic exploration on a number of prospects in the extensive land package containing over 50km long prospective corridor on a 350km^2 land package. Work programs will focus on Lagonda, Daimler, Bentley North, and Bentley South for potential VMS systems similar in style to the Bentley and Jaguar deposits. Exploration will also be completed on a number of early stage gold prospects including Charlie Chicks, Garden Well #1, Halloween and JunHill.

Table 9: Jaguar Operation – 30 June 2016 Mineral Resources (and 2015 comparison)

		Mineral Resources - 30 June 2015					Mineral Resources - 30 June 2016				
		Tonnes	Grade				Tonnes	Grade			
	Classification	(t)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	(t)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Bentley	Measured	529,000	2.1	11.5	159	0.8	402,000	1.8	11.5	177	0.9
	Indicated	1,252,000	1.6	7.3	118	0.8	1,418,000	1.0	11.0	161	1.0
	Inferred	1,113,000	1.0	8.8	149	1.1	282,000	0.7	5.3	107	1.0
	Stockpiles	13,000	1.1	9.2	121	0.6	5,000	2.0	8.9	131	0.8
	Sub-Total	2,907,000	1.5	8.6	138	0.9	2,107,000	1.2	10.3	157	1.0
		Mineral Resources – 30 August 2009					Mineral Resources – 30 August 2009				
Teutonic Bore	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		4,461,000	1.5	6.5	107	-	3,661,000	1.4	7.0	111	0.6

Notes:
1. FY15 Mineral Resources include massive sulphide and stringer sulphide mineralisation. Massive sulphide resources are geologically defined; stringer sulphide resources for FY15 are reported above a cut-off grade of 0.7% copper. No economic mining constraints were applied to the FY15 Mineral Resource.
2. FY16 massive sulphide Mineral Resource is reported above a cut-off of $96/t NSR. Stringer sulphide (incremental resources) reported above a cut-off of $60/t NSR. Economic mining constraints have been applied to the FY16 Mineral Resource. The change in methodology has resulted in a reduction in the reported tonnages and the associated contained metal between FY15 and FY16. On a like for like basis the FY16 Mineral Resource would have delivered an additional 15% zinc, 31% copper, 14% silver, and 28% gold contained metal.
3. Block modelling mainly used ordinary-kriging grade-interpolation methods within wireframes for all elements and density.
4. All Mineral Resources are depleted for mining
5. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
6. Mineral Resources are inclusive of Ore Reserves.
7. The Teutonic Bore Resource estimate is reported in accordance with JORC Code 2012 reporting guidelines. The model is unchanged from the 2009 model.
8. JORC Code (2012) Table 1 Parameters are contained within Appendix D of this release.

Table 10: Jaguar Operation – 30 June 2016 Ore Reserves (and 2015 comparison)

		Ore Reserves - 30 June 2015					Ore Reserves - 30 June 2016				
		Tonnes	Grade				Tonnes	Grade			
	Classification	(t)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	(t)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Bentley	Proved	323,000	2.0	10.8	155	0.8	277,000	1.8	9.7	157	0.8
	Probable	821,000	1.6	6.3	115	0.7	1,157,000	1.0	9.5	142	0.7
	Sub-Total	1,144,000	1.7	7.6	126	0.7	1,434,000	1.1	9.5	145	0.8
Stockpiles	Proved	13,000	1.1	9.2	121	0.6	4,000	1.7	9.3	138	0.7
GRAND TOTAL		1,157,000	1.7	7.6	126	0.7	1,438,000	1.1	9.5	145	0.8

Notes:
1. Cut-off values were based on NSR values of $134/t ore or direct mill feed and $80/t ore for marginal feed.
2. Revenue factor inputs (US$): copper price $5,540/t, zinc price $2,020/t, silver price $17.00/oz, gold price $1,200/oz and foreign exchange rate of A$1.00 : US$0.75.

3. The following metallurgical recovery factors have been used: 85.0% copper recovery into copper concentrate, 45.0% silver recovery into copper concentrate, 32.0% gold recovery into copper concentrate, 86.0% zinc recovery into zinc concentrate and 16.0% silver recovery into the zinc concentrate.
4. Longitudinal sub-level long hole stoping with unconsolidated rock fill is the primary method of mining.
5. All Measured Resources and associated dilution was classified as Proved Reserves. All Indicated Resources and associated dilution was classified as Probable Reserves. No Inferred Resources has been converted into Reserves.
6. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
7. JORC Code (2012) Table 1 Parameters are contained within Appendix D of this release.

Stockman Project

The Stockman Project is located in the East Gippsland region of eastern Victoria, 460km by road from Melbourne and approximately 19km east-south-east of Benambra. The Stockman Project was acquired as part of IGO's acquisition of Jabiru Metals Ltd in 2011.

The Stockman Project encompasses two copper-zinc-lead-silver-gold VMS deposits, Wilga and Currawong, and various exploration prospects and targets. The larger Currawong deposit is intact, whilst a core of copper-rich ore from the Wilga deposit was mined and processed onsite between 1992 and 1996.

Feasibility level studies have been completed on the Stockman Project with design studies based on a 1.0Mtpa differential flotation concentrator to produce approximately 150ktpa of copper and zinc concentrates over a Project life of approximately ten years.

The Environment Effects Statement (EES) for the Stockman Project, the overarching permitting instrument for the project under the Victorian Environmental Effects Act 1978, received a positive assessment from the Victorian Government, and Project approval from the Federal Government, subject to conditions, in November 2014. This allowed the Stockman Project to proceed to the licensing phase, which is currently being advanced.

As the project approval process progresses, IGO continues to evaluate the Project's economics and strategic fit. A final investment decision will likely only occur once approvals are finalised.

Geology

The Wilga and Currawong deposits are hosted within a Silurian age sedimentary basin that forms part of the Lachlan Fold Belt. Both deposits are hosted by fine-grained sediments and intermediate to felsic volcanics of the Gibson's Folly Formation. Mineralisation occurs as stratiform massive sulphide lenses. Both deposits are approximately 350m in strike and dip extent, dip shallowly to the north and are located some 100m below the surface. The Currawong deposit comprises five massive-sulphide lenses and associated stringer-style mineralisation, stacked by a series of post-mineralisation faults. Located approximately 4km to the south, the Wilga deposit comprises a single massive sulphide lens with an extensive zone of stringer-style mineralisation situated down-dip and along-strike to the west of the massive sulphide mineralisation. The sulphide mineralogy in both deposits is predominantly pyrite, sphalerite, chalcopyrite and minor galena.

Mineral Resource

The Mineral Resource estimate is unchanged since 2012, consisting of 14.0Mt at 2.1% Cu, 4.3% Zn, 38g/t Ag and 1.0g/t Au.

Ore Reserve

The Ore Reserve for the Stockman Project remains unchanged from 30 June 2015 at 9.0Mt at 2.1% Cu, 4.5% Zn, 39g/t Ag and 1.1g/t Au.

Exploration

There remains significant exploration potential on the Stockman Project for repetition of the VMS styles of mineralisation as observed at Wilga and Currawong. Limited exploration programs are planned for FY17 on a number of the prospects.

Table 11: Stockman Project – 30 June 2016 Mineral Resources (and 2015 comparison)

		Mineral Resources - 30 June 2015					Mineral Resources - 30 June 2016				
		Tonnes	Grade				Tonnes	Grade			
	Classification	(Mt)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	(Mt)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Currawong	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	9.5	2.0	4.2	42	1.2	9.5	2.0	4.2	42	1.2
	Inferred	0.8	1.4	2.2	23	0.5	0.8	1.4	2.2	23	0.5
	Sub-Total	10.3	2.0	4.0	40	1.1	10.3	2.0	4.0	40	1.1
Wilga	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	3.0	2.0	4.8	31	0.5[4]	3.0	2.0	4.8	31	0.5[4]
	Inferred	0.7	3.7	5.5	34	0.4	0.7	3.7	5.5	34	0.4
	Sub-Total	3.7	2.3	4.9	32	0.5[4]	3.7	2.3	4.9	32	0.5[4]
GRAND TOTAL		14.0	2.1	4.3	38	1.0[4]	14.0	2.1	4.3	38	1.0[4]

Notes:

1. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
2. The Mineral Resource estimate is unchanged since 2012.
3. 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.
4. Au grades for Wilga are all Inferred due to paucity of Au data in historic drilling.
5. Block modelling used ordinary-kriging grade-interpolation methods within wireframes for all elements and density.
6. Mining as at end of historic mine life (1996) has been removed from the Mineral Resource estimate for Wilga.
7. Mineral Resources are inclusive of Ore Reserves.
8. JORC Code (2012) Table 1 Parameters are contained within Appendix E of this release.

Table 12: Stockman Project – 30 June 2016 Ore Reserves (and 2015 comparison)

		Ore Reserves - 30 June 2015					Ore Reserves - 30 June 2016				
		Tonnes	Grade				Tonnes	Grade			
	Classification	(Mt)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	(Mt)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Currawong	Proved	-	-	-	-	-	-	-	-	-	-
	Probable	7.4	2.1	4.3	40	1.2	7.4	2.1	4.3	40	1.2
	Sub-Total	7.4	2.1	4.3	40	1.2	7.4	2.1	4.3	40	1.2
Wilga	Proved	-	-	-	-	-	-	-	-	-	-
	Probable	1.6	2.1	5.6	31	0.5[2]	1.6	2.1	5.6	31	0.5[2]
	Sub-Total	1.6	2.1	5.6		0.5[2]	1.6	2.1	5.6		0.5[2]
GRAND TOTAL		9.0	2.1	4.5	39	1.1[2]	9.0	2.1	4.5	39	1.1[2]

Notes

1. All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur due to rounding.
2. 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 $8.65/g 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 18% of the Total Ore Reserve.
3. The Ore Reserve was estimated using the NSR method. The NSR value represents unit revenue per tonne net of all off-site costs. These off-site costs included road transport, sea transport, treatment charges, refining costs and state royalties. The NSR value did not include site costs such as mining, geology, processing and site administration. These site costs were applied in the form of an NSR cut-off, used to guide the limits of a practical and economic mining envelope. The Currawong NSR cut-off was $97/t and for Wilga it was $105/t.

4. Revenue factor inputs (US$): Cu $6,591/t, Zn $2,979/t, Ag $20.17/oz, Au $1,146/oz. Exchange rate A$1.00 : US$0.84.
5. Metallurgical recoveries – 81.5% Cu, 40.7% Ag, and 20.4% Au in Cu concentrate; 76.4% Zn and 18.5% Ag in Zn concentrate.
6. Long hole open stoping with cemented paste backfill is the primary method of mining proposed at Stockman.
7. Historic mining at Wilga has been removed from the Ore Reserve estimate.
8. The Ore Reserve estimate includes Inferred and unclassified material in the form of mining dilution estimated to be approximately 780,000t at 0.31 Cu%, 1.0 Zn%, 5.2g/t Ag and 0.1g/t Au.
9. JORC Code (2012) Table 1 Parameters are contained within Appendix E of this release.

JORC Code (2012) Competent Persons Statement

Nova Project Resources and Reserves

The information that relates to the Nova Project Mineral Resources is based on, and fairly represents information and supporting documentation compiled by Mr Mark Drabble and Mr David Hammond. Mr Hammond is an employee of IGO and Mr Drabble is Principal Consultant-Geology of consultancy group Optiro Pty Ltd. Both are members of The Australasian Institute of Mining and Metallurgy and both have sufficient experience relevant to the type and style of mineral deposit under consideration, and to the activity which has been undertaken, to qualify as Competent Persons as defined in the 2012 edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code). Mr Drabble and Mr Hammond consent to the inclusion in this report of the Nova Bollinger Mineral Resource estimate, based on their information in the form and context in which it appears.

The information that relates to the Nova Project Ore Reserves is based on, and fairly represents information and supporting documentation compiled by Mr Brett Hartmann who is a Member of The Australasian Institute of Mining and Metallurgy. Mr Hartmann is a full-time employee of IGO. Mr Hartmann has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration, and to the activity which has been undertaken, to qualify as a Competent Person as defined in the 2012 edition of the JORC Code. Mr Hartmann consented to the inclusion in this report of the Nova Bollinger Ore Reserve estimate, based on his information, in the form and context in which it appears.

Tropicana Gold Mine Resources and Reserves

The information that relates to the Tropicana Mineral Resources is based on, and fairly represents information and supporting documentation compiled by Mr Mark Kent, a full-time employee and security holder 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 2012 edition of the JORC Code. Mr Kent consented to the inclusion in this report of the Tropicana Mineral Resource estimate, based on the information in the form and context in which it appears.

The information that relates to the Tropicana Ore Reserves is based on, and fairly represents information and supporting documentation compiled by Mr Jason Vos, a full-time employee and security holder of AngloGold Ashanti Australia Limited, who is a member of The Australasian Institute of Mining and Metallurgy. Mr Vos 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 2012 edition of the JORC Code. Mr Vos consented to the inclusion in this report of the Tropicana Ore Reserve estimate, based on the information, in the form and context in which it appears.

Long Operation Resources and Reserves

The information in this report that relates to the Long Operation's Mineral Resources is based on, and fairly represents information and supporting documentation compiled by Ms Somealy Sheppard. The information in this report that relates to the Long Operation's Ore Reserves is based on information compiled by Mr Brett Hartmann. Ms Sheppard is a full-time employee of IGO and is a member of the Australian Institute of Geoscientists. Mr Hartmann is a full-time employee of IGO 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. Ms Sheppard 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.

Jaguar Operation Bentley / Teutonic Bore Resources and Reserves

The information in this report that relates to the Bentley Mineral Resources is based on, and fairly represents information and supporting documentation compiled by Mr William Stewart. The information in this report that relates to the Teutonic

Bore Mineral Resources is based on information compiled by Mr Stewart. Mr Stewart is a full-time employee of IGO and member of The Australasian Institute of Mining and Metallurgy and member of Australian Institute of Geoscientists.

The information in this report that relates to the Bentley Ore Reserves is based on information compiled by Mr Shane McLeay who is a Fellow of The Australasian Institute of Mining and Metallurgy. Mr McLeay is a full-time employee of Entech Pty Ltd. Mr Stewart and Mr McLeay have sufficient experience 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 Stewart and Mr McLeay consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

Stockman Project Currawong and Wilga Resources and Reserves

The information in this report that relates to the Stockman Mineral Resources is based on, and fairly represents information and supporting documentation compiled by Mr Matthew Dusci. Mr Dusci is a full-time employee of IGO and is a member of the Australian Institute of Geoscientists. Mr Dusci 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 Dusci 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 Reserves is based on, and fairly represents information and supporting documentation compiled by Mr Geoff Davidson who is a Fellow 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.

Annual Report Mineral Resource and Ore Reserve Statement

The information in this report that relates to the Independence Group Annual Report Mineral Resources and Ore Reserves Statement as a whole is based on information compiled by Mr. Dusci who is a member of Australian Institute of Geoscientists and is a full-time employee of IGO. The Annual Report Mineral Resources and Ore Reserves Statement is based on, and fairly represents, information and supporting documentation prepared by the above-named Competent Persons. The Annual Report Mineral Resources and Ore Reserves Statement has been issued with the prior written consent of Mr. Dusci, in the form and context in which it appears in the Annual Report.

Mineral Resource and Ore Reserve Governance

In estimating Mineral Resources and Ore Reserves the Competent Person(s) for each estimate is (are) responsible for:

- *Adopting annual Board approved metal prices and foreign exchange assumptions for use in estimates*
- *Monitoring the planning, progress, estimation and reporting of Mineral Resources and Ore Reserves to meet IGO standards and timelines*
- *JORC Code compliant reporting*
- *Periodic internal review of process, data, estimates and reports*
- *Periodic external review of data, Estimates and reports for new or materially changed estimates.*

Independence Group NL reports its Mineral Resources and Ore Reserves on an annual basis, in accordance with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code) 2012 Edition. Mineral Resources are quoted inclusive of Ore Reserves.

Competent Persons named by Independence Group NL are Members or Fellows of the AusIMM and/or the Australian Institute of Geoscientists, and qualify as Competent Persons as defined in the JORC Code.

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APPENDIX A

Nova Bollinger Mineral Resource and Ore Reserve 2016

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques **Note: Due to the similarity of the deposit setting, procedures and estimation these tables present the combined Nova-Bollinger tabulation. All references to the Bollinger deposit are in bold font, and Nova is in normal font.**	**The Bollinger deposit was sampled using diamond drill holes (DD) on a nominal 25m x 25m to 50m x 50m grid spacing. A total of 72 DD holes were drilled for 35,935m. Holes were generally angled towards grid west between -60^0 and -90^0 to optimally intersect the mineralised zones.** The Nova deposit was sampled using Reverse Circulation (RC) and diamond drill holes (DD) on nominal 25m x 25m grid spacing. A total of 15 RC and 163 DD holes were drilled for 2,910m and 63,099m respectively. Holes were generally angled towards grid west at varying angles to optimally intersect the mineralised zones.
	Bollinger is defined by diamond drilling only, and uses the same measures employed at Nova for controls and sample representivity. The drill hole locations were picked up and downhole surveyed by survey contractors. Initial RC drilling identified the Nova target and diamond core was used to delineate the resource. The RC samples were collected by cone or riffle splitter. Diamond core was used to obtain high quality samples that were logged for lithological, structural, geotechnical, density and other attributes. Sampling was carried out under Sirius protocols and QAQC procedures as per industry best practice.
	Diamond core is HQ and NQ2 size, sampled on geological intervals (0.2m to 1.2m), cut into half (NQ2) or quarter (HQ) core to give sample weights under 3kg. Samples were crushed, dried and pulverised (total prep) to produce a sub-sample for analysis by four acid digest with an ICP/OES or ICP/MS finish and fire assay (Au, Pt, Pd) with MS finish. Diamond core is HQ (metallurgical holes) or NQ2 size, sampled on geological intervals (0.2m to 1.3m), cut into half (NQ2) or quarter (HQ met) core to give sample weights under 3kg. Samples were crushed, dried and pulverised (total prep) to produce a sub-sample for analysis by four acid digest with an ICP/OES or ICP/MS finish and fire assay (Au, Pt, Pd) with MS finish. Reverse circulation drilling was used to obtain 1m samples from which 3kg was pulverised (total prep) to produce a sub-sample for assaying as above.
Drilling techniques	**Diamond drilling accounts for 100% of the current drilling at Bollinger and comprises NQ2 or HQ sized core. Pre-collar depths range from 20m to 84m and hole depths range from 450m to 667m. The core was oriented using a Camtech orientation tool.** Diamond drilling accounts for 96% of the drilling in the resource area and comprises NQ2 or HQ sized core. Pre-collar depths range from 6m to 150m and hole depths range from 144m to 667m. The core was oriented using a Camtech orientation tool with 71% of orientations rated as "good". RC drilling accounts for 4% of the total drilling and comprises 140mm diameter face sampling hammer drilling. Hole depths range from 90m to 280m.
Drill sample recovery	Diamond core and RC recoveries are logged and recorded in the drill hole database. Overall recoveries are >95% for Nova and **Bollinger** and there are no core loss issues or significant sample recovery problems.
	Diamond core at Nova and **Bollinger** is reconstructed into continuous runs on an angle iron cradle for orientation marking. Depths are checked against the depth given on the core blocks and rod counts are routinely carried out by the drillers. RC samples were visually checked for recovery, moisture and contamination.
	The Bollinger mineralisation is defined by diamond core drilling, which has high recoveries. The bulk of the Nova resource is defined by diamond core drilling, which has high recoveries. The massive sulphide style of mineralisation and the consistency of the mineralised intervals are considered to preclude any issue of sample bias due to material loss or gain.
Logging	Geotechnical logging at Nova and **Bollinger** was carried out on all diamond drillholes for recovery, RQD and number of defects (per interval). Information on structure type, dip, dip direction, alpha angle, beta angle, texture, shape, roughness and fill material is stored in the structure table of the database.
	Logging of diamond core and RC samples at Nova and **Bollinger** recorded lithology, mineralogy, mineralisation, structural (DD only), weathering, colour and other features of the samples. Core was photographed in both dry and wet form.

Criteria	Commentary
	All drillholes were logged in full, apart from rock roller diamond hole pre-collar intervals of between **20m to 60m depth (Bollinger)** and 20m to 60m (Nova).
Sub-sampling techniques and sample preparation	Core for Nova and **Bollinger** were cut in half (NQ2) and quarter core (HQ) onsite using an automatic core saw. All samples were collected from the same side of the core.
	RC samples were collected on the rig using cone splitters. All samples in mineralised zones were dry.
	The sample preparation of diamond core for Nova and **Bollinger** follows industry best practice in sample preparation involving oven drying, coarse crushing of the half core sample down to ~10mm followed by pulverisation of the entire sample (total prep) using Essa LM5 grinding mills to a grind size of 85% passing 75µm. The sample preparation for RC samples is identical, without the coarse crush stage.
	Field QC procedures involve the use of certified reference material as assay standards, along with blanks, duplicates and barren washes. The insertion rate averaged 1:15 **for both projects**, with an increased rate in mineralised zones.
	No field duplicates were taken for Bollinger. Samples were selected to weigh less than 3kg to ensure total preparation at the pulverisation stage. Field duplicates were taken on 1m composites for RC, using a riffle splitter. One twinned diamond hole was drilled at Nova. This hole supported the location of the geological intervals intersected in the first drillhole (no assays were taken as this is a metallurgical hole).
	The sample sizes are considered appropriate to correctly represent the sulphide mineralisation at Bollinger based on: the style of mineralisation (massive sulphides), the thickness and consistency of the intersections, the sampling methodology and percent value assay ranges for the primary elements. The sample sizes are considered appropriate to correctly represent the sulphide mineralisation at Nova based on: the style of mineralisation (massive sulphides), the thickness and consistency of the intersections, the sampling methodology and percent value assay ranges for the primary elements.
Quality of assay data and laboratory tests	**The analytical techniques used a four acid digest multi element suite with ICP/OES or ICP/MS finish (25 gram FA/MS for precious metals).** The analytical techniques used a four acid digest multi element suite with ICP/OES or ICP/MS finish (25 gram or 50 gram FA/MS for precious metals). The acids used are hydrofluoric, nitric, perchloric and hydrochloric acids, suitable for silica based samples. The method approaches total dissolution of most minerals. Total sulphur is assayed by combustion furnace.
	No geophysical tools were used to determine any element concentrations used in either resource estimate.
	Sample preparation checks for fineness were carried out by the laboratory as part of their internal procedures to ensure the grind size of 85% passing 75µm was being attained. One diamond hole had duplicates taken from the half core after coarse crushing and the results were within 3% of the original sample values. Laboratory QAQC involves the use of internal lab standards using certified reference material, blanks, splits and replicates as part of the in-house procedures. Umpire laboratory campaigns with two other laboratories have been carried out as independent checks of the assay results using **201 pulp samples and standards sent to ALS,** (Nova 2,590 samples) and these show good precision. Certified reference materials, having a range of values, were inserted blindly and randomly. Results highlight that sample assay values are accurate and that contamination has been contained. The diamond drilled core pulp duplicates had more than 90% of its pairs with differences (half absolute relative differences or HARD values) below 10% (Ni, Cu, Co), which concurs with industry best practice results. Repeat or duplicate analysis for samples reveals that precision of samples is within acceptable limits.
Verification of sampling and assaying	**Both the Managing and the Technical Director of Sirius have visually verified significant intersections in diamond core from Bollinger in 2013. Optiro has viewed the intersections of metallurgical core and checked core photos against the assay and geology logs.** Optiro has visually verified significant intersections in diamond core as part of the resource estimation process.
	No twin holes were drilled at Bollinger to date. Two PQ and one HQ metallurgical holes have been drilled at Nova since March 2013 and the logging supports the interpreted geological and mineralisation domains. One hole at Nova was twinned (SFRD0117 and SFRD0117W1M). The results confirmed the initial intersection geology. The twin (suffixed W1M) was used as a metallurgical hole.
	Primary data were collected for **both projects** using a set of standard Excel templates on Toughbook laptop computers using lookup codes. The information was sent to ioGlobal for validation and compilation into a SQL database server.
	No adjustments or calibrations were made to any assay data used in **either** estimate.

Criteria	Commentary
Location of data points	Hole collar locations for **all holes** were surveyed by Whelans Surveyors of Kalgoorlie using RTK GPS connected to the state survey mark (SSM) network. Elevation values were in AHD RL and a value of +2,000m was added to the AHD RL for local co-ordinate use. Expected accuracy is +/– 30mm for easting, northing and elevation coordinates. Downhole surveys used single shot readings during drilling (at 18m, then every 30m) and Gyro Australia carried out gyroscopic surveys using a Keeper high speed gyroscopic survey tool with readings every 5m, after completion of the drill hole. Stated accuracy is +/-0.25° in azimuth and +/-0.05° in inclination. QC involved field calibration using a test stand. Only gyro data are used in the resource estimate.
	The grid system for **Nova-Bollinger** is MGA_GDA94, zone 51 (local RL has 2,000m added to value). Local easting and northing are in MGA.
	Topographic surface for **Nova-Bollinger** uses 2012 Lidar 50cm contours.
Data spacing and distribution	**The nominal drill hole spacing is 25m (northing) by 25m (easting) in the core of the deposit, and is up to 50m by 50m on the margins.** The nominal drill hole spacing is 25m (northing) by 25m (easting) for Nova.
	The mineralised domains for **Nova-Bollinger** have demonstrated sufficient continuity in both geological and grade continuity to support the definition of Mineral Resources and Reserves, and the classifications applied under the 2012 JORC Code.
	Samples have been composited to one metre lengths for **both projects**, and adjusted where necessary to ensure that no residual sample lengths have been excluded (best fit).
Orientation of data in relation to geological structure	**The deposit is drilled towards grid west at angles varying from -60° and -90° to intersect the mineralised zones at a close to perpendicular relationship for the bulk of the deposit.** The deposit is drilled to grid west, which is slightly oblique to the orientation of the mineralised trend; however the intersection angles for the bulk of the drilling are nearly perpendicular to the mineralised domains. Structural logging based on oriented core indicates that main sulphide controls are largely perpendicular to drill direction.
	No orientation based sampling bias has been identified at **Nova-Bollinger** in the data.
Sample security	Chain of custody was managed by the company. Samples for **Nova-Bollinger** were stored on site and were either delivered by company staff to Perth and then to the assay laboratory, or collected from site by Centurion transport and delivered to Perth, then to the assay laboratory. Whilst in storage, samples were stored in a locked yard. "Tracking sheets" track the progress of batches of samples.
Audits or reviews	A review of the sampling techniques and data was carried out by Optiro as part of each resource estimate and the database is considered to be of sufficient quality to carry out resource estimation. An internal system audit was undertaken by Sirius in November 2012.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	**Nova and Bollinger** are located wholly within M28/376. IGO has a 100% interest in the ML following acquisition of Sirius effective 22 September 2015. The tenement is located within the Ngadju Native Title Determination Area.
	The tenement is in good standing and no known impediments exist.
Exploration done by other parties	No previous systematic exploration has been undertaken at the Nova-**Bollinger** Project prior to the work by Sirius.
Geology	The global geological setting is a Proterozoic aged gabbroic intrusion(s) within metasediments situated in the Albany Fraser mobile belt. It is a high grade metamorphic terrane. The sulphide mineralisation is related to, and is part of the intrusive event. The deposits are analogous to other mafic-hosted nickel-copper deposits which occur worldwide.
Drill hole Information	No new exploration data are announced within this report.
Data aggregation methods	No new exploration data are announced within this report.
Relationship between mineralisation widths and intercept lengths	The Nova deposit is moderately east dipping in the west, flattening to shallow dipping in the east. The fans of drillholes are inclined between -54° and -90° to the west to allow intersection angles with the mineralised zones to approximate the true width. **The Bollinger deposit is dominantly flat lying and is drilled to grid west with drill holes inclined between -60° and -90°. The intersection angles for the drilling appear to be close to perpendicular to the mineralised zones, therefore reported downhole intersections approximate true width.**
Diagrams	No new exploration data are announced within this report.

Criteria	Commentary
Balanced reporting	No new exploration data are announced within this report.
Other substantive exploration data	All samples are measured for their bulk density which in the Nova-**Bollinger** deposit range from 2.90 g/cm^3 to 4.66g/cm^3. Multi element assaying is conducted routinely on all samples for a suite of potentially deleterious elements including Arsenic, Sulphur, Zinc and Magnesium. Geotechnical logging was carried out on all diamond drillholes for recovery, RQD and number of defects (per interval). Information on structure type, dip, dip direction, alpha angle, beta angle, texture, shape, roughness and fill material is stored in the structure table of the database.
Further work	Underground mapping in decline development confirms geotechnical assumptions. Mapping of ore development commenced in early 2016 in parallel with diamond grade control drilling conducted from a service drives above the Nova and **Bollinger** orebodies to a nominal 12.5m x 12.5m spacing. This work will start in early 2016 and will be an ongoing process to allow the geological model to be refined for final underground detailed design. The Mineral Resource will be updated with this new data in late 2016.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Data templates with lookup tables and fixed formatting are used for logging, spatial and sampling data at **Nova-Bollinger**. Data transfer is electronic via e-mail. Sample numbers are unique and pre-numbered bags are used. These methods all minimise the potential for errors.
	Data validation checks are run by database management consultancy “ioGlobal” using their proprietary software (“ioHub”). ioGlobal have their own database model with a production and quarantine database for each client. Data are validated from quarantine to upload using a set of validation rules developed by Sirius and ioGlobal. **Data for Nova-Bollinger are stored in a single database.**
Site visits	**Mark Drabble (Principal Consultant - Optiro), who is acting as Competent Person, viewed the metallurgical drill core at AMMTEC on 28th June 2013.** Optiro carried out a site visit to the Nova deposit on the 21st of February 2013. Mark Drabble inspected the deposit area, the core logging and sampling facility and density measurement area. During this time, notes and photos were taken along with discussions were held with site personnel regarding the available drill core and procedures. Diamond core was also viewed in the Sirius offices in Perth on three occasions. A number of minor recommendations were made on procedures but no major issues were encountered. In addition, Mr Drabble viewed drill core in the Sirius offices in Balcatta on a number of occasions in 2013.
Geological interpretation	The confidence in the geological interpretation of Nova and **Bollinger** is considered good. The global geological setting is a gabbroic intrusion(s) within metasediments within a high grade metamorphic terrane. The sulphide mineralisation is related to, and part of, the intrusive event. **The Bollinger deposit appears to be intimately related to the Nova deposit and represents part of a number of intrusive events that transgress sedimentary layers to the immediate east of Nova**. The **Nova-Bollinger** deposit appears similar in style to many mafic-hosted nickel-copper deposits.
	Petrography and litho-geochemistry have been used to assist identification of the rock type subdivisions applied in the interpretation process.
	The **Nova-Bollinger** deposit is generally tabular in geometry, with clear boundaries which define the mineralised domains. Infill drilling has supported and refined the model and the current interpretation is thus considered to be robust.
	Geological controls and relationships were used to define sub-domains. Key features are sulphide content, form and multi-element geochemistry relationships.
	The Bollinger disseminated zone has small intervals of massive sulphide that required sub-domaining to constrain the estimation of metal around these samples. The Nova lower breccia zone has mixed grade populations due to variable clast versus massive sulphide content. This can be seen in the MgO and nickel grade relationships and influences the local rather than the global grade estimate. These factors have been addressed via the resource estimation process applied.
Dimensions	**The Bollinger Mineral Resource area abuts** the Nova area **and has dimensions of 300m**

Criteria	Commentary
	(north) by 400m (east) and 125m (elevation). The Bollinger resource has a maximum depth of 450m below surface. The Nova and Bollinger deposits are conjoined by a feeder zone. The two resources areas are arbitrarily split along a North-South line defined by the 518,600mE MGA grid line. The Nova Mineral Resource starts at a depth of 40m below surface. The Resource area has dimensions of 450m (north) by 550m (east) and 400m (elevation).
Estimation and modelling techniques	Grade estimation using Ordinary Kriging (OK) was completed for Nova and **Bollinger**. CAE Studio 3 software was used to estimate six elements; Ni%, Cu%, Co%, Fe%, Mg (ppm) and S%, as well as bulk density. Drill grid spacing ranges from 25m to 50m. Drillhole sample data were flagged using domain codes generated from three dimensional mineralisation domains and oxidation surfaces. Sample data were composited per element to a one metre downhole length using a best fit-method. There were consequently no residuals. Intervals with no assays were excluded from the compositing routine. The influence of extreme sample distribution outliers was reduced by top-cutting where required. The top-cut levels were determined using a combination of top-cut analysis tools (grade histograms, log probability plots and CVs). Top-cuts were reviewed and applied on a domain basis. Due to the folded nature of the Lower Massive domain at Nova **and the Massive domain at Bollinger,** an industry accepted unfolding routine was carried out using CAE Studio 3 software. Variography and grade estimation of these domains were completed in unfolded space. It was noted that the Lower Massive domain at Nova **and the Massive and the Carapace domain at Bollinger** showed evidence of sub-populations within the domains which were not able to be wireframed separately at the available grid spacing. A categorical indicator approach using three grade bins at Nova **and two grade bins within the Bollinger domains was considered appropriate to sub-domain these populations.** It was interpreted that these sub-domains represented massive, breccia and/or low-grade mineralisation. **Several domains which demonstrated a moderate degree of folding at Bollinger were estimated using flattening routines or Dynamic Anisotropy in order to optimise the grade estimation. Variography of these domains was completed in 2D space.** For all domains, directional variograms were modelled using traditional variograms or normal scores transformations. Nugget values are moderate to high (Nova <0.5**, Bollinger <0.3).** Grade continuity was variable in either resource depending on mineralisation styles and ranged from 50m to 170m in the major direction. Small or poorly sampled domains where robust variography could not be generated used the variography of a geologically similar domain. Estimation searches for all elements were set to the ranges of the nickel variogram for each domain.
	No previous mining activity has taken place in this area. Check estimates have been complted during the development drilling of the deposit and have produced very similar global estimates for the Nova-Bollinger deposit.
	The main by-product of the resource is cobalt and recovery will be as a by-product with the pentlandite. This is dependent on any off-take agreement and may realise a credit.
	The non-grade elements estimated are Fe%, Mg% and S%.
	A single block model for Nova**-Bollinger** was constructed using an 8mE by 12mN by 4mRL parent block size with subcelling to 1mE by 1mN by 0.25mRL for domain volume resolution. All estimation was completed at the parent cell scale. Kriging neighbourhood analysis was carried out for Nova in order to optimise the block size, search distances and sample numbers used. Discretisation was set to 4 by 6 by 2 for all domains. The size of the search ellipse per domain was based on the nickel variography, due to the moderate-strong correlation of nickel with the other elements. Three search passes were used for each domain. In general, the first pass used the ranges of the nickel variogram and a minimum of 8 and maximum of 30 samples. In the second pass the search ranges were changed to double the ranges of the nickel variogram, maintaining a minimum of 8 samples. The third pass ellipse was extended to **3 times the range of the variograms for Bollinger** and 5 times for Nova. A minimum of 4 and a maximum of 30 samples were applied. A maximum of 5 samples per hole were used. In the majority of domains, most blocks were estimated in the first pass (particularly for the main domains); however, some more sparsely-sampled domains were predominantly estimated on the second or third pass. Non-estimated blocks, i.e. those outside the range of the third pass, were assigned the estimated domain mean and lower resource confidence classification. Hard boundaries were applied between all estimation domains, excluding the alteration envelope

Criteria	Commentary
	at Nova where a soft boundary with the disseminated domain was used.
	No selective mining units were assumed in this estimate.
	Neural networking (3D spatial analysis) was used to determine relationships between the variables at Nova in the initial estimate. These were then incorporated into the domain interpretation process. Strong positive correlation exists between nickel and all other elements estimated, with the exception of copper. The correlation between nickel and copper is variable; based on domain and mineralisation style. All elements within a domain used the same sample selection routine for block grade estimation.
	The geological interpretation correlated the sulphide mineralisation to geological and structural elements at Nova- **Bollinger**. The structural framework and understanding of primary magmatic and remobilised mineralisation was used to refine the mineralisation domains. These domains were used as hard boundaries to select sample populations for variography and estimation.
	Statistical analysis showed the populations in each domain at Nova and **Bollinger** to generally have a low coefficient of variation but it was noted that a very small number of estimation domains included outlier values that required top-cut values to be applied.
	Validation of the block model included a volumetric comparison of the resource wireframes to the block model volumes. Validating the estimate compared block model grades to the input data using tables of values, and swath plots showing northing, easting and elevation comparisons. Visual validation of grade trends and metal distributions was carried out. Minor mining has taken place but broken stocks remain unprocessed.
Moisture	The tonnages are estimated on a dry basis.
Cut-off parameters	A nominal grade cut-off of 0.4% Ni appears to be a natural grade boundary between disseminated and trace sulphides for the Nova-**Bollinger** mineralised system. This cut-off grade was used to define the mineralised envelope within which the higher grade sub-domains were interpreted. Mineral Resources are reported above a 0.6% NiEq Cut-off grade. NiEq% = ((Cu % x 0.95) x ($7,655/$16,408)) + (Ni % x 0.89).
Mining factors or assumptions	Mining of the Nova-**Bollinger** deposit will be dominantly by underground mining methods involving mechanised mining techniques. The geometry of the deposit will make it amenable to mining methods currently employed in many underground operations in similar deposits around the world. No assumptions on mining methodology have been made.
Metallurgical factors or assumptions	Mineralogy shows the main sulphide minerals as chalcopyrite, pentlandite and pyrrhotite. Chalcopyrite is largely liberated, however some fine pentlandite is associated with the pyrrhotite. Gangue minerals include olivine/pyroxene, amphibole, feldspars, garnets, quartz which are un-altered. The Concentrator is designed for a nominal 1.5Mtpa capacity. Processing will comprise conventional crushing, milling and classification circuits followed by dual flotation circuits to produce separate nickel (+cobalt) and copper (+silver) concentrates. Detailed testwork in 2013/2014 has developed a split concentrate flowsheet that has achieved separation between copper and nickel for production of separate concentrates with acceptable recoveries. The results to date show a robust processing flowsheet than can consistently achieve a copper concentrate grading 27 – 31% Cu for 95% overall recovery and a nickel concentrate grading 13 - 17% Ni for 89% overall recovery.
Environmental factors or assumptions	No assumptions have been made.
Bulk density	Bulk density has been estimated from density measurements carried out on **7,950 (Bollinger)** and 12,429 (Nova) full length core samples using the Archimedes method of dry weight versus weight in water. The use of wax to seal the core was trialled but was shown to make less than 1% difference. Density standards were used for QAQC using an aluminium billet, and pieces of core with known values. The density ranges for the mineralised units are listed below: Massive sulphides 2.0 to 4.7g/cm^3 (average: 3.9g/cm^3), net textured sulphides 3.0 to 4.4g/cm^3 (average: 3.6g/cm^3) and disseminated sulphides 2.5 to 4.6g/cm^3 (average: 3.5g/cm^3).
	The host geology comprises high grade metamorphic rocks that have undergone granulite facies deformation. The rocks have been extensively recrystallised and are very hard and competent. Vugs or large fracture zones are generally annealed with quartz or carbonate in breccia zones. Porosity in the mineralised zone is low. Sensitivity to these issues is thus low.
	The bulk density values were estimated using the nickel search parameters and **7,950** plus 12,429

Criteria	Commentary
	density samples taken within the geological domains.
Classification	**The Mineral Resource classification at Bollinger is based on good confidence in the geological and grade continuity, along with 25m by 25m spaced drill hole density in the core and bulk of the deposit, and 50m x 50m on the margins.** The Mineral Resource classification at Nova is based on good confidence in the geological and grade continuity, along with 25m by 25m spaced drill hole density throughout. Estimation parameters including Kriging efficiency have been utilised during the classification process.
	The input data are comprehensive in coverage of the mineralisation and do not favour or misrepresent *in situ* mineralisation. Geological control at Nova-**Bollinger** consists of a primary mineralisation event modified by metamorphism and structural events. The definition of mineralised zones is based on a high level of geological understanding producing a robust model of mineralised domains. This model has been confirmed by infill drilling which supported the initial interpretation. The validation of the block model shows good correlation of the input data to the estimated grades.
	The Mineral Resource estimate appropriately reflects the view of the Competent Person.
Audits or reviews	This is the maiden **Bollinger** Mineral Resource estimate and an update of the Nova March 2013 Mineral Resource estimate. The Nova resource was reviewed by Optiro and some improvements made to the geological domains as a result of the new information at Bollinger.
Discussion of relative accuracy/confidence	The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per the guidelines of the 2012 JORC Code.
	The statement relates to global estimates of tonnes and grade.
	No production data are available for comparison and reconciliation. The boxcut has been completed and decline access to the orebody is underway.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The Underground Ore Reserve estimate is based on the Mineral Resource estimate carried out by Optiro Pty Ltd. *ASX announcement 15 July* 2013 “Maiden Bollinger Resource and Scoping Study Update”. The June 2016 Ore Reserves are December 2015 with minor broken stocks
	The Mineral Resources reported are inclusive of the Ore Reserve.
Site visits	The Competent Person is employed by IGO on the Nova site.
Study status	Feasibility level studies have been completed for all areas of the Nova Nickel Project and the project is under construction. The current Ore Reserve estimates are based around the assumptions completed for the Nova Nickel Project Feasibility Study and enhanced in the IGO Optimisation Study.
Cut-off parameters	In order to determine the economically mineable part of the resource, the total value of the mineralised material was calculated, including recognition of the value of nickel, copper and cobalt in the ore. This value, commonly referred to as a Net Smelter Return (NSR) is calculated in Australian dollars per ore tonne and represents the value of the products produced from one tonne of ore if sold at the mill gate. It is calculated from the revenue received from the payable metal (mill recovered) contained in the products less all costs and charges downstream of the site including transportation, smelting, refining and metal loss throughout these stages. The NSR cut-off calculations were conducted by prior to designing the underground mine, and again following completion of the design, scheduling and cost modelling. The initial estimation that was used for Feasibility Study mine design purposes was based on processing, treatment, refining, mining, administration and operating cost estimates from the Sirius Scoping Study. The operating cost generated from the Nova Underground financial model is $105/tonne of ore comprising: • Mining cost of $55/t • Processing cost of $38/t • Admin cost of $12/t. Metal prices are based on 12 month averages (not volume-weighted) of spot prices from the London Metal Exchange between June 2012 and July 2013 and were provided by Sirius, prices are

Criteria	Commentary
	as follows: • Nickel – US$7.44/lb • Copper – US3.47/lb • Cobalt – US$12.00/lb • Exchange rate - $A 1: $US 0.90
Mining factors or assumptions	The Ore Reserve estimate has been calculated by generating detailed mining shapes for each stoping block as well as development. Designed stope shapes include planned dilution, being waste material that is located within the mineable stope shape. Additional unplanned dilution is also generally incurred from the walls of stopes due to re-distribution of stress within the rock mass as voids are created in the mine, blast damage, poor mining practice (such as poor blasthole drilling setup). This additional material is also included in Ore Reserve Estimate. A 7% unplanned dilution factor has been calculated by Entech in consultation with SRK based on kinematic and empirical methods.
	The selected mining methods for the Nova Project is long-hole sub-level open-stoping which is widely used in many underground mines in Western Australia and is deemed appropriate considering the nature of the ore body, and the desire to extract the maximum value from the deposit.
	Stope sizes are generally 25mW by 25mH by the orebody width and have been created to suit the Mineral Resource model. As the resource changes in width and dip the mining method changes from large multi-lift stopes to echelon retreat single access stopes. Geotechnical assessment of the mineralised zone is also favourable for the selected mining method. In consultation with SRK geotechnical parameters have been set out for the size of the stoping blocks as well as support standards and development stand-off distances. All mining shapes included in the Ore Reserve estimate abide by the recommendations supplied by SRK.
	A mining recovery factor of 95% has been applied post geological interrogation to generate the final diluted and recovered Ore Reserve estimate. This mining recovery is applied to allow for any ore loss due to mining related issues such as; underbreak due to poor drilling and blasting techniques, stope bridging or freezing or material being left in stopes due to inaccessibility.
	Minimum mining width for stoping is 4m.
	Grade control drilling is underway from UG drill platforms on a nominal 12.5m x 12.5m pattern on the footwall.
	No Inferred Mineral Resources have been included in the Ore Reserve Estimate. Any Inferred Mineral Resource contained within a mining block (stope or development) is classified as waste and is used to dilute the overall Ore Reserve.

	Infrastructure build for Nova is still underway, with the remaining been accounted for and included in all work leading to the generation of the Ore Reserve estimate. The Nova Nickel Project infrastructure includes:
	• All site surface infrastructure, including:
	  o Processing facilities, including crushing, grinding, flotation and dewatering
	  o Tailings storage facility
	  o Offices, workshops, warehouses and associated facilities
	  o Borefield and pipeline
	  o Camp
	  o Airstrip
	  o Access Road
	  o Power Station
	• Paste filling infrastructure. The backfilling of the production stopes is an integral component of the mining method at Nova for all stope sizes and configurations. Paste fill utilising classified live tailings is the nominated fill type. A Paste Plant will be located above the orebody on the surface and will comprise: tailings storage tank(s); filter; binder storage; mixer and associated facilities. Paste will be delivered underground by gravity through a reticulation system consisting of boreholes and horizontal piping.
	• All power and pumping reticulation will be fed through decline development, ventilation rises and service holes drilled in close proximity to the decline to minimise cable and pipe runs along the decline path.
	• Ventilation fans will be installed underground at the base of a raisebored shaft to supply fresh air to underground workings. Return air ventilation system to be located on opposite side of the deposit to the decline to allow for flow through ventilation. Caged ladderways will be installed in fresh air ventilation rises to establish a second means of egress from underground project.

Metallurgical factors or	Mineralogy shows main sulphide minerals as chalcopyrite, pentlandite and pyrrhotite. Chalcopyrite

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td rowspan="5">assumptions</td><td>is largely liberated, however some fine pentlandite is associated with the pyrrhotite. Gangue minerals include olivine/pyroxene, amphibole, feldspars, garnets, quartz which are un-altered.

The nameplate of the processing facility is 1.5Mtpa capacity plant. Processing will comprise conventional crushing, milling and classification circuits followed by dual flotation circuits to produce separate nickel (+cobalt) and copper (+silver) concentrates.</td></tr>
<tr><td>The Nova-Bollinger deposit is different from other local nickel deposits of Norlisk – Lake Johnston, Western Areas – Forrestonia, Panoramic – Lanfranchi and Mincor - Widgemooltha which are near Norseman to the West and North.

The nearest analogous deposits are in Canada such as Thompson (owned by Vale), Raglan (owned by Xstrata) and Voisey's Bay (owned by Vale) who are using fresh water in the processing.</td></tr>
<tr><td>The split concentrate flowsheet has achieved separation between copper and nickel for production of separate concentrates with acceptable recoveries. The results to date show a robust processing flowsheet than can consistently achieve a copper concentrate grading 27 – 31% Cu for 95% overall recovery and a nickel concentrate grading 13 - 17% Ni for 89% overall recovery.

The copper concentrate is low in nickel (<0.5%) and represents <0.5% nickel recovery. It should be noted that the grades and recoveries reported cannot be paired as there are middling streams in the batch flowsheet that are yet to be allocated to either concentrate or tails. That is, recoveries are rougher recoveries, whilst grades are cleaner grades. The final copper and nickel flotation recovery for this flowsheet will be determined from planned locked cycle testwork.

The testing is investigating two potential reagent regimes for split flotation, one using TETA (tri-ethylene-tetra-amine) with sodium sulphite and Cytec Industries polymeric depressant (7261A). These are all used in commercial flotation processes, more commonly in North America, less commonly in Australia. Selective sulphide flotation is considered a well-tested technology.

Flotation testing has shown the ability to produce a combined bulk concentrate or a separate split concentrate in hyper-saline site water.

Economic evaluations concluded that a split concentrate option will achieve a higher revenue than a combined concentrate, due to the increased pay-ability of the copper. Split concentrate offers flexibility, marketing options and was adopted as the preferred flowsheet for the Feasibility Study.

Composite A was formulated as the main testing composite to be used in further development testwork. Composite A is based on the following criteria:
Year 1-3 stoping material
All MET holes below 2005 RL
Including mining dilution as advised by Entech, and agreed by Sirius, nominally 2.5m HW and 0.5m FW.
Every second metre from 9 holes.
Composites B - P includes all major material types of Disseminated in Gabbro, Stringer in Sediment, Upper Massive, Lower massive/breccia and Net-textured, including dilution coming from HW Waste, FW Waste and HW Gabbro Disseminated. All metallurgical composites represent 83% of the known ore resource. Detailed modelling of the metallurgical recoveries by ore type and ore zones has been applied to the mining schedule to determine the overall recoveries used in the financial modelling. These are as follows:

<table><tr><th>Metallurgical Recoveries</th><th>Copper Concentrate</th><th>Nickel Concentrate</th></tr><tr><td>Ni</td><td>1%</td><td>89%</td></tr><tr><td>Cu</td><td>95%</td><td>3%</td></tr><tr><td>Co</td><td>1%</td><td>85%</td></tr></table>Note: Nickel Recoveries are based on Mill Feed Grades.</td></tr>
<tr><td>No deleterious elements were observed in the concentrates, with the exception of chloride from the process water. Concentrate washing has been investigated to determine the required amount.</td></tr>
<tr><td>Copper Concentrate specification – Cu 27-31%, S 29-33%, Fe 29-30%, MgO <1%, SiO_2 <2.5%, As 0.005%, Sb 0.001%, Bi 0.003%, Cd <0.002%, Pb 0.016%, Zn 0.046%, Ni 0.64%, Co 0.02 %, Cl + F <300 ppm, Hg <1 ppm, Al_2O_3 0.56%.

Nickel Concentrate specification – Ni 13-17%, Cu 0.20-0.6%, Co 0.43-0.49%, Au 0.05g/t, Ag 4.8g/t, S 31-34%, Fe 41-44%, MgO <1.5%, SiO_2 <3.0%, As 0.002%, Pb 0.005%, Zn 0.020%, Cl + F</td></tr>
</table>

Criteria	Commentary
	<300 ppm, Al_2O_3 0.9%. The main minerals of chalcopyrite, pentlandite and pyrrhotite can be defined by Cu, Ni, Fe and S grades. The deposit has been modelled with Ni, Cu, Co, Fe, S and MgO for all major material domains.
Environmental	All environmental approvals for the proposed mining activities have been secured. Waste rock and tailings characterisation studies have been completed. Negligible waste rock will be disposed of on surface. Tailings are highly acid-forming and the costs of appropriate impoundments have been allowed. Construction of the Tailings Storage Facility (TSF) has been completed to full Life of Mine capacity.
Infrastructure	The majority of the significant surface infrastructure for the Nova Nickel Project has been constructed, or is currently under construction. The concentrator is planned to be commissioned in late 2016. Decline access to the orebody is well advanced, having commenced in May 2015 and will begin supplying ore to the concentrator in line with the commissioning schedule. The proposed infrastructure lies partly on Fraser Range Station (a pastoral lease administered by Pastoral Lands Board) and unallocated crown land. Some infrastructure (access road, borefield, pipeline) is located on mining tenure held by other companies, and appropriate access agreements have been entered into. It has been modelled that there will be sufficient water available to develop the Nova Nickel Project. Dewatering of a confined aquifer overlaying the ore zone (the Botryoidal Aquifer) is well advanced and this water is being stored in the TSF for use in the initial years of processing. Further exploration for Life of Mine (LOM) water supply is continuing.
Costs	Capital costs used in the production of the Ore Reserve estimate have been gathered from budget pricing or from a cost database. In the case where database costs have been used, contingencies have been applied. Major capital items are based on estimates prepared by experienced independent engineers, including: • Processing Plant – Ausenco • TSF, Access Road, Aerodrome – GHD Australia Pty Ltd • Borefields – MSP Engineering Pty Ltd • Underground (Fixed Plant) – Entech Ltd As firm contracts have been let during the implementation phase, costs have generally being seen to be in line with, or less than those used in the Feasibility Study and the Ore Reserve estimate.
	Operating costs for the underground operation are based on a budget estimate from a leading underground mining contractor. Major operating costs are based on estimates by Sirius and experienced independent engineers, including: • Underground Contract Mining – Barminco • Processing Costs (based on Sirius reagent consumption) – Ausenco A capital and operating cost model has been developed in Excel and has been used to complete a life of mine cash flow estimate.
	Smelter terms have been determined from typical contracts and include: • Nickel payability and TC. • Copper payability (Ni Concentrate) • Copper concentrate copper payability and TC/RC • The presence of deleterious elements has been assessed and it has been determined that no penalties will be applied • Estimates of smelter terms have been determined in-house.
	Product inland transport costs have been estimated by an experienced contractor, Qube Logistics. Shipping costs from the Port of Esperance have been estimated by an experienced shipping broker, Braemar Seascope.
	Royalty allowances are in accordance with Division 5 of the WA Mining Act (Ni and Co = 2.5% of gross FOB metal value in $A; Cu = 5% of nett FOB metal value in A$). In addition a 0.5% of gross Ni FOB value in A$ is payable under Native Title agreements.
Revenue factors	Head grade of the project is dependent on the material scheduled to be mined from underground. Treatment and transportation charges applied in the economic evaluation as outlined previously.
	Revenue has been based on the commodity price and exchange rates commented on above. Metal prices are based on 12 month averages (not volume-weighted) of spot prices from the London Metal Exchange between June 2012 and July 2013 and were provided by Sirius. Prices are as follows:

Criteria	Commentary
	• Nickel – US$7.44/lb • Copper – US3.47/lb • Cobalt – US$12.00/lb Exchange rate - $A 1: $US 0.90
Market assessment	Demand for concentrate has been derived from international metals market analysts – Wood Mackenzie, who prepared a commissioned nickel & copper market study, dated 18 June, 2014.
	Customer and competitor analysis is based on research provided by Wood Mackenzie, plus input from Wood Mackenzie Nickel Industry Cost Service. Also, a commissioned research report into Nickel West furnished by Vector Solutions Pty Ltd, entitled "Value-in-Use Assessment of Nova Concentrate to Nickel West".
	The price and volumes forecast are based on information provided by Wood Mackenzie's Long Term Outlook Reports for Nickel and Copper, June 2014 editions, the commissioned research by Wood Mackenzie, and pricing forecasts by Consensus Economics Inc.
	Potential customers have received and approved representative samples, and received detailed specifications.
Economic	The Ore Reserve estimate is based on a financial model that has been prepared at a "Feasibility Study" level of accuracy. All inputs from underground operations, processing, transportation and sustaining capital as well as contingencies have been scheduled and evaluated to generate a full life-of-mine cost model. Economic inputs have been sourced from suppliers or generated from database information relating to the relevant area of discipline. A discount rate of 8% has been applied.
	The NPV of the project is strongly positive at the assumed commodity prices.
Social	IGO has engaged in discussions with key project stakeholders including: • The Fraser Range Pastoral Lessees, • Southern Hills Pastoral Lessees, • The Esperance Ports Sea and Land; and • Shires of Esperance and Dundas. None has expressed material concerns with the proposed development. Apart from the Fraser Range homestead and caravan park, there are no permanent residences within the Project Area or its environs. All agreements with key stakeholders including traditional owner claimants have either been issued, or are expected to be issued in due course. Those not yet finalised will not affect the Ore Reserve estimate.
Other	Groundwater model simulations indicate that the Nova Project shall have excess water for the first 12 months during construction and development. However, as the aquifers are successfully dewatered and mineral processing commences, the Project is likely to fall into a water deficit scenario. Three additional water supply bores have been identified and these shall need to be drilled, constructed and equipped. Additional bores may be required within a further 2 years, but this can be re-assessed once dewatering and other pumping data becomes available. Although it is not expected, if further groundwater resources are necessary later in the Project's life, there are multiple options for further groundwater resource development within a 50km radius of the Project. These include: • additional discrete fractures that could be identified within the Nova lease • off-lease palaeochannel aquifers • off-lease fractured rock aquifers. A Reverse Osmosis (RO) plant will be required to produce all potable water requirements including concentrate washing. This plant will be designed to treat water quality expected from the borefield. The RO plant will produce the Project's potable water requirement which is then distributed across the site and to the accommodation village.
	Nova and Bollinger are located wholly within Mining Lease M28/376. IGO has a 100% interest in the tenements. The tenements sit within the Ngadju Native Title Determination.
Classification	The Ore Reserve is based on Probable Ore Reserves, no Proved Ore Reserves are reported. Indicated Mineral Resources have been converted to a Probable Ore Reserve. No Measured category Mineral resources have been estimated to date.

Criteria	Commentary
	The Competent Person is satisfied with the classification of the Underground Mineral Resource and hence the conversion to Ore Reserve is appropriate.
Audits or reviews	The Ore Reserve has been peer reviewed internally and is in line with current industry standards.
Discussion of relative accuracy/confidence	The Ore Reserve has been completed to a Definitive Feasibility standard; hence confidence in the resulting figures is high.
	Confidence in the mine design and schedule are high.
	All modifying factors have been applied to designed mining shapes on a global scale as there are limited local data.

APPENDIX B

Tropicana Mineral Resources and Ore Reserves 2016

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	AngloGold Ashanti Australia (AGA) has carried out all the drilling within the Tropicana deposit, with sampling from Reverse Circulation (RC) and diamond drilling predominantly from one metre sample intervals, for 50g gold fire assay. The sampling methodology with RC drilling has changed over time. Sample collection prior to 2007 was via a cyclone, dust collection system and multi-stage riffle splitter attached to the drill rig. From the beginning of 2007 sample collection was via a cyclone, dust collection system and cone splitter attached to the drill rig. RC samples are collected from one metre intervals for resource definition drill holes, with two metre sample intervals from RC pre-collar drilling introduced in 2016. All NQ2 and HQ diamond holes have been half-core sampled over prospective mineralised intervals determined by the geologist, where sample intervals are generally one metre samples. In 2016 two metre samples were processed from un-mineralised core to collect additional geometallurgical data (hyperspectral and XRF) for waste rock characterisation. Within fresh rock, core is oriented for structural/geotechnical logging wherever possible. In oriented core, one half of the core was sampled over one metre intervals and submitted for fire assay. The other half of the core, including the bottom-of-hole orientation line, was retained for geological reference and potential further sampling such as metallurgical test work. In intervals of un-oriented core, the same half of the core has been sampled where possible, by extending a cut line from oriented intervals through into the un-oriented intervals. The lack of a consistent geological reference plane, (such as bedding or a foliation), precludes using geological features to orient the core.
Drilling techniques	Reverse Circulation drilling has been utilised to an average depth of 150m in the shallower, up-dip, western portions of the Resource and as pre-collars to diamond holes. All Reverse Circulation drilling has been via face sampling hammer. Diamond drilling has predominantly been NQ2 with limited HQ2, HQ3 and PQ in the upper saprolite and for holes drilled for geotechnical and metallurgical purposes. The majority of diamond holes have been drilled as tails to RC drilling. From 2011 many deeper holes were drilled with shorter RC pre-collars (~60m), or HQ from surface to minimise deviation.
Drill sample recovery	The sample recovery is currently recorded on selected intervals to assess that the sample is being adequately recovered during RC drilling. Prior to April 2008, no systematic assessment of sample recovery data was made for RC drilling. A subjective visual estimate was used where weights were recorded as 25, 50, 75 or 100%. Since April 2008 a systematic sample recovery program has been implemented where for 1:25 intervals, the Primary (lab weight), Secondary (archive weight) and Reject splits are weighed and recorded in the database. These weights are combined and then compared to a theoretical recovery of the interval based on the regolith and rock type of the interval being analysed. For diamond drilling recovered core for each drill run is recorded and measured against the expected core from that run. Core recovery is consistently very high, with minor loss occurring in regolith and heavily fractured ground.
Logging	All RC chips and diamond drill cores have been geologically logged for lithology, regolith, mineralisation and alteration utilising AGA's standard logging code library. RC Sample quality data recorded includes recovery, sample moisture (i.e. whether dry, moist wet or water injected) and sampling methodology. Diamond core has also been logged for geological structure and geotechnical properties. Diamond drill holes are routinely orientated, photographed and structurally logged with the confidence in the orientation recorded. Geotechnical data recorded includes QSI, RQD, matrix, and fracture categorisation. Bulk density determinations have been routinely collected for diamond drill core over one to five meter intervals using water immersion methods. A coherent segment of core (>10cm length), representative of the meter interval is selected. Laboratory bulk density determination is completed on selected 'core from surface' diamond holes to collect bulk density data for oxide and transitional rock types, and from fresh rock types to ensure water immersion methods used onsite is accurate.

Criteria	Commentary
	All logging data are digitally captured via Field Marshall Software (upgraded to Micromine Geobank platform 2016) and the data are validated in Vulcan prior to being uploaded to an SQL database. DataShed has been utilised for the majority of the data management of the SQL database. The SQL database utilises referential integrity to ensure data in different tables are consistent and restricted to defined logging codes.
Sub-sampling techniques and sample preparation	Since the commencement of exploration activities at Tropicana, sample preparation and analysis has been carried out by three laboratories, as detailed below: Prior to November 2006 - SGS (formerly Analabs) Welshpool performed all gold and multi-element analysis. SGS routinely prepared half-core diamond samples by crushing in a jaw crusher followed by pulping in an LM5 to 90% passing 75µm. One metre RC samples are pulped in an LM5 to 90% passing 75µm. 50-gram samples are then assayed by fire assay. Sieve tests are carried out on 5% of samples. November 2006 to 2014 – Genalysis Perth has performed all gold and multi-element analyses and hyperspectral scans. The 2015 Boston Shaker infill drilling was analysed at the Tropicana onsite lab, with sample preparation conducted by AGAA staff operating an automated circuit, and SGS conducting the fire assay and analysis. June 2016 to current, infill drilling has been analysed at the Tropicana onsite lab, with sample preparation conducted by AGAA staff operating an automated circuit, and SGS conducting the fire assay and analysis. At Genalysis, half core samples weighing approximately 2.5kg are prepared via a robot. The samples are then crushed to <3mm in a Boyd crusher and automatically split, down to a sample of ~1kg for pulping and analysis. The remainder of the material was retained as a coarse split for metallurgical test-work. One metre RC samples were pulped in a mixer mill to 90% passing 75µm. Wet sieve tests were carried out on 5% of the samples. The Tropicana laboratory uses a linear automated process to prepare the samples. Samples, from RC and diamond drilling, are loaded onto racks at the lab. Each sample bag has a unique bar-code attached to the bag. Samples are dried and weighed. Small samples (<800g) are manually pulverised in an LM2 mill to 90% passing 75µm. Acceptable weight samples (>800g) are loaded into tubs and the samples passed under a Terraspec Hyperspectral camera. Samples are then passed through a Boyd crusher, reducing the particle size to 90% passing 2mm before being split via a Linear Sample Divider. Coarse duplicates are assayed at a rate on 1 in 20 within the assaying of the batch. Primary samples then get pulverised to 90% passing 75µm and the resultant product split into a 50g sample or fire assay and a 500g sample. The 500g sample passes under a portable XRF scanner for analysis of secondary elements (that are not used in the Mineral Resource estimate). The 500g sample is retained for check assay work. Routinely, coarse blank samples are inserted as the first sample in each laboratory job. The purpose of this sample is to check that laboratory crushing and grinding equipment is kept clean. Coarse blanks samples are also inserted into the sequence of samples before each zone of mineralisation. Standards are inserted into batches of samples at a frequency of three standards in every 100. Sieve tests are carried out on 5% of samples. Coarse duplicates are completed on 20% of the samples.
Quality of assay data and laboratory tests	At SGS 50-gram samples were assayed by fire assay. SGS inserted blanks and standards (one in 20 samples) in every batch. Every 20th sample was selected as a duplicate from the original pulp packet and then analysed. Repeat assays were completed at a frequency of one in 20 and were selected at random throughout the batch. In addition, further repeat assays were selected at random by the quality control officer, the frequency of which was batch dependent. Analysis was by fire assay with similar quality assurance (QA) for RC and half core samples. Genalysis inserted internal standards and blanks randomly through each batch. Every 25th sample was selected as a duplicate from the original pulp packet and then analysed at the end of the batch. Finally, 6% of the batch was selected for re-analysis. Internal laboratory checks and internal and external check assays such as repeats and check assays enable assessment of precision. Contamination between samples is checked for by the use of blank samples. Assessment of accuracy is carried out by the use of certified Standards (CRM). Check assay campaigns generally coincide with each Resource update. QAQC results are reviewed on a batch-by-batch and monthly basis. Any deviations from acceptable precision or indications of bias are acted on with repeat and check assays. Overall performance of both laboratories has been satisfactory.

Criteria	Commentary
Verification of sampling and assaying	On receipt of assay results from the laboratory the results are verified by the Data Manager and by geologists who compare results with geological logging. Analysis of twinned drill holes showed that no significant down-hole smearing was occurring in RC holes when compared to the twinned diamond holes in Tropicana and Havana.
Location of data points	All hole locations within the resource area to date have been pegged with a standard GPS, or by RTK GPS. Once the holes are drilled the collar location is then surveyed with an RTK GPS. A regional Digital Terrain Model was then created to cover the Tropicana JV tenement area from Shuttle Radar Topography Mission (SRTM) data. The data were sampled at 3 arc-seconds, which is 1/1200th of a degree of latitude and longitude, or about 90 metres. Eastman single shot instruments were used routinely for down-hole surveys prior to 2007. From 2007, gyro surveying instruments have been used to complete downhole surveying.
Data spacing and distribution	Drill hole spacing on sections, and between sections, typically range from 25 x 25m to 100 x 100m. The majority of the Open Pit Resource area has been drill tested at a nominal density of 50 x 50m with the spacing closed up to 25m x 25m within the upper levels of the deposit. The down-plunge extension of the Havana Deeps area is drilled at 100m x 100m or 100 x 50m closer to the pit area. 1m samples are composited to 2m prior to Resource Estimation.
Orientation of data in relation to geological structure	The majority of drilling is orientated to intersect normal to mineralisation. The chance of bias introduced by sample orientation is thus considered minimal.
Sample security	Samples are sealed in calico bags, which are in turn placed in large poly-weave bulka-bags for transport. Filled poly-weave bulk-bags are secured on wooden crates 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 AGA 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, the pulp packets are returned to the AGA warehouse on secure pallets where they are documented for long term storage and retrieval.
Audits or reviews	Field quality control and assurance has been assessed on a daily, monthly and quarterly basis. Field QA/QC was assessed by Quantitative Group (QG) as part of their audits of the Tropicana and Havana Resource between 2007 and 2009.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	Tropicana is a joint venture between AngloGold Ashanti Australia Limited (AGA) and Independence Group NL (IGO) (AGA:IGO, 70:30) AGA is the manager of the JV.
	There is no known heritage or environmental impediments over the leases where significant results were received. The tenure is secure at the time of reporting. No known impediments exist to operate in the area.
Exploration done by other parties	AGA has carried out all the drilling within the Tropicana deposit.
Geology	The Tropicana and Havana gold deposit host rocks are predominantly gneisses.
Drill hole Information	No new exploration data are announced within this report.
Data aggregation methods	No new exploration data are announced within this report.
Relationship between mineralisation widths and intercept lengths	No new exploration data are announced within this report.
Diagrams	No new exploration data are announced within this report.
Balanced reporting	No new exploration data are announced within this report.
Other substantive exploration data	No new exploration data are announced within this report.

Criteria	Commentary
Further work	No new exploration data are announced within this report.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	AGA uses various software programs to collect the different forms of drilling data obtained during exploration. The main packages are from Microsoft (SQL Server and Access) an, Maptek Pty Ltd. (Vulcan), Micromine Pty Limited (Micromine, Field Marshall and Geobank), Aranz Geo Limited (Leapfrog), Maxwell Services Limited (DataShed) and Karjeni Pty Limited (dPipe). The database is managed with Microsoft's SQL Server and Maxwell's DataShed. DataShed was developed as a front end interface to MS Access or SQL Server. DataShed was specifically created for the exploration and mining community and contains queries and data management utilities unique to the mining industry. Many of these or additional processes have been modified or added to by AGA. Drilling data are captured in the field directly into handheld, for example Husky, LXE, Toughbook or laptop computers with Field Marshall and Geobank software. Daily drilling forms (Plods) are completed by the driller in hard copy and signed off by the geologist and entered into DataShed. ampling, bulk densities, Hardness and Magnetic Susceptibility (MagSus) readings are also recorded digitally into handheld devices. Up to end of 2015, the merging of logging data into the database was semi- automated via a file transfer program called dPipe. Karjeni Pty Limited developed dPipe to facilitate the transfer of data from one format into another into SQL databases. This program has the ability to read a file to split, composite and append data into the desired format. From 2016, logging data is synchronised from Geobank directly into Datashed, and field data, such as RTK collar coordinates and downhole surveys are loaded via DataShed importers. Assay results received from the laboratories are emailed to geologists and stored on the server. An invoice is mailed to AGA along with a hard copy or digital PDFs of the results. The hard copies are filed in folders and PDFs stored on the network for future auditing purposes. Assay data files are loaded via DataShed importers, and loading procedures include QA/QC checks to ensure standards and blanks have returned acceptable results. Rigorous data validation procedures are in place to identify data issues.
Site visits	Mining activities are ongoing and the site is visited regularly by the Competent Persons.
Geological interpretation	3D solids are created or mineralised zones, dykes, shears and garnet gneiss using Leapfrog. The mineralised domains are created by flagging intervals at a 0.3g/t gold rade cut-off with internal lower grades included in the model. The Dykes, Shears an Garnet Gneiss units are selected by flagging intervals based on logged lithology, as they are the most visually distinctive units, are the least subjective when being logged and therefore are considered to have a high level of confidence in interpretation. The dykes are locally important as they post-date mineralisation and are generally barren of mineralisation. Modelling of the shears is critical to understanding geotechnical aspects and assessing the spatial controls on the mineralisation. The Garnet Gneiss units are important because they are generally found in the hanging wall and as a precursor to mineralisation, as well as being the dominant waste rock unit.
Dimensions	The Open Pit Mineral Resource is reported within an A$1817 optimisation shell that is 4.7km long, up to 1km wide, and up to 450m deep. The Underground Mineral Resource extends to a depth of approximately 1km below surface.

Criteria	Commentary
Estimation and modelling techniques	The Mineral Resource is reported from open pit and underground Mineral Resource models, estimated with differing estimation techniques and with different cut-off grades applied to each model. The Open Pit Mineral Resources have been estimated using the geostatistical technique of Localised Uniform Conditioning using average drill hole intercepts and is reported above a marginal (break-even) cut-off grade of 0.3g/t for oxide material and 0.4g/t for transitional and fresh material. The Havana Deeps Underground Mineral Resource has been estimated at a cut-off grade of 2.0g/t using the geostatistical technique of Ordinary Kriging using average drill hole intercepts. The Underground cut-off grade calculation is based on an underground Pre-Feasibility study completed in late 2013, and a gold price of US$1400 (A$1704). 2m down-hole composites are used for both estimates. Gold is the only element modelled, as no other significant element has been detected in sampling to date which would be deleterious to mine and mill performance. The Open Pit estimate uses block sizes of 15m (X) by 30m (Y) by 10m (Z) with an SMU of 5m (X) by 7.5m (Y) by 3.33m (Z). The Underground estimate uses a block size of 10m (X) by 10m (Y) by 2m (Z), with blocks dipping 30° to the (grid) east, parallel to the majority dip of the orebody, with the resulting estimate filtered to remove isolated blocks that cannot be mined individually. Both Resource Estimates are compared to the input data using swath plots to check for bias in the estimation, no bias was noted in the plots. Mining has been ongoing since 2012 and reconciliations to date indicate that the Mineral Resource model has reconciled well with grade control.
Moisture	Tonnage estimates are on a dry tonne basis.
Cut-off parameters	The Open Pit Mineral Resources use a cut-off grade of 0.3g/t for oxide material and 0.4g/t for transitional and fresh material, based on contract mining costs, budgeted processing and administration costs, and a gold price of US$1400 (A$1817). The Underground Mineral Resource has been estimated at a cut-off grade of 2.0g/t. The cut-off grade calculation is based on an underground Pre-Feasibility study completed in late 2013, and a gold price of US$1400 (A$1704).
Mining factors or assumptions	Open Pit mining assumes selectivity of SMU's of 5m (X) by 7.5m (Y) by 3.33m (Z) for Havana and Tropicana. Boston Shaker uses an SMU of 5m (X) by 7.5m (Y) by 2.5m (Z). No external dilution accounted for in the Mineral Resource. Underground mining is based on a modified Long-Hole Open Stope method, with 20m vertical intervals between ore drives. The Mineral Resource is filtered based on the average grade of surrounding blocks to remove isolated blocks from the Mineral Resource total. No external dilution is included in the Mineral Resource Estimate.
Metallurgical factors or assumptions	Metallurgical recovery is taken into account in the optimisation of both Open Pit and Underground Resource optimisations, with an average project recovery of 90.3% assumed, based on extensive metallurgical test work completed as part of the Feasibility Study for the Havana Open Pit.
Environmental factors or assumptions	Tropicana Gold Mine (TGM) operates under an environmental management plan that meets or exceeds all environmental and legislative requirements. TGM holds the license to operate and it is valid for the life of the Ore Reserve. Environmental rehabilitation plans are produced and cost of the rehabilitation work is accounted for in the financial evaluation model.
Bulk density	Dry Bulk Density (DBD) determinations have been routinely collected on the mineralised zones in all DDH core at one-metre intervals using water immersion methods. A coherent segment of core (>10cm length), representative of the metre interval, is selected. The weight is measured dry, in air, then measured submerged in water. Core was left to dry naturally on the core racks. Dry Bulk Density has been estimated using Ordinary Block Kriging, with areas with insufficient data to generate a kriged estimate being assigned the average measured value for that lithology and regolith type. Density values within units show little variation.
Classification	The estimates of the Mineral Resources presented in this Report have been carried out in accordance with the principles and guidelines of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012). Mineral Resources have been classified based on the 15% rule whereby a Measured Resource should reconcile within plus or minus 15% over quarterly production volumes, 90% of the time, and an Indicated Resource should reconcile within plus or minus 15% over yearly volumes, 90% of the time, as per internal AGA guidelines. This criterion defines a drill spacing of approximately 25 x 25 m to define a Measured Resource, and 50m x 50 m to define an Indicated Resource. Inferred

Criteria	Commentary
	Resources are defined when evidence of geological and grade continuity exists sufficient to generate an estimated grade. The average data spacing for Inferred Resources varies, but is generally 100m x 100m or less. The Resource classification is consistent between the Open Pit and Underground estimates, given that the underground mining will focus on large tonnage, low cost methods and the Resource is mined at a relatively low cut-off grade. Material defined by relatively few drill-holes was manually recoded out of Resource classifications, and not reported as part of the Tropicana Mineral Resource.
Audits or reviews	The Open Pit Mineral Resource has been audited previously as part of the BFS by Quantitative Group (QG) between 2007 and 2009. An additional external review of the Mineral Resource was also completed in 2011. Golder Associates audited the 2015 Mineral Resource estimate, and supported the estimate with some recommendations which have been adopted for the current update.
Discussion of relative accuracy/confidence	The relative accuracy of the Mineral Resource Estimates is reflected in the Resource Classification. A trial grade control pattern of ~100m by 100m was drilled during the BFS which provided confidence that the Mineral Resource Estimate was accurate in that volume. Reconciliations of the Resource model to date indicate no significant flaws in the grade estimate, with some additional lower grade material being mined than was predicted from the Resource.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	All Ore Reserves estimated for Tropicana Gold Mine are based on the Mineral Resource model. No Ore Reserve exists outside of the Mineral Resource base.
	Mineral Resources are reported inclusive of Ore Reserves.
Site visits	The Competent Person is based on site and it is part of his daily activities to inspect the mining areas.
Study status	A Feasibility Study was completed in 2010, which determined a technically achievable and financially economic mine plan. The Ore Reserves are designed based on the current operational practices of the mine. All Ore Reserves are estimated by reporting physicals (volumes, tonnes, grades, material types, etc.) against the Resource model within detailed staged pit designs. Ore Reserve physicals are then put through a financial model for economic evaluation. Performance of the on-going mining activities has demonstrated that current mine plans are technically achievable and economically viable considering the material modifying factors.
Cut-off parameters	The cut-off grades are determined based on the net return from the gold produced at the processing plant for each material type. Only the ore that has a grade above the cut-off grades are included in the Ore Reserves.
Mining factors or assumptions	The Ore Reserves are reported within detailed operational designs that are developed based on the geological Resource model, geotechnical studies and financial information.
	Open pit mining method is based on using excavator and truck fleet system. The staged pit designs used for Ore Reserves are generated as three dimensional designs considering operational requirements such as equipment access. Mining operations at Tropicana Gold Mine started in July 2012 and the operation has proven that the designs and plans are technically achievable; no issue preventing access or pre-strip is experienced or envisaged for the Ore Reserves.
	Overall pit slope angles for oxide and fresh rock types are assumed to be 36° and 60°, respectively. External and internal Geotechnical studies carried out to evaluate the operational designs have confirmed that the pit designs do not violate the geotechnical guidelines developed during Feasibility study. Grade control drilling is completed prior to ore mining on a 12 x 12m pattern using reverse circulation drill rigs.
	The Mineral Resource model used to develop the Ore Reserves uses blocks in 15m x 30m horizontal dimensions and 10m (7.5m for Boston Shaker) vertical bench height that are mined in 3 flitches (3.3m in average height and Boston Shaker 2.5m in average height), with a minimum SMU 5m x 7.5m x 3.33m (Boston Shaker 5m x 7.5m x 2.5m). The grades within the resource model have been diluted to reflect the average grade of this mineable block size. Therefore, no other mining dilution is applied.
	Mining recovery factor used is 1.0.
	In the designs, a minimum of 50m width is implemented for a pit base or some location with only one bench height, where it is technically possible to access. In the design work, a minimum of 80m

Criteria	Commentary
	mining width is implemented as a generic rule.
	Inferred material is excluded from the Ore Reserves and treated as waste material, which incurs a mining cost but is not processed and hence does not generate any revenue. The total quantity of the Inferred material is less than 0.3% the Ore Reserve. Hence the reported Ore Reserve's financial outcome is not sensitive to the Inferred material within the pit designs.
	There is no infrastructure to be completed.
Metallurgical factors or assumptions	The metallurgical process, which was proposed and is currently in operation, was developed through a comprehensive series of test programs at Scoping, Pre-Feasibility and Feasibility study levels. Test work was mostly at batch scale but, where considered advisable, at pilot and demonstration plant scale.
	The majority of the process uses highly mature technology. The sole exception is the use of High Pressure Grinding Rolls to prepare ball mill feed. The equipment used for this technology itself dates back over twenty years, and is mature. Developments for the hard rock industry are more recent, but have now been successfully used in a number of plants worldwide and this is the part of the process that was extensively tested in a range of machines from pilot up to demonstration scale.
	Metallurgical test work consisted of comprehensive testing of a number of composite samples to develop the process design basis, and supplementary testing of a much larger number of samples to establish variability. These variability samples were taken on a grid pattern to ensure even coverage of the entire deposit. No metallurgical domains have been recognised to date other than by regolith type and some minor variation in one northern section of the deposit.
	The ore is exceptionally free of deleterious elements and base metals. No allowances have been made or are considered necessary.
	Pilot scale test work utilised PQ diameter core. Whilst only a relatively small number of PQ holes were drilled, their position was selected based on the prior variability test work to provide samples considered to be adequately representative of the orebody as a whole. The samples were also characterised by standard batch scale and geometallurgical style tests so that results could be related to the wider orebody.
	As a gold mine, the product is not defined by specification. No problems are envisaged, or have been encountered, in producing gold bars of saleable quality.
Environmental	Tropicana Gold Mine (TGM) operates under an environmental management plan that meets or exceeds all environmental and legislative requirements. TGM holds the license to operate and it is valid for the life of the Ore Reserve. Environmental rehabilitation plans are produced and cost of the rehabilitation work is accounted for in the financial evaluation model.
Infrastructure	Adequate infrastructure has been completed and sustaining cost of the infrastructure (maintenance and replacement) is accounted for in the financial model.
Costs	Capital costs of removing waste over ore are included in the evaluations for the applicable pits.
	Mining operating costs are provided by the contractor Macmahon as rates (from an annual rate review conducted between AGAA and Macmahon). Processing operating costs have been derived from variety of sources including first principle estimates, metallurgical test work results, budget quotations for consumables and vendors, consultant advice on wear rates/component replacement frequency, baseline input parameters such as exchange rates, power cost, labour numbers etc., AGA Australia Ltd advice, Lycopodium and sub-consultants data and experienced based on similar sized operations.
	No allowances have been made or are considered necessary for the content of deleterious elements.
	Transportation cost for the produced gold doré is relatively small and charged on a contract base with the refinery.
	The source of the treatment and refinery charges is the contract with refinery and there is no specification and no penalty is considered for not meeting specifications.
	Total royalty cost allowance is 2.5% of the total revenue.
Revenue factors	The assumption made for the gold price is US$1,100/oz, A$1,436/oz and the exchange rate is US$0.77 per Au$1.0.
	The assumptions are derived after reviewing historic commodity prices and exchange rates.
Market assessment	Long term market assessments are provided by a number of independent companies. AGA does not provide advice or endorsement for using a specific forecasting company.

Criteria	Commentary
Economic	Tropicana Gold Mine (TGM) is now operating with mining costs based on contractor mining rates. Processing costs have been derived via comprehensive test work and studies. TGM is therefore not highly exposed to uncertainty in, or to inaccuracy in estimation of, mining or processing costs. The inflation rates assumed are based on prior AGAA Treasury guidance provided, whilst discount rate utilised at AGA is derived from the weighted average cost of capital for Australia.
	Sensitivity studies are carried out on various parameters including mining cost, processing cost, gold price and discount rate. Gold price is the most sensitive input for NPV and a 10% reduction would eliminate about 30,000 ounces (~0.80%) from the Reserves.
Social	Tenement status is in good standing.
Other	There is no foreseeable TGM specific risk. There are typical risks of an open pit mining operation such as heavy rain events and geotechnical risks. These risks are managed through implementation of various risk management mechanisms as much as practical.
Classification	Exploration drill-hole spacing is the basis of the classification. Proved material is defined for the areas drilled with 25m spacing and Probable is defined on 50m drill spacing.
	The methodology of classification is appropriate for the deposit.
	Proportion of the Proved Ore Reserves is a sub-set of Measured Mineral Resources. Probable Ore Reserves are derived from Indicated Mineral Resources.
Audits or reviews	A Mineral Resource and Ore Reserve audit was completed in 2011. No unexpected results came from the audit.
Discussion of relative accuracy/confidence	As part of the Ore Reserve estimation process, a review is performed for the actual reconciled extraction against previous year's Reserve estimation.
	Reconciliation of the Ore Reserves to actual mined during the 2014 year showed that Ore Reserve estimation is slightly conservative.

APPENDIX C

Long Operation Mineral Resources and Ore Reserves 2016

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 electromagnetic (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 was 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 3kg. 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 the 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 contract drillers. HQ 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. They are all located in very competent fresh material so any loss of fine material would be negligible.
Logging	Geotechnical logging was carried out on all recent diamond drill holes for recovery, RQD, and number of fractures (per interval). The information is captured in the main drill hole 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 drill hole 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. All core samples were collected from the same side of the core. Extremely broken core is sampled by visually selecting 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 is 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

Criteria	Commentary
	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 75µm. 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 1in 10 blank samples and 1 in 20 standard samples. The performance of the blank returned 97% of results within acceptable limits as opposed to 88% reported for the last estimation period. Results of standards and blanks from each batch are scrutinised at the time they are received, and compared with expected values. Variation outside two standard deviations from the expected result is reported to the lab for checking, and re-assaying if required. In-house QAQC reports are produced quarterly and annually to examine variability in standard and blanks performance and reliability.
Quality of assay data and laboratory tests	The analytical techniques used a 400mg sub sample digested in mixed four acid digest (Nitric, Perchloric, Hydrochloric 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 removed from the hot plate and Hydrochloric Acid is added. The beaker is 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 again removed from the hotplate and allowed to cool. De-iodised water is added to the beaker to bring the volume of the solution up to a standard 18ml and the solution is 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 or XRF tool was used to determine element concentrations used in the Resource estimate. Sample preparation checks for grain size were carried out by the contract laboratory as part of its internal checks to ensure the grind size of 90% passing 75µm. Greater than 90% of all sizing tests were within acceptable limits. Blank samples returned results of 88% of the samples within acceptable limits. Work is ongoing with the current laboratory to reduce contamination through the crushing and pulverising stages. Diamond core samples are taken for field duplicates and submitted to the laboratory as separate batches with overall 22% returned results outside acceptable limits. 52% of the field duplicates were from diamond quarter core samples. These returned with 53% results outside acceptable limits. The remaining 48% field duplicates were from diamond half core samples. These returned with 14% results outside acceptable limits. The majority of the results reported outside acceptable limits were below 1% Ni, where the lab accuracy is decreased. This is considered to be due to heterogeneous core samples and accuracy of analysis decreasing as the value approaches the detection limit. The half core, sampled at 0.1m to 1.1m intervals is 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. 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 laptop computers and acQuire database logging templates. The

Criteria	Commentary
	data were transferred into main drill database (acQuire Database version 4.5.0.1) with SQL2008 database server backend. There was no adjustment to assay data. Assay results are received 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 main drill hole database through a database importer protocol.
Location of data points	Planned drill collars for underground diamond drill holes are laid out by marking 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 dip and azimuth set up is carried out by the drill contractor using "Azimuth Aligner" North Seeking Gyro with drill rig alignment accuracy of < 0°Lat RMS True North, and dip accuracy of <0.01°. The collar position is later surveyed by the Company Surveyor using a Viva TS15 Total Station Theodolite, locating the exact position of the drill hole collar. The collar coordinates are stored in the main drill hole database. Historical downhole surveys were completed using Eastman and Reflex cameras 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 trace. No gyroscopic validation of down hole surveys was undertaken in the drilling from January 2014 to December 2014, 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 Resource is estimated in Local Grid (KNO-Grid). It is a non-linear projection of MGA_GDA94, Zone52 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 main drill hole 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 are on a nominal 20m northing with 10m easting drill spacing with 5m by 5m closer-spaced drilling. Moran is on a nominal 20m northing with 10m easting drill spacing with some up to 10m by 10m closer-spaced drilling. The data spacing and distribution are considered to be sufficient to establish the degree of geological and grade continuity to support the Mineral Resource estimation and classification applied.
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. Underground diamond drill holes collars are generally fanned off sections but kept to near true width as much as possible. In Long, Victor South and McLeay ore bodies 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 hanging wall geometry only and the assay results were not used for 2D Resource estimation for the Long orebody as they do not represent the entire mineralised width. Some grade control drill holes (holes drilled within the ore bodies and within the ore drives) were used to determine footwall or hanging wall geometry and the assay results were used in the 3D Resource estimation for Victor South, McLeay and Moran ore bodies.
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 reporting of results. Half core is kept for reference and 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 main drill hole database.
Audits or reviews	The sampling data are collected and managed by IGO geologists familiar with the local rock-types and data collection process established by 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 geologist and is considered to be of sufficient quality to carry out Resource

Criteria	Commentary
	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 an "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 leases and no other known impediments.
Exploration done by other parties	Exploration was initially undertaken by WMC who eventually commissioned the Long Shaft and Victor decline mine development. The data are of high quality with most of the historic drilling 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 intrusion by multiple felsic to intermediate igneous dykes and sills.
Drill hole Information	Drill hole data have been collected from this area since 1978 with over 2,800 drill holes completed. Reproduction of this number of drill holes, the majority of which have been mined out, is not feasible for this report. Material drill holes have been reported to the ASX in previous public releases.
Data aggregation methods	Exploration results are calculated as the length and density-weighted average to 1% nickel cut-off grade. Maximum internal waste of 2m may be included however the total nickel composite average grade must be >1% nickel. Intercepts are length and 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	No new exploration data are announced within this report.
Balanced reporting	No new exploration data are announced within this report.
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 will test for potential new ore channel positions.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Primary data were collected using laptop computers and acQuire drill hole database logging templates. The data were transferred into acQuire Database version 4.5.0.1 with SQL2008

Criteria	Commentary
	database server backend. All validation is completed by IGO 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 lab 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 carried out.
Site visits	The Competent Person employed by IGO. 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 estimation process.
Geological interpretation	Geological interpretation has a high to moderate confidence as up/down dip and plunge continuity is well established. Barren porphyry dykes are irregularly spaced and orientated so geological interpretation is considered to be moderate in 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 intrusion and faults. The mineralised komatiite volcanic flows continue past the off-sets.
Dimensions	Long deposit consists of 26 mineralised surfaces and is approximately 2.2km down plunge, 3m thick and 500m down dip in extent. The surfaces are narrow and ribbon-like accumulates of massive and semi-massive sulphides and start 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. Moran deposit consists of 3 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 modelling techniques	Surpac v6.3 and v6.6 modelling software was used for the variography and block modelling. Wireframes for all mineralised domains are interpreted as strings in section or plan orientation to honour geological contacts and to 1% nickel cut-off grade. Victor South disseminated zone cut-off grade was 0.6% nickel. The string interpretations are used to generate digital terrain wireframes (DTMs) which are finally validated using Surpac 3D modelling software and set as solids. The Long ore body was estimated using a 2D metal accumulation. 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. Therefore the 2D longitudinal modelling approach is based on an accumulation variable incorporating mineralised domain horizontal width, intercept grade and density. Each sample within a mineralised domain was assigned a unique code. This coding was used to control compositing and mineralised domain grades were composited across the entire coded interval resulting in a single intercept composite. The 2D modelling approach uses Ordinary Kriging to estimate accumulation and horizontal width variables and the final grade is back calculated. The Long 2D block estimates were based on a parent block size of 10mYx8mX in the longitudinal plane. The minimum number of samples used in the estimate was 6 and the maximum was 24. The search ellipse radius used was 200m. The final Long 3D block model parent cell size was 10mYx4mXx8mZ sub-celling to 1.25mYx0.25mXx0.25mZ. The 2D grade variables were imported into the 3D "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. Victor South, McLeay and Moran ore bodies were estimated using Ordinary Kriging of 1m downhole composites. During estimation, a local rotation was applied to both the variogram model and search ellipsoid. The orientation of this local rotation was controlled by the trend of individual DTM surfaces modelled to reflect the general trend of each domain. The rotations were interpolated into the

Criteria	Commentary
	volume intermediate to and beyond the controlling surfaces for use in the grade interpolation. The local rotations were used to orient both the variogram model and search neighbourhood. Domain surfaces which were believed to be similar in nature in regards to mineralisation controls were combined for the purpose of grade estimation. The McLeay, Victor South and Moran block 3D models parent cell size was 10mYx4mXx4mZ sub-celling to 5mYx0.5mXx0.5mZm. The minimum number of samples used in the estimate was 4 and the maximum was 12. The search ellipse radius used was between 60 and 100m. 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 20m northing with 10m easting drill spacing with some up to 10m by 10m closer-spaced drilling. Block sizes are considered appropriate for the data spacing and search employed. Porphyry wireframes were generated as 3D solid models (3DMs). The porphyry wireframes were used to flag a porphyry code of 999 into the 3D block model. Block nickel grade (ni) was reset to 0.01% Ni and density (density) was reset to 2.7 g/cm^3 within the flagged blocks. Fields ni_orig and density_orig retain the original estimated values prior to porphyry flagging and resetting. 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. 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 all have measured bulk density values. Top-cutting was not applied to the nickel assays as the data do not present any apparent outliers. Densities were checked against density vs grade regression curves and 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 as SWATH plots for comparing sectional divisions along the length of each ore body. Monthly mine reconciliation is completed and updated each time the Resource estimate is updated. Other than for comparative purposes, the reconciliation data are not used in the block model.
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 in fresh rock. The ore treatment processes are undertaken using the BHP Billiton (BHP_B) nickel concentrator located within 5km of the mine. This process plant has been in use for over 30 years and is appropriate for nickel ore sourced from this area.
Environmental factors or assumptions	Waste is trucked to the surface or used for backfill 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 a regression curve formula.

Criteria	Commentary
	This water immersion technique accounts for vugs and porosity as it 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 mineralisation zones so bulk density values will not cross different rock types and mineralisation styles. Bulk density is estimated into the block model using ordinary kriging. Porphyry intrusions are assigned a bulk density value of 2.7g/cm^3 as this represents the average value of porphyry bulk density measurements.
Classification	Mineralisation classification is conducted 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 20mY x 10mX grid for all Long, Victor South and McLeay deposits and 40mY x10mX for Moran. Mineralisation modelled with a drilling density up to 40m x 40m is classified as Inferred Resource provided there is a reasonable assumption of grade continuity. The classification scheme takes into account all of the relevant factors when assigning the Resource category. This result appropriately reflects the Competent Person's view of the deposit.
Audits or reviews	No review of the previous Resource estimate was conducted as all only depletion was calculated in 2016. The variography and estimation parameters used in the estimation for all the deposits have been generated and validated by Cube Consulting prior to estimation.
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. Confidence in the Mineral Resource estimate is moderate to high in the mine development areas and/or drilling with a 20m x 20m pattern or greater. Confidence is moderate to poor in areas with broader drill spacing. Reconciliation with production data is completed monthly and updated each time a Mineral Resource estimate is completed.

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 Operation are a sub-set of the Long Operation Mineral Resources. No Reserves exist outside of the Mineral Resource base.
	Mineral Resources are inclusive of Ore Reserves.
Site visits	Rob Dennis as the General Manager of Operations frequently visits the operation and inspects working areas within the mine.
Study status	The Long Operation has a history of being mined by Lightning Nickel since October 2002. The mine Reserves have been designed based on the current operational practices of the mine. All Ore Reserves are estimated by constructing three dimensional mine designs and reporting 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 compare 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 where appropriate.

Criteria	Commentary
	In certain cases where a mined stope contains both Indicated and Inferred Mineral Resources, the stope was only designed around the Indicated Resources, however in some cases 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	Independence Group 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	Independence Group operates under an environmental management plan, which meets or exceeds all environmental legislative requirements. Independence Group's license to operate is in good standing. Environmental rehabilitation plans are constructed and progressively acted upon. The costing of the rehabilitation works is accounted for in the operations Life of Mine model.
Infrastructure	The current infrastructure at the Long Operation 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. 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 Nickel 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$11,766 per tonne, Copper price US$ 5,173 per tonne and Foreign exchange rate of AUD$1.00 : US$0.74.
	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 from a small proportion of Inferred Mineral Resources.

Criteria	Commentary
Audits or reviews	An independent audit is periodically undertaken on both the Mineral Resource and Reserve process. No external audit was undertaken this year.
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.

APPENDIX D

Jaguar Operation (Bentley) Mineral Resource and Ore Reserve 2016

JORC Code, 2012 Edition – Table 1

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 ½ core samples were minimum length 0.3m and maximum length 1m. BQ core was submitted as whole core samples. Only 11 holes were drilled BQ. Core was cut with an automated core cutter after orientation and mark-up. Drillhole spacing is described in the sub-section "*Data spacing and distribution".*
	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. Analytical techniques are described in the sub-section "*Quality of assay data and laboratory tests".*
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 (now First Drilling) and holes were NQ2 core size. In 2013 BQ core size was tested but later discontinued. 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 is measured and marked up on angle iron in continuous runs. 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. Core samples and core duplicate masses are compared to monitor core cutting bias. Action is taken to stop bias when bias is noted.
	Most core is competent and cuts well with minimal loss of fines. No sample bias from core drilling or core recovery is suspected.
Logging	Core was photographed both dry and wet and copies of the digital images stored on the site server. All core holes are logged via laptop into an acquire database. 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 >30m 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 sub-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 for the majority of samples. These techniques are appropriate for base metals samples.

Criteria	Commentary
	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 are used to measure and monitor the fraction of pulp passing the 75 micron threshold. Samples through mineralised zones that fail the sieve test, plus samples either side of the failed test, are recombined with residues and pulverised again.
	Field duplicates in the form of second half- sampling are inserted at a rate of 2 per 100 samples or better in the underground drilling. The sampling is representative of the material drilled. Precision is lower this year, with 72% of the field duplicate samples in the 2014-2015 drilling being within +/-20% relative difference for Zn, 77% for Ag and 66% for Cu. More stringer duplicates were submitted this year, showing greater variability as expected, with core cutting issues identified and rectified during the year potentially contributing to some of the poorer precision. Stringer mineralisation, by nature, is also expected to be less precise in core duplicates.
	Sample sizes are appropriate for the material sampled.
Quality of assay data and laboratory tests	At the exploration stage (surface drilling), 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 original 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. The 2014 surface drilling samples were subjected to a four acid digest and ICP-MS or OES finish. Gold was originally analysed by 50g fire assay to 0.01ppm detection limit, but in 2014 a 25g fire assay with solvent extraction and AAS finish was used. For the underground drill samples similar digest methods were used but in Feb 2012 the 50g fire assay was reduced to a 25g charge (or less) due to high sulphide/copper 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. Umpire check-assay samples identified poor precision for Au (78% within +/-20% RD), due to the need to reduce the catchweight in high sulphide samples, sometimes to 5g (from 25g) at the primary laboratory, which impacts the repeatability of the Au assays. The Ag, Cu, Zn and Pb analyses were shown to be reasonably precise (>90% within +/-20% RD for Ag, Cu, Pb and Zn; 73% within +/-10% RD for Ag, 80% for Cu, 82% for Pb, 91% for Zn) and only low bias (<5% for Ag, Cu, Pb; and <1% for Zn) was observed for these elements in the 2015-16 drilling. Bias is monitored and addressed throughout the year with re-assaying of samples when required. 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 and improve precision.
Verification of sampling and assaying	Significant intersections are checked by company personnel to see they meet the known geological and 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 twin 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 with geological and assay information plotted for review.
	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 non-gyro tool) with surveys at 4m intervals, accuracy to +/-0.01° Azimuth (per station) and +/-0.2° Dip, or more recently a Reflex MEMS referencing gyro with surveys at 3m intervals, accuracy to +/-0.1° Azimuth (per 100m) and +/-0.2° Dip. Mine workings and underground hole collars are surveyed by the on-site surveyors using a Leica TCRP1203 instrument to an accuracy of +/- 3mm, or from November 2013 a Leica TS15P 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

Criteria	Commentary
	workings in the modelled mineralisation.
	All resource work has been conducted on the local mine grid co-ordinate system.
	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 50m x 50m 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 110m (Inferred Resource) for the Arnage lens. Indicated drill spacing is generally less than 40m (northing) x 40m (RL).
	The data spacing and distribution are sufficient to establish the geological and grade continuity for the classifications applied. The wide spaced drilling below 3625mRL, with low intersection angles, has resulted in a resource classification of Inferred, until greater confidence through drilling at appropriate spacing and intersection angles 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 drilling has been undertaken from footwall and hangingwall positions to enable appropriate drill intercept angles to the mineralisation.
	No orientation biased sampling is suspected or has been identified in the data above the 3625mRL. Below that level, drilling is at very low angles to the mineralisation and is classified as inferred or unclassified.
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, and which are also recorded on 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. A data audit was completed by an independent consultancy in January 2015 for all Bentley surface and underground drillholes, in preparation for the resource estimate. Improvements in data collection processes as a result of that audit are being introduced. Data validation commenced with the resource relevant data verified first and other drillhole data to follow.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The Bentley deposit is within mining lease M37/1290 (expiry date is 2 February 2031) held 100% by Independence Jaguar Limited, 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 Jabiru Metals Limited (JML) in 2008. JML was taken over by IGO in 2011. 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 six discrete lenses (Arnage, Mulsanne, Brooklands, Comet, Flying Spur and Zagato). 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.

Criteria	Commentary
Drill hole Information	Holes drilled into the Bentley deposit are described in Section 1. Current drilling is from underground and involves infill grade control drilling known mineralised zones within the resource envelope to a nominal 20m x 20m hole spacing. Resource conversion drilling is drilled at 40m x 40m hole spacing with resource addition drilling on 80m x 80m hole spacing.
Data aggregation methods	There are no exploration results reported for the immediate Bentley mine area.For the resource drilling, top-cuts applied are described in the "*Estimation and modelling techniques"* in Section 3.
	Samples are composited to 1m downhole length using the best fit compositing method. Composite grades are length and density-weighted prior to grade interpolation.
	No metal equivalent values are used.
Relationship between mineralisation widths and intercept lengths	There are no exploration results reported for the immediate Bentley mine area. Orientation of mineralisation with 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	Drilling will continue to focus on delineating additional resources at depth following the main plunge of the mineralised system. Exploration drilling will continue in FY2017.

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, William Stewart, 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 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 logging was 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 a length of 400m along strike (north-south) with a steep, southerly down-plunge length of 890m and a maximum thickness of 30m. It sits 160m below the surface and extends a vertical

Criteria	Commentary
	depth of 1000m. Mulsanne is about 220m along strike, 160m vertical extent, and approximately 3m thick. Brooklands is about 100m along strike, 230m vertical extent, and approximately 4m thick. The Comet lens is about 200m along strike, 160m vertical extent and approximately 5m thick. The Flying Spur lens is about 370m along strike, 300m vertical and averages 2m in thickness. The Arnage Deeps zone sits adjacent to the Arnage lens, approximately 1000m below surface.
Estimation and modelling techniques	Statistics and variography were completed using Supervisor v6.5 software. Ordinary Kriging was used for grade estimation utilising Surpac v6.3.2 software. Block modelling was also completed in Surpac. 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. Mild top-cuts were used to reduce the impact of extremely high grades. Search distances were up to 230m for Pass 1 and up to 382m for Pass3. The minimum number of samples was reduced to 5 for the majority of lenses in Pass 2.
	This estimate compares well with previous estimates for the Bentley deposit. Reconciliation of the undiluted resource model with mine production shows the model tends to underestimate ore tonnes (-20%) and overestimate grades (between 16-22%). The tonnage and grade difference is due to dilution during mining.
	No assumptions have been made regarding the recovery of by-products.
	Zn, Cu, Ag, Au, Fe, Pb, As, Sb, and S have been estimated
	Drill intercept spacing is nominally 20m x 20m in the developed portion of the mine and nominally 40m x 40m in deeper portions of the mineralised envelope (below current development). Kriging Neighbourhood Analysis was used to determine modelling parameters. The parent block size was set at 15m Y x 5mX x 15mZ and grades were interpolated into these blocks. Parent block grades are assigned to sub-blocks within the parent block and the constraining wireframe. Sub-celling is used for better volume resolution. Search dimensions and orientations were set from variography.
	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 independently.
	The block model cells were coded according to which style of mineralisation and which wireframe they were within. Corresponding composite sample files for each wireframe were used for grade interpolation into block model cells inside each of the wireframe domains. Each wireframe was used as a hard boundary during estimation.
	Top-cut grades were determined from a review of the composite sample data statistics, histograms and log-probability plots. Massive sulphide domain top-cuts are applied to Cu (14%), Pb (10%), Ag (1200ppm), Au (12ppm), As (2500ppm). Stringer sulphide domain top-cuts Pb (1.7%), Zn (13%), Ag (400ppm), Au (2.5ppm), As (14000ppm), Sb (300ppm). Azure lens top-cuts are applied for Cu (15%), Zn (18%), Ag (130ppm), Au (0.75ppm).
	The block model is checked visually first, in Surpac graphics, and compared with drilling data, then checked on a section, bench 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. No reconciliation factors are applied to the resource estimate.
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	Resource block cut-off grade for massive sulphide domain of $96/t Net Smelter Return (NSR) was employed. Stringer sulphide domain employed a resource cut-off grade of $60/t NSR. Both cut-offs are derived from estimated future mining and processing costs applied to fully costed massive sulphide domain and incremental stringer domains.
Mining factors or assumptions	Economic grade and mineralisation continuity were taken into account with some areas on the periphery of the ore body excluded from the resource estimate.
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 5 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 test work on almost all core samples that were submitted to the laboratory for assay. All density measurements have been determined using the simple water

Criteria	Commentary
	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 a nominal +/-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 composite grades used in grade estimation.
Classification	Kriging efficiency (KE), slope of regression (RS), drill hole spacing, geological continuity, and mine development was used for 2016 classification. • Measured resources: KE >0.6, RS >0.75, drill spacing < 20m along strike and down dip, high confidence of geological continuity, areas with lateral mine development. • Indicated resources: KE >0.3, RS >0.5, drill spacing < 40m along strike and down dip, high to moderate confidence of geological continuity. • Inferred resources: KE <0.3, RS <0.5, drill spacing > 40m along strike and down dip, moderate to low confidence of geological continuity Unclassified resources: one or few drill intercepts with very low confidence of geological continuity.
Audits or reviews	No audits were conducted for 2016 resource estimate. Optiro Pty Ltd assisted and mentored the 2016 resource estimate for Bentley with supporting documentation complied in the 2016 Mineral Resource report.
Discussion of relative accuracy/confidence	Confidence of the Mineral Resource estimate is high within the measured resource envelope. High to moderate confidence of the Mineral Resource within the Indicated Resource envelope. Moderate to low confidence within the Inferred Resource envelope. Factors considered in classifying the resource estimate were estimators KE, RS, development exposures, drill spacing, quantity of numbers of drillholes and samples for good grade estimation, and mineralisation intersection angles. Sample quality was excellent and did not factor into the classification. The estimate is a local estimate and is suitable for mine planning within areas classified as measured and indicated resources. Reconciliation with production data takes place each time a new Mineral Resource estimate is completed. Reconciliation for the life of mine to 29 February 2016 was carried out in March 2016 and commented on in the sub-section on “Estimation and Modelling Techniques”, above.
Resource Model number	BT_RSC_2016_06

Jaguar Operation (Teutonic Bore) Mineral Resources and Ore Reserves 2016

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Almost all sampling used in this estimate has been core sampling. Mostly sawn half-core samples of NQ or quarter-core samples of HQ core, varying in length up to 1m and adjusted to geological boundaries, for the JML drilling. Historic surface holes were filleted with about 1/3 core diameter used as the sample, up to 2m sample lengths but usually 1.5m. Poorly mineralised zones were chip sampled at about 15cm intervals bulked over 1.5-3m lengths. Sample quality in the JML holes is considered very good and is considered moderate in the historic holes. Underground holes were sampled as sawn half-core BQ core in strong mineralisation and not sampled in weak mineralisation. Drillhole spacing is described in the sub-section *"Data spacing and distribution".*
	Zinc and copper mineralisation is visible and zones containing sphalerite and chalcopyrite, whether in massive sulphide or stringer form, are sampled by JML, along with a 5m buffer zone either side of the mineralised interval.
	Core was cut with an automated core saw or a diamond core saw after orientation, mark-up, logging and photography. The same side of the core is always selected for sampling. Analytical techniques are described in the sub-section *"Quality of assay data and laboratory tests".*
Drilling techniques	Percussion drilling, diamond drilling - some with percussion pre-collars. The surface diamond holes are HQ and NQ core sizes. The underground holes are BQ core size. Core from Jabiru Metals Limited (JML) work was oriented using a Reflex Ace Core Orientation tool. Few percussion drilling holes were used in the 2009 estimate.
Drill sample recovery	Core is measured and marked up on angle iron in continuous runs. For JML holes, core recovery was good to excellent, except where drillholes intersected old underground workings. 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.
	JML core was cut with an automated core saw after orientation, mark-up, logging and photography. The same side of the core was always selected for sampling. Core was saw cut but no information was available for the measures taken to maximise sample recovery and ensure representative nature of the samples in the historic drilling.
	Most core is competent and cuts well with minimal loss of fines. No sample bias is suspected.
Logging	JML 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. Surface drilled holes were logged on paper and subsequently data entered and loaded into the acQuire database. Underground holes were logged but not photographed by Australian Selection. 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.
Sub-sampling techniques and sample preparation	JML core was cut with a diamond core saw or Almonte automated core saw after orientation and mark-up. Core sample sizes are discussed in the *Sampling Techniques* sub-section of Section 1.
	All samples were oven dried, crushed and pulverised. Sample preparation techniques at Genalysis for the JML samples were industry standard. Sample preparation for the historic holes was industry standard for those days but would be considered of moderate quality today.
	Quartz washes at the pulverising stage were implemented between every sample to combat sample carryover (contamination) during the sample preparation process.
	Field duplicates in the form of second quarter-core samples were submitted from the JML drilling. The results showed there is a significant difference in assays between the two quarters. Cu, Zn and Ag show poor reproducibility with numerous samples outside the 20% relative difference lines (32% of Cu analyses, 50% of Zn analyses, and 55% of Ag analyses). These results are indicating that either there is a large sampling error or there is a large inherent nugget effect. The dataset was small (40) and further data are needed.
	Sample sizes are appropriate for the material sampled.

Criteria	Commentary
Quality of assay data and laboratory tests	All JML samples were crushed and pulverised, then a subsample digested using a four-acid digest (digest A or AX) with an AAS finish, at Genalysis. Detection limits for the A digest were 1ppm for Cu, Zn, Ag, and 5ppm for Pb. Detection limits for the AX digest were 0.01% for Cu, Zn, Pb and 5ppm for Ag. Historic sampling (surface holes) was assayed by Australian Selection in house using a 3 acid digest with AAS finish (Cu, Pb. Zn to 0.01% and Ag to 0.2, 2 or 10ppm.). Underground samples were prepared on site and assayed by Analabs in Kalgoorlie using an *aqua regia* digest. The majority of the assay techniques are for total digestion of the sulphides and are considered appropriate for this type of mineralisation. The *aqua regia* digest is a partial digest and those assays (underground drilling) may underestimate true grades of the samples.
	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 for the JML work. Standards and blanks were inserted into the sample sequence in the 2005-2007 campaigns at the rate of about 1 in 40, increasing to 1 in 20 for standards and decreasing to 1 in 50 for blanks in 2008. Check assays on pulps were also carried out, both using the primary lab Genalysis and another lab Ultra Trace. Standards and check assays showed reasonable levels of accuracy and precision in the JML samples. Blanks showed some contamination was occurring and procedures were changed to include barren flushes between samples. Duplicate sampling showed large variation of grades between the two quarter-core samples for the same interval, up to 40% relative difference for Cu and Zn. The analytical technique for Ag in 2008 was not appropriate for the stringer mineralisation grade range and samples were re-analysed using a more suitable technique. Australian Selection assayed 10% of samples in duplicate and if the assays varied by more than 5% the entire batch was re-assayed. Quality control procedures were adequate for the different eras but new work will need to have improved QC applied.
Verification of sampling and assaying	Significant intersections are checked by company personnel to see they meet the known geological and mineralisation models.
	No twin holes were drilled for any of the drilling programs.
	Primary data for JML were collected in Excel spreadsheets or using off-line acQuire data entry objects on Toughbooks. Data were imported directly to the database with importers and have built in validation rules. Assay data were imported directly from digital assay files and were merged in the database with sample information. All holes had a hard copy summary plotted for review with geological and assay information. Historic data were collected on paper logs and hard copy printouts. In 2006 the historic data were compiled by JML, validated and imported to the acQuire database.
	No adjustments are made to assay data once accepted into the database.
Location of data points	All recent drillhole collar positions were surveyed by licensed or company surveyors using either GPS or dGPS. Original Australian Selection surface holes were measured by tape from the nearest grid peg and are considered to have +/-3m level of accuracy. Underground holes have been measured from plans and sections and are considered to be to a +/-5m level of accuracy. Dip and Azimuth readings – generally good quality surveys using Eastman down hole camera shots at 40m intervals down the historic surface holes, and gyro surveys for the recent surface holes to 2007. JML holes in 2008 were downhole surveyed at 20m intervals using a Reflex EZ-Trac digital downhole camera. Underground holes have been measured from plans and sections and only have collar azimuth and dip. Collar and downhole surveys are considered good to moderate accuracy. Mined volume at Teutonic Bore has been removed from the Resource estimate using void wireframes based on historical plans and sections and the surface topography from photogrammetry. Void wireframes are considered accurate to about +/-3m and have been confirmed by intersections during JML's drilling.
	All Resource work has been conducted on the local mine grid.
	Topographic control was from photogrammetry following aerial photography flown in October 2008. Five co-ordinates were used to transform photo centres and observations to the ground co-ordinate system (GDA94/51). The registered photography was then transformed to the Teutonic Bore local grid for use in modelling.
Data spacing and distribution	Diamond drill coverage at Teutonic Bore is on a nominal 20 x 20m (massive) to 40 x 40m (stringer) pattern with stringer mineralisation closer to the massive sulphide having closer spaced drilling. Twin holes have not been drilled.

Criteria	Commentary
	The data spacing and distribution is sufficient to establish geological and grade continuity appropriate for the Mineral Resource estimation procedure and classification applied (Indicated) in the massive sulphide and Indicated or Inferred classification in the stringer mineralisation. Samples were composited to 1m length with an acceptable minimum of 0.6m, using length and density-weighting for Cu, Zn, Pb and length-weighting for Ag and density.
Orientation of data in relation to geological structure	Surface drilling intersects the massive sulphide lenses almost perpendicular to the lens orientation. Holes drilled in 2008 for the stringer zone were drilled from the east side of the pit to avoid the underground workings. The stringer zone is intersected at moderate to low angles to the mineralisation. The underground fan drilling mostly intersects the massive sulphide zone at a variety of angles. Two of the underground holes were removed prior to the estimate due to inappropriate dip orientations. No orientation biased sampling was used in the Resource estimate.
Sample security	All JML 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. JML 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. No information on sample security is available for the historic drilling.
Audits or reviews	Sampling techniques and data collection processes are reviewed regularly by IGO staff. No external review has been conducted. A program of data compilation and validation was completed in 2006 for all historic drillholes in preparation for Resource estimation.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	Teutonic Bore is located within mining lease M37/44. There are no Native Title Claims registered over the lease and no other known impediments. Heritage sites registered at the Department of Aboriginal Affairs (DAA) surround the area but the immediate pit area does not encroach on these. The tenure is secure and expires on 17 December 2026. Other than the heritage sites mentioned above there are no known impediments to obtaining a licence to operate in the area.
Exploration done by other parties	JML acquired the Teutonic Bore mining leases from Mount Isa Mines (MIM) in August 1997. No exploration is being conducted by other parties in or around the Teutonic Bore deposit.
Geology	Teutonic Bore is a V(H)MS style deposit, occurring as a polymetallic (pyrite-sphalerite-chalcopyrite) massive sulphide lens within a volcano-sedimentary succession. An extensive feeder zone below the massive sulphide lens (in the footwall) has produced a large sulphide stringer zone. The mineralisation dips steeply (70-80°) to the west (local grid) and plunges gently (20°) to the north.
Drill hole Information	Holes drilled into the Teutonic Bore deposit are described in Section 1. No new drilling is being reported here. A summary of drillholes is not considered applicable in this instance as no new drilling is being reported.
Data aggregation methods	There are no exploration results reported for the immediate Teutonic Bore mine area.
Relationship between mineralisation widths and intercept lengths	There are no exploration results reported for the immediate Teutonic Bore mine area. Orientation of mineralisation with drilling angles has been covered in Section 1.
Diagrams	There are no exploration results reported for the immediate Teutonic Bore mine area.
Balanced reporting	There are no exploration results reported for the immediate Teutonic Bore mine area.
Other substantive exploration data	There are no exploration results reported for the immediate Teutonic Bore mine area.
Further work	Further drilling to better define the stringer mineralisation may follow if preliminary mine planning studies show positive results.

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 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, Will Stewart, 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 is high for the geological interpretation of the massive sulphide and is moderate for the stringer zone. Vein orientation is not well understood in the stringer zone and drilling density sparser, with mineralisation boundaries defined by cut-off grade rather than geologically defined units. As the cut-off grade increases, continuity of mineralised stringer zones reduces.
	Good geological cross-sectional interpretations were available to guide modelling of the mineralisation. No underground mapping was available to aid interpretation.
	There have been no alternative interpretations of the mineralisation model, with the Teutonic Bore massive sulphide deposit being well regarded as a V(H)MS style orebody. It is relatively planar and simple in geometry. The stringer interpretation is based on assay cut-offs and could be interpreted slightly differently; however this is not expected to have a major impact on the stringer Resource estimate. Should additional sampling be added to the database for the historic drilling, then this could impact on the stringer Resource estimate.
	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.
	See above. There are no known intrusives or significant faults disrupting the mineralisation at Teutonic Bore.
Dimensions	The massive sulphide (pre-mining) is a tabular body about 250m long and 17m thick (true width), extending down dip for about 190m. The remnant mineralisation is located 240m below surface, below the previously stoped mineralisation, as well as in fingers to the south and north ends of the open pit and stoped areas. The stringer mineralisation occurs in the footwall to the massive sulphide zone over a strike length of about 245m. It is up to 50m thick and extends down dip about 200m.
Estimation and modelling techniques	GeoAccess software was used for statistical analysis of the composites. Surpac software v6.1 was used for the variography and block modelling. Ordinary kriging (with top-cuts) was used for grade interpolation, based on the variography and validation of the search orientations in Surpac. Block cells had been coded with the wireframe name and only composite samples from that zone were used to interpolate grades into that zone. All grade interpolation was constrained to within the massive and stringer sulphide wireframes. The massive sulphide was domained into a fresh rock and a transitional rock domain for statistics and variography. Both these domains were further subdivided for search ellipse orientation changes due to changes in their geometry in the south end. The largest of the stringer zones was used to establish kriging parameters and these were applied to the other stringer zones with an appropriate search ellipse orientation change. Search distances were generally 150m along the major axis, up to 140m in the semi-major direction and up to 40m in the minor direction (18m in the massive zone). Ordinary kriging is considered an appropriate estimation technique for this style of deposit.
	Both the massive and stringer estimates compare well with previous estimates. The Mineral Resource estimate is undiluted. It does not take into account any historic production data, other than to have the previous workings model used to deplete the Resource estimate. Block model cells were coded as mined if within the open pit or void wireframes and were excluded from the estimate. The void wireframes were expanded slightly to remove any skins of mineralisation that might be left behind through the coding of the cells within the wireframes.

Criteria	Commentary
	No assumptions have been made regarding the recovery of by-products.
	Economic minerals estimated are Zn, Cu, Ag and Pb (sub-economic). Density was also estimated. Insufficient assay data for Au, Fe and S were available for grade interpolation. There are no known deleterious elements in the Teutonic Bore mineralisation.
	The block model had extents of 700m in Y, 500m in X and 410m in the Z direction. The parent cell size was 5 x 5 x 5m sub-celling to 1.25 x 1.25 x 1.25m. The parent cell size was a compromise between close-spaced drilling in the massive sulphide and wider-spaced drilling in the stringer zone. Sub-cell size was determined more for an open-cut mining scenario rather than underground, and could be reduced further for better resolution in an underground mining scenario.
	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 for each wireframe were used for grade interpolation into block model cells inside each of the wireframe domains. Each wireframe was used as a hard boundary during estimation.
	Top-cut grades for massive and stringer mineralisation were defined using log-probability plots and identifying the inflexion point indicating deviation from log-normality. Top-cut grades applied were: Massive sulphide fresh rock 18% Cu, 3.8% Pb, 880ppm for Ag and no top-cut for Zn; massive sulphide transitional rock 17% Cu, 33% Zn, 3.6% Pb and 440ppm Ag; Stringer sulphide 7% Cu, 12% Zn, 2% Pb and 350ppm Ag.
	The block model is checked visually first, in Surpac graphics, and compared with drilling data, then checked on a section, bench and lens basis by comparing composite sample grades with block model grades in swath plots. For the massive sulphide the grade comparison is adequate and for the stringer the grade comparison is very close.
Moisture	Tonnages have been estimated using densities that contained natural moisture. The natural moisture of the Teutonic Bore massive sulphides is typically very low (<1%).
Cut-off parameters	No cut-off grade was applied to the massive sulphide as the mineralisation was defined geologically. A modelling cut-off grade of 0.5% Cu was applied to the stringer mineralisation. A block cut-off grade of 0.7% 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 was in production from 1980-1984 with a conventional flotation processing plant, where mostly massive sulphide ore was processed. The stringer mineralisation would be amenable to open cut mining methods or underground sub-level caving.
Metallurgical factors or assumptions	No metallurgical factors or assumptions have been made; the Jaguar mill on site is a flotation plant which generates two concentrate types, and has treated Jaguar and Bentley massive sulphide and stringer ore proficiently over numerous years. There is no known reason why Teutonic Bore ore could not be successfully treated on site, given the production history of the old Teutonic Bore mine.
Environmental factors or assumptions	No environmental factors or assumptions have been made. The waste dump and tailings storage facilities for the nearby Bentley mine are well established with approval from the Department of Mines and Petroleum (DMP). There are however, areas surrounding the old open pit that have been rehabilitated and signed off by the Department of Mining and Petroleum (DMP), and if IGO/JML were to re-start mining there could be significant environmental liability issues to deal with for past disturbances and dumping. Any renewed mining activities would require a new mining proposal and works approval which require DMP and Department of Environment Regulation (DER) approvals respectively.
Bulk density	JML performed density testwork on almost all core samples that were submitted to the laboratory for assay. Historic density data were restored during the 2006 data compilation stage. Most samples had measured densities determined using the simple water immersion technique. Densities were checked against density vs grade regression curves and outliers were replaced with calculated densities or in the case of the stringer mineralisation, a nominal density of 2.95g/cm^3. The density dataset is quite large and in good condition. Densities were used for compositing Cu, Zn and Pb grades and were interpolated into the block model in the same way as a grade.

Criteria	Commentary
Classification	The massive sulphide mineralisation was classified as Indicated because it has closely spaced drilling and a production history, as well as good confidence in the geological model. The stringer mineralisation was classified as Indicated where drill spacing was about 20x20m and Inferred where drill spacing was about 40x40m. Stringer mineralisation also had some historic holes drilled through it that were not sampled and these areas, if not sampled with JML drilling, were classified as Inferred. Mineralisation modelled but with drilling density sparser than 40x40m was not classified as Resource.
	Input data are partly historic (of reasonable quality) and more recent data are of excellent quality. There is high confidence in the massive sulphide mineralisation interpretation and moderate confidence in the stringer interpretation. Confidence in grade and tonnage estimates where the drilling is at a >40m spacing is low, and is reflected in the classification. More drilling into the stringer zone along with sampling of previously unsampled intervals in the historic drilling will improve the interpretation, confidence and Resource estimate for the stringer zone.
	The classification of the Mineral Resource reflects the Competent Person's view of the confidence in the estimate.
Audits or reviews	A review of the Resource estimate was conducted by Runge Limited in 2009 which identified no significant issues other than some aspects of the variography, derivation of kriging parameters and search neighbourhoods. Subsequent review (by Wildfire Resources Pty Ltd and JML staff) of these aspects concluded that there was no material issue that required action.
Discussion of relative accuracy/confidence	Confidence in the Mineral Resource estimate for massive sulphide is high in the areas with mine development and/or drilling with a 20m x 20m pattern. Confidence is moderate to low in the stringer mineralisation. 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, sampling and assaying quality, and mineralisation intersection angles. Assay quality was varied with recent JML sampling and assaying considered more reliable than historic sampling and assaying. The main factors that could affect the accuracy of the estimate, in particular the stringer Resource estimate, is the historic assaying methods, the selective sampling of intervals with some of the low grade intervals not sampled, and drillhole spacing.
	The estimate is a global estimate and is suitable for long term mine planning.
	The Resource estimate is a remnant Resource estimate and as such has no production data available for comparison.
Further work	Historic core that has not been sampled and is in suitable condition may be sampled to improve the detail of the Resource estimate prior to mining. Similarly JML core that was not sampled but lies within the mineralised envelope may be sent for assaying. Core trays for historic drilling have been rehabilitated and are in a suitable condition for longer term storage. Preliminary mine planning studies into the viability of the Teutonic Bore project is the next step. Other testwork would be completed as part of a Pre-feasibility study if the preliminary mine planning studies meet the company's expectations.
Resource Model number	TB_RSC_2009_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 Bentley Mineral Resources. No reserves exist outside of the Mineral Resource base.
	Mineral Resources are inclusive of Ore Reserves.
Site visits	The competent person has not visited the site. The competent person is comfortable relying on reports from other independent consultants, and other Entech staff, who have visited site and other operations in the area respectively.
Study status	The mine Reserves have been designed based on the current operational practices of the mine. All Ore Reserves are estimated by construction of 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 cost model where each stope is economically evaluated and the total reserve is evaluated to assess its economic viability.

Criteria	Commentary
	Previous mine performance has demonstrated that the current mining methods are technically achievable and economically viable. The modifying factors are based on historical data utilising a similar mining method.
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. All designed stopes and development are then assessed individually to verify that they are above the NSR cut-off and can be economically mined. For longhole stoping, the mill feed NSR cut-off used is $134/t with a marginal cut-off NSR value of $80 /t. For development a $57/t cut-off NSR value has been used. Airleg stoping is an incremental mining method and a $161/t cut-off NSR value has been used.
Mining factors or assumptions	Three dimensional mine designs are designed based on known information about the orebodies physical characteristics and the geotechnical environment. The designs are consistent with what has been in practice on site. Modifying factors such as unplanned dilution (10-20% for Long hole stoping depending on stope size) and mining recovery (90-95% for stoping, depending on mining method and stope size) are applied based on the chosen mining method. In some cases, additional recovery losses have been applied where pillars are required. These modifying factors are based on recent historical data. Minimum mining widths 1.5 m and 1 m were used for longhole stopes and airleg stopes respectively. Current infrastructure supports mining of the Reserve. Any additional capital required has been included in the cost model.
	In cases where Inferred Mineral Resources are present in a design, this material has been assigned as waste and re-evaluated to ensure it is economically viable.
Metallurgical factors or assumptions	Ore from Bentley is processed at the Jaguar processing facilities. The process and recovery of contained metal is well understood and reasonably consistent in performance. The following metallurgical recovery factors have been used: 85.0% Cu recovery into Cu concentrate, 45.0% Ag recovery into Cu concentrate, 32.0% Au recovery into Cu concentrate, 86.0% Zn recovery into Zn concentrate and 16.0% Ag recovery into the Zn concentrate.
Environmental	Independence Group operates under an environmental management plan, which meets or exceeds all environmental legislative requirements. Independence Group's license to operate is in good standing. Environmental rehabilitation plans are constructed and progressively acted upon. The costing of the rehabilitation works is accounted for in the operations Reserve model at a value of $11,000,000 AUD.
Infrastructure	The current infrastructure at the Jaguar Operation is adequate for the extraction of the Bentley underground Reserves. Maintenance costs for current equipment were included in the Reserve economic model.
Costs	Capital costs for decline development and accesses were included in the financial evaluation. An allowance per ore tonne is also made for ongoing exploration costs. Operating costs were modelled on recent actual mining costs from site. Concentrate payables, which includes accounting for any deleterious elements, has been calculated and used within the NSR evaluation process.
Revenue factors	The assumptions made for commodity prices are: Copper price US$ 5,540 per tonne, Zinc price US$2,020 per tonne, Silver price US$17.00 per troy ounce, Gold price US$1,200 per troy ounce and Foreign exchange rate of AU$1.00 : US$0.75.
	These prices are based on the 50th percentile of the Consensus Economics commodity price assumptions as well as the 50th percentile of Bloomberg foreign exchange rates (US$ to AU$).
	During the calculation of Reserves, the Metal prices and exchange rates were assumed fixed for the life of the project.
Market assessment	The volume and high quality of concentrate produced is expected to continue to attract a ready market domestically and internationally.
Economic	The reserve shows a robust NPV. The confidence in the inputs is consistent with the assigned Proved and Probable classifications of the Ore Reserve.
	Sensitivity analysis work has been undertaken on variables such as mining costs, processing costs, foreign exchange rate and metal price, with the NPV proving particularly sensitive to changes in the exchange rate.
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.

Criteria	Commentary
	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 Mineral Resources.
Audits or reviews	The Ore Reserve has been peer reviewed internally and is in line with current industry standards.
Discussion of relative accuracy/confidence	The Ore Reserve is based on recent operational performance and costs at the mine; hence confidence in the resulting figures is high.
	Confidence in the mine design and schedule are high as mining rates and modifying factors are based on actual site performance. Mine design is consistent with what has been effective previously.

APPENDIX E

Stockman Mineral Resources and Ore Reserves 2016

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 37 additional diamond drill holes 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

Criteria	Commentary
	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 75µm. 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. 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

Criteria	Commentary
	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 considered material.
Sample security	Drill core was transported from the drilling site to the Stockman core yard by staff 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.

Criteria	Commentary
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.

Criteria	Commentary
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 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)

Criteria	Commentary
	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.
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.
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.

Criteria	Commentary
	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 and has remained unchanged. 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 Tailings Storage Facility (TSF) of Lake St Barbara, the old plant site at Waxlip spur, existing Wilga portal and site of the proposed Currawong portal.
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 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.
	The cut-off NSR value was determined from the site operating costs including mining, processing and site administration and overhead costs. The cut-off was estimated to be between \$97 and \$105 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. The incremental NSR cut-off was applied to ore drive development to determine its inclusion in the Ore Reserve estimate.
Mining factors or assumptions	The Ore Reserve was determined by digitising practical stope wireframes around contiguous blocks of Indicated material above the NSR cut-off value. The wireframes include a nominal 0.5 m of unplanned mining dilution from over-break. An additional 2% dilution was included for all stopes due to fall off from paste walls. A further 2% dilution allowance was applied to secondary stopes which will have more than one exposed wall 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. In addition, any development, such as ore drives, outside the stope wireframes, which reported an average value above an incremental NSR cut-off of \$60 per tonne, were also included in the Reserve.
	The Ore Reserves for both Currawong and Wilga were determined on the basis of long hole open stoping using cemented 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

Criteria	Commentary
	deposits at Currawong and Wilga.
	Stope spans and other ground support requirements were based on analysis conducted by geotechnical consultants Mining One Pty Ltd. Grade control methods would entail methods used by Independence Group NL (IGO) at their existing operations in WA and will include stope definition diamond drilling, face and stockpile sampling.
	The Mineral Resource model was originally prepared and reported by IGO geologists in accordance with the JORC Code (2004), and the report 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 and Fe within wireframe constraints.
	Sufficient detailed analysis was carried out to provide confidence in key assumptions such as stability of stope spans and mining rate. Recent testwork of the paste backfill using cycloned tailings to remove the ultrafine fraction of the tailings stream has demonstrated over time (260 days) that a number of 4% binder mixes can achieve acceptable early strengths, good long terms strengths with no signs to date of fill degradation. The final tests were at 365 days and were due in February 2015.
	The total dilution (planned plus unplanned) included in the stope wireframes was estimated to be 14% at grades reported from the Mineral Resource model and within the diluted stope wireframe envelopes. The mining method requires total extraction within the stoping envelope; therefore, no losses will occur from sterilisation of ore in pillars. A nominal 5% ore loss was applied for reasons as described above. An approximate minimum mining width of 3 m (true width) was used when creating the stope wireframes.
	The underground capital infrastructure will include 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 workforce refuge chambers. In addition, a backfill paste plant will need to be constructed and paste reticulated throughout the stoping areas of the Currawong mine. At Wilga, tailings filter cake will be trucked to from Currawong to a surface stockpile adjacent to the Wilga portal where it will be used to produce paste on surface, then pumped into the mine and reticulated throughout the workings.
	The Ore Reserve estimate includes Inferred and unclassified material in the form of mining dilution estimated to be approximately 780,000 t at 0.31 Cu%, 1.0 Zn%, 5.2 g/t Ag and 0.1 g/t Au.
	The Wilga Ore Reserve was derived from the Indicated Mineral Resource. The Indicated classification was based on the confidence in copper, zinc and silver grades; however, 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 $8.65 per gram of gold in situ and within the mining envelope. This 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 18% of the total Ore Reserve tonnage.
Metallurgical factors or assumptions	The metallurgical process will use differential flotation to produce separate concentrates of copper and zinc minerals.
	The 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 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; therefore, recoveries vary depending on the combination of minerals present in the feed at any increment in time. The life of mine average recoveries for copper concentrate were as follows: • Copper = 81.5% • Silver = 40.7% • Gold = 20.4% The life of mine average recoveries for zinc concentrate were as follows: • Zinc = 76.4 % • Silver = 18.5%
	Metallurgical test work 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%).

Criteria	Commentary
	The penalty element assays were generally low, but where slightly elevated, remained in the negotiable range for settlement. Deductions for penalty elements were 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.
	Bulk flotation testwork was carried out on ten 50 kg samples as part of the 2014 Optimisation Study. The testwork demonstrated the geo-metallurgical algorithms to be reliable.
	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 zinc and lead for copper concentrate and iron 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 (TSF).
	Tailings produced from on-site processing will be either returned to the underground workings as backfill or disposed of in an approved 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. Detailed geotechnical and hydrogeological assessment of the proposed TSF impoundment is still to be carried out to validate the design parameters. No material changes to the design parameters are anticipated.
	Water produced from dewatering the underground workings will be treated and recycled for use in the mining or processing operations. Surplus 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 project also triggers Commonwealth assessment which utilises the Victorian EES via a State-Federal bilateral agreement. 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.
	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. The Victorian Minister for Planning has provided a positive assessment of the EES and associated Inquiry Panel report. The approval process by the federal Environment Minister is still in progress.
	There are no known impediments to the outstanding portions of 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 which will need to be upgraded to bring these services to the site. Power will be generated on site using natural gas sourced from the Victorian natural gas infrastructure. Water balance modelling has indicated the project will have a near neutral requirement for supplementary water; however, contingent sources of groundwater have been identified beneath the Benambra plains and various locations adjacent to the site access road. 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

Criteria	Commentary
	impediments to the granting of this license.
Costs	Capital costs for the project were based on 2014 budget quotations provided by potential vendors based on 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 or sourced internally from the IGO existing operations.
	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 similar mining projects within Victoria.
	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. A number of export port options are available; however, the costs used contemplated export via Port Anthony. 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 forecasts provided by recognised market analysts.
	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 considered to be typical 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.
	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 real pricing and were based on forecast pricing provided by recognised market analysts. The project is therefore leveraged to the rising future zinc price forecast by most industry analysts.
	The average Life of Mine metal prices and foreign exchange rate used in the cash flow model included the following: • Copper $US 6,591 per tonne of copper metal • Zinc $US 2,979 per tonne of zinc metal • Gold $US 1,146 per ounce troy of gold • Silver $US 20.17 per ounce troy of silver • Exchange rate of $0.84 $AU per $US
	The cash flow was modelled in real terms and no price or cost escalation was applied.
Market assessment	In its September 2014 "Q3 2014 Global copper long-term outlook", Wood Mackenzie forecast the need for a long-term incentive price of US$3.50/lb Cu (in 2014 dollars) to encourage sufficient investment in mine capacity to ensure the market does not slip into structural deficit. This equates to a tonnage price of US$7,700/t Cu (in 2014 dollars).
	In its September 2014 "Q3 2014 Global zinc long-term outlook", Wood Mackenzie forecast the need for a long-term incentive price to encourage mine development to address market deficits. For the period between 2013 to 2035 Wood Mackenzie have forecast the price to average US$2997/t (in 2014 dollars).

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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 determined by recognised market analysts. The cash flow was modelled in real terms, hence no price or cost escalation was applied. A discount rate of 10% was applied 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 25%.
	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 process plant feed grade, capital and operating costs, metal prices, foreign exchange rate. The results were evaluated on the basis of pre-tax NPV. All parameters tested returned a positive NPV over the range.
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 maximise 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 agreement to access the area for the purposes of confirmatory site investigation works is currently being sought from the state government. The state government has acknowledged the need to access the site without transferring liability of the facility; however, details are still to be finalised. The project sanction will be contingent on confirming the site conditions and the proposed tailings management strategy. An application for an Infrastructure Mining License will be made following project sanction by IGO. There are no known impediments to the granting of this license.
	The project is located in state forest and 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: • Local pastoralist for land access to the Stockman Village site and a potential borefield site. • East Gippsland Shire Council for road improvements and maintenance, land (road verge) access for a potential borefield pipeline and high voltage underground power cable.
	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.
	The Victorian Minister for Planning’s positive assessment of the Stockman Environmental Effects Statement (EES) and Inquiry Panel report was released on 30 October 2014. The Minister’s Assessment contains no unexpected conditions that are material to the project. The specific detail of the various licenses that are required for project operations will now be scoped and agreed with Government agencies. It is expected that detailed licensing will take approximately 12 months. The Victorian Minister for Planning’s assessment report was also provided at the same time to the

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	Commonwealth Minister for Environment for his consideration under the Environment Protection and Biodiversity Conservation (EPBC) Act. The Federal decision is expected to be announced by early 2015, and no significant issues are foreseen.
	There are currently no unresolved matters with third parties.
Classification	The Ore Reserve was classified in accordance with the guidelines in the JORC Code (2012). Standard modifying factors and conversions were applied as described above. The Currawong portion of the Ore Reserve comprised approximately 95% Indicated Mineral Resource with the balance comprising dilution from Inferred and unclassified material. The Wilga portion of the Ore Reserve comprised approximately 85% Indicated Mineral Resource with the balance comprising dilution from Inferred and unclassified material. The Ore Reserve estimate includes Inferred and unclassified material in the form of mining dilution estimated to be approximately 780,000 t at 0.31 Cu%, 1.0 Zn%, 5.2 g/t Ag and 0.1 g/t Au. Given these proportions and the manner in which the modifying factors were applied, the entire estimate was classified at Probable Ore Reserve. No known conditions or issues existed that warranted downgrading of this classification.
	Gold (Au) grades are Inferred at Wilga due to a paucity of gold assays in historic drilling.
	The Ore Reserve estimation and classification methods used were 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 Mineral Resources largely classified as Indicated with the balance comprising either Inferred or unclassified material incorporated as mining dilution. 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.
	A sensitivity analysis was conducted on the cash flow model over a +/-20% range of variability of key parameters including copper and zinc grade, capital and operating cost, copper and zinc price and exchange rate. The results were evaluated on the basis of pre-tax cash flow. All parameters tested returned a positive pre-tax cash flow over the range.
	The nature of the deposit is such that the economic mining envelope is dependent on metal price and foreign exchange assumptions. Material changes in price assumptions could alter the outcome of the Ore Reserve estimate.
	Discrepancy exists between the historical tonnes reported as mined at Wilga (about 956 kt) and those accounted for in current digital wireframe of the workings (about 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 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.
	Vendor quotation used in the cost estimates were requested on the basis of +/-10% to 15% accuracy.
	Revenue assumptions were based on real forward-pricing calculated by combining Consensus forecasts (sourced from eight separate banks &/or research firms) and Wood Mackenzie forecasts. Forecast periods varied across the sources, ranging from two years to eight years. Where the project life exceeded a reasonable number of available forecasts, the last period price was held constant (or "flat-lined") in the cash flow model to the end of the project. The Life of Mine (LoM) average forward copper pricing (US$6,591/t) is 11% below the September 2014 long term forecast copper pricing by Wood Mackenzie which covers the mine production period (US$7,446/t). The LoM average forward zinc pricing (US$2,979/t) is 1% below the September 2014 long term forecast zinc pricing by Wood Mackenzie which covers the mine production period (US$3,020/t). The foreign exchange rate used the IGO-derived consensus rate calculated by combining forecasts from three separate bank &/or research firm's forecasts. As with the copper and zinc pricing, the foreign exchange rate was flat-lined from the last period forecast to the end of the project.
Reserve Model Number	SM_OS_RSV_2014_11