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PANAUST

ASX ANNOUNCEMENT
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2014 Mineral Resource and Ore Reserve Statements

Following are the PanAust Limited statements of Mineral Resources and Ore Reserves for material projects as at 31 December 2013, reported under The JORC Code, 2012 Edition.

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SUMMARY OF MINERAL RESOURCES AND ORE RESERVES

Mineral Resources

Phu Kham (including KTL)

Depletion from mining during 2013 and block model adjustments to marginally mineralised material in contact with the footwall fault account for the changes to the 31 December 2013 Phu Kham Mineral Resource when compared with the 31 December 2012 estimate. No updated geology interpretation or resource interpolation was completed on the 2012 model during 2013.

During the year there was a downgrading of resource blocks at the contact and proximal to the fault contact between the mineralised Phu Kham volcanics and the un-mineralised red bed (basement) material. This reduction was a result of review of the continuity of the mineralisation at the contact, based on mining experience over the past two years. The mineralisation downgraded to waste was Indicated and Inferred in classification and had marginal copper and gold grades. The blocks screened out were also in un-minable configurations and would likely be diluted with un-mineralised fault clay and/or red bed material during mining.

Additional exploration and infill drilling at the Khamthonglai (KTL) copper-gold deposit during 2013 led to an updated model that formed a basis for the KTL 2013 study. The updated model incorporated drill results; geologic interpretation; grade estimation; and Mineral Resource classification.

Study work indicated that a higher copper cut-off than had previously applied would be required for the preferred development option of operating KTL as a satellite open-pit to Phu Kham. Based on the work completed in the study combined with continuity of mineralisation the reporting copper cut-off for the resource was raised to 0.5% copper. The classification of the Mineral Resource is nearly an even split of Measured and Indicted with no Inferred Resource reported.

Phu Kham (0.2% copper cut-off)	**31 December 2013**				**31 December 2012**			
Class	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
Measured	118	0.53	0.23	1.9	138	0.51	0.23	1.9
Measured (stockpiles)	3	0.31	0.17	1.2	0.3	0.50	0.24	2.2
Indicated	72	0.46	0.21	2.3	75	0.45	0.21	2.3
Sub-total (M+I)	**192**	**0.50**	**0.22**	**2.0**	**214**	**0.49**	**0.22**	**2.0**
Inferred	12	0.37	0.21	1.9	14	0.36	0.21	1.8
TOTAL	**204**	**0.49**	**0.22**	**2.0**	**227**	**0.48**	**0.22**	**2.0**
KTL	**KTL (0.5% copper cut-off)**				**KTL (0.25% copper cut-off)**			
Measured	9	1.13	0.57	3.4	22	0.65	0.36	2.3
Indicated	10	0.78	0.27	3.9	62	0.40	0.15	2.3
Sub-total (M+I)	**19**	**0.94**	**0.41**	**3.7**	**83**	**0.47**	**0.21**	**2.3**
Inferred	0	0.52	0.02	0.3	9	0.33	0.05	1.5
Total	**19**	**0.94**	**0.41**	**3.7**	**92**	**0.45**	**0.19**	**2.3**
Measured	127	0.57	0.25	2.0				
Measured (stockpiles)	3	0.31	0.17	1.2				
Indicated	82	0.50	0.22	2.5				
Sub-total (M+I)	**211**	**0.54**	**0.24**	**2.2**				
Inferred	12	0.37	0.21	1.9				
TOTAL	**223**	**0.53**	**0.24**	**2.1**				

The main difference when compared with the KTL Mineral Resource estimate as at 31 December 2012 is the application of a higher copper cut-off grade. The increase in copper cut-off grade from

0.25% copper to 0.50% copper was implemented to better reflect the mining and operational costs to exploit the deposit as a satellite open-pit to Phu Kham rather than as a standalone operation. The change in cut-off grade reduced the Mineral Resource tonnes by approximately 79% but captures 43% of the contained copper.

The KTL deposit has not previously been reported as part of the Phu Kham operation. The inclusion of the KTL deposit within the Phu Kham Mineral Resource has largely compensated for tonnes depleted due to mining at Phu Kham during 2013. Total Mineral Resource tonnes for the combined Phu Kham – KTL Mineral Resource as at 31 December 2013 declined by 2% when compared with the Phu Kham Mineral Resource as at 31 December 2012 and contained copper increased by 8%.

Other minor changes resulted from incorporating results from drilling during 2013 and subsequent resource modelling. No mining has occurred on the deposit and all mineralised material at the previous reporting cut-off remains in-situ.

Ban Houayxai

Mining depletion accounted for all the changes to the 31 December 2013 Ban Houayxai resource. No updated geology interpretation or resource interpolation was completed on the 2012 model during 2013.

Ban Houayxai	**31 December 2013**				**31 December 2012**			
Class	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
Oxide (0.25g/t gold cut-off)								
Measured	0	-	0.57	3.1	2	-	0.83	3.3
Indicated	7	-	0.70	3.7	8	-	0.70	3.7
Sub-total (M+I)	**7**	**-**	**0.70**	**3.7**	**9**	**-**	**0.73**	**3.6**
Inferred	1	-	0.38	1.7	1	-	0.38	1.7
Sub-total (Oxide)	**8**	**-**	**0.66**	**3.5**	**10**	**-**	**0.70**	**3.4**
Transitional	**0.30g/t gold cut-off**				**0.35g/t gold cut-off**			
Measured	3	-	0.83	10.6	4	-	1.00	10.4
Indicated	14	-	0.83	9.0	12	-	0.92	9.3
Sub-total (M+I)	**16**	**-**	**0.83**	**9.3**	**16**	**-**	**0.94**	**9.6**
Inferred	0	-	0.45	3.4	0	-	0.51	3.7
Sub-total (Transitional)	**17**	**-**	**0.82**	**9.1**	**16**	**-**	**0.93**	**9.5**
Primary (0.40g/t gold cut-off)								
Measured	1	-	1.10	10.3	1	-	1.10	10.3
Indicated	30	-	1.04	7.6	30	-	1.04	7.6
Sub-total (M+I)	**31**	**-**	**1.04**	**7.7**	**31**	**-**	**1.04**	**7.7**
Inferred	7	-	0.87	5.9	7	-	0.87	5.9
Sub-total (Primary)	**37**	**-**	**1.01**	**7.4**	**37**	**-**	**1.01**	**7.4**
Combined Oxide, Transitional, Primary								
Measured	4	-	0.85	9.8	7	-	0.97	8.7
Measured (stockpiles)	2	-	0.38	2.4	1	-	0.41	1.5
Indicated	50	-	0.94	7.4	49	-	0.96	7.4
Sub-total (M+I)	**56**	**-**	**0.91**	**7.4**	**57**	**-**	**0.95**	**7.4**
Inferred	8	-	0.80	5.4	8	-	0.81	5.4
TOTAL	**64**	**-**	**0.90**	**7.1**	**65**	**-**	**0.93**	**7.2**

Long Chieng Track (LCT)

Long Chieng Track (0.3g/t gold cut-off)	31 December 2013				31 December 2012			
Class	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
Measured	7	0.02	0.72	4.0	3	0.02	0.72	2.3
Indicated	12	0.05	0.81	5.2	5	0.05	0.65	4.7
Sub-total (M+I)	**19**	**0.04**	**0.78**	**4.7**	**8**	**0.04**	**0.67**	**3.8**
Inferred	11	0.15	0.72	2.4	25	0.15	0.80	5.3
TOTAL	**31**	**0.08**	**0.76**	**3.9**	**32**	**0.08**	**0.72**	**4.3**

During the year the Long Chieng Track (LCT) gold–silver Mineral Resource was updated with new drill results; geologic interpretation; grade estimation; and Mineral Resource classification. The main difference in the new Mineral Resource estimate is the updated drill results and modelling. No mining has occurred on the deposit and all mineralised material at the previous reporting cut-off remains in-situ.

The results of a scoping study have been helpful in setting on-going exploration priorities and have confirmed the requirement for the identification of additional resources at LCT or the nearby Nam Ve prospect before further studies are warranted.

Inca de Oro and Carmen

Inca de Oro	31 December 2013				31 December 2012			
Heap leach								
Inca de Oro Oxide and Mixed (0.25% copper cut-off)[1]								
Class	**Tonnes (Mt)**	**Cu Soluble (%)**			**Tonnes (Mt)**	**Cu Soluble (%)**		
Measured	11	0.22			11	0.22		
Indicated	54	0.23			54	0.23		
Sub-total (M+I)	**64**	**0.22**			**64**	**0.22**		
Inferred	7	0.14			7	0.14		
TOTAL	**71**	**0.20**			**71**	**0.20**		
Flotation								
Inca de Oro Supergene and Primary (0.25% copper cut-off)								
Class	**Tonnes (Mt)**	**Cu Total (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu Total (%)**	**Au (g/t)**	**Ag (g/t)**
Measured	186	0.44	0.13	2.0	186	0.44	0.13	2.0
Indicated	126	0.35	0.08	1.7	126	0.35	0.08	1.7
Sub-total (M+I)	**312**	**0.41**	**0.11**	**1.8**	**312**	**0.41**	**0.11**	**1.8**
Inferred	77	0.30	0.06	1.4	77	0.30	0.06	1.4
TOTAL	**389**	**0.39**	**0.10**	**1.7**	**389**	**0.39**	**0.10**	**1.7**

Carmen Transitional and Primary (0.25% copper cut-off)	31 December 2013				31 December 2012			
Class	**Tonnes (Mt)**	**Cu Total (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu Total (%)**	**Au (g/t)**	**Ag (g/t)**
Measured	5	0.33	0.42	1.1	5	0.33	0.42	1.1
Indicated	7	0.35	0.43	1.3	7	0.35	0.43	1.3
Sub-total (M+I)	**12**	**0.34**	**0.43**	**1.2**	**12**	**0.34**	**0.43**	**1.2**
Inferred	34	0.34	0.31	1.0	34	0.34	0.31	1.0
TOTAL	**46**	**0.34**	**0.34**	**1.0**	**46**	**0.34**	**0.34**	**1.0**

[1] The Inca de Oro oxide and mixed Mineral Resource estimate was based on a total copper cut-off. The likely process route for this mineralisation is heap leach and as such only the soluble copper component of the Mineral Resource estimate has been included in this table.

The Inca de Oro (IDO) resource was not re-estimated and no further drilling or geologic modelling was completed in 2013. The resource remains unchanged from that reported as at 31 December 2012. The Carmen resource remains unchanged from that reported as at 31 December 2012. An updated Mineral Resource estimate is scheduled for completion during the June quarter 2014.

Ore Reserves

Phu Kham (including KTL)

The changes to the 31 December 2013 Phu Kham open-pit Ore Reserve when compared with the 31 December 2012 estimate largely reflect depletion due to mining and processing during 2013. Lower metal price assumptions of US$3.20/lb copper and US$1,300/oz gold were used in the 31 December 2013 estimate (US$3.50/lb copper and US$1,600/oz gold for the 31 December 2012 estimate) together with updated cost estimates derived from the 2013 life-of-mine plan and higher achieved ore processing rates.

It is noted that the Ore Reserve is relatively insensitive to copper metal price with no change to the estimated tonnage and grade at metal prices down to US$2.75/lb Cu and US$1,300/oz Au.

The addition of the KTL open-pit Ore Reserve derives from the recently completed Pre-Feasibility Study that evaluated the development of the KTL copper-gold deposit as a satellite open-pit to Phu Kham.

Phu Kham	**31 December 2013**				**31 December 2012**			
Class	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
Proved	102	0.52	0.23	1.9	124	0.51	0.23	1.8
Proved (stockpiles)	3	0.31	0.17	1.2	-	-	-	-
Probable	52	0.46	0.22	2.1	52	0.46	0.23	2.1
Sub-total	**157**	**0.50**	**0.22**	**1.9**	**176**	**0.50**	**0.23**	**1.9**
KTL								
Proved	7	1.09	0.70	3.2	-	-	-	-
Probable	1	0.94	0.46	5.2	-	-	-	-
Sub-total	**8**	**1.06**	**0.66**	**3.5**	**-**	**-**	**-**	**-**
Total Phu Kham Operations								
Proved	112	0.55	0.26	2.0	124	0.51	0.23	1.8
Probable	53	0.47	0.22	2.2	52	0.46	0.23	2.1
TOTAL	**165**	**0.52**	**0.25**	**2.0**	**176**	**0.50**	**0.23**	**1.9**

The KTL deposit has not previously been reported as part of the Phu Kham operation. The inclusion of the KTL deposit within the Phu Kham Ore Reserve has partly compensated for tonnes depleted due to mining at Phu Kham during 2013. Total Ore Reserve tonnes for the combined Phu Kham – KTL Ore Reserve as at 31 December 2013 declined by 11 million tonnes (6%) when compared with the Phu Kham Ore Reserve as at 31 December 2012 and contained copper decreased by 3%.

Ban Houayxai

The changes to the 31 December 2013 Ban Houayxai Ore Reserves when compared with the 31 December 2012 estimate largely reflect depletion due to mining during 2013; lower revenue assumptions; and reduced unit operating costs derived from the 2013 life-of-mine plan. Lower revenue assumptions result from lower metal price assumptions of US$1,300/oz gold and US$20/oz silver used in the 31 December 2013 estimate (US$1,600/oz gold and US$28/oz silver for the 31 December 2012 estimate), and updated realisation charges and metallurgical recovery factors derived from operating experience.

Ban Houayxai	31 December 2013				31 December 2012			
Class	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)
Proved	4	-	0.79	9.5	8	-	0.82	8.4
Proved (stockpiles)	2	-	0.36	2.3	1	-	0.41	1.5
Probable	30	-	0.85	8.3	33	-	0.82	8.1
TOTAL	**36**	**-**	**0.81**	**8.0**	**41**	**-**	**0.81**	**7.9**

General notes

- The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Ore Reserves.
- The Mineral Resources and Ore Reserves estimates are reported on a 100% ownership basis. PanAust has a 90% beneficial interest in Phu Kham, Ban Houayxai, KTL and LCT; a 60.45% interest in Inca de Oro; and a 100% interest in Carmen.
- The tonnes and grades are stated to a number of significant digits reflecting the confidence of the estimate. Since each number and total is rounded individually, the table may show apparent inconsistencies between the sum of rounded components and the corresponding rounded total.
- The Phu Kham Ore Reserve is estimated at commodity prices of US$3.20/lb copper and US$1,300/oz gold and reflects the non-mining break-even value of US$8.73/t processed subject to a minimum cut-off grade of 0.20% Cu.
- The KTL Ore Reserve is estimated at commodity prices of US$3.20/lb copper and US$1,300/oz gold and reflects the non-mining break-even value of US$29.30/t processed.
- The Ban Houayxai Ore Reserve is estimated at a gold price of US$1,300/oz and is reported at cut-off grades of 0.32 g/t Au for oxide material, 0.40 g/t Au for transitional material and 0.44 g/t Au for primary material.

Summary of material differences between the estimate as at 31 December 2013 and the estimate as at 31 December 2012

- Phu Kham Copper-Gold Ore Reserve and Mineral Resources: depletion relative to the 2013 data due to mining during 2013.
- KTL Copper-Gold Ore Reserve: inaugural Ore Reserve for 2014 derived from the Pre-Feasibility Study conducted during 2013 which supported a relatively high-grade development providing crushed ore to the Phu Kham process plant.
- KTL Copper-Gold Mineral Resources: The 2014 estimate utilised a 0.5% copper cut-off (2013 estimate 0.25% copper cut-off) to reflect the economics of the 2013 Pre-Feasibility study mining strategy which resulted in a reduction in overall Mineral Resource tonnes and an increase in average grades commensurate with a high-grade satellite pit development providing crushed ore to the Phu Kham process plant.

- Ban Houayxai Ore Reserve and Mineral Resources: depletion relative to the 2013 data due to mining during 2013 with the Ore Reserve estimate also reflecting lower revenue assumptions and a reduced unit operating cost.
- LCT Gold-Silver Mineral Resources: substantial increase in Measured and Indicated resources reflects the increased confidence in the estimate following inclusion in the geological database of results from resource definition drilling during 2013.

Competent Person Statements

- **Mineral Resources**

The data in this report that relate to Mineral Resources are based on information reviewed by Mr Daniel Brost who is a Member and Chartered Professional (Geology) of the Australasian Institute of Mining and Metallurgy (MAusIMM CP) and a Registered Member of the Society for Mining, Metallurgy & Exploration (SME).

Mr Brost is a full time employee of PanAust Limited. Mr Brost has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.

Mr Brost consents to the inclusion in the report of the Mineral Resources in the form and context in which they appear.

- **Ore Reserves**

The data in this report that relate to Ore Reserves are based on information reviewed by Dr Peter Trout who is a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM).

Dr Trout is a full time employee of PanAust Limited. Dr Trout has sufficient experience relevant to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.

Dr Trout consents to the inclusion in the report of the Ore Reserves in the form and context in which they appear.

Forward-Looking Statements

This announcement includes certain “Forward-Looking Statements”. All statements, other than statements of historical fact, included herein, including without limitation, statements regarding financial, production and cost performances, potential mineralisation, exploration results and future expansion plans and development objectives of PanAust Limited are forward-looking statements that involve various risks and uncertainties. There can be no assurance that such statements will prove to be accurate and actual results and future events could differ materially from those anticipated in such statements.

JORC 2012 Edition Table 1 Reporting for Mineral Resources

Phu Kham

Section 1. Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 1,493 holes for 128,000m have been drilled from surface for the Phu Kham deposit, with approximately 65% of the samples from diamond drilling, with the remaining 35% from reverse circulation. Nearly all of the reverse circulation samples are from the upper portions of the deposit and have been mined out in the gold cap operations. The majority (93%) of the samples collected from the drilling are 2m in length, the remainder vary in length from 0.2m through to 11m. Many of the longer samples are from areas that are outside the mineralised zone. QAQC samples consisting of field duplicates (additional split from reverse circulation drilling or a quarter core samples from diamond drilling), standards and blank material were included at the rate of 1 in 30. All samples were sent to ALS for sample preparation and analysis. Recent sample preparation was completed in ALS Vientiane, ALS Perth (Australia) prior, and all analysis has been completed at ALS Perth. The complete data set includes 55,180 copper samples and is consists of diamond drilling and reverse circulation drilling.
Drilling techniques	Approximately 35% of the samples were from reverse circulation drilling (RC drilling). The majority of these were relatively short holes or were pre-collars for diamond tails, the RC portion of holes extended to a length of 185m. The RC drilling consisted of a down the hole hammer and face sampling bit that was 146mm in diameter. Diamond drill holes were drilled using triple tube in two major sizes, PQ (83.1mm) and HQ (61.1mm) with minor amounts of NQ (45.1mm) samples. The average step down depth from PQ to HQ is 83m and from HQ to NQ is 333m down hole. Most of the diamond hole core was orientated using a long, thin, tapered steel rod (spear) with either a tungsten spike or RED “Chinagraph” pencil inserted in the end. The spear, which will naturally reside on the bottom side of the hole under gravity, is lowered slowly down an inclined drill hole to mark the lower edge of the core stub that will be retrieved in the following core run.
Drill sample recovery	RC sample recoveries are calculated from estimation the theoretical weight of the material collected compared with the actual weight. The theoretical weight is estimated using the following calculation – RC Sample Recovery can be calculated by dividing the measured mass (kg) by the theoretical mass (kg) using the following formulas: Theoretical Volume = π × ((drill bit diameter + 4mm)/2)^2 × h Theoretical Mass = (Density × 1000) × Theoretical Volume Where: • Drill Bit Diameter = Measured in millimetres at the start of hole, every change of bit and at the end of the hole. The diameter must be converted to meters for the calculation. • 4mm = a constant and allows for 2mm hole over break either side of

Criteria	Commentary
	the drill bit. The over break must be converted to meters for the calculation. • h = Sample interval in metres • Density = Nominated value in grams/cubic cm Diamond core recovery is calculated by geo-technicians by measuring the length of the recovered core and comparing this with the sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. The recoveries and geo-technicians mark-ups are checked by the site geologist during logging. During logging the geologist will assign the loss of core to either core loss due to drilling error, bad ground (fault material) or voids. An analysis of the assay data and recoveries does not indicate that there is any correlation of core loss to high or low grade intervals. The lithology within the mineralised zone are competent and this is reflected in the good recoveries achieved, the average recovery from all drilling programs exceeded 95%.

Logging	Each day a geologist inspects the core at the drill site and identifies the major zones and lithology. The core is then transported to the core yard using specifically designed cages to ensure the security of the samples. Once at the core yard the core is cleaned and checked for completeness and to ensure that all core block depths are correct and consistent with the daily drilling sheets.
	
	Reverse circulation chips are logged by a geologist at the drill rig during drilling. No geotechnical logging is possible on reverse circulation samples.
	
	In the core yard, geo-technicians start to geotechnically log the core applying a standard set of criteria to a high standard and level of detail. This data is recorded validated and entered into the acQuire database.
	
	The geotechnical logging captures rock quality designation (RQD) data, core run details (core recovery, core loss type, length of core within run), material properties (strength, consistency) and structures. In addition, point load testing was completed on each dry bulk density sample (cylinder); the majority of point load testing is diametrical. If axial point load testing is, required guidance is sought from the supervising geologist at the start of the drilling programme.
	
	Once the geotechnical logging is completed, site geologists log the geology, mineralisation and alteration in detail using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. This data is logged onto paper forms or entered directly into the acQuire database using tablet computers. All paper forms are entered into the acQuire database by data entry staff and then validated by the logging geologist. A summary of the drill hole, including logging is produced on finalisation of the assay data.
	
	• There are separate logging sheets to capture:
	  • Bulk density
	  • Orientation data
	  • Geotechnical data
	  • Weathering (Oxidation state)

Criteria	Commentary
	• Lithology • Alteration • Veining • Mineralisation • Structure • Magnetic Susceptibility On completion of mark up and geotechnical and geological logging, all core was photographed before sampling. Core was photographed wet and in direct sunlight. The photographs were downloaded onto the site server and are available for checking and validation.
Sub-sampling techniques and sample preparation	Reverse circulation chip samples were collected from each 1m sampling interval from the rig mounted cyclone. This sample was riffle split to produce a sample that represents 12.5% of the initial sample collected. When a field duplicate is required (1 in 30) another 12.5% sample was collected from the splitter. A reference sample is collected from the splitter, this represents 25% of the primary sample. Any wet RC chip samples were allowed to dry before any sampling using the riffle splitter was attempted. This is to reduce the potential for sample loss or contamination due to wet samples. Only a very small number of RC samples were wet (1%). All diamond drill holes were logged and photographed in the core yard before sampling. The site geologists oversee all sampling and ensure that representative samples are collected by defining the cutting line on the core. Sampling consisted of cutting the core lengthways using a diamond core saw along the predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals is placed into a pre-numbered calico bag. Where a field duplicate sample was collected (approximately 1 in 30) for diamond holes, the half core cut for assaying was cut again to produce two quarters. The left had side was sent for assay and the right hand side was sent as the duplicate sample. These represent two separate samples sent to the laboratory. Samples were sent to ALS in Perth for sample preparation until 2005, since then all sample preparation was completed at ALS in Vientiane. The sample preparation process is an ALS standard process used for all drill hole samples, it includes- • Drying at 1100C in LPG gas ovens controlled by thermostat. • Crushing to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. More than 5% of the samples are quality control tested to ensure sizing conformance. • Samples received with a weight greater than 1.2kg were split to 1-1.2kg using the adjustable rotary sample divider. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm. More than 5% of the samples are quality control tested to ensure sizing conformance. Holes drilled in 1996 and 1997 have unknown laboratory methods and no QAQC results. Given the unknown nature of the analysis or quality of these samples, these

Criteria	Commentary
	holes were excluded from Resource estimation. This data set consists of 64 holes; the majority of these holes that are within the mineralised area are covered by new drilling.

Quality of assay data and laboratory tests	All sample analysis was completed at ALS laboratories in Perth (Australia). Base metals were analysed in Perth by ME-ICP41 (aqua-regia acid digestion inductively coupled plasma - atomic emission spectroscopy (ICP -AES)) with a sample charge of 20g. The lower detection limit for copper and silver are 0.2g/t and 1g/t while the upper detection limits are 100g/t and 10,000g/t respectively. ME-OG62, (four acid digest analysed by inductively coupled plasma - atomic emission spectroscopy) method was used to analyse higher grade silver, copper and zinc (greater than 100g/t) for accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively.
	Gold was analysed by ALS at the Perth laboratory using the Au-AA26 fire assay method analysed by atomic absorption spectroscopy with a 50g sample charge. This method has a lower detection limit of 0.02g/t and upper detection limit of 100g/t. When gold is greater than 100g/t the Au-AA26-DIL method is used (lower detection of 1g/t).
	Cyanide leach for copper is triggered when the copper result is greater than 0.05%. The AA17 (cyanide leach Atomic absorption spectrometry (AAS)) method was used with a 200g sample charge. The lower detection limit for the AA17 method for copper is 0.01% while the upper detection limit is15%.
	The suit of elements analysed include:
	• Au-AA26
	  • Gold
	• MEICP41
	  • Silver, arsenic, bismuth, copper, iron, potassium, lead, sulphur, sulphur and zinc
	• Cu-PK06 package
	  • Total copper, cyanide copper, soluble copper, residual copper
	• Cu-AA17 (0.05% copper trigger)
	  • Cyanide leach for copper.
	All of the analytical techniques applied are considered total or near total.
	The company inserted certified standards, blanks and field duplicates every 30m (10%) in that order.
	Moungcha limestone was used as a blank sample for this project. Blank samples are inserted to check for contamination in field sampling, laboratory's sample preparation and analysis, the result of the blank material should be below detection limits.
	The base metal and gold standards used were sourced from Geostats Pty Ltd and Gannet Holdings with copper certified values ranging between 2897g/t and 13903g/t. Standard reference materials are used to check accuracy and bias of the analytical method, the results should be very similar to the standard concentration for the specific standard.
	QAQC samples were monitored on a batch-by-batch basis. An assay batch is accepted if the blank samples are within the acceptable limits (5 times the lower detection limit) and the standards are within the ± 3SD (standard deviations). One failed standard can cause rejection if the results around the failed standard are not in normal grade range. A batch is also re-assayed when several standards assay results are outside the acceptable limits. The inserted blank materials did not show any consistent issues with

Criteria	Commentary
	sample contamination. Any batches that were rejected based on the QAQC results were re-assayed at ALS, the subsequent results were again reviewed, and if acceptable were loaded into the acQuire database as the primary sample result. The inserted blank material did not show any consistent issues for gold and silver, however there were a few (37%) copper results that were greater than the upper limit (10 times the detection limit). These results most likely indicate that the standard (Moungcha Limestone) contains minor amounts of copper, as any contamination issues would have also been indicated in gold and silver. The company certified standards did not indicate any consistent bias, three out of the four standard returned results that were within the acceptable limits (3SD from the standard value), with the less than 1% of the results falling outside the acceptable limits for the fourth. All gold standards were generally within acceptable limits (3SD from the standard value), with a small percentage of results falling outside these limits. The performance of field duplicates in core samples is good for copper and gold. Silver results indicate some variability in the results; this may be due to the style of mineralisation, or issues with the imitations of the analysis method. The method for determining silver results is aqua-regia acid digestion inductively coupled plasma (MEICP41). The preferred method for silver assay is Atomic Absorption Sepctroscopy (AAS). This introduces some uncertainty in the results for silver. However project to date reconciliation indicated that the mill (analysis of the mill feed) is achieving a silver grade that is 8% higher than the estimated grade in the Resource model. The ALS laboratory also inserted QAQC samples to internally test the quality of the analysis. These results are received with the assay results in each batch. The ALSQAQC included standards, blanks and duplicates for independent quality control. The results of the lab standards were also monitored on a batch to batch basis by the Data and Resources geologist. The results did not show any issues with the laboratory.
Verification of sampling and assaying	Round Robin analysis for 490 samples assayed between 2011 and 2012 was conducted in 2013. Check pulps were submitted to ALS Perth, SGS Perth, Intertek Indonesia and Standard Reference Lab Perth. The pulps were analysed for gold with fire assays and copper using as close as possible to the same analysis methods as the initial ALS laboratory (MEI-CP41 and ME-OG62). Correlation for gold ranged between 0.96 and 0.97 while copper had correlation between 0.93 and 0.99. The check samples and the original did not show any bias though an improvement in pulp splitting and sample preparation was recommended for better precision. All assay data was accepted into the database as supplied by the laboratory. Where a duplicate sample was taken (quarter core), the original and duplicate assays were averaged to maintain the sample support of half core.
Location of data points	Drill hole collars were set-up using Garmin 76S hand held GPS units on WGS84 UTM Zone 48N. During drilling, or once completed the hole collar pick-up was completed by the Phu Kham Mine survey department using the Total Station method on WGS84_48N. A down hole survey was completed every 30 m, using a Reflex single shot camera, Eastman single shot camera, Proshot or a simple shot camera. The majority of the surveys were completed using a Reflex single shot camera, which was used exclusively for diamond holes. RC holes were generally surveyed with a Simple Shot camera. The

Criteria	Commentary
	down hole surveys were collected and examined each day by the site geologist. Any surveys that were spurious were retaken. Once validated the surveys were entered into the acQuire data base by data entry personnel. All down hole cameras were tested once a week in the presence of a geologist during the drilling programme using an on-site reliability jig. Current topography is determined by the Phu Kham mine survey department.
Data spacing and distribution	The nominal drill hole spacing within the oxide gold portion of the deposit is 25x25m. The nominal drill hole spacing within the main copper portion of the deposit is 50x75m.The spacing is quite consistent through the main resource area however the spacing may extend out to 100x100m in some areas of the deposit. The drill hole spacing is sufficient to estimate the grades in the deposit; the mineralisation is reasonably consistent within domains. The definition of the 0.9% copper zones are limited by the drill hole spacing, however this should only have a local impact on the estimated grades. The Resource Classification applied to the Resource model reflects the variability in drill hole spacing and confidence in the estimation. All samples were composited to 2m lengths due to the high proportion of 2m samples (93%).
Orientation of data in relation to geological structure	The majority of drill holes that pass through the mineralised zones are drilled to the east (090deg) at a dip of 60deg. These holes are close to perpendicular to the mineralised packages. A secondary direction is to the west (270deg); these holes target the western boundary of the mineralised zone and basement. The majority of short holes are drilled vertically; the holes targeted the gold cap. This material has been removed by mining. The direction of the drilling was designed to best capture the variability of grade; the structural measurements from orientated core enabled an analysis of the controls of mineralisation and structures to ensure that the direction of drilling was optimal. Some recent drill holes have been drilled in an opposing direction to confirm the location of the western basement contact.
Sample security	Drill core and reverse circulation samples were picked up from the drilling site on a daily basis and transported to the core yard. The geologist recorded the intervals of core being picked and some completed brief notes on the main features before the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is under 24 hour security guard. All logging and sampling of diamond drill core was completed in the core yard while RC logging is completed at the rig and samples are bagged in the field and delivered to the core yard. All samples to be dispatched were packed in rice bags, clearly labelled with a submission number, project and prospect number, sample type and date of dispatch. The sample dispatches were accompanied by supporting documentation signed by the geologist and showing the sample submission number, analysis suite, and number of samples. ALS picks up samples at the core yard and transport to the Vientiane laboratory by ALS employees for sample preparation. On site, prior to dispatch, a log book is signed by an ALS personnel and the site geological technicians. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples.

Criteria	Commentary
	The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators in Vientiane and loaded into acQuire through an automated process. QAQC on import is completed before the results are finalised.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects. AMC (Brisbane) completed a desktop Resource review in 2008; this study included a review of the drilling, survey, sampling, logging and analytical inputs, QAQC, interpretation and 3D modelling of the Phu Kham Resource. This study did not identify any high risk issues. In 2013 BDA Minerals Industry Consultants completed an independent audit of the Phu Kham and Ban Houayxai projects. This was a high level review that was a post completion audit for the increased recovery project, however the report included commentary on the confidence in the reliability of the drill hole data set (post 2000).

Section 2. Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited ('Phu Bia Mining'). The Government of the Lao PDR holds a 10% interest in Phu Bia Mining. Phu Bia Mining's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 square kilometres, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.
Exploration done by other parties	No significant exploration has been completed by other companies on the Phu Kham project area. The prospect was discovered between 1996 and 2004 by Phu Bia Mining (PBM), a Lao-registered company 90% owned by PanAust.
Geology	The Phu Kham deposit is thought to represent the distal aspect of a conventional copper-gold porphyry system with mineralisation occurring in skarn and stock work styles. The system occurs in a sequence of deformed, hydrothermal altered tuffs, volcaniclastics and carbonate sediments of Permo-Carboniferous age. The most abundant skarn is banded to massive pyrite. Mineralisation in the pyrite skarns occurs as disseminated grains of bornite and chalcopyrite with traces of sphalerite and galena. Stockwork mineralisation occurs in the altered tuffs, volcaniclastic sediments and porphyry dykes most of the mineralisation occurs as fine fracture networks overprinting

Criteria	Commentary
	an early set of grey granular quartz veins.
Drill hole Information	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Relationship between mineralisation widths and intercept lengths	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Orientation of data in relation to geological structure".
Diagrams	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.

Section3. Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	AcQuire using a Microsoft SQL Server 2005 running the acQuire data model (3500) is the database used for storing all drill hole data. The central database is located in the Vientiane office (capital of Lao). A recent database audit was completed in July 2013 by acQuire Technology solutions. No potential risks with the database structure or application were reported. Historical data was manually entered using an acQuire data entry form, and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using Motion computing tablets computers with acQuire offline data entry forms. The logging data was electronically transferred to the

Criteria	Commentary
	site server and validated by the geologists. After validation on site, data was routinely synchronised with the central acQuire server in Vientiane. Assay results were sent electronically from the laboratory directly to the Database Administrator in Vientiane for validation and loading into acQuire, no manual data entry or intervention is required. The Data Resources Geologist checks the QAQC of the received assays and advised the Databases Administrator to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, Data and Resources geologist and if required queries are made with ALS. After a batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. Significant intercept tables for the accepted assay data were produced and distributed as a further confirmation. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste interval of 4m. The geologists on site reviewed the QAQC data and the received assays against the logging intervals. The received assays were reviewed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. Checking of assay data within the acQuire database indicate that there are no systematic issues with the importing, storage or extraction of assay data from the acQuire database. The site server and the Vientiane server are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. A high degree of confidence in the geological interpretation can be gained by the minor adjustments required by subsequent drilling programs. The recent drilling program resulted in very minor changed to the geology model. This is further supported by observations of the geology in the pit. Site geologists developed a sectional interpretation every 25m for the major lithology and structures. Alteration, structure, surface mapping, mineralisation style and the assay values were used to generate the major lithological domains. The sections are confirmed with plan and long section interpretations. The sectional interpretations along with structural information were used to generate a 3D model. Site generated a series of surfaces to represent the weathering profile (base of leached zone (BOCO), base of oxide (BOX), base of upper transitional (BOCC) and base of lower transitional (BOT)). This defines the Leached, Oxide, Transitional (upper and lower) and Primary zones. The weathering surfaces were determined by a combination of geological logging, cyanide soluble copper assaying and consideration of sulphur

Criteria	Commentary
	grades. A series of lithological solids were generated on site to represent the mineralised package and different basement lithology. To best represent the higher grade zone, nominally the skarn lithology, a series of 0.9% solids were generated. These were completed in section using the drill hole sample grades incorporating the grade control drilling and pit mapping.
Dimensions	The main mineralised zone is approximately 1,700m long striking to the North, is approximately 200m in width and dips shallowly to steeply to the east to approximately 300m below the surface.
Estimation and modelling techniques	Copper occurs as Chalcopyrite and Bornite in both the Skarn and stockwork mineralisation. These minerals occur as disseminations within the skarns and as a stockwork in the mineralised Tuffs. The domaining for estimation closely represented the style of mineralisation seen, with additional criteria of weathering and a significant difference in the orientation of mineralisation north and south. The Resource model extents cover the mineralised zone and extend to include waste material that may be included in a pit shell. The model consists of 20x20x10m (X, Y, Z) blocks that are sub-blocked to 5x5x5m (X, Y, Z) on the mineralisation solid boundaries. The Resource estimate used a combination of the mineralisation, lithology and weathering zones to define 35 domains to control the estimation. The 35 estimation domains consist of eight lithology units cut into north and south zones (representing the different orientations of grade continuity) and then cut by weathering zones (leached, oxide, upper transitional, lower transitional and primary). Some of these domains were combined due to geostatistically similar characteristics. Some of these domains shared estimation parameters due to a lack of data to determine reasonable parameters. The majority of these were as considered as hard boundaries; the only soft boundary was in the southern zone when estimating outside the 0.9% mineralisation solids. This was considered a soft boundary due to the uncertainty in modelling these highly variable thin zones. Data validation and preliminary statistical work was completed on the drill holes and geology models. This data was provided to AMC (Brisbane to complete further statistical work and data analysis and Resource estimation. The Resource estimate was completed within CAE Studio 3 (Datamine) utilising Ordinary Kriging (OK) as the estimation method. Multiple domains (35) were used to control the estimation and account for different grade distribution and mineralisation characteristics. The Leached zone, Oxide/Upper transitional, Lower transitional and Primary zones were estimated as a separate domains. The Oxide and Upper transitional zones were combined mainly due to a lack of data in the separate domains. The lithology and copper mineralisation (0.9% copper solids) were cut by the weathering zones. In addition to this, there was a major separation on the background and high grade mineralised zones north and south. These major zones were separated by a surface that ran north/east-south/west approximately through the centre of the deposit. This represented a significant change in the orientation of the copper mineralisation. The orientation of the continuity of mineralisation changed from moderately west dipping in the southern zone to sub-horizontal in the north. These were critical components to control the estimation. A boundary analysis on these solids indicated

Criteria	Commentary
	that there was a significant change in grade across this boundary. Supervisor (Snowden software) and Isatis were used for data analysis and variogram modelling for all 35 domains, in the end some of the domains shared variogram models due to similarities and/or a lack of data to define well formed variograms. However the domains restricted the samples available for estimation. In most cases the variograms were well formed (where there was sufficient data support), where this was not possible omni-directional variogram models were produced, or variogram models from other related domains were applied. The average nugget effect applied to the copper variogram models was 37%. The range of the variogram model averaged 250m for the mineralised domains. A correlation matrix was generated for all metals in the major domains to determine any possible correlations. The analysis indicated that there was no strong correlation copper and any other metal. Kriging Neighbourhood Analysis (KNA) was undertaken to better define the estimation parameters. These tests used the Kriging Efficiency, Slope of Regression and Kriging Variance to assisted in defining the best parameters for- • Block size • Search distances • Minimum and maximum number of samples to be used for an estimate • Octant restraints on the sample selection Search distances and directions vary for each domain; they range in maximum values between 60m and 150m, and 20m to 60m in the short axis. In the high grade zones the search will be constrained by the solid more than the range of the search ellipse. The shape and direction of the copper searches vary for the domains. The ellipse in the high grade zone follows the major direction of the solids (moderate westerly dip in the south and near horizontal in the north). Outside the high grade zone the main direction of continuity also follows the same orientation, so the search ellipse maintains this orientation. The ellipse orientation was varied for some domains in the oxide and transitional zones. The number of samples used during estimation to inform a block ranged from a minimum of 4 to a maximum of 16. A larger number of samples were considered, however the data indicated that this had a negative effect on the quality of the estimate. The estimate was completed using ordinary kriging in three separate estimation passes. Each pass used larger searches and more relaxed minimum/maximum sample numbers. This was done to ensure that distal blocks were estimated. The estimation pass was a consideration when assigning a Resource classification. Second and third estimated blocks potentially received a lower confidence classification. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tool was the use of SWATH plots. These are plots of the average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Both validation tools indicated that the Resource estimate was a fair representation of the sample grades, given the estimation process and the distribution of samples. Comparisons of the estimated Resource model with previous models indicated that the changes in the tonnes, grades and classification were consistent with the impact of the

Criteria	Commentary
	additional drilling and changes to the mineralisation solids and estimation parameters.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut off used for reporting was selected on economic factors and the modelled recoveries. The selected cut off is supported by several years of mine operation. The cut offs applied for mining operations in 2013 changed mid-year in response to changing economic criteria. For the first half of the year a 0.15% copper cut off was applied and for the remainder of the year a 0.2% copper cut off was used. The cut off is not impacted by the weathering zones.
Mining factors or assumptions	The Phu Kham open pit is mined as a conventional truck and shovel operation with 10 m high benches. The majority of the ore and waste material is drilled and blasted before being excavated by hydraulic shovels and excavators. There has been no change to mining practice and mining practices are well established and appropriate to the style of deposit. Project to date reconciliation figures indicate that the Mill has achieved a lower tonnage (5% lower) and lower copper grades (2% lower). These variances are well within the expected limitations of a Resource to Mill actual reconciliation.
Metallurgical factors or assumptions	The estimated recovery of copper is related to the copper and sulphur head grades, at the deposit copper grade of 0.5%, the recovery is predicted to be approximately 75%. This assumes a long term average ore processing rate of 18Mtpa and 23% copper grade in concentrate. Black shale material was excluded from the Resource due to the poor recovery of copper making it uneconomic.
Environmental factors or assumptions	Existing governmental approvals, permits and infrastructure at the Phu Kham operation will be capable of dealing with the expected by-products of mining the reported material.

Bulk density	Bulk density determinations were undertaken at three separate points, complete core tray determination, calliper measurements and water immersion by ALS. Each method provides a different set of data that had its own bias. The least bias data set is considered to be the ALS water immersion. Over 60% of the samples have an ALS bulk density value, every sample has an ALS bulk density value from the 2011 drilling onwards (within the mineralised zone).
	
	• The ALS method (OA-GRA08) consists of-
	    • Sample was dried
	    • Placed into a stocking
	    • Weighed in air
	    • Weighed in water
	
	No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal.
	
	An analysis of the density data failed to find strong correlations with mineralisation, weathering or lithology. However a combination of weathering and lithology generally showed good separation of data. This combination was used as the basis to assign density to the Resource model. Average bulk density values for the 26 estimation

Criteria	Commentary
	domains were derived, these have been assigned to corresponding blocks in the Resource model. Preference was given to ALS density values where available, then the calliper measurements were used. Any extreme and unrealistic values were not used to generate the average value.
Classification	The Resource classification assigned to the model considered the drill hole spacing, geological controls, grade continuity and the robustness of the estimate. The robustness of the estimate is primarily determined by the estimation pass, the lower the pass number (1-3) the more robust the estimate. The kriging variance and slope of regression values that are produced by the estimation process are also taken into account as a guide to the robustness of the estimate. These considerations were used by the estimation geologist to generate solids representing the confidence in the estimate. These solids were used to assign the Resource classifications to the model. Blocks classified as measured are generally made up of blocks that were estimated in pass 1 and were in the region of drilling that was on 25m to 75m spacing. Indicated blocks were mostly pass 1 and pass 2 and/or outside the Measured zone with an approximate drill hole spacing of 75m. Blocks supported by a drill hole spacing of 100m were generally classified as Inferred. Nearly all of these blocks were estimated in the first two passes.
Audits or reviews	The Resource estimate was completed by an external consultancy, AMC consultants Brisbane. The work and resulting Resource model was reviewed both internally by AMC and within PanAust. In 2013 BDA Minerals Industry Consultants completed an independent audit of the Phu Kham and Ban Houayxai projects. This was a high level review that was a post completion audit for the increased recovery project, however the report included commentary on the confidence in the geology and estimation. In 2011 AMC consultants Brisbane conducted a review of the reconciliation at the Phu Kham mine. Significant changes to the site processes and reconciliation process have been completed since this review.
Discussion of relative accuracy/ confidence	The nature of bulk density samples introduces a risk to the estimated tonnages. The three methods used to determine bulk density values all have an implied bias, the ALS data set is the best representation. The actual results and the assignment of bulk to the classified mineralisation has an associated risk. Mining data to date does potentially suggest that improvements in this process can be made. Another risk is the modelling of the skarn zone (0.9% copper solid). The impact of the possible variability of these solids potentially will be restricted to local variations and should not impact greatly on the global Resource. These local variations will be identified before mining by the close spaced grade control drilling and modelling. The majority of the mineralised material within the current pit shell is well supported by drilling, any changes in the mineralisation or lithology solids should only have a localised impact and not alter the overall Resource.

Ban Houayxai

Section 1. Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 891 holes for 125,129m have been drilled from surface for the Ban Houayxai deposit with approximately 74% of the samples from diamond drilling and the remaining 26% from reverse circulation drilling. The majority (79%) of samples for this project are 1m in length; however, sample length range from 0.2m through to 5m for samples outside the mineralised zone. QAQC samples consisting of field duplicates (additional split from reverse circulation drilling or a quarter core samples from diamond drilling), standards and blank material were included at the rate of 1 in 30. The half core samples from diamond drilling averaged 5kg in weight, the reverse circulation samples averaged 27kg. All of these samples were sent to ALS Perth (Australia) for sample preparation and analysis. Recent sample preparation was completed in ALS Vientiane, ALS Perth prior, and all analysis has been completed at ALS Perth. The complete data set includes 97,250 gold samples and consists of diamond and reverse circulation samples.
Drilling techniques	Approximately 26% of the samples were from reverse circulation drilling (RC drilling), with the majority being short holes, measuring less than 100m in length. Six of these holes were pre-collars for diamond tails, the RC portion was up to 84m in depth in 5 holes, one hole had a 200m pre-collar. The RC drilling consisted of a down the hole hammer and face sampling bit that was 146mm in diameter. Diamond drill holes were drilled using triple tube in two major sizes, PQ (83.1mm) and HQ (61.1mm) with minor amounts of NQ (45.1mm) samples. The average step down depth from PQ to HQ is 50m and from HQ to NQ is 170m down hole. Most of the diamond hole core is orientated using a long, thin, tapered steel rod (spear) with either a tungsten spike or RED "Chinagraph" pencil inserted in the end. The spear, which will naturally reside on the bottom side of the hole under gravity, is lowered slowly down an inclined drill hole to mark the lower edge of the core stub that will be retrieved in the following core run. Recent diamond drilling has been orientated using a Reflex ACT II RD tool. The Reflex ACT II RD is a digital core orientation device which comprises two components, one residing within the inner tube and backend assembly and the other as a core barrel extension. Accuracy is reportedly better than 1° while orientation can be still achieved in drill holes up to an inclination of -88°.
Drill sample recovery	RC sample recoveries are calculated from estimation the theoretical weight of the material collected compared with the actual weight. The theoretical weight is estimated using the following calculation – *RC Sample Recovery can be calculated by dividing the measured mass (kg) by the theoretical mass (kg) using the following formulas:* *Theoretical Volume =* $\pi \times ((\text{drill bit diameter} + 4\text{mm})/2)^2 \times h$ *Theoretical Mass = (Density × 1000) × Theoretical Volume* *Where:* • *Drill Bit Diameter = Measured in millimetres at the start of hole, every change of bit and at the end of the hole. The*

Criteria	Commentary
	diameter must be converted to meters for the calculation. • *4mm = a constant and allows for 2mm hole over break either side of the drill bit. The over break must be converted to meters for the calculation.* • *h = Sample interval in metres* • *Density = Nominated value in grams/cubic cm* Diamond core recovery is calculated by geo-technicians by measuring the length of the recovered core and comparing this with the sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. The recoveries and geo-technicians mark ups are checked by the site geologist during logging. Any loss of core is assigned to either core loss due to drilling error, bad ground (fault material) or voids. An analysis of the assay data does not indicate that there is any correlation of core loss to high or low grade intervals. The lithology within the mineralised zone is competent and this is reflected in the good recoveries achieved, the average recovery from all drilling programs exceeded 98%.

Logging	Each day a geologist inspects the core at the drill site and identifies the major zones and lithology. The core is then transported to the core yard using specifically designed cages to ensure the security of the samples. Once at the core yard the core is cleaned and checked for completeness and to ensure that all core block depths are correct and consistent with the daily drilling sheets.
	
	Reverse circulation chips are logged by a geologist at the drill rig during drilling. No geotechnical logging is possible on reverse circulation samples.
	
	In the core yard, geo-technicians start to geotechnical log the core applying a standard set of criteria to a high standard and level of detail. This data is recorded validated and entered into the acQuire database.
	
	The geotechnical logging captures rock quality designation (RQD) data, core run details (core recovery, core loss type, length of core within run), material properties (strength, consistency) and structures. In addition, point load testing was completed on each dry bulk density sample (cylinder), the majority of point load testing is diametrical. If axial point load testing is, required guidance is sought from the supervising geologist at the start of the drilling programme.
	
	Once the geotechnical logging is completed, site geologists log the geology, mineralisation and alteration in detail using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. This data is logged onto paper forms or entered directly into the acQuire database using tablet computers. All paper forms are entered into the acQuire database by data entry staff and then validated by the logging geologist. A summary of the drill hole, including logging is produced on finalisation of the assay data.
	
	• There are separate logging sheets to capture-
	  • Bulk density
	  • Orientation data
	  • Geotechnical data
	  • Weathering (oxidation state)

Criteria	Commentary
	• Lithology • Alteration • Veining • Mineralisation • Structure • Magnetic Susceptibility On completion of mark up and geotechnical and geological logging, all core was photographed before sampling. Core was photographed wet and in direct sunlight. The photographs were downloaded onto the site server and are available for checking and validation.
Sub-sampling techniques and sample preparation	RC chip samples were collected from each sampling interval from the rig mounted cyclone. This sample was riffle split to produce a sample that represents 12.5% of the initial sample collected. Another 25% sample is retained as a reference sample and when required (1 in 30) another 12.5% sample was collected as a field duplicate. Any wet RC chip samples were allowed to dry before any sampling using the riffle splitter was attempted. Only a very small number of RC samples were wet (1%). All diamond drill holes were logged and photographed in the core yard before sampling. The site geologists oversee all sampling and ensured that representative samples are collected by defining the cutting line on the core. Sampling consisted of cutting the core lengthways using a diamond core saw along the predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals is placed into a pre-numbered calico bag. Where a field duplicate sample was collected (approximately 1 in 30), the half core for assaying was cut again to produce two quarters. The left had side was sent for assay and the right hand side was sent as the duplicate sample. Sample preparation was done at ALS Vientiane as follows: • All samples were dried at 110^{0}C in LPG gas ovens controlled by thermostat. • All RC and core samples were crushed to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. More than 5% of the samples are quality control tested to ensure sizing conformance. • Samples received with a weight greater than 1.2kg were split to 1-1.2kg using the adjustable rotary sample divider. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm. More than 5% of the samples are quality control tested to ensure sizing conformance.
Quality of assay data and laboratory tests	All the analytical techniques used are considered total through four acid digestion or sample fusion. Gold was analysed by ALS at Perth (Australia) using the Au-AA26 fire assay with atomic absorption spectroscopy method with a 50g sample charge. This method has a lower detection limit of 0.02g/t and upper detection limit of 100g/t. When gold is greater than 100g /t the Au-DIL (gold by dilution) method is used (lower detection of 1g/t). A cyanide leach analysis for gold is triggered by a gold result that is greater than 0.10g/t. Before 2009 the ALS method AA13 (cyanide leach Atomic absorption spectrometry (AAS)) was done in ALS Perth (30g charge), after 2009 AA15ve

Criteria	Commentary
	accelerated cyanide leach was used with a 200g sample charge. The lower detection limit for the AA15ve method for gold and copper is 0.02g/t and 0.1g/t while the upper detection limits are 300g/t and 20% respectively. Base metals were analysed in Perth by ME-ICP41 aqua-regia acid digestion with Inductively coupled plasma (ICP) method with a sample charge of 20g. The lower detection limit for silver and copper are 0.2g/t and 1g/t while the upper detection limits are 100g/t and 10,000g/t respectively. ME-OG62 (four acid digest analysed by inductively coupled plasma - atomic emission spectroscopy), was used to analyse higher grade copper, lead and zinc (greater than 100g/t) for better accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively. The Ban Houayxai analysis suite includes ALS methods Au-AA26 for gold and ME-ICP41 (aqua-regia acid digestion inductively coupled plasma - atomic emission spectroscopy (ICP -AES)) for silver, arsenic, copper, mercury, lead, sulphur, antimony and zinc. A trigger of gold greater than 0.10g/t is used to select samples for AA15ve and AA15ve. The company inserted certified field duplicates, blanks and standards every 30m (10%) in that order. Three types of blanks have been used since 2006, Mekong sand, Moungcha limestone and Vang Vieng limestone. Blank samples are inserted to check for contamination in field sampling, laboratory's sample preparation and analysis, the result of the blank material should be below detection limits. The gold and base metal standards were sourced from Geostats Pty Ltd and Gannet Holdings with gold certified values ranging between 0.21g/t and 3.14g/t. Standard reference materials are used to check accuracy and bias of the analytical method, the results should be very similar to the standard concentration for the specific standard. QAQC samples were monitored on a batch-by-batch basis. An assay batch is accepted if the blank samples are within the acceptable limits (5 times the lower detection limit) and the standards are within the $\pm$ 3SD (standard deviations). One failed standard can cause rejection if the results around the failed standard are not in the normal grade range. A batch is also re-assayed when two or more standards assay results are outside the acceptable limits. The inserted blank materials did not show any consistent issues with sample contamination. 97% of the gold standards assays were within acceptable limits with a slight bias low. 55% of the gold standards show a low bias though they are within the acceptable limits. ALS is investigating the low bias in gold standards. 99% of the base metal standards all performed well, with results within the expected limits. The performance of field duplicates in core samples is generally reasonable and the variations are related to the style of mineralisation. ALS conducted test work on the low precision of the gold duplicates and found that the deposit contained coarse gold and may explain the variance in duplicates assays. A QAQC review by AMC also confirmed the presence of coarse gold and suggested that changes to the sub-sampling processes would improve the results achieved. Further work is being considered to determine the optimum sub-sample size to ensure the best representation of a sample. The ALS laboratory also inserted QAQC samples to internally test the quality of the analysis. These results are received with the assay results in each batch. The ALSQAQC included standards, blanks and duplicates for independent quality control. The results of the lab standards were also monitored on a batch to batch basis by the Data and

Criteria	Commentary
	Resources geologist. The results did not show any issues with the laboratory.
Verification of sampling and assaying	Round Robin analysis for 490 samples assayed between 2011 and 2012 was conducted in 2013. Check pulps were submitted to ALS Perth, SGS Perth, Intertek Indonesia and Standard Reference Lab Perth. The pulps were analysed for gold with fire assays and copper using as close as possible to the same analysis methods as the initial ALS laboratory (MEI-CP41 and ME-OG62). Correlation for gold ranged between 0.96 and 0.97 while copper had correlation between 0.93 and 0.99. The check samples and the original did not show any bias though an improvement in pulp splitting and sample preparation was recommended for better precision. Parts of eleven reverse circulation and diamond drill holes were twinned. The comparison of the assays between these holes showed the similar trends with no major differences down hole in grade variation. The original half core assay results were compared additional with half core duplicates (total core consumed) to check for grade variance between original assays and duplicate values to determine if the sample size is sufficient for the style of mineralisation. Five holes were selected for the exercise, four were in high grade zones in historical holes while one hole plotted adjacent to a historical hole which had high grades. The log correlation coefficient of the original and half core duplicates is 0.91 which suggest that the results generally illustrate a fair repeatability. However, variances in grade were noted and were related to mineralisation style (veining). All assay data was accepted into the database as supplied by the laboratory. Where a duplicate sample was taken (quarter core), the original and duplicate assays were averaged to maintain the sample support of half core.
Location of data points	Before 2009, drill hole collars were set-up using Garmin 76CS, after this a Garmin 60CSx was then used using WGS84 UTM Zone 48N. On completion of drilling collars were surveyed using differential GPS (DGPS). For recent holes, collar pick-up was completed by the BHX Mine survey using DGPS Trimble R8 GNSS on WGS84 datum, UTM Zone 48N. Approximately 176 collar locations were validated by AAM (Thailand) Co. Ltd. during a 2009 topographical survey. A number of inconsistencies were identified with the survey of these holes; these and the deficiencies in the process were corrected. A down hole survey was completed every 30 m, using a Reflex single shot camera, Eastman single shot camera, Proshot or a simple shot camera. The majority of the surveys were completed using a Reflex single shot camera, which was used exclusively for diamond holes. RC holes were generally surveyed with a Simple Shot camera. The down hole surveys were collected and examined each day by the site geologist. Any surveys that were spurious were retaken. Once validated the surveys were entered into the acQuire data base by data entry personnel. All down hole cameras were tested once a week in the presence of a geologist during the drilling programme using an on-site reliability jig.
Data spacing and distribution	The majority of the drill holes are spaced on a 50x50m grid, the spacing reduces to 25x25 in some areas, particularly in the upper zones. The drill hole spacing adequately enables the estimation of the Resource on a medium scale, there will be small scale variability that the drilling will not identify, however close spaced grade control drilling (5x10m) to date has indicated that the Resource model is a fair representation. The classification of the Resource model reflects the variability of the drill hole spacing at

Criteria	Commentary
	Ban Houayxai. Sample data was composited to 2m for estimation, the mean raw sample length was 1.3m, a 2m composite was considered appropriate for the raw sample length and the mineralisation style.
Orientation of data in relation to geological structure	60% of the holes were drilled at an inclination of -60 deg drilled to the south (180deg), these were designed to best capture the interpreted dip and strike of the mineralisation. Diamond drilling is angled to be perpendicular to the trend of lithology and structure, to as much as possible obtain true width samples. Drilling before 2009 were in various directions due to uncertainty in the orientation of the mineralisation and structures. In 2009 an update and review of the geology, mineralisation and structures was completed that enabled the current (optimum) drilling direction to be determined. Structural measurements from the orientated core enabled an analysis of the controls of mineralisation and structures to continuously evaluate the direction of drilling to ensure that it was optimal.
Sample security	Drill core and reverse circulation samples were picked up from the drilling site on a daily basis and transported to the core yard. The geologist recorded the intervals of core being picked and some completed brief notes on the main features before the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is under 24 hour security guard. All logging and sampling of diamond drill core was completed in the core yard while RC logging is completed at the rig and samples are bagged in the field and delivered to the core yard. All samples to be dispatched were packed in rice bags, clearly labelled with a submission number, project and prospect number, sample type and date of dispatch. The sample dispatches were accompanied by supporting documentation signed by the geologist and showing the sample submission number, analysis suite, and number of samples. ALS picks up samples at the core yard and transport to the Vientiane laboratory by ALS employees for sample preparation. On site, prior to dispatch, a log book is signed by an ALS personnel and the site geological technicians. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples. The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators in Vientiane and loaded into acQuire through an automated process. QAQC on import is completed before the results are finalised.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects. AMC Brisbane reviewed the Ban Houayxai QAQC data in January 2010; recommendations included reviewing the procedures for sub-sampling. H&S consultants have reviewed the data as part of the preparation steps for Resource estimation. Any discrepancies found were corrected within the acQuire database.

Section 2. Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited ('Phu Bia Mining'). The Government of the Lao PDR holds a 10 % interest in Phu Bia Mining. Phu Bia Mining's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 square kilometres, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.
Exploration done by other parties	No significant exploration has been completed by other companies on the Phu Kham project area. The prospect was discovered between 1996 and 2004 by Phu Bia Mining (PBM), a Lao-registered company 90% owned by PanAust.
Geology	The Ban Houayxai deposit is a narrow vein, structurally controlled Gold - Silver deposit of Permo-Carboniferous age (285Ma). Mineralised veins are predominantly hosted within intermediate volcanics which have been subject to greenschist facies metamorphism and structurally juxtaposed against a siliciclastic package that is probably of lower metamorphic grade. The siliciclastic package contains mineralisation hosted within bodies of volcaniclastic siltstone and volcanogenic sandstone.
Drill hole Information	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Relationship between mineralisation widths and intercept lengths	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Orientation of data in relation to geological structure".
Diagrams	No exploration results are included in this release, all information relates to the Mineral

Criteria	Commentary
	Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.

Section3. Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	AcQuire using a Microsoft SQL Server 2005 running the acQuire data model (3500) is the database used for storing all drill hole data. The central database is located in the Vientiane office (capital of Lao). A recent database audit was completed in July 2013 by acQuire Technology solutions. No potential risks with the database structure or application were reported. Historical data (approximately 7% of the data set) was manually entered using an acQuire data entry form, and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using Motion computing tablets computers with acQuire offline data entry forms. The logging data was electronically transferred to the site server and validated by the geologists. After validation on site, data was routinely synchronised with the central acQuire server in Vientiane. Assay results were sent electronically from the laboratory directly to the Database Administrator in Vientiane for validation and loading into acQuire, no manual data entry or intervention is required. The Data Resources Geologist checks the QAQC of the received assays and advised the Databases Administrator to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, Data and Resources geologist and if required queries are made with ALS. After a batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. All significant intercepts start and end on a grade interval (0.3g/t gold) with a minimum length of 4m, a total grade of or greater than 0.3g/t gold, and a maximum continuous internal waste interval of 4m. BHX may report gold intercepts down to 2m if the gold grade is significant. Once finalised, the accepted assays were transferred to the acQuire database on site. The geologists on site reviewed the QAQC data and the received assays against the logging intervals. The received assays were reviewed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. In October 2013, the highest 50 gold and 50 silver results from the Ban Houayxai

Criteria	Commentary
	project were extracted from the acQuire database and compared against the original ALS laboratory data. This analysis indicated that all assays for gold and silver had been imported correctly and that the export process correctly exports the data without error. Some differences were observed in the export for minor elements at low levels, this has been caused by rounding of data on export. The gold results were from 5 reverse circulation holes and 28 diamond drill holes dispatched in 33 batches between 2006 and 2013. The silver results were selected from 1 reverse circulation hole and 38 diamond drill holes dispatched in 39 batches between 2006 and 2013. The down hole surveys from 859 holes were checked for completeness and for excessive deviation (greater than 10 degrees). A detailed report is in progress, however the preliminary analysis does not indicate any material concerns. The site server and the Vientiane server are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. A high degree of confidence in the geological interpretation can be gained by the minor adjustments required by subsequent drilling programs. The recent drilling program resulted in very minor changed to the geology model. Additional support is gained from mining operations, grade control drilling and in-pit mapping. Site geologists developed a sectional interpretation every 25m for the major lithologies. Alteration, structure, surface mapping, mineralisation style and the assay values were used to generate the major lithological domains. The sections are confirmed with plan and long section interpretations. The sectional interpretations along with structural information were used to generate a 3D model. Site generated a series of surfaces to represent the weathering profile; this consists of the base of complete oxidation (BOCO) and base of partial oxidation (BOPO). This defines the Oxide, Transitional and Primary zones. BOCO and BOPO surfaces were produced using drill hole logging information and then refined using sulphur and Zinc assays where available. Within the BOCO, zones of elevated sulphur and/or zinc were generally excluded, and zones of very low sulphur and/or zinc were sometimes included, depending on lithology. Gold occurs in in narrow veins and is structurally controlled. Mineralisation is predominantly contained within sparsely distributed east-west striking moderate north dipping quartz-carbonate-sulphide veins. The grade of mineralised veins (quartz pyrite ± carbonate ± base metals ± electrum ± native silver) and the mineralised volcaniclastic siltstone generally increase with increasing intensity of deformation. Mineralisation solids were developed using the major lithological domains and sample grades. The Resource estimate used a combination of the mineralisation and weathering zones as domains to control the estimation.

Criteria	Commentary
Dimensions	The main mineralised zones (Northern and southern zones) are approximately 1200m long striking in a North-south direction, vary in width from 20m to 300m in and steeply dips to approximately 350m below the surface. The Northern mineralised zone is generally deeper than the South mineralised zone. The Resource has been reported above 311mRL.
Estimation and modelling techniques	The Resource estimate was completed within CAE Studio 3 (Datamine). Multiple indicator kriging (MIK) was used for gold and silver utilising the Hellman & Schofield (H&S) GS3 software while Ordinary Kriging (OK) method was used for copper, lead, sulphur, zinc and gold in the colluvium employing the Datamine software. The MIK estimates generated by H&S GS3 software were converted to Datamine block and combined with OK model for validation and reporting. Geological modelling was completed by site geologists and reviewed by H&S. QAQC and data validation was completed by Data and Resource Geologists along with preliminary data analysis. H&S reviewed this work and used this data to complete geostatistical analysis and Resource estimation. Multiple domains were used to control the estimation and account for different grade distribution and mineralisation characteristics. Sample data was composited to 2m for estimation. No top caps were applied to any of the data sets used for estimation. Mineralisation solids were generated by combining lithology solids that had similar geostatistical characteristics, a total of eight zones were defined. These solids were combined with the weathering surfaces to produce 24 estimation domains. The domains used are a combination of soft and hard boundaries during estimation. For gold and copper estimation, boundaries between oxides and transitional and primary were soft while the boundaries between ore and waste domains were hard. For silver, sulphur, lead and zinc estimation, the oxide-transitional and ore and waste were boundaries were considered hard while transition-primary boundary was soft. For silver estimation, the primary domains were combined with the exception of the Southeast siltstone. Some of the variograms were shared between domains due to data distribution or similarities; however, the domains restricted the samples available for estimation. The Resource model covers the mineralised zone and extends to include waste material that may be included in a pit shell. The model consists of 25mx12.5mx10m (X, Y, Z) blocks that are sub-blocked to 5mx2.5mx2.5m (X, Y, Z) on the mineralisation solid boundaries. Variogram maps of gold for the main north and south mineralised zones indicate a relatively steep northerly dipping trend supporting the general geologically interpreted trend of the mineralisation. The variogram is a measure of variability; it increases as samples become more dissimilar. Sets of Gold indicator variograms were modelled in the main mineralised domains and applied to other mineralised zones. Oxidation zones were combined for gold variography. A nugget of 20% was used and the nugget represents the geological microstructure and measurement error within the data. The ranges for the shortest and longest structures varied between 20m and 250m. Search distances and directions applied during estimation vary for each domain. They range in maximum values between 35m to 140m, and 10m to 40m in the short axis. The orientation and dip of the gold searches area- • Mineralisation zones 1, 3-6: 60^0 North

Criteria	Commentary
	• Mineralisation zone 2: 80^0 North • Colluvium: 20^0 west The estimation of gold and copper only considered mineralisation zones while estimation for silver, lead, zinc and sulphur considered weathering zones and were broken into east, central (easterly dipping), central (flat lying) and western (westerly dipping). The number of samples used to inform a block estimate ranged from 8 to 48 for MIK and 8 to 32 for OK. The estimate was completed using OK and MIK in four separate estimation passes. Each pass used larger searches and more relaxed minimum/maximum sample numbers. This was done to ensure that distal blocks were estimated. Comparisons of the Resource model with 2010 model indicates an increase in gold grade (8%) and a decrease in tonnes (3%) resulting in an increase in gold metal of 5% for Measured, Indicated and Inferred Resources. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tool was the use of SWATH plots. These are plots of the average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Both validation tools indicated that the Resource estimate was a fair representation of the sample grades, given the estimation process and the distribution of samples.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut off grades used for reporting were selected on the basis economic factors and mining recoveries for the oxides, transition and primary materials. They also broadly represent the mining cutting offs used in the past 12 months. In 2013, mining operations used a gold equivalent cut-off of 0.23g/t for oxides and transitional materials. No primary material was mined in 2013.
Mining factors or assumptions	The deposit will be mined by a selective open pit method where the ore is defined before mining using grade control drilling and mark out. Pit slope parameters have been derived from recommendations by specialist consultants Pells Sullivan Meynink (PSM). Dilution and mining control have been derived from industry standards, taking into consideration the mining to date. Pit shells that have been generated on previous Resource estimates were used to guide the consideration of prospects for eventual economic extraction. Only Mineral resources above 311RL (50m below the current ultimate pit outline) were reported and considered to be economic for eventual extraction. Project to date reconciliation indicates that the Mill is achieving a higher tonnage (9%) and higher grades (3%) compared with the Resource model depletions. These are within the expected limitations of a comparison of Mill actual numbers and Recourse model.
Metallurgical factors or assumptions	Metallurgical test work has been completed on samples of the mineralised zone. The estimated gold recoveries applied of 93% and 83% for oxide and transitional ores, result from 18 months of operating history supported the metallurgical tests that gold at Ban Houyxai is extractable using industry standard methods. An estimated gold recovery of 78% for primary ore was derived in the Feasibility study due to a lack of operating data. The ore is processed using a conventional crush, grind and carbon in leach (CIL)

Criteria	Commentary
	processes to recover gold and silver to a dore product that is exported for refining into fine gold and silver.
Environmental factors or assumptions	Existing governmental approvals and infrastructure at the BHX operation will cover the planned volumes of waste and residue from the mining operation.
Bulk density	Bulk density determinations were undertaken at three separate points, complete core tray determination, calliper measurements and water immersion by ALS. Each method provides a different set of data that had its own bias. The least bias data set is considered to be the ALS water immersion. Approximately 25% of the samples were selected for ALS bulk density value testing. Density measurements were made on each sample for the entire selected hole. The ALS method (OA-GRA08) consists of- • Sample was dried • Placed into a stocking • Weighed in air • Weighed in water No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal. Average ALS bulk density values for eight lithology units that were further broken up by the three weathering zones were determined and assigned to corresponding blocks in the Resource model. Any extreme and unrealistic values were not used to generate the average value.
Classification	Solids were generated to represent Measured, Indicated and Inferred zones based on drill hole spacing, geological confidence and estimation pass. Measured blocks were made up of blocks that were estimated in pass 1 and were in the region of drilling that was generally 25m to 25m spacing. Indicated blocks were mostly pass 1 and pass 2 and/or outside the Measured zone with an approximate drill hole spacing of 50m. Blocks supported by a drill hole spacing of 100m were generally classified as Inferred. Nearly all of these blocks were estimated in the third pass. The majority of Measured material is contained within the transitional zone, with the bulk of the Indicated and Inferred material being within the primary zone. The resource classification results appropriately reflect the Competent Person's view of the deposit.
Audits or reviews	The Resource estimate has been completed by Hellman & Schofield consultants Pty Ltd of Sydney. The Resource estimation process and results have been audited and verified by PanAust personnel. In 2013 BDA Minerals Industry Consultants completed an independent audit of the Phu Kham and Ban Houayxai projects. This was a high level review that was a post completion technical audit for the Ban Houayxai project, however the report included commentary on the confidence in the geology and estimation.
Discussion of relative accuracy/ confidence	The nature of bulk density samples introduces a risk to the estimated tonnages. The three methods used to determine bulk density values all have an implied bias, the ALS data set is the best representation. The actual results and the assignment of bulk to the classified mineralisation has an associated risk. Mining data to date does not suggest

Criteria	Commentary
	any significant issues with this data set. The estimation parameters used for the domains with limited data were derived from similar neighbouring domains. This was a fair application however, local variations may occur. Additional data from infill drilling will improve estimation in these domains. The majority of the mineralised material within the current pit shell is well supported by drilling, so any changes in the mineralisation or lithology solids should only have a localised impact and not alter the overall Resource.

KTL

Section 1. Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 302 holes for 66,563 m have been drilled from surface for the Khamthonglai (KTL) deposit with approximately 93% of the samples from diamond drilling and the remaining 75% from reverse circulation drilling. The majority (96%) of samples for this project are 2m in length; however, samples range from 0.2m through to 12.1m for samples outside the mineralised zone. Half core samples from diamond drilling averaged 6kg in weight and the reverse circulation samples averaged 29kg. QAQC samples consisting of field duplicates (additional split from reverse circulation drilling or a quarter core samples from diamond drilling), standards and blank material were included at the rate of 1 in 30. All samples were sent to ALS Townsville (Australia) for sample preparation and analysis prior to 2009. Recent sample preparation was completed in ALS Vientiane, ALS Townsville prior, and gold analysis has been completed at ALS Vientiane while base metal analysis was completed at ALS Brisbane. The complete data set includes 32,345 copper samples and 32,249 gold samples and consists of diamond and reverse circulation samples.
Drilling techniques	Approximately 7% of the samples were from reverse circulation drilling (RC drilling). The majority of these holes were less than 150m in length. Six of these holes were pre-collars for diamond tails, the RC portion was up to 60m in length. The RC drilling consisted of a down the hole hammer and face sampling bit that was 146mm in diameter. Diamond drill holes were drilled using triple tube in two major sizes, PQ (83.1mm) and HQ (61.1mm) with minor amounts of NQ (45.1mm). The average step down depth from PQ to HQ is 60m and from HQ to NQ is 270m depth down hole. Before 2009, diamond hole core was orientated using a spear, a long, thin, tapered steel rod (spear) with either a tungsten spike or RED “Chinagraph” pencil inserted in the end. The spear, which will naturally reside on the bottom side of the hole under gravity, is lowered slowly down an inclined drill hole to mark the lower edge of the core stub that will be retrieved in the following core run. The core drilled after 2009 has been orientated using a Reflex ACT II RD tool. The Reflex ACT II RD is a digital core orientation device which comprises two components, one residing within the inner tube and backend assembly and the other as a core barrel extension. Accuracy is reportedly better than 1° while orientation can be still achieved in drill holes up to an

Criteria	Commentary
	inclination of -88°.
Drill sample recovery	RC sample recoveries are calculated from estimation the theoretical weight of the material collected compared with the actual weight. The theoretical weight is estimated using the following calculation – RC Sample Recovery can be calculated by dividing the measured mass (kg) by the theoretical mass (kg) using the following formulas: Theoretical Volume = π × ((drill bit diameter + 4mm)/2)^2 × h Theoretical Mass = (Density × 1000) × Theoretical Volume Where: • Drill Bit Diameter = Measured in millimetres at the start of hole, every change of bit and at the end of the hole. The diameter must be converted to meters for the calculation. • 4mm = a constant and allows for 2mm hole over break either side of the drill bit. The over break must be converted to meters for the calculation. • h = Sample interval in metres • Density = Nominated value in grams/cubic cm Diamond core recovery is calculated by geo-technicians by measuring the length of the recovered core and comparing this with the sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. The recoveries and geo-technicians mark-ups are checked by the site geologist during logging. During logging the geologist will assign the loss of core to either core loss due to drilling error, bad ground (fault material) or voids. An analysis of the assay data and recoveries does not indicate that there is any correlation of core loss to high or low grade intervals. The lithology within the mineralised zone is competent and this is reflected in the good recoveries achieved, the average recovery from all drilling programs exceeded 96%.
Logging	Each day a geologist inspects the core at the drill site and identifies the major zones and lithology. The core is then transported to the core yard using specifically designed cages to ensure the security of the samples. Once at the core yard the core is cleaned and checked for completeness and to ensure that all core block depths are correct and consistent with the daily drilling sheets. Reverse circulation chips are logged by a geologist at the drill rig during drilling. No geotechnical logging is possible on reverse circulation samples. In the core yard, geo-technicians start to geotechnically log the core applying a standard set of criteria to a high standard and level of detail. This data is recorded validated and entered into the acQuire database. The geotechnical logging captures rock quality designation (RQD) data, core run details (core recovery, core loss type, length of core within run), material properties (strength, consistency) and structures. In addition, point load testing was completed on each dry bulk density sample (cylinder); the majority of point load testing is diametrical. If axial point load testing is, required guidance is sought from the

Criteria	Commentary
	supervising geologist at the start of the drilling programme. Once the geotechnical logging is completed, site geologists log the core in detail using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. This data is logged onto paper forms or entered directly into the acQuire database using tablet computers. All paper forms are entered into the acQuire database by data entry staff and then validated by the logging geologist. A summary of the drill hole, including logging is produced on finalisation of the assay data. There are separate logging sheets to capture- • Bulk density • Orientation data • Geotechnical data • Weathering (oxidation state) • Lithology • Alteration • Veining • Mineralisation • Structure • Magnetic Susceptibility On completion of mark up and geotechnical and geological logging, all core was photographed before sampling. Core was photographed wet and in direct sunlight. The photographs were downloaded onto the site server and are available for checking and validation.
Sub-sampling techniques and sample preparation	RC chip samples were collected from each sampling interval from the rig mounted cyclone. This sample was riffle split to produce a sample that represents 12.5% of the initial sample collected. Another 25% sample is retained as a reference sample and when required (1 in 30) another 12.5% sample was collected as a field duplicate. Any wet RC chip samples were allowed to dry before any sampling using the riffle splitter was attempted. A number of the RC samples were wet (19%). All diamond drill holes were logged and photographed in the core yard before sampling. The site geologists oversee all sampling and ensure that representative samples are collected by defining the cutting line on the core. Sampling consisted of cutting the core lengthways using a diamond core saw along the predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals is placed into a pre-numbered calico bag. Where a field duplicate sample was collected (approximately 1 in 30) for diamond holes, the half core cut for assaying was cut again to produce two quarters. The left had side was sent for assay and the right hand side was sent as the duplicate sample. These represent two separate samples sent to the laboratory. Sample preparation was done at ALS Vientiane. • Samples were dried at 110^0C in LPG gas ovens controlled by thermostat. • All RC and core samples were crushed to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. More than 5% of the samples are quality control tested to ensure sizing conformance.

Criteria	Commentary
	• Samples received with a weight greater than 1.2kg were split to 1-1.2kg using the adjustable rotary sample divider. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm. More than 5% of the samples are quality control tested to ensure sizing conformance
Quality of assay data and laboratory tests	All the analytical techniques used are considered total through four acid digestion or sample fusion. Base metals were analysed in Brisbane (Australia) since 2005 by ME-ICP41 aqua-regia acid digestion with Inductively coupled plasma (ICP) method with a sample charge of 20g. The lower detection limit for copper and silver are 1% and 0.2g/t while the upper detection limits are 1% and 100g/t respectively. ME-OG62, a four acid digest method was used to analyse higher grade copper, lead and zinc (greater than 1%) for better accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively. Cyanide leach for copper and gold is triggered when copper is greater than 0.1%. Before 2009 the AA13 (cyanide leach Atomic absorption spectrometry (AAS)) method was done in ALS Perth (30g charge), after 2009 AA15ve cyanide leach was used with a 200g sample charge. The lower detection limit for the AA15ve method for gold and copper is 0.02g/t and 0.1ppm while the upper detection limits are 300g/t and 20% respectively. Gold was analysed by ALS at Townsville laboratory before 2009. After 2009, samples have been prepared and analysed at ALS Vientiane (Lao) using the Au-AA26 (fire assay fire assay with atomic absorption spectroscopy) method with a 50g sample charge. This method has a lower detection limit of 0.02g/t and upper detection limit of 100g/t. When gold is greater than 100g /t the Au-AA26-DIL (gold by dilution) method is used (lower detection of 1g/t). The Khamthonglai analysis suite includes ALS method Au-AA26 for gold and MEICP41 for silver, arsenic, bismuth, cadmium, cobalt, copper, iron, mercury, manganese, molybdenum, nickel, lead, sulphur, antimony, uranium, vanadium, zinc and copper and gold AA15ve triggered when copper is greater than 1000g/t. The company inserted field duplicates, blanks and standards every 30m (10%) in that order. Mekong sand was used as a blank for Khamthonglai project. Blank samples are inserted to check for contamination in field sampling, laboratory's sample preparation and analysis, the result of the blank material should be below detection limits. The blank results indicated that Mekong Sand performed well for gold but was not a suitable blank material for copper. A new blank source to replace Mekong Sand was recommended for Khamthonglai project. The base metal and gold standards were sourced from Geostats Pty Ltd and Gannet Holdings with copper certified value ranging between 14g/t and 5578g/t and gold certified values ranging between 0.21g/t and 4.15g/t. Standard reference materials are used to check accuracy and bias of the analytical method, the results should be very similar to the standard concentration for the specific standard. QAQC samples were monitored on a batch-by-batch basis. An assay batch is accepted if the blank samples are within the acceptable limits (5 times the lower detection limit) and the standards are within the ± 3SD (standard deviations). One failed standard can cause rejection if the results around the failed standard are not in normal grade range. A batch is also re-assayed when two or more standards assay

Criteria	Commentary
	results are outside the acceptable limits. The inserted blank materials did not show any consistent issues with sample contamination. Any batches that were rejected based on the QAQC results were re-assayed at ALS, the subsequent results were again reviewed, and if acceptable were loaded into the acQuire database as the primary sample result. 99% of the base metal standards all performed well and were within acceptable limits. All gold standards were generally within acceptable limits with 78% of the standards showing a slight bias low. ALS is investigating the low bias in gold standards. The performance of field duplicates in core samples is generally reasonable and the variations are possibly related to the style of mineralisation. The ALS laboratory also inserted QAQC samples to internally test the quality of the analysis. These results are received with the assay results in each batch. The ALSQAQC included standards, blanks and duplicates for independent quality control. The results of the lab standards were also monitored on a batch to batch basis by the Data and Resources geologist. The results did not show any issues with the laboratory.
Verification of sampling and assaying	Round Robin analysis for 490 samples assayed between 2011 and 2012 was conducted in 2013. Check pulps were submitted to ALS Perth, SGS Perth, Intertek Indonesia and Standard Reference Lab Perth. The pulps were analysed for gold with fire assays and copper via MEI-CP41 and ME-OG62. Correlation for gold ranged between 0.96 and 0.97 while copper had correlation between 0.93 and 0.99. The check samples and the original did not show any bias though an improvement in pulp splitting and sample preparation was recommended for better precision. Parts of five drilled holes were twinned, this included reverse circulation and diamond drill holes. There were no unexpected down hole variations in grade between the holes. The results indicated that the Khamthonglai project assays are of good quality and reproducible. All assay data was accepted into the database as supplied by the laboratory. Where a duplicate sample was taken (quarter core), the original and duplicate assays were averaged to maintain the sample support of half core.
Location of data points	Drill hole collars were set-up using Garmin 72 before 2009. The Garmin GPSmap62s was then used up to 2011 and holes that are more recent have been picked up using the Garmin 60CSx on WGS84 UTM Zone 48N. On completion of drilling collars were surveyed using differential GPS (DGPS). Approximately 51 collar locations were validated by AAM (Thailand) Co. Ltd. during a 2011 topographical survey. Only 2 collar elevations were identified to be inconsistent with a 0.5% error. A down hole survey was completed every 30 m, using a Reflex single shot camera, Proshot or a simple shot camera. The majority of the surveys were completed using a Simple shot camera. The down hole surveys were collected and examined each day by the site geologist. Any surveys that were spurious were retaken. Once validated the surveys were entered into the acQuire data base by data entry personnel. All down hole cameras were tested once a week in the presence of a geologist during the drilling programme using an on-site reliability jig.
Data spacing	The majority of the drill holes are spaced on a 50x50m grid. The spacing is variable

Criteria	Commentary
and distribution	due to topography. The spacing reduces to 25x50m in some areas, particularly central part of the deposit. The drill holes spacing adequately enables the estimation of the Resource on a medium scale. Resource model is a fair representation of the drill spacing and geological confidence. The Resource Classification applied to the Resource model reflects the variability in drill hole spacing. Sample data was composited to 2m for estimation. The 2m composite sample length was selected for estimation based on the majority sample interval (96% of the samples were 2m in length).
Orientation of data in relation to geological structure	99% of the holes were drilled with a northerly azimuth (approximately 360^0) with dips ranging between -45^0 and -90^0. The direction of the drilling was designed to best capture the interpreted dip of the mineralisation. The diamond drilling is angled to be perpendicular to the trend of lithology and structure. 30 scout holes and 6 RC holes drilled between 1996 and 2005 were vertical. Structural measurements from the orientated core enabled an analysis of the controls of mineralisation and structures to continuously evaluate the direction of drilling to ensure that it was optimal.
Sample security	Drill core and reverse circulation samples were picked up from the drilling site on a daily basis and transported to the core yard. The geologist recorded the intervals of core being picked and some completed brief notes on the main features before the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is manned by employees during the day and contracted security guards during the night time. All logging and sampling of diamond drill core was completed in the core yard while RC logging is completed at the rig and samples are bagged in the field and delivered to the core yard. All samples to be dispatched were packed in rice bags, clearly labelled with a submission number, project and prospect number, sample type and date of dispatch. The sample dispatches were accompanied by supporting documentation signed by the geologist and showing the sample submission number, analysis suite, and number of samples. ALS picks up samples at the core yard and transport to the Vientiane laboratory by ALS employees for sample preparation. On site, prior to dispatch, a log book is signed by an ALS personnel and the site geological technicians. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples. The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators in Vientiane and loaded into acQuire through an automated process. QAQC on import is completed before the results are finalised.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects. In 2011, AMC Consultants Melbourne reviewed Phu Bia Sampling Handling protocols report and Field Technical Procedures report as part of the preparation steps for Resource estimation. They found them appropriate.

Section 2. Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited ('Phu Bia Mining'). The Government of the Lao PDR holds a 10% interest in Phu Bia Mining. Phu Bia Mining's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 square kilometres, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.
Exploration done by other parties	No significant exploration has been completed by other companies on the Phu Kham project area. The prospect was discovered between 1996 and 2004 by Phu Bia Mining (PBM), a Lao-registered company 90% owned by PanAust.
Geology	The Khamthonglai deposit comprises Late Carboniferous–Early-Permian volcano-sedimentary rocks of shallow marine and volcaniclastic origin which have been intruded by rift associated calc-alkali stocks of dioritic and tonalitic affinities. Mineralisation occurs within strata bound porphyry-skarn style copper gold system which predominantly contains skarn and stock work quartz-sulphide copper-(molybdenum-gold) veining. Low to moderate grade copper-molybdenum-gold mineralisation is hosted within multi-phase stock works, sheeted quartz-sulphide veins and disseminated-aggregate sulphide mineralisation. High grade copper-gold is associated with banded, semi-massive to massive sulphides hosted within exo-skarns.
Drill hole Information	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Relationship between mineralisation widths and intercept	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Orientation of data in relation to geological

Criteria	Commentary
lengths	structure".
Diagrams	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource. Feasibility studies are currently in progress for this project, any further work will be relevant to the outcomes of these studies.

Section 3. Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	AcQuire using a Microsoft SQL Server 2005 running the acQuire data model (3500) is the database used for storing all drill hole data. The central database is located in the Vientiane office (capital of Lao). The most recent database audit was completed in July 2013 by acQuire Technology solutions. No potential risks with the database structure or application were reported. Historical data (approximately 13% of the data set) was manually entered using an acQuire data entry form, and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using Motion computing tablets computers with acQuire offline data entry forms. The logging data was electronically transferred to the site server and validated by the geologists. After validation on site, data was routinely synchronised with the central acQuire server in Vientiane. Assay results were sent electronically from the laboratory directly to the Database Administrator in Vientiane for validation and loading into acQuire, no manual data entry or intervention is required. The Data Resources Geologist checks the QAQC of the received assays and advised the Databases Administrator to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, Data and Resources geologist and if required queries are made with ALS. After the batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste interval of 4m. Once finalised, the accepted assays were transferred to the acQuire database on site. The geologists on site reviewed the QAQC data and the received assays against the

Criteria	Commentary
	logging intervals. The received assays were reviewed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. The highest 50 copper results from the Khamthonglai project were extracted from the acQuire database and compared against the original ALS laboratory data. This analysis indicated that all assays for copper had been imported correctly and that the export process correctly exports the data without error. Some differences were observed in the export for minor elements at low levels, this has been caused by rounding of data on export. The down hole surveys were checked in holes drilled with inclination greater than 70 degrees for completeness and for excessive deviation (greater than 10 degrees). The maximum deviation obtained in dip was 1.9 degrees while a maximum deviation of 6.3 degrees was recorded in azimuth. Khamthonglai holes have minimum deviation in dip and azimuth for holes drilled with an inclination greater than -70 degrees. The site server and the Vientiane server are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. Site geologists developed a sectional interpretation every 50m for the major lithologies. Alteration, structure, surface mapping and mineralisation style were used to generate the major lithological domains. The sections are confirmed with plan and long section interpretations. The sectional interpretations along with structural information were used to generate a 3D model. Site generated a series of surfaces to represent the weathering profile (base of complete oxidation (BOCO) and base of partial oxidation (BOPO)). This defines the Oxide, Transitional and Primary zones. BOCO and BOPO surfaces were produced using drill hole logging information, and were then refined using Lead and Zinc assays which generally have higher concentrations above the BOCO. Higher Sulphur values are normally expected in the transitional and primary zones. Mineralisation occurs within strata bound porphyry-skarn style copper gold system which predominantly contains skarn and stock work quartz-sulphide copper – (molybdenite-gold). Mineralisation occurs as low to moderate grade copper-molybdenite-gold hosted within multi-phase stock works, sheeted quartz-sulphide veins and disseminated-aggregate sulphide High grade copper-gold is associated with banded, semi-massive to massive sulphides hosted within exo-skarns form by metasomatic hydrothermal activity. The dominant sulphide minerals within KTL are pyrite, chalcopyrite and pyrrhotite, with less common molybdenite, bornite, sphalerite and galena. Mineralisation solids were developed using the site interpretation and sample grades. The Resource estimate used a combination of the mineralisation and

Criteria	Commentary
	weathering zones as domains to control the estimation.
Dimensions	The main mineralised zone is approximately 2500m long striking in an East- West direction, varies in width from 10m to 100m (may consist of multiple lenses) and dips 40-45^0 to approximately 400m below the surface.
Estimation and modelling techniques	The Resource estimate was completed within CAE Studio 3 (Datamine) utilising Ordinary Kriging (OK) as the estimation method. Multiple domains were used to control the estimation and account for different grade distribution and mineralisation characteristics. Sample data was composited to 2m for estimation (96% are 2m in length). No top caps were applied to the samples within the mineralised zone during estimation. Three mineralisation solids, 0.2% copper, 0.5% copper and the back ground material were generated, and the 0.5% copper solid represents the higher grade copper zone. The mineralisation zones in conjunction with the weathering surfaces, oxides, transitional and primary zones were used to generate the nine domains. Spherical variograms were generated for the 0.2% copper zone and applied to 0.5% copper solid due to data distribution and similarities; however, the domains restricted the samples available for estimation. The variogram is a measure of variability; it increases as samples become more dissimilar. The nugget for the copper variograms range from 34% (in the higher grade zone) to 37% for the moderate and background grade zones. The variogram range shows the maximum distance at which pairs of samples are related. The longest ranges for the variograms (165m to 370m) are in the strike and down dip directions while the shortest range (28m to 62m) is in the Z direction. The resulting ellipsoid had along axis dipping to the south. The Resource model extents cover the mineralised zone and extend to include waste material that may be included in a pit shell. The model consists 15mx15mx5m (X,Y,Z) blocks that are sub-blocked to 5mx5mx5m (X,Y,Z) on the mineralisation solid boundaries. Data analysis and variography was completed in Supervisor (Snowden software) for the major domains. Kriging Neighbourhood Analysis (KNA) was undertaken to better define the estimation parameters. These tests used the Kriging Efficiency, Slope of Regression and Kriging Variance to assisted in defining the best parameters for- • Block size • Search distances • Minimum and maximum number of samples to be used for an estimate • Octant restraints on the sample selection Search distances and directions vary for each domain; they range in maximum values between 100m to 400m, and 40m to 160m in the short axis. The direction of the gold searches area - • Long axis dipping at 40-45^0 to the south for transitional, primary and oxide zones in copper, gold, sulphur, silver and arsenic. • Flat E_W strike for iron, lead, Zinc and Molybdenum in oxides, transitional and primary zones The number of samples used to inform a block estimate ranged from a minimum of 4 to a maximum of 35. The estimate was completed using ordinary kriging in three separate estimation passes. Each pass used larger searches and more relaxed

Criteria	Commentary
	minimum/maximum sample numbers. This was done to ensure that distal blocks were estimated. The estimation pass was a consideration when assigning a Resource classification. Second and third estimated blocks potentially received a lower confidence classification. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tool was the use of SWATH plots. These are plots of the average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Comparisons of the Resource model with previous models indicated that the changes in the Resource were consistent with the impact of the additional drilling and changes to the mineralisation solids. Both validation tools indicated that the Resource estimate was a fair representation of the sample grades, given the estimation process and the distribution of samples.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut off used for reporting were selected from the initial economic evaluations completed for the feasibility study.
Mining factors or assumptions	Initial findings of the feasibility study have been used to limit the material reported for the Resource model. One of the larger pits from Whittle studies was used to determine a depth for possible economic extraction. The lower limit was selected that was 20m below this pit (1200mRL). This pit was also used to guide the lateral limits of the Resource. A limit of 318800mE was selected as the western extent of the Resource reporting. A cut off of 0.5% copper, above 1200mRL and to the east of 318800mE was applied for reporting the Resource for this project.
Metallurgical factors or assumptions	Khamthonglai material will be processed through the existing Phu Kham concentrator. The Phu Kham flow sheet uses a conventional crush, grind and flotation process to recover a saleable copper concentrate that contains copper, gold, and silver. The metallurgical test work demonstrated similar mineralogy to the Phu Kham deposit which will enable Khamthonglai material to be directly blended with Phu Kham ore and treated through the Phu Kham concentrator. Metallurgical recovery estimates for these metallurgical domains are based on results from ongoing batch flotation test work. Higher metallurgical recovery values (85% copper, 65% gold and 45% silver) were obtained for primary mineralisation with lower recoveries for transitional mineralisation.
Environmental factors or assumptions	Possible sites for waste material storage and process residue disposal have been identified within the current exploration area. The Khamthonglai deposit occurs in an area with unexploded ordnance (UXO). All working areas were cleared of any UXO before any exploration work commenced. The company employs Milsearch Pty Limited which has extensive experience in the assessment and remediation of unexploded ordnance. All cleared areas are plotted on a base map for future references.
Bulk density	Bulk density determinations were undertaken at three separate points, complete core tray determination, calliper measurements and water immersion by ALS. Each

Criteria	Commentary
	method provides a different set of data that had its own bias. The least bias data set is considered to be the ALS water immersion. Over 80% of the samples have an ALS bulk density value; every sample has an ALS bulk density value from the 2010 drilling onwards. The ALS method (OA-GRA08) consists of- • Sample was dried • Placed into a stocking • Weighed in air • Weighed in water No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal. Average ALS bulk density values for six lithology units that were further broken up by the three weathering zones were determined and assigned to corresponding blocks in the Resource model. Any extreme and unrealistic values were not used to generate the average value.
Classification	Solids were generated to represent Measured, Indicated and Inferred zones based on drill hole spacing, geological confidence and estimation pass. The Measured blocks are made up of blocks that were estimated in pass 1 and were in the region of drilling that was on 70m to 70m spacing. Indicated blocks were mostly pass 1 and pass 2 and/or outside the Measured zone with an approximate drill hole spacing of 100m. Blocks supported by a drill hole spacing up to 300m were generally classified as Inferred. The majority of Measured material is contained within the Transitional zone, with the bulk of the Indicated and Inferred material being within the primary zone. The resource classification results appropriately reflect the Competent Person's view of the deposit
Audits or reviews	The geology and estimation process and results have been internally audited. Previous estimates were completed by external consultants (AMC Consultants Melbourne and Cube Consulting Perth) under the guidance PanAust and reviewed both externally and internally.
Discussion of relative accuracy/ confidence	The bulk of the classified mineralisation is within the defined mineralised zones. The geological understanding of the deposit increased with more drilling information being available, however additional drilling did not require significant changes to the geology modelling. This indicates that the geology model is robust. The majority of the mineralised material within the current pit shell is well supported by drilling, so any changes in the mineralisation or lithology solids should only have a localised impact and not alter the overall Resource.

LCT

Section 1. Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	A total of 249 holes for 46,901m have been drilled from surface for the Long Chieng Track (LCT) deposit with approximately 88% of the samples from diamond drilling and the remaining 125 from reverse circulation drilling. The majority (97%) of samples for this project are 1m in length; however, samples range from 0.1m through to 59m for samples outside the mineralised zone. QAQC samples consisting of field duplicates (additional split from reverse circulation drilling or a quarter core samples from diamond drilling), standards and blank material were included at the rate of 1 in 30. The half core samples from diamond drilling averaged 4kg in weight, the reverse circulation samples averaged 26kg. All of these samples were sent to ALS Perth (Australia) for sample preparation and analysis. Recent sample preparation was completed in ALS Vientiane, ALS Perth (Australia) prior, and all gold analysis has been completed at ALS Vientiane. The base metal analysis has been done at ALS Perth until May 2012 and ALS Brisbane (Australia) thereafter. The complete data set includes 40,276 gold samples and consists of diamond and reverse circulation samples
Drilling techniques	Approximately 12% of the samples were from reverse circulation drilling (RC drilling). The majority of these were short holes, less than 50m in length. Three of these holes were pre-collars for diamond tails, with the RC portion up to 100m in length. The RC drilling consisted of a down the hole hammer and face sampling bit that was 146mm in diameter. Diamond drill holes were drilled using triple tube in two major sizes, PQ (83.1mm) and HQ (61.1mm) with minor amounts of NQ (45.1mm) samples. The average step down depth from PQ to HQ is 75m and from HQ to NQ is 390m down hole. Recent diamond drilling has been orientated using a Reflex ACT II RD tool. The Reflex ACT II RD is a digital core orientation device which comprises two components, one residing within the inner tube and backend assembly and the other as a core barrel extension. Accuracy is reportedly better than 1° while orientation can be still achieved in drill holes up to an inclination of -88°.
Drill sample recovery	RC sample recoveries are calculated from estimation the theoretical weight of the material collected compared with the actual weight. The theoretical weight is estimated using the following calculation – RC Sample Recovery can be calculated by dividing the measured mass (kg) by the theoretical mass (kg) using the following formulas: Theoretical Volume = $\pi \times ((\text{drill bit diameter} + 4\text{mm})/2)^2 \times h$ Theoretical Mass = (Density × 1000) × Theoretical Volume Where: • Drill Bit Diameter = Measured in millimetres at the start of hole, every change of bit and at the end of the hole. The diameter must be converted to meters for the calculation. • 4mm = a constant and allows for 2mm hole over break either side of the drill bit. The over break must be converted to meters

Criteria	Commentary
	for the calculation. • h = Sample interval in metres • Density = Nominated value in grams/cubic cm Diamond core recovery is calculated by measuring the length of the recovered core and comparing this with the sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. The recoveries and geo-technicians mark ups are checked by the site geologist during logging. Any loss of core is assigned to either core loss due to drilling error, bad ground (fault material) or voids. An analysis of the assay data does not indicate that there is any correlation of core loss to high or low grade intervals. The lithology within the mineralised zone is competent and this is reflected in the good recoveries achieved, the average recovery from all drilling programs exceeded 98%.
Logging	Each day a geologist inspects the core at the drill site and identifies the major zones and lithology. The core is then transported to the core yard using specifically designed cages to ensure the security of the samples. Once at the core yard the core is cleaned and checked for completeness and to ensure that all core block depths are correct and consistent with the daily drilling sheets. Reverse circulation chips are logged by a geologist at the drill rig during drilling. No geotechnical logging is possible on reverse circulation samples. In the core yard, geo-technicians start to geotechnically log the core applying a standard set of criteria to a high standard and level of detail. This data is recorded validated and entered into the acQuire database. The geotechnical logging captures rock quality designation (RQD) data, core run details (core recovery, core loss type, length of core within run), material properties (strength, consistency) and structures. In addition, point load testing was completed on each dry bulk density sample (cylinder); the majority of point load testing is diametrical. If axial point load testing is, required guidance is sought from the supervising geologist at the start of the drilling programme. Once the geotechnical logging is completed, site geologists log the geology, mineralisation and alteration in detail using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. This data is logged onto paper forms or entered directly into the acQuire database using tablet computers. All paper forms are entered into the acQuire database by data entry staff and then validated by the logging geologist. A summary of the drill hole, including logging is produced on finalisation of the assay data. There are separate logging sheets to capture- • Bulk density • Orientation data • Geotechnical data • Weathering (Oxidation state) • Lithology • Alteration • Veining • Mineralisation

Criteria	Commentary
	• Structure • Magnetic Susceptibility. On completion of logging and mark up, all core was photographed before sampling. Core was photographed wet and in direct sunlight. The photographs were downloaded onto the site server and are available for checking and validation.
Sub-sampling techniques and sample preparation	RC chip samples were collected from each sampling interval from the rig mounted cyclone. This sample was riffle split to produce a sample that represents 12.5% of the initial sample collected. Another 25% sample is retained as a reference sample and when required (1 in 30) another 12.5% sample was collected as a field duplicate. Any wet RC chip samples were allowed to dry before any sampling using the riffle splitter was attempted. Only a very small number of RC samples were wet (1%). All diamond drill holes were logged and photographed in the core yard before sampling. The site geologists oversee all sampling and ensured that representative samples are collected by defining the cutting line on the core. Sampling consisted of cutting the core lengthways using a diamond core saw along the predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals is placed into a pre-numbered calico bag. Where a field duplicate sample was collected (approximately 1 in 30), the half core for assaying was cut again to produce two quarters. The left had side was sent for assay and the right hand side was sent as the duplicate sample. Sample preparation was done at ALS Vientiane as follows: • All Samples were dried at 110^{0}C in LPG gas ovens controlled by thermostat. • All RC and core samples were crushed to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. More than 5% of the samples are quality control tested to ensure sizing conformance. • Samples received with a weight greater than 1.2kg were split to 1-1.2kg using the adjustable rotary sample divider. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm. The samples are also quality control tested to ensure sizing conformance.
Quality of assay data and laboratory tests	All the analytical techniques used are considered total through four acid digestion or sample fusion. Gold was analysed by ALS at Vientiane (Lao) laboratory after 2006 and ALS Perth before this using the Au-AA26 fire assay with atomic absorption spectroscopy method with a 50g sample charge. This method has a lower detection limit of 0.02g/t and upper detection limit of 100g/t. When gold is greater than 100g/t the Au-AA26-DIL (gold by dilution) method is used (lower detection of 1g/t). A cyanide leach analysis for gold is triggered by a gold result that is greater than 0.10g/t. Before 2009 the ALS method AA13 (cyanide leach Atomic absorption spectrometry (AAS)) was done in ALS Perth (30g charge), after 2009 AA15ve accelerated cyanide leach was used with a 200g sample charge. The lower detection limit for the AA15ve method for gold and copper is 0.02g/t and 0.1g/t while the upper detection limits are 300g/t and 20% respectively. Base metals were analysed in Perth until May 2012 and then ALS Brisbane by ME-ICP41

Criteria	Commentary
	aqua-regia acid digestion with Inductively coupled plasma (ICP) method with a sample charge of 20g. The lower detection limit for silver and copper are 0.2g/t and 1g/t while the upper detection limits are 100g/t and 10,000g/t respectively. ME-OG62 (four acid digest analysed by inductively coupled plasma - atomic emission spectroscopy), was used to analyse higher grade copper, lead and zinc (greater than 100g/t) for better accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively. The Long Chieng Track analysis suite includes ALS methods Au-AA26 for gold and ME-ICP41 (aqua-regia acid digestion inductively coupled plasma - atomic emission spectroscopy (ICP -AES)) for silver, arsenic, copper, mercury, lead, sulphur, antimony and zinc. A trigger of gold greater than 0.10g/t is used to select samples for gold, silver and copper AA15ve. Screen fire assay test work to ascertain the concentration of gold in the coarse fraction was completed on a limited number of holes where the gold grade was above 5g/t. The company inserted certified field duplicates, blanks and standards every 30m (10%) in that order. The blank used for the majority of the drilling into the mineralised zone is a locally sourced Moungcha limestone. Blank samples are inserted to check for contamination in field sampling, laboratory's sample preparation and analysis, the result of the blank material should be below detection limits. The gold and base metal standards were sourced from Geostats Pty Ltd and Gannet Holdings with gold certified values ranging between 0.21g/t and 1.40g/t. Standard reference materials are used to check accuracy and bias of the analytical method, the results should be very similar to the standard concentration for the specific standard. QAQC samples were monitored on a batch-by-batch basis. An assay batch is accepted if the blank samples are within the acceptable limits (5 times the lower detection limit) and the standards are within the $\pm$ 3SD (standard deviations). One failed standard can cause rejection if the results around the failed standard are not in normal grade range. A batch is also re-assayed when two or more standards assay results are outside the acceptable limits. The inserted blank material did not show any consistent issues with sample contamination. 91% of the gold standards were within acceptable limits with a slight bias low. ALS is investigating the low bias in gold standards. Base metal standards all performed well with less than 1% of the standards outside the acceptable limit (3 SD from the certified value). The performance of field duplicates in core samples is generally reasonable and the variations are possibly related to the style of mineralisation. The ALS laboratory also inserted QAQC samples to internally test the quality of the analysis. These results are received with the assay results in each batch. The ALS QAQC included standards, blanks and duplicates for independent quality control. The results of the lab standards were also monitored on a batch to batch basis by the Data and Resources geologist. The results did not show any issues with the laboratory.
Verification of sampling and assaying	Round Robin analysis for 490 samples assayed between 2011 and 2012 was conducted in 2013. Check pulps were submitted to ALS Perth, SGS Perth, Intertek Indonesia and Standard Reference Lab Perth. The pulps were analysed for gold with fire assays and copper using as close as possible to the same analysis methods as the initial ALS laboratory (MEI-CP41 and ME-OG62). Correlation for gold ranged between 0.96 and 0.97 while copper had correlation between 0.93 and 0.99. The check samples and the original did not show any bias though an improvement in pulp splitting and sample

Criteria	Commentary
	preparation was recommended for better precision. All assay data is accepted as supplied by the laboratory. Where a duplicate sample was taken (quarter core), the original and duplicate assays were averaged to maintain the sample support of half core.
Location of data points	Drill hole collars were set-up using Garmin 60CSx, Rhino 659 and 62S hand held GPS units on WGS84 UTM Zone 48N. On completion of drilling collars were surveyed using differential GPS (DGPS). Initial collar pick-up between 2003 and 2006 were completed by GEOMAP (external consultant). Between 2010 and September 2012, the Phu Kham Mine survey department completed collar pickups. Succeeding collar surveys covering the remaining holes were conducted by the LCT exploration survey team using an Ashtech PM100 series DGPS system, utilising multiple base stations at the LCT office, LCT helipad and LNE. A down hole survey was completed every 30 m, using a Reflex single shot camera or an Eastman single shot camera (for the earlier drilling). The majority of the surveys were completed using a Reflex single shot camera, which was used exclusively for diamond holes. RC holes were generally surveyed with a Simple Shot camera. The down hole surveys were collected and examined each day by the site geologist. Any surveys that were spurious were retaken. Once validated the surveys were entered into the acQuire data base by data entry personnel. All down hole cameras were tested once a week in the presence of a geologist during the drilling programme using an on-site reliability jig.
Data spacing and distribution	The majority of the drill holes are spaced on a 50x100m grid. This is variable in some areas due to topography, the spacing is closer or more widely spaced in areas. The Resource Classification applied to the Resource model reflects the variability in drill hole spacing. No sample compositing was undertaken for estimation. This was not considered necessary due to the high proportion of 1m samples (97%).
Orientation of data in relation to geological structure	All of the RC holes have been drilled vertically, the predominant direction for diamond holes is -60 deg drilled to the south (180deg).The direction of the RC drilling was designed to best capture the variability of grade for oxide mineralisation, while the diamond drilling is angled to be perpendicular to the trend of lithology and structure of transitional and primary mineralisation. A small number of drilling in other directions was completed to ensure that the drilling direction best intersected the major lithology and structures. Structural measurements from the orientated core enabled an analysis of the controls of mineralisation and structures to continuously evaluate the direction of drilling to ensure that it was optimal.
Sample security	Drill core and reverse circulation samples were picked up from the drilling site on a daily basis and transported to the core yard. The geologist recorded the intervals of core being picked and some completed brief notes on the main features before the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is under 24 hour security guard. All logging and sampling of diamond drill core was completed in the core yard while RC logging is completed at the rig and samples are bagged in the field and delivered to the core yard. All samples to be dispatched were packed in rice bags, clearly labelled with a

Criteria	Commentary
	submission number, project and prospect number, sample type and date of dispatch. The sample dispatches were accompanied by supporting documentation signed by the geologist and showing the sample submission number, analysis suite, and number of samples. ALS picks up samples at the core yard and transport to the Vientiane laboratory by ALS employees for sample preparation. On site, prior to dispatch, a log book is signed by an ALS personnel and the site geological technicians. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples. The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators in Vientiane and loaded into acQuire through an automated process. QAQC on import is completed before the results are finalised.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects.

Section 2. Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited ('Phu Bia Mining'). The Government of the Lao PDR holds a 10% interest in Phu Bia Mining. Phu Bia Mining's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 square kilometres, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.
Exploration done by other parties	No significant exploration has been completed by other companies on the Phu Kham project area. The prospect was discovered between 1996 and 2004 by Phu Bia Mining (PBM), a Lao-registered company 90% owned by PanAust.
Geology	The majority of the mineralisation occurred within quartz-patchy potassium-feldspar altered felsic porphyry and adjacent contact breccia zones within volcaniclastics. Mineralisation occurs as aggregates of pyrite, sphalerite, galena and chalcopyrite occurring within multiphase deformed quartz-carbonate veining. The mineralisation itself is structurally controlled and complex, with multiple overlapping styles and truncated by several intrusive bodies. Mineralisation is north-north-west dipping extending down-dip and along strike and appears to be wrapping around to the south.
Drill hole	No exploration results are included in this release, all information relates to the Mineral

Criteria	Commentary
Information	Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Relationship between mineralisation widths and intercept lengths	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Orientation of data in relation to geological structure".
Diagrams	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Additional drilling programs will be reviewed on the completion of further study work.

Section 3. Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	AcQuire is the database used for storing all drill hole data, an Microsoft SQL Server 2005 running the acQuire data model (3500) is located in the Vientiane office (capitol of Lao). A recent database audit was completed in July 2013 by acQuire Technology solutions. No potential risks with the database structure or application were reported. Historical data (approximately 13% of the data set) was manually entered using an acQuire data entry form, and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using Motion computing tablets computers with acQuire offline data entry forms. The logging data was electronically transferred to the site server and validated by the geologists. After validation on site, data was routinely synchronised with the central acQuire server in Vientiane. Assay results were sent electronically from the laboratory directly to the Database Administrator in Vientiane for validation and loading into acQuire, no manual data

Criteria	Commentary
	entry or intervention is required. The Data Resources Geologist checks the QAQC of the received assays and advised the Databases Administrator to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, Data and Resources geologist and if required queries are made with ALS. After a batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. All significant intercepts start and end on a grade interval (0.3g/t gold) with a minimum length of 4m, a total grade of or greater than 0.3g/t gold, and a maximum continuous internal waste interval of 4m. Once finalised, the accepted assays were transferred to the acQuire database on site. The geologists on site reviewed the QAQC data and the received assays against the logging intervals. The received assays were analysed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. The highest 50 gold and 50 copper results were extracted from the acQuire database and compared against the original ALS laboratory results. This analysis indicated that all assays for gold and copper had been imported correctly and that the export process correctly exports the data without error. Some differences were observed in the export for minor elements at low levels, this has been caused by rounding of data on export. Ten drill holes were selected from the acQuire database and compared against the original ALS laboratory assay data. No significant inconsistencies were found. The down hole surveys from all holes were checked for completeness and for excessive deviation (greater than 10 degrees). Only 14% of the down hole surveys had a deviation of greater than 10 degrees, the majority of these were validated and appeared to be reasonable. The site server and the Vientiane server are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. A high degree of confidence in the geological interpretation can be gained by the minor adjustments required by subsequent drilling programs. Site geologists developed a sectional interpretation every 50m for the major lithologies. Alteration, structure, surface mapping, and mineralisation style were used to generate the major lithological domains. The sectional interpretations along with structural information were used to generate a 3D model. A series of surfaces to represent the weathering profile (base of complete oxidation (BOCO) and base of partial oxidation (BOPO) were generated on site. This defines the oxide, transitional and primary zones. BOCO and BOPO surfaces were produced using

Criteria	Commentary
	drill hole logging information, and were then refined using copper cyanide solubility results, then lead and zinc assays which may increase in concentrations at both BOCO and BOPO margins. Higher sulphur values are normally expected in the transitional and primary zones. Mineralisation solids were developed using the site interpretation and sample grades. The Resource estimate used a combination of the mineralisation and weathering zones as domains to control the estimation. Gold occurs in three broad styles and several minor and disjointed zones. The most abundant association is with quartz-sphalerite-galena base metal veins. The second style is associated with brecciation and silicification of groundmass. The last type occurs as semi-massive pyrite and chalcopyrite vein styles.
Dimensions	The main mineralised zone is approximately 900m long striking in a North-North/West direction, varies in width from 50m to 270m in width (may consist of multiple lenses) and steeply dips to approximately 600m below the surface.
Estimation and modelling techniques	The Resource estimate was completed within CAE Studio 3 (Datamine) utilising Ordinary Kriging (OK) as the estimation method. Multiple domains were used to control the estimation and account for different grade distribution and mineralisation characteristics. No compositing was done on the samples (97% are 1 m in length) and no top caps were applied to the samples within the mineralised zone. A top cap at the 99.99 percentile was applied to samples in the background zone. This impacted three samples. The oxide zone (BOPO) was estimated as a separate domain to the combined transitional/primary zone. For gold, 0.1g/t and 0.3g/t mineralisation solids were used as domains in conjunction with the BOPO surface. For copper, lead and zinc, 0.2% mineralisation solids were used as domains in conjunction with the BOPO surface. For silver, a 1.0g/t mineralisation solid was used as a domain in conjunction with the BOPO surface. Some of the variograms were shared between domains due to data distribution or similarities; however, the domains restricted the samples available for estimation. Variograms were generated where possible on each domain. The variogram is a measure of variability, it increases as samples become more dissimilar. These were combined where the differences were not significant or where there was insufficient data to support the definition of a variogram. Several of the domains had sufficient data to form reasonable experimental variograms; these were applied to less informed domains. The gold variograms had nugget values that ranged from 20% for oxide domains to 35% for primary domains, the ranges varied from 80m in the sort direction up to 800m in the long direction. The search ellipse ranges were shorter than the variogram ranges. The Resource model extents cover the mineralised zone and extend to include waste material that may be included in a pit shell. The model consists of 20x20x10m (X, Y, Z) blocks that are sub-blocked to 10x10x5m (X, Y, Z) on the mineralisation solid boundaries. Data analysis and variography was completed in Supervisor (Snowden software) for the major domains. Kriging Neighbourhood Analysis (KNA) was undertaken to better define the estimation parameters. These tests used the Kriging Efficiency, Slope of Regression and Kriging Variance to assisted in defining the best parameters for-

Criteria	Commentary
	• Block size • Search distances • Minimum and maximum number of samples to be used for an estimate • Octant restraints on the sample selection Search distances and directions vary for each domain; they range in maximum values between 60m to 80m, and 10m to 26m in the short axis. The shape and direction of the gold searches area- • Oxide - Cigar shaped search striking N-E with a shallow plunge and shallow dip to the S-E. • Transitional/Primary - Disk shaped search striking N-E with a moderate dip (50^0) to the N/N-W. The number of samples used to inform a block estimate ranged from a minimum of 8 to a maximum of 20. The estimate was completed using OK in three separate estimation passes. Each pass used larger searches and more relaxed minimum/maximum sample numbers. This was done to ensure that distal blocks were estimated. Second and third estimated blocks potentially received a lower confidence classification. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tool was the use of SWATH plots. These are plots of the average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Both validation tools indicated that the Resource estimate was a fair representation of the sample grades, given the estimation process and the distribution of samples. Comparisons of the Resource model with previous models indicated that the changes in the Resource were consistent with the impact of the additional drilling and changes to the mineralisation solids.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut-off grade used for reporting was selected on the basis of other operations operated by the company in close proximity (Phu Kham and BHX).
Mining factors or assumptions	Pit shells that have been generated on previous Resource estimates were used to guide the consideration of prospects for eventual economic extraction.
Metallurgical factors or assumptions	Metallurgical test work is continuing to determine the optimal methods to extract gold, copper, lead and zinc utilising the existing Phu Kham mine infrastructure as much as possible.
Environmental factors or assumptions	Existing infrastructure at the Phu Kham operation will be capable of dealing with the small amount of additional waste residue. Sites for additional waste material storage have been identified within the current mining operation area.
Bulk density	Bulk density determinations were undertaken at three separate points, complete core tray determination, calliper measurements and water immersion by ALS. Each method provides a different set of data that had its own bias. The least bias data set is considered to be the ALS water immersion. Over 60% of the samples have an ALS bulk density value; every sample has an ALS bulk density value from the 2011 drilling onwards. The ALS method (OA-GRA08) consists of-

Criteria	Commentary
	• Sample was dried • Placed into a stocking • Weighed in air • Weighed in water • No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal. Average ALS bulk density values for eight lithology units that were further broken up by the three weathering zones were determined and assigned to corresponding blocks in the Resource model. Any extreme and unrealistic values were not used to generate the average value.
Classification	Solids were generated to represent Measured, Indicated and Inferred zones based on drill hole spacing and estimation pass. Measured blocks are made up of blocks that were estimated in pass 1 and were in the region of drilling that was on 25m to 50m spacing. Indicated blocks were mostly pass 1 and pass 2 and/or outside the Measured zone with an approximate drill hole spacing of 75m.Blocks supported by a drill hole spacing of 100m were generally classified as Inferred. Nearly all of these blocks were estimated in the first two passes. The majority of Measured material is contained within the Transitional zone, with the bulk of the Indicated and Inferred material being within the primary zone.
Audits or reviews	The geology and estimation process and results have been internally audited.
Discussion of relative accuracy/ confidence	The bulk of the classified mineralisation is within the defined mineralised zones. A major risk would be any significant changes to the modelled mineralised zone due to faulting or differences in lithology that have not been accounted for in this model. The majority of the mineralised material within the current pit shell is well supported by drilling, so any changes in the mineralisation or lithology solids should only have a localised impact and not alter the overall Resource. The presence of coarse gold needs to be reviewed and the impact of this on sampling and Resource estimation need to be determined.

Inca de Oro

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	The Inca de Oro (IDO) deposit is mostly drill tested by diamond core drilling with occasional reverse circulation drilling. The most recent drill program, between the 14 April 2011 and the 13 March 2012, six drilling campaigns were completed (among which four campaigns were undertaken using the diamond core drilling method while the remaining two programs were performed using the reverse circulation technique). A total of 33.812 meters, corresponding to 98 drill-holes, was drilled and divided as follows:

Criteria	Commentary
	• Geo-metallurgy campaign: 39 diamond drill-holes were completed for a total of 16,146.37m drilled. • Hydrology campaign: 8 reverse circulation drill-holes were completed for a total of 1,240 meters drilled. • Geotechnical campaign: 6 diamond drill-holes were completed for a total of 1,746.4m drilled. • Resource extension campaign: 5 diamond drill-holes were completed for a total of 2,254.35m drilled. • Condemnation campaign: 15 reverse circulation drill-holes were completed for a total of 3,741.50m drilled. • Chalcocite campaign: 25 diamond drill-holes were completed for a total of 8,682.30m drilled. The above drilling comes in addition to the drilling undertaken by CODELCO between the years 2004 and 2009. CODELCO completed a total of 160 drill-holes for a total drilled of 55,892m. In 2004, 49 drill-holes totalizing 9,828m using the reverse circulation technique were completed. Between 2005 and 2006, 90 drill holes, mixing drilling techniques (reverse circulation and diamond drilling) and drilling diameter (HQ and NQ diameters) were completed for a total of 41,567m. In 2009, CODELCO completed 21 diamond drill holes, in diameters PQ and HQ for a total of 4,497 meters. Samples for this project are 1.5m in length (except for the hydrology campaign where sampling was done on a 1m basis). All reverse circulation samples were weighted. Other work studies between 2002 and 2007: • Geochronology: 4 samples were analysed and dated • Petrography: 8 samples • Porosity: 145 samples were analysed • 600 samples were analysed for alteration with PIMA • Point load testing: 144 measurements were done • A study on the deposition of gravel and ash layer overlying the deposit • In 2006, was generated a geotechnical model for the deposit • Geophysics campaigns with some magnetometry profiles completed In the 2011-2012 campaign, a total of 16,238 samples were analysed. In addition to these analyses, 1,021 CODELCO pulps derived from samples obtained during the 2004-2009 drilling campaigns and located in the chalcocite-rich enrichment zone were sent for sequential copper analyses. These analyses come in addition to the 29,401 samples analysed by CODELCO between 2004 and 2009. Subsamples were taken from splits of the reverse circulation chips or half of the core samples and sent to ALS for sample preparation and analysis.
Drilling techniques	All drilling has been undertaken with diamond core technique and occasional reverse circulate drilling. From April 2011, all exploratory and Infill drilling was performed with standard PQ-sized tube (equivalent to core diameter of 85mm).
Drill sample	Diamond core recovery is calculated by measuring the length of the recovered

Criteria	Commentary
recovery	core and comparing this with the theoretical sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. Any loss of core is assigned to either core loss due to drilling error or bad ground (fault material). An analysis of the assay data does not indicate that there is any correlation of core loss to high or low grade intervals. Core loss is frequently associated with structures (faults). RC sample recoveries are calculated from estimation the theoretical weight of the material collected compared with the actual weight. If RC samples are wet, they are allowed to dry before determination of actual weight. An analysis of the assay data and recoveries does not indicate that there is any correlation of core loss to high or low grade intervals. The lithologies within the mineralised zone are competent and this is reflected in the good recoveries achieved, the average recovery from all drilling programs exceeded 95%.
Logging	Each day a geologist inspects the core at the drill site and identifies the major zones and lithologies. Basic geotechnical data (recovery and RQD) is taken at rig site by the rig controller. The core is then transported to the core yard using specifically designed cages to ensure the security of the samples. Core received at the core yard is checked for completeness and to ensure that all core block depths are correct. Core was photographed wet and in direct sunlight. The photographs were downloaded onto the site server and are available for checking and validation. Geological logging is completed by site geologists using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. The supervising geologist also marks core for cutting. A detailed logging is undertaken after receiving assay results from the lab. All logged data is recorded onto paper forms. All paper forms are entered into the local Excel database. There are separate logging sheets to capture- • Basic drill-hole information • Geo-technical • Lithology • Weathering (oxidation state) • Alteration • Mineralisation • Veinlets • Structure On completion of mark up and geotechnical and geological logging, core is then marked for sample splitting.
Sub-sampling techniques and sample preparation	Reverse circulation chip samples were collected from each 1m sampling interval from the rig mounted cyclone. This sample was riffle split to produce a sample that represents 12.5% of the initial sample collected. When a field duplicate is required (1 in 30) another 12.5% sample was collected from the splitter. A reference sample is collected from the splitter, this represents 25% of the

Criteria	Commentary
	primary sample. Any wet RC chip samples were allowed to dry before any sampling using the riffle splitter was attempted. This is to reduce the potential for sample loss or contamination due to wet samples. Only a very small number of RC samples were wet (1%). The site geologists oversee all sampling and insure that representative samples are collected by defining the cutting line on the core. Sampling consists of cutting the core lengthways using a diamond core saw along a predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals. Half core is packed in plastic bags and labelled with sample ID stapled within sample bag. Listings with sample IDs are generated by the onsite senior geology technician. The logging geologist controls procedure quality control. Where a field duplicate sample was collected (approximately 1 in 25), the half core for assaying was cut again to produce two quarters. One quarter was sent for assay and the other quarter was sent as the duplicate sample. Sample preparation was done at ALS laboratory. Laboratory code for the preparation is PREP-31 • Drying at 110^{0}C in LPG gas ovens controlled by thermostat. • All core samples were crushed to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm.
Quality of assay data and laboratory tests	Analyses were performed by ALS laboratories in Santiago (Chile) and Lima (Peru). Four acid digestion coupled to ICP analyses were undertaken for 33 elements going from silver to zinc (ME-ICP61 lab package). Base metals were analysed in Perth by ME-ICP41 (aqua-regia acid digestion inductively coupled plasma - atomic emission spectroscopy (ICP -AES)) with a sample charge of 20g. The lower detection limit for copper and silver are 0.2g/t and 1g/t while the upper detection limits are 100g/t and 10,000g/t respectively. ME-ICP61, (four acid digest analysed by inductively coupled plasma - atomic emission spectroscopy) method was used to analyse higher grade silver, copper and zinc (greater than 100g/t) for accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively Additionally, cyanide leaching method was performed to obtain oxide gold grades (Au-AA13lab package). Poor correlation with previous drill results twins and a check sample and re-assay program was initiated. Gold fire assay check samples were sent and analysed at ACME laboratory in Santiago. A gross 7,500 samples was sent for check and re-assay analyses. As a result the cyanide leach method was dropped from the assay suite for the project and the gold analysis is entirely completed by fire assay (Au-AA23). Using the Au-AA23 fire assay method with atomic absorption spectroscopy utilising a 50g sample charge. This method has a lower detection limit of

Criteria	Commentary
	0.02g/t and upper detection limit of 100g/t. Final, copper grades were also obtained using the four acid near-total digestion methods coupled to ICP analyses (Cu-AA62 lab package). When copper grades were greater than 0.25%, sequential copper analyses were undertaken (CuCN-AN06, CuR-AN06 and CuS-AN06 lab packages). The AA17 (cyanide leach Atomic absorption spectrometry (AAS)) method was used with a 200g sample charge. The lower detection limit for the AA17 method for copper is 0.01% while the upper detection limit is 15%. All of the analytical techniques applied are considered total or near total. QAQC measures include inserting blanks, standards and duplicate samples within the batches of sample sent to the laboratory. Blanks, standards and duplicates represent 10% of the total samples to analyse. Six base metal standards have been used. Three from the previous drilling and three matrix matched prepared from material onsite. All standards have certificates. Copper grades of these certificates range from 0.22% to 0.90% copper. Analyses of standard results show that 100% of results are acceptable within a 3 standard deviation error and over 90% of analyses are within a 2 standard deviation error. Data is also reviewed over time. Adequate precision and accuracy of results correlate well with time. Blanks were initially made of white quartz and were later on substituted by cement. The inserted blank material did not show any consistent issues with sample contamination. Assay quality control protocols were not developed for the geo-metallurgy and condemnation campaigns. As most of the core was used for destructive sampling and testing. ALS internal results QAQC were relied on in this case. Analyses of duplicate and original samples show that 91% of data present differences no greater than 30%. ALS laboratory also inserted standards, blanks and duplicates independently for quality control. The results did not show any issues with the laboratory. A total of 187 duplicate samples were also inserted in all the drill holes from the chalcocite and resource infill campaigns. Analyses show that 91% of data present differences no greater than 30%. ALS laboratory also inserted standards, blanks and duplicates independently for quality control. The results of the lab standards were also monitored on a batch to batch basis by the senior geologist. The results did not show any issues with the laboratory. Sample batches are monitored for deviation in performance against standards, duplicates, and blanks. If the 3 standard deviation level is reached in any batch then a re-assay is requested.
Verification of sampling and assaying	Significant intercepts are verified by the company Qualified Person. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4.5m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste interval of 4.5m.

Criteria	Commentary
	Onsite, all paper documents related to each drill hole (basic information on the drill hole, down hole survey data, geological logging, geotechnical information, assay results, certificates) are filed and stored in a data room. Geological logging, geotechnical data, density measurement data are transferred from paper format to digital format by technical staff. All data is transferred into a single database built in Excel. It is in process to develop the database in Access supported by a Microsoft SQL server. Once transferred, the digital geological data is checked by the geologist who undertook the logging. Complete data validation is undertaken by the database administrator with automated check. Assays received are checked against the significant mineralisation intercepts for mineralised holes by the site geologists. Gaps in assay sequences are checked and validated. Also excessive deviations of drill holes dips are checked in 3-D while excessive survey interval are also validated. Regular back-ups are done on external hard drives. Installation (underway) of internet connections with optic fibre will allow the automatized back-ups on a server All assay data is accepted as supplied by the laboratory certificates. Where a duplicate sample is taken (quarter core) the original and duplicate assays are averaged to maintain the sample support of half core.
Location of data points	The IDO project, survey work uses the UTM Zone 19 (S) projection and PSAD 56 horizontal and vertical datum. All drill collars and mapping stations are surveyed in this datum. Topographic surveying was ensured by third party registered surveyors. In the field, the crew used a TRIMBLE GPS, model 5800, of double frequency with surveying in real time. The project coordinate system is UTM and datum is PSAD56 with local total station control points established. Surveyed data is geo-referenced using a topographic point (IGM INCA) belonging to the national geodetic network and certified by the Military Geographic Institute of Chile. As part of the drilling quality control process, Comprobe Ltda, a certified (ISO 9001, ISO 14001 and OHAS 18001) company was contracted to check deviation of bore-holes. A high precision, digital recording, SRG gyroscope was used. Data collection was based on 50 m spacing. All data was compiled in a database Equipment is not affected by magnetite, delivers data in real time and makes the appropriate correction of deviation. Data from each drill-hole is received by email the following day of survey in an Excel format. Later on, is received the data in certified paper format. This document corresponds to the print obtained from the field equipment and includes the vertical, horizontal and 3D projections.
Data spacing and distribution	The nominal drill hole spacing within the main copper portion of the deposit is 50x75m.The spacing is quite consistent through the main resource area

Criteria	Commentary
	however the spacing may extend out to 100x100m in some areas of the deposit. Cross sections are spaced at 50m for geologic data aggradation. The Resource Classification applied to the Resource model reflects the variability in drill hole spacing and confidence in the estimation.
Orientation of data in relation to geological structure	All of the RC holes have been drilled vertically, the predominant azimuths for diamond holes are 090°, 180° and 360° while dips vary between -45° and -80°. Most holes are close to perpendicular to the mineralisation. A secondary direction is to the east (090^{o}) has been developed; these holes target additional information in the mineralised zone. The direction of the drilling was designed to best capture the variability of grade and deliver true widths or approximate true widths. Diamond drilling was set to intersect perpendicularly the main porphyry mineralisation. Minor drilling was undertaken in other directions for exploration purposes. The average dip of the ore zone is sub-vertical, 80^{o}-90^{o} .The leach, oxide and chalcocite enriched zone is more planar with slight undulation and zone of copper depletion.
Sample security	Drill core and samples were picked up from the drilling site on a daily basis and transported to the core shed. The geologist recorded the intervals of the core being picked and some brief notes on the core the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is under 24 hour security guard. All logging and bagging of diamond drill samples were done in the core yard. All samples to be dispatched were packed in plastic bags, clearly labelled with a submission number, project and prospect number, sample type and date of dispatch. The samples were accompanied by supporting documentation showing submission number, analysis suite, and number of samples. ALS picks up the samples at site and truck to Vientiane laboratory by ALS employees. On site, prior to dispatch, a log book is signed by ALS and supervising geologists. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples. The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators, manager, and senior geologist onsite. After review the results are loaded into Excel then Access through a merging process. QAQC on import is completed before the results are finalized.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects. AMC (Brisbane) completed a desktop due diligence and resource review in 2009; this study included a review of the drilling, survey, sampling, logging and

Criteria	Commentary
	analytical inputs, QAQC, interpretation and 3D modelling of the IDO resource. This study did not identify any high risk issues. AMC also performed initial mineral resource estimation in 2011 and 2012 on the completed data sets and not high risk issues were identified. A medium risk was identified around the exact boundaries of weathering as the drill density may not be optimal for a detailed estimate of oxides and chalcocite. Also geometry and nature of the oxide material should be finessed and reconciled with supporting data. Some concerns around uncertainty associated with local controls on copper mineralisation within primary mineralisation have also been flagged.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	In March 2011, the Chilean National Corporation for Copper, CODELCO, signed a Joint Venture agreement with Australian listed PanAust Ltd to undertake the feasibility study on the Inca de Oro Project (IDO). PanAust holds 66% interest in the company IDO S.A. and CODELCO hold the remaining 34% The project covers an area of 4698 hectares and includes mining and exploration concessions, established and registered, and in the process of registration, that are owned by IDO SA . The project is covered by a total of 2,398 ha of mining concessions, 2,300 ha of first level exploration concessions. The IDO property mining and exploration concessions have been surveyed and monumented according to the Chilean mining regulations. Per the mining laws of Chile, mining concessions can be held in perpetuity provided that the appropriate annual payments have been made. There are no requirements that the property be put into production within some specified time frame. There are no other surface uses designated for the land which has been acquired for mining purposes at the property, and the owner(s) to the mining rights therefore has first right to the use of the surface, subject to the appropriate permitting.
Exploration done by other parties	A significant amount of modern exploration work has been completed on the IDO property to date including geological mapping, geophysical TEM survey, diamond and RC drilling. An extensive compilation of the data that has been accumulated to date and which has resulted in the definition of a porphyry copper-gold mineral deposit. The first modern exploration drilling activities in the region were undertaken in 1978 in the search for copper, and were centred on the potential beneath the valley gravels. Exploration and drilling was carried out by Utah International employing the same concept that later led to the discovery of the buried Escondida deposit. However, there were only two holes drilled in the valley 4 km west of the town, with no mineralisation encountered. CODELCO exploration team identified the area surrounding the town of Inca de Oro as an exploration target for a porphyry-style deposit with secondary enrichment beneath gravels.

Criteria	Commentary
	Between 1985 and 1988 geological and geophysical surveys (magnetometry, gravimetry and induced polarisation) were carried out, resulting in the identification of two polarisation anomalies: one to the east of the town of Inca de Oro and another around 5.5 km northeast. Between 1991 and 1992 a campaign of reverse circulation (RC) drilling was completed, amounting to a total of thirty holes over the IP anomalies and other prospective areas. The drilling intersected only weak copper mineralisation. The recent drilling comes in addition to the drilling undertaken by CODELCO between the years 2004 and 2009. CODELCO completed a total of 160 drill-holes for a total drilled of 55,892m. In 2004, 49 drill-holes totalizing 9,828m using the reverse circulation technique were completed. Between 2005 and 2006, 90 drill holes, mixing drilling techniques (reverse circulation and diamond drilling) and drilling diameter (HQ and NQ diameters) were completed for a total of 41,567m. In 2009, CODELCO completed 21 diamond drill holes, in diameters PQ and HQ for a total of 4,497 meters. All of the CODELCO data, including drill core, coarse rejects, sample pulps, RC chips and supporting data have been preserved. The previous data sets and core form all drilling is available and stored onsite. 1,021 CODELCO pulps derived from samples obtained during the 2004-2009 drilling campaigns that were located in the chalcocite enrichment zone were sent for sequential copper analyses. The total copper results were compared with the previous results and correlation was 94%. The entire data set obtained from CODELCO is considered to be of high quality and acceptable. Comparative statistics including lognormal correlation of the Orion and Southwestern copper and gold assays indicate little bias in the two data sets. Sectional interpretation on 100m spacing was completed by the previous companies and forms the basis for the current 50m spaced lithology and mineralisation models.
Geology	The IDO deposit is a blind composite porphyry copper-gold type. The entire mineralisation is covered by post mineralisation colluvium that ranges from 30m to 60m in depth. The bedrock geology/ stratigraphy comprise volcanic, volcano-sedimentary and sedimentary formations which date from Jurassic to Quatenary, and plutonic intrusive complexes of the Upper Cretaceous and the Lower Tertiary. The IDO project relates to a porphyry copper system with mineralisation composing copper, gold and molybdenum. The deposit relates to the emplacement of a complex of several successive phases of porphyries with a tonalite to granodiorite composition of the upper Cretaceous period and of dykes of andesitic to latitic composition. The intrusive rock complex has a clear north to northeast orientation and is emplaced in a host rock sequence of sedimentary-volcanic rocks of the lower Cretaceous correlated to the Punta del Cobre Formation. The deposit is characterised by a core of potassic alteration that has been

Criteria	Commentary
	partially replaced by an intermediate argillic alteration and by a sericite alteration associated to hydrothermal veins that contribute pyrite and chalcopyrite to the system. The deposit is a composite porphyry type comprising a zone of copper depletion – leached zone, extensive oxide and acid soluble copper development, a secondary enrichment zone of chalcocite that form over a broad zone of primary sulphide copper. Towards the margins of the intrusive bodies, propylytic facies with epidote and chlorite alteration are seen. The following rock units are recognized at the IDO property, chiefly thorough observation in core: • Andesitic lavas, breccias and tuffs • Fine porphyritic andesites • Andesitic tuff and breccias. • Tonalitic porphyry • Dioritic porphyry Mineralisation at Inca de Oro is associated with a porphyry and breccia complex which intrudes andesitic volcanics. The un-mineralised footwall is composed of andesite tuffs and breccias.
Drill hole Information	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Sampling techniques", "Drilling techniques" and "Drill sample recovery".
Relationship between mineralisation widths and intercept lengths	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-"Orientation of data in relation to geological structure".
Diagrams	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Historical data was manually entered using keyed data entry and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using merging function from electronic assay files from the labs, geologists and technicians including offline data entry forms. The logging data was electronically transferred to the site server and validated by the geologists. After validation on site, data was sent to Brisbane to be entered into the central acQuire database. Field limitations are incorporated with all new data to insure the data is correct data type (character, integer, real) and within limits of the particular record (i.e. x< 50). Data read/ write privileges are limited to the site manager and Brisbane general manager. Assay results were sent electronically from the laboratory directly to the manager onsite for validation and loading into local databases, no manual data entry or intervention is required. The senior geologist checks the QAQC of the received assays and advised the manager to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, senior geologist or manager and if required queries are made with ALS. After a batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. Significant intercept tables for the accepted assay data were produced and distributed as a further confirmation. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4.5m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste interval of 4.5m. The geologists on site reviewed the QAQC data and the received assays against the logging intervals. The received assays were reviewed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. The Brisbane server and database are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter. AMC has conducted a site vista and inspect the core and project area in 2012.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. A high degree of confidence in the geological interpretation can be gained by the minor adjustments required by subsequent drilling programs. The recent drilling program resulted in minor change to the geology model.

Criteria	Commentary
	Local confidence varies depending upon the density of available data points, but is still considered to be acceptable. No assumptions are made regarding the data, all geologic interpretation are based on logged drill hole data, surface mapping and assay results. Site geologists developed a sectional interpretation every 50m for the major lithologies and structures. Alteration, structure, surface mapping, mineralisation style and the assay values were used to generate the major lithological domains. The sections are confirmed with plan and cross section interpretations. The sectional interpretations along with structural information were used to generate a 3D model. Geological interpretation of lithology and oxidation state is based on drilling information nominally spaced at 50 m to 100 m intervals in cross section and 50 m intervals in plan. Two main lithologies were interpreted comprising tonalitic porphyry and andesite In addition; a surface defining the extent of the surficial gravel horizon that overlies the basement rocks hosting mineralisation was created. Site generated a series of surfaces to represent the weathering profile (base of leached zone, base of oxide, base of enrichment. This defines the leached, oxide, chalcocite and primary zones. The weathering surfaces were determined by a combination of geological logging, cyanide soluble copper assaying and consideration of sulphur grades that are reconciled to the logging interpretation Six oxidation states were interpreted comprising leached, green oxide, black oxide, and oxide-dominant mixed, sulphide-dominant mixed, supergene and primary sulphides. A series of lithological solids were generated on site to represent the mineralised package and different deposit lithologies. Weathering surfaces were also generated. These were completed in section using the drill hole sample grades incorporating the reconciled geology interpretations Mineralisation is generally consistent along the north-northeast axis and vertical to sub-vertical down dip for the primary sulphides and a more planer flat for the other weather features. Lithology and alteration envelopes were interpreted on drill section using logged lithology and elemental geochemistry. The gold mineralised envelop was interpolated using a >0.1 g/t gold indicator. Solids were used to separate background copper and gold grades from mineralisation. The solids were generated at ½ the section spacing beyond the last drill section. Down dip mineralisation was extrapolated less than 50m below the deepest data point.
Dimensions	The drilling that has been conducted centred on a + 50 meter grid within the approximate centre of the currently defined mineral deposit (and as contained within a 1000m x 1000m area). Several holes were drilled in long section orientated to the north to better reconcile the geologic interpretation. Scattered peripheral holes have been drilled to the east and west of the main

Criteria	Commentary
	zone as well as on trend to the north in combination with sufficient sterilisation and condemnation drilling. The main mineralised zone is approximately 1,000m long striking to the North, is approximately 200m to 500m in width and dips steeply to the east to approximately 300m below the surface.
Estimation and modelling techniques	No updated resource interpolation was completed for the IDO deposit. The existing 2012 interpolation was re-reported at the same copper cut-off grade. Data validation and preliminary statistical work was completed on the drill holes and geology models in the due diligence phase before acquisition by AMC and they are regarded as have sufficient knowledge of the project data and geology. This data was provided to AMC Brisbane to complete further statistical work and data analysis and resource estimation. The Resource estimate was completed by AMC Brisbane within CAE Studio 3 (Datamine) utilising Ordinary Kriging (OK) as the estimation method. Multiple domains (8) were used to control the estimation and account for different grade distribution and mineralisation characteristics. A boundary analysis on these solids indicated that there was a significant change in grade across the mineralised domain boundaries. The 2012 resource estimate used hard domain boundaries based on the lithological and oxidation wireframes. Some domains were combined to increase the number of composites informing the estimate inside each domain. This resulted in 15 estimation domains. Material interpreted to be leached is excluded from mineral resources, but was estimated. Statistical analysis of drill hole data resulted in the decision to use a composite length of 6.0 m based on the following considerations: • Mining would be a bulk-mining method, with nominally 10 m benches, suggesting a composite length of less than 5 m is too small. • Primary mineralisation is relatively homogenous; also suggesting a composite length of less than 5 m is too small. • Supergene, mixed oxide-sulphide, and oxide mineralisation is more variable than primary mineralisation, but these material types make up a relatively small proportion of the mineral resources. • The median composite length is 1.5 m, so it is logical to consider composite intervals in multiples of 1.5 m. A review of the need for grade capping found that it was appropriate to cap some copper grades. Generally, the grade capping was very conservative, reflecting the lower grades and lack of significant outlier values in the copper grade distribution. No grade caps were considered necessary for gold, silver, molybdenum, iron, or arsenic in any of the other estimation domains Supervisor (Snowden software) and Isatis were used for data analysis and variogram modelling for all 8 domains; in the end some of the domains shared variogram models due to similarities and/or a lack of data to define well formed variograms. Variography was undertaken for copper, gold, silver, molybdenum, iron,

Criteria	Commentary
	arsenic, sulphur, acid-soluble copper, cyanide-soluble copper, and residue copper respectively. Gaussian experimental semivariograms in most cases displayed improved directional semivariogram structures compared to their raw composite counterparts. Where good structural models were developed, these were used to generate raw modelled semivariograms for grade estimation purposes. The predominant major direction of continuity for copper is the northwest to southeast direction for the oxide material, and the north–south direction for the combined mixed oxide-sulphide and supergene material, all with a relatively flat, tabular shape. For primary copper, the major direction of continuity is vertical and oriented northwest to southeast for tonalitic porphyry, and northeast to southwest for andesite. Inca de Oro has variable data density ranging from 30 m by 30 m in the highly drilled areas, to 100 m by 100 m in the less well-drilled areas. A block size of 25 m by 25 m by 10 m was chosen based on the average drillhole spacing in the more densely-drilled areas of the deposit. Grade continuity ranges, as determined from the modelled semivariograms, were in excess of 500 m giving further support to the chosen grid spacing. Grade estimation was undertaken using ordinary kriging. Estimation was undertaken for copper, gold, silver, molybdenum, arsenic, iron, sulphur, acid-soluble copper, cyanide-soluble copper, and residue copper variables. A minimum of four composites and a maximum of 16 composites were used for most grade estimation. To support this decision, an investigation of different sample numbers was conducted to consider the most robust parameters. The copper variable was used to assess the minimum and maximum number of composites due to its economic significance. A quadrant search was not used for the grade estimation process. This decision was made because of the pseudo-regular grid pattern which results in a good drillhole distribution pattern for the grade estimation process. Using the composited assay grades and estimating into 25 m by 25 m by 10 m blocks resulted in the block discretisation of 5 (NS) by 5 (EW) by 2 (RL). This results in an even distribution of points within the block for the volume variance calculations to correctly calculate the kriging weights. A three-pass search was used for grade estimation. Search ellipses used for grade estimation have the same orientation as the main orthogonal directions as defined in the semivariogram modelling process. The first pass ensures good quality resource estimates in the well-informed areas. The second search dimensions are double the first pass with the intention of estimating most of the remaining unestimated blocks due to sparse sample density. The third search dimensions are triple the first pass with the intention of estimating all remaining unestimated blocks due to sparse sample density in some of the subdomains. For all domains, the first pass estimated the majority of the blocks. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tools were comparative statics and SWATH plots by northing easting and mRL. These are plots of the

Criteria	Commentary
	average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Visual comparison of drill holes and the block model grades suggests that the resource model appears to be reasonable, and a robust estimate (Figure 14.3). In general, the resource model block grades correlate well with the composite grades, except where the geometry of the mineralisation domain and the orientation of the search ellipses are complex. Validation indicates that the Resource estimate was an adequate representation of the sample grades, given the estimation process and the distribution of samples. Comparisons of the estimated resource model with previous models indicated that the changes in the tonnes, grades and classification were consistent with the impact of the additional drilling and changes to the mineralisation solids and estimation parameters.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut off used for reporting was selected on economic factors of similar type copper – gold deposits. The cut-off for reporting the rescore is a straight application of a copper % value of 0.20.
Mining factors or assumptions	The resources are constrained within the limits of domained lithology and weathering surface wireframes. The assumed mining method is open pit and the block size takes into account grade dilution within the domain.
Metallurgical factors or assumptions	The copper flotation testwork has confirmed that a conventional flotation circuit using rougher flotation, regrinding of rougher concentrate followed by three stages of cleaner flotation, produces a high quality concentrate (free of deleterious elements) at an average life of mine recovery for copper of 87% and for gold of 70%. Locked cycle tests in sea water indicates copper recoveries between 80-87%, with concentrate copper grade over 25% for low pyrite and higher pyrite ore types. Approximately 75% copper recovery into a 24% copper concentrate was indicated to be achievable for supergene material.
Environmental factors or assumptions	There are no known environmental concerns at this time with regard to the work that has been performed at IDO. Under Chilean environmental regulations, as they apply to mining exploration activities, it will not be required to initiate environmental permitting until it starts actual development (as opposed to exploration) work on the property; it is anticipated there will no difficulty in obtaining the requisite work, and other permits, to develop and mine the property at the appropriate time. Chile is a mining oriented country and mining is both socially and politically viewed favourably. All drill pads and tracks have been rehabilitated and approved by the regulating authorities. Only main trunk roads are maintained at the deposit. Drill collars were marked by monuments in the field. All work was conducted under the guidelines set forth by the “Declaration of Environmental Impact”. Environmental, social, and community issues have been professionally managed through pro-active planning, and the development of environmental management systems and protocols based on baseline studies and modelling.

Criteria	Commentary
	There appears to be no technical environmental issue likely to constrain the proposed operation.
Bulk density	Previous dry bulk density work was completed by oxidation/ weather state and contains over 500 measurements across the data set. The recent bulk density determinations were completed by water immersion on site. Approximately 140 new density measurements were made in the past drilling programs. There is very little difference in density between the various lithology types and material types. The global mean is 2.65g/cm3. The mean oxide material type is 2.41 g/cm^3, enriched supergene is 2.52 g/cm^3 and 2.64 g/cm^3 for primary. The on-site method consists of; • Samples are of 15cm in length and are taken every 25m • Sample was dried • Placed into a stocking • Weighed in air • Weighed in water No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal. An analysis of the density data failed to find strong correlations with mineralisation, weathering or lithology. However a combination of weathering and lithology generally showed good separation of data. This combination was used as the basis to assign density to the Resource model. Average bulk density values for the 8 estimation domains were derived; these have been assigned to corresponding blocks in the resource model.
Classification	The resource classification took into account confidence in the geological interpretation of lithology and oxidation state, drill hole spacing, and a number of geostatistical resource estimation parameters, including the estimation pass, distance to the nearest composite, number of composites used, and the kriging variance divided by the sill, to generate confidence criteria thresholds to code into the block model. These confidence values were then displayed in section and used to guide the digitisation of broad resource category wireframe solids. Resource classification considered the drill hole spacing, geological controls, grade continuity and the robustness of the estimate. The robustness of the estimate is primarily determined by the estimation pass, the lower the pass number (1, 2 or 3) the more robust the estimate. To refine the classification an indicative confidence in resource category wireframes was generate based on visual combination of drill density and drill support at depth. The envelopes were generated by digitising strings on 50m sections and creating wireframes that flagged the block model. The criterion used for the confidence envelopes is as follows. • Measured blocks are generally made up of blocks that were estimated in pass 1 and were in the region of drilling that was <50m spacing and within the measured wireframe.

Criteria	Commentary
	• Indicated blocks were mostly pass 1 and outside the measured zone with a drill spacing between 50- 80m and pass 2 within the indicated wireframe. • Blocks supported by a drill hole spacing of >80m, pass 2 in the inferred wireframe and pass 3 RL were classified as Inferred • All remaining blocks are undefined classification.
Audits or reviews	The data review and resource estimate was completed by an external consultancy, AMC Brisbane under the supervision of the general manager geology. The work and resulting resource model was peer reviewed both internally by AMC and within PanAust. AMC conducted a site visit in 2012.
Discussion of relative accuracy/ confidence	The model reported provides a reasonable global estimate of the available copper and gold resources. The model has been validated visually against drilling and statistically against input data on a domain and swath basis. Actual local estimates are likely to vary with the current drill spacing and considerable in-fill will be required to raise the confidence to measured and indicated to support further pit optimisation and scheduling work. Geo -metallurgy and ore types need to be developed as the current ore-types based on oxidation could be too broad. Assess the variability of oxide copper and chalcocite mineralisation boundaries as pockets of the zones could be copper depleted along faults. This will be achieved by completing additional drilling to target oxide mineralisation and the interface with the enriched sulphide mineralisation. Drill spacing of 25m could be required for adequate resolution of the areas of mineralisation. Another risk is the modelling of the lithology and alteration domains. The impact of the possible variability of these solids potentially will be restricted to local variations and should not impact greatly on the global Resource. The domain could over-smooth the metal grade and will likely impact local grade distributions. Evaluation of likely separate estimation models for gold, silver and molybdenum. Also the higher grade primary mineralisation could be further domained and further resolve the copper distribution of the deposit. The grade estimation needs to be combined with the structural model to assess the likelihood that mineralisation could terminated across major fault displacements or be affected by weathering along the fault zones. The geostatiscal modelling could prove to be too general and further contribute to smoothing of metal grades. Once further drilling has been accomplished, further detailed data analysis will refine the grade interpolation. The majority of the mineralised material within the current pit shell is supported by drilling.

Carmen

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	The Carmen deposit is dominantly drill tested by diamond core drilling with rare reverse circulation drilling. Between October 2012 and January 2014, a total of 90 diamond drill-holes for a total of 34,194 meters were completed. These drill holes come to the addition of 84 diamond drill-holes which were completed by three different companies that held the Carmen concession prior to PanAust. These drill holes include: • 23 drill holes (for a total of 2,989m drilled) completed in 1988-1990 by Orion. • 33 drill holes (for a total of 11,796m drilled) completed in 1996-1997 by Southwestern Mining. • 28 drill holes (for a total of 12,377m drilled) completed in 2007-2008 by Peñoles. In the 2012-2014 campaign, and to date, 18,259 sample analyses were received (some results are still pending). These analyses come in addition to analyses completed by prior companies • Orion analysed 2060 core samples for gold and total copper. There are no certificates for these analyses. • Southwestern analysed 8420 core samples for gold, Ag and molybdenum, copies of ordinals assay certificates are on record • Finally, Peñoles analysed 6643 core samples for gold, total Copper and 33 other elements (by ICP). All certificates are present Among the 90 drill holes completed during the 2012-2014 campaign, 78 served exploratory and infill purposes (for a total of 31,468.7m) and 12 drill holes were done for metallurgy purposes (for a total of 2,725m) All exploratory and infill drill holes were systematically sampled on a 1.5m basis. Subsamples were taken from splits of half of the core samples and sent to ALS for sample preparation and analyses.
Drilling techniques	All drilling has been undertaken with diamond core technique. Until December 2013, all exploratory and Infill drilling was performed with standard NQ-sized tube (equivalent to core diameter of 47.6mm). From December 2013, drilling was undertaken with HQ size rods (diameter of core is of 63.5mm). Metallurgical drilling campaigns (2 took place in 2013) were performed with HQ and PQ diameter (which corresponds to a core diameter of 85.0mm).
Drill sample recovery	Diamond core recovery is calculated by measuring the length of the recovered core and comparing this with the theoretical sample length. Any over or under runs is carried to subsequent samples to account for minor core block errors or stick up. Any loss of core is assigned to either core loss due to drilling error or bad ground (fault material). An analysis of the assay data does not indicate that there is any correlation of core loss to high or low grade intervals. Core loss is always associated to structure (faults). The average recovery from all drilling programs exceeded 98%.
Logging	Basic geotechnical data (recovery and RQD) is taken at rig site by the rig controller. Core received at the core yard is checked for completeness and to ensure that all core block depths are correct. Core is then photographed.ensure that all core block depths are correct.

Criteria	Commentary
	Geological logging is completed by site geologists using standard logging sheets and well defined look up tables to ensure that all data is collected consistently. The supervising geologist also marks core for cutting. A detailed logging is undertaken after receiving assay results from the lab. All logged data is recorded onto paper forms. All paper forms are entered into the actual Excel database. There are separate logging sheets to capture- • Basic drill-hole information • Lithology • Weathering (oxidation state) • Alteration • Mineralisation • Veinlets • Structure On completion of logging and mark up, density measurements are undertaken. To date, over 4500 density data has been taken. Core is then split.
Sub-sampling techniques and sample preparation	The site geologists oversee all sampling and insures that representative samples are collected by defining the cutting line on the core. Sampling consists of cutting the core lengthways using a diamond core saw along a predetermined line to generate two halves of core. Where the core was not solid, a divider is placed along the centre of the broken core to equally separate the material. Once cut or separated, the left hand side of the core or material for the selected intervals. Half core is packed in plastic bags and labelled with sample ID stapled within sample bag. Listings with sample IDs are generated by the onsite senior geology technician. The logging geologist controls procedure quality control. Where a field duplicate sample was collected (approximately 1 in 25), the half core for assaying was cut again to produce two quarters. One quarter was sent for assay and the other quarter was sent as the duplicate sample. Sample preparation was done at ALS laboratory. Laboratory code for the preparation is PREP-31 • Drying at 110^{0}C in LPG gas ovens controlled by thermostat. • All core samples were crushed to better than 70% passing 2mm using Rocklabs Boyd crushers with a rotary sample divider attached. • The samples were pulverised using the Labtech ESSA LM2 pulverising mills to better than 85% passing 75µm.
Quality of assay data and laboratory tests	Analyses were performed by ALS laboratories in Santiago (Chile) and Lima (Peru). Four acid digestion coupled to ICP analyses were undertaken for 33 elements going from silver to zinc (ME-ICP61 lab package). Base metals were analysed in Perth by ME-ICP41 (aqua-regia acid digestion inductively coupled plasma - atomic emission spectroscopy (ICP -AES)) with a sample charge of 20g. The lower detection limit for copper and silver are 0.2g/t and 1g/t while the upper detection limits are 100g/t and 10,000g/t respectively. ME-ICP61, (four acid digest analysed by inductively coupled plasma - atomic emission spectroscopy) method was used to analyse higher grade silver, copper and zinc

Criteria	Commentary
	(greater than 100g/t) for accuracy and precision. The lower detection limit for copper, lead and zinc are all 0.001% while the upper detection limits are 40%, 20% and 30% respectively Additionally, cyanide leaching method was performed to obtain oxide gold grades (Au-AA13lab package). Poor correlation with previous drill results twins and a check sample and re-assay program was initiated. Gold fire assay check samples were sent and analysed at ACME laboratory in Santiago. A set of 7,500 samples was sent for check and re-assay analyses. As a result the cyanide leach method was dropped from the assay suite for the project and the gold analysis is entirely completed by fire assay (Au-AA23). Using the Au-AA23 fire assay method with atomic absorption spectroscopy utilising a 50g sample charge. This method has a lower detection limit of 0.02g/t and upper detection limit of 100g/t. Final, copper grades were also obtained using the four acid near-total digestion methods coupled to ICP analyses (Cu-AA62 lab package). When copper grades were greater than 0.25%, sequential copper analyses were undertaken (CuCN-AN06, CuR-AN06 and CuS-AN06 lab packages). The AA17 (cyanide leach Atomic absorption spectrometry (AAS)) method was used with a 200g sample charge. The lower detection limit for the AA17 method for copper is 0.01% while the upper detection limit is15%. All of the analytical techniques applied are considered total or near total. QAQC measures include inserting blanks, standards and duplicate samples within the batches of sample sent to the laboratory. Blanks, standards and duplicates represent 10% of the total samples to analyse. Six base metal standards have been used. All standards have certificates. Copper grades of these certificates range from 0.11% to 0.90% copper. Two gold standards with gold grades of 0.17 and 0.40 g/t gold. Analyses of standard results show that 100% of results are acceptable within a 3⍰⍰error and over 97% of analyses are within a 2⍰⍰error. Data is also reviewed over time. Adequate precision and accuracy of results correlate well with time. Blanks were initially made of white quartz and were later on substituted by cement. The inserted blank material did not show any consistent issues with sample contamination. 83.40% of blank samples did not show copper grades greater than 0.003% copper. Analyses of duplicate and original samples show that 73% of data present differences no greater than 30%. ALS laboratory also inserted standards, blanks and duplicates independently for quality control. The results did not show any issues with the laboratory. ALS laboratory also inserted standards, blanks and duplicates independently for quality control. The results of the lab standards were also monitored on a batch to batch basis by the senior geologist. The results did not show any issues with the laboratory. Sample batches are monitored for deviation in performance against standards, duplicates, and blanks. If the 3⍰⍰level is reached in any batch then a re-assay is

Criteria	Commentary
	requested.
Verification of sampling and assaying	Significant intercepts are verified by the company Qualified Person. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4.5m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste interval of 4.5m. Recently there have been 14 twin holes drilled. Generally there is +90% correlation for copper and less for gold. Detailed analysis is pending Onsite, all paper documents related to each drill hole (basic information on the drill hole, down hole survey data, geological logging, geotechnical information, assay results, certificates) are filed and stored in a data room. Geological logging, geotechnical data, density measurement data are transferred from paper format to digital format by technical staff. All data is transferred into a single database built in Excel. It is in process to develop the database in Access supported by a Microsoft SQL server. Once transferred digital geological data is checked by the geologist who undertook the logging. Complete data validation is undertaken by the database administrator with automated checks. Assays received are checked against the significant mineralisation intercepts for mineralised holes by the site geologists. Gaps in assay sequences are checked and validated. Also excessive deviations of drill holes dips are checked in 3-D while excessive survey interval are also validated. Regular back-ups are done on external hard drives. Installation (underway) of internet connections with optic fibre will allow the automatized back-ups on a server All assay data is accepted as supplied by the laboratory certificates. Where a duplicate sample is taken (quarter core) the original and duplicate assays are averaged to maintain the sample support of half core.
Location of data points	All drill collars and mapping stations are surveyed in this datum. The Carmen project, survey work uses the UTM Zone 19 (S) projection and PSAD 56 horizontal and vertical datum. Topographic surveying was ensured by third party registered surveyors. In the field, the crew used a TRIMBLE GPS, model 5800, of double frequency with surveying in real time. The project coordinate system is UTM and datum is PSAD56 with local total station control points established. Surveyed data is geo-referenced using a topographic point (IGM INCA) belonging to the national geodetic network and certified by the Military Geographic Institute of Chile. As part of the drilling quality control process, Comprobe Ltda, a certified (ISO 9001, ISO 14001 and OHAS 18001) company was contracted to check deviation of bore-holes. A high precision, digital recording, SRG gyroscope was used. Data collection was based on 50 m spacing. All data was compiled in a database Equipment is not affected by magnetite, delivers data in real time and makes

Criteria	Commentary
	the appropriate correction of deviation. Data from each drill-hole is received by email the following day of survey in an Excel format. Later on, is received the data in certified paper format. This document corresponds to the print obtained from the field equipment and includes the vertical, horizontal and 3D projections.
Data spacing and distribution	The nominal drill hole spacing within the main copper portion of the deposit is 50x75m.The spacing is quite consistent through the main resource area however the spacing may extend out to 100x100m in some areas of the deposit. Cross sections are spaced at 50m for geologic data aggradation. The Resource Classification applied to the Resource model reflects the variability in drill hole spacing and confidence in the estimation.
Orientation of data in relation to geological structure	The majority of drill holes that pass through the mineralised zones are drilled to the west (270deg) at a dip of -60deg. These holes are close to perpendicular to the mineralised packages. A secondary direction is to the east (090deg) has been developed; these holes target the eastern boundary of the mineralised zone. The direction of the drilling was designed to best capture the variability of grade and deliver true widths or approximate true widths. Diamond drilling was set to intersect perpendicularly the main structural feature (i.e. the Canada Fault). Minor drilling was undertaken in other directions for exploration purposes. The average dip of the ore zone is sub-vertical, 80^{o}-90^{o} . The leached and transitional zones are more planar and tend to mirror topography.
Sample security	Drill core and samples were picked up from the drilling site on a daily basis and transported to the core shed. The geologist recorded the intervals of the core being picked and some brief notes on the core the core is moved from the rig. Core received at the core yard is cleaned and checked for completeness. The core yard is a fully fenced and secure location for all core storage; this facility is under 24 hour security guard. All logging and bagging of diamond drill samples were done in the core yard. All samples to be dispatched were packed in plastic bags, clearly labeled with a submission number, project and prospect number, sample type and date of dispatch. The samples were accompanied by supporting documentation showing submission number, analysis suite, and number of samples. ALS picks up the samples at site and truck to Vientiane laboratory by ALS employees. On site, prior to dispatch, a log book is signed by ALS and supervising geologists. Supporting dispatch documentation is emailed to ALS. When the samples were received at the laboratory, a receipt notification was sent to the database administrator confirming the number of samples. The chain of custody is maintained by ALS once the samples are collected from site and a full audit trail for every sample is available through the ALS' Webtrieve application. Assay results are emailed to the database administrators, manager, and senior

Criteria	Commentary
	geologist onsite. After review the results are loaded into Excel then Access through a merging process. QAQC on import is completed before the results are finalized.
Audits or reviews	The sampling process is fully documented as a standard company process and has been reviewed previously for other projects. AMC (Brisbane) completed a desktop Resource review in 2011; this study included a review of the drilling, survey, sampling, logging and analytical inputs, QAQC, interpretation and 3D modelling of the Carmen resource. This study did not identify any high risk issues. AMC also performed initial mineral resource estimation in 2011 and 2013 on the completed data sets and not high risk issues were identified. A medium risk was identified around the correlation of gold leach assays and fire assays in the data and this finding led to a gold re-assay program and check of gold assay in the recent drilling.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The concessions that have been acquired by PanAust cover an area totalling 970 hectares. PanAust owns a 100 per cent interest in the property through a Chilean based subsidiary. The Carmen Property mining concessions have been surveyed and monumented according to the Chilean mining regulations. Per the mining laws of Chile, mining concessions can be held in perpetuity provided that the appropriate annual payments have been made. There are no requirements that the property be put into production within some specified time frame. There are no other surface uses designated for the land which has been acquired for mining purposes at the Carmen Property, and the owner(s) to the mining rights therefore has first right to the use of the surface, subject to the appropriate permitting. The owners likewise will have full legal rights to the water that is found within the Carmen property. There are no known environmental concerns at this time with regard to the work that has been performed at Carmen. Under Chilean environmental regulations, as they apply to mining exploration activities, the company will not be required to initiate environmental permitting until it initiates actual project development work on the property. It is expected that the company will have no difficulty in obtaining the requisite work, and other permits, to develop and mine the property at the appropriate time.
Exploration done by other parties	The Carmen property has been previously mined on a small scale by “pirquineros” for placer gold and for gold (copper?) from a number of shallow trenches and a small shaft. The production that has been taken from the property is not known, but is estimated to be minimal. The Pique (Shaft) Carmen is located on/or proximal to a narrow zone of silicified copper oxide stained gold bearing vein/veinlets and which was glory-holed over a zone + 3m wide x 120m long and 30m deep.

Criteria	Commentary
	A significant amount of modern exploration work has been completed on the Carmen property to date including geological mapping, a magnetometer survey, extensive trenching and sampling, diamond drilling (61,356 meters in 174 holes), and an extensive compilation of the data that has been accumulated to date and which has resulted in the definition of a copper-gold mineral deposit. The previous exploration has been completed by three different companies in the project area. The prospect was discovered between 1988 and 2008 84 diamond drill-holes which were completed by three different companies that held the Carmen concession prior to PanAust. These drill holes include: • 23 drill holes (for a total of 2,989m drilled) completed in 1988-1990 by Orion Mining. • 33 drill holes (for a total of 11,796m drilled) completed in 1996-1997 by Southwestern Mining. • 28 drill holes (for a total of 12,377m drilled) completed in 2007-2008 by Peñoles Exploration. None of the Orion drill core has been preserved. Approximately 80% of the Southwestern drill core is available and stored onsite and all of the Peñoles core is available and stored onsite. The mineralised Orion holes have been redrilled over the subsequent exploration programs and correlation to the previous assays is considered acceptable. Documentation, drill logs and assay certificates exist for the Southwestern data and selected mineralised holes were redrilled in subsequent Peñoles and PanAust exploration programs. The Southwestern data is considered acceptable. Comparative statistics including lognormal correlation of the Orion and Southwestern copper and gold assays indicate little bias in the two data sets. The Peñoles data is considered the best documented and validated data set. All of the Peñoles data is considered to be of high quality and acceptable. Orion/Southwestern/Peñoles mapped the property (2,000m x 800m) at a scale of 1:1,000 and generated sectional geologic modelling. Geophysics: Between 1996 and 1997, Southwestern Mining completed 3 geophysical campaigns. First one was of magnetometry. 100m-spaced lines were set with 25m-spaced stations. Positive anomalies were obtained of up to 900 nT. Two other campaigns were of TEM in drills with induction frequency of 25 Hz, line spacing of 200m and a depth limit of 150-200m. Contacts between low resistivity and medium resistivity were localized. Peñoles undertook to measure physical parameters such as, chargeability, resistivity and magnetic susceptibility in core dills completed by Southwestern. Sectional interpretation on 100m spacing was completed by the previous companies and forms the basis for the current lithology and mineralisation

Criteria	Commentary
	models.
Geology	The Carmen Property is underlain primarily by volcanic rocks of the Punta del Cobre Formation which consist of andesitic lava flows, breccias and tuffs. They are overlain, discordantly, by calcareous sediments of the Chañarcillo Group. These units are intruded by a tonalitic porphyry, which has been termed the Carmen Porphyry. The following rock units are recognized at the Carmen property: • Andesitic lavas, breccias and tuffs • Fine porphyritic andesites • Dacitic tuffs • Latitic tuffs • Grey calcareous sandstones • Dark red andesitic flows and breccias • Calcareous sandstones and tuffs • Andesitic tuff and breccias. • Tonalitic Porphyry The geology of the property consists of a complex of tonalitic (and felsic) porphyries which intrude the volcanic-sedimentary sequence. A major fault, the Canada Fault, which trends 350 º to due north and dipping -75ºeast to subvertical, cross cuts the centre of the mineralization. This major through-going fault is probably closely related to the northerly trending gold-copper vein swarms. The Carmen copper-gold mineralisation is hosted within a hornblende tonalite/ feldspar porphyry intrusive complex into andesitic volcanics and metasomatised tuffaceous and calcareous sediments (skarns). The intrusive/volcanic/sedimentary host rocks show various alteration assemblages which is characteristic of a low sulphadation system that overprints an earlier iron-oxide-copper-gold type system. The copper gold mineralization at Carmen is associated with the intrusive, intrusive-andesite contact zone and into the skarn altered tuffs and sediments. The main type of mineralisation is a network of quartz-calcite-sulphide-magnetite-(hematite) bearing veinlets have been developed with the previous mineralized stockworks formed by early magnetite-chlorite veinlets.
Drill hole Information	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-“Sampling techniques”, “Drilling techniques” and “Drill sample recovery”.
Data aggregation methods	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-“Sampling techniques”, “Drilling techniques” and “Drill sample recovery”.
Relationship between	No exploration results are included in this release, all information relates to the

Criteria	Commentary
mineralisation widths and intercept lengths	Mineral Resource, as such this section is not relevant. Comments relating to drill hole information relevant to the Mineral Resource Estimate can be found in Section1-“Orientation of data in relation to geological structure”.
Diagrams	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Balanced reporting	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Other substantive exploration data	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.
Further work	No exploration results are included in this release, all information relates to the Mineral Resource, as such this section is not relevant.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Historical data was manually entered using keyed data entry and validated by site geologists, senior geologists and database administrators. The rest of the data capturing was completed electronically using merging function from electronic assay files from the labs, geologists and technicians including offline data entry forms. CODELCO data was validated by verifying a total of 5% of the assay database and lithology codes. Against the assay certificates and drill logs. The results produced 100% correlation. The logging data was electronically transferred to the site server and validated by the geologists. After validation on site, data was sent to Brisbane to be entered into the central acQuire database. Field limitations are incorporated with all new data to insure the data is correct data type (character, integer, real) and within limits of the particular record (i.e. x< 50). Data read/ write privileges are limited to the site manager and Brisbane general manager. Assay results were sent electronically from the laboratory directly to the manager onsite for validation and loading into local databases, no manual data entry or intervention is required. The senior geologist checks the QAQC of the received assays and advised the manager to accept or reject the batch depending on the QAQC performance. Any concerns with the QAQC results are reviewed by the site geologists, senior geologist or manager and if required queries are made with ALS. After a batch of results was accepted in the database, significant intercept tables were produced and distributed; these are reviewed and considered in relation to other intercepts and the logged geology as a further confirmation. Significant intercept tables for the accepted assay data were produced and distributed as a further confirmation. All significant intercepts start and end on a grade interval (0.3% copper) with a minimum length of 4.5m, a total grade of or greater than 0.3% copper, and a maximum continuous internal waste

Criteria	Commentary
	interval of 4.5m. The geologists on site reviewed the QAQC data and the received assays against the logging intervals. The received assays were reviewed to ensure they match the drill hole logging and the geological and mineralisation interpretation. Reconciliation of the new assays and the neighbouring holes on section was done to ascertain the results. Once validated, the QAQC data along with the logged data was reported in a drill hole report. The Brisbane server and database are automatically backed up on daily basis with the process being overseen by the IT department.
Site visits	The project site and the core processing areas are regularly visited by the competent person. Site processes, QAQC and geological interpretation are reviewed and guidance given on future programs or developments. The frequency of these visits varies, however it would average at least one visit every quarter. AMC has conducted a site vista and inspect the core and project area in 2012.
Geological interpretation	The geology model has evolved over multiple drilling programs and completed to a high standard. A high degree of confidence in the geological interpretation can be gained by the minor adjustments required by subsequent drilling programs. The recent drilling program resulted in moderate changed to the geology model. Local confidence varies depending upon the density of available data points, but is still considered to be acceptable. No assumptions are made regarding the data, all geologic interpretation are based on logged drill hole data, surface mapping and assay results. Site geologists developed a sectional interpretation every 50m for the major lithologies and structures. Alteration, structure, surface mapping, mineralisation style and the assay values were used to generate the major lithological domains. The sections are confirmed with plan and cross section interpretations. The sectional interpretations along with structural information were used to generate a 3D model. Site generated a series of surfaces to represent the weathering profile (base of leached zone, base of oxide, ND base of transitional. This defines the leached, oxide, transitional and primary zones. The weathering surfaces were determined by a combination of geological logging, cyanide soluble copper assaying and consideration of sulphur grades that are reconciled to the logging interpretation. A series of lithological solids were generated on site to represent the mineralised package and different deposit lithologies. To best represent the higher grade zone, a series of copper% and gold g/t solids were generated. These were completed in section using the drill hole sample grades incorporating the reconciled geology interpretations. Mineralisation is generally consistent along the north south axis and down dip. Mineralised envelopes were interpreted on drill section using copper grade (>0.1% copper) and elemental geochemistry. The gold mineralised envelop was

Criteria	Commentary
	interpolated using a >0.1 g/t gold indicator. Solids were used to separate background copper and gold grades from mineralisation. The solids were generated at ½ the section spacing beyond the last drill section. Down dip mineralisation was extrapolated less than 50m below the deepest data point.
Dimensions	Mineralised veins outcrop with the intrusive and are slightly oxidised (transitional ore type) to 30-50m in depth. The drilling that has been conducted (identified) to date is generally centred on a + 50 meter grid within the approximate centre of the currently defined mineral deposit (and as contained within a 500m x 900m area). A number of holes (infill and check holes) were drilled at intervals varying from 5m to 20m offsets, in either an east-west, or a north south direction. Scattered peripheral holes have been drilled to the east and west of the main zone as well as on trend to the north. The main mineralised zone is approximately 1,000m long striking to the North, is approximately 50m to 200m in width and dips steeply to the east to approximately 300m below the surface.
Estimation and modelling techniques	No updated resource interpolation was completed for the Carmen deposit. The existing 2011 interpolation was re-reported at the same copper cut-off grade. Data validation and preliminary statistical work was completed on the drill holes and geology models. This data was provided to AMC Brisbane to complete further statistical work and data analysis and resource estimation. The Resource estimate was completed by AMC Brisbane within CAE Studio 3 (Datamine) utilising Ordinary Kriging (OK) as the estimation method. Multiple domains (8) were used to control the estimation and account for different grade distribution and mineralisation characteristics. A boundary analysis on these solids indicated that there was a significant change in grade across the mineralised domain boundaries. Compositing to 2m lengths was done on the assay samples. Approximately 50% of the data was sampled at 1.0 m and 30%, at 2.0m intervals and remainder at various interval lengths. Composites were generated down hole and honoured the mineralised envelopes. No top cuts to copper or gold were applied to any of the composite samples for estimation. The Transitional and Primary zones were estimated as separate domains. The mineralisation domains (0.1% copper solid and 0.1 g/t gold) were cut by the weathering zones, resulting in 4 domains to be estimated for copper, gold, silver and molybdenum and 4 domains outside mineralisation to be estimated for background copper, gold, silver and molybdenum. Supervisor (Snowden software) and Isatis were used for data analysis and variogram modelling for all 8 domains; in the end some of the domains shared variogram models due to similarities and/or a lack of data to define well formed variograms. Variography was undertaken for all domains, in the end some of the domains shared variograms due to similarities and/or a lack of data to define well formed variograms, however the domains restricted the samples available for

Criteria	Commentary
	estimation. Omni directional variograms were generated and modelled for the estimated metals. Kriging Neighbourhood Analysis (KNA) was undertaken to better define the estimation parameters. These tests used the Kriging Efficiency, Slope of Regression and Kriging Variance to assisted in defining the best parameters for- • Block size • Search distances • Minimum and maximum number of samples to be used for an estimate • Octant restraints on the sample selection Search distances and directions were omnidirectional and unchanged for each domain. The first pass search ellipse was 75m in the north-south direction by 35m in the east-west direction and 50m vertically. A second pass was run with search ranges doubled. This results in most of the blocks being estimated. The number of samples used during estimation to inform a block ranged from a minimum of 2 to a maximum of 20. The model was validated in section by visually comparing the blocks grades with the sample grades. The other major validation tools were comparative statics and SWATH plots by northing easting and mRL. These are plots of the average block and sample grades for a slice through the mineralisation. These slices are usually in the same orientation as the major search direction. Both validation tools indicated that the Resource estimate was an adequate representation of the sample grades, given the estimation process and the distribution of samples. Comparisons of the estimated resource model with previous models indicated that the changes in the tonnes, grades and classification were consistent with the impact of the additional drilling and changes to the mineralisation solids and estimation parameters.
Moisture	All tonnages stated are dry tonnes. These are based on density determinations that are completed on dry samples. No moisture content estimates have been completed.
Cut-off parameters	The cut off used for reporting was selected on economic factors of similar type copper – gold deposits. The cut-off for reporting the rescore is a straight application of a copper % value of 0.20.
Mining factors or assumptions	The resources are constrained within the limits of domained copper and gold mineralisation wireframes. The assumed mining method is open pit and the block size takes into account grade dilution within the domain.
Metallurgical factors or assumptions	The resource is low sulphur and iron sulphides. Initial metallurgical recovery work has indicated that + 90% copper recovery is attained in the primary mineralisation.
Environmental factors or assumptions	There are no known environmental concerns at this time with regard to the work that has been performed at Carmen. Under Chilean environmental regulations, as they apply to mining exploration activities, it will not be required to initiate environmental permitting until it starts actual development (as opposed to exploration) work on the property; it is anticipated there will no difficulty in obtaining the requisite work, and other permits, to develop and mine the property at the appropriate time. Chile is a mining oriented country

Criteria	Commentary
	and mining is both socially and politically viewed favourably. All drill pads and tracks have been rehabilitated (or are in progress) and approved by the regulating authorities. Only main trunk roads are maintained at the deposit. Drill collars were marked by monuments in the field. All work was conducted under the guidelines set forth by the "Declaration of Environmental Impact". Environmental, social, and community issues have been professionally managed through pro-active planning, and the development of environmental management systems and protocols based on baseline studies and modelling. There appears to be no technical environmental issue likely to constrain the proposed operation.
Bulk density	Bulk density determinations were completed by water immersion on site. Approximately 5,000 density measurements were made in the past drilling. While another 500 have been added in the most recent drilling. There is very little difference in density between the various lithology types and material types. The global mean is 2.65 g/cm3. The mean transitional material type is 2.58 g/cm3 and 2.64 g/cm3 for primary. The on-site method consists of; • Samples are of 15cm in length and are taken every 25m • Sample was dried • Placed into a stocking • Weighed in air • Weighed in water No waxing was undertaken due to the issues with sample preparation and assaying waxed samples. If the method was completed efficiently, the amount of water adsorbed is minimal. An analysis of the density data failed to find strong correlations with mineralisation, weathering or lithology. However a combination of weathering and lithology generally showed good separation of data. This combination was used as the basis to assign density to the Resource model. Average bulk density values for the 8 estimation domains were derived; these have been assigned to corresponding blocks in the resource model.
Classification	Resource classification considered the drill hole spacing, geological controls, grade continuity and the robustness of the estimate. The robustness of the estimate is primarily determined by the estimation pass, the lower the pass number (1-2) the more robust the estimate. To refine the classification an indicative confidence in resource category was generate based on visual combination of drill density and drill support at depth. The envelopes were generated by digitising strings on 50m sections and creating wireframes that flagged the block model. The criterion used for the confidence envelopes is as follows. • Measured blocks are generally made up of blocks that were estimated in pass 1 and were in the region of drilling that was <50m spacing and above the 2050 mRL

Criteria	Commentary
	• Indicated blocks were mostly pass 1 and outside the measured zone with a drill spacing between 50- 80m and between the 1975 and 2050 mRL • Blocks supported by a drill hole spacing of >80m, pass 2 and <1975m RL were classified as Inferred
Audits or reviews	The data review and resource estimate was completed by an external consultancy, AMC Brisbane under the supervision of the general manager geology. The work and resulting resource model was peer reviewed both internally by AMC and within PanAust.
Discussion of relative accuracy/ confidence	The model reported provides a reasonable global estimate of the available copper and gold resources. The model has been validated visually against drilling and statistically against input data on a domain and swath basis. Actual local estimates are likely to vary with the current drill spacing and considerable in-fill will be required to raise the confidence to measured and indicated to support further pit optimisation and scheduling work. Geo -metallurgy and ore types need to be developed as the current ore-types based on oxidation could be too broad. Another risk is the modelling of the copper and gold mineralised domains (0.1% copper and 0.1g/t gold solids). The impact of the possible variability of these solids potentially will be restricted to local variations and should not impact greatly on the global Resource. The domain could over-smooth the metal grade and will likely impact local grade distributions. The grade estimation needs to be combined with the structural model to assess the likelihood that mineralisation could terminated across major fault displacements. The geostatistical modelling could prove to be too general and further contribute to smoothing of metal grades. Once further drilling has been accomplished, further detailed data analysis will refine the grade interpolation. The majority of the mineralised material within the current pit shell is supported by drilling, but significant drilling will be required to elevate the confidence of the estimate.

JORC 2012 Edition Table 1 Reporting for Ore Reserves

Phu Kham

Section 4. Estimation and Reporting of Ore Reserves

Criteria	Explanation
Mineral Resource estimate for conversion to Ore Reserves	The Phu Kham Ore Reserve estimate is based on the 31 December 2013 Mineral Resource estimate. Dan Brost is the Competent Person responsible for the Mineral Resource estimate. The Mineral Resource estimate is derived from a model prepared by AMC Consultants Pty Ltd (AMC). PanAust geology staff familiar with the deposit were responsible for providing guidance to the geological interpretation and domain wireframe generation used in the creation of the model. The Mineral Resource estimate has changed from the previous estimate as a result of depletion from mining during 2013. The Mineral Resource model is an Ordinary Kriged model having a parent block size of 20 m by 20 m by 10 m (East x North x RL) with regular sub-blocks of 5 m by 5 m by 5 m at domain boundaries. Densities are assigned by domain using the mean value in each respective lithology/oxide combination. The Mineral Resource model is re-blocked at 10 m by 10 m by 10 m dimensions for use in the Ore Reserve model. Material with Black Shale lithology is not considered to be Mineral Resource. Although it can contain copper and gold mineralisation, the recovery of metal from this material is very low which prevents it from being economic. The Mineral Resource estimate is reported inclusive of the Ore Reserve estimate.
Site visits	PanAust staff responsible for the preparation of the Ore Reserve estimate have made several visits to the Phu Kham Mine. The Competent Person has visited the Phu Kham Mine.
Study / Project Status	Phu Kham is an active mining operation. Mining of the gold enriched, copper depleted cap to the deposit commenced in 2005 followed by mining from the copper–gold pit in 2006. Excavation of the open pit has progressed to 390 mRL with most mining activity occurring along the western pit wall. The copper–gold ore processing plant has operated since March 2008 at rates that have risen from 12 Mtpa to a planned 18.5 Mtpa in 2014. Plant modifications have enabled the capacity increase and the recently completed Increased Recovery Project has enabled higher metallurgical recovery since mid-2013. The Ore Reserve estimate is supported by an open pit design, detailed budget estimate, life of mine plan and operational history that exceeds the level of detail and precision found in a Feasibility Study.
Cut-off parameters	The economic cut-off was determined from the non-mining breakeven value for those Ore Reserve model blocks having a minimum 0.2% Cu grade. The economic breakeven value was calculated for each block in the Ore Reserve model and incorporates metal price, selling cost, product transport cost, ore processing cost, tailings storage facility construction cost and site administration cost. Revenue from copper and gold is considered in this calculation. Only blocks having a positive value and a minimum grade of 0.2% Cu are reported in the Ore Reserve. This is consistent with mine operating practice. A lower cut-off grade may be economically viable but has not been applied due to uncertainty in metal recovery estimates at very low copper grades (below 0.2% Cu).

Criteria	Explanation
	Test work is being performed to determine the metallurgical recovery characteristics of this material. Mining costs are not considered in the cut-off grade criteria. The impact of mining costs is accounted for in the calculation of the optimal pit shell including differences between ore and waste mining due to variance in drilling and blasting and truck haulage distance.
Mining factors or assumptions	The Phu Kham open pit is mined as a conventional truck and shovel operation with 10 m high benches. The majority of the ore and waste material is drilled and blasted before being excavated by hydraulic shovels and excavators. There has been no change to mining practice since the previous Ore Reserve estimate. Mining practices are well established and appropriate to the style of deposit. Slope design recommendations for the final pit walls have been provided by geotechnical consultants (PSM) and the Company's technical staff. Slope design parameters are based on drill hole information, mapping, observation and site experience with pit wall behaviour. The recommended slope designs were applied to the pit optimisation and design used for the Ore Reserve estimate. The Mineral Resource model is converted to an Ore Reserve model as part of the Ore Reserve estimation process. The Mineral Resource model is re-blocked at 10 m x 10 m x 10 m to align with the chosen SMU and mining bench height. Dilution is included in the Ore Reserve estimate through two mechanisms. • The Mineral Resource model grade is interpolated as an average value inside blocks with dimensions of 20 m by 20 m in plan and 10 m vertically. Mining selectivity with the equipment working in the pit is considered to be smaller at approximately 10 m by 10 m by 10 m. Hence the whole block average grade effectively incorporates internal dilution into the grade estimate. • External (edge) dilution is quantified in the Ore Reserve model for each regularised 10m by 10m by 10m block. A dilution skin of 1.5 m from each neighbouring block to the east, west, north and south, at the grades of the adjacent blocks, is applied to calculate a diluted grade for each of the Ore Reserve model blocks. The process preserves the total mass of material with each block gaining and losing the same volume of material. This skin dilution process makes allowance for dilution across the boundaries between ore and waste zones. The Ore Reserve is not modified to account for ore loss. The ore zones are generally broad and well defined by close spaced grade control drilling prior to mining. Given the ore zone geometry it is considered that ore loss is minimal. The Inferred Mineral Resource is not considered for conversion to Ore Reserve. The pit design contains approximately 3 Mt of Inferred Mineral Resource and this small quantity will have no significant impact on pit design. The Ore Reserve is estimated within a pit design completed by mining engineering consultants AMDAD Pty Ltd in 2013 under the direction of PanAust. The pit design includes ramps and safety berms on the pit walls. The design is similar to prior pit designs but does not extend as deep in the north end of the pit. Optimisation of the pit shape was completed using the Lerchs Grossman algorithm as implemented in the Whittle 4X software. Optimisation considered a range of metal

Criteria	Explanation
	price scenarios: • copper prices from $2.50/lb to $4.00/lb • gold prices from $1,100/oz to $1,600/oz. The pit shell that generated the highest discounted cashflow was selected as basis for the final pit limit design. The optimisation process showed that this optimal pit shell is relatively insensitive to metal price assumptions and contains over 80% of the Mineral Resource. The detailed open pit design prepared from the optimal pit shell was used for Ore Reserve estimation. A LOM production schedule was prepared during 2013 using the open pit design and 31 December 2012 Ore Reserve model that forms the basis for the current Ore Reserve estimate. The production schedule showed that ore can be delivered to the processing plant in sufficient quantity for each year of the mine life to satisfy the assumptions associated with the costs used in the Ore Reserve estimate. The waste movement required to extract the Ore Reserve is 248 Mt inclusive of 15 Mt of waste rock from the Phu Kham quarry. The quarry waste rock was excluded from the previous Ore Reserve estimate. Infrastructure exists at the Phu Kham site to support the open pit mining operation and includes maintenance workshops, refuelling facilities, an engineered tailings storage facility (TSF), waste dumps and ore stockpiles.
Metallurgical factors or assumptions	The Phu Kham ore treatment plant commenced operation in 2008 and uses a conventional crush, grind and flotation process to recover a saleable copper concentrate that contains copper, gold and silver. The concentrate typically grades 23% copper. The ore processing circuit has been designed and progressively enhanced to suit the characteristics of the Phu Kham deposit. The metallurgical recovery estimate includes the copper recovery uplift from the Increased Recovery Project (IRP) that was commissioned in 2013. Copper metal recovery to copper concentrate is estimated by a formula that uses copper grade and the sulphur to copper ratio in the ore with an upper limit of 90% copper recovery. The formula was developed from metallurgical test work and historical plant performance data. Operating performance following the commissioning of the IRP has validated the metallurgical recovery formula. Gold metal recovery to copper concentrate is assumed to be 48% which represents the historic average recovery. Further plant performance data is required to determine the impact of the IRP circuit on gold recovery. The Phu Kham ore contains arsenic as the major deleterious element. Ore and concentrate blending has successfully controlled the arsenic grade in copper concentrate below contractual penalty levels. Hence no arsenic penalty was assumed for the Ore Reserve estimate. The processing plant was upgraded in 2012 and has a nominal capacity of 16 Mtpa on hard ores. Higher throughput rates are being achieved on softer ores and a rate is 18.5 Mtpa is planned for 2014. Approaches to offset the lower throughput rate on hard ore have been identified and include higher plant utilisation, finer blast fragmentation and SAG mill pebble crushing. There is limited data to support the prediction of metal recovery from low grade ore (below 0.2% Cu). The selection of the cut-off grade at 0.2 Cu% is heavily influenced by the lack of confidence in estimates of metal recovery below this grade. Metallurgical

Criteria	Explanation
	test work is being performed to determine the recovery properties of this material.
Environmental	The Phu Kham Operation has been approved by the Government of Laos and has obtained the necessary environmental licences and permits. Tailings from ore processing are permanently stored in an engineered TSF. The TSF is a valley fill behind an earthen dam wall. Tailings are deposited into the valley and stored under water to prevent acid generation. The TSF earth wall is raised progressively in advance of the dam fill rate. Waste mined from the pit is classified into four categories according to the potential acid forming character. Based on these categories, waste rock is disposed in different ways to manage potential environmental impacts and satisfy mine closure requirements. Waste rock with the potential to generate acid rock drainage (ARD) is either encapsulated within benign waste rock or placed sub-aqueously within the TSF. Non-ARD waste rock is placed in conventional dumps. The Phu Kham Operation has a positive water balance. Water management practices are applied to use water efficiently, maintain appropriate water holdings and discharge water in a controlled manner throughout the year. Plans are in place to reclaim and progressively rehabilitate land to a standard which aims to minimise environmental impact and maximise land use during and after mining and ore processing.
Infrastructure	All necessary supporting infrastructure is in place to support the Phu Kham Operation. No significant additional infrastructure is required to enable mining of the Ore Reserve.
Costs	Phu Kham is an operating mine with no major planned expansionary capital expenditure. No allowance is included for any expansion capital expenditure and the Ore Reserve estimate does not depend on any such projects. Sustaining capital will be required on a regular basis for the replacement of mining fleet, minor infrastructure works and maintenance of the ore processing plant. A sustaining capital allowance of \$0.125/t mined was included in the pit optimisation. Operating costs have been estimated through a combination of historical actual performance and planned expenditure at a long term ore processing rate of 18 Mtpa. An average LOM mining operating unit cost of \$3.00/t mined was calculated for open pit optimisation. An average LOM ore processing operating cost of \$6.45/t processed was applied on the basis of actual and planned performance. The progressive construction of the TSF dam wall was estimated at \$0.68/t of ore processed over the remaining ore tonnage. General and Administration costs were estimated at \$1.41/t processed based on actual and planned performance. Off site administration costs were not included. Selling costs including deleterious element penalties and smelting and refining charges were derived from existing sales contracts. Concentrate transport charges were derived from calculated costs and the intended distribution of concentrate volumes through ports in Thailand and Vietnam. A net smelter return (NSR) royalty of 6% of net revenue after transport, smelting,

Criteria	Explanation
	refining and selling costs is paid to the Government of Laos.
Revenue factors	Revenue calculations were based on the diluted Ore Reserve block model grades, long term metal prices for copper and gold, and contractual terms for treatment charges and metal payability. These values were incorporated into the NSR calculation used to determine the economically mineable portion of the deposit. Copper ($3.20/lb) and gold ($1,300/oz) prices based on PanAust's assessment of long term market conditions were used to prepare the Ore Reserve estimate. These price assumptions are below the three year trailing average of $3.66/lb Cu and $1,550/oz Au. Silver provides a negligible contribution to revenue and was excluded from the revenue calculation. The Ore Reserve is relatively insensitive to copper metal price with no change to the estimated tonnage and grade at metal prices down to $2.75/lb Cu and $1,300/oz Au. Smelting and refining charges assumed in the Ore Reserve estimate were derived from existing contracts and PanAust's assessment of future copper concentrate sale terms.
Market assessment	Copper concentrate is widely traded in international markets. PanAust has long term sales contracts in place for Phu Kham's copper concentrate. There is strong demand for the Phu Kham copper concentrate in the Asian region and PanAust expects this demand to remain for the life of the mine.
Economic	All costs and revenues were prepared on a US dollar basis with no exchange rate assumption and no escalation. PanAust maintains a commercial model that is used to estimate the value of the Phu Kham Operation. The model shows a net present value that exceeds that carrying value of the asset.
Social	The company maintains a social license to operate through sustainable development programs and government and community consultation forums. There are no known social issues that threaten the license to operate.
Other	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited (PBM). The Government of the Lao PDR holds a 10 % interest in PBM. PBM's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 km^2, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.
Classification	All critical assumptions applied to mining, ore processing, waste disposal, cost and revenue are generally considered to be at a level of confidence appropriate for a Proved Ore Reserve estimate. The confidence classification is therefore predominately dependent on the category of the Mineral Resource estimate. The Proved Ore Reserve estimate is classified as the economically mineable part of the Measured Mineral Resource except for the portion of Measured Mineral Resource

<table>
<tr><th>Criteria</th><th colspan="3">Explanation</th></tr>
<tr><td></td><td colspan="3">contained within the final southern cutback and ore at depth below 280 mRL in the north of the deposit.

The Probable Ore Reserve estimate is the economically mineable part of the Indicated Mineral Resource plus:

• 2 Mt at 0.35% Cu, 0.18 g/t Au and 3.3 g/t Ag of Measured Mineral Resource contained within the final south cutback that has been modified to Probable Ore Reserve due to infrastructure risks associated with the mining of this cutback, and
• 6 Mt at 0.50% Cu, 0.23 g/t Au and 1.2 g/t Ag of Measured Mineral Resource in the north of the deposit below 280mRL that has been modified to Probable Ore Reserve due to mining access and economic risks associated with mining this material.

Mining of the final southern cutback will increase the risk of instability of the ore crushing and conveying infrastructure. The risk of compromising this infrastructure combined with the moderate grade of the ore in this cutback creates uncertainty with respect to the eventual economic extraction of this ore and warrants reporting at Probable Ore Reserve.

Ore located below 280 mRL in the north end of the deposit requires movement of the Nam San creek diversion or another method of managing the creek water. The depth of the ore and high waste to ore strip ratio of the required pit cutback needs a sufficiently high metal price at the time of mining to justify extraction late in mine life. In view of the uncertainty over the creek diversion and economic value, all of the Ore Reserve in the northern pit area below 280 mRL has been classified as Probable.

These classifications appropriately reflect the Competent Person’s view of the deposit.</td></tr>
<tr><td>Audits or reviews</td><td colspan="3">The Ore Reserve estimate has been prepared internally by PanAust and draws upon the work of mining engineering consultants AMDAD Pty Ltd for pit optimisation and design. No external audits have been performed.

There is no information that contradicts any of the assumptions or models used in the preparation of the estimate or indicate any significant errors in the estimation process.</td></tr>
<tr><td rowspan="5">Discussion of relative accuracy / confidence.

Rated from 1-5 with 1 being the highest level of accuracy / confidence.</td><td>Criteria</td><td>Risk Rating</td><td>Comment and Controls</td></tr>
<tr><td>Mineral Resource</td><td>2</td><td>The 2012 Mineral Resource model provides an acceptable reconciliation against the principal payable metals with a slight overestimation of mineralised quantities.</td></tr>
<tr><td>Project Status</td><td>1</td><td>The Phu Kham Operations has been operating for more than seven years.</td></tr>
<tr><td>Cut-off Parameters</td><td>2</td><td>The cut-off grade equates to a copper price that is significantly below the value assumed for this Ore Reserve estimate. Pit optimisation and cashflow modelling have demonstrated that the project is robust over a reasonable range of metal prices.</td></tr>
<tr><td>Mining Factors</td><td>3</td><td>The risk of dilution and ore loss is considered to be low due to the broad and well defined ore zones.

Geotechnical instability on the west wall of interim</td></tr>
</table>

Criteria	Explanation		
			pit stages presents a moderate risk to the continued mining of ore in the lower benches at the north end of the pit. Mining of a portion of the Measured Mineral Resource in the deep north and final south cutback has moderate risk and has therefore been classified as Probable Ore Reserve.
	Metallurgy Factors	2	The ore processing plant has operated for several years and has generally outperformed the design criteria. Initial metallurgical recovery results have exceeded those predicted by the recovery model used for the Ore Reserve estimate however further plant data from a greater variety of ores is required. Measures to offset the reduced milling rate for hard ores have been identified but not demonstrated at a plant scale. Uncertainty associated with metallurgical recovery below 0.2% Cu has been removed by applying a minimum copper grade criteria to the selection of mineralisation reported in the Ore Reserve.
	Environmental	2	Tailings, ARD waste rock and water management are considered to be of a high standard. There remains a residual risk with the environment impact for closure of the open pit and post mine management of the TSF dam.
	Infrastructure	1	The infrastructure required to operate the mine is well established.
	Cost Estimates	2	Cost estimates are considered reliable based on historical performance. Some uncertainty exists with the estimation of future costs however this risk is considered to be consistent with industry standards.
	Revenue Factors	3	The forecasting of long term copper and gold prices represents a residual risk.
	Market assessment	1	The copper concentrate market was assessed as being low risk. Movements in the copper concentrate supply and demand balance are likely to affect price rather than volume. Arsenic and other impurities have been successfully blended below contract limits for several years and any future issues are expected to be of a short duration.
	Economic	2	The Phu Kham Operation does not require further major expansionary capital. The operation remains cashflow positive at copper

Criteria	Explanation		
			prices significantly lower than those assumed for this Ore Reserve estimate.
	Social	1	No known issues.
	Classification	1	Modifying factors have been applied where there is a risk.

KTL Copper-Gold

Section 4. Estimation and Reporting of Ore Reserves

Criteria	Explanation
Mineral Resource estimate for conversion to Ore Reserves	The KTL Ore Reserve estimate is based on the 31 December 2013 KTL Mineral Resource estimate. Dan Brost is the Competent Person responsible for the 2013 KTL Mineral Resource estimate. The KTL deposit is located near the town of Phonsavan, the capital of Xieng Khouang Province in Laos, and is the only deposit considered for this Ore Reserve estimate. PanAust geology personnel familiar with the KTL deposit were responsible for providing guidance to the geological interpretation and domain wireframe generation used in the creation of the model. The Mineral Resource model is an Ordinary Kriged model having a parent block size is 20 m by 20 m by 10 m (East x North x RL) with regular sub-blocks of 5 m by 5 m by 5 m at domain boundaries. Densities are assigned by domain using the mean value in each respective lithology/oxide combination. The KTL Mineral Resource estimate is reported inclusive of the KTL Ore Reserve estimate.
Site visits	PanAust staff involved in the preparation of the KTL Ore Reserve estimate have made several visits to the project site and the Phu Kham Operation. The Competent Person for the Ore Reserve estimate has visited both the KTL site and the Phu Kham Operation.
Study / Project Status	This is the initial Ore Reserve estimate for the KTL deposit. The Ore Reserve estimate is based on designs and cost estimates reported in the Phonsavan Copper-Gold Project Prefeasibility Study that determined the viability of mining the KTL deposit as a satellite open-pit to the Phu Kham Operation. Ore will be trucked along public roads over a distance of 110 km for processing at the Phu Kham copper–gold concentrator which has operated since March 2008. Prefeasibility Study costs were derived from a mixture of vendor estimates and historical data from PanAust's other operating mines in Laos and are considered to be at ±25% level of accuracy. No mining excavation or production has occurred at the KTL project site.
Cut-off parameters	The economic cut-off for ore selection was the non-mining breakeven value. This value is calculated for each block in the Ore Reserve model and incorporates mining dilution and ore loss, road haulage to Phu Kham, processing recoveries, ore processing cost, TSF construction cost, KTL site administration cost, concentrate transport cost, selling cost and metal prices. Revenue from copper and gold is considered in the value calculation. Only blocks having a positive value are reported in the Ore Reserve estimate. Mining costs are not considered in the cut-off criteria for ore selection. The impact of mining costs is accounted for in the determination of the optimal pit shell.
Mining factors or assumptions	The KTL open pit will be mined as a conventional truck and shovel operation with 5 m high benches in ore. The majority of the ore and waste material will be drilled and blasted before being excavated by hydraulic shovels and excavators. Similar mining practices are well established at the Phu Kham and Ban Houayxai operations, both of which are owner operated by PanAust's 90% owned subsidiary company. Waste pre-stripping of 7 Mt is required to establish mining areas suitable for the planned mining fleet. Routine grade control drilling will be used to delineate the ore boundaries of the deposit. The mining method is considered appropriate to the surface topography and

Criteria	Explanation
	the deposit's geometry, grade and geological continuity. Slope design recommendations for the final pit walls have been provided by geotechnical consultants PSM. The slope design is based on drill hole information, mapping and hydrogeological data. The recommended slope designs were applied to the open pit optimisation and design used for the Ore Reserve estimate. The open pit slope design is based on recommendations for a larger open pit configuration and, while appropriate for Prefeasibility Study evaluation, may require modification for final design prior to mining. The Mineral Resource model is converted to an Ore Reserve model as part of the Ore Reserve estimation process. Dilution is included in the Ore Reserve estimate through two mechanisms. • The Mineral Resource model grade is estimated as an average value inside blocks with dimensions of 5 m by 5 m in plan and 5 m vertically. • External (edge) dilution is quantified in the Ore Reserve model for each block. A dilution skin of 0.5 m from each neighbouring block to the east, west, north and south, at the grades of the adjacent blocks, is applied to calculate a diluted grade for each of the Ore Reserve model blocks. The process preserves the total mass of material with each block gaining and losing the same volume of material. This skin dilution process makes allowance for dilution across the boundaries between ore and waste zones. The Ore Reserve is further modified to account for an assumed ore loss of 5% to represent ore grade material that cannot be extracted within a practical mining shape. Optimisation of the pit shape was completed using the Lerchs Grossman algorithm as implemented in the Whittle 4X software. Optimisation considered a range of metal price scenarios: • Copper price from $3.00/lb to $4.50/lb • Gold price from $1,200/oz to $1,400/oz. The open pit shell used for design was selected from the pit optimisation analysis at metal prices of $3.20/lb Cu and $1,300/oz Au. The Ore Reserve was estimated within a preliminary open pit design prepared by mining engineering consultants AMDAD Pty Ltd in late 2013. The design includes ramps and safety berms on the open pit walls. 61 Mt of waste rock (inclusive of pre-stripped material) will be stored in an engineered emplacement adjacent to the open pit. The Inferred Mineral Resource is not considered for conversion to Ore Reserve. There are no blocks classified as Inferred Resource within the open pit design. A life of mine production schedule was prepared from the open pit design used for the Ore Reserve estimate. The production schedule showed that ore can be delivered to the Phu Kham concentrator in sufficient quantity in each year of the mine life to satisfy the assumptions associated with the costs used in the Ore Reserve estimate. Infrastructure will be required at Phonsavan to support open pit mining of the KTL deposit and will include a site access road, primary and secondary crushers, maintenance workshop, vehicle refuelling facility and waste dumps.
Metallurgical factors or assumptions	The KTL deposit will deliver ore feed to the Phu Kham concentrator. The Phu Kham flowsheet uses a conventional crush, grind and flotation process to recover a saleable copper concentrate that contains copper, gold, and silver.

Criteria	Explanation
	A variability metallurgical test work program was conducted using batch kinetic flotation tests across grade, weathering, lithology and alteration based sample sets from the KTL deposit. These sample sets are considered spatially representative of the KTL deposit. Batch kinetic tests were conducted under Phu Kham reagent and grinding conditions. The copper concentrate obtained from the KTL flotation test work typically graded 23% Cu and contained no significant deleterious elements. Copper recoveries were very consistent while gold recovery demonstrated greater variability. The metallurgical test work demonstrated similar mineralogy to the Phu Kham deposit which will enable KTL ore to be directly blended with Phu Kham ore and treated through the Phu Kham concentrator. KTL samples are classified as moderately hard to hard for grinding based on preliminary tests. Metallurgical domains were developed for primary and transitional mineralisation. Metallurgical recovery estimates for these metallurgical domains are based on results from ongoing batch flotation test work. Metallurgical recovery values (85% Cu, 65% Au and 45% Ag) were obtained for primary mineralisation with lower recoveries for transitional mineralisation. A recovery formula was derived from the sulphur to copper ratio for transitional mineralisation and capped at 85% Cu recovery. Average gold recovery for transitional ore was 60%. Transitional ore accounts for approximately 13% of planned concentrator feed. No oxide material is planned to be treated. No bulk sample or pilot scale test work has been performed given the similarity with Phu Kham ores.
Environmental	A draft Environmental and Social Impact Assessment (ESIA) has been prepared for the Phonsavan Copper-Gold Project. The ESIA report forms part of the Impact Assessment documents for the project and has been prepared based on the requirements of the Government of Laos Environmental Impact Assessment Guidelines (2012). It is expected that the ESIA will be submitted in 2014. The ESIA identifies the likely types of impacts and assesses the likelihoods and magnitudes of the potential impact of the proposed project development. Consistent with the Decree on Environmental Impact Assessment, the ESIA includes consideration of both environmental and social aspects and impacts. Standalone social and health assessment reports are also provided as part of the appendices to the ESIA. A framework for further community and Government consultation is also provided. Earth Systems is the consultant for the ESIA and has demonstrable skills and experience in mining ESIA preparation including extensive experience in Laos. Earth Systems has also engaged additional specialists and sub-consultants from Laos and other international specialists. Resettlement or relocation of any settlement areas will be required for the project. The project has been designed to minimise physical environmental impacts and associated biological and social impacts. The main components with physical impacts are the open pit mine and waste rock dump. The waste rock dump is located and designed to minimise environmental risk and social impact whilst preserving project economics. Waste rock mined from the open pit is classified into multiple categories according to the potential acid forming characteristics. Waste rock from the different categories is disposed in different ways to meet mine closure requirements. Ore processing will occur at the Phu Kham Operation which has been approved by the Government of Laos and has obtained the necessary environmental licences and

Criteria	Explanation
	permits. Tailings from ore processing will be permanently stored in the existing engineered TSF at the Phu Kham site. The TSF is formed by a valley fill behind an earthen dam wall. Tailings are deposited into the valley and stored under water to prevent sulphide oxidation. The TSF earth wall is raised progressively in advance of the dam fill rate. Plans are in place to reclaim and progressively rehabilitate land to a standard which aims to minimise environmental impact and maximise land use during and after mining and ore processing.
Infrastructure	The KTL deposit is accessible by air and road access via the Phonsavan township which has a population of 57,000. Phonsavan is serviced by sealed roads to the Lao capital city of Vientiane through to the southern parts of Laos and Thailand. The distance to Vientiane is 430 km via Route 7 west and then Route 13 south. In addition, sealed roads run from Phonsavan to Vietnam via Route 7 east with the distance to the Vietnam border being 120 km and the ports of Cua Lo, Hong La and Vung Ang a further 370 km. The most significant infrastructure requirement is the ore haulage route between Phonsavan and the Phu Kham Operation. A significant portion of this route relies upon the construction of a new road by the Government of Laos. Construction work has commenced as part of a separate initiative to improve transport links between Vientiane and China. The completion of this road is essential for the project and, based on current knowledge, is expected to be completed within the timeframes contemplated by the Prefeasibility Study. The Ore Reserve estimate is contingent upon the continued operation of the Phu Kham processing plant with all fixed costs of the processing plant attributed to the Phu Kham operation. The Thai port of Sriracha, through which some copper concentrates will be shipped, lies some 950 km south of the Phu Kham Operation. Copper concentrate is also transported to the Vietnamese port of Vung Ang which is located approximately 380 km east-southeast of the Phu Kham Operation. These copper concentrate transport routes are currently used by the Phu Kham operation and will accommodate the additional concentrate production from mining of the KTL deposit. Other infrastructure requirements include an on-site crushing plant, mobile mining equipment maintenance facility, road haulage fleet maintenance facility and administrative buildings. No permanent camp is required as accommodation is available in the Phonsavan township. On-site supplemental water requirements are minimal and will be sourced from local streams. The majority of the workforce will be recruited and trained locally. The project site is located with the Company's existing Contract Area which enables land access to develop the KTL deposit.
Costs	Operating costs have been estimated by both external consultants and PanAust. The proposed organisational structure was used to derive total employment costs for each employee. Mining operating costs were established from first principles using mobile equipment hours calculated from haulage profiles and mining productivity data for excavation rates. Haul cycle data for the trucking fleet was based on the haul road layout provided by the mine design consultants. Haul cycles were generated from a simulation of total

Criteria	Explanation
	truck cycle times for each mining bench. Conventional drilling and blasting has been costed for both 5 m and 10 m bench heights commensurate with the planned excavation methods. In addition, the drill and blast costing allows for various rock types, explosive types, explosive weights and drill productivities. The balance of the mining operating cost is based on the support equipment required to match the primary load and haul fleet operation. The average mining unit cost was calculated to be $2.63/t mined. The road haulage operating costs were generated from first principles based on fuel burn rates, tyre wear assumptions and estimated maintenance parts consumption. An extensive fuel burn trial was conducted along the current concentrate haulage route to establish both loaded and unloaded conditions which provided a high level of confidence to the estimate given the similar distance and topographic profile. Tyre life and maintenance parts were established from the Company's maintenance database utilising data from many years of concentrate haulage operations. All rates were then benchmarked against recent bids by multiple tenderers for concentrate haulage operations in Laos. Processing at Phu Kham is primary crusher constrained so crushed KTL ore will be delivered directly to the coarse ore stockpile as supplemental feed to existing ore processing capacity. Ore processing costs were therefore assumed to be incremental to the existing Phu Kham operating cost which required the exclusion of primary crushing costs and fixed costs from the calculated unit cost of processing KTL ore. The cost of expanding the Phu Kham TSF was modelled as a unit cost of ore processed for the purpose of the NSR calculation and is based upon the 2013 Phu Kham TSF life of mine plan. An average ore processing cost of $26.40/t processed was calculated inclusive of road haulage costs. General and Administrative expenses attributable to the mining of the KTL deposit were calculated to be $2.90/t processed. The mobile mining fleet and the road haulage trucking fleet capital costs were sourced from single quotations from existing suppliers to the Company's other operations in Laos. Sustaining capital is not required for the replacement of mining fleet or ore haulage fleet due to the relatively short mine life. Ore crushing station equipment costs were sourced from vendor proposals. Facility construction costs were sourced from quotations for the supply of packages relating to fuel pumps, tanks, office facilities and roof structures. Concrete pads and earthworks were estimated from rates established for the Company's other recent projects in Laos and escalated where required. Road construction and upgrade costs were sourced from engineering designs for the proposed route coupled with cost rates sourced from recent road upgrade contracts provided by multiple contractors on a dry season construction basis. Mobile equipment finance lease expenses were costed into the financial model at commercial rates. Selling costs and smelting and refining charges are derived from existing sales contracts. Concentrate transport charges were derived from calculated costs and the planned distribution of concentrate volumes through ports in Thailand and Vietnam. A net smelter return (NSR) royalty of 6% of net revenue after transport, smelting,

Criteria	Explanation
	refining and selling costs is paid to the Government of Laos.
Revenue factors	Revenue calculations were based on Ore Reserve block model grades, long term metal price assumptions for copper and gold and contractual terms for treatment charges and metal payability. These values were incorporated into the NSR calculation used to determine the economically mineable portion of the deposit. Long term copper ($3.20/lb) and gold ($1,300/oz) prices were used to prepare the Ore Reserve estimate. These price assumptions are below the three year trailing average of $3.66/lb Cu and $1,550/oz Au. Smelting and refining charges assumed in the definition of Ore Reserve are based on existing contracts and PanAust's assessment of future copper concentrate sale terms.
Market assessment	KTL ore will be treated with Phu Kham ore to produce a blended concentrate. PanAust has long term sales contracts in place for Phu Kham's copper concentrate. There is strong demand for the Phu Kham copper concentrate in the Asian region and PanAust expects this demand to remain for the life of the project to accommodate the sale of additional concentrate from the processing of KTL ore.
Economic	All costs and revenues were prepared on a US dollar basis with no exchange rate assumption and no escalation. A detailed cash flow model was created and modelled in real terms (i.e. no price or cost escalation was applied). A discount rate of 8% was used to determine the Net Present Value (NPV) of the project cash flows. The project provides a positive NPV at the assumed price for Ore Reserve estimation. The project remains NPV positive at a $3.00/lb copper price. Sensitivities of ±5% were performed for copper price, copper recovery, copper grade, operating cost, gold price and project development capital. The project financial outcomes are most sensitive to copper price and metallurgical recovery, followed by copper grade and operating costs.
Social	The company maintains a social license to operate through sustainable development programs and government and community consultation forums. There are no known social issues that threaten the license to operate. Several local social issues were identified and addressed during the ESIA process. • Socio-economic benefits associated with employment, workforce training, revenue generation, improvements to local services and infrastructure (roads, health, education etc.). • Impacts associated with loss of productive land and/or loss of access to productive land, and the need for the Company to implement appropriate livelihood restoration. • Protection of the water intakes for the Phonsavan town water supply. The Company has already supported the improvement of water treatment infrastructure for Phonsavan. Implementation of proposed management and monitoring measures is expected to protect the Phonsavan water supply. • Potential impacts, both positive and negative, associated with population in-migration. • Potential noise and air quality impacts on villages located close to project activities. • Community health and safety impacts associated primarily with the high

Criteria	Explanation
	frequency and volume of road transport.
Other	PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited (PBM). The Government of the Lao PDR holds a 10 % interest in PBM. PBM's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 km^2, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations. The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time. The project will be subject to a permitting process including an ESIA and a Feasibility Study in order to obtain approvals to commence operation. The company is experienced in permitting mining operations in Laos. The project will be reassessed based on the conditions imposed by these approvals. No material changes to the project are anticipated as a result of any foreseeable permit conditions. The project also requires completion of a new section of road between Phonsavan and the Phu Kham Operation. The Government of Laos has commenced construction of this road section however its suitability for ore haulage has yet to be determined. Current progress provides reasonable grounds for construction to be completed in a timeframe and to a standard that will support the development of the KTL deposit.
Classification	All critical assumptions applied to mining, ore processing, waste disposal, cost and revenue are considered to be at a level of confidence appropriate for a Prefeasibility Study and an Ore Reserve estimate. The confidence classification is therefore based on the category of the Mineral Resource estimate with Proved Ore Reserves derived from the Measured Mineral Resource and Probable Ore Reserves converted from the Indicated Mineral Resource. The classification methods used are considered by the Competent Person to be appropriate for the style and nature of the deposit.
Audits or reviews	The Ore Reserve estimate has been prepared internally by PanAust and is based upon open pit optimisation and design work performed by mining engineering consultants AMDAD Pty Ltd. No independent external audits have been performed. There is no information that contradicts any of the assumptions or models used in the preparation of the estimate or indicate any significant errors in the estimation process.

Discussion of relative accuracy / confidence. Rated from 1-5 with 1 being the highest level of accuracy / confidence.	Criteria	Risk Rating	Comment and Controls
	Mineral Resource	3	The Mineral Resource estimate risk assessment highlights data density and distribution as a medium risk and tonnage factors as medium level risks. The current drilling spacing shows minimal copper grade variability with distance however there are localised areas of rapid copper grade changes.

Criteria	Explanation		
			It is recommended that more density measurements in different lithologies be collected. No mining or ore processing reconciliation data exists as the KTL deposit has not been developed.
	Project Status	3	The project will be permitted upon acceptance of the ESIA by the Government of Laos and the subsequent issuance of a Permit to Mine. The Ore Reserve is contingent upon the continued operation of the Phu Kham processing plant with all fixed costs of the processing plant attributed to the Phu Kham operation. There is a low risk that the project will not be permitted in time to allow the complete Ore Reserve to be extracted over the remaining Phu Kham open pit mine life. The Ore Reserve will be re-assessed over time with consideration of the permitting timeframe and remaining Phu Kham mine life.
	Cut-off Parameters	3	Pit optimisation and cash flow modelling have demonstrated that the project is post-tax NPV positive above a $3.00/lb copper price. The project may not be viable at copper prices substantially below $3.00/lb. The distribution of mineralisation grades and pit optimisation analysis shows that the Ore Reserve is sensitive to mined grade.
	Mining Factors	3	The risk of dilution and ore loss is considered to be moderate due to the distinct transition between the ore and waste boundaries. The applied dilution and ore loss factors are considered appropriate for the Prefeasibility Study but require validation through mining. The geotechnical assessment was based on a substantially larger pit. There is a risk that the slope angles

Criteria	Explanation		
			may not be appropriate for the smaller pit dimensions used to develop the Ore Reserve estimate.
	Metallurgy Factors	2	Copper metallurgical recoveries exhibited consistent results across all test work conducted. Gold recoveries demonstrated higher variability and there is a medium risk that actual gold recovery may be 10% lower than estimated. Gold accounts for approximately 22% of revenue at Ore Reserve commodity prices. Metallurgical test work is on-going and the project will be revised as further results become available. The Phu Kham ore processing plant has operated for several years and is suitable for the processing of ores from the KTL deposit.
	Environmental	2	Tailings, ARD waste rock management and water management are consistent with operating practices at other PBM operations in Laos and are considered to be of an appropriate standard.
	Infrastructure	2	The processing plant infrastructure required to operate the mine is well established. However, a portion of the road between Phu Kham and Phonsavan is not yet constructed. There is a risk that this portion may not be constructed in the timeframe and/or to the required standard as the construction is largely contingent on third parties. PBM continues to monitor the progress of the road construction and has reasonable confidence that the road will be completed within the required timeframe to an acceptable standard.
	Cost Estimates	3	Cost estimates are considered reliable based on the historical performance at Phu Kham for processing rates, mining costs and G&A costs. However, a medium risk exists for the ore haulage cost between Phonsavan and Phu Kham as the final road is not yet constructed. This risk will be

Criteria	Explanation		
			mitigated following completion of the road by third parties and will involve a fuel burn trial at the appropriate time.
	Revenue Factors	3	The forecasting of copper and gold prices represents a residual risk.
	Market assessment	1	The copper concentrate market was assessed as being low risk. Movements in the copper concentrate supply and demand balance are likely to affect price rather than volume.
	Economic	3	The project may not be viable at copper prices substantially below $3.00/lb or if actual operating costs significantly exceed those developed for the Prefeasibility Study.
	Social	2	Several social impacts were identified during the ESIA process along with appropriate control measures. Sediment control is a sensitive issue due to potential impact of uncontrolled releases on neighbouring farming activities and the town water supply.
	Classification	1	No known issues.

Ban Houayxai Gold-Silver

Section 4. Estimation and Reporting of Ore Reserves

Criteria	Discussion
Mineral Resource estimate for conversion to Ore Reserves	The Ore Reserve Estimate is based on the 31 December 2012 Mineral Resource estimate. Dan Brost is the Competent Person responsible for the Mineral Resource estimate. The Mineral Resource estimate is based on a geologically domained model prepared by H&SC (H&S Consultants Pty Ltd). PanAust geology staff familiar with the deposit were responsible for providing guidance to the geological interpretation and domain wireframe generation used in the creation of the model. The Mineral Resource estimate has changed from the previous estimate as a result of depletion due to mining in 2013. The Mineral Resource model is a Multiple Indicator Kriged (MIK) model with gold as the indicator metal. Silver grades are interpolated using a separate MIK run with the results then averaged so the silver grades are presented as a single block average grade known as an "e-type" grade estimate. The block model has parent block size of 25 m by 12.5 m by 10 m (East x North x RL) with regular sub-blocks of 12.5 m x 6.25 m x 5 m at domain boundaries. The MIK adjustment for support is based on a mining selectivity (SMU) of 5 m x 2.5 m x 2.5 m (East x North x RL). Drill hole data is constrained within each domain for estimation. Densities are assigned by domain using the mean value in the data each respective lithology/oxide combination. The lowest gold grade threshold for an indicator bin is 0.2 g/t Au. The Mineral Resource estimate is reported to the volume between the surface and 310 mRL. The pit design that supports the Ore Reserve estimate has its lowest bench at 440 mRL. The Mineral Resource estimate is reported inclusive of the Ore Reserve estimate.
Site visits	PanAust staff responsible for the preparation of the Ore Reserve estimate have made several visits to the Ban Houayxai Operation. The Competent Person has visited the Ban Houayxai Operation.
Study / Project Status	The Ban Houayxai pit is an active mining operation. Mining commenced at the beginning of 2012 and ore processing commenced in second quarter 2012. Most assumptions relevant to the Ore Reserve estimation have been tested by practical experience in 18 months of operation. The expected average processing rate of 4.5 Mtpa has been achieved on a combination of oxide and transitional ore. No significant primary ore has been treated as yet. Processing rates are predicted to vary with ore type. The Ore Reserve estimate is supported by an open pit design, detailed budget estimate, life of mine plan and operational history that exceeds the level of detail and precision found in a Feasibility Study.

Criteria	Discussion
Cut-off parameters	Gold grade is used as the cut-off grade criteria since gold metal dominates the economic value of the deposit and the structure of the MIK model allows only the indicator grade to be used for selection. The cut-off grade varies with the oxidation state of the ore since mining and ore processing costs and metallurgical recoveries vary with oxidation state (material type). • Oxide ore: Gold greater or equal to 0.32 g/t • Transitional ore: Gold greater or equal to 0.40 g/t • Primary ore: Gold greater or equal to 0.44 g/t These marginal cut-off grades were developed at a gold price of $1,300/oz. The silver grade assigned to the Mineral Resource is a block average grade. Given that silver metal should follow gold metal the application of a block average silver grade will be an underestimation of the actual grade in the Mineral Resource and hence in the Ore Reserve. There has been no attempt to adjust the silver grade for this effect in the Ore Reserve since its impact on economic value will be modest. The open pit optimisation process used a different ore selection methodology to that applied in the final Ore Reserve estimate. Pit optimisation calculates a net value per tonne as its selection criteria using both gold and silver revenue contributions. The silver grade used in this net value calculation is the block average silver grade. The gold grade is the grade in each MIK grade bin. The pit optimisation considers positive net value parcels as “mill feed” and negative net value parcels are “waste”. Mill feed is defined on a marginal economic basis for pit optimisation, with the selection method considering whether material hauled to the pit exit will realise more value as mill feed than as waste. The mill feed calculated by the pit optimisation process will not be the same as the Ore Reserve due to the different estimation methodologies . Furthermore the Ore Reserve estimate is derived from gold revenue only which provides a more conservative cut-off grade value. The operating mill feed cut-off grade used in 2013 operations was higher than the December 2012 Ore Reserve cut-off grade. This approach was applied to maximise mill feed grade during a period where mined ore quantities exceeded the ore processing rate. The low grade untreated ore is sent to stockpile for treatment at a later time. In 2013, an additional 0.9 Mt at 0.32 g/t Au and 3.8 g/t Ag was added to stockpile.

Criteria	Discussion
Mining factors or assumptions	Selective open pit mining techniques are applied at the Ban Houayxai Operation. Ore and waste are drilled and blasted on 5 m high benches before mining in two flitches. Blasting on 10 m benches has recently been trialled with mining on four flitches. Loading and haulage is achieved by a conventional truck and backhoe excavator fleet with two Komatsu PC1250 excavators, a Caterpillar 990 front end loader and seven Caterpillar 777 haul trucks. Grade control drilling and detailed geology modelling is carried out before mining to identify and define ore blocks. Movement during blasting is measured and the location of ore blocks is adjusted for blast movement prior to excavation. Ore mining is carried out under geological control. Dilution in the broad oxide and transitional ore zones is low (demonstrated by reconciliation data) with only minor quantities of primary ore being mined to date. The harder primary ore requires higher blast powder factors and more dilution is expected due to blast movement. The primary ore zones are narrower than the oxide and transitional ore zones and greater dilution and ore loss is assumed for primary ore mining. Dilution and ore loss is included in the Ore Reserve estimate through two mechanisms. • Mineral Resource modelling. The MIK model estimates a grade that includes an allowance for internal dilution and ore loss. • External dilution and ore loss factors are applied to account for the losses at the edge of mining blocks following blasting and losses from mining practices. These factors are a simple percentage change in grade and tonnes applied across all ore blocks on the basis of oxidation state (material type). For the oxide material type, no additional dilution factor was applied and 1% ore loss factor was applied. Dilution of 3% and ore loss of 2% were applied to the transitional material type. In the primary material type, dilution was assumed to be 10% and ore loss 5%. Dilution was applied prior to the ore loss adjustment. The grade of the diluting waste is assumed to be 0g/t for both gold and silver. The use of 0 g/t grade for the dilution is considered to be conservative for the Ban Houayxai deposit as there is likely to be some gold grade in the diluting material. With no experience in mining primary ore, and a plan to move to 10 m bench height, a dilution factor of 10% at zero grade is considered to be appropriate. These factors will require revision when sufficient reconciliation data is available from primary ore mining. The Inferred Mineral Resource was not considered for conversion to Ore Reserve. Final pit slope design parameters were recommended by specialist geotechnical consultants, PSM. The design is based on drill hole information. There is limited exposure of interim and final pit walls from mining. Pit wall mapping has been performed and new geotechnical drilling data will be used to prepare an updated set of slope design criteria. The Ore Reserve is estimated within an open pit design that includes ramps and safety berms on the pit walls. Optimisation of the pit shape was completed in early 2013 using the Lerchs Grossman algorithm as coded to the Whittle 4X software. Optimisation considered a range of metal price scenarios: • Gold price from $1,100/oz to $1,600/oz Silver price of $28/oz.

Criteria	Discussion
Mining factors or assumptions **(Continued)**	The final pit shell was selected from the optimisation analysis and used as the basis for detailed open pit design. The Ore Reserve was estimated from the design open pit using the surveyed topographical surface as at 31 December 2013 to account for mining depletion. A life of mine production schedule was generated in 2013 using the same ore loss and dilution factors as those used in the 31 December 2013 Ore Reserve estimate. The production schedule showed that ore can be presented to the processing plant in sufficient quantity in each year of the mine life to satisfy the assumptions regards costs used in the 31 December 2013 Ore Reserve estimate. Significant stocks of low grade ore are scheduled to accumulate during mining for treatment late in the mine life. Mining of 69 Mt of waste rock is required to extract the material reported in the Ore Reserve estimate. Infrastructure exists at the Ban Houayxai site to support the open pit mining operation and includes maintenance workshops, refuelling facilities, waste dumps and ore stockpiles.
Metallurgical factors or assumptions	The Ban Houayxai gold ore treatment plant uses conventional crush, grind, gravity separation and carbon in leach (CIL) processes to recover gold and silver to a doré product that is exported for refining into fine gold and silver. Processing recovery estimates are based on metallurgical test work and plant experience since 2012 for oxide and transitional material types. Limited primary ore has been treated to date. The estimated recoveries for the different ore types are: • Oxide: 93% Au and 55% Ag • Transitional: 83% Au and 60% Ag • Primary: 78% Au and 75% Ag Recoveries assume grind sizes of 75 µm for oxide ore and 150 µm for transitional and primary ore. The processing plant has a capacity of 3.4 Mtpa to 5.4 Mtpa depending on the mix between harder primary ore and softer oxide ore. Processing unit costs per tonne are adjusted by ore type to reflect the change in processing rate.

Criteria	Discussion
Environmental	The Ban Houayxai Operation has been approved by the Government of Laos. An assessment of environmental impact, waste storage, social and permitting issues was completed by Earth Systems Pty Ltd (ES) during the Feasibility Study and has been detailed in an Environmental and Social Impact Assessment report. Based on this work, PBM has implemented an environmental and social management and monitoring plan for the site and there are no factors are likely to have a material impact on the Ore Reserve estimate. PanAust, of which PBM is a subsidiary, is signatory to the International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold (Cyanide Code) and the Ban Houayxai facility has been designed, constructed and operated in accordance with the guidelines contained in the Cyanide Code. Ore processing tailings are retained in a tailing storage facility (TSF). The TSF is a valley fill behind an earth dam wall with tailings being deposited into the valley and stored under water. Waste mined from the pit is placed into a valley fill dump site located to the south-east of the pit. Some waste rock has the potential to be acid-generating under certain conditions and is encapsulated in cells within the dump surrounded by non-acid forming rock. Low grade ore will be stored adjacent to the waste in the same valley until it is reclaimed for treatment late in mine life.
Infrastructure	All necessary supporting infrastructure is in place to support the Ban Houayxai Operation. No significant additional infrastructure is required to enable mining of the Ore Reserve.

Criteria	Discussion
Costs	Ban Houayxai is an operating mine with no major planned expansionary capital expenditure. No allowance is included for any expansion capital expenditure and the Ore Reserve estimate does not depend on any such projects. Sustaining capital expenditure for the replacement of mining fleet, minor infrastructure works and maintenance of the ore processing plant has been included in pit optimisation at the rate of $0.07/t mined. The remaining mine life requires minimal replacement equipment. All existing mining fleet capital is considered a sunk cost. No consideration of the fleet financing costs (lease arrangements) is included in pit optimisation since this debt does not change the decision to mine a block from within the pit. Likewise no consideration is given to the potential residual value of the mining fleet at end of mine life. The TSF requires construction of a single dam wall that will provide storage sufficient over the mine life. Construction of the existing wall is considered a sunk cost and there is only minor sustaining capital cost requirement for wall raises over the life of the mine. Mine operating costs have been estimated with a combination of first principle calculations, historical performance, budget estimates for the years 2014 to 2016 and life of mine (long term) cost estimates. Mining costs vary with open pit depth and the ore or waste rock classification. Ore has a higher blasting cost, higher grade control charges and higher loading cost that is partly offset by a lower haulage cost compared to waste rock. Ore processing operating costs are based on 2013 costs with allowance for future variation in processing rate as ore hardness changes. Processing fixed costs are distributed over the range of processing throughput rates for the purposes of estimating a total unit cost of processing: • Oxide: $8.53/t • Transitional: $9.81/t • Primary: $10.35/t. General and Administration unit costs for the site are estimated at $3.00/t ore processed. Cost of major consumables (fuel, electrical power, tyres, steel, chemicals) are based on assumptions in the 2013 operations budget that are in turn based on a combination of supplier contracts and market intelligence. Gold-silver doré refining cost, transport and royalty charges are based on the current contract rates and realised costs. A royalty of 6% of revenue is paid to the Government of Laos.
Revenue factors	A long term gold price of $1,300/oz was used to prepare the Ore Reserve estimate relative to the three year trailing average price of $1,550/oz Au. Silver was not included in the cut-off grade calculation for the Ore Reserve estimate.
Market assessment	There is no market limit for sales of gold and silver from Ban Houayxai.

<table>
<tr><th>Criteria</th><th colspan="3">Discussion</th></tr>
<tr><td>Economic</td><td colspan="3">All costs and revenues were prepared on a US dollar basis with no exchange rate assumption and no escalation.

PanAust maintains a commercial cashflow model that is used to calculate the value of the Ban Houayxai Operation. The model shows a positive net present value of the operation within a range of metal price assumptions.</td></tr>
<tr><td>Social</td><td colspan="3">The company maintains a social license to operate through sustainable development programs and government and community consultation forums. There are no known social issues that threaten the license to operate</td></tr>
<tr><td>Other</td><td colspan="3">PanAust owns a 90% interest in the Lao-registered company, Phu Bia Mining Limited (PBM). The Government of the Lao PDR holds a 10 % interest in PBM.

PBM's Mineral Exploration and Production Agreement (MEPA) was signed on 26 January 1994 and has been amended three times: in 1996, 2002, and 2007. The MEPA, as amended, defines a Contract Area of more than 2,600 km^2, regulates exploration, development and mining activities within the Contract Area, and sets out tax and royalty obligations.

The validity and effectiveness of the MEPA is extended for each additional approved mining operation, and the MEPA continues in force until the expiration of the Operating Period for the last Mining Area in the Contract Area. There is potentially no limit to the number of Mining Areas that can be commenced under the MEPA or that can be in operation at any given time.</td></tr>
<tr><td>Classification</td><td colspan="3">All critical assumptions relevant to mining, ore processing, waste disposal, cost and revenue are generally considered at a level of confidence appropriate for Proved Ore Reserve Estimation. The confidence classification is therefore predominately dependent on the category of the Mineral Resource Estimate.

• The Proved Ore Reserve Estimate is the economically mineable part of the Measured Mineral Resource Estimate.
• The Probable Ore Reserve Estimate is the economically mineable part of the Indicated Mineral Resource Estimate.</td></tr>
<tr><td>Audits or reviews</td><td colspan="3">The Ore Reserve estimate has been prepared by PanAust. An external audit of the underlying open pit optimisation process was completed by AMC Consultants Pty Ltd in 2013 to confirm the process used by PanAust. There are no audit findings or reviews that contradict any of the assumptions or models used in the preparation of the estimate or indicate any significant errors in the estimation process.</td></tr>
<tr><td rowspan="4">Discussion of relative accuracy / confidence.

Rated from 1-5 with 1 being the highest level of accuracy / confidence.</td><td>Criteria</td><td>Risk Rating</td><td>Comment and Controls</td></tr>
<tr><td>Mineral Resource</td><td>2</td><td>Industry standard risk for gold deposit Mineral Resource estimation.</td></tr>
<tr><td>Project Status</td><td>1</td><td>Constructed, commissioned and in normal operation.</td></tr>
<tr><td>Cut-off Parameters</td><td>3</td><td>The cut-off grade for the Ore Reserve estimate is sensitive to gold price.

The Ore Reserve cut-off grade is considered to be conservative as it excludes the contribution of silver revenue.</td></tr>
</table>

Criteria	Discussion		
	Mining Factors	2	Internal and external dilution and ore loss have been applied. Production reconciliation is robust. Primary ore mining and increased blast bench height have greater dilution and ore loss risk which has been incorporated through higher assumed dilution of ore loss values.
	Metallurgy Factors	3	Plant performance is better than predicted on oxide and transitional ore. Primary ore recovery and throughput performance has yet to be demonstrated.
	Environmental	2	No material environmental issues have been identified. Cyanide, tailings and waste rock controls and practices are of a high standard.
	Infrastructure	1	All required infrastructure is in place.
	Cost Estimates	2	Mine operating experience provides confidence in the cost estimates.
	Revenue Factors	4	Metal price represent a significant uncertainty given the sensitivity of the Ore Reserve to gold price. Stockpiled low grade ore within the Ore Reserve may not be processed late in the mine life if the gold price at the time is materially lower than assumed.
	Market assessment	1	No material risk.
	Economic	3	The project is constructed with no further significant capital expenditure. The primary risk relates to downward metal price movement.
	Social	1	No known issues.
	Classification	1	Ore Reserve classification is solely based on Mineral Resource classification. No other factors were applied to alter the classification.