CENTRAL TANAMI PROJECT TOTAL MINERAL RESOURCE INCREASES TO 2.8 MOZ

- Updated Mineral Resource estimates for gold deposits across the Central Tanami Project in the Northern Territory have returned a total Mineral Resource of **31 Mt at 2.8 g/t gold for 2.8 Moz**, as at 30 September 2025, including **11 Mt at 3.3 g/t gold for 1.2 Moz** from the key Groundrush Gold Deposit.
- The total Mineral Resource represents an 18% increase in tonnes, **9% increase in contained ounces**, and an 8% decrease in grade compared to the previously reported Mineral Resource estimate dated 30 June 2023.
- Mineral Resources have been stringently **constrained by Pit and Stope Optimisations**, using deposit-specific cut-off grades based on a **A$3,500 per ounce gold price**, haulage to the existing Central Tanami mill site, benchmark operating costs, and processing recoveries based on free-milling material through a conventional CIL circuit, and the production and sale of a concentrate for mineralisation classified as refractory.

Perth, Australia, 7 November 2025: Tanami Gold NL (**ASX:TAM**) (**Tanami Gold** or the **Company**) is pleased to advise that the Mineral Resource Estimates (MRE) for the Central Tanami Project Joint Venture (**CTPJV**) in the Northern Territory have been updated. This update forms part of the transition to report all Mineral Resources on the Central Tanami Project in accordance with the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (**JORC Code**).

The updated CTPJV MRE totals **31 Mt at 2.8 g/t gold for 2.8 Moz** as at 30 September 2025, representing an 18% increase in tonnes, 9% increase in contained ounces, and an 8% decrease in grade compared with the previously reported MRE dated 30 June 2023 (refer to 14 September 2023 – Annual Mineral Resource Statement).

The increase primarily reflects drilling completed since 30 June 2023, higher gold price assumptions, and improved metallurgical recoveries for fresh material classified as refractory.

The updated estimates were compiled by mining consultant MoJoe Mining Pty Ltd (**MJM**) using revised geological models that more accurately represent the mineralised systems. Reported Mineral Resources have been rigorously constrained by Pit and Stope Optimisations, with deposit-specific cut-off grades based on a A$3,500/oz gold price, haulage to the existing Central Tanami mill site, benchmark operating costs, and recoveries derived from a free-milling CIL circuit (**carbon-in-leach**) or concentrate production for refractory mineralisation.

This approach satisfies the JORC Code requirement that there are Reasonable Prospects for Eventual Economic Extraction (**RPEEE**). The updated estimates do not include all results from drilling completed during the 2025 field season at the key targets Jims Gold Mine and Groundrush Gold Deposit, both of which remain open down plunge.

The CTPJV is a 50/50 Joint Venture between Tanami Gold and ASX-listed Northern Star Resources Limited (**ASX:NST**) (**Northern Star**), established to advance exploration across the 2,108 km^2 tenement holding in the Tanami region.

On 17 July 2025, Mount Gibson Iron Limited (**ASX:MGX**) announced it had entered into an agreement to acquire Northern Star's 50% interest in the CTPJV, along with adjacent 100%-owned exploration tenure *(refer ASX announcement dated 17 July 2025 – "Agreement to Acquire 50% of the Central Tanami Gold Project from Northern Star").*

Table 1 - Mineral Resource estimates for the Central Tanami Project as of 30 September 2025. (Tanami Gold 50% Northern Star 50%)

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML33760		**Southern[1], Bouncer[1], Bumper[1], Gatling[1], Miracle[1], Tombola[1], Assault[1], Bastille[1], Battey[1], South Temby[1], Dinky[1], Dice[1], Thrasher[1], Airstrip[1], Hurricane[1] & Repulse**											
Open Pit	0.5-0.7	11	1.5	1	1,700	2.5	140	1,900	2.8	170	3,700	2.6	310
Underground	1.7-1.8	-	-	-	370	2.2	27	1,300	2.8	120	1,700	2.7	150
Stockpiles	0.6	-	-	-	13	1.1	0	-	-	-	13	1.1	0
Sub-Total		**11**	**1.5**	**1**	**2,100**	**2.5**	**170**	**3,300**	**2.8**	**290**	**5,400**	**2.7**	**460**
EL26926		**Thrasher[1] & Gallifrey**											
Open Pit	0.6-0.7	-	-	-	4	1.6	0	99	1.5	5	100	1.5	5
Underground	1.5	-	-	-	2	2.0	0	110	2.2	8	110	2.2	8
Stockpiles	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-Total		**-**	**-**	**-**	**6**	**1.7**	**0**	**210**	**1.8**	**12**	**210**	**1.8**	**13**
ML(S)167		**Carbine[1], Phoenix[1], Inca[1], Daddy[1], Funnelweb[1], Harleys[1], Huntsman[1], Huntswoman[1], Katipo[1], Money[1], Redback Rise[1], Redback SE[1], Redback SW[1], Bulldog[1], Dogbolter[1], Dogbolter NE[1], Kelpie[1], Lynx[1] & Legs[1]**											
Open Pit	0.6-0.7	9	2.4	1	2,400	3.3	260	290	2.9	27	2,700	3.3	280
Underground	1.7-1.7	0	3.3	0	1,200	2.8	110	2,200	3.1	220	3,400	3.0	330
Stockpiles	0.6	470	0.6	9	210	0.7	4	-	-	-	680	0.6	14
Sub-Total		**480**	**0.7**	**10**	**3,800**	**3.0**	**370**	**2,400**	**3.1**	**240**	**6,800**	**2.9**	**630**
ML(S)168		**Camel Bore & Jims**											
Open Pit	0.6-0.7	150	2.0	9	560	2.4	43	55	1.6	3	760	2.2	55
Underground	1.4-1.6	-	-	-	140	2.2	10	1,800	3.0	170	1,900	2.9	180
Stockpiles	0.6	550	0.7	13	26	0.9	1	-	-	-	580	0.7	14
Sub-Total		**700**	**1.0**	**22**	**730**	**2.3**	**54**	**1,800**	**2.9**	**170**	**3,200**	**2.4**	**250**
ML(S)180 & EL26925		**Beaver, Banjo, Bonsai, Orion, Pendragon[2] & Cheeseman**											
Open Pit	0.6	-	-	-	370	3.0	35	130	3.2	14	500	3.1	49
Underground	1.5	-	-	-	280	2.7	24	390	2.5	31	670	2.6	55
Stockpiles	0.6	160	0.6	3	-	-	-	-	-	-	160	0.6	3
Sub-Total		**160**	**0.6**	**3**	**640**	**2.9**	**59**	**520**	**5.7**	**45**	**1,300**	**2.5**	**110**
EL28282		**Crusade[1]**											
Open Pit	0.7-0.8	-	-	-	1,500	2.2	100	79	1.5	4	1,500	2.1	100
Underground	1.8	-	-	-	83	2.6	7	1	1.8	0	84	2.6	7
Stockpiles	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-Total		**-**	**-**	**-**	**1,500**	**2.2**	**110**	**80**	**1.5**	**4**	**1,600**	**2.2**	**110**
ML22934		**Groundrush & Ripcord**											
Open Pit	0.6	-	-	-	1,000	2.0	65	150	1.5	8	1,200	1.9	73
Underground	1.5-1.6	-	-	-	5,500	3.1	550	5,900	3.5	660	11,000	3.3	1,200
Stockpiles	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-Total		**-**	**-**	**-**	**6,500**	**2.9**	**610**	**6,100**	**3.4**	**670**	**13,000**	**3.2**	**1,300**
Total Open Pit		**170**	**2.0**	**10**	**7,500**	**2.6**	**640**	**2,700**	**2.6**	**230**	**10,000**	**2.6**	**880**
Total Underground		**0**	**3.1**	**0**	**7,600**	**3.0**	**730**	**12,000**	**3.2**	**1,200**	**19,000**	**3.1**	**1,900**
Total Stockpiles		**1,200**	**0.7**	**25**	**250**	**0.7**	**6**	**-**	**-**	**-**	**1,400**	**0.7**	**31**
Total		**1,300**	**0.8**	**36**	**15,000**	**2.8**	**1,400**	**14,000**	**3.1**	**1,400**	**31,000**	**2.8**	**2,800**

Notes:

Mineral Resource estimates are not precise calculations, as they rely on the interpretation of limited information regarding the location, shape, and continuity of mineralisation, as well as the available sampling data. The quantities presented in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimates. Consequently, rounding may result in minor discrepancies in the totals.

All Mineral Resources are reported on a dry, in-situ basis and in accordance with the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.

The open pit estimates are reported at cut-off grades ranging from 0.6 g/t gold to 0.8 g/t gold and are constrained within optimised pit shells based on a A$3,500 per ounce gold price. Underground Mineral Resources are reported at cut-off grades ranging from 1.4 g/t gold to 1.8 g/t gold, incorporating all material contained within stope optimisation wireframes (inclusive of planned MSO mining dilution) and using the same long-term gold price assumption of A$3,500 per ounce.

Note[1]: Deposits containing fresh material have been identified as potentially refractory; therefore, the applied cut-off grades incorporate additional costs and adjusted recovery factors to reflect the production and sale of a gold concentrate.

Pendragon[2] is located on Exploration Licence EL26925, which surrounds Mineral Lease ML(S)180.

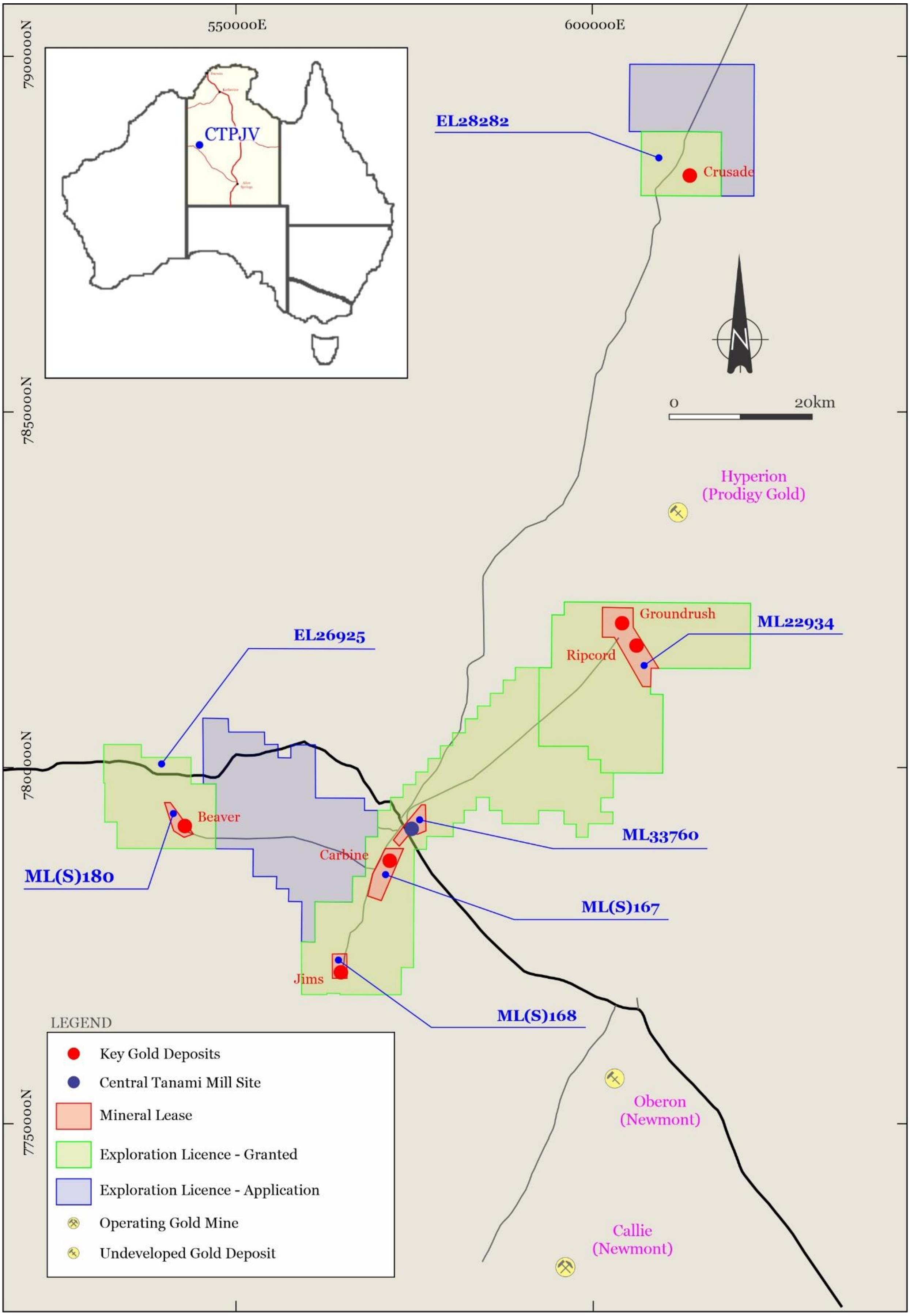

Figure 1 – CTPJV Project Tenure

ML22934

Mineral Lease ML22934 covers an area of 3,950 hectares, encompassing the Groundrush and Ripcord gold deposits. These deposits are located approximately 45 kilometres north-east of the Central Tanami mill site.

The Mineral Resource for ML22934 totals 13 Mt at 3.2 g/t gold for 1.3 Moz, representing both open pit and underground material. This includes 11 Mt at 3.3 g/t gold for 1.2 Moz from the key Groundrush Gold Deposit.

Table 2 - Mineral Resource estimates for ML22934 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML22934													
Groundrush	0.6-1.6	-	-	-	5,300	3.1	530	5,900	3.5	660	11,000	3.3	1,200
Ripcord	0.6-1.5	-	-	-	1,200	2.0	79	220	1.8	13	1,400	2.0	92
Total		-	**-**	**-**	**6,500**	**2.9**	**610**	**6,100**	**3.4**	**670**	**13,000**	**3.2**	**1,300**

Notes:

Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

ML22934 has been explored since the mid-1980s. Numerous companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), Tanami Gold and Northern Star have been active in the area.

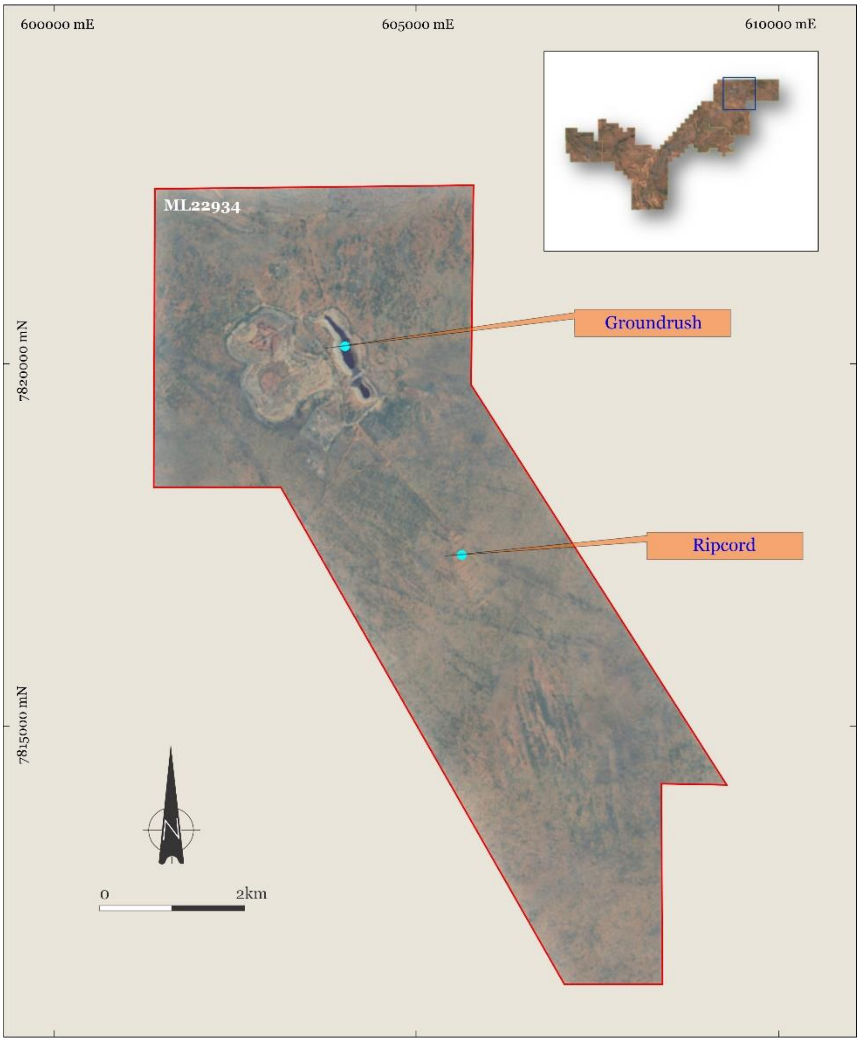

Figure 2 – ML22934

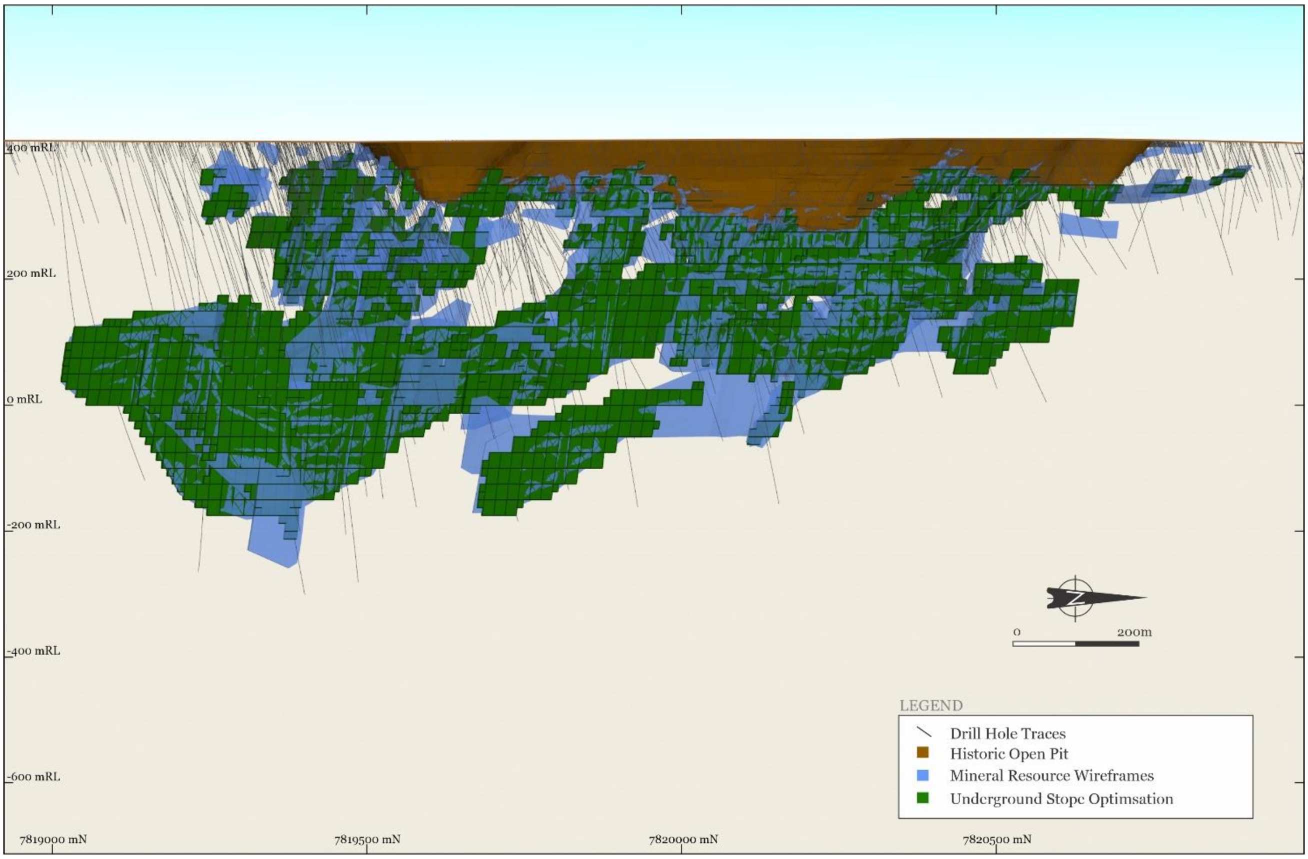

Figure 3 – Groundrush Gold Deposit

Geology and Geological Interpretation

Rocks of the Killi Killi Formation host the Groundrush and Ripcord deposits, which are exposed in a narrow north to north-northwest-trending corridor flanked by lobes of the younger Frankenia Dome granite. Groundrush and Ripcord therefore occur within rocks of a similar age to those hosting The Granites and Dead Bullock Soak gold deposits, located approximately 100 km to the south, but are older than the Mount Charles Formation, which hosts the Tanami gold deposits some 50 km to the south-west. Less than one kilometre north of Groundrush, the Killi Killi beds are truncated by a fault-bounded outlier of younger sediments belonging to the Mount Charles Formation.

At Groundrush, a package of relatively undeformed, steeply west-dipping sedimentary rocks is intruded by a fractionated dolerite unit broadly conformable with bedding. The main dolerite body exposed in the open pit consists of a coarser-grained, leucocratic quartz dolerite.

Gold mineralisation at Groundrush is primarily hosted within quartz–sulphide veins and stockwork zones developed along steeply dipping shear zones in the quartz dolerite unit, as well as within gently dipping quartz–sulphide brittle fracture veins.

The Ripcord deposit is a Palaeoproterozoic, dolerite- and sediment-hosted, quartz vein-mineralised system within the Granites–Tanami Inlier. Gold mineralisation is controlled by a brittle fracture network associated with large regional-scale structures that cross-cut a shallowly south-east-plunging regional anticline. Mineralisation is predominantly hosted within dolerite and sedimentary rocks, occurring as quartz-vein-hosted and shear-hosted mineralisation respectively.

Drilling Information and Sampling

Sampling was completed using reverse circulation (RC) and diamond core (DD) drilling. Some drillholes were pre-collared with RC methods and completed with diamond tails, while others were drilled entirely as RC or diamond core from surface.

Diamond drilling utilised a combination of HQ and NQ2-sized core. HQ core was drilled until competent ground was intersected, then continued in NQ2. Drill core was oriented, aligned, and half-cut using metre-based and geologically determined intervals (maximum 1.2 m, minimum 0.3 m), with geological boundaries taking precedence.

Sampling of diamond holes was completed using a core saw, with half-core samples taken on 0.3–1.2 m intervals honouring lithological boundaries.

RC drilling employed a 5.25-inch face-sampling hammer bit with samples taken at 1 metre intervals.

The number of drill holes shown by type for Groundrush and Ripcord are in the following tables.

Table 3 - Summary of Groundrush Drill hole database and holes used in modelling

In Groundrush. Database			In UG Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	16	882		
DD	263	76,019.20	222	3,324.13
GC	41,183	321,743	17,018	82,908.58
RB	611	32,125		
RC	470	54,178	304	4,921.55
RC_DD	113	43,959.06	91	942.54
TR	8	11		
VC	897	8,283.00		
Total	**43,561**	**537,200.56**	**17,635**	**92,096.80**

Table 4 - Summary of Ripcord Drill hole database and holes used in modelling

In Ripcord Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	84	4,785		
DD	6	1,088	5	83.16
RB	256	12,806.20		
RC	287	37,856	165	1599.48
VC	83	776.4		
Total	**716**	**57,311**	**170**	**1682.64**

Sample Preparation and Analysis

Sample preparation was completed at a range of laboratories depending on the drilling campaign and was considered appropriate for the material type.

- Pre 2000 Normandy completed sample preparation in Alice Springs.
- 2000 – 2005 Newmont samples were sent to ALS Alice Springs for 50 g fire assay (method Au-AA26). Preparation involved jaw-crushing followed by pulverisation in an LM5 mill. Barren quartz flushes were inserted between samples to minimise contamination.
- In 2012, samples were sent to Intertek Genalysis, with preparation in Alice Springs and analysis in Townsville. Samples were dried (~120 °C), crushed, rotary-split where required, and pulverised.
- Northern Star drilling samples were prepared at ALS Perth, commencing with sorting, checking, and drying at <110 °C to prevent sulphide breakdown. Samples were jaw-crushed to <6 mm, then reduced (if >3 kg) using a Boyd crusher and rotary splitter to <3 mm. The entire sample (or sub-sample) was pulverised to 90% passing 75 µm using an LM5 bowl pulveriser. A 300 g pulp was then scooped and stored in labelled packets.

Gold concentration was determined through several analytical methods depending on the campaign:

- Normandy samples were analysed at Analabs Adelaide using methods P603 (Acid Digest, Carbon Rod Finish), P625 (Acid Digest, AAS Finish), P630 (30 g Fire Assay, AAS Finish), and P650 (50 g Fire Assay, AAS Finish). Aqua regia digestion was the default unless visible gold was logged, in which case fire assay or screen fire assay was used. Samples >2 ppm Au were re-assayed by fire assay; those >7–8 ppm were re-assayed by screen fire assay.

- Tanami Gold (2011) samples were analysed at SGS Perth using 50 g fire assay with AAS finish (FAA505), measuring total gold.

- Intertek Genalysis (2012) analyses used a 50 g lead collection fire assay with aqua regia digestion of the prill and flame AAS determination to 0.005 ppm (FA50/AA).

- Northern Star samples analysed by ALS Perth used 50 g lead collection fire assay with MP-AES finish, measuring total gold.

Estimation Methodology and Classification

Grade estimation was completed using Ordinary Kriging (OK) with an oriented 'ellipsoid' search, implemented in Surpac software.

Three-dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data were composited to 1 m downhole intervals using the best fit method for Groundrush and the fixed length method for Ripcord.

Extreme grade outliers were managed through top-cuts determined via statistical analysis (histograms, log-probability plots, and coefficients of variation using Supervisor software. Only gold was interpolated into the block model.

Block model parameters:

- Groundrush underground: 10 m x 5m x 5m parent block size, sub-blocked to 2.5 m × 0.625 m × 0.625 m.

- Ripcord: 10 m × 5 m × 5 m parent blocks, sub-blocked to 2.5 m × 1.25 m × 1.25 m.

Block dimensions were selected at approximately half the average drill spacing in well-drilled areas, with adequate resolution in the across-strike and down-dip directions.

Interpolation employed an oriented ellipsoid search aligned to the average strike and dip of mineralisation.

- Groundrush and Ripcord:
    - Pass 1: 20–40 m radius, 2–6 samples
    - Pass 2: 40–80 m radius, 2–6 samples
    - Pass 3: 80–160 m radius, 2–4 samples
    - Maximum samples: 8–14 (Groundrush), 10–12 (Ripcord)

Unfilled blocks: estimated

- using nearest neighbour in a fourth pass

Validation included qualitative visual checks (sectional comparison of block grades versus drilling) and quantitative comparisons between composite input grades and block output grades. Trend analyses (eastings and elevations) confirmed good correlation between composite and estimated grades.

The Groundrush Mineral Resource has been constrained by wireframed mineralised envelopes and an A$3,500/oz stope optimisation, inclusive of dilution.

The Ripcord open-pit Mineral Resource is constrained by mineralised wireframes and reported above a 0.6 g/t gold cut-off within an A$3,500/oz pit shell, undiluted by external waste. The Ripcord underground Mineral Resource is similarly constrained by A$3,500/oz stope optimisation and includes all material within the optimised shapes.

The Mineral Resource estimate complies with the JORC Code (2012 Edition) and is classified as Indicated and Inferred based on data quality, drill spacing, and lode continuity.

Indicated Mineral Resources are defined in areas drilled at approximately 20–25 m × 25 m spacing with strong geological and grade continuity.

Inferred Mineral Resources are assigned to areas with wider or insufficient drill coverage where continuity is inferred from limited data. A minimum of three drillholes defining strike and dip was required for Inferred classification.

Mining, Metallurgy and Other Modifying Factors

It is assumed that the Groundrush Deposit will be mined by underground methods, while the Ripcord Deposit will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in CIL processing recoveries for Ripcord of 97% oxide, 90% transitional, 89% fresh and 94% for Groundrush, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix within Section 3.

Groundrush was previously mined by open pit methods from 2001 to 2005 by Newmont and produced 4.76 million tonnes @ 4.03 g/t Au for 611,000 ounces of gold. A review of the Newmont mill reconciliation data gives a claimed gold recovery of 98.9% from all material that was processed.

ML33760

Mineral Lease ML33760 covers an area of 1,120.34 hectares and forms part of the Central Tanami Project. It encompasses the Central Tanami mill site, located approximately 1.0 to 3.3 kilometres from the Bastille, Battery, Assault, South Temby, Dinky and Dice, Hurricane–Repulse, Southern, Thrasher, Tombola, Miracle, Gatling, Bouncer, and Bumper gold deposits.

The Mineral Resource for ML33760 totals 5.4 Mt at 2.7 g/t gold for 460 koz, representing open-pit, underground, and stockpiled material derived from 16 gold occurrences.

Table 5 - Mineral Resource estimates for ML33760 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML33760													
Southern	0.6-1.7	-	-	-	330	2.3	25	390	2.3	29	730	2.3	53
Bouncer	0.6-1.7	-	-	-	5	2.9	0	41	2.5	3	46	2.6	4
Bumper	0.6-1.7	-	-	-	1	2.9	0	3	2.5	0	4	2.6	0
Gatling	0.6-1.7	-	-	-	70	2.6	6	79	1.7	4	150	2.1	10
Miracle	0.6-1.7	-	-	-	58	2.5	5	60	2.1	4	120	2.3	9
Tombola	0.6-1.7	-	-	-	180	2.1	12	100	2.1	7	280	2.1	19
Assault	0.6-1.7	11	1.5	1	17	2.0	1	28	2.4	2	56	2.1	4
Bastille	0.6-1.7	-	-	-	130	2.6	11	21	1.5	1	150	2.5	12
Battery	0.6-1.7	-	-	-	110	3.0	11	-	-	-	110	3.0	11
South Temby	0.6-1.7	-	-	-	89	2.1	6	26	2.0	2	120	2.1	8
Dinky	0.6-1.7	-	-	-	140	2.1	9	200	2.1	14	340	2.1	23
Dice	0.6-1.7	-	-	-	54	1.7	3	64	2.2	5	120	2.0	8
Thrasher	0.6-1.7	-	-	-	97	2.2	7	46	2.2	3	140	2.2	10
Airstrip	0.6-1.7	-	-	-	210	1.9	13	340	2.1	23	550	2.1	36
Hurricane	0.6-1.7	-	-	-	190	3.0	18	1,300	3.5	150	1,500	3.5	160
Repulse	0.6-1.7	-	-	-	420	2.9	39	590	2.6	50	1,000	2.8	89
Stockpiles	0.6	-	-	-	13	1.1	0	-	-	-	13	1.1	0
Total		**11**	**1.5**	**1**	**2,100**	**2.5**	**170**	**3,300**	**2.8**	**290**	**5,400**	**2.7**	**460**

Notes:

Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

ML3360

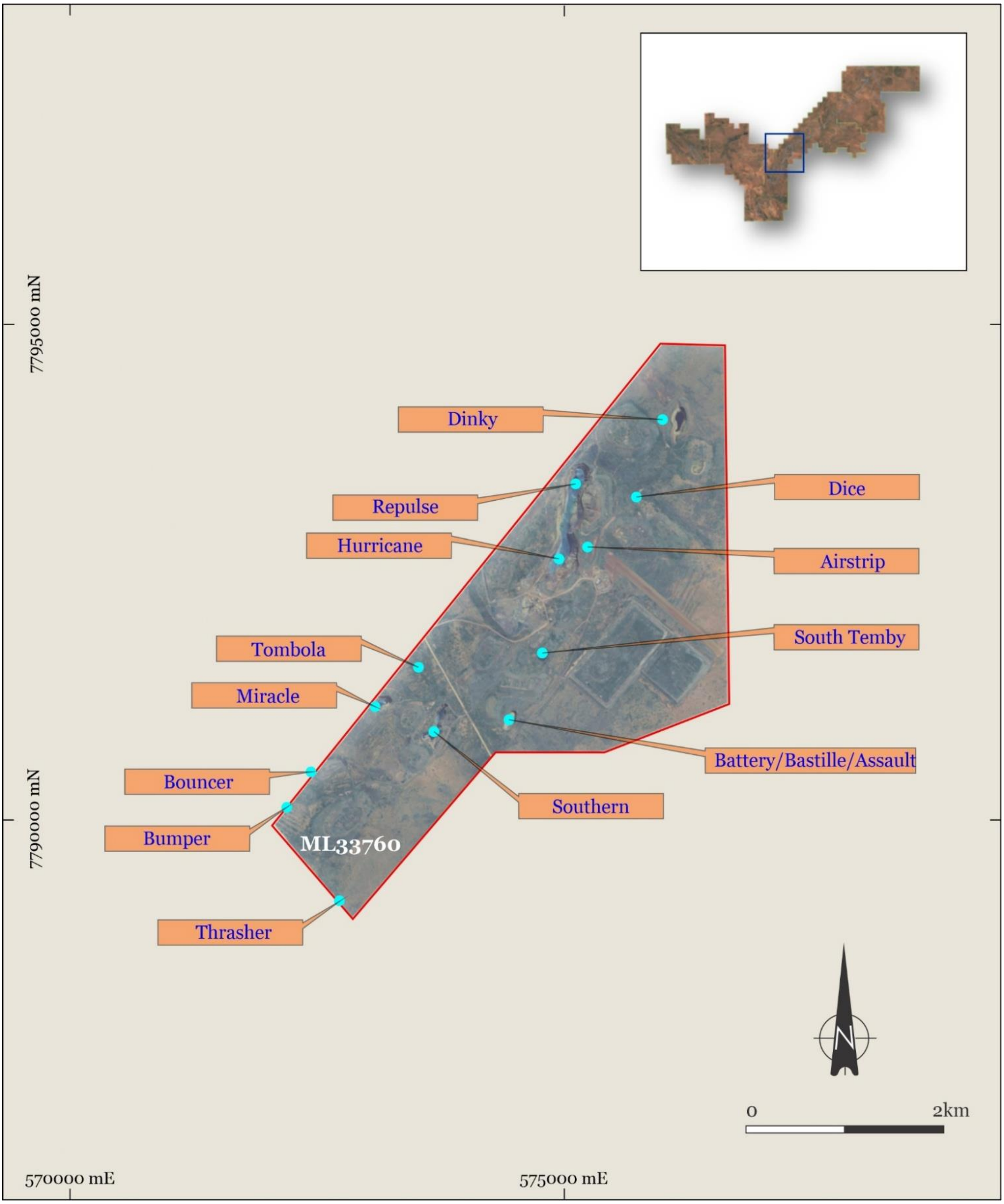

Figure 4 – ML33760

The Bastille, Battery, Assault, South Temby, Dinky and Dice Hurricane-Repulse, Southern, Thrasher Tombola, Miracle, Gatling, Bouncer and Bumper areas have been explored since the early 1990's. Several companies including, Zapopan NL, Otter Gold Mines, Newmont (Asia Pacific), and Tanami Gold have been active in the area. The Hurricane-Repulse area has been explored since the mid-1980's. Several companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area.

Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold and Northern Star.

Geology and Geological Interpretation

The Bastille, Battery, Assault, South Temby, Dinky and Dice, Southern, Thrasher Tombola, Miracle, Gatling, Bouncer and Bumper deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that are part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures, which crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts.

Dinky and Dice Deposits

The Dinky-Dice area is overlain by thin transported colluvium, sand, silt and laterite, with weathering extending to 70 to 110 metres below the surface. The surface horizons are typically 1 to 2 metres thick.

The Dinky and Dice gold deposits are largely hosted by basalt with lesser sandstones and siltstones. The strike and dip of the basalt / sediment contacts are about 035° and -70° northwest. The Dinky gold deposits strike between 010° to 035° and dips -25° to -65° SE depending on the lode. The Dice gold deposits strike between 000° to 38° and dips -40° to -85° SE. Both Dinky and Dice had significant supergene gold mineralisation.

Battery, Assault and South Temby Deposits

The Battery, Assault and South Temby area is overlain by thin transported colluvium, sand, silt and laterite, with weathering extending to 65 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick.

The strike and dip of the basalt / sediment contacts are about 035° and -75° west. The Bastille gold deposit is predominantly hosted by sandstones and siltstones with lesser basalt. Mineralisation at Battery, Assault and South Temby is largely hosted by basalt with lesser sedimentary units.

The majority of the mineralisation appears to be associated with basalt / sedimentary contacts that are complex and numerous with a number of cross cutting faults. Gold mineralisation is associated with sericite-quartz-carbonate-pyrite alteration. The Bastille Battery gold deposits strike between 020° to 036° and dips -65° to -75° SE depending on the lode. The Assault gold deposits strike between 13° to 49° and dips -62° to -70° SE. The South Temby gold deposits strike between 16° to 37° and dips -50° to -85° SE.

Hurricane-Repulse Deposits

The Hurricane-Repulse deposits are hosted by mafic volcanic flows (pillowed, vesicular and massive basalt flows) some volcanic flow breccias, sequences of lithic sandstones, siltstones and mudstones, occasional coarse sediments consisting of very proximal volcanic fragments, and more minor to rare siliceous/cherty horizons, and rare graphitic mudstones.

Southern Deposit

The local geology at the Southern Gold Deposit consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures.

The area is overlain by thin transported colluvium, sand, silt and laterite, with weathering extending to 60 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick.

The Southern Gold Deposit is spatially associated with the contact between the Hurricane Sediment and the Redback Basalt Complex. The sediment / basalt contacts are complex and numerous. Doran (2013) proposed that the mineralisation was hosted by southeast dipping shear zones that encompass northwest dipping vein sets. The deposit mostly strikes between 007° to 062° and dips -65° to -85° SE depending on the lode. There are some minor lodes that dip 35° to 57° West, but they do not appear to be economically significant.

Thrasher Deposit

The local geology at the Thrasher Gold Deposit consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures.

The area is overlain by thin transported colluvium, sand, silt and laterite, with weathering extending to 70 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick.

The Thrasher Gold Deposit is spatially associated with the contact between the Hurricane Sediment and the Redback Basalt Complex. The sediment / basalt contacts are complex and numerous.

Tombola, Miracle, Gatling, Bouncer and Bumper Deposits

The local geology in the Tombola, Miracle, Gatling, Bounce and Bumper area consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures.

The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 60 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick.

The Tombola gold deposit historically has been considered to be mineralisation north of the Miracle Open Pit giving a total strike length of 900 metres. The Tombola Gold Deposit is mostly associated with the contact of an offset of the Bouncer Basalt and the Hurricane Sediment unit. An 085° trending fault creates the offset in the basalt / sediment sequence. The Basalt north of the fault strikes at 037° to 040° and dips -60° NW. The gold mineralisation in this area strikes at between 042° to 044° and dips -20° to -60° SE.

The Basalt south of the fault strikes at 045° to 065° and dips -50° to -60° NW. Gold mineralisation strikes at 050° to 060° and dips -50° to -60° SW. The local geology consists of north westerly dipping basalt, sandstone, siltstone and mudstone. Mineralisation is hosted by sediments and lesser basalt.

The Miracle gold deposit has been defined as the mineralisation around the Miracle Open Pit and to the southwest of the pit. The Bouncer Basalt in this area is discontinuous and strikes at about 060° and dips -60° NW. Gold mineralisation in this area strikes between 060° to 070° and dips -60° SE and is hosted mainly by sediment and lesser basalt.

The Gatling gold mineralisation is considered to be the area south and east of the Miracle Open Pit and north of the filled in Bouncer Open Pit. The Basalt in this area strikes at about 040° and dips -60° NW. Gold mineralisation is associated with the contacts between the basalt and sediment and strikes between 040° and 075° and dips -60° to -70° SE.

The Bouncer Gold deposit is the area around the now back filled Bouncer Open Pit. The Basalt in this area is striking at about 040° to 055° and dips -50° to -65° NW. Gold mineralisation is closely associated with the basalt / sediment contacts. Gold mineralisation strikes between 045° to 080° and dip -50° to -75° SE.

The Bumper Gold deposit is the area around the now back filled Bumper Open Pit. The Basalt in this area is striking at 050° and dipping -50° to -60° NW and appears to consist of two separate units separated by thin sediment units. Gold mineralisation strikes at about 097° to 098° and dips -60° to -70° S.

Drilling Information and Sampling

Sampling was completed using reverse circulation (**RC**) and diamond core drilling. RC drilling was completed using a 5.75" face sampling hammer drill bit, whilst diamond core drilling was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or similar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.

The number of drill holes shown by type for the ML3370 mineral resources are in the following tables.

Table 6 - Summary of Bastille, Battery, Assault and South Temby Drill hole database and holes used in modelling

In Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
DD	3	511.79		
DW	4	314		
RB	524	20139.5		
RC	342	22753.76	167	1543
WB	4	306	1	27
Grand Total	**877**	**44,025.05**	**168**	**1,570.00**

Table 7 - Summary of Dinky & Dice Drill hole database and holes used in modelling

In Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	2	103		
RB	863	27393		
RC	526	29148	341	2775
SL_RC	9	378		
TR	16	1939		
WB	4	330	1	10
Grand Total	**1414**	**57,603**	**338**	**2,785**

Table 8 - Summary of Hurricane Repulse Drill hole database and holes used in modelling

In Hurricane-Repulse Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
DD	75	13940.19	56	745.77
RB	1352	48429		
RC	1080	82946.47	646	6538.8
TR	6	945		
BH	241607	705655		
WB	5	404		
Grand Total	**244121**	**852,319.66**	**702**	**7284.57**

Table 9 - Summary of Southern Drill hole database and holes used in modelling

In Southern Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
DD	19	3600.12	16	168.99
DW	23	794		
RB	145	5906		
RC	486	31158.46	303	2504.46
WB	3	276		
Grand Total	**676**	**41,734.58**	**319**	**2,673.45**

Table 10 - Summary of Thrasher Drill hole database and holes used in modelling

In Thrasher Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	3	218		
DW	43	4088		
RB	262	8361.1		
RC	73	4549	54	311
Grand Total	**381**	**17,216.10**	**54**	**311**

Table 11 - Summary of Tombola, Miracle, Bouncer, Bumper and Gatling Drill hole database and holes used in modelling

In Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	21	1464		
DD	32	5918.52	11	65.65
DW	62	4826		
RB	889	36155		
RC	17547	91375.94	6,955	12,328.43
WB	3	276		
Grand Total	**18542**	**138,039.46**	**6,966**	**12,394.08**

Sample Preparation and Analysis

RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg.

The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample.

RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter.

For RC holes drilled by Tanami Gold in 2010 to 2012 samples were taken at 1 metre intervals from the cyclone and collected from a 75:25% riffle splitter in prenumbered sample bags. The samples varied from wet to dry. Samples were generally around 3 kg in weight

For CTP holes, 1m RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole.

For RC holes drilled by Zapopan NL in the late 1980s and early 1990s samples were taken at 1 metre intervals from the cyclone and placed into labelled plastic bags. Samples were combined into 2 metre composites, but the method is unknown.

RC holes drilled by OGM in the mid-1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a four-deck riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40-litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples. During mining operations (mid-1990s to 2001) under the Tanami Gold Joint Venture (OGM) drill samples were analysed offsite at ALS Alice Springs however with the availability of the onsite laboratory, the database does include some onsite analysis. There was no fixed procedure for selecting on- or offsite analysis; rather the choice was governed by onsite laboratory availability. Analysis (both on and offsite) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot whereas ALS use a 50ml aliquot for all AAS readings.

RC holes drilled by Tanami Gold were collected through a 75:25% riffle splitter in prenumbered bags. The samples varied from wet to dry. The samples varied from wet to dry. Samples were generally around 3 kg in weight. TAM (2010 - 2012) sent samples to SGS Laboratories in Perth for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). This method had a 0.01 ppm gold detection limit.

Diamond holes drilled by Tanami Gold were completed using NQ2 size core. Half core samples were taken down the length of the hole.

For RC holes drilled by Northern Star in 2024 samples were taken at 1 metre intervals using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. Northern Star (2024) sent samples to ALS Adelaide, commencing with sorting, checking, and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg, a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets. Gold concentration was determined for Northern Star samples sent to ALS in Adelaide by fire assay using the lead collection method with a 50g sample charge weight. MP-AES instrument finish was used to measure gold levels. The methodology used measures total gold.

Gold concentration was determined in various ways depending on the drilling program. From the late 1980s to about March 1994, most of the samples collected by Zapopan NL were assayed for gold by fire assay with a 0.01 ppm detection limit at the onsite laboratory. Samples collected during mining operations (the mid-1990s to 2001) under the Tanami Gold Joint Venture were submitted to the onsite laboratory or ALS in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings.

Tanami Gold (2010-2012) sent samples to SGS Laboratories in Perth for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). This method had a 0.01 ppm gold detection limit.

Samples collected by CTP were sent to ALS in Malaga, Perth. Gold concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility.

This was common to both Diamond Core and RC Chip sample collection.

Estimation Methodology and Classification

Ordinary Kriging (OK) interpolation with an oriented ‘ellipsoid’ search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the ‘fixed length’ method. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV’s, and summary statistics) using Supervisor software.

MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the deposits.

Only gold was interpolated into the block model, estimation of deleterious elements was not carried out.

The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation. A first pass of radius 20-60m with a minimum number of samples of 2-6 samples and a second pass of radius 40-120m with a minimum number of 2-6 samples were used. A third pass of search radius 80-240m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples was set at 10 to 12. Blocks that did not fill after 3 passes were given a 4th pass using nearest neighbour estimation. Lodes that were defined by RAB drilling were assigned a 0.5 g/t gold grade and pass 5.

To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.

The Mineral Resource estimates were constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.6 g/t gold, 0.7 g/t gold, and 0.7 g/t gold cut-off grade in Oxide, Transitional and Fresh, respectively for open pit material within a $A3500 per ounce pit shell.

The only exception to the cut off grades is Repulse where the grades are reported above 0.5 g/t gold, 0.5 g/t gold, 0.7 g/t gold cut-off grade in Oxide, Transitional and Fresh, respectively for open pit material within a $A3500 per ounce pit shell. This is due to favourable metallurgical gold recoveries.

The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an Inferred Resource is 3 drill holes spaced apart so that strike and dip can be determined.

Validation of the block model shows good correlation of the input data to the estimated grades where there were sufficient composites for kriging to be effective.

Mining, Metallurgy and Other Modifying Factors

Historical processing results and supporting testwork confirm that refractory deposits are predominantly associated with zoned arsenian pyrite + arsenopyrite, particularly around the existing mill on ML33760, MLS167 and north of Groundrush.

MineScope and AFX Commodities completed a Scoping Study evaluating the potential to treat semi-refractory, fresh material. The study utilised old historical core from the main deposits Hurricane, Carbine, and Legs, assuming the existing processing plant is upgraded to a 1.5 Mtpa design capacity.

Based on the resulting metallurgical recoveries, SWOT analysis, capital/operating benchmarks, and MineScope's database a concentrate production for export was considered for incorporation into the MRE update.

The study concluded an additional operating cost of A$3.95/t of plant feed and capital cost of A$34M to produce concentrate. The post mine-gate, concentrate costs were estimated at a net smelter return of 85.1% including:

- Concentrate Transport of US$370/dmt conc
- Concentrate Treatment & Refining US$148/dmt conc
- Payable Au Factor of 93.0%

It is assumed that the ML33760 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill and additional circuit to produce a concentrate.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in processing recoveries in Table 12, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 2 within Section 3.

Table 12 - ML33760 Processing Recovery

Deposit	Processing Recovery			
	Oxide	Transitional	Fresh CIL	Fresh Floatation
Bastille	90.0%	76.0%	10.1%	85.1%
Dinky Dice	86.0%	75.0%	10.1%	85.1%
Hurricane	86.0%	75.0%	10.1%	85.1%
Repulse	95.0%	94.0%	74.0%	0.0%
Southern	90.0%	76.0%	10.1%	85.1%
Thrasher	90.0%	76.0%	10.1%	85.1%
Tombola Miracle	86.0%	75.0%	10.1%	85.1%

EL26926

Exploration Licence EL26926 covers an area of approximately 649.03 km^2 (204 blocks) and forms part of the Central Tanami Project. It is situated 29 kilometres southwest of the Central Tanami Mill site, encompassing the Galifrey Gold Deposit.

The Mineral Resource on EL26926 totals 214kt grading 1.8 g/t gold for 13koz, representing open-pit and underground material.

Table 13 - Mineral Resource estimates for EL26926 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
EL26926													
Thrasher	0.6-1.7	-	-	-	5	1.7	0	29	2.3	2	34	2.2	2
Galifrey	0.6-1.5	-	-	-	-	-	-	180	1.8	10	180	1.8	10
Total		**-**	**-**	**-**	**6**	**1.7**	**0**	**210**	**1.8**	**12**	**210**	**1.8**	**13**

Notes:
Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

The Galifrey area has been explored since 1989. Several previous companies, Otter Gold Mines, Zapopan NL, and Tanami Gold NL have been active in the area.

Geology and Geological Interpretation

The Galifrey deposit is a Palaeoproterozoic, granite / felsic intrusive and sediment shear hosted deposit that is part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a quartz stockwork that is associated with granite / felsic intrusive and sediment contacts.

Gold mineralisation at Galifrey consists of two separate shear zones with strike lengths of about 2,400 and 500 metres respectively. Strikes of individual lenses of primary mineralisation vary from 295° to 335° and dip steeply. There are some minor, near surface regolith hosted laterite and mottled clay mineralisation that are flat lying. The strike length of individual lenses of gold mineralisation varies from 25 to 500 metres but are more typically 50 to 60 metres. True thickness varies from 1 to 2 metres to several metres. The down dip extent is typically of the order of 50 to 100 metres.

Drilling Information and Sampling

Sampling was completed using RC and diamond core drilling.

RC drilling was completed using a 5.75" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.

The number of drill holes shown by type for the Galifrey mineral resources are in the following tables.

Table 14 - Summary of Galifrey Drill hole database and holes used in modelling

In Galifrey Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	162	12264		
DD	8	1306.45	5	88
RB	1451	37682		
RC	42	5157	17	371
Grand Total	**1663**	**56,409.45**	**22**	**459**

Sample Preparation and Analysis

RC drillholes were sampled using either a cyclone rotary splitter mounted on the RC drill rig for an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg.

The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample.

RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter.

Northern Star Stage-1 RC drilling saw all bulk material collected on a 1m basis directly from cyclone in pre labelled green plastic mining bags.

Northern Star Stage-2 RC drilling saw single metre (1m) samples collected from a trailer mounted static cone splitter. Approximately 12.5% of each meter sample was collected in a pre-labelled calico bag with the depth while the remaining 87.5% was collected in a green mining bag and retained.

All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.

Estimation Methodology

Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software.

Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation. A first pass of radius 20-50m with a minimum number of samples of 2-6 samples and a second pass of radius 40-100m with a minimum number of 2-6 samples were used for Galifrey. A third pass of search radius 80-200m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 18-28 depending on the number of samples in the domain. Blocks that did not fill after 3 passes were given a 4th pass using nearest neighbour estimation.

To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.7g/t gold cut-off grade for oxide, 0.6 g/t gold for transitional and fresh for open pit material within a $A3500 pit shell. For underground all material within a A$3500 stope optimisation is reported.

The Mineral Resource was classified as Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Mining, Metallurgy and Other Modifying Factors

MineScope and AFX Commodities completed a Scoping Study evaluating the potential to treat semi-refractory, fresh material. The study utilised old historical core from the main deposits Hurricane, Carbine, and Legs, assuming the existing processing plant is upgraded to a 1.5 Mtpa design capacity.

Based on the resulting metallurgical recoveries, SWOT analysis, capital/operating benchmarks, and MineScope's database a concentrate production for export was considered for incorporation into the MRE update.

The study concluded an additional operating cost of A$3.95/t of plant feed and capital cost of A$34M to produce concentrate. The post mine-gate, concentrate costs were estimated at a net smelter return of 85.1% including:

- Concentrate Transport of US$370/dmt conc
- Concentrate Treatment & Refining US$148/dmt conc
- Payable Au Factor of 93.0%

It is assumed that the ML33760 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill and additional circuit to produce a concentrate.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in processing recoveries, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 3 within Section 3.

Galifrey is in an Inferred Mineral Resource that may be mined by open pit and underground methods if further drilling improves the resource classification. There is no metallurgical data from the Galifrey prospect. It has been assumed for RPEEE purposes that this deposit may have similar recoveries to the Jims deposit. The following gold recoveries were used.

- CIL Processing Recovery 76% oxide, 95% transitional, 92% fresh

ML(S)167

Mineral Lease (Southern) MLS167 covers an area of 1,877 ha and encompasses the Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca gold deposits approximately 10 kilometres southwest of the Central Tanami Mill site.

The Mineral Resource on ML(S)167 totals 6,800kt grading 2.9 g/t gold for 630koz, representing open-pit, underground and stockpile material.

Table 15 - Mineral Resource estimates for ML(S)167 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML167													
Carbine	0.6-1.7	-	-	-	1,100	3.3	120	1,400	3.3	150	2,600	3.3	270
Phoenix	0.6-1.7	8	2.2	1	140	2.6	12	120	2.6	10	270	2.5	22
Inca	0.6-1.7				87	2.4	7	8	1.7	0	94	2.4	7
Daddy	0.6-1.7				63	2.3	5	1	1.7	0	64	2.2	5
Funnelweb	0.6-1.7				180	2.0	11	60	1.9	4	239	2.0	15
Harleys	0.6-1.7	0	3.1	0	17	3.7	2	1	4.1	0	18	3.7	2
Huntsman	0.6-1.7				23	3.9	3	-	-	-	23	3.9	3
Huntswoman	0.6-1.7				43	2.7	4	190	3.3	21	240	3.2	24
Katipo	0.6-1.7				64	2.9	6	3	2.0	0	67	2.9	6
Money	0.6-1.7	1	4.0	0	27	3.3	3	-	-	-	28	3.4	3
Redback Rise	0.6-1.7				84	2.2	6	48	2.6	4	130	2.4	10
Redback SE	0.6-1.7				170	4.4	24	41	3.9	5	210	4.3	29
Redback SW	0.6-1.7				280	2.8	25	42	2.7	4	320	2.8	29
Bulldog	0.6-1.7				250	2.2	18	54	2.5	4	310	2.2	22
Dogbolter	0.6-1.7				74	4.2	10	75	2.2	5	150	3.2	15
Dogbolter NE	0.6-1.7				52	5.2	9	6	3.3	1	58	5.0	9
Kelpie	0.6-1.7				-	-	-	63	3.1	6	63	3.1	6
Lynx	0.6-1.7				360	2.8	32	36	1.8	2	390	2.7	34
Legs	0.6-1.7				580	4.0	74	250	3.0	24	830	3.7	98
Stockpiles	0.6	470	0.6	9	210	0.7	4	-	-	-	680	0.6	14
Total		**480**	**0.7**	**10**	**3,800**	**3.0**	**370**	**2,400**	**3.1**	**240**	**6,800**	**2.9**	**630**

Notes:
Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

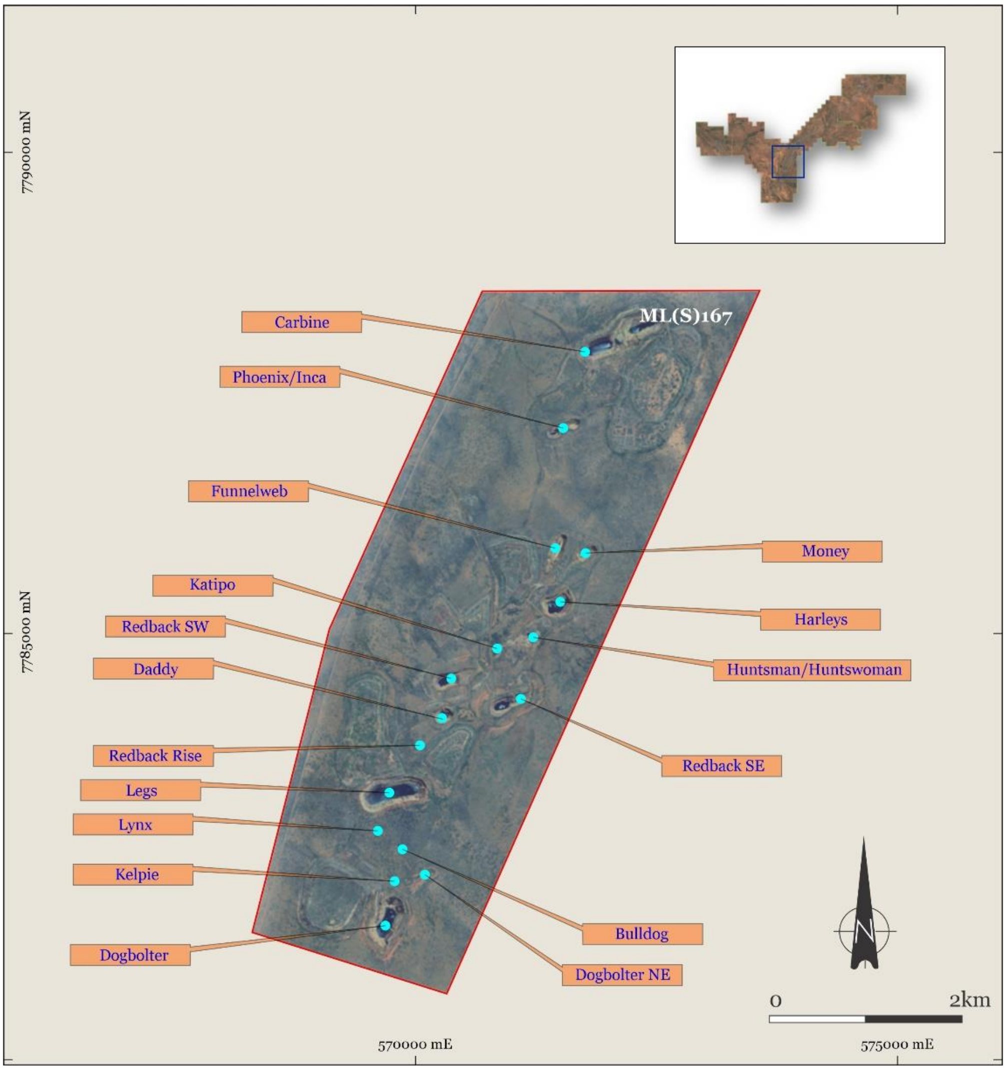

Figure 5 – ML(S)167

The Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca areas have been explored since the early 1990's. Several previous companies, Zapopan NL, Otter Gold Mines, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area.

Geology and Geological Interpretation

The Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca area deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that are part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts.

Carbine

The Carbine deposit is hosted within the Redback basalt of the Tanami Group (1838+/-6Ma), a basaltic sequence with intercalated sediments. Dolerite sills up to 200+m thick intrude the Tanami Group.

The stratigraphic sequence from south to north at the Carbine open pit consists of north dipping Hurricane sediments, Redback basalt with intercalated sediments from 0.5 to 10 metres that are unconformably overlain by Gardiner sandstone. The sequence has been interpreted as forming within an intracratonic setting. This is supported by abundant hematite and metamorphic detritus within the intercalated sediments.

Mineralisation at Carbine is mainly hosted by quartz veining within strongly altered basalt and is structurally controlled within a dominant 060° to 070° and a lesser 020° to 045° striking structures. Most of the ore mined (+90%) was hosted by pillow basalt.

The main mineralised trend at Carbine strikes at 060° to 070° and dips -55° to -80° SE and plunges about 15° SW. A lesser cross cutting zone of mineralisation strikes at 020° to 045° and dips -60° to -70° SE and plunges about 15° SE.

Dogbolter to Lynx Area Deposits

The Dogbolter gold deposit strikes between 000° to 020° and dips 60° to 70° East depending on the lode. The local geology consists of northwest dipping basalt and haematitic siltstone with minor lithic siltstone, granite and dacite. The strike and dip of the basalt / sediment contacts are about 020° and -60° west. Mineralisation is largely hosted by basalt.

Kelpie gold mineralisation strikes between 5° and 17° and dips 60° to 80° East. The local geology consists of northwest dipping basalt and lithic siltstone. The area is structurally complex with small scale faulting and folding. Overall, the strike and dip of the basalt / sediment contact is about 015° to 020° and -60° west. Mineralisation is largely hosted by siltstone with minor basalt.

The Bulldog area is along strike of the Kelpie gold mineralisation. Gold mineralisation consists of veins with 2 distinct orientations. The main lens of mineralisation strikes at 010° to 015° and dips -50° to -60° east while a cross-cutting set strikes between 35° to 70° and dips -40° to -65° southeast. 3 separate lenses of basalt intercalated with sediments strike at about 025° and dip -50° to -60° west. Several faults have been interpreted to offset the stratigraphy by up to 15 metres. Gold mineralisation is hosted by both basalt and sedimentary units.

Lynx gold mineralisation is located about 200 to 300 metres northwest of the Bulldog mineralisation that strikes and dips at 060° to 070° and -30° to -50° southeast. The local geology was interpreted from geology intersected in drilling and the airborne aeromagnetic data and consists of intercalated basalt and sediment that strike at between 040° to 050° and dip -40° to -55° northwest.

Dogbolter NE is located about 200 to 600 metres northeast of the Dogbolter main gold mineralisation. The main veins strike at between 050° to 075° and dip -60° to -85° southeast. A few minor vein sets strike at 020° to 025° and dip -50° to -55° east. Interpreted basalt units in this area strike overall at 020° and dip -50° to -60° west. The western pit was interpreted as being within the Redback Basalt complex while eastern pit was within the Harleys sediment package.

Legs Deposit

The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and a slight inflexion in the strike of the units.

The main structures within the basalt strike between 050° to 075° and dip 50° to 75° southeast while structures within the sedimentary horizons strike at about 030° to 045° and dip 30° to 80° east southeast. OGM carried out pit mapping while mining and found a number of gold bearing structures that were striking between 340° to 10° and dipping 50° to 80° North. The structures appear to be bounded by the major structures previously mentioned.

Redback Area Deposits

The gold mineralisation in the Redback area has three dominant strike directions and is structurally complex. The gold mineralisation occurs at the boundaries between basalt and sediment units defined by OGM and a slight inflexion in the strike of the units. The local geology consists of northwest dipping basalt, sandstone, and siltstone.

There are 3 main gold mineralisation trends with the Redback area that appear to be controlled by faulting and rheology contrast provided by the contacts of sedimentary and basaltic units. The Funnelweb trend strikes at about 020° and occurs within the Redback basalt sequence. This sequence is intercalated with conglomerates, sandstone and siltstone. A second major trend occurs from Redback SW to Harley's open pit with the mineralisation striking overall at 060°. The third trend occurs from Daddy open pit to Redback SE open pit where the overall strike of the mineralisation varies from 060° to 090°. The Money open pit does not align with other deposits but is associated with a 060° trend in the footwall of the Harley's sediments in the footwall basalt complex.

The Funnelweb mineralisation is hosted within the Redback basalt sequence. The basalt in the area of the Funnelweb open pit is striking at 020° and dipping about -65° west. The basalt in this area is 50 to 80 metres thick and is bounded on the eastern side by conglomerate and sandstone / siltstone and mudstone on the western contact. A number of cross cutting fault zones have been mapped but displacement is not shown. The sequences are unconformably overlain by the Gardiner Sandstone in the northern part of the pit.

Geological mapping of the Money open pit has shown the rocks consist of the Footwall Basalt Complex and Harley's sediments that are unconformably overlain by the Gardiner Sandstone in the northern part of the pit. Harley's sediments in this area consist of conglomerate, sandstone, and siltstone. A well-developed laterite zone occurred over the northwest portion of the open pit. The overall interpreted strike and dip of the basalt sequence in this area is between 040° to 055° and -40° to -50° northwest.

Geological mapping of the Harley's open pit showed that the majority of the open pit is within the Harley's sediment sequence that consists of conglomerate, sandstone, siltstone, and mudstone intercalated with dacite and basalt. The overall strike and dip of the lithologies varies in strike from 010° to 040° and -60° west. There appears to be at 3 basalt units that are distinguishable in the open pit. The basalt exposed in the western wall is interpreted to part of the Redback Basalt sequence while the remainder form part of the Footwall Basalt Complex. The Redback Basalt sequence has a thickness of about 100 metres in this area and appears to have been structurally thickened. The Footwall Basalt

Complex units are only 10 to 15 metres thick and discontinuous.

Geological mapping of the Huntsman open pit showed that it is mainly within the Harley's sediments.

The Redback Basalt sequence is interpreted to be transecting the western side of the pit. The Harley's sediments in this area have been described as micaceous sandstone, feldspathic sandstone, siltstone, and mudstone intercalated with basalt. The strike and dip of the sediment sequence is about 020° to 025° and -40° to -45° west. The eastern side of the open pit had a well-developed laterite zone. The Huntsman open pit has been backfilled and is now a small waste dump.

Huntswoman was located in Harley's sediments and strongly brecciated basalt. Recent interpretation has shown that Huntswoman had a well-developed laterite along a 060° trend.

No geological mapping could be located for the Katipo open pit, but it has been noted that it had a well-developed laterite and was hosted by the Redback Basalt sequence. The laterite is striking at about 060°but this may reflect a major structural feature. The basalt is at least 100 metres thick in this area and most likely structurally thickened. The overall strike of the basalt is likely to be around 030°.

Geological mapping of the Redback SE open shows that the western end of the pit is within Harley's sediments while the eastern part of the pit is within the Footwall Basalt Complex. Harley's sediments in this area consists of sandstone, feldspathic sandstone, hematitic sandstone, and siltstone while Footwall Basalt Complex has been described as porphyritic, pillow, vesicular and fine-grained basalt. The strike and dip of the sediments is about 020° to 030° and dip -65° west. The Footwall Basalt Complex appears to truncate some of the sediments and has an apparent strike of about 060° and dip of -50° to -60° northwest. This may reflect the structures cutting through the area. The eastern area of the open pit had a well-developed laterite.

Geological mapping of the Redback SW open pit shows that Hurricane sediments are exposed in the pit and the sequence consists of siltstone and mudstone intercalated with basalt sequences. The basalt is interpreted to be 30 to 40 metres thick and striking overall at about 010° and dipping -50° to -65° west. Laterite is developed in the eastern end of the pit. There appears to be an 060° structural trend cutting the geology.

Geological mapping of the Daddy open pits shows that the western end of the open was excavated in Hurricane sediments while the eastern portion contains basalt and interflow sediment. The sediments consist of siltstone, conglomerate, and sandstone that are striking at about 020° and dipping -50° west. The overall strike of the basalt varies from 025° to 030° and dips -45° to -65°. Major structures cutting through the open pits vary from 060° to 095° and have maximum thicknesses of 60 metres. The eastern pit has been backfilled.

Phoenix Inca Deposits

The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures.

The Phoenix gold deposit strikes between 055° to 080° and dips -65° to -85° NW depending on the lode. The local geology consists of westerly dipping basalt, conglomerate, sandstone, and intercalated siltstone and mudstone. The strike and dip of the basalt / sediment contacts are about 020° and -60° west. Mineralisation is largely hosted by basalt. Kelpie gold mineralisation strikes between 5° and 17° and dips 60° to 80° East. The local geology consists of northwest dipping basalt and lithic siltstone. The area is structurally complex with small scale faulting and folding. Overall, the strike and dip of the basalt / sediment contact is about 015° to 020° and -60° west. Mineralisation is largely hosted by siltstone with minor basalt. The Phoenix gold resource is located about 900 metres south of the Carbine gold mineralisation and shows similarities with larger deposit.

The Inca trend gold mineralisation strike between 015° to 040° and dips -50° to -75° NW depending on the lode. The local geology consists of basalt and intercalates sedimentary units that are striking at about 030° and dipping steeply northwest.

Drilling Information and Sampling

Sampling was completed using RC and diamond core drilling.

RC drilling was completed using a 5.75” face sampling hammer drill bit. Diamond core drilling was completed using HQ or NQ utilizing triple tube recovery.

RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a four-deck riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40-litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples.

RC holes drilled by Tanami Gold were collected through a 75:25% riffle splitter in prenumbered bags. The samples varied from wet to dry.

Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.

The number of drill holes shown by type for the MLS167 mineral resources are in the following tables.

Table 16 - Summary of Carbine Drill hole database and holes used in modelling

In Carbine Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	1	64.00		
DD	116	35,350.80	92	1291.36
DW	2689	85,678.00		
RB	382	15,524.00		
RC	691	40,238.41	289	2393.31
WB	11	1,240		
Total	**3890**	**178,095**	**381**	**3684.67**

Table 17 - Summary of Dogbolter to Lynx Drill hole database and holes used in modelling

In Dogbolter to Lynx Database			In Resource Model	
Hole Type	No. Holes	Metres drilled	No. Holes	Intersection Metres
DD	107	13101.3	101	12475.4
DW	2119	82567		
RB	2252	87430		
RC	2098	88072.82	2053	86227.1
Grand Total	**6576**	**271,171**	**2154**	**98,702.5**

Table 18 - Summary of Legs Drill hole database and holes used in modelling

In Legs Database			In Resource Model	
Hole Type	No. Holes	Metres drilled	No. Holes	Intersection Metres
DD	34	5387.65	23	268.57
DW	2133	81425.58		
RB	936	37299		
RC	897	48337.42	293	2480
WB	13	983	8	110
Grand Total	**4013**	**173,433**	**324**	**2,858.57**

Table 19 - Summary of Redback area Drill hole database and holes used in modelling

In Redback Rise Database			In Money Resource	
Hole Type	No. Holes	Metres drilled	No. Holes	Intersection Metres
DD	129	16532	2	20
DW	4346	147017.3		
RB	2842	123549		
RC	3468	129157.52	88	859.99
WB	1	145		
Grand Total	**10786**	**416,401**	**90**	**879.99**

Table 20 - Summary of Redback area holes used in modelling

In Funnelweb Resource		In Harleys Resource	
No. Holes	Intersection Metres	No. Holes	Intersection Metres
4	21	7	103
199	1189	132	945.98
203	**1,210.00**	**139**	**1,048.98**

Table 21 - Summary of Redback area holes used in modelling

In Huntsman Resource		In Huntswoman Resource	
No. Holes	**Intersection Metres**	**No. Holes**	**Intersection Metres**
1	8	6	141
173	1047	141	861.91
174	**1,055.00**	**147**	**922.91**

Table 22 - Summary of Redback area holes used in modelling

In Katipo Resource		In Redback SW Resource	
No. Holes	**Intersection Metres**	**No. Holes**	**Intersection Metres**
3	13	40	267
69	354.68	414	2442.92
72	**367.68**	**454**	**2,709.92**

Table 23 - Summary of Redback area holes used in modelling

In Redback SE Resource		In Daddy Resource	
No. Holes	**Intersection Metres**	**No. Holes**	**Intersection Metres**
30	360	3	153
257	2371	31	963
287	**2,731.00**	**34**	**1,116.00**

Sample Preparation and Analysis

RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg.

The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample.

RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter.

Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate.

Zapopan NL completed all sample preparation pre-1994 at the on-site laboratory.

During mining operations drill samples were prepped either onsite or at the ALS facility in Alice Springs to industry standards. The Otter Gold Mines data does include some onsite analysis at the mine laboratory.

Drill samples collected by Tanami Gold were submitted to SGS Laboratories in Perth and assayed using a 50g fire assay charge for gold with an atomic spectrometer finish. This method had a 0.01ppm detection limit. Sample weights were generally around 3kg in size.

Estimation Methodology and Classification

Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data.

Sample data was composited to 1m down hole lengths using the 'fixed length' method for Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca area deposits. Sample data was composited to 1m down hole lengths using the 'best fit" method for Carbine. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software.

Only gold was interpolated into the block model. The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation. A first pass of radius 10-40m with a minimum number of samples of 2-6 samples and a second pass of radius 20-80m with a minimum number of 2-6 samples were used. A third pass of search radius 40-160m was used with a minimum number of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8-20 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass using nearest neighbour.

To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above 0.6 g/t gold, 0.7 g/t gold, 0.7 g/t gold Cut-off in Oxide, Transitional, Fresh Rock within an optimised pit shell using A$3,500/oz. Underground resources are reported within an A$3,500/oz optimised stope below the open pit shell. This includes planned dilution.

The Mineral Resource was classified as Measured, Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression.

The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Mining, Metallurgy and Other Modifying Factors

MineScope and AFX Commodities completed a Scoping Study evaluating the potential to treat semi-refractory, fresh material. The study utilised old historical core from the main deposits Hurricane, Carbine, and Legs, assuming the existing processing plant is upgraded to a 1.5 Mtpa design capacity.

Based on the resulting metallurgical recoveries, SWOT analysis, capital/operating benchmarks, and MineScope's database a concentrate production for export was considered for incorporation into the MRE update.

The study concluded an additional operating cost of A$3.95/t of plant feed and capital cost of A$34M to produce concentrate. The post mine-gate, concentrate costs were estimated at a net smelter return of 85.1% including:

- Concentrate Transport of US$370/dmt conc
- Concentrate Treatment & Refining US$148/dmt conc
- Payable Au Factor of 93.0%

It is assumed that the ML33760 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill and additional circuit to produce a concentrate.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in processing recoveries, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 4 within Section 3.

- CIL Processing Recovery 90% oxide, 76% 95% (Legs, Dogbolter, Lynx), 90% (Carbine, Inca, Phoenix, Redback Area)
- CIL Processing Recovery transitional, 76% (Legs, Dogbolter, Lynx, Redback Area), 75% (Carbine, Inca, Phoenix)
- CIL Processing Recovery fresh tailings 10.1% of total gold fresh tailings Floatation 85.1% of total gold recovered as concentrate

ML(S)168

Mineral Lease (Southern) MLS168 covers an area of 711.9 ha and encompasses the Jims and Camels Bore gold deposits approximately 23 kilometres southwest of the Central Tanami Mill site.

The Mineral Resource on ML(S)168 totals 3,200kt grading 2.4 g/t gold for 250koz, representing open-pit, underground and stockpile material.

Table 24 - Mineral Resource estimates for ML(S)168 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML168													
Camel Bore	0.6-1.4	-	-	-	26	2.3	2	69	2.4	5	95	2.4	7
Jims	0.6-1.6	145	2.0	9	680	2.3	51	1,700	2.9	160	2,600	2.7	220
Stockpiles	0.6	550	0.7	13	26	0.9	1	-	-	-	580	0.7	14
Total		**700**	**1.0**	**22**	**730**	**2.3**	**54**	**1,800**	**2.9**	**170**	**3,200**	**2.4**	**250**

Notes:

Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

The Jims and Camel Bore area has been explored since the early 1990's. Several previous companies, Newmont (Asia Pacific), and Tanami Gold and Northern Star have been active in the area.

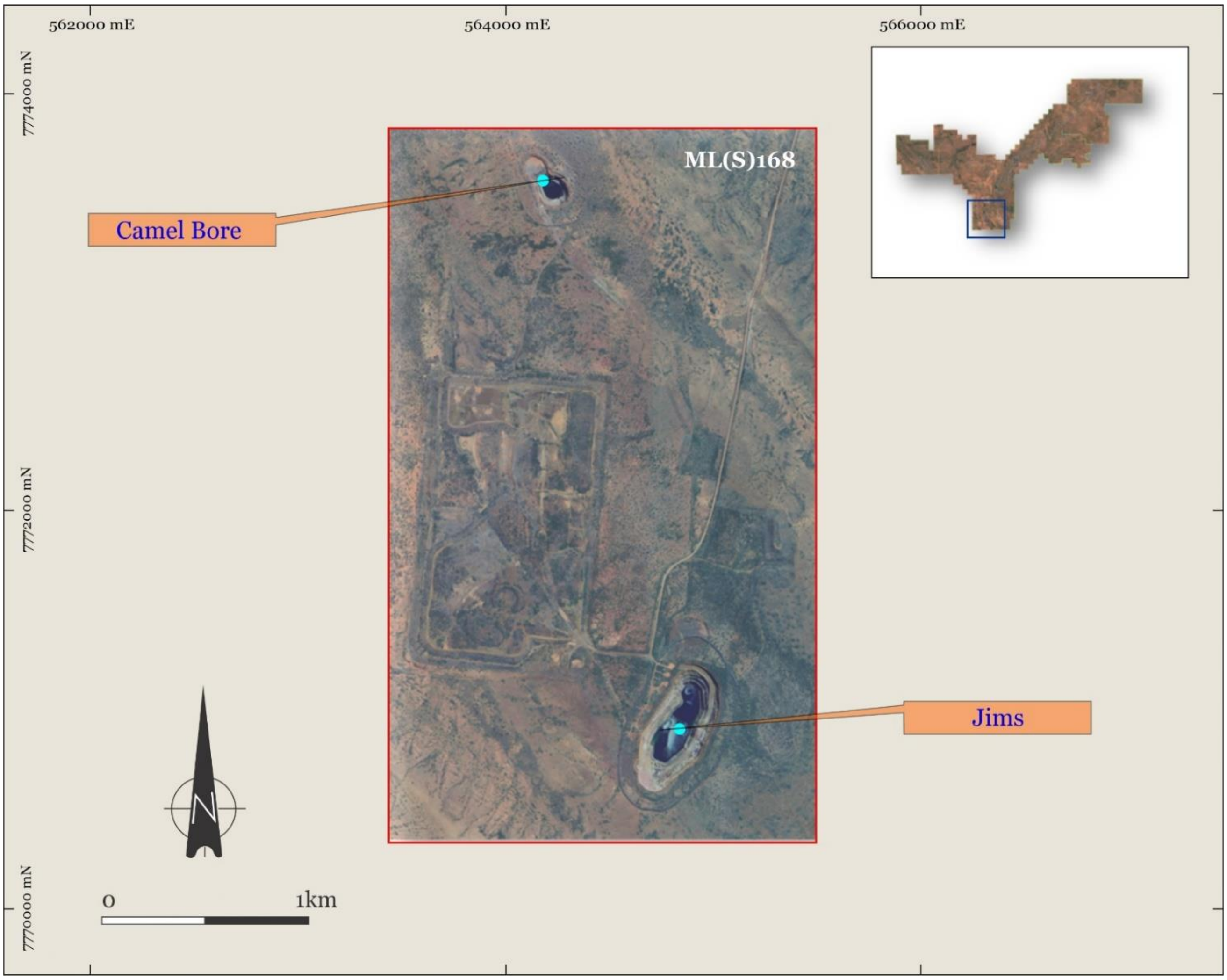

Figure 6 – ML(S)168

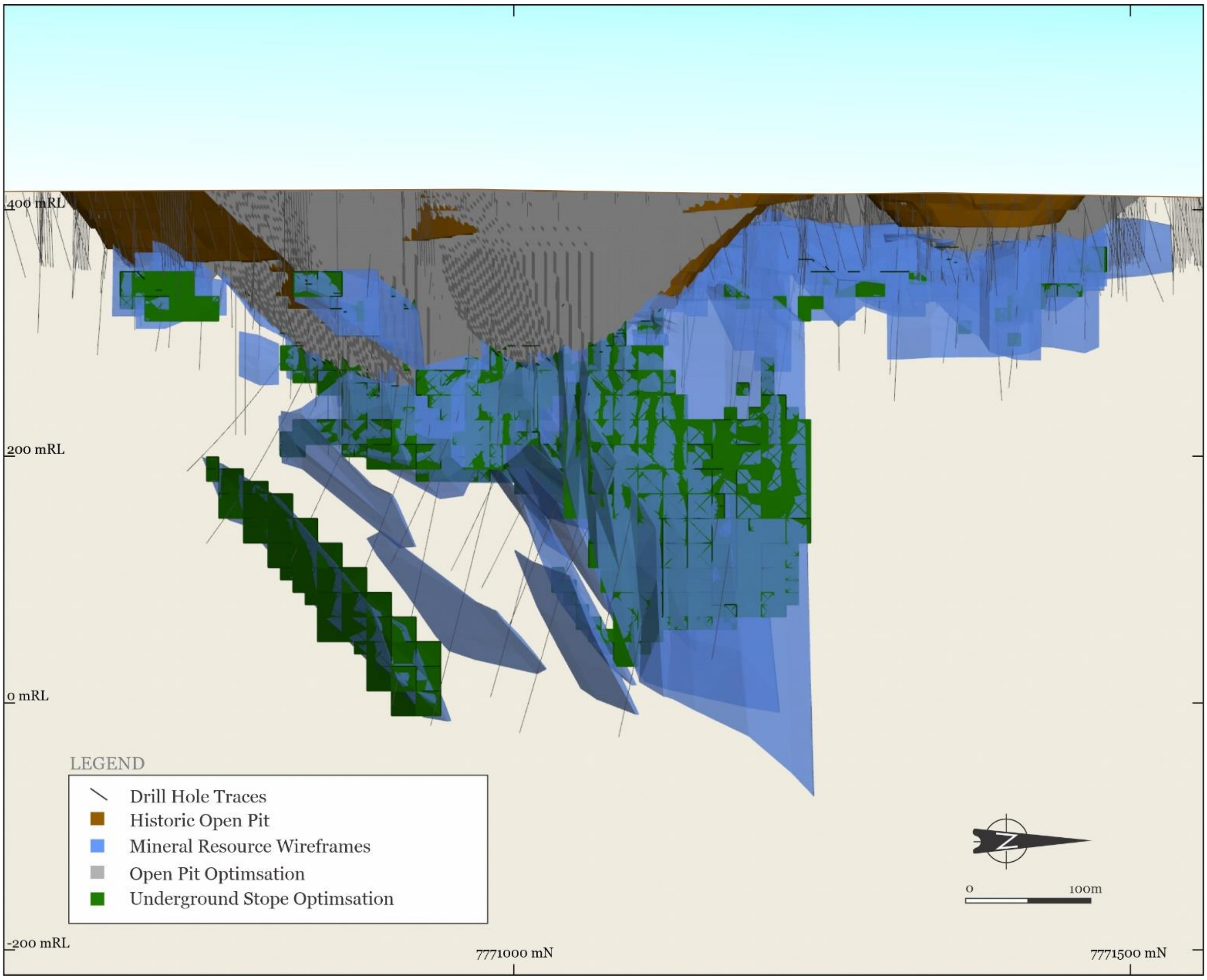

Figure 7 – Jims Gold Mine

Geology and Geological Interpretation

The Jims and Camel Bore deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that are part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts.

Drilling Information and Sampling

Sampling was completed using RC and diamond core drilling. RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core drilling was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or similar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.

RC drillholes were sampled using either a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg.

The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample.

The CTP collected samples at 1m intervals at the rig, representing the cutting's coarse fraction. For CTP drillholes, all samples were taken at 1-metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site.

RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter.

RC drill holes drilled by Tanami Gold between 2010 to 2011 samples were collected on a one metre basis through a 75:25% riffle splitter and placed into pre-numbered sample bags.

Northern Star Stage-1 RC drilling saw all bulk material collected on a 1m basis directly from cyclone in pre labelled green plastic mining bags.

Northern Star Stage-2 RC drilling saw single metre (1m) samples collected from a trailer mounted static cone splitter. Approximately 12.5% of each meter sample was collected in a pre-labelled calico bag with the depth while the remaining 87.5% was collected in a green mining bag and retained.

All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.

Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference.

The number of drill holes shown by type for the MLS168 mineral resources are in the following tables.

Table 25 - Summary of Camel Bore area Drill hole database and holes used in modelling

In Camel Bore Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	23	1297		
DW	372	12473		
RB	189	9586.5		
RC	141	8951	62	445
SL_RC	16	930	8	48
WB	9	1684		
Grand Total	**750**	**34,922**	**70**	**493**

Table 26 - Summary of Jims area Drill hole database and holes used in modelling

In Jims Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	57	4581		
DD	64	11509.41	55	1062.38
DW	1944	95327		
RB	1146	53346		
RC	3212	77993.1	1677	12910.91
RC_DD	7	2710	7	158.66
VC	58	67		
WB	39	5302	12	311.59
Grand Total	**6527**	**250,836**	**1751**	**14443.54**

Sample Preparation and Analysis

Samples collected during mining operations were submitted to the onsite laboratory or the ALS facility in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings.

Samples collected by Northern Star were sent to ALS in Malaga, Perth. Gold concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC samples.

Estimation Methodology and Classification

Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by CTPJV and checked by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software.

Only gold was interpolated into the block model. The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes.

Jims

Three expanding passes were used in the estimation. A first pass of radius 15-30m with a minimum number of samples of 2-6 samples and a second pass of radius 30-100m with a minimum number of 2-6 samples were used for Jims. A third pass of search radius 60-200m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4-14 depending on the number of samples in the domain.

Camel Bore

Three expanding passes were used in the estimation. A first pass of radius 20-25m with a minimum number of samples of 2-6 samples and a second pass of radius 40-50m with a minimum number of 4-6 samples were used for Camel Bore. A third pass of search radius 80-100m was used with a minimum of 2-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4-28 depending on the number of samples in the domain.

Blocks that did not fill after 3 passes were given a 4th pass using nearest neighbour estimation.

To validate the models, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.

The Mineral Resource has been constrained by the wireframed mineralised envelopes, are undiluted by external waste and reported above 0.7 g/t gold, 0.6 g/t gold and 0.6 g/t gold cut-off in Oxide, Transitional & Fresh within an optimised pit shell using A$3,500/oz. Underground resources are reported within an optimised stope below the open pit shell. Underground tonnes and grade include planned dilution in the stope optimisation.
The Mineral Resource estimates are reported in accordance with JORC and were classified as Measured, Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity.

The Measured Mineral Resource is located below Jims Main Open Pit and has already in part been subjected grade control drilling.

The Indicated Mineral Resources were defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. The Inferred Mineral Resources were assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Mining, Metallurgy and Other Modifying Factors

It is assumed the ML(S)168 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in CIL processing recoveries of 76% oxide, 95% transitional, 92% fresh, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 5 within Section 3.

ML(S)180 & EL26925

Mineral Lease (Southern) MLS180 covers an area of 803.6 ha encompassing the Molech gold deposits, Banjo, Beaver, Bonsai, Cheeseman and Orion, while the Pendragon gold deposit is located on the surrounding Exploration Licence EL26925 that covers an area of approximately 190.01 km^2 (60 blocks). These deposits are situated approximately 35km west of the Central Tanami Mill site.

The Mineral Resource on ML(S)168 totals 3,200kt grading 2.4 g/t gold for 250koz, representing open-pit, underground and stockpile material.

Table 27 - Mineral Resource estimates for ML(S)180 & EL26925 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
ML(S)180 & EL26925													
Beaver	0.6-1.5	-	-	-	250	3.3	27	200	3.0	20	460	3.2	46
Banjo	0.6-1.5	-	-	-	160	2.9	15	39	1.9	2	200	2.7	18
Bonsai	0.6-1.5	-	-	-	130	2.1	9	130	2.3	10	260	2.2	19
Orion	0.6-1.5	-	-	-	80	2.6	7	31	3.1	3	110	2.7	10
Pendragon	0.6-1.5	-	-	-	-	-	-	49	2.2	3	49	2.2	3
Cheeseman	0.6-1.5	-	-	-	12	4.4	2	68	3.1	7	81	3.3	8
Stockpiles	0.6	160	0.6	3	-	-	-	-	-	-	160	0.6	3
Total		**160**	**0.6**	**3**	**640**	**2.9**	**59**	**520**	**5.7**	**45**	**1,300**	**2.5**	**10**

Notes:
Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

The Molech area has been explored since the mid 1980's. Numerous companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area.

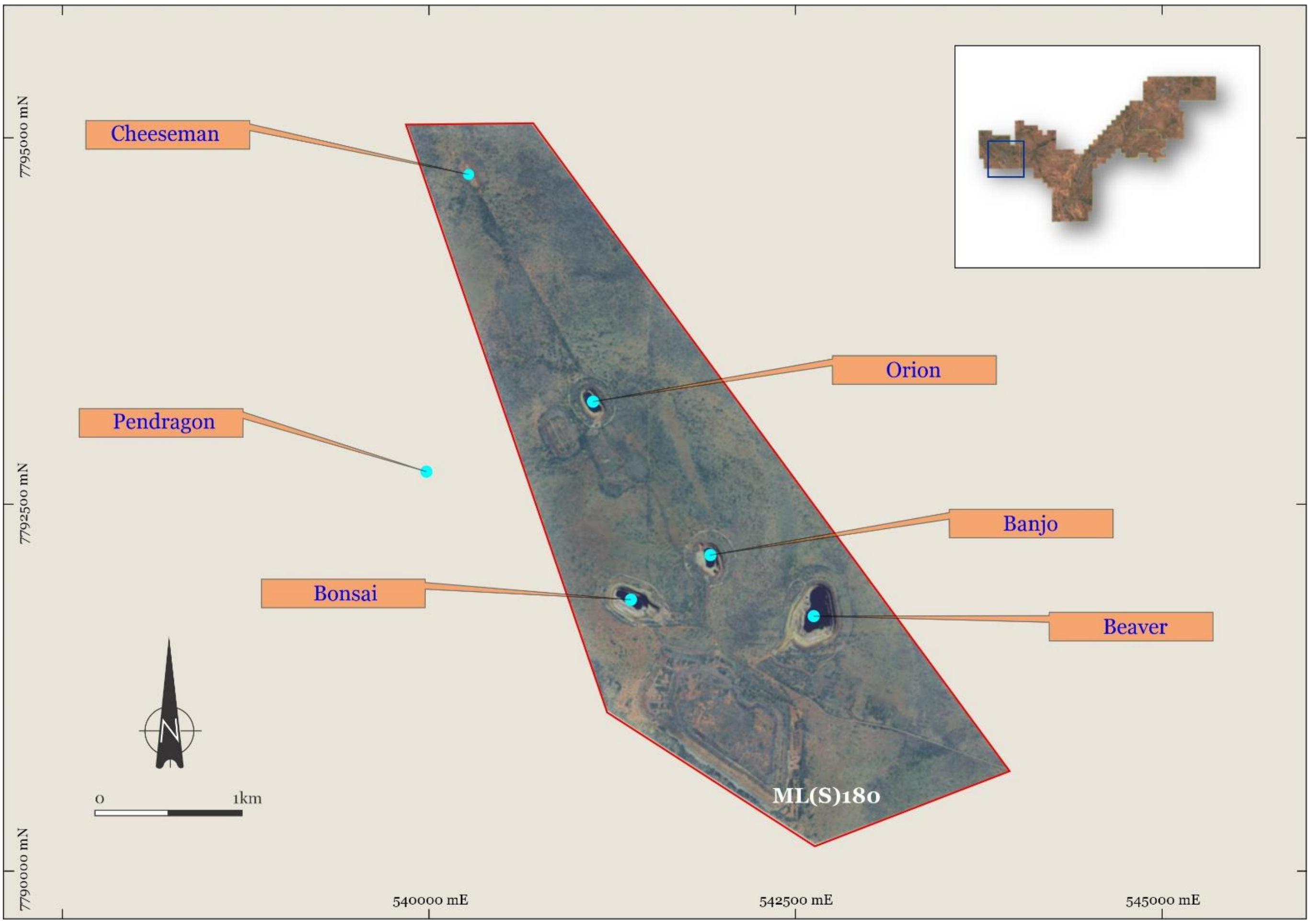

Figure 8 – ML(S)180

Geology and Geological Interpretation

Banjo

The Banjo deposit is hosted by sandstone, mudstone, chert and basalt from the Mt Charles Formation.

Geological interpretations from drill logging, aeromagnetic data and pit mapping suggest that the basalt and sediments are striking about 272° and dipping about -80° South. A 340° trending shear transects the local stratigraphy and has been described as being about 40 metres wide.

Beaver

The Beaver deposit is hosted by intercalated mudstone, siltstone, sandstone, coarse grained volcanoclastic units and undifferentiated basalt from the Mt Charles Formation (Thomson, 2012).

Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 315° and dipping steeply. Mapping from the open pit describes the lithology as thick sequence of mudstone to siltstone that strike 315° and dip 70° South.

Bonsai

The Bonsai deposit is hosted within a 290° trending shear zone that transect basalt and interbedded siltstone and sandstone from the Mt Charles formation.

Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 280° to 335°, dipping steeply and display several fault offsets.

Cheeseman

The Cheeseman deposit is hosted by regional shear that generally trends at about 340°. In the Cheeseman area there is an apparent inflection in this zone where the shear changes from about 330° to 320°. The host rocks consist of basalt and siltstone and sandstone. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Interpreted basalt outside of the shear has an apparent strike of between 1° to 20° and is steeply dipping. Within the shear the basalt has an apparent strike that is parallel to the shear zone.

Orion

The Orion deposits are hosted by a regional shear that generally trends between 325° to 340° and is interpreted to be the same structure that hosts Banjo in the south and Cheeseman in the north. The local geology consists of siltstone, sandstone, and basalt with minor felsic units. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Basalt outside of the shear strikes at about 330° and has apparent steep dip and is 50 to 60 metres thick. Basalt within the shear is discontinuous and is up to 15 to 20 metres thick and steeply dipping.

Pendragon

The Pendragon deposit is hosted within a 300° trending shear zone that transect basalt and interbedded siltstone and sandstone from the Mt Charles formation. This appears to be a similar structure to the shear that hosts the Bonsai deposit.

Drilling Information and Sampling

Sampling was completed using RC and diamond core drilling.

RC Drilling was completed using a 5.25” face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes, the device is unknown.

For RC holes drilled in the 1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter. Historically, where wet samples were encountered the entire sample was collected into a 40-litre plastic bucket before being tipped into discrete piles whereupon scoop samples through the spoil pile were taken.

For drillholes completed by Tanami Gold, all samples were taken at 1 metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site.

All diamond drill holes drilled from 1990s to 2011 were photographed and half core assayed in 1 metre intervals with the remainder retained for future reference. Core is stored in racks or on pallets at the core yard located at the old exploration camp, approximately 5km to the south of the Central Tanami Mill Site.

Table 28 - Summary of Molech area Drill hole database and holes used in modelling for Banjo and Beaver

In Molech Database			In Banjo Resource Model		In Beaver Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**	**No. Holes**	**Intersection Metres**
AC	158	13753				
DD	17	2639.37	2	17	5	43.35
DW	3113	118578.9				
RB	1112	56955				
RC	1947	124001.2	136	1371	349	2636.5
SL_RC	12	320				
WB	24	1764				
Grand Total	**6383**	**318,011**	**138**	**1388**	**354**	**2679.85**

Table 29 - Summary of Molech area holes used in modelling for Bonsai, Cheeseman and Orion

Hole Type	In Bonsai Resource Model		In Cheeseman Resource Model		In Orion Resource Model	
	No. Holes	**Intersection Metres**	**No. Holes**	**Intersection Metres**	**No. Holes**	**Intersection Metres**
AC						
DD	4	13.6				
DW						
RB						
RC	199	1516.9	79	572	137	909
SL_RC						
WB						
Grand Total	**203**	**1530.5**	**79**	**572**	**137**	**909**

Table 30 - Summary of EL26925 area Drill hole database and holes used in modelling

In EL26925 Database			In Pendragon Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	152	13084		
DW	17	2980		
RB	3074	73834		
RC	122	9116	11	85
WB	2	168		
Grand Total	**3367**	**99,182**	**11**	**85**

Sample Preparation and Analysis

During mining operations (1990s to 2001) under Otter, drill samples were analysed offsite at ALS Alice Springs, however with the availability of the onsite laboratory, the database does include some onsite analysis. There was no fixed procedure for selecting onsite or offsite analysis, rather the choice was governed by onsite laboratory availability. Analysis (both onsite and offsite) was by AAS with selective Fire Assay checks. It should be noted that all onsite analysis was performed with a 20ml aliquot whereas ALS use a 50ml aliquot for all AAS readings.

The onsite procedure incorporated the inclusion of a check sample, quartz wash and a standard sample per batch of 30 samples. On a monthly basis one pulp per shift (i.e. two per day) was selected and analysed offsite by AAS and Fire Assay for repeatability. Additional check samples were selected and assayed offsite as required by the geological staff.

Tanami Gold (2010 – 2012) sent samples to the Genalysis Laboratory in Alice Springs for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). No data has been located for the QAQC samples submitted during the RC drilling campaigns completed by Tanami Gold.

Estimation Methodology and Classification

Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CVs, and summary statistics) using Supervisor software. Top cuts were done on a lode basis and prior to estimation.

Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Search ellipses and the minimum and maximum number of samples were lode dependent and varied considerably. A first pass search radius of 25 to 50 metres with a minimum number of samples of 2-6 samples and a second pass radius of 50 to 100 metres with a minimum number of 2-6 samples were used. A third pass search radius of 100-200m was used with 2-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 3-26 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass using nearest neighbour estimation.

To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the lodes. This analysis was completed for northings and elevations across each deposit. Validation plots showed good correlation between the composite grades and the block model grades.

The Mineral Resource estimate was constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.6g/t gold cut-off grade for open pit material within a $A3500 pit shell. The underground resource is reported within a $A3500 optimised stope and is diluted by waste.

The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC). The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. No Measured Resource was categorised due to no actual bulk density data, no QAQC data, open pits finishing early due to geotechnical issues, water ingress and loss of the detailed mining history. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25 by 25 metre, where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined. Validation of the block model shows good correlation of the input data to the estimated grades.

Mining, Metallurgy and Other Modifying Factors

It is assumed the ML(S)180 & EL26925 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill.

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Differences in mining cut-off grades reflect variations in processing recoveries of 88% oxide, 88% transitional, 90% fresh, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 6 within Section 3.

EL28282

Exploration Licence EL28282 covers an area of approximately 101.07 km^2 (35 blocks) encompassing the Crusade gold deposit. This deposit is situated approximately 100km northeast of the Central Tanami Mill site.

The Mineral Resource on EL28282 totals 1,600kt grading 2.2 g/t gold for 110koz, representing open-pit and underground material.

Table 31 - Mineral Resource estimates for EL28282 as of 30 September 2025.

UG+OP Deposit	COG	Measured			Indicated			Inferred			Total		
		Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)	Tonnes (000's)	Grade (g/t)	Ounces (000's)
EL28282													
Crusade	0.7-1.8	-	-	-	1,500	2,2	110	80	1.5	4	1,600	2.2	110
Total		**-**	**-**	**-**	**1,500**	**2.2**	**110**	**80**	**1.5**	**4**	**1,600**	**2.2**	**110**

Notes:
Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The quantities contained in the above table have been rounded to two significant figures to reflect the relative uncertainty of the estimate. Rounding may cause values in the table to appear to have computational errors.

Mineral Resources are reported on a dry in-situ basis.

The Crusade area has been explored since the mid 1990's. Several companies, including Newmont (Asia Pacific) and Tanami Gold NL have been active in the area. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.

Geology and Geological Interpretation

The Crusade deposit is a Paleoproterozoic, mafic-hosted vein-mineralized deposit that is part of the Granites-Tanami Inlier. Mineralisation occurs within quartz veins which are parallel to the contact between the Nany Goat Volcanics and the Killi Killi Formation along a regional fault structure. Specifically, the deposit lies on the northerly striking and westerly dipping contact between biotite dacite and mafic volcanics. The contact dips between 60 to 70 degrees west and strikes at about 020 degrees.

Primary mineralisation being associated with hydrothermal veining and vein brecciation that are dominated by quartz enclosing lesser amounts of pyrite, illite/sericite and tourmaline. Accessory ore minerals associated with higher gold values include chalcopyrite, galena and sphalerite. The mineralisation appears to be thickest highest grade at the intersection of the regional fault and the dacite / basalt contact.

Drilling Information and Sampling

Sampling was completed using RC and diamond core drilling.

RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core drilling was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or similar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.

Table 32 - Summary of Crusade area Drill hole database and holes used in modelling

In Crusade Database			In Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
DD	20	3,466.70	19	429.14
RB	294	7,352		
RC	98	10,121	38	497.61
Total	**412**	**20,940**	**57**	**926.75**

Sample Preparation and Analysis

One metre RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole. For RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter

Sampling of diamond core drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.

Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate.

Northern Star drilling samples were prepared at ALS Perth, commencing with sorting, checking, and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg, a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets.

Samples collected during the 1990s were analysed by AAS with selective FA checks with a 20ml aliquot. It is unknown where the samples were analysed.

Samples collected by Northern Star were sent to ALS in Malaga, Perth. Gold concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC Chip sample collection.

Estimation Methodology and Classification

Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations.

Three dimensional mineralised wireframes (interpreted by CTPJV and checked by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates.

The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CVs, and summary statistics) using Supervisor software.

MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Crusade deposit.

Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation. A first pass of radius 40-60m with a minimum number of samples of 4-6 samples and a second pass of radius 80-120m with a minimum number of 4-6 samples were used for Crusade. A third pass of search radius 160-240m was used with 3-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8-24 depending on the number of samples in the domain. Blocks that did not fill after 3 passes were given a 4th pass.

To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades. The Open Pit Mineral Resource Estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.7g/t gold cut-off grade in Oxide and Transition and 0.8 g/t gold cut-off in Fresh for open pit material within a $A3,500 pit shell. The underground Mineral resource Estimate reports all material with a A$3,500 stope optimisation including planned dilution.

The Mineral Resource estimate is reported here in compliance with JORC. The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC drilling of 40m by 40m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Mining, Metallurgy and Other Modifying Factors

MineScope and AFX Commodities completed a Scoping Study evaluating the potential to treat semi-refractory, fresh material. The study utilised old historical core from the main deposits Hurricane, Carbine, and Legs, assuming the existing processing plant is upgraded to a 1.5 Mtpa design capacity.

Based on the resulting metallurgical recoveries, SWOT analysis, capital/operating benchmarks, and MineScope's database a concentrate production for export was considered for incorporation into the MRE update.

The study concluded an additional operating cost of A$3.95/t of plant feed and capital cost of A$34M to produce concentrate. The post mine-gate, concentrate costs were estimated at a net smelter return of 85.1% including:

- Concentrate Transport of US$370/dmt conc
- Concentrate Treatment & Refining US$148/dmt conc
- Payable Au Factor of 93.0%

It is assumed that the ML33760 deposits will be mined by both open pit and underground methods with all material to be processed through the refurbished existing Central Tanami Project (CTP) free milling CIL mill and additional circuit to produce a concentrate.

Differences in mining cut-off grades reflect variations in processing recoveries, haulage distances, mining conditions, and open pit and underground mining techniques conceptually applied to the individual deposits detailed in the JORC Table1 Appendix 7 within Section 3.

- CIL processing recovery 90% oxide, 86% transitional, 10.1% of total in fresh tailings
- Flotation processing recovery 85.1% of total gold in concentrate
- Processing cost contingency 10%

To satisfy the requirement for Reasonable Prospects for Eventual Economic Extraction (RPEEE), the open pit Mineral Resource estimates are constrained by optimised pit shells developed using reasonable operating cost assumptions and a long-term gold price of A$3,500 per ounce.

Underground Mineral Resources are reported within volumes generated through a Mineable Shape Optimiser (MSO) process and include planned mine dilution consistent with the defined stope geometry including practical minimum width. The underground Mineral Resource is reported outside the open pit Mineral Resource optimisation shells.

Information on Tanami's projects can be found on the Company's website at https://www.tanami.com.au

This announcement has been authorised by the Board of Directors of Tanami Gold NL for release on 7 November 2025.

Arthur Dew
Chairman
Tanami Gold NL

Competent Persons Statements

The information in this release that relates to the Mineral Resource estimates on ML33760, EL26926, ML(S)167, ML(S)168, ML(S)180 & EL26925, EL28282 and ML22934 is based on information compiled by Mr. Graeme Thompson, who is a Member of the Australasian Institute of Mining and Metallurgy, and is an employee of MoJoe Mining Pty Ltd. Mr Graeme Thompson has sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activity, which he has undertaken to qualify as a Competent Person, as defined in the 2012 Edition of the Australasian Code for the Reporting of Mineral Resources and Ore Reserves.

Mr Graeme Thompson has provided written consent approving the inclusion of the Mineral Resource Estimates in the report in the form and context in which they appear.

The information in this report that relates to Exploration for the Mineral Resource estimates on ML33760, EL26926, ML(S)167, ML(S)168, ML(S)180 & EL26925, EL28282 and ML22934 fairly represents information and supporting documentation compiled by Mr. Neale Edwards BSc (Hons), a Fellow of the Australian Institute of Geoscientists, who is a Director of the Company and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the 2012 Edition of the Australasian Code of Reporting for Exploration Results, Mineral Resources and Ore Reserves. Mr. Neale Edwards has provided written consent approving the inclusion of the Exploration Results in the report in the form and context in which they appear.

Appendix 1 - JORC Table 1

ML22934

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DDH) core drilling. Some drill holes were pre-collared using RC drilling methods and completed with DDH tails, while some were drilled diamond core or reverse circulation from the surface.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	Diamond drilling used a combination of HQ and NQ2-sized core. HQ core was drilled until competent ground was intersected, then NQ2 core was drilled. Drill core was oriented, aligned, and half-cut using metre intervals and geologically determined intervals (max 1.2 metres and min 0.3 metres), with geologically determined intervals taking precedence. RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	One metre RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole. Sampling of DDH drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Reflex ACT, EZY MARK, Boart Longyear TruCore, or Axis Champ Ori equipment. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample. RC recovery in the completed campaigns were considered consistent.

Criteria	JORC Code Explanation	Commentary
		DDH core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists.
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by the geologists. The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks. This was checked and compared to the core measurements by the geological team. Any issues were communicated back to the drilling contractor, and necessary adjustments were made.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged by geologists at the drill rig to a high level of detail to support resource estimation, mining studies and metallurgical studies. RC logging is undertaken on a metre by metre basis at the time of drilling. Geologists log DDH core to industry standards. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples are logged for lithology, alteration, mineralisation. Logging is a mix of qualitative and quantitative observations. Visual estimates are made of sulphide, quartz, and alteration as percentages. RC samples are not photographed. All DDH logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and diamond core hole was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference.
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	Depending on the drilling campaign, RC samples were sampled using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. Primary analysis on some RC drilling was determined using 4m speared composite samples at the geologist's discretion. Composite samples with a grade above 0.5 g/t gold had single metre bulk samples riffle split (using a 3-tier riffle splitter) and reanalysed. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate. Normandy completed sample preparation in Alice Springs

Criteria	JORC Code Explanation	Commentary
		All Newmont samples were sent to ALS in Alice Springs for 50g fire assay (method Au-AA26). Sample preparation included jaw crushing all the interval then pulverisation by an LM5. Barren quartz flushes were inserted between each sample to minimise sample cross-contamination. In 2012 samples were sent to Intertek Genalysis (Genalysis) with preparation completed in Alice Springs and analysis done in Townsville. Samples are dried at approximately 120°C, crushed and rotary split (where required), and fine pulverised. Northern Star drilling samples were prepared at ALS Perth, commencing with sorting, checking, and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg, a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	Grind checks are performed at both the crushing stage (3mm) and pulverising stage (75µm), requiring 90% of the material to pass through the relevant size.
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate and to industry standard. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 20 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology and assay value ranges for gold.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Gold concentration was determined by several methods from several campaigns, including: Samples from Normandy were sent to Analabs in Adelaide utilising several assay techniques for gold including P603 (Acid Digest, Carbon Rod Finish), P625 (Acid Digest, AAS Finish), P630 (30g Fire Assay, AAS Finish), P650 (50g Fire Assay, AAS Finish). Normandy procedures dictated that aqua regia was to be utilised for all samples unless visible gold was observed during logging. If the gold assay returned was greater than 2 ppm, the sample was resubmitted for a fire assay; if it was greater than 7-8 ppm, then it was re-submitted for a screen fire assay. If visible gold was observed during logging, screen fire assay was the preferred technique. Samples by Tanami Gold in 2011 were sent to SGS in Perth where gold grades were determined by 50 g Fire Assay with AAS finish (Ore grade analysis FAA505).50 g Fire Assay with AAS finish (Ore grade analysis FAA505) fire assay using the lead collection method with a 50g sample charge weight. MP-AES instrument finish was used to measure gold levels. The methodology used measures total gold. In 2012 samples were sent to Intertek Genalysis with preparation completed in Alice Springs and analysis done in Townsville. Analysis for gold was completed using a 50-gram lead collection fire assay with aqua regia digestion of the prill and flame AAS determination of the gold to 0.005 ppm (FA50/AA). Gold concentration was determined for Northern Star samples sent to ALS in Perth by fire assay using the lead collection method with a 50g sample charge weight. MP-AES instrument finish was used to measure gold levels. The methodology used measures total gold.

Criteria	JORC Code Explanation	Commentary
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	Historical QAQC data from the Normandy and Newmont drilling has not been located but reports reference that QAQC was completed. It is likely that the drilling was of good quality as the area was mined by open pit methods. QAQC programs were completed by Tanami Gold and Northern Star as lined out below. QAQC protocols include the use of commercially prepared certified reference materials ("CRM") that are inserted at a rate of 1 in 20 samples. The CRM is not identifiable to the laboratory and is assessed on import to the database and reported monthly, quarterly, and annually. Values outside of 3 standard deviations were re-assayed with a new CRM. Failed standards are followed up by re-assaying a second 50g pulp sub-sample of all samples in the batch above 0.1 ppm gold by the same method at the primary laboratory. Laboratory QAQC protocols include repeat analysis of pulp samples at a rate of 1 in 20 samples. Screen tests (percentage of pulverised sample passing the 75µm mesh) are undertaken at a rate of 1 in 40 samples. The laboratory reports its QAQC data regularly. The laboratory's standards are routinely loaded into the database. The accuracy component (CRMs) and the precision component (duplicates and repeats) of the QAQC protocols are thought to provide an acceptable level of accuracy and precision. Blanks were routinely inserted into the sample sequence at a rate of 1 per 25 samples and again specifically after potential or existing high-grade mineralisation to test for contamination. Failures of blanks above 0.2g/t were followed up, and re-assayed. New pulps were prepared if failures continued.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	All significant intersections were verified by Geologists on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled for this data set.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data is either entered directly or imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Assay files are received in .csv format and loaded directly into the SQL acQuire database by geologists or database administrators. Hardcopy and electronic copies of the data is stored for future reference. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates. A systematic procedure utilising several re-assays and/or check assays are employed to determine if/when the first (primary) gold assay is changed for the final assay.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource*	Planned drillholes were sited either with a handheld global positioning system (GPS) or a differential global positioning system (DGPS), and the initial drillhole pickup is usually with a handheld GPS, as well, with accuracy between ± 0.3 to 1m. After

Criteria	JORC Code Explanation	Commentary
	estimation.	program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm. During drilling, single-shot surveys were taken every 30m to ensure the hole remains close to the design. Down-hole surveys were performed using Reflex ACT, EZY MARK, Boart Longyear TruCore, or Axis Champ Ori equipment., recording the down-hole dip and magnetic azimuth. These results were then uploaded into the database.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition, or a DGPS elevation with an accuracy of ± 10mm is used.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	Drillhole spacing at Groundrush varies, although minimum 25m spacing was targeted during the design and drilling phases. Drillhole spacing at Ripcord varies; the indicated mineral resource was defined within areas of RC drilling of nominally 20m by 25m
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution from the reported campaigns is sufficient to establish geological and/or grade continuity. Further drilling will be required to ensure that it is appropriate for resource estimation and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends.
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	The chain of custody of samples was managed by geologists and geotechnicians. Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email or uploaded to an FTP site. Sample pulp splits are stored for a time at the laboratory. Retained pulp packets are returned to the Central Tanami Mine for storage.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Geologists have undertaken internal reviews of applied sampling techniques and data. The completed reviews raised no issues.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Groundrush and Ripcord Gold Deposits are located in the Tanami Region in the Northern Territory on Mining Licence ML22934, approximately 45km northeast of the Central Tanami Mill site. ML22934 covers an area of 3,950 ha and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108km^2 tenement area in the Tanami Region held by the CTPJV is registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. Mineral Leases have a 25-year life and are renewable for 25 years. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	ML22934 is granted and in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Groundrush area, which includes the Ripcord Prospect, has been explored since the mid 1980's. Numerous companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area. Drilling reported with this release is contiguous with the Groundrush open-cut mine and Ripcord area. Previous drilling at this project adds gold grade and geological context to the subsequent Northern Star Resources interpretation of the area as tested by the drill holes covered by this report. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.
Geology	• *Deposit type, geological setting and style of mineralisation.*	Rocks of the Killi Killi Formation host the Groundrush and Ripcord deposits exposed in a narrow N- to NNW-trending corridor flanked by lobes of the younger Frankenia Dome granite. Groundrush and Ripcord thus lies within rocks of a similar age to the host rocks of The Granites and Dead Bullock Soak gold deposits 100km to the south, but older than the Mount Charles Formation, which hosts the Tanami gold deposits 50km southwest. Less than 1 km to the north of Groundrush, the Killi Killi beds are truncated by a fault-bounded outlier of younger sediment of the Mount Charles Formation. At Groundrush, a package of relatively undeformed, steeply west-dipping, sedimentary rocks is intruded by two tabular dolerite units broadly conformable with bedding. The main dolerite body exposed in the open pit consists of a coarser-grained leucocratic quartz dolerite. Groundrush gold mineralisation is mainly hosted in quartz-sulphide veins and stockwork zones within steeply dipping shear zones in the quartz dolerite unit and flat dipping quartz-sulphide brittle fracture veins. The Ripcord deposit is a Palaeoproterozoic, dolerite, and sediment-hosted vein-mineralized deposit that is part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with larger regional-scale structures that crosscut a regional scale southeast, shallowly plunging anticline. Mineralisation is predominantly hosted in

Criteria	JORC Code Explanation	Commentary
		dolerite and sediment, in either quartz vein or shear hosted, respectively.
Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. Exploration rresults are reported as weighted averages using a nominal 0.5 g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied.
	• *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. Any high-grade zones above 15g/t gold within a reported intercept are also reported as included intervals.
	• The assumptions used for any reporting of metal equivalent values should be clearly stated.	No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	• These relationships are particularly important in the reporting of Exploration Results.	The reported drillholes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends.
	• If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.	The orientation of the Groundrush mineralised system is generally well understood. The geometry of the mineralisation to drillhole intercepts generally at a high angle, often nearing perpendicular. There is enough historic exploration and production data at Groundrush to infer geological continuity in mineralisation reported. Gold mineralisation at Ripcord is dipping -60° to -70° west and strikes between 320° to 350°. Drilling was carried perpendicular to these trends and often at a high angle.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this*	When exploration results were previously disclosed, only downhole lengths were reported. True widths are not known.

Criteria	JORC Code Explanation	Commentary
	effect (e.g. 'down hole length, true width not known').	
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Appropriate diagrams accompany this release.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Planned drillholes are sited with a handheld global positioning system (GPS), and the initial drillhole pickup is usually with a handheld GPS, as well; with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths. All intercepts for all holes have been reported regardless of grade.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	Drilling is planned for CY25 and CY26 to infill and expand the current resource envelopes.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records.

Criteria	JORC Code Explanation	Commentary
		There are some minor overlap errors in the RC and diamond drill holes where 4 metre samples overlapped later 1 metre samples but the occurrence was not significant
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	The competent person Graeme Thompson, Principal Resource Geologist of MoJoe Mining have made a number of site visits
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to high as there are current no exposures and it is based upon RC exploration and grade control drill holes and diamond drill holes. The Groundrush gold mineralisation has been previously mined from 2000 to 2005 and the current interpretation is based upon detailed RC grade control data. The mineralisation was projected below the historical open pit to areas of exploration drilling. At Ripcord the interpretation is based upon RC and diamond drilling with no exposures at surface. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. The Groundrush deposit is hosted within the Killi Killi Formation of the Tanami Group (1838+/-6Ma) (Huston, 2006 in Hillyard, 2013), a turbiditic siltstone and sandstone (arkose and greywacke) unit up to 4 Km thick that has been intruded by a fractionated dolerite sill. This unit conformably overlies the Dead Bullock Formation, composed of graphitic units with minor chert and iron rich horizons. Dolerite sills up to 200+m thick intrude the Tanami Group. Gold mineralisation is primarily hosted within a fractionated dolerite sill, with minor mineralisation extending into turbiditic sediments. The mineralisation has been described in detail by Stephens (2004), and Majoribanks (2011), Mason (2011) and Pascoe & Eggers (2012). The major conclusion is that the Groundrush mineralisation represents an orogenic, reverse-fault stacked gold lode system. Neilsen (2023) describes the Groundrush deposit as reflecting an early conjugate, flat tensional veining event that was progressively sheared and overprinted by a steep shear vein event. The intersections of S1 and S2 resulting in development of high-grade shoots. Bland & Annison (2016) state that the Groundrush deposit sits in an almost arcuate belt of sediments belonging to the Killi Killi formation, it lies between two major granitoid intrusions: The Coomarie Dome to the Northwest and the Frankenia Dome to the Southeast. Sediments dip steeply to the Southwest and exhibit three dolerite intrusions of which there is one containing the bulk of Groundrush gold mineralisation. Other intrusions at Groundrush include dolerite, tonalite porphyry, andesite and quartz monzodiorite. Overall, the deposit can be referred to as a reverse fault orogenic system; mineralisation is typically hosted in stacked vein sets with a variety of orientations as well as sub-vertical quartz-filled shear zones. Along with the various orientations of veining there also exists a variety of types: shear, extensional and also a shear-extension hybrid style of veining. The host dolerite unit at Ripcord shows similar fractionation textures as per Groundrush, with fractionated quartz dolerite bounded on both sides by transitional quartz dolerite zones (Hillyard, 2013). The dolerite is striking at about 340° and dipping -70o to – 85° west and is bounded either side by other dolerites that are striking 320° to 325° and dipping about -70° southwest. There appears to be structural thickening of the main dolerite sequence in the southern part of the Ripcord deposit. Interpreted

Criteria	JORC Code Explanation	Commentary
		faults by Neilson (2023) cut through this area and are trending at 005° to 007° and dipping steeply to the west. The sedimentary sequences at Ripcord consist of fine-grained sandstone and siltstone. The main sedimentary sequence is located in the hanging wall of the main 320° trending dolerite and varies in thickness between tens of metres to 180 metres thick. The apparent strike and dip of this sequence mirrors the bounding dolerite sequences. Gold mineralisation is hosted by quartz veins in both the sedimentary sequence and dolerite. There is a strong spatial association between the dolerite-sediment contact and gold mineralisation. The southern 200 metres of the Ripcord deposit is mostly hosted by dolerite while the remainder of the mineralisation is hosted by sediments with lesser mineralisation hosted by dolerite. Neilson (2023) describes veining at Ripcord to be dominated by a series of flat-tensional conjugate array developed in response to initial shortening that was superposed by a steep shear array. Flattening of gold mineralisation is seen locally surrounding 007° faulting that also appears to offset the mineralisation in the southern part of the Ripcord deposit. North of this area the majority of the veining is steeper between -60° to -70° west. Approximately 60% of the gold mineralisation is hosted by dolerite while the other 40% is hosted by the sedimentary sequence. The average grade of the gold mineralisation is about 1.91 g/t in dolerite and 1.76 g/t au in the sedimentary sequence Cross cutting faults are known to exist in the prospect area. These could have an effect of the geometry and continuity of the gold mineralisation.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	Gold mineralisation at Groundrush is primarily hosted within a fractionated dolerite sill, with minor mineralisation extending into turbiditic sediments. Previous mining of the open pit concentrated on ore zones that incorporated many of the vein sets within an economic zone. Fogden (2012) completed a geological analysis and modelled vein sets and lodes identified in close spaced grade control drilling and regional data sets. Three main orientations were identified; steep west dipping shear hosted veins, moderate east dipping veins and moderate west dipping veins. 106 lodes were modelled. The main steep lodes make up 85% of the resource and confidence in these lodes is high. There are 14 shallow dipping lodes The main lodes in Groundrush are generally striking around 340° but varied from 307° degrees to 356° and have a total strike length of 1900 metres. They dip about -60° to -70° west but range from -50° to -85° west. These lodes are mostly plunging to the south at about 15° but individual plunges can vary. The strike length of the lodes varies from 50 to 970 metres, and they extend down dip from 50 to a maximum of 350 metres. The true thickness of the lodes varies from 1-2 to 35 metres thick. The geometry of these lodes is as stacked lenses within the Groundrush dolerite. These lenses are still open down plunge. The Groundrush flat lying lodes are only well established in the mined-out areas where they were defined by closed spaced grade control drilling. These lodes are splays from the main lodes and are difficult to interpret from the exploration drilling data. They are largely confined to areas of dolerite and strike between 325° to 360°, dip from -8° to -46° west and plunge southwest between 0° to 40°. These lodes are best developed in mined out areas of the open where supergene effects have played a role in their enrichment. The strike length of these lodes varies from 50 to a maximum of 600 metres with a true thickness in fresh material of 1-2 metres. The down dip extent varies from 15 to 100 metres. The Western Dolerite gold mineralisation is striking between 314° to 338°, plunges generally to the northwest and dips between -50°

Criteria	JORC Code Explanation	Commentary
		to -75° west. The strike length varies between 25 to 310 metres with true thickness varying from 1-2 metres to 8 metres. The down plunge extent varies from 25 to 130 metres. The strike of the mineralised zone at Ripcord is 1160 metres and the known down dip extent from drill data is about 200 metres. The width of the zone of primary mineralisation is of the order of 40 to 120 metres but individual veins are much thinner. Individual veins vary in strike length from 25 to 385 metres and are 1 – 2 metres to 30 metres thick and have down dip extents of 20 to 115 metres. The strike of the veins varies from 320° to 350°, plunges from 0 to -25° north and dip from being flat to -80° west. A total of 56 quartz veins and 5 supergene / alluvium lodes have been modelled.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by CTPJV and checked by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Groundrush deposit. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The Groundrush block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m for the open pit model and 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 0.625m by 0.625m for the underground model. The Ripcord block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization An orientated 'ellipsoid' search was used to select data for interpolation. The ellipsoid was oriented to the average strike and dip of the mineralised zones. The Groundrush block model was filled using a first pass of radius 20-40m with a minimum number of samples of 2-6 samples and a second pass of radius 40-80m with a minimum number of 2-6 samples were used for Groundrush. A third pass of search radius 80-160m was used with 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8-14 depending on the number of samples in the domain. The Ripcord block model was filled using a first pass of radius 20-40m with a minimum number of samples of 2-6 samples and a second pass of radius 40-80m with a minimum number of 2-6 samples were used for Ripcord. A third pass of search radius 80-160m was used with 2-4 samples to ensure all blocks within the

Criteria	JORC Code Explanation	Commentary
		mineralised lodes were estimated. The maximum number of samples was set at 10 to 12. Blocks that did not fill after 3 passes were given a fourth pass using nearest neighbour. Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource estimate at Groundrush has been constrained by the wireframed mineralised envelopes. The underground resource is reported by a AU$3500 stope optimisation that includes dilution. The Ripcord Open Pit Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.6g/t gold cut-off grade with a $3500 open pit optimised shell. The Ripcord underground mineral resource estimates is constrained by AU$3,500 gold price stope optimisation and includes all material within the optimised stope. These figures were based upon financial studies by MoJoe Mining Pty Ltd
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	It is assumed the Groundrush deposit will be mined by underground methods when a new mining operation can be established. The following mining factors and costs were used for the Deswik optimisation of the underground resource: Deswik Underground Stope Optimiser Assumptions • Minimum Mining Width 3.75 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 25 metres • Optimise grade or metal: Grade • Stope Strike ±20 degrees • Stope Dip – Minimum 40 degrees • Sub Stope Shapes 2 U / 2 V • Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • CIL Processing recovery 94% • UG Stoping Costs $88/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $35.46/tonne ore

Criteria	JORC Code Explanation	Commentary
		• ROM to Mill transport $4.12/tonne ore • Admin (G&A) $4.50/tonne ore • Contingency Factor (10%) $15.81/tonne ore • Au Royalty 5.5% • Au Price AU$3500/troy ounce It is assumed the Ripcord deposit will be mined by open pit and underground methods when a new mining operation can be established. The following mining factors and costs were used for the Deswik optimisation of the open pit and underground resource: Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10% • Pit Slopes – Oxide 39° • Pit Slopes – Other 45° • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10% • Mine ROM to Mill ROM Haulages $0.10/t per km ore • Mill Opex cost $35.46 per tonne • Admin (G&A) $4.50 per tonne • CIL Processing Recovery 97% oxide, 90% transitional, 89% fresh • Processing cost contingency 10% • Au Price AU$3500 per troy ounce • Au Royalty 5.5% • Discount Rate 8% • Mining Rate 20 Mtpa rock • Ripcord haulage 41.2 km Deswik Underground Stope Optimiser Assumptions • *Minimum Mining Width 2.4 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 20 metres • Optimise grade or metal: grade • Stope Strike ±40 degrees • Stope Dip – Minimum 40 degrees • Sub Stope Shapes 2 U / 2 V • Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • CIL Processing recovery 89% • UG Stoping Costs $89/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $35.46/tonne ore • ROM to Mill transport $4.12/tonne ore • Admin (G&A) $4.50/tonne ore

Criteria	JORC Code Explanation	Commentary
		• Contingency Factor(10%) $12.41/tonne ore • Au Royalty 5.5% • Au Price AU$3500/troy ounce
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	Twelve composite samples were collected for the TGNL Definitive Feasibility Study (DFS) Extraction Test Work that included a total of 18 individual samples from within the model mineralised lodes of the Groundrush deposit. A total of 5 extra samples (including 6 individual samples from within the model mineralised lodes) that were originally used in the Pre-Feasibility Study (PFS) have also been included in the data (Smith, 2013). The samples were derived from ¼ NQ2 diamond core and sent to ALS Laboratories. Tailings Sample test work were collected and tested at the same time. The metallurgical data shows excellent gold recoveries that range from 86.7% to 99.3% with an average of 94.3% Tanami Gold NL submitted 9 composite RC samples for metallurgical testing in 2013 at Ripcord. These samples were ground to 150 microns and tested for recovery of gold. This data was collated by oxidation state (weathering), summarised and the average was assigned to the Ripcord block model. • Oxide mineralisation 97.2% recovery • Transitional mineralisation 90.1% recovery • Fresh mineralisation 89.9% recovery
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	No assumptions have been made regarding environmental factors.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	Bulk density measurements were routinely taken by Tanami in 2011-2012 using the wet and dry emulsion method. Measurements were taken at regular 10 m intervals downhole and are chosen to be representative of the surrounding geology. This data was analysed by Tanami and summarised by mineralisation, lithology and elevation. Bulk density measurements were taken from multiple sections (mineralized and waste) throughout pre-determined new resource definition drilling holes (normally holes ending in 0 and 5). A total of 845 bulk density measurements were taken from 20 drill holes. These results were the validated against previous bulk densities in the 2012 Optiro/Tanami resource model. Measurements were taken using the immersion method and related back to dominant rock code. No oxide and transitional zone were measured in the program. The following values were assigned to the block models for bulk density.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>

Rock Type	Oxidation State	Density
SED	OX	2.53
SED	TRANS	2.62
SED	FRESH	2.74
GOD	OX	2.55
GOD	TRANS	2.9
GOD	FRESH	2.94
GQD	OX	2.4
GQD	TRANS	2.76
GQD	FRESH	2.86
WOD	OX	2.55
WOD	TRANS	2.9
WOD	FRESH	2.99
TOD	OX	2.55
TOD	TRANS	2.9
TOD	FRESH	2.94
BF/WD		2.2

No bulk density measurements were available for the Ripcord Deposit. The bulk density measurements were taken from the Groundrush deposit 3 km to the northwest and the NST engineering team for cover material.

Oxidation State	Rock Type		
	Dolerite	Turbiditic sediment	Alluvium / Colluvium
Oxide	2.4	2.32	2.2
Transitional	2.7	2.58	
Fresh	2.85	2.7	

At this stage of the project, it is assumed that these values will be close to the real values</td></tr>
<tr><td>Classification</td><td>The basis for the classification of the Mineral Resources into varying confidence categories.
Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
Whether the result appropriately reflects the Competent Person's view of the deposit.</td><td>The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC). The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC drilling of 20-25m by 25m, and where the continuity and predictability of the lode positions was good and the quality of the estimation was also good. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Validation of the block model shows good correlation of the input data to the estimated grades.

The result reflects the competent person's view that the classification is Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>The results of any audits or reviews of Mineral Resource estimates.</td><td>Internal reviews have been conducted by Northern Star Resources resource geologists.</td></tr>
<tr><td>Discussion of relative accuracy/ confidence</td><td>Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent</td><td>The Groundrush Mineral Resource Estimate has been reported with a moderate degree of confidence.

The Indicated Mineral Resource is based upon 25 by 25 metre RC drilling of acceptable quality. It is assumed that the mineralisation in this area is continuous between drill sections.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
	Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	The Groundrush deposit has been previously mined by Normandy / Newmont between 2001 and 2005 by open pit. Groundrush has a recorded historical production of 611,000 ounces (4.76Mt @ 4.03 g/t Newmont Production database) via open pit mining. The current model at a low grade cut off of 1 g/t Au produces 4Mt @ 4.9 g/t Au for 639k ounces of gold. Once mining factors are taken in account of about 10% dilution, 2% ore loss and mill recovery of 98-99% the total ounces closely match the production of ounces of gold. It must be noted that there are uncertainties to the exact low grade cut off grade used by Newmont. The Ripcord Mineral Resource Estimate has been reported with a moderate degree of confidence. The Indicated Mineral Resource is based upon 25 by 20 metre RC drilling of acceptable quality. It is assumed that the mineralisation in this area is continuous between drill sections. The project is in area of no previous mining however the Groundrush deposit is located 3 km to the northwest. The Mineral Resource statement relates to global estimates of tonnes and grade.

Appendix 2 - JORC Table 1
ML33760

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) drilling. Sampling of RC chips was completed on RC drillholes. Sampling of diamond holes utilised half cut core.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled. Diamond holes were marked up against the core runs.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	For RC holes drilled by Zapopan NL in the late 1980s and early 1990s samples were taken at 1 metre intervals from the cyclone and placed into labelled plastic bags. Samples were combined into 2 metre composites, but the method is unknown. RC holes drilled by OGM in the mid 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a four-deck riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40 litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples. During mining operations (mid 1990s to 2001) under the Tanami Gold Joint Venture (OGM) drill samples were analysed offsite at ALS Alice Springs however with the availability of the onsite laboratory, the database does include some onsite analysis. There was no fixed procedure for selecting on- or offsite analysis; rather the choice was governed by onsite laboratory availability. Analysis (both on and offsite) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot whereas ALS use a 50ml aliquot for all AAS readings RC holes drilled by TAM were collected through a 75:25% riffle splitter in prenumbered bags. The samples varied from wet to dry. The samples varied from wet to dry. Samples were generally around 3 kg in weight. TAM (2010 - 2012) sent samples to SGS Laboratories in Perth for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). This method had a 0.01 ppm Au detection limit. Diamond holes drilled by TAM, NQ2 size core was collected. Half core samples were taken down the length of the hole. For RC holes drilled by Northern Star in 2024 samples were taken at 1 metre intervals using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle

Criteria	JORC Code Explanation	Commentary
		splitter down to approximately 2kg. Northern Star (2024) sent samples to ALS Adelaide, commencing with sorting, checking, and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg, a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets. Gold concentration was determined for Northern Star samples sent to ALS in Adelaide by fire assay using the lead collection method with a 50g sample charge weight. MP-AES instrument finish was used to measure gold levels. The methodology used measures total gold.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.75" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or simiilar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample. DD core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists.
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by geologists. The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks. This was checked and compared to the core measurements by the geological team. Any issues were communicated back to the drilling contractor, and necessary adjustments were made.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged on 1 metre intervals with data subsequently merged into an access database. A representative portion of each RC metre was retained in chip trays and stored on site.

Criteria	JORC Code Explanation	Commentary
		Geologists log DD core. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples are logged for lithology, alteration, mineralisation. Logging is a mix of qualitative and quantitative observations. Visual estimates are made of sulphide, quartz, and alteration as percentages. RC samples are not photographed. All DDH logging was quantitative where possible and qualitative elsewhere.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and DD was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference.
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample. RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter. For RC holes drilled by TAM in 2010 to 2012 samples were taken at 1 metre intervals from the cyclone and collected from a 75:25% riffle splitter in prenumbered sample bags. The samples varied from wet to dry. Samples were generally around 3 kg in weight For CTP holes 1m RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole.
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	During mining operations drill samples were prepped either at onsite or at ALS in Alice Springs to industry standards. The Otter Gold Mines data does include some onsite analysis at the mine laboratory. Drill samples collected by TAM were submitted to SGS Laboratories in Perth and assayed using a 50g fire assay charge for gold with an atomic spectrometer finish. This method had a 0.01ppm detection limit. Sample weights were generally around 3kg in size. For CTP RC samples are dried at 100°C to constant mass, all samples below approximately 3kg are pulverised in LM5's to nominally 85% passing a 75µm screen. Samples generated above

Criteria	JORC Code Explanation	Commentary
		4kg are crushed to <6mm and cone split to nominal mass before pulverisation. For RC samples, no formal heterogeneity study has been carried out or monographed. An informal analysis suggests that the sampling protocol currently in use is appropriate to the mineralisation encountered and should provide representative results.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	Grind checks are performed at both the crushing stage (3mm) and pulverising stage (75µm), requiring 90% of the material to pass through the relevant size.
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 20 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology, and assay value ranges for gold.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Gold concentration was determined in various ways depending on the drilling program. From the late 1980s to about March 1994, most of the samples collected by Zapopan NL were assayed for gold by fire assay with a 0.01 ppm detection limit at the onsite laboratory. Samples collected during mining operations (the mid-1990s to 2001) under the Tanami Gold Joint Venture were submitted to the onsite laboratory or ALS in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings. TAM (2010-2012) sent samples to SGS Laboratories in Perth for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). This method had a 0.01 ppm Au detection limit. Samples collected by CTP were sent to ALS in Malaga, Perth. Gold (Au) concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC Chip sample collection.
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	1994 to 2001 Analysis (both on and offsite) was by AAS with selective FA checks. The onsite procedure incorporates the inclusion of a check sample, quartz wash and a standard sample per batch of 30 samples. On a monthly basis one pulp per shift (ie two per day) was selected and analysed offsite by AAS and Fire assay for repeatability. Additional check samples were selected

Criteria	JORC Code Explanation	Commentary
		and assayed offsite as required by the geological staff. This data has not been located. For TAM drilling in 2010 to 2012 certified reference material (CRM) were inserted every 30 samples and blanks every 50 samples. Any CRM that fell outside of 2 standard deviations of the expected value was followed up to determine the cause. The CTP QAQC protocols used include the following for all drill samples: • Field QAQC protocols used for all drill samples include commercially prepared certified reference materials (CRM) inserted at an incidence of 1 in 20 samples. The CRM used is not identifiable to the laboratory with QAQC data is assessed on import to the database and reported monthly, quarterly and yearly. • NSR RC Resource definition drilling routinely inserts field blanks and monitor their performance. • Laboratory QAQC protocols used for all drill samples include repeat analysis of pulp samples occurs at an incidence of 1 in 20 samples and screen tests (percentage of pulverised sample passing a 75µm mesh) are undertaken on 1 in 40 samples. • The laboratories' own standards are loaded into the database and the laboratory reports its own QAQC data monthly. • Blanks were routinely inserted into the sample sequence at a rate of 1 per 25 samples and again specifically after potential or existing high-grade mineralisation to test for contamination. Failures of blanks above 0.2g/t were followed up, and re-assayed. New pulps were prepared if failures continued. • Failed standards are generally followed up by re-assaying a second 30g pulp sample of all samples in the fire above 0.1ppm by the same method at the primary laboratory. • Both the accuracy component (CRM's and third-party checks) and the precision component (duplicates and repeats) of the QAQC protocols are thought to demonstrate acceptable levels of accuracy and precision.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	The majority of the data is historical from the period when Otter Gold Mines ran the operation and when TAM was the sole operator. 6 RC holes were drilled by CTP in 2024. All significant intersections were verified by Geologists on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC. During the CTP JV (2017-present) significant intersections were verified by a Northern Star Senior Geologist on-site during the drill-

Criteria	JORC Code Explanation	Commentary
		hole validation process and later signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled for this data set.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data was imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	All collar locations were set from surveyed grids or individually surveyed into position and subsequently surveyed by the mine survey department after drilling. This data was transformed using appropriate grid transformations in MGA94 Zone 52.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition with the addition of drill hole collar pick-ups from qualified mine surveyors.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 12 to 25m or closer (with some infill), where the continuity and predictability of the lode positions were good. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider-spaced drilling or insufficient drilling in smaller lodes.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends.
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	The security measures for holes from 1990 to 2001 have not been recorded. For TAM the chain of custody of samples was managed by geologists and geotechnicians.

Criteria	JORC Code Explanation	Commentary
		Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at the on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email. Sample pulp splits are stored for a time at the laboratory. Retained pulp packets are returned to the Central Tanami Mine for storage.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Audits or reviews for holes from 1990 to 2001 have not been located. For TAM results were reviewed by geologists to ensure sampling returned a representative sample. Geologists have undertaken internal reviews of applied sampling techniques and data. The completed reviews raised no issues.

Section 2 Reporting of Exploration Results
(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Bastille, Battery, Assault, South Temby, Dinky and Dice, Hurricane-Repulse, Southern, Thrasher, Tombola, Miracle, Gatling, Bouncer and Bumper Gold Deposits are located in the Tanami Region in the Northern Territory on Mineral Lease ML33760, approximately 1 to 3.3 km northeast of the Central Tanami Mill site. ML33760 covers an area of 1120.34 ha and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108 km2 tenement area in the Tanami Region held by the CTPJV are registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. Mineral Leases have a 25-year life and are renewable for 25 years. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	ML33760 is granted and in good standing.

Criteria	JORC Code Explanation	Commentary
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Bastille, Battery, Assault, South Temby, Dinky and Dice Hurricane-Repulse, Southern, Thrasher Tombola, Miracle, Gatling, Bouncer and Bumper areas have been explored since the early 1990's. Several previous companies, Zapopan NL, Otter Gold Mines, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area. The Hurricane-Repulse area has been explored since the mid-1980's. Several companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area. Drilling reported with this release is contiguous with the Hurricane-Repulse open-cut mine. Previous drilling at this project adds gold grade and geological context to the subsequent Northern Star Resources interpretation of the area as tested by the drill holes covered by this report. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The Bastille, Battery, Assault, South Temby, Dinky and Dice, Southern, Thrasher Tombola, Miracle, Gatling, Bouncer and Bumper deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that is part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts. The Hurricane-Repulse deposit is hosted by mafic volcanic flows (pillowed, vesicular and massive basalt flows), some volcanic flow breccias, sequences of lithic sandstones, siltstones and mudstones, occasional coarse sediments consisting of very proximal volcanic fragments, and more minor to rare siliceous/cherty horizons, and rare graphitic mudstones. Vein stages have been identified from crosscutting relationships in several areas of the mine leases, with gold mineralisation associated with both: • grey quartz ± sericite ± pyrite ± chlorite ± sphalerite ± arsenopyrite ± gold; or • ankerite-quartz ± chalcopyrite ± chlorite ± gold ± sericite ± pyrite ± calcite. Gold occurs in grains up to 15 µm within pyrite in the first vein style and chalcopyrite in the second vein style. The overall strike length of the known gold mineralisation on the Hurricane-Repulse trend is 1,750 metres and has a variable down dip extent of about 180 metres. The true thickness of gold mineralisation varies from less than a metre to 10 metres. The host to the mineralisation in the Hurricane pit is interbedded sandstone and siltstone. In this area, the strike of the mineralisation is about 030°, and the strike length of individual lenses varies between 80 to 120 metres. The down-dip extent of lenses varies from 10 to 80 metres and the true thickness from 0.6 to several metres. The shapes of the mineralisation are irregular

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>and are interpreted to reflect the rheology contrasts between the siltstone and sandstone. The dips of the mineralisation varied from 30° to 75° southeast.

In the northern part of the Hurricane pit, the mineralisation changes strike to about 010° as the mineralisation approaches the boundary between the sediments and basalt. The strike length of the mineralisation increases to 180 metres and there are several cross-cutting structures that vary in strike from 040° to 075° close to the basalt/sediment contact. This pattern continues into the basalt</td></tr>
<tr><td>Drill hole information</td><td>A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:
easting and northing of the drill hole collar
elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar
dip and azimuth of the hole
down hole length and interception depth
hole length</td><td>All drill hole information is historical drilled from 1990 to 2024. 12 RC holes were drilled in 2024 and are listed below.
<table>
<tr><th>BHID</th><th>EAST</th><th>NORTH</th><th>RL</th><th>LENGTH</th><th>AZIMUTH</th><th>DIP</th></tr>
<tr><td>MWRC0001</td><td>573020</td><td>7791032</td><td>432.2</td><td>100</td><td>346.3</td><td>-55.5</td></tr>
<tr><td>MWRC0002</td><td>573043</td><td>7790992</td><td>431.8</td><td>180</td><td>341.1</td><td>-55.5</td></tr>
<tr><td>MWRC0003</td><td>573046</td><td>7790937</td><td>431.8</td><td>234</td><td>345.3</td><td>-55.3</td></tr>
<tr><td>MWRC0004</td><td>572976</td><td>7791024</td><td>433.2</td><td>100</td><td>346.0</td><td>-56.0</td></tr>
<tr><td>MWRC0005</td><td>572982</td><td>7790977</td><td>433.1</td><td>160</td><td>345.0</td><td>-56.8</td></tr>
<tr><td>MWRC0006</td><td>572999</td><td>7790918</td><td>432.8</td><td>228</td><td>346.6</td><td>-56.6</td></tr>
<tr><td>MWRC0007</td><td>572929</td><td>7791013</td><td>433.9</td><td>100</td><td>348.6</td><td>-56.1</td></tr>
<tr><td>MWRC0008</td><td>572938</td><td>7790966</td><td>434.1</td><td>160</td><td>346.7</td><td>-55.9</td></tr>
<tr><td>MWRC0009</td><td>572953</td><td>7790919</td><td>433.7</td><td>234</td><td>345.5</td><td>-56.4</td></tr>
<tr><td>MWRC0010</td><td>572884</td><td>7791005</td><td>435</td><td>100</td><td>354.0</td><td>-56.2</td></tr>
<tr><td>MWRC0011</td><td>572892</td><td>7790954</td><td>434.7</td><td>160</td><td>351.1</td><td>-55.9</td></tr>
<tr><td>MWRC0012</td><td>572905</td><td>7790904</td><td>434.6</td><td>220</td><td>346.9</td><td>-55.6</td></tr>
</table></td></tr>
<tr><td></td><td>If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.</td><td>Dinky and Dice Deposits

4 RC holes for 354 metres were excluded due to apparent wrong locations or missing assays.

Bastille, Battery, Assault, South Temby Deposits

5 RC holes for 438 metres were excluded due to apparent wrong locations or missing assays.

Southern Deposit

2 RC holes for 117 metres were excluded due to apparent wrong locations

Tombola, Miracle, Gatling, Bouncer and Bumper Deposits

3 RC holes for 136.08 metres were excluded due to apparent inconsistencies with other holes.</td></tr>
<tr><td>Data aggregation methods</td><td>In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.</td><td>This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.

In the reporting of exploration results, results are reported as weighted averages using a nominal 0.5 g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied</td></tr>
<tr><td></td><td>Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure</td><td>Most drill hole information is historical drilled from 1990 to 2012. It is unknown whether the results have been released previously. There are 6 RC holes drilled in 2024.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
	used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends
	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Mineralisation is dipping to at 0° to -85° southeast therefor core angles do vary.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	Only downhole lengths have been reported. True widths are not known.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Appropriate plans and sections have been included.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Drill holes were surveyed in local grids by mine surveyors when the mine was operating from the period 1990 to 2001. The holes were transformed to GDA94 zone 52 using appropriate transforms. Holes drilled by TAM in 2010 to 2012 had the collars picked up by a licensed surveyor using a RTK GPS with an accuracy of ± 30 mm horizontal and ± 50 mm vertical in GDA94 Zone 52. Down hole surveys were conducted by TAM every 30 metres down hole using a Reflex digital camera. The camera was calibrated once a week and the surveys were assessed for quality. Where concerns existed TAM ran a GYRO down the holes that was operated by Surtron Technologies CTP planned drillholes are sited with a handheld global positioning system (GPS), and the initial drillhole pickup is usually with a handheld GPS, as well; with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater,*	Data from the period 1990 to 2001 is historical from the period when Zapopan NL and Otter Gold Mines was operating the Tanami Gold Mine. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties

Criteria	JORC Code Explanation	Commentary
	geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	Upon receipt of all results, a review of the drilling completed is required before further work is planned.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	A number of site visits have been conducted by Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to good as there are some open pit exposures and it is based upon RC. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. **Dinky and Dice Deposits** The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 70 to 110 metres below the surface. The surface horizons are typically 1 to 2 metres thick. The Dinky and Dice gold deposit was largely hosted by basalt with lesser sandstones and siltstones. The strike and dip of the basalt / sediment contacts are about 035° and -70° northwest. The Dinky gold deposits strike between 010° to 035° and dips -25° to -65° SE depending on the lode. The Dice gold deposits strike between

Criteria	JORC Code Explanation	Commentary
		000° to 38° and dips -40° to -85° SE. Both Dinky and Dice had significant supergene gold mineralisation. **Battery, Assault and South Temby Deposits** The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 65 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick. The strike and dip of the basalt / sediment contacts are about 035° and -75° west. The Bastille gold deposit was largely hosted by sandstones and siltstones with lesser basalt. Mineralisation at Battery, Assault and South Temby is largely hosted by basalt with lesser sedimentary units. The majority of the mineralisation appears to be associated with basalt / sedimentary contacts that are complex and numerous with a number of cross cutting faults. Gold mineralisation is associated with sericite-quartz-carbonate-pyrite alteration. The Bastille Battery gold deposits strike between 020° to 036° and dips -65° to -75° SE depending on the lode. The Assault gold deposits strike between 13° to 49° and dips -62° to -70° SE. The South Temby gold deposits strike between 16° to 37° and dips -50° to -85° SE. **Hurricane-Repulse Deposits** The Hurricane-Repulse deposits are hosted by mafic volcanic flows (pillowed, vesicular and massive basalt flows) some volcanic flow breccias, sequences of lithic sandstones, siltstones and mudstones, occasional coarse sediments consisting of very proximal volcanic fragments, and more minor to rare siliceous/cherty horizons, and rare graphitic mudstones. **Southern Deposit** The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures. The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 60 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick. The Southern Gold Deposit is spatially associated with the contact between the Hurricane Sediment and the Redback Basalt Complex. The sediment / basalt contacts are complex and numerous. Doran (2013) proposed that the mineralisation was hosted by southeast dipping shear zones that encompass northwest dipping vein sets. The deposit mostly strikes between 007° to 062° and dips -65° to -85° SE depending on the lode. There are some minor lodes that dip 35° to 57° West, but they do not appear to be economically significant. **Thrasher Deposit** The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures.

Criteria	JORC Code Explanation	Commentary
		The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 70 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick. The Thrasher Gold Deposit is spatially associated with the contact between the Hurricane Sediment and the Redback Basalt Complex. The sediment / basalt contacts are complex and numerous. **Tombola, Miracle, Gatling, Bouncer and Bumper Deposits** The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures. The area is overlain by thin transported colluvium, sand, silt and laterite and weathering extends to 60 to 90 metres below the surface. The surface horizons are typically 1 to 2 metres thick. The Tombola gold deposit historically has been considered to be mineralisation north of the Miracle Open Pit giving a total strike length of 900 metres. The Tombola gold deposit mostly associated with the contact of an offset of the Bouncer Basalt and the Hurricane Sediment unit. An 085° trending fault creates the offset in the basalt / sediment sequence. The Basalt north of the fault strikes at 037° to 040° and dips -60° NW. The gold mineralisation in this area strikes at between 042° to 044° and dips -20° to -60° SE. The Basalt south of the fault strikes at 045° to 065° and dips -50° to -60° NW. Gold mineralisation strikes at 050° to 060° and dips -50° to -60° SW. The local geology consists of north westerly dipping basalt, sandstone, siltstone and mudstone. Mineralisation is hosted by sediments and lesser basalt. The Miracle gold deposit has been defined as the mineralisation around the Miracle Open Pit and to the southwest of the pit. The Bouncer Basalt in this area is discontinuous and strikes at about 060° and dips -60° NW. Gold mineralisation in this area strikes between 060° to 070° and dips -60° SE and is hosted mainly by sediment and lesser basalt. The Gatling gold mineralisation is considered to be the area south and east of the Miracle Open Pit and north of the filled in Bouncer Open Pit. The Basalt in this area strikes at about 040° and dips -60° NW. Gold mineralisation is associated with the contacts between the basalt and sediment and strikes between 040° and 075° and dips -60° to -70° SE. The Bouncer Gold deposit is the area around the now back filled Bouncer Open Pit. The Basalt in this area is striking at about 040° to 055° and dips -50° to -65° NW. Gold mineralisation is closely associated with the basalt / sediment contacts. Gold mineralisation strikes between 045° to 080° and dip -50° to -75° SE. The Bumper Gold deposit is the area around the now back filled Bumper Open Pit. The Basalt in this area is striking at 050° and dipping -50° to -60° NW and appears to consist of two separate

Criteria	JORC Code Explanation	Commentary
		units separated by thin sediment units. Gold mineralisation strikes at about 097° to 098° and dips -60° to -70° S.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	**Dinky and Dice Deposits** The Dinky and Dice Gold Deposits are spatially associated with contacts between Sedimentary units and Basalt. The sediment / basalt contacts are complex and numerous with a number of cross cutting faults. Gold mineralisation is associated with sericite-quartz-carbonate-pyrite alteration. Dinky gold mineralisation has a strike length of about 400 metres and is hosted mainly by basalt with lesser sediments. Strikes of individual lodes of gold mineralisation vary from 010° to 035° and dips -25° to -65° SE depending on the lode. The strike length of individual lodes varies from 40 to 370 metres while down dip extent varies from 10 to 150 metres. True thickness varies from 2 to 15 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -10° NE. Dice gold mineralisation has a strike length of about 270 metres and is hosted mainly by basalt with lesser sedimentary units. Strikes of individual lodes of gold mineralisation vary from 000° to 38° and dips -40° to -85° SE. The strike length of individual lodes varies from 25 to 170 metres while down dip extent varies from 10 to 90 metres. True thickness varies from 2 to 10 metres. The plunge of the gold mineralisation appears to be mostly flat however some lodes plunge up to-25° NE. **Battery, Assault and South Temby Deposits** Bastille and Battery gold mineralisation has a strike length of about 320 metres and is hosted mainly by sandstone and siltstone with lesser basalt. Strikes of individual lodes of gold mineralisation vary from 020° to 036° and dips -65° to -75° SE depending on the lode. The strike length of individual lodes varies from 25 to 315 metres while down dip extent varies from 15 to 90 metres. True thickness varies from 2 to 15 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -5° NE and one lode appears to be plunging -18° SW. Assault gold mineralisation has a strike length of about 275 metres and is hosted mainly by basalt with lesser sedimentary units. Strikes of individual lodes of gold mineralisation vary from 13° to 49° and dips -62° to -70° SE. The strike length of individual lodes varies from 30 to 90 metres while down dip extent varies from 15 to 60 metres. True thickness varies from 2 to 10 metres. The plunge of the gold mineralisation appears to be mostly flat however one lode plunges -20° SW. South Temby gold mineralisation has a strike length of about 475 metres and is hosted by basalt and sediment. Strikes of individual lodes of gold mineralisation vary from 16° to 37° and dips -50° to -85° SE. The strike length of individual lodes varies from 20 to 160 metres while the true thickness varies from 2 to 20 metres. The plunge of the mineralisation varies from being flat to 10° SW. **Hurricane-Repulse Deposits** The overall strike length of the known gold mineralisation on the Hurricane-Repulse trend is of the order of 1750 metres and has a

Criteria	JORC Code Explanation	Commentary
		variable down dip extent of about 180 metres. True thickness of gold mineralisation varies from less than a metre to 10 metres. Mineralisation on the Airstrip trend strikes at about 045° and dips between 45° to 50° southeast. The overall strike length is about 900 metres, but individual lenses vary from about 100 to 350 metres while the true thickness varies from less than a metre to several metres. The down dip extent has been interpreted to be up to 170 metres **Southern Deposit** Southern gold mineralisation has a strike length of about 540 metres and is hosted mainly by basalt with lesser sandstone. Strikes of individual lodes of gold mineralisation vary from 007° to 062° and dips -65° to -85° SE depending on the lode. There are some minor lodes that dip 35° to 57° West. The strike length of individual lodes varies from 15 to 270 metres while down dip extent varies from 15 to 120 metres. True thickness varies from 2 to 8 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -30° NE and one lode appears to be plunging -25° NW. **Thrasher Deposit** Thrasher gold mineralisation has a discontinuous strike length of about 860 metres and is hosted mainly by basalt with lesser sandstone. Strikes of individual lodes of gold mineralisation vary from 000° to 077° and dips -0° to -80° SE depending on the lode. The strike length of individual lodes varies from 10 to 170 metres while down dip extent varies from 15 to 125 metres. True thickness varies from 2 to 8 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -10° NE. Tombola, Miracle, Gatling, Bouncer and Bumper Deposits The Tombola, Miracle, Bouncer, Bumper and Gatling Gold Deposits are spatially associated with the contact between the Bouncer Basalt and the Hurricane Sediment. The sediment / basalt contacts are complex and numerous. The overall strike length of gold mineralisation in the Tombola Miracle area is approximately 1300 metres. Gold mineralisation generally cuts across the stratigraphy and is hosted by sediments and basalt. Strikes of individual lodes of gold mineralisation vary from 010° to 080° and dips -30° to -75° SE depending on the lode. The strike length of individual lodes varies from 11 to 245 metres while down dip extent varies from 11 to 100 metres. True thickness varies from 2 to 20 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -18° SW and -12° NE. The overall strike length of gold mineralisation in the Gatling Bouncer and Bumper is just over 1000 metres but the 3 areas are not continuous. Gold mineralisation at Gatling strikes between 035° to 075° and dips -60° to -75° SE. The plunges tend to be flat but can vary from -15° NE to -15° SW. Strike lengths vary from 10 to 200 metres while true thickness varies from 2 to 6 metres. Gold mineralisation at Bouncer strikes from 18° to 74° and dips from -45° to 85° SE. Plunges in this area can vary from being flat to 45° SW. Strike lengths vary from 10 to 100 metres and true thickness varies from 2 to 12 metres.

Criteria	JORC Code Explanation	Commentary
		Gold mineralisation at Bumper strikes between 56° to 106° with dips from -25° to -85° S. The overall strike direction is about 095° while the plunge is generally flat. Strike lengths vary from 10 to 205 metres with true thickness from 2 to 30 metres.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Dinky and Dice deposits. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization. An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation (20-60, 40-120 and 80-240 metres). A first pass of radius 20-60m with a minimum number of samples of 2-6 samples and a second pass of radius 40-120m with a minimum number of 2-6 samples were used. A third pass of search radius 80-240m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples was set at 10 to 12. Blocks that did not fill after 3 passes were given a 4th pass using nearest neighbour estimation. Lodes that were defined by RAB drilling were assigned a 0.5 g/t gold grade and pass 5 Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.</td></tr>
<tr><td>Moisture</td><td>Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.</td><td>Tonnages and grades were estimated on a dry in situ basis.</td></tr>
<tr><td>Cut-off parameters</td><td>The basis of the adopted cut-off grade(s) or quality parameters applied.</td><td>The Mineral Resource estimates has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.6g/t, 0.7 g/t, 0.7 g/t gold cut-off grade in Oxide, Transitional and Fresh for open pit material within a $AU3500 per ounce pit shell.

The only exception to the cut off grades is Repulse where the grades are reported above 0.5g/t, 0.5 g/t, 0.7 g/t gold cut-off grade in Oxide, Transitional and Fresh for open pit material within a $AU3500 per ounce pit shell. This is due to favourable metallurgical gold recoveries.

The potential underground resource is all material that is within 3D shapes derived from Deswik's stope optimiser software at AU$3500 per ounce.</td></tr>
<tr><td>Mining factors or assumptions</td><td>Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.</td><td>It is assumed the ML33760 deposits will be mined by open pit and underground methods when a new mining operation can be established. This model is only suitable for open pit purposes although it can be used for a preliminary assessment of underground potential.

Concentrate assumptions
Concentrate transport $370.04 per dry metric tonne of concentrate
Concentrate treatment and refining costs $147.58 per dry metric tonne of concentrate
Gold in concentrate payability 93%
Deswik Open Pit Assumptions:
Mining Ore Loss 2%
Open Pit dilution 10%
Mining Cost Insitu Rock $4.50 per tonne rock
Mining Cost Loose Rock $2.60 per tonne rock
Mining Fixed and Grade Control Costs $5.30 per tonne of ore
Mining Cost Contingency 10%
Mine ROM to Mill ROM Haulages $0.10/t per km ore
Mill CIL Opex cost $35.46 per tonne
Mill Floatation Opex cost additional $3.94 per tonne (excluding Repulse)
Admin (G&A) $4.50 per tonne
<table>
<tr><th></th><th colspan="4">Processing Recovery</th></tr>
<tr><th></th><th>Oxide</th><th>Transitional</th><th>Fresh CIL</th><th>Fresh Floatation</th></tr>
<tr><td>Bastille</td><td>90.0%</td><td>76.0%</td><td>10.1%</td><td>85.1%</td></tr>
<tr><td>Dinky Dice</td><td>86.0%</td><td>75.0%</td><td>10.1%</td><td>85.1%</td></tr>
<tr><td>Hurricane</td><td>86.0%</td><td>75.0%</td><td>10.1%</td><td>85.1%</td></tr>
</table></td></tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><td>Repulse</td><td>95.0%</td><td>94.0%</td><td>74.0%</td><td>0.0%</td></tr>
<tr><td>Southern</td><td>90.0%</td><td>76.0%</td><td>10.1%</td><td>85.1%</td></tr>
<tr><td>Thrasher</td><td>90.0%</td><td>76.0%</td><td>10.1%</td><td>85.1%</td></tr>
<tr><td>Tombola Miracle</td><td>86.0%</td><td>75.0%</td><td>10.1%</td><td>85.1%</td></tr>
</table>

Processing cost contingency 10%
Au Price AU$3500 per troy ounce
Au Royalty 5.5%
Discount Rate 8%
Mining Rate 20 Mtpa rock
Dinky Dice haulage 0.0km
Bastille haulage 1.5km
Hurricane-Repulse 0.0km
Southern haulage 1.5km
Thrasher haulage 5.6km
Tombola Miracle haulage 2.1 km

Deswik Underground Stope Optimiser Assumptions

*Minimum Mining Width 2.4 metres
Minimum Pillar Width 5 metres
Stope Strike Length 20 metres
Sub-level Interval 20 metres
Optimise grade or metal: grade
Stope Strike ±40 degrees
Stope Dip – Minimum 40 degrees
Sub Stope Shapes 2 U / 2 V
Smoothing None
*Minimum Mining Width includes allocation for HW and FW dilution
UG mining unplanned recovery 5%
UG mining unplanned dilution 5%
CIL Processing recovery 75% (Repulse)
CIL Processing recovery 10.1% of total in tailings (Bastille, Dinky Dice, Hurricane, Southern, Thrasher, Tombola Miracle)
Floatation processing recovery 85.1% of total gold in concentrate (Bastille, Dinky Dice, Hurricane, Southern, Thrasher, Tombola Miracle)
UG Stoping Costs $75/tonne ore
UG Opex Fixed Cost $5/tonne ore
Mill Opex cost $35.46/tonne ore
ROM to Mill transport $0.0/tonne ore
Admin (G&A) $4.50/tonne ore
Mining Cost Contingency 10%
Processing cost contingency 10%v
Au Royalty 5.5%
Au Price AU$3500/troy ounce

</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the</td><td>No metallurgical data could be found for the Bastille, Battery, Assault, South Temby, Thrasher and Tomola Miracle, Bouncer and Bumper gold deposits. The deposits were mined by July 1991. Recoveries from October 1990 through to June 1991 appear to</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
	process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	have varied from 83% to 90%. The nearest deposit with metallurgical data is Southern and the metallurgical testing is in line with the historical mill statistics. Preliminary metallurgical testing MineScope Services Pty Ltd on historical diamond drill core at Central Tanami completed in 2025 suggests that using flotation, concentrate export and leaching of the flotation tails produces a total gold recovery of 95.7% in fresh rock. Further testing is still required **Dinky** Metallurgical testing was carried out on the Dinky deposit by OGM (Hood, 2001). Seven RC holes were drilled at the base of the Dinky Open Pit. 5 composites weighing 10 – 15 kg were created from the drill samples and were submitted for 24-hour bottle roll gold recovery testing. All of the samples are either oxidised or transitional material. No samples are indicative of fresh rock. Gold recovery results for the Dinky Open Pit ranged from 86.7 to 95.4%. These results are in line with historical recoveries from the Tanami Mill recoveries when Zapopan and Otter Gold operated the mine. **Hurricane-Repulse** The Hurricane-Repulse pits were mined from the late 1980s to March 1994. Several satellite pits were also included in the production figures but no break down of the source feed to the mill has been found. These pits may have included Dinky, Airstrip, Temby, Dingo, Central, Bastille, Reward, Southern, Bumper and Bouncer. For the period from October 1990 to November 1991 1.36 million tonnes @ 2.34 g/t Au were fed into a CIL plant for an overall recovery of 87%. For the period from October 1992 to March 1994 1.87 million tonnes were fed into the CIL plant for an overall recovery of 85.2%. A metallurgical study was completed in 2016 following the diamond drilling of HRDD0004 to HRDD0011 to determine fold recovery in fresh rock. Anon (2016) states four holes were chosen to be included Hurricane metallurgical studies; HRDD0004, HRDD0007, HRD0010, HRDD0011. The samples selected were gold mineralisation found in fresh basalt. The results indicate that for the Hurricane area that only a 51-56% gold recovery can be achieved in fresh rock using a 75 µm grind size. This grind size is finer than would be achieved in a CIL plant. The available data suggests that metallurgical gold recovery in oxide and transitional material is in the vicinity of 85 to 87%. Fresh rock gold recovery appears to be far more complex. The Repulse area may have gold recoveries of up to 87% in fresh rock but this is based upon one sample and may not be representative of the entire Repulse pit. Leachwell assays and fire assays of residual material was completed on 29 samples from RC drilling in 2024 from Repulse. Based upon the logging of the weathering and oxidation from the onsite geologists it appears that Leachwell gold recoveries were 95% in oxide, 94% in transitional material and 74% in fresh. It must be noted that Leachwell results overestimate actual recoveries that will be achieved in a CIL plant due to the finer grind size of 75 µm and ideals conditions for gold extraction. **Southern**

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>Zapopan NL collected and tested 4 samples in 1992 for metallurgy from the Southern Gold Deposit (Jobson, 1992). The samples represented the 3 main ore types, oxide, oxide-transition and transition sulphide. In 1992 the gold recovery was considered to be 94.3% for oxide, 92.3% for oxide transition and 84.4% for transition-sulphide.

Otter Gold Mines conducted metallurgical test work on Southern Open Pit ore in 1999.They concluded that that oxide material produces good recoveries of 78% and 91% while fresh material had extremely low recoveries from 8% to 18%. Further metallurgical testing on RC hole STR0112. The results for gold recovery ranged from 78 to 95%.

TAM conducted further gold recovery work on the Southern Gold deposit on diamond drill core in fresh rock that confirmed previous test work. Gold recovery in fresh rock ranged from 5.8 to 40.3%.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.</td><td>No assumptions have been made regarding environmental factors. The area has a history of mining with existing open pits and waste dumps located in the area.</td></tr>
<tr><td>Bulk density</td><td>• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
• The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
• Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>No historical bulk density measurements from the mining activities at for the Assault, Bastille, Battery, Dinky and Dice, Southern Open Pits could be located but Zapopan NL is reported as using a bulk density of 2.3 for a number of deposits.

The closest gold deposit with actual data is the Hurricane – Repulse Open Pit.

Dinky and Dice

Bulk density was applied through oxidation state and rock type. These values were derived from data collected by TAM in 2011 from diamond drill holes HRDD0005 to HRDD0013.

<table><tr><th rowspan="2">Oxidation State</th><th colspan="4">Material Type</th></tr><tr><th>Basalt</th><th>Sediments</th><th>Waste Dump</th><th>Backfill</th></tr><tr><td>Oxide</td><td>2.29</td><td>2.51</td><td rowspan="3">2.2</td><td rowspan="3">2.2</td></tr><tr><td>Transitional</td><td>2.6</td><td>2.65</td></tr><tr><td>Fresh</td><td>2.84</td><td>2.87</td></tr></table>
Assault, Bastille, Battery, Thrasher, Tomola Miracle Bouncer</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary

Bumper Gatling Deposits

No historical bulk density results could be located from these deposits. The closest deposit to with bulk density data is Southern.

Density values were derived from taking average values for sediments and basalt and adjusting for oxidation by RL and reconciling against the average bulk density from the data collected from drill core. These values may not be correct.

Rock type	RL		Bulk Density
	From	To	gcc
BF	surface		2.2
WD	Surface		2.2
TR	surface	380	2.2
Sedimentary	440	420	2.2
	420	405	2.3
	405	390	2.4
	395	375	2.5
	375	360	2.6
	360	345	2.7
	345	100	2.8
Basalt	440	405	2.3
	405	390	2.4
	390	375	2.5
	375	360	2.6
	360	345	2.7
	345	100	2.8

Hurricane-Repulse

Bulk density data was located from Tanami Gold NL drilling data HRDD0005 to HRDD0013. All sample densities were calculated using the water displacement/air water method [Density = Weight of sample in air / (weight of sample in air – weight of sample in water)]. Results were highly variable.

Hillyard (2011) does not mention whether the samples were oven dried before being weighed so there is some uncertainty as to whether the values represent a wet or dry bulk density. There is also no mention of whether void space was considered

Bulk densities were applied to the model by rock type and oxidation state. The top of fresh rock and base of complete oxidation surfaces were re-interpreted to consider the latest drilling from 2016.

Oxidation State	Material Type	
	Basalt	Sediments
Oxide	2.29	2.51
Transitional	2.6	2.65
Fresh	2.84	2.87

Densities of 2.2 were applied to backfill and waste dumps.

Southern

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
TAM recorded 129 bulk density measurements from the 2010 drilling campaign at the Southern Gold Deposit from 7 diamond holes (SODD0001 – SODD0005, SODD0007, SODD0009) and 10 RC holes (SORC0001, SORC0003, SORC0005 – SORC0006, SORC0009 – SORC0011, SORC0014 – SORC0016). The RC data was not used as it does not take into account pore space and will typically overestimate the values and no calibration holes could be located.
Density values were derived from taking average values for sediments and basalt and adjusting for oxidation by RL and reconciling against the average bulk density from the data collected from drill core. These values may not be correct.
<table>
<tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>gcc</th></tr>
<tr><td>BF</td><td>surface</td><td></td><td>2.2</td></tr>
<tr><td>WD</td><td>Surface</td><td></td><td>2.2</td></tr>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.2</td></tr>
<tr><td rowspan="7">Sedimentary</td><td>440</td><td>420</td><td>2.2</td></tr>
<tr><td>420</td><td>405</td><td>2.3</td></tr>
<tr><td>405</td><td>390</td><td>2.4</td></tr>
<tr><td>395</td><td>375</td><td>2.5</td></tr>
<tr><td>375</td><td>360</td><td>2.6</td></tr>
<tr><td>360</td><td>345</td><td>2.7</td></tr>
<tr><td>345</td><td>100</td><td>2.8</td></tr>
<tr><td rowspan="6">Basalt</td><td>440</td><td>405</td><td>2.3</td></tr>
<tr><td>405</td><td>390</td><td>2.4</td></tr>
<tr><td>390</td><td>375</td><td>2.5</td></tr>
<tr><td>375</td><td>360</td><td>2.6</td></tr>
<tr><td>360</td><td>345</td><td>2.7</td></tr>
<tr><td>345</td><td>100</td><td>2.8</td></tr>
</table>
</td></tr>
<tr><td>Classification</td><td>

The basis for the classification of the Mineral Resources into varying confidence categories.
Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
Whether the result appropriately reflects the Competent Person’s view of the deposit.

</td><td>
The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the ‘Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves’ by the Joint Ore Reserves Committee (JORC).
The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an Inferred Resource is 3 drill holes spaced apart so that strike and dip can be determined.
Validation of the block model shows good correlation of the input data to the estimated grades where there were sufficient composites for kriging to be effective.
</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
		The result reflects the competent person's view that the classification is Indicated and Inferred.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	No audits or reviews of this estimate have been conducted.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	**Dinky and Dice** The Dinky and Dice area Mineral Resource Estimate has been reported with a moderate degree of confidence. The Measured Mineral Resource was defined within areas of RC drilling of 20 by 15m or closer where the continuity and predictability of the lode positions was excellent, and the estimation had good slopes of regression. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. The current published resource is combined with all resources listed under ML33760 (formerly MLS153, MLS122, MLS123). Mineral Resource estimates have previously been estimated and reported for the Dinky and Dice gold deposits. Zapopan NL reported resources for the deposits in 1992 through to 1994 using a 0.5 g/t Au cut off, 10 g/t Au top cut and a bulk density of 2.3. Zapopan NL reported a remnant resource in 1994 for Dice of 130,000 tonnes @ 2.4 g/t Au for about 10k ounces of gold. In the same report they reported a remnant resource of 80,000 tonnes @ 1.8 g/t Au for 4.6k ounces of gold for Dinky. In 2010 TAM reported a remnant resource of 335,942 tonnes @ 1.64 g/t Au for 17.7k ounces of gold for Dinky. It is unknown whether this resource also takes in the Dice deposit. The Dinky and Dice gold deposits were mined by Zapopan NL in the early 1990s. No individual production could be located. The current model when queried gives about 590,550 tonnes @ 1.74 g/t Au for 33K ounces for mined material from the 2 open pits. The Dice Open Pit is now backfilled with tailings while the Dinky Pit is still open but has water in the base making a determination of the stability of the walls difficult. **Bastille, Battery, Assault** Mineral Resource estimates have previously been estimated and reported for the Bastille, Battery, Assault deposits however they have been lumped in with other deposits making direct comparison extremely difficult. Bastille, Battery, Assault and South Temby were mined by Zapopan NL in the early 1990s. No individual production could be located. The current model when queried gives about 428,000 tonnes @ 2.5 g/t Au for 34K ounces for mined material from the 4 open pits. The Battery and Assault Pits are backfilled with tailings and partially covered by waste dumps. The South Temby Pit is also backfilled with tailings. The Bastille Pit is the only one that is still open but currently has water in the pit making it difficult to determine the status of the walls and whether backfill. **Hurricane Repulse** The Hurricane-Repulse Mineral Resource Estimate has been reported with a moderate degree of confidence.

Criteria	JORC Code Explanation	Commentary
		The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. Production figures were recovered for 2 periods of mining. For the period from October 1990 to November 1991 the Hurricane-Repulse area produced 1.44 million tonnes @ 2.26 g/t Au of high grade and 0.25 million tonnes @ 0.6 g/t Au of low grade. For the period from November 1992 to January 1994 the production was 1.49 million tonnes @ 2.37 g/t Au. **Southern** The Southern area Mineral Resource Estimate has been reported with a moderate degree of confidence. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20m by 20m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. Mineral Resource estimates have previously been estimated and reported for the Southern deposit. Zapopan NL published the resource figures as Measured and Indicated in 1992 through to 1994 with no breakdown of the categories with a 0.5 g/t Au low grade cut off and a 10 g/t Au high grade cut. In 1992 the Measured and Indicated Mineral Resource was 346,000 tonnes @1.90 g/t Au and in 1994 it was stated at 334,000 tonnes @ 1.9 g/t Au. The 2001 resource by Otter Gold Mines is constrained to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. A 20 g/t Au high grade cut was applied to the sample data. They quoted a Measured, Indicated and Inferred Mineral Resource at 423,000 tonnes @ 2.30 g/t Au. The 2011 resource appears to have used grade restriction rather than high grade cuts and has only used a 0.7 g/t au low grade cut off with no consideration for economic parameters. The Measured, Indicated and Inferred Mineral Resource was quoted as 1,084,960 tonnes @ 2.31 g/t Au. The current published resource is combined with all resources listed under MLS153. The Southern Open Pit was mined by Zapopan NL from March 1991 to November 1992. The actual production is unknown. The open pit was mined to a depth of 45 metres. The base of the open pit was about at the 380 m RL while the surface was around 225 m RL. The current model produces 242,334 tonnes @ 2.19 g/t Au for 17,030 ounces of gold for the mined portion at 0.5 g/t Au low grade cut off. Total volume (including ore and waste) removed from the open pit is about 1.11 million m3 or 2.57 million tonnes. **Thrasher** The Thrasher area Mineral Resource Estimate has been reported with a moderate degree of confidence. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20m by 20m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. Mineral Resource estimates have previously been estimated and reported for the Thrasher deposit. In 2001 OGM published an Indicated and Inferred Mineral Resource of 77,000 tonnes @ 2.4 g/t Au. This resource is constrained to pit shells (Lerch-Grossman

Criteria	JORC Code Explanation	Commentary
		algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. A 20 g/t Au high grade cut was applied to the sample data. In 2010 TAM produced an Indicated and Inferred Mineral Resource of 160,859 tonnes @ 2.2 g/t Au using a low grade cut off of 0.7 g/t Au. No open pit optimisation constraints were applied to this resource. In 2011 TAM updated the Thrasher Mineral Resource Estimate. They produced and Indicated and Inferred Resource of 197,534 tonnes @ 2.1 g/t Au using a 0.7 g/t Au low grade cut off with no other economic constraints. **Tomola Miracle Bouncer Bumper Gatling Deposits** Mineral Resource estimates have previously been estimated and reported for the Miracle deposit. Zapopan NL published the resource figures as Measured and Indicated in 1992 through to 1994 with no breakdown of the categories with a 0.5 g/t Au low grade cut off and a 10 g/t Au high grade cut. In 1992 the Measured and Indicated Mineral Resource was 454,000 tonnes @1.7 g/t Au and in 1994 it was stated at 324,000 tonnes @ 1.6 g/t Au. No production records have been located for the Miracle deposit but the current model estimates that the open pit produced about 64,000 tonnes @ 2.5 g/t Au at a 0.5 g/t Au low grade cut off. No production records have been located for the Bumper and Bouncer open pits. The current model estimates that they produced about 258,000 tonnes @ 2.8 g/t Au and 206,000 tonnes @ 2.8 g/t Au respectively at a 0.5 g/t Au low grade cut off. The 2001 resource by Otter Gold Mines is constrained to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. A 20 g/t Au high grade cut was applied to the sample data. They quoted a Measured, Indicated and Inferred Mineral Resource for Tombola at 114,000 tonnes @ 2.30 g/t Au and for Gatling at 70,000 tonnes @ 3.3 g/t Au. In 2010 TAM produced a Measured, Indicated and Inferred resource for the Gatling / Bastille area of 1.03 million tonnes @ 2.4 g/t Au. In 2013 this resource was updated and increased 3.09 million tonnes @ 2.0 g/t Au.

Appendix 3 - JORC Table 1
EL26926

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) core drilling. Sampling of RC chips was completed on RC drillholes and half core sampling on diamond drillholes was completed.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled. Diamond drilling used NQ sized core and was half cut using metre intervals.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a four-deck riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40 litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples. For RC Holes drilled by Northern Star Resources (NST) from 2018 to 2019, 1m RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole. Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on metre intervals.
Drilling techniques	*Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.75" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample. DD core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists.

Criteria	JORC Code Explanation	Commentary
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by geologists.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged on 1 metre intervals with data subsequently merged into an access database. A representative portion of each RC metre was retained in chip trays and stored on site. Geologists log DD core. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples are logged for lithology, alteration, mineralisation. Logging is a mix of qualitative and quantitative observations. Visual estimates are made of sulphide, quartz, and alteration as percentages. RC samples are not photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and DD hole was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample. RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter. Northern Star Stage-1 RC drilling saw all bulk material collected on a 1m basis directly from cyclone in pre labelled green plastic mining bags. Northern Star Stage-2 RC drilling saw single metre (1m) samples collected from a trailer mounted static cone splitter. Approximately 12.5% of each meter sample was collected in a pre-labelled calico bag with the depth while the remaining 87.5% was collected in a green mining bag and retained. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.

Criteria	JORC Code Explanation	Commentary
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	During mining operations drill samples were prepped either at onsite or at ALS in Alice Springs to industry standards. The Otter Gold Mines data does include some onsite analysis at the mine laboratory. Northern Sar sample preparation was conducted at ALS Perth, commencing with sorting, checking and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	Grind checks are performed at both the crushing stage (3mm) and pulverising stage (75µm), requiring 90% of the material to pass through the relevant size.
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate and to industry standard. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 20 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology, and assay value ranges for gold.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples collected during mining operations were submitted to the onsite laboratory or ALS in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings. Samples collected by Northern Star were sent to ALS in Malaga, Perth. Gold (Au) concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC Chip sample collection.
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	1994 to 2001 Analysis (both on and offsite) was by AAS with selective FA checks. The onsite procedure incorporates the inclusion of a check sample, quartz wash and a standard sample per batch of 30 samples. On a monthly basis one pulp per shift (ie two per day) was selected and analysed offsite by AAS and Fire assay for repeatability. Additional check samples were selected and assayed offsite as required by the geological staff.

Criteria	JORC Code Explanation	Commentary
		The Northern Star QAQC protocols used include the following for all drill samples: • Field QAQC protocols used for all drill samples include commercially prepared certified reference materials (CRM) inserted at an incidence of 1 in 20 samples. The CRM used is not identifiable to the laboratory with QAQC data is assessed on import to the database and reported monthly, quarterly and yearly. • NSR RC Resource definition drilling routinely inserts field blanks and monitor their performance. • Laboratory QAQC protocols used for all drill samples include repeat analysis of pulp samples occurs at an incidence of 1 in 20 samples, and screen tests (percentage of pulverised sample passing a 75µm mesh) are undertaken on 1 in 40 samples. • The laboratories' own standards are loaded into the database, and the laboratory reports its own QAQC data monthly. • Blanks were routinely inserted into the sample sequence at a rate of 1 per 25 samples and again specifically after potential or existing high-grade mineralisation to test for contamination. Failures of blanks above 0.2g/t were followed up, and re-assayed. New pulps were prepared if failures continued. • Failed standards are generally followed up by re-assaying a second 30g pulp sample of all samples in the fire above 0.1ppm by the same method at the primary laboratory. The accuracy component (CRMs and third-party checks) and the precision component (duplicates and repeats) of the QAQC protocols are thought to demonstrate acceptable levels of accuracy and precision.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	All data is historical from the period when Otter Gold Mines ran the operation. Significant intersections were signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled for this data set.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data was imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates.

Criteria	JORC Code Explanation	Commentary
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	All collar locations were set from surveyed grids or individually surveyed into position and subsequently surveyed by the mine survey department after drilling. This data was transformed using appropriate grid transformations in MGA94 Zone 52.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition with the addition of drill hole collar pick-ups from qualified mine surveyors.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider-spaced drilling or insufficient drilling in smaller lodes.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends.
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	Unknown for holes from 1994 to 2001
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Unknown For Northern Star the chain of custody of samples was managed by geologists and geotechnicians. Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at the on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email or uploaded to an FTP site. Sample pulp splits are stored for a time at the laboratory.

Criteria	JORC Code Explanation	Commentary
		Retained pulp packets are returned to the Central Tanami Mine for storage.

Section 2 Reporting of Exploration Results
(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Galifrey Resource is on Exploration Lease EL26926, 3.9 km west southwest of Jims Open Pit and approximately 29 kilometres southwest of the Central Tanami Mill. EL26926 forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108km2 tenement area in the Tanami Region held by the CTPJV are registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	EL26926 is granted and in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Galifrey area has been explored since 1989. Several previous companies, Otter Gold Mines, Zapopan NL, and Tanami Gold NL have been active in the area.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The Galifrey deposit is a Palaeoproterozoic, granite / felsic intrusive and sediment shear hosted deposit that is part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a quartz stockwork that is associated with granite / felsic intrusive and sediment contacts.

Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:*
	  • *easting and northing of the drill hole collar*
	  • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar*
	  • *dip and azimuth of the hole*
	  • *down hole length and interception depth*
	  • *hole length*
		All drill hole information is historical drilled from 1994 to 2019.

	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly*	No holes are excluded from this report.

Criteria	JORC Code Explanation	Commentary
	explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. In the reporting of exploration results, results are reported as weighted averages using a nominal 0.5g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied.
	• *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends
	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Mineralisation is sub-vertical to vertical.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	Only downhole lengths have been reported. True widths are not known.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Appropriate plans and sections have been included.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Drill holes were surveyed in local grids by mine surveyors when the mine was operating from the period 1990 to 2001. The holes were transformed to GDA94 zone 52 using appropriate transforms. Holes drilled later than this were recorded in GDA94 zone 52 with a DGPS.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater,*	Data from the period 1994 to 2001 is historical from the period when Otter Gold Mines was operating the Tanami Gold Mine. Northern Star Resources (NST) drilled holes from 2018 to 2019.

Criteria	JORC Code Explanation	Commentary
	geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	Upon receipt of all results, a review of the drilling completed is required before further work is planned.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records. There are some minor overlap errors in the RC and diamond drill holes where 4 metre samples overlapped later 1 metre samples, but the occurrence was not significant
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	A number of site visits have been conducted by Mr Joe McDiarmid, Director of MoJoe Mining Pty Ltd and Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is low to moderate as it is based upon a relatively small number of holes with variable spacing. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. The Galifrey deposit strikes about 310° and dips steeply. The 310° structural trend is interpreted to be a shear zone (Galifrey Fault). The local geology consists of intercalated granite / felsic intrusive, sandstone, and siltstone.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise),*	Galifrey gold mineralisation consists of two separate shear zones with strike lengths of about 2400 and 500 metres respectively.

Criteria	JORC Code Explanation	Commentary
	plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	Strikes of individual lenses of primary mineralisation vary from 295° to 335° and dip steeply. There are some minor near surface regolith hosted laterite and mottled clay mineralisation that are flat lying. The strike length of individual lenses of gold mineralisation varies from 25 to 500 metres but are more typically 50 to 60 metres. True thickness varies from 1 to 2 metres to several metres. The down dip extent is typically of the order of 50 to 100 metres.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the deposit. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization. An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation (20-50, 40-100 and 80-200 metres). A first pass of radius 20-50m with a minimum number of samples of 2-6 samples and a second pass of radius 40-100m with a minimum number of 2-6 samples were used for Galifrey. A third pass of search radius 80-200m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 18-28 depending on the number of samples in the domain. Blocks that did not fill after 3 passes were given a 4th pass using nearest neighbour estimation. Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite

Criteria	JORC Code Explanation	Commentary
		file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.7 g/t gold cut-off grade for oxide, 0.6 g/t Au for transitional and Fresh for open pit material within a $AU3500 pit shell. For underground all material within a AU$3500 stope optimisation is reported.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	It is assumed the Galifrey deposit will be mined by open pit and underground methods when a new mining operation can be established. This model is only suitable for open pit purposes. The following mining factors and costs were used for the Deswik optimisation of the open pit and underground resource: Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10% • Pit Slopes – Oxide 39° • Pit Slopes – Other 45° • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10% • Mine ROM to Mill ROM Haulages $0.10/t per km ore • Mill Opex cost $35.46 per tonne • Admin (G&A) $4.50 per tonne • CIL Processing Recovery 76% oxide, 95% transitional, 92% fresh • Processing cost contingency 10% • Au Price AU$3500 per troy ounce • Au Royalty 5.5% • Discount Rate 8% • Mining Rate 20 Mtpa rock • Galifrey haulage 29 km Deswik Underground Stope Optimiser Assumptions • *Minimum Mining Width 2.4 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 20 metres • Optimise grade or metal: grade • Stope Strike ±40 degrees • Stope Dip – Minimum 40 degrees

Criteria	JORC Code Explanation	Commentary
		• Sub Stope Shapes 2 U / 2 V • Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • CIL Processing recovery 92% • UG Stoping Costs $75/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $35.46/tonne ore • ROM to Mill transport $2.90/tonne ore • Admin (G&A) $4.50/tonne ore • NT Factor (10%) $12.79/tonne ore • Au Royalty 5.5% • Au Price AU$3500/troy ounce
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	There is no metallurgy for the Galifrey deposit. Recovery data is assumed
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	No assumptions have been made regarding environmental factors.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the*	No bulk density data from the Galifrey prospect could be located. Density values were derived from taking average values for sediments and granite / felsic intrusive and adjusting for oxidation by RL. The following values were applied.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td>measurements, the nature, size and representativeness of the samples.
• The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
• Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>
<table>
<tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>gcc</th></tr>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.2</td></tr>
<tr><td rowspan="6">Sedimentary</td><td>440</td><td>340</td><td>2.2</td></tr>
<tr><td>340</td><td>330</td><td>2.3</td></tr>
<tr><td>330</td><td>320</td><td>2.4</td></tr>
<tr><td>320</td><td>300</td><td>2.5</td></tr>
<tr><td>300</td><td>290</td><td>2.6</td></tr>
<tr><td>290</td><td>180</td><td>2.7</td></tr>
<tr><td rowspan="7">Granite / Felsic</td><td>surface</td><td>400</td><td>2.1</td></tr>
<tr><td>400</td><td>360</td><td>2.2</td></tr>
<tr><td>360</td><td>340</td><td>2.3</td></tr>
<tr><td>340</td><td>320</td><td>2.4</td></tr>
<tr><td>320</td><td>300</td><td>2.5</td></tr>
<tr><td>300</td><td>280</td><td>2.6</td></tr>
<tr><td>280</td><td>180</td><td>2.7</td></tr>
</table>
</td></tr>
<tr><td>Classification</td><td>• The basis for the classification of the Mineral Resources into varying confidence categories.
• Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
• Whether the result appropriately reflects the Competent Person's view of the deposit.</td><td>The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC).
The Mineral Resource was classified as Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.
Validation of the block model shows good correlation of the input data to the estimated grades where there were .
The result reflects the competent person's view that the classification is Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>• The results of any audits or reviews of Mineral Resource estimates.</td><td>No audits or reviews of this estimate have been conducted.</td></tr>
<tr><td>Discussion of relative accuracy/ confidence</td><td>• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.</td><td>The Galifrey Mineral Resource Estimate has been reported with a low to moderate degree of confidence.
No previous mining has occurred in this area.
The Mineral Resource statement relates to global estimates of tonnes and grade.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
	• *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

Appendix 4 - JORC Table 1
ML(S)167

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) drilling. Sampling of RC chips was completed on RC drillholes. Sampling of diamond holes utilised half cut core.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled. Diamond drilling used a combination of HQ and NQ2-sized core. HQ core was drilled until competent ground was intersected, then NQ2 core was drilled. Drill core was oriented, aligned, and half-cut using metre intervals and geologically determined intervals (max 1.2 metres and min 0.3 metres), with geologically determined intervals taking precedence. Diamond holes were marked up against the core runs
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a four-deck riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40 litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples. RC holes drilled by TAM were collected through a 75:25% riffle splitter in prenumbered bags. The samples varied from wet to dry. Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.75" face sampling hammer drill bit. Diamond drilling was completed using HQ or NQ utilizing triple tube recovery.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample.

Criteria	JORC Code Explanation	Commentary
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks. This was checked and compared to the core measurements by the geological team. Any issues were communicated back to the drilling contractor, and necessary adjustments were made.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged on 1 metre intervals with data subsequently merged into an access database. A representative portion of each RC metre was retained in chip trays and stored on site. Geologists log DD core. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples are logged for lithology, alteration, mineralisation. Logging is a mix of qualitative and quantitative observations. Visual estimates are made of sulphide, quartz, and alteration as percentages. RC samples are not photographed. All DDH logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample. RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter.
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate.

Criteria	JORC Code Explanation	Commentary
		Zapopan NL completed all sample preparation pre-1994 at the on site laboratory. During mining operations drill samples were prepped either at onsite or at ALS in Alice Springs to industry standards. The Otter Gold Mines data does include some onsite analysis at the mine laboratory. Drill samples collected by TAM were submitted to SGS Laboratories in Perth and assayed using a 50g fire assay charge for gold with an atomic spectrometer finish. This method had a 0.01ppm detection limit. Sample weights were generally around 3kg in size.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	No formal heterogeneity study has been completed or monographed. An informal analysis suggests that the sampling protocol currently in use is appropriate to the mineralisation encountered and should provide representative results
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 30 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology, and assay value ranges for gold.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples collected during mining operations were submitted to the onsite laboratory or ALS in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings. Tanami Gold sent RC samples to SGS Laboratories in Perth for the 2010 to 2011 drilling. They were assayed using a 50g fire assay charge for gold with an atomic spectrometer finish and a 0.01 ppm detection limit.
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	1994 to 2001 Analysis (both on and offsite) was by AAS with selective FA checks. The onsite procedure incorporates the inclusion of a check sample, quartz wash and a standard sample per batch of 30 samples. On a monthly basis one pulp per shift (ie two per day) was selected and analysed offsite by AAS and Fire assay for repeatability. Additional check samples were selected and assayed offsite as required by the geological staff. For TAM drilling in 2010 certified reference material (CRM) were inserted every 30 samples and blanks every 50 samples. Any CRM that fell outside of 2 standard deviations of the expected value was followed up to determine the cause.

Criteria	JORC Code Explanation	Commentary
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	The majority of data is historical from the period when Otter Gold Mines ran the operation and when TAM was the sole operator. Significant intersections were signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled for this data set.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data was imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	All collar locations were set from surveyed grids or individually surveyed into position and subsequently surveyed by the mine survey department after drilling. This data was transformed using appropriate grid transformations in MGA94 Zone 52.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition with the addition of drill hole collar pick-ups from qualified mine surveyors.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	The Measured and Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m or closer (with some infill), where the continuity and predictability of the lode positions were good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider-spaced drilling or insufficient drilling in smaller lodes.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends.
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.

Criteria	JORC Code Explanation	Commentary
Sample security	• *The measures taken to ensure sample security.*	The security measures for holes from 1994 to 2001 have not been recorded. For TAM the chain of custody of samples was managed by geologists and geotechnicians. Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at the on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email. Sample pulp splits are stored for a time at the laboratory. Retained pulp packets are returned to the Central Tanami Mine for storage.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Audits or reviews for holes from 1994 to 2001 have not been located. For TAM results were reviewed by geologists to ensure sampling returned a representative sample..

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca, Gold Deposits are located in the Tanami Region in the Northern Territory on Mineral Lease (Southern) MLS167, approximately 10km southwest of the Central Tanami Mill site. MLS167 covers an area of 1877ha and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108km2 tenement area in the Tanami Region held by the CTPJV are registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. Mineral Leases have a 25-year life and are renewable for 25 years. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	MLS167 is granted and in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca areas have been explored since the early 1990's. Several

Criteria	JORC Code Explanation	Commentary
		previous companies, Zapopan NL, Otter Gold Mines, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The Carbine, Dogbolter to Lynx, Legs, Redback Area, Phoenix and Inca area deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that are part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts.
Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length*	All drill hole information is historical drilled from 1994 to 2010. **Phoenix and Inca** 181 RC holes for 5476.3 metres were excluded due to missing assays. These were mainly grade control holes.
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	No holes are excluded from this report.
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	The reporting of exploration results would have largely been by Otter Gold Mines between 1994 to 2001. The reports for this era have not been located. This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. In the reporting of exploration results, results are reported as weighted averages using a nominal 0.5 g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied
	• *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	All drill hole information is historical drilled from 1994 to 2010. It is unknown whether the results have been released previously. This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.
Relationship between	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends

Criteria	JORC Code Explanation	Commentary
mineralisation widths and intercept lengths	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Mineralisation is dipping to at 30° to 90° southeast therefor core angles do vary.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	Only downhole lengths have been reported. True widths are not known.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Appropriate plans and sections have been included.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Drill holes were surveyed in local grids by mine surveyors when the mine was operating from the period 1994 to 2001. The holes were transformed to GDA94 zone 52 using appropriate transforms. Holes drilled by TAM in 2010 had the collars picked up by a licensed surveyor using a RTK GPS with an accuracy of ± 30 mm horizontal and ± 50 mm vertical in GDA94 Zone 52. Down hole surveys were conducted by TAM every 30 metres down hole using a Reflex digital camera. The camera was calibrated once a week and the surveys were assessed for quality. Where concerns existed TAM ran a GYRO down the holes that was operated by Surtron Technologies
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Data from the period 1994 to 2001 is historical from the period when Otter Gold Mines was operating the Tanami Gold Mine. TAM drilled 4 diamond and 7 RC holes in 2010 however these were targeted at the nearby Carbine deposit.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	Upon receipt of all results, a review of the drilling completed is required before further work is planned.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	A number of site visits have been conducted by Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to good as there are exposures in the open pits and it is based upon RC and Diamond drill holes. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. **Carbine** The Carbine deposit is hosted within the redback basalt of the Tanami Group (1838+/-6Ma), a basaltic sequence with intercalated sediments. Dolerite sills up to 200+m thick intrude the Tanami Group. The stratigraphic sequence from south to north at the Carbine open pit consists of north dipping Hurricane sediments, Redback basalt with intercalated sediments from 0.5 to 10 metres that are unconformably overlain by Gardiner sandstone. The sequence has been interpreted as forming within an intracratonic setting. This is supported by abundant hematite and metamorphic detritus within the intercalated sediments. Mineralisation at Carbine is mainly hosted by quartz veining within strongly altered basalt and is structurally controlled within a dominant 060° to 070° and a lesser 020° to 045° striking structures. Most of the ore mined (+90%) was hosted by pillow basalt. The main mineralised trend at Carbine strikes at 060° to 070° and dips -55° to -80° SE and plunges about 15° SW. A lesser cross cutting zone of mineralisation strikes at 020° to 045° and dips -60° to -70° SE and plunges about 15° SE. **Dogbolter to Lynx Area Deposits** The Dogbolter gold deposit strikes between 000° to 020° and dips 60° to 70° East depending on the lode. The local geology consists of northwest dipping basalt and haematitic siltstone with minor

Criteria	JORC Code Explanation	Commentary
		lithic siltstone, granite and dacite. The strike and dip of the basalt / sediment contacts are about 020° and -60° west. Mineralisation is largely hosted by basalt. Kelpie gold mineralisation strikes between 5° and 17° and dips 60° to 80° East. The local geology consists of northwest dipping basalt and lithic siltstone. The area is structurally complex with small scale faulting and folding. Overall, the strike and dip of the basalt / sediment contact is about 015° to 020° and -60° west. Mineralisation is largely hosted by siltstone with minor basalt. The Bulldog area is along strike of the Kelpie gold mineralisation. Gold mineralisation consists of veins with 2 distinct orientations. The main lens of mineralisation strikes at 010° to 015° and dips -50° to -60° east while a cross cutting set strikes between 35° to 70° and dips -40° to -65° southeast. 3 separate lenses of basalt intercalated with sediments strike at about 025° and dip -50° to -60° west. A number of faults have been interpreted to offset the stratigraphy by up to 15 metres. Gold mineralisation is hosted by both basalt and sedimentary units. Lynx gold mineralisation is located about 200 to 300 metres northwest of the Bulldog mineralisation that strikes and dips at 060° to 070° and -30° to -50° southeast. The local geology was interpreted from geology intersected in drilling and the airborne aeromagnetic data and consists of intercalated basalt and sediment that strike at between 040° to 050° and dip -40° to -55° northwest. Dogbolter NE is located about 200 to 600 metres northeast of the Dogbolter main gold mineralisation. The main veins strike at between 050° to 075° and dip -60° to -85° southeast. A few minor vein sets strike at 020° to 025° and dip -50° to -55° east. Interpreted basalt units in this area strike overall at 020° and dip -50° to -60° west. The western pit was interpreted as being within the Redback Basalt complex while eastern pit was within the Harleys sediment package. **Legs Deposit** The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and a slight inflexion in the strike of the units. The main structures within the basalt strike between 050° to 075° and dip 50° to 75° southeast while structures within the sedimentary horizons strike at about 030° to 045° and dip 30° to 80° east southeast. OGM carried out pit mapping while mining and found a number of gold bearing structures that were striking between 340° to 10° and dipping 50° to 80° North. The structures appear to be bounded by the major structures previously mentioned. **Redback Area Deposits** The gold mineralisation in the Redback area has three dominant strike directions and is structurally complex. The gold mineralisation occurs at the boundaries between basalt and sediment units defined by OGM and a slight inflexion in the strike

Criteria	JORC Code Explanation	Commentary
		of the units. The local geology consists of northwest dipping basalt, sandstone, and siltstone. There are 3 main gold mineralisation trends with the Redback area that appear to be controlled by faulting and rheology contrast provided by the contacts of sedimentary and basaltic units. The Funnelweb trend strikes at about 020° and occurs within the Redback basalt sequence. This sequence is intercalated with conglomerates, sandstone and siltstone. A second major trend occurs from Redback SW to Harley's open pit with the mineralisation striking overall at 060°. The third trend occurs from Daddy open pit to Redback SE open pit where the overall strike of the mineralisation varies from 060° to 090°. The Money open pit does not align with other deposits but is associated with a 060° trend in the footwall of the Harley's sediments in the footwall basalt complex. The Funnelweb mineralisation is hosted within the Redback basalt sequence. The basalt in the area of the Funnelweb open pit is striking at 020° and dipping about -65° west. The basalt in this area is 50 to 80 metres thick and is bounded on the eastern side by conglomerate and sandstone / siltstone and mudstone on the western contact. A number of cross cutting fault zones have been mapped but displacement is not shown. The sequences are unconformably overlain by the Gardiner Sandstone in the northern part of the pit. Geological mapping of the Money Open Pit has shown the rocks consist of the Footwall Basalt Complex and Harley's sediments that are unconformably overlain by the Gardiner Sandstone in the northern part of the pit. Harley's sediments in this area consist of conglomerate, sandstone, and siltstone. A well-developed laterite zone occurred over the northwest portion of the open pit. The overall interpreted strike and dip of the basalt sequence in this area is between 040° to 055° and -40° to -50° northwest. Geological mapping of the Harley's open pit showed that the majority of the open pit is within the Harley's sediment sequence that consists of conglomerate, sandstone, siltstone, and mudstone intercalated with dacite and basalt. The overall strike and dip of the lithologies varies in strike from 010° to 040° and -60° west. There appears to be at 3 basalt units that are distinguishable in the open pit. The basalt exposed in the western wall is interpreted to part of the Redback Basalt sequence while the remainder form part of the Footwall Basalt Complex. The Redback Basalt sequence has a thickness of about 100 metres in this area and appears to have been structurally thickened. The Footwall Basalt Complex units are only 10 to 15 metres thick and discontinuous. Geological mapping of the Huntsman open pit showed that it is mainly within the Harley's sediments. The Redback Basalt sequence is interpreted to be transecting the western side of the pit. The Harley's sediments in this area have been described as micaceous sandstone, feldspathic sandstone, siltstone, and mudstone intercalated with basalt. The strike and dip of the sediment sequence is about 020° to 025° and -40° to -45° west. The eastern side of the open pit had a well-developed laterite zone.

Criteria	JORC Code Explanation	Commentary
		The Huntsman open pit has been backfilled and is now a small waste dump. Huntswoman was located in Harley's sediments and strongly brecciated basalt. Recent interpretation has shown that Huntswoman had a well-developed laterite along a 060° trend. No geological mapping could be located for the Katipo open pit, but it has been noted that it had a well-developed laterite and was hosted by the Redback Basalt sequence. The laterite is striking at about 060°but this may reflect a major structural feature. The basalt is at least 100 metres thick in this area and most likely structurally thickened. The overall strike of the basalt is likely to be around 030°. Geological mapping of the Redback SE open shows that the western end of the pit is within Harley's sediments while the eastern part of the pit is within the Footwall Basalt Complex. Harley's sediments in this area consists of sandstone, feldspathic sandstone, hematitic sandstone, and siltstone while Footwall Basalt Complex has been described as porphyritic, pillow, vesicular and fine-grained basalt. The strike and dip of the sediments is about 020° to 030° and dip -65° west. The Footwall Basalt Complex appears to truncate some of the sediments and has an apparent strike of about 060° and dip of -50° to -60° northwest. This may reflect the structures cutting through the area. The eastern area of the open pit had a well-developed laterite. Geological mapping of the Redback SW open pit shows that Hurricane sediments are exposed in the pit and the sequence consists of siltstone and mudstone intercalated with basalt sequences. The basalt is interpreted to be 30 to 40 metres thick and striking overall at about 010° and dipping -50° to -65° west. Laterite is developed in the eastern end of the pit. There appears to be an 060° structural trend cutting the geology. Geological mapping of the Daddy open pits shows that the western end of the open was excavated in Hurricane sediments while the eastern portion contains basalt and interflow sediment. The sediments consist of siltstone, conglomerate, and sandstone that are striking at about 020° and dipping -50° west. The overall strike of the basalt varies from 025° to 030° and dips -45° to -65°. Major structures cutting through the open pits vary from 060° to 095° and have maximum thicknesses of 60 metres. The eastern pit has been backfilled. **Phoenix Inca Deposits** The local geology consists of northwest dipping basalt, sandstone, and siltstone. The gold mineralisation occurs at the boundary between basalt and the Hurricane sediment unit defined by OGM and 020° and 060° structures. The Phoenix gold deposit strikes between 055° to 080° and dips -65° to -85° NW depending on the lode. The local geology consists of westerly dipping basalt, conglomerate, sandstone, and intercalated siltstone and mudstone. The strike and dip of the basalt / sediment contacts are about 020° and -60° west. Mineralisation is largely hosted by basalt. Kelpie gold

Criteria	JORC Code Explanation	Commentary
		mineralisation strikes between 5° and 17° and dips 60° to 80° East. The local geology consists of northwest dipping basalt and lithic siltstone. The area is structurally complex with small scale faulting and folding. Overall, the strike and dip of the basalt / sediment contact is about 015° to 020° and -60° west. Mineralisation is largely hosted by siltstone with minor basalt. The Phoenix gold resource is located about 900 metres south of the Carbine gold mineralisation and shows similarities with larger deposit. The Inca trend gold mineralisation strike between 015° to 040° and dips -50° to -75° NW depending on the lode. The local geology consists of basalt and intercalated sedimentary units that are striking at about 030° and dipping steeply northwest.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	**Carbine** The overall strike length of the mineralisation is of the order of 1250 metres while the dip extent is about 600 metres. Individual lenses of mineralisation vary in strike length from 60 to 600 metres and have down dip extents of 10 to 160 metres with true thickness varying from 2 to 10 metres. The arrangement of the mineralisation can be described as a stacked lens array. The true thickness of the material mined in the open pit was around 10 metres **Dogbolter to Lynx Area Deposits** Dogbolter main mineralisation has a strike length of about 490 metres and is hosted mainly by basalt with lesser siltstone. Strikes of individual lenses of gold mineralisation vary from 358° to 039° and dip -60° to -90° east. The strike length of individual lenses varies from 20 to 370 metres while down dip extent varies from 15 to 105 metres. True thickness varies from 2 to 16 metres. The plunge of the gold mineralisation appears to be flat to 5° either north or south. The ore zone is described as a major quartz vein and stockwork with the width blowing out to 30 metres where the main zone intersected the Kelpie zone. Further, they describe the best grades as occurring in basalt with slightly lower grade and width where the east dipping mineralisation cuts across the west dipping sedimentary units. Gold mineralisation appears to be offset by faulting down dip however evidence for this could not be confirmed. A surficial mineralised laterite / colluvial horizon mimicked the strike of the main zone of mineralisation and had a strike of 015° and flat dip. The strike of the regolith hosted gold was 200 metres with a width that varied from 10 to 40 metres and a thickness from 2 to several metres. Kelpie gold mineralisation has a strike length of about 350 metres and is hosted by sedimentary units and basalt. Strikes of individual lenses of gold mineralisation vary from 8° to 17° and dip -60° to -75° east. The strike lengths of individual lenses of gold mineralisation vary from 35 to 300 metres while the down dip extent varies from 20 to 90 metres. True thickness varies from 2 to 12 metres. The plunge of the gold mineralisation appears to be flat to 5° either north or south. The sediments hosting the mineralisation have been described as haematitic siltstone, sandstone, and basalt. The area is structurally complex, and several small-scale folds and faults

Criteria	JORC Code Explanation	Commentary
		have been mapped in the ramp of the Dogbolter main open pit. Makar (2001) describes Kelpie as a narrow quartz and stockworks developed within a shear zone, 4 to 5 metres thick. Gold mineralisation pinches and swells along strike with the best grades in basalt. Further Makar (2001) states that there are noticeable blow outs in width and grade at intersecting 060° trending structures. The Bulldog gold mineralisation is along strike of the Kelpie deposit and has a strike length of about 290 metres and is hosted by intercalated basalt and sedimentary units. Strike of individual lenses of gold mineralisation vary in strike from 010° to 060° and vary in dip from -35° to -65° east. The strike length of individual lenses varies from 30 to 290 metres while the down dip extent varies from 30 to 150 metres. True thickness varies from 2 to 5 metres. The plunge of the gold mineralisation appears to be relatively flat. The gold mineralisation at Lynx is north of Bulldog and appears to have similarities with the Legs deposit located about 250 metres further north. Strike of individual lenses of gold mineralisation vary from 50° to 85° and dip -30° to -55° south. The strike lengths of individual lenses vary from 30 to 200 metres while the down dip extent varies from 30 to 160 metres. True thickness varies from 2 to 8 metres. The plunge of the gold mineralisation appears to be flat to 5° either north or south. The mineralisation is hosted by intercalated basalt and sedimentary units that dip to the north. Dogbolter NE has a strike length of about 380 metres and is hosted by basalt with minor sedimentary units in the southwest pit and mainly sediments in the junior pit. The majority of the mineralisation is developed along the 060° trend however there was a well-developed vein along the 020° trend. Strike length varies from 15 to 170 metres and the down dip extent ranges from 10 to 100 metres. True thickness varies from 2 to 15 metres. The strike and dip of the 060° trending lenses varies from 050° to 075° and dips from -60° to -85° south. Strike and dip of the 020° trending veins vary from 020° to 023° and dip -50° to -55° east. The plunge of the gold mineralisation appears to be flat to 10°. Makar (2001) describes the gold mineralisation in the southern pit as a 4 to 5 metre thick quartz vein and stockwork in a bleached basalt with pyrite and sericite alteration. Further, the north pit is described a quartz stockwork with intense silicification and bleaching of the host sedimentary units. **Legs** Legs gold mineralisation has a main zone strike length of about 650 metres and is hosted by sandstone and siltstone in the southwest and basalt with minor sediment in the northeast. The strike of mineralisation varies depending upon rock type. Strikes of individual lenses of gold mineralisation in the sedimentary units vary from 030° to 045° and dip 30° to 80° east. The strike length of individual lenses of gold mineralisation varies from 40 to 200 metres but are more typically 100 metres. True thickness varies from 2 metres to 25 metres. The down dip extent is typically of the order of 50 metres and plunges vary from about 10° to 25° to the southwest. Strikes of individual lenses of main gold mineralisation in the basalt vary from 050° to 075° and dip 50° to 75° southeast. The

Criteria	JORC Code Explanation	Commentary
		strike length of individual lenses varies from 40 to 230 metres whilst the down dip extent varies from 30 metres to 200 metres. These lenses have greater down dip extents than the gold mineralisation hosted by sedimentary units. True thickness varies from 2 to 8 metres whilst the plunge varies from 10° to 50° southwest. Strikes of lenses of gold mineralisation hosted by basalt but with a spatial association with the basalt / sediment contact and the main zone in basalt vary in strike from 340° to 10° and dip 50° to 80° North. These lenses are at about 45 to the main zone and are interpreted to be a ladder vein system. **Funnelweb** The Funnelweb gold mineralisation is hosted by the Redback Basalt sequence and has a strike length of about 640 metres. The mineralisation has been described as discontinuous quartz veining within a sheared and altered basalt with decreasing grade and width where the shear crosscuts sedimentary units. Laterite gold mineralisation strikes at 025°, was flat lying and had a strike length of 270 metres. Widths varied from 5 to 55 metres while true thickness was up to 5 metres. This unit was mined, and no descriptions have been located. Primary gold mineralisation has strike lengths varying from 20 to 165 metres, down dip extents of 10 to 110 metres and true thickness of 1-2 to 10 metres. Plunges are generally flat but can vary with local geology. Historical grades average around 2.4 g/t Au while the overall strike length was of the order of 600 metres. **Money** The gold mineralisation at Money is hosted by the footwall of the Harley's sediments and the Footwall Basalt Complex. Mined grade was 3.5 g/t Au. The overall strike of the gold mineralisation was about 195 metres. Mined laterite gold mineralisation had a strike of 065° and a strike length of 110 metres, width from 20 to 60 metres and a true thickness of up to 5 metres. Primary gold mineralisation consists of a massive quartz vein with strikes of 065° to 070°, dips -65° to -70° southeast and plunges 10° to 18°. The strike lengths of individual lodes varied from 10 to 175 metres with down dip extents of 10 to 90 metres and true thickness of 10 to 20 metres. Gold mineralisation is terminated abruptly in the southwest and there appears to be limited exploration potential. **Harleys** The gold mineralisation at Harley's is developed along the intersection of 020° and 060° structural trends with the best grades being associated with massive quartz veining along the plunge direction. The gold mineralisation is largely hosted by the Harley's sediment sequence that consists of coarse-grained ferruginous greywacke that becomes silicified and bleached when mineralised. There was minor mineralisation in the basalt. Mined gold bearing laterite had a strike of 060° and was flat lying. The strike length was of the order of 160 metres and the width varied from 13 to 50 metres with a thickness of 2 to 3 metres. Primary gold mineralisation strikes between 030° to 050°, dips -25° to -75° southwest and plunges 0° to 24°. The strike lengths of

Criteria	JORC Code Explanation	Commentary
		individual lodes of gold mineralisation varies from 15 to 165 metres with down dip length varying from 10 to 75 metres. The true thickness varies from 2 to 15 metres. The overall strike length of the Harley's deposit was about 240 metres while the average mined grade was 4.1 g/t Au. Virtually gold mineralisation was taken in the mining process leaving little exploration potential. **Huntsman / Huntswoman** The Huntsman gold mineralisation is developed along a 060° trend and is hosted largely by the Harley's sediments. The mineralisation is bleached and silicified, pyritic quartz veins and stockworks within the sediment that carry in excess of 4 g/t Au. Gold mineralisation in basalt consists of narrow quartz veins that only carry 1 to 2 g/t Au. Mined gold bearing laterite at Huntsman open pit had a strike of 060° and was 10 to 65 metres wide with a true thickness between 2 and 10 metres. The average thickness was 5 metres. Primary gold mineralisation at Huntsman had flat plunges with strikes between 040° to 075° and dips -47° to -62° southeast. Individual strike lengths varied from 30 to 60 metres with down dip extents of 20 to 60 metres and true thickness of 2 to 12 metres. There is a major cross cutting fault just south of Huntsman that offsets this gold mineralisation from that of Huntswoman. Huntswoman also sits upon an 060° trend and is hosted largely by Harley's sediments. A mineralised shear on the 060° trend was mined down to a depth of 5 metres from Huntswoman to Katipo. This is likely to have been supergene or lateritic in nature. The lode had a strike of 160 metres and was 10 to 20 metres wide with an average thickness of 1 to 2 metres. Gold mineralisation below this horizon was likely to be shear related. Primary gold mineralisation at Huntswoman generally had flat plunges and varied in strike from 045° to 073° with dips between -25° to -75° southeast. Strike length varied from 30 to 120 metres with down dip extents of 10 to 170 metres but generally closer to 50 metres. True thickness varied from 2 to 15 metres. High grade gold mineralisation was associated with bleaching, silicification and pyritic quartz veins and stockworks. Both pre and post mineralisation faulting and high-grade gold mineralisation at depth have been noted. **Katipo** The gold mineralisation at Katipo is developed along a 060° trend and appears to be the strike extension of the Huntswoman mineralisation. The mineralisation is within the footwall side of the Redback Basalt Complex. The average grade from the Katipo open pit was 2.4 g/t Au. Flat lying gold bearing laterite was mined that had a strike length of about 150 metres, varied in width from 20 to 35 metres and had a true thickness 2 to 5 metres. Primary mineralisation consists of lodes that strike and dip between 060° to 084° and -65° to -80° southeast. The strike lengths vary from 20 to 100 metres, with down dip extents from 20 to 70 metres and true thickness that vary from 2 to metres. **Redback SE** The gold mineralisation at Redback SE is developed along a 065° to 070° trend and is hosted by Harley's sediments in the west and

Criteria	JORC Code Explanation	Commentary
		the Footwall Basalt Complex in the east. Higher grades were noted in sediments and discrete veins with lower grade were found in basalt. The average grade of mineralisation derived from Redback SE was 2.9 g/t Au. Gold bearing laterite or supergene mineralisation was mined in the eastern side of the open pit. Flat lying laterite or supergene had an overall strike of about 070° and varied in width from 10 to 40 metres while the true thickness was 2 to 4 metres. Below this mineralisation were 3 small lodes striking 065° to 075° and dipping -70° southeast. These lodes had strike length between 40 and 95 metres and down dip extents of 20 to 65 metres. The true thickness of these lodes varied from 2 to 10 metres. Gold mineralisation mined in the western side of the open pit consists of a main lode with a strike length of about 320 metres, down dip extent of up to 130 metres and a true thickness of 10 to 15 metres. The overall strike and dip of this lode is 065° and -60° to -70° southeast. The plunge is about 15° to 20° to the southwest. There are a number of smaller lodes that were mined in the open pit, but they have short strike lengths and may be splays from the main lode. **Redback SW** Gold mineralisation at Redback SW is developed along a 070° trend and has been interpreted to be along the same trend as Huntswoman. The gold mineralisation is hosted by Hurricane sediments in the western side of the pit and intercalated basalt and sediments becoming dominant in the rest of the pit. The average grade of gold mineralisation from the Redback SW was 2.6 g/t Au. The eastern end of the pit is now backfilled. Gold bearing laterite or supergene mineralisation was mined in the eastern portion of the pit. Flat lying laterite or supergene had an overall strike of 065°. The strike length was about 160 metres and varied in width from 15 to 40 metres while the true thickness was about 4 to 5 metres. The main lode at Redback SW strikes at 075° and dip -60° southeast and plunges about -10° southwest. The lode has a strike length of about 400 metres and a down dip extent of 130 metres and true thickness 5 to 10 metres. Other lodes in this area appear to be splays from the main lode. **Daddy** The gold mineralisation at Daddy is developed along an East West trend and was interpreted to be the southern extent of the Redback SE mineralised trend. The western end of the mineralisation was hosted by the Hurricane sediments while the eastern gold mineralisation was hosted by basalt with minor intercalated sediments. The strike of the gold lodes varied from 055° to 095° while the dip varied from -50° to -75° south. Plunges were generally flat. Strike lengths at Daddy vary from 15 to 110 metres while the down dip extent varies from 10 to 110 metres. True thickness of the gold mineralisation varies from 2 to 8 metres. **Phoenix and Inca** Phoenix gold mineralisation has a strike length of about 740 metres and is hosted mainly by basalt with lesser sandstone. Strikes of individual lodes of gold mineralisation vary from 055° to

Criteria	JORC Code Explanation	Commentary
		080° and dips -65° to -85° NW. The strike length of individual lodes varies from 15 to 200 metres while down dip extent varies from 15 to 130 metres. True thickness varies from 2 to 10 metres. The plunge of the gold mineralisation appears to be mostly flat however individual lodes can plunge up to -20° SW and one lode appears to be plunging -10° NE. Makar (2001) describes the southwestern zone of the open pit as being along the basalt / Hurricane sediment contact with little potential to be deepened. The northeast portion of the gold mineralisation is further described as being narrow vein quartz vein / stockworks up to 5 to 6 metres thick and discontinuous and being of low grade between 2 to 2.5 g/t gold. The Inca trend gold mineralisation has a discontinuous strike length of about 500 metres and is largely hosted by basalt but with a strong association with the contacts of sedimentary units. Strike of individual lodes varies from 15° to 40° and dips -50° to -75° NW. The strike length of individual lodes varies from 40 to 220 metres. The plunge of the gold mineralisation is generally flat but can be up to 10° SW. Makar (2001) describes the Inca trend as being a splay from the main 060° gold mineralisation trend. Further he describes the Inca structure as being offset by faulting and becoming two separate targets.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Redback area deposits. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. The Carbine block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 0.625m by 0.625m. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization. An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was

Criteria	JORC Code Explanation	Commentary
	• *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation (10-40, 20-80 and 40-160 metres). A first pass of radius 10-40m with a minimum number of samples of 2-6 samples and a second pass of radius 20-80m with a minimum number of 2-6 samples were used. A third pass of search radius 40-160m was used with a minimum number of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8-20 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass using nearest neighbour. Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above 0.6 g/t, 0.7 g/t, 0.7 g/t gold Cut-off in Oxide, Transitional, Fresh Rock within an optimised pit shell using AU$3,500/oz. Underground resources are reported within an AU$3,500/oz optimised stope below the open pit shell. This includes planned dilution.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	It is assumed the deposits will be mined by open pit and underground methods when a new mining operation can be established. This model is only suitable for open pit purposes although it can be used for a preliminary assessment of underground potential. Concentrate assumptions • Concentrate transport $370.04 per dry metric tonne of concentrate • Concentrate treatment and refining costs $147.58 per dry metric tonne of concentrate • Gold in concentrate payability 93% Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10% • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10%

Criteria	JORC Code Explanation	Commentary
		- Mine ROM to Mill ROM Haulages $0.10/t per km ore - Mill CIL Opex cost $35.46 per tonne - Mill floatation Opex cost additional $3.94 per tonne - Admin (G&A) $4.50 per tonne - CIL Processing Recovery 90% oxide, 76% 95% (Legs, Dogbolter, Lynx), 90% (Carbine, Inca, Phoenix, Redback Area) - CIL Processing Recovery transitional, 76% (Legs, Dogbolter, Lynx, Redback Area), 75% (Carbine, Inca, Phoenix) - CIL Processing Recovery fresh tailings 10.1% of total gold fresh tailings Floatation 85.1% of total gold recovered as concentrate - Processing cost contingency 10% - Au Price AU$3500 per troy ounce - Au Royalty 5.5% - Discount Rate 8% - Mining Rate 20 Mtpa rock - Carbine haulage 7.0 km - Dogbolter to Lynx haulage 12.6km - Legs haulage 10.9km - Phoenix haulage 7.0km - Redback Area haulage 10.3km Deswik Underground Stope Optimiser Assumptions *Minimum Mining Width 2.4 metres - Minimum Pillar Width 5 metres - Stope Strike Length 20 metres - Sub-level Interval 20 metres - Optimise grade or metal: grade - Stope Strike ±40 degrees - Stope Dip – Minimum 40 degrees - Sub Stope Shapes 2 U / 2 V - Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution - UG mining unplanned recovery 5% - UG mining unplanned dilution 5% - CIL Processing recovery 35% - UG Stoping Costs $75/tonne ore - UG Opex Fixed Cost $5/tonne ore - Mill Opex cost $39.40/tonne ore - ROM to Mill ROM Haulage $0.10/t per km ore - Carbine haulage 7.0 km - Dogbolter to Lynx haulage 12.6km - Legs haulage 10.9km - Phoenix haulage 7.0km - Redback Area haulage 10.3km - Admin (G&A) $4.50/tonne ore

TANAMI GOLD NL

Criteria	JORC Code Explanation	Commentary
		• Mining Cost Contingency 10% • Processing Cost Contingency 10% • Au Royalty 5.5% • Au Price AU$3500/troy ounce
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	There is very little metallurgy for the Legs deposit. During mining it was noted that the oxide zone had 95% recovery of gold while the transitional to fresh zones had <30% to 40% recovery. Further Makar (2001) noted that the gold was in the <5 µm range and was occluded with silicates, arsenopyrite and locked in pyrite and goethite. Metallurgy was carried out on samples from Redback SE, Redback SW, and Harley's Open Pits for gold recovery from weathered gold bearing ore in 1995. 8 weathered gold bearing ore composites from the Redback SE & SW mineralisation using optimum conditions for gold recovery were tested. The optimum leach conditions for weathered gold bearing ore from the Central desert Joint Venture were. • Grind P80=75 µm • Pulp density 40% solids • Leach time 20 hours • pH 10-10.5 • Initial cyanide strength 0.075% The optimum leach conditions for transition / fresh ore differed only in the leach time being extended to 24 hours. Under these conditions 90% of greater gold were achieved. A further study looked at recoveries at coarser grind sizes. Grinding was carried in a 200 mm diameter by 200 mm long stainless steel mill with an 11 kg rod charge. It was concluded that gold recoveries are relatively insensitive to the grind size at up to a P80 of 110 – 125 µm. Metallurgical test work on sediment saprolite clay from the Harley's pit resource was conducted in 1995. Ten samples from 8 RC drill holes for 78.6 intersection metres were received from site and subdivided to make a 78 kg composite for laboratory testing. The test procedure involved taking 4 kg of ore that was ground to 80% passing 75 microns and passed through a Knelson concentrator and collecting a concentrate about 2.5% by weight. The concentrate was examined microscopically and gold flakes up to 250 microns but mainly <50 microns were observed. Coarse gold was removed and assayed by amalgamation with mercury prior to recombining the concentrator and amalgamation tailings. 1kg of material was subsampled and leached. Gold bearing oxide material is likely to have 90 to 95% recovery in the Redback rise area however the majority of the economic ores have already been mined. No metallurgical data on transitional to fresh gold bearing mineralisation has been located. The closest metallurgical data from transitional and fresh gold bearing mineralisation is from the Legs deposit where it is likely that gold recoveries of 76% in transitional and about 35% in fresh rock would be achievable with a CIL plant. It is not known whether these recovery estimates are applicable to the area. Preliminary metallurgical testing MineScope Services Pty Ltd on historical diamond drill core at Central Tanami completed in 2025 suggests that using floatation, concentrate export and leaching of the flotation tails produces a total gold recovery of 95.7% in fresh rock. Further testing is still required.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td>Environmental factors or assumptions</td><td>Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.</td><td>No assumptions have been made regarding environmental factors.</td></tr>
<tr><td>Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>No bulk density data from the Daddy, Redback SW & SE, Katipo, Funnelweb, Huntsman, Huntswoman, Harleys and Money prospect could be located. The only bulk density data available was taken from the Open Pit Closure Reports that gave the average density for each pit.
<table>
<tr><th rowspan="2">Pit</th><th colspan="2">Ore</th><th rowspan="2">Bulk density</th></tr>
<tr><th>BCM</th><th>Tonnes</th></tr>
<tr><td>Money</td><td>89513</td><td>232250</td><td>2.59</td></tr>
<tr><td>Harleys</td><td>184048</td><td>466765</td><td>2.54</td></tr>
<tr><td>Funnelweb</td><td>44876</td><td>106367</td><td>2.37</td></tr>
<tr><td>Huntsman</td><td>72687</td><td>185155</td><td>2.55</td></tr>
<tr><td>Katipo</td><td>18787</td><td>43955</td><td>2.34</td></tr>
<tr><td>Redback SE</td><td>141340</td><td>327292</td><td>2.32</td></tr>
<tr><td>Redback SW</td><td>177974</td><td>427006</td><td>2.40</td></tr>
<tr><td>Daddy</td><td>80543</td><td>191842</td><td>2.38</td></tr>
</table>
Density values were derived from taking average values for sediments and basalt and adjusting for oxidation by RL to match the bulk density derived from the mining of ore from the open pits. These values may not be correct. It is recommended that further searches through the historical data to try and find the bulk density measurements.
No bulk density data from the Phoenix and Inca area could be located. The following table contains the values that were used.
<table>
<tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>gcc</th></tr>
<tr><td>BF</td><td>surface</td><td></td><td>2.2</td></tr>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.2</td></tr>
<tr><td rowspan="5">Sedimentary</td><td>440</td><td>415</td><td>2.2</td></tr>
<tr><td>415</td><td>405</td><td>2.3</td></tr>
<tr><td>405</td><td>395</td><td>2.4</td></tr>
<tr><td>395</td><td>360</td><td>2.5</td></tr>
<tr><td>360</td><td>290</td><td>2.6</td></tr>
</table>
</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary

	290	100	2.7
Basalt	440	405	2.3
	405	395	2.4
	395	360	2.5
	360	320	2.6
	320	300	2.7
	300	100	2.8

No bulk density data from the Legs prospect could be located. The following table contains the values that were used.

Rock type	RL		Bulk Density
	From	To	Gcc
WD	surface		2.2
TR	surface	380	2.2
Sedimentary	440	340	2.2
	340	330	2.3
	330	320	2.4
	320	300	2.5
	300	290	2.6
	290	180	2.7
Basalt	surface	400	2.1
	400	360	2.2
	360	340	2.3
	340	320	2.4
	320	300	2.5
	300	280	2.6
	280	180	2.7

No bulk density data from the Dogbolter to Lynx area could be located. Density values were derived from taking average values for sediments and basalt and adjusting for oxidation by RL. These values may not be correct.

Rock type	RL		Bulk Density
	From	To	Gcc
WD	surface		2.2
SP	surface		2.2
BF	surface		2.2
TR	surface	380	2.2
Sedimentary	440	380	2.2
	380	360	2.3
	360	340	2.4
	340	320	2.5
	320	290	2.6

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><td></td><td>290</td><td>100</td><td>2.7</td></tr>
<tr><td rowspan="8">Basalt</td><td>surface</td><td>400</td><td>2.1</td></tr>
<tr><td>400</td><td>380</td><td>2.3</td></tr>
<tr><td>380</td><td>360</td><td>2.4</td></tr>
<tr><td>360</td><td>340</td><td>2.5</td></tr>
<tr><td>340</td><td>320</td><td>2.6</td></tr>
<tr><td>320</td><td>300</td><td>2.7</td></tr>
<tr><td>300</td><td>100</td><td>2.8</td></tr>
</table>
There has been sporadic collection of bulk density data from the Carbine deposit but poor documentation of that data. During mining of the oxide, transitional and partial fresh zones the average bulk density was reconciled at 2.5. Previous workers) have applied bulk density values by RL. Given the lack of documented data the same approach was taken in the current model.
<table>
<tr><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>g/cc</th></tr>
<tr><td>380</td><td>Surface</td><td>2.1</td></tr>
<tr><td>360</td><td>380</td><td>2.2</td></tr>
<tr><td>340</td><td>360</td><td>2.3</td></tr>
<tr><td>330</td><td>340</td><td>2.4</td></tr>
<tr><td>320</td><td>330</td><td>2.5</td></tr>
<tr><td>300</td><td>320</td><td>2.6</td></tr>
<tr><td>290</td><td>300</td><td>2.7</td></tr>
<tr><td>-300</td><td>290</td><td>2.8</td></tr>
</table>
</td></tr>
<tr><td>Classification</td><td>

The basis for the classification of the Mineral Resources into varying confidence categories.
Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
Whether the result appropriately reflects the Competent Person’s view of the deposit.

</td><td>The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the ‘Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves’ by the Joint Ore Reserves Committee (JORC).

The Mineral Resource was classified as Measured, Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20-25m by 20-25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Validation of the block models show good correlation of the input data to the estimated grades where there were sufficient composites for kriging to be effective.

The result reflects the competent person’s view that the classification is Measured, Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>

The results of any audits or reviews of Mineral Resource estimates.

</td><td>No audits or reviews of this estimate have been conducted.</td></tr>
<tr><td>Discussion of relative</td><td>

Where appropriate a statement of the relative accuracy and confidence level in the Mineral

</td><td>The Redback Area Mineral Resource Estimates has been reported with a low to moderate degree of confidence.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
accuracy/ confidence	*Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. Mineral Resource Estimates have previously been estimated and reported for the Daddy, Redback SW & SE, Katipo, Funnelweb, Huntsman, Huntswoman, Harleys and Money deposits. The current published resource for Redback Rise area is combined and listed under MLS167 These figures are derived from 2011 and is from a MineMap inverse distance squared model. The 2011 MRE is unconstrained by economics. The last published resource for Redback Rise area from Otter Gold Mines (OGM) was in June 2001. OGM constrained open pit resources to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. The Katipo, Huntsman, Huntswoman, Harley's and Money deposits are not mentioned in the 2001 resource statement and are only mentioned in reconciliation data. These prospects are included in the 2011 resource under the Redback Rise Region where no economic parameters were applied. The economically constrained resource from 2001 at greater than 0.5 g/t Au was 332,000 tonnes @ 3.53 g/t Au for 37.3k ounces of gold. The economically unconstrained resource from 2011 at greater than 0.7 g/t Au was 1,045,198 tonnes @ 2.87 g/t Au for 96.3k ounces of gold. The Funnelweb Open Pit was mined from July 1997 to April 1998 and produced 106,367 tonnes @ 2.25 g/t Au for 7,687 ounces of gold. The pit was mined to a depth of 45 metres. This pit produced almost 3,000 ounces less than the OGM model predicted. The Money Open Pit was mined from April 1997 to January 1998 and produced 232,250 tonnes @ 3.63 g/t au for 27,086 ounces of gold. The pit was mined to an RL of 358 metres or 62 metres depth. The pit produced about 5,000 ounces more than the OGM model predicted. The Harley's Open Pit was mined from November 1996 to April 1998 and produced 466,765 tonnes @ 3.69 g/t Au for 55,394 ounces of gold. The pit was mined to a depth of 94 metres or the 335 m RL. No reconciliation data has been located. The Huntsman / Huntswoman pits were mined from February 1996 to July 1997 and produced 185,155 tonnes @ 4.14 g/t Au for 24,664 ounces of gold. The pits were mined to a depth of 55 metres or the 374 m RL. This was 9,000 ounces more than the OGM model predicted. The Katipo Open Pit was mined from July 1997 to March 1998 and produced 43,955 tonnes @ 2.39 g/t Au for 3,373 ounces of gold. The pit was mined to a depth of 33 metres or the 386 m RL. 3,373 ounces of gold were produced versus the model prediction of 2,043 ounces. The Redback SE Open Pit was mined from December 1995 to December 1997 and produced 327,292 @ 2.93 g/t Au for 30,817 ounces of gold. The pit was mined to a depth of 77 metres or the 342 m RL. The predicted ounces from the OGM model were 24,293 ounces.

Criteria	JORC Code Explanation	Commentary
		The Redback SW Open Pit was mined from November 1995 to May 1997 and produced 427,006 tonnes @ 2.63 g/t Au for 36,106 ounces of gold. The pit was mined to a depth of 80metres or the 336 m RL. 36,106 ounces were produced versus 32,582 ounces predicted from the model. The Daddy Open Pits were mined December 1997 to April 2000 and produced 191,842 tonnes @ 2.11 g/t Au for 13,006 ounces of gold. The pits were mined down to 62 metres in the west and 35 metres in the east or 345 m RL and 376 m RL respectively. 13,006 ounces were produced versus 13,997 ounces predicted from the model. The current model was compared to the actual production data and reconciled grade and ounces using a low grade cut off of 0.8 g/t Au and the mined field. The overall result was that the September 2023 model was within 4.1% of actual tonnages, -3.4% on grade and 0.8% of ounces of gold. The Phoenix and Inca area Mineral Resource Estimate has been reported with a moderate degree of confidence. The Measured Mineral resource was confined to areas below the mined open pit where the geology was well established and there was close spaced drilling. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 20m by 20m (with some infill), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. Mineral Resource estimates have previously been estimated and reported for the Phoenix deposit. The current published resource is combined and listed under MLS167. These figures are derived from Makar (2011) and are from a MineMap inverse distance squared model. The 2011 MRE is unconstrained by economics. The last published resource for Phoenix s from Otter Gold Mines (OGM) was in June 2001 (Weedon, 2001). OGM constrained open pit resources to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. The Phoenix Open Pit was mined by OGM from April 1997 to May 1998 and produced 214,657 tonnes @ 2.36 g/t Au for 16,268 ounces of gold. The open pit was mined to a depth of 60 metres. The Legs Mineral Resource Estimate has been reported with a low to moderate degree of confidence. Mineral Resource estimates have previously been estimated and reported for the Legs deposit. The current published resource for Legs is combined and listed under MLS167 These figures are derived from Makar (2011) and is from a MineMap inverse distance squared model. The 2011 MRE is unconstrained by economics. The last published resource for Legs from Otter Gold Mines (OGM) was in June 2001 (Weedon, 2001). OGM constrained open pit resources to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. The Legs Open Pit was mined by OGM from May 1998 to May 2001 and produced 1,077,731 tonnes @ 3.51 g/t Au for 121,621 ounces of gold. The open pit was mined down to the 304 mRL. The current model at a low grade cut off of 0.67 g/t Au produces 980,559 tonnes @ 3.7 g/t Au for 116,298 ounces. The shortfall in tonnes and ounces can be explained by not all of the smaller lodes in the mined area were modelled and the current model does not take

Criteria	JORC Code Explanation	Commentary
		mining dilution into account. This result is within 5% of ounces and 10% of tonnes which is an acceptable result. the 304 m RL. The Dogbolter to Lynx area Mineral Resource Estimate has been reported with a moderate degree of confidence. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. Mineral Resource estimates have previously been estimated and reported for the Dogbolter, Bulldog and Lynx deposits. The current published resources are combined and listed under MLS167 These figures are derived from Makar (2011) and is from a MineMap inverse distance squared model. The 2011 MRE is unconstrained by economics. The last published resource for this area from Otter Gold Mines (OGM) was in June 2001 (Weedon, 2001). OGM had an underground resource for Dogbolter and open pit resources for Lynx and Bulldog prospects. OGM constrained open pit resources to pit shells (Lerch-Grossman algorithm) using a AU$750 gold price and 0.5 g/t Au low grade cut off. The Dogbolter Main Open Pit was mined by OGM from January 1996 to November 1998 and produced 722,362 tonnes @ 4.15 g/t Au for 96,456 ounces of gold. The open pit was mined down to the 290 m RL. Makar (2001) commented that there were geotechnical concerns along the eastern wall due to west dipping sediments and the south, north and west walls had collapses due to intense weathering coupled with intense jointing and shearing. Further, Makar (2001) states they were good recoveries in the sulphides although they were very hard and required blending. The current model for Dogbolter Main Open Pit at a low grade cut off of 0.8 g/t Au produces 710,578 tonnes @ 4.15 g/t Au for 94,695 ounces. This result is within 2% of tonnes and grade and 1% of actual ounces produced. This is considered to be a good reconciliation between production and the current model. Dogbolter NE and NE Junior were mined by OGM from July 1996 to November 1997 and produced 154,651 @ 4.25 g/t Au for 21,134 ounces of gold. The open pits were mined down to the 350m RL and 358m RL respectively. Makar (2001) states that both pits were not mined to the final design due to pit wall failures along the southern walls. The current model for Dogbolter NE and NE Junior at a low grade cut off of 0.8 g/t Au produces 134,898 tonnes @ 5.05 g/t Au for 21,897 ounces of gold. The tonnes are 12.8% short of actual mined tonnes but the grade from model is 19% higher. The produced ounces of gold are within 4%. This is considered to be a reasonable reconciliation between the current model and actual production. It is likely that the differences in tonnages and grade may have been due to dilution. The Carbine Mineral Resource Estimate has been reported with a moderate degree of confidence. The Indicated Mineral Resource is based upon 25 by 20 metre RC drilling of acceptable quality. It is assumed that the mineralisation in this area is continuous between drill sections. The project is in area of previous mining. The Carbine Open pit was mined from August 1997 through to December 2000 and produced

Criteria	JORC Code Explanation	Commentary
		1,050,658 tonnes @ 2.66 g/t Au for about 90k ounces. The waste to ore ratio was 16.8 to 1 and the pit made AU$2.4 million profit (Makar, 2001a) The low grade cut off was 0.8 g/t Au. Weedon (2001) states that the reconciled production from the Carbine Open Pit as June 2001 was 1,010,885 tonnes @ 2.69 g/t Au for 87.4K ounces. It is not known why there is a difference between Makar (2001a) and Weedon (2001). The current model for the mined area produces 932,096 tonnes @ 2.86 g/t Au for 85.6K oz. If dilution is considered, the tonnes and grade are very similar to the actual production results. The main risks associated with the mining operation is the geotechnical stability of the pit walls and the lower recoveries in the fresh sulphide mineralisation. Approximately 10,000 tonnes @ 4 g/t Au of blasted oxide ore in the Carbine North ramp goodbye cut was lost due to the east pit wall collapse The Mineral Resource statement relates to global estimates of tonnes and grade.

TANAMI GOLD NL

Appendix 5 - JORC Table 1
ML(S)168

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) core drilling. Sampling of RC chips was completed on RC drillholes, and half core sampling on diamond drillholes was completed.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled. Diamond drilling used a combination of HQ and NQ2-sized core. HQ core was drilled until competent ground was intersected, then NQ2 core was drilled. Drill core was oriented, aligned, and half-cut using metre intervals and geologically determined intervals (max 1.2 metres and min 0.3 metres), with geologically determined intervals taking precedence.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	1m RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole. For RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter. This generated a 2 to 4 kg sample. Where wet samples were encountered the entire sample was collected in a 40-litre bucket before being tipped into discreet piles. A scoop sample was taken from wet samples. Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or similar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample.

Criteria	JORC Code Explanation	Commentary
		RC recovery in the completed campaigns were considered consistent. DD core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists. For RC holes drilled between 1994 to 2001 no data is available on sample recoveries.
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by geologists. The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks. This was checked and compared to the core measurements by the geological team. Any issues were communicated back to the drilling contractor, and necessary adjustments were made.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged by geologists at the drill rig to a high level of detail to support resource estimation, mining studies and metallurgical studies. RC logging is undertaken on a metre by metre basis at the time of drilling. Geologists log DD core to industry standards. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples are logged for lithology, alteration, mineralisation. Logging is a mix of qualitative and quantitative observations. Visual estimates are made of sulphide, quartz, and alteration as percentages. RC samples are not photographed. All DD logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and diamond core hole was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference.
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. The Central Tanami Gold Joint Venture (Otter and Acacia) during the early 1990s, collected samples at 1 metre intervals via a rig-

Criteria	JORC Code Explanation	Commentary
		mounted cyclone and collected into plastic bags. All holes were originally sampled on a 3-metre composite using a PVC spear to obtain a 2kg sample. The CTP collected samples at 1m intervals at the rig, representing the cutting's coarse fraction. For CTP drillholes, all samples were taken at 1-metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site. RC holes drilled in the mid-1990s to 2001 samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter. RC drill holes drilled by Tanami Gold between 2010 to 2011 samples were collected on a one metre basis through a 75:25% riffle splitter and placed into pre-numbered sample bags. Northern Star Stage-1 RC drilling saw all bulk material collected on a 1m basis directly from cyclone in pre labelled green plastic mining bags. Northern Star Stage-2 RC drilling saw single metre (1m) samples collected from a trailer mounted static cone splitter. Approximately 12.5% of each meter sample was collected in a pre-labelled calico bag with the depth while the remaining 87.5% was collected in a green mining bag and retained. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	During mining operations drill samples were prepped either at onsite or at ALS in Alice Springs to industry standards. The Otter Gold Mines data does include some onsite analysis at the mine laboratory. Northern Sar sample preparation was conducted at ALS Perth, commencing with sorting, checking and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	Grind checks are performed at both the crushing stage (3mm) and pulverising stage (75µm), requiring 90% of the material to pass through the relevant size.
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate and to industry standard. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 20 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology, and assay value ranges for gold.

Criteria	JORC Code Explanation	Commentary
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples collected during mining operations were submitted to the onsite laboratory or ALS in Alice Springs. Analysis (both on and off-site) was by AAS with selective FA checks. It should be noted that all onsite analysis was performed with a 20ml aliquot, whereas ALS used a 50ml aliquot for all AAS readings. Samples collected by Northern Star were sent to ALS in Malaga, Perth. Gold (Au) concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC Chip sample collection.
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	For holes drilled between 1994 to 2001 Analysis (both on and offsite) was by AAS with selective FA checks. The onsite procedure incorporates the inclusion of a check sample, quartz wash and a standard sample per batch of 30 samples. On a monthly basis one pulp per shift (ie two per day) was selected and analysed offsite by AAS and Fire assay for repeatability. Additional check samples were selected and assayed offsite as required by the geological staff. The Northern Star QAQC protocols used include the following for all drill samples: • Field QAQC protocols used for all drill samples include commercially prepared certified reference materials (CRM) inserted at an incidence of 1 in 20 samples. The CRM used is not identifiable to the laboratory with QAQC data is assessed on import to the database and reported monthly, quarterly and yearly. • NSR RC Resource definition drilling routinely inserts field blanks and monitor their performance. • Laboratory QAQC protocols used for all drill samples include repeat analysis of pulp samples occurs at an incidence of 1 in 20 samples, and screen tests (percentage of pulverised sample passing a 75µm mesh) are undertaken on 1 in 40 samples. • The laboratories' own standards are loaded into the database, and the laboratory reports its own QAQC data monthly. • Blanks were routinely inserted into the sample sequence at a rate of 1 per 25 samples and again specifically after potential or existing high-grade mineralisation to test for contamination. Failures of blanks above 0.2g/t were followed up, and re-assayed. New pulps were prepared if failures continued.

Criteria	JORC Code Explanation	Commentary
		• Failed standards are generally followed up by re-assaying a second 30g pulp sample of all samples in the fire above 0.1ppm by the same method at the primary laboratory. The accuracy component (CRMs and third-party checks) and the precision component (duplicates and repeats) of the QAQC protocols are thought to demonstrate acceptable levels of accuracy and precision.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	All significant intersections were verified by Geologists on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled for this data set.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data is either entered directly or imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Assay files are received in .csv format and loaded directly into the SQL acQuire database by geologists or database administrators. Hardcopy and electronic copies of the data is stored for future reference. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates. A systematic procedure utilising several re-assays and/or check assays are employed to determine if/when the first (primary) gold assay is changed for the final assay.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Planned drillholes were sited either with a handheld global positioning system (GPS) or a differential global positioning system (DGPS), and the initial drillhole pickup is usually with a handheld GPS, as well, with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm. During drilling, single-shot surveys were taken every 30m to ensure the hole remains close to the design. Down-hole surveys were performed using Boart Longyear TruCore, Axis Champ Ori, or similar equipment., recording the down-hole dip and magnetic azimuth. These results were then uploaded into the database.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition, or a DGPS elevation with an accuracy of ± 10mm is used.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions were good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider-spaced drilling or insufficient drilling in smaller lodes.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.

Criteria	JORC Code Explanation	Commentary
	Reserve estimation procedure(s) and classifications applied.	
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	The chain of custody of samples was managed by geologists and geotechnicians. Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at the on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email or uploaded to an FTP site. Sample pulp splits are stored for a time at the laboratory. Retained pulp packets are returned to the Central Tanami Mine for storage.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Geologists have undertaken internal reviews of applied sampling techniques and data. The completed reviews raised no issues.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	Jims and Camel Bore Gold Deposits are located in the Tanami Region in the Northern Territory on Mineral Lease (Southern) MLS168, approximately 23km southwest of the Central Tanami Mill site. MLS168 covers an area of 711.9ha and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108km2 tenement area in the Tanami Region held by the CTPJV are registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. Mineral Leases have a 25-year life and are renewable for 25 years. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.

Criteria	JORC Code Explanation	Commentary
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	MLS168 is granted and in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Jims and Camel Bore area has been explored since the early 1990's. Several previous companies, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area. Drilling reported with this release is contiguous with the Jims open-cut mine. Previous drilling at this project adds gold grade and geological context to the subsequent Northern Star Resources interpretation of the area as tested by the drill holes covered by this report. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The Jims and Camel Bore deposits are Palaeoproterozoic, basalt and sediment-hosted vein-mineralised deposits that is part of the Granites-Tanami Inlier. Gold mineralisation is controlled by a brittle fracture system associated with regional-scale structures that crosscut a regional-scale southeast, shallowly plunging anticline. Mineralisation occurs within a series of vein and breccia lodes developed near basalt-sediment contacts.
Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length*	Not applicable
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	No holes are excluded from this report.
data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. In the reporting of exploration results, results are reported as weighted averages using a nominal 0.5 g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied.
	• *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. Any high-grade zones above 15g/t gold within a reported intercept are also reported as included intervals.
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.

Criteria	JORC Code Explanation	Commentary
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends
	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Gold mineralisation is sub-vertical to vertical.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	Only downhole lengths have been reported. True widths are not known.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Appropriate plans and sections have been included.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Planned drillholes are sited with a handheld global positioning system (GPS), and the initial drillhole pickup is usually with a handheld GPS, as well; with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths. All intercepts for all holes have been reported regardless of grade.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	Upon receipt of all results, a review of the drilling completed is required before further work is planned.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records. There are some minor overlap errors in the RC and diamond drill holes where 4 metre samples overlapped later 1 metre samples but the occurrence was not significant
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	The competent person, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd has visited site has made a number of visits to the Tanami JV area.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to good as there are exposures in the open pit and it is based upon RC and diamond drill holes and grade control data from mining. There are also geophysical images to help guide the interpretation. Mineralisation was based upon close spaced grade control sections within the open pit. The interpretation from this data was used to extrapolate gold mineralisation into areas of exploration drilling. Previous interpretations only used the exploration data. This resulted in quite differing interpretations. The Jims deposits are located mostly on the north-eastern side of an interpreted north-northwest trending regional fault. The mineralisation is hosted by pillow and undifferentiated basalt intercalated with minor sediments. Prior to mining documentation by Makar (2001) suggests that the area had an intact regolith profile with a lateritic cap. The mineralised trend at Jims main pit has been described by Makar (2001) as striking North-South with flexures and dipping moderate to steep west in the upper extent but changes to steep to east dipping below the 320m RL. The Camel Bore deposit strikes about 330° and dips steeply. The 330° structural trend is interpreted to be a shear zone. The local geology consists of intercalated basalt, sandstone, and siltstone.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	**Jims Gold Deposit** The main ore zone has a true thickness of 15 to 25 metres but has areas up to 60 metres thick. The strike length of the Jims Main mineralisation is of the order of 300 metres and is hosted by basalt. The mineralisation has been interpreted down to 500 metres below the surface or 395 metres down dip. Further Makar (2001) states that secondary ore zones identified by mining indicate moderate to flat east dipping, limited strike length lenses that are generally lower grade +1.5g/t Au, but can contain narrow discrete high grade +5g/t Au pods. These were found in the

Criteria	JORC Code Explanation	Commentary
		southern areas of the open pit and decreased in the number of zones with depth. The gold mineralisation has a sharp boundary with sheared zones. Alteration associated with mineralisation consists of sericite, carbonate, chlorite, silicification and pyrite. The gold mineralisation was re-interpreted in March 2025 using the close spaced grade control data. The addition of this data added considerably more geological detail and showed that the mineralisation has a northerly plunge. Drilling conducted in 2022 through to 2024 was also added and showed the mineralisation has a northerly plunge of -20° to -25° North. The mineralisation at Jims Central appears to be the northern strike extension of the Jims Main mineralisation. The mineralisation has a strike of about 200 metres and is 2 to several metres thick and has been interpreted to a depth of 150 metres below the surface. The open pit was abandoned in depleted zone due to not achieving the predicted tonnes or grade that the model predicted. Makar (2001) states that prior to stopping the pit costeans were dug across the mineralised zones and the walls were mapped and sampled. The results indicated that high grade gold was associated with thin flat lying quartz veins stacked at intervals of +1 metre with subgrade between the veins. These appear to have been ladder veins in between shear zones. Jims Central mineralisation was re-interpreted with this knowledge to determine whether any further mineralisation can be economically extracted. Jims West is adjacent to the current waste dump and occurs close to the north-northwest striking regional fault. Mineralisation is striking about North-South and dips approximately 45 degrees West. The strike length of Jims West is of the order of 150 metres with true thickness between 1 – 7 metres and individual lenses have been interpreted up to 120 metres down dip. The area has not previously been mined. **Camel Bore Gold Deposit** Camel Bore gold mineralisation has a strike length of about 460 metres and is hosted by sandstone and siltstone with minor basalt. Strikes of individual lenses of mineralisation vary from 315° to 330° and dip steeply east and west. The strike length of individual lenses of gold mineralisation varies from 25 to 200 metres but are more typically 50 to 60 metres. True thickness varies from 1 to 2 metres to several metres. The down dip extent is typically of the order of 50 metres and plunges vary from about 20° to the northwest to 25° to the southeast. Makar (2001) describes the ore within the open pit as being contained by two discreet steeply plunging shoots dipping vertical to steeply east. Further, Makar (2001) states that the best grades are down plunge and are erratic along strike that required close spaced trenching and RC drilling. No descriptions of the mineralisation have been located but it is assumed that it is similar to the Jims mineralisation that is located 2 km to the southeast. There appears to be a close association between the mineralisation and contacts between basalt and sediment.

Criteria	JORC Code Explanation	Commentary
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by CTPJV and checked by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CV's, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Jims deposit. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block models used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. **Jims** Three expanding passes were used in the estimation (15-50, 30-100 and 60-200 metres). A first pass of radius 15-30m with a minimum number of samples of 2-6 samples and a second pass of radius 30-100m with a minimum number of 2-6 samples were used for Jims. A third pass of search radius 60-200m was used with a minimum of 2-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4-14 depending on the number of samples in the domain. **Camel Bore** Three expanding passes were used in the estimation (20-25, 40-50 and 80-100 metres). A first pass of radius 20-25m with a minimum number of samples of 2-6 samples and a second pass of radius 40-50m with a minimum number of 4-6 samples were used for Camel Bore. A third pass of search radius 80-100m was used with a minimum of 2-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4-28 depending on the number of samples in the domain. Blocks that did not fill after 3 passes were given a 4^{th} pass using nearest neighbour estimation.

Criteria	JORC Code Explanation	Commentary
		Selective mining units were not modelled. The block size used in the resource models was based on drill sample spacing and lode orientation. To validate the models, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource Estimates have been constrained by the wireframed mineralised envelopes, are undiluted by external waste and reported above 0.7 g/t, 0.6 g/t, 0.6 g/t gold cut-off in Oxide, Transitional & Fresh within an optimised pit shell using AU$3,500/oz. Underground resources are reported within an optimised stope below the open pit shell. Underground tonnes and grade include planned dilution in the stope optimisation
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	It is assumed the Jims and Camel Bore deposits will be mined by open pit and underground methods when a new mining operation can be established. Both models are only suitable for open pit purposes although they can be used for a preliminary assessment of underground potential. The following mining factors and costs were used for the Deswik optimisation of the open pit and underground resource: Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10% • Pit Slopes – Oxide 39° • Pit Slopes – Other 45° • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10% • Mine ROM to Mill ROM Haulages $0.10/t per km ore • Mill Opex cost $35.46 per tonne • Admin (G&A) $4.50 per tonne • CIL Processing Recovery 76% oxide, 95% transitional, 92% fresh • Processing cost contingency 10% • Au Price AU$3500 per troy ounce • Au Royalty 5.5% • Discount Rate 8% • Mining Rate 20 Mtpa rock • Jims haulage 23.9 km

Criteria	JORC Code Explanation	Commentary
		• Camel Bore haulage 23.6 km Deswik Underground Stope Optimiser Assumptions • *Minimum Mining Width 2.4 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 20 metres • Optimise grade or metal: grade • Jims Stope Strike ±50 degrees • Camel Bore Stope Strike ±40 degrees • Stope Dip – Minimum 40 degrees • Sub Stope Shapes 2 U / 2 V • Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • CIL Processing recovery 92% • Jims UG Stoping Costs $88/tonne ore • Camel Bore UG Stoping Costs $73/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $35.46/tonne ore • Mine ROM to Mill ROM transport $0.10/t per kmore • Jims haulage 23.9 km • Camel Bore haulage 23.6 kmAdmin (G&A) $4.50/tonne ore • Mining Cost Contingency 10% • Processing cost contingency 10% • Au Royalty 5.5% • Au Price AU$3500/troy ounce
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	Metallurgical testing was carried out in 1993 by Laurie Smith & Associates and AMDEL. Sighter test work was carried from a 12 metre intersection in JRC043 that had an average grade of 2.94 g/t Au. 25% of the of the gold was recovery by gravity concentration but it was noted that it was locked up in heavy particles. Further they noted that Leach kinetics were also good and gold recovery was of the order of 96% with much of the extraction in the first 8 hours. Leach tests indicated that Jims weathered low and high grade had slow leaching times and after 40 hours recovery was 76% and 78%. The other 3 samples, (Jims Mottled Zone, Jims transitional and Jims Primary) after 40 hours were 93 to 95%. The following recoveries were assigned to the block model. **Material Type** / **Recovery%** Oxide / 83 Transitional / 94.6 Fresh / 92.9

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>There is no metallurgy for the Camel Bore deposit. The closest deposit with metallurgical data is Jims which is located 2 km to the southeast.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.</td><td>No assumptions have been made regarding environmental factors.</td></tr>
<tr><td>Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>No bulk density data from the Jims prospects could be located. Density values were estimated by taking average bulk densities for basalt and sedimentary units and adjusting for weathering and matching the average density for each open pit. These values may not be correct.
<table>
<tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>gcc</th></tr>
<tr><td>WD</td><td>surface</td><td></td><td>2.2</td></tr>
<tr><td>BF</td><td>surface</td><td></td><td>2.2</td></tr>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.2</td></tr>
<tr><td rowspan="7">Sedimentary</td><td>440</td><td>420</td><td>2.2</td></tr>
<tr><td>420</td><td>405</td><td>2.2</td></tr>
<tr><td>405</td><td>390</td><td>2.3</td></tr>
<tr><td>390</td><td>375</td><td>2.4</td></tr>
<tr><td>375</td><td>360</td><td>2.5</td></tr>
<tr><td>360</td><td>345</td><td>2.6</td></tr>
<tr><td>345</td><td>-100</td><td>2.7</td></tr>
<tr><td rowspan="6">Basalt</td><td>440</td><td>405</td><td>2.2</td></tr>
<tr><td>405</td><td>390</td><td>2.3</td></tr>
<tr><td>390</td><td>375</td><td>2.4</td></tr>
<tr><td>375</td><td>360</td><td>2.5</td></tr>
<tr><td>360</td><td>345</td><td>2.6</td></tr>
<tr><td>345</td><td>-100</td><td>2.7</td></tr>
</table>
No bulk density data from the Camel Bore prospect could be located. The only density value that was found was for the entire Camel Bore open pit that averaged 2.18 tonnes per metre3. This was derived from the pit closure report.</td></tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>Density values were derived from taking average values for sediments and basalt and adjusting for oxidation by RL to try and match the average pit value. The following values were applied.
<table>
<tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr>
<tr><th>From</th><th>To</th><th>gcc</th></tr>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.2</td></tr>
<tr><td rowspan="6">Sedimentary</td><td>440</td><td>340</td><td>2.2</td></tr>
<tr><td>340</td><td>330</td><td>2.3</td></tr>
<tr><td>330</td><td>320</td><td>2.4</td></tr>
<tr><td>320</td><td>300</td><td>2.5</td></tr>
<tr><td>300</td><td>290</td><td>2.6</td></tr>
<tr><td>290</td><td>180</td><td>2.7</td></tr>
<tr><td rowspan="7">Basalt</td><td>surface</td><td>400</td><td>2.1</td></tr>
<tr><td>400</td><td>360</td><td>2.2</td></tr>
<tr><td>360</td><td>340</td><td>2.3</td></tr>
<tr><td>340</td><td>320</td><td>2.4</td></tr>
<tr><td>320</td><td>300</td><td>2.5</td></tr>
<tr><td>300</td><td>280</td><td>2.6</td></tr>
<tr><td>280</td><td>180</td><td>2.7</td></tr>
</table></td></tr>
<tr><td>Classification</td><td>The basis for the classification of the Mineral Resources into varying confidence categories.
Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
Whether the result appropriately reflects the Competent Person's view of the deposit.</td><td>The Mineral Resource estimates are reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC).

The Mineral Resources were classified as Measured, Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity.

The Measured Mineral Resource is located below Jims Main Open Pit and has already been grade controlled drilled in part.

The Indicated Mineral Resources were defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. The Inferred Mineral Resources were assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Validation of the block model shows good correlation of the input data to the estimated grades.

The result reflects the competent person's view that the classification is Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>The results of any audits or reviews of Mineral Resource estimates.</td><td>No audits or reviews of this estimate have been conducted.</td></tr>
<tr><td>Discussion of relative</td><td>Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed</td><td>The Jims and Camel Bore Mineral Resource Estimates have been reported with a moderate degree of confidence.

The Measured Mineral Resource is located below Jims Main Open Pit and has already been grade controlled drilled in part.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
accuracy/ confidence	*appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	The Indicated Mineral Resources were defined within areas of RC and diamond drilling of 25m by 25m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. Jims Main was successfully mined from the 30th January 1998 to 25th June 2001 and produced 1,383,585 @ 2.62 g/t Au for 116,386 ounces. A block model report for this area from ctp_jims_Mar2025.mdl results in a resource that was mined of 1.43 million tonnes @ 2.95 g/t Au for 127,921 ounces. This is within 4% of tonnes mined and 5.5% of grade and 9% of total ounces produced. Once ore loss and dilution are considered this result is acceptable. Jims Central produced 3,069 tonnes @ 2.67 g/t Au for the period from 10th June 1998 to 1st April 1999 for 263 ounces. The pit was abandoned because tonnes and ounces were not reconciling with the model. A block model report with a low grade cut off from 1.1 g/t Au from ctp_jims_Mar2025.mdl produces 1,942 tonnes @ 1.94 g/t Au for 121 ounces. This zone is just below the depletion zone noted in Makar (2001) and Anon (2001b) and variable results can be expected. The Camel Bore Open Pit was mined by Otter Gold Mines from September 2000 to May 2001 and produced 72,000 tonnes @ 2.45 g/t Au for 5,678 ounces of gold. The open pit was mined down to the 340 m RL. The production figures were sourced from the pit closure report. Makar (2001) stated that tonnes and grade were achieved in the upper pit levels, but poor reconciliations occurred in the bottom 30 metres due to the ingress of groundwater. The groundwater hampered grade control and mining resulting in high dilution and ore loss. The open pit produced 5,678 ounces compared to the predicted 8,008 ounces by Otter Gold modelling. This was largely due to an achieved grade of 2.45 g/t Au versus the predicted grade of 3.45 g/t Au. The current model returns a grade of 3.49 g/t au but only 59,638 tonnes for 6,695 ounces for the mined area It appears that the RC drilling may be overestimating the grade of the gold mineralisation or there is significant low grade that was incorporated in the model. The Mineral Resource statement relates to global estimates of tonnes and grade.

Appendix 6 - JORC Table 1
ML(S)180 & EL26925

Section 1 Sampling Techniques and Data
(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) core drilling. Sampling of RC chips was completed on RC drillholes, and half core sampling on diamond drillholes was completed.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.2-2.0m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes, the device is unknown.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	DD core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists.
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by geologists.

Criteria	JORC Code Explanation	Commentary
		The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No evaluation has been carried to date, to determine if a relationship exists between sample recovery and grade or if bias may have occurred due to preferential loss/gain of fine/coarse material
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged by geologists at the drill rig to a high level of detail to support resource estimation and mining studies. RC logging is undertaken on a metre-by-metre basis at the time of drilling. Geologists log DD core. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples were logged for lithology, alteration, mineralisation. Logging was a mix of qualitative and quantitative observations. Visual estimates were made of sulphide, quartz, and alteration as percentages. RC samples were not photographed. All DD logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and diamond core hole was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.2-2.0m in length honouring lithological boundaries.
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	No historic record has been located to describe the method used to sub-sample non-core samples and the method used if the sample was wet
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	No historic record has been located to describe the quality control procedures adopted for all sub-sampling stages
	• *Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.*	No historic record has been located to describe the measures taken to ensure that the sampling is representative of the in-situ material collected
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology and assay value ranges for gold.
Quality of assay data	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples collected during the 1990s were analysed by AAS with selective FA checks with a 20ml aliquot. It is unknown where the samples were analysed.

Criteria	JORC Code Explanation	Commentary
and laboratory tests	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	The laboratory procedure during the 90s to early 2000s is unknown.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	All significant intersections were verified by Geologists on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	No twinned holes were drilled
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data is either entered directly or imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Assay files are received in .csv format and loaded directly into the SQL acQuire database by geologists or database administrators. Hardcopy and electronic copies of the data is stored for future reference. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates. A systematic procedure utilising several re-assays and/or check assays are employed to determine if/when the first (primary) gold assay is changed for the final assay.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	During drilling, single-shot surveys were taken every 30m to ensure the hole remains close to the design. Down-hole surveys were performed using Boart Longyear TruCore, Axis Champ Ori, or similar equipment., recording the down-hole dip and magnetic azimuth. These results were then uploaded into the database.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52 for holes drilled between 2010 and 2012. The original holes drilled by Otter Gold Mines from 1990 to 2001 were picked up by site surveyors in either the Molech or Orion grids. These grids were well established and converted to MGA94 zone 52.
	• *Quality and adequacy of topographic control.*	All open pits had been surveyed in either the local Molech or Orion grids. These were converted to MGA94 Zone 52 in Surpac software using parameters established during mining. Topographic control was established using the drill hole collars that were surveyed during mining and converted to MGA94 zone 52.
	• *Data spacing for reporting of Exploration Results.*	Drillhole spacing across the area varies from 20 by 20 metres or closer where grade control drilling has taken to broader spacing of 50 by 50 metres.

Criteria	JORC Code Explanation	Commentary
Data spacing and distribution	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	No historic record has been located that outlines the measures taken to ensure sample security
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	No historic record has been located that details the results of any audits or reviews of sampling techniques and data

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Molech Gold Deposits, Beaver, Orion, Bonsai, Banjo and Cheeseman are located in the Tanami Region in the Northern Territory on Mineral Lease (Southern) 180 ("MLS180"), whilst the Pendragon Gold Deposit is located on Exploration Licence 26925 ("EL26925"). Collectively the deposits are located approximately 35km west of the Central Tanami Mill site. MLS180 covers an area of 803.6ha and EL26925 60 blocks (190.01 km^2) and are registered to Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. They form part of the 2,108km^2 Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Resources Limited. The Central Tanami Project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	MLS180 and EL26925 are granted and in good standing

Criteria	JORC Code Explanation	Commentary
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Molech area has been explored since the mid 1980's. Numerous companies, including Zapopan NL, Otter Gold NL, Normandy Mining Ltd, Newmont (Asia Pacific), and Tanami Gold NL have been active in the area.
Geology	• *Deposit type, geological setting and style of mineralisation.*	**Banjo** The Banjo deposit is hosted by sandstone, mudstone, chert and basalt from the Mt Charles Formation. Geological interpretations from drill logging, aeromagnetic data and pit mapping suggest that the basalt and sediments are striking about 272° and dipping about -80° South. A 340° trending shear transects the local stratigraphy and has been described as being about 40 metres wide. **Beaver** The Beaver deposit is hosted by intercalated mudstone, siltstone, sandstone, coarse grained volcanoclastic units and undifferentiated basalt from the Mt Charles Formation (Thomson, 2012). Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 315° and dipping steeply. Mapping from the open pit describes the lithology as thick sequence of mudstone to siltstone that strike 315° and dip 70° South. **Bonsai** The Bonsai deposit is hosted within a 290° trending shear zone that transect basalt and interbedded siltstone and sandstone from the Mt Charles formation. Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 280° to 335°, dipping steeply and display several fault offsets. **Cheeseman** The Cheeseman deposit is hosted by regional shear that generally trends at about 340°. In the Cheeseman area there is an apparent inflection in this zone where the shear changes from about 330° to 320°. The host rocks consist of basalt and siltstone and sandstone. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Interpreted basalt outside of the shear has an apparent strike of between 1° to 20° and is steeply dipping. Within the shear the basalt has an apparent strike that is parallel to the shear zone. **Orion** The Orion deposits are hosted by a regional shear that generally trends between 325° to 340° and is interpreted to be the same structure that hosts Banjo in the south and Cheeseman in the north. The local geology consists of siltstone, sandstone, and basalt with minor felsic units. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Basalt outside

Criteria	JORC Code Explanation	Commentary
		of the shear strikes at about 330° and has apparent steep dip and is 50 to 60 metres thick. Basalt within the shear is discontinuous and is up to 15 to 20 metres thick and steeply dipping. **Pendragon** The Pendragon deposit is hosted within a 300° trending shear zone that transect basalt and interbedded siltstone and sandstone from the Mt Charles formation. This appears to be a similar structure to the shear that hosts the Bonsai deposit.
Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length*	This report pertains to the reporting of Mineral Resources. Exploration results have previously been reported to the ASX by the various previous explorers.
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	This report pertains to the reporting of Mineral Resources. Exploration results have previously been reported to the ASX by the various previous explorers
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	This report pertains to the reporting of Mineral Resources. Exploration results have previously been reported to the ASX by the various previous explorers
	• *Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralization.
	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Mineralisation structures are vertical to sub-vertical.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this*	When exploration results were previously disclosed, only downhole lengths were reported. True widths are not known

Criteria	JORC Code Explanation	Commentary
	effect (e.g. 'down hole length, true width not known').	
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Appropriate diagrams accompany this report.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Surveys were conducted by Otter Gold NL in local grids by qualified surveyors during the period 1999 to 2001. The data has been transformed to MGA94 Zone 52 and the locations coincide with surveys flown in 2008 and with drilling conducted in 2024.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths. All intercepts for all holes have been reported regardless of grade.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Exploration results have previously been regularly reported to the ASX by the previous various explorers.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	No further work is currently planned until mining recommences at Central Tanami.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate Diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table.

Criteria	JORC Code Explanation	Commentary
	used.	• Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records. There are some minor overlap errors in the RC and diamond drill holes where 4 metre samples overlapped later 1 metre samples, but the occurrence was not significant
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	The competent person, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining has made a number of visits to the Tanami JV area
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to good as there are open pit exposures and it is based upon RC and several diamond drill holes. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. The local pit geology was georeferenced in Surpac and used to help interpret the geology. The Molech and Pendragon deposits are hosted by mudstone, sandstone, mudstone, chert, volcanoclastic units and basalt from the Mt Charles Formation. **Banjo Deposit** Geological interpretations from drill logging, aeromagnetic data and pit mapping suggest that the basalt and sediments are striking about 272° and dipping about -80° South. A 340° trending shear transects the local stratigraphy and has been described as being about 40 metres wide. The southern portion of the open pit was mostly basalt while the northern part is dominated by sediments. The mapping shows that there are numerous faults cutting the stratigraphy. Gold mineralisation is hosted by the shear and has been described as associated with intense silicification and pyrite. The best grades were located within the sedimentary units and only low-grade pods were encountered in the basalt. Mining within the Banjo open pit concentrated on a north plunging lens of gold mineralisation. **Beaver Deposit** Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 315° and dipping steeply. Mapping from

Criteria	JORC Code Explanation	Commentary
		the open pit describes the lithology as thick sequence of mudstone to siltstone that strike 315° and dip 70° South. Total production from the Beaver open pit was 536,225 tonnes @ 3.33 g/t Au for 57,381 ounces of gold. Most of this production came from two dominant re structures locally named the main and east lodes. Both lodes are offset by cross cutting faults with a displacement of 15 to 20 metres. The main lode is terminated in the south by a 290° trending structure that hosts the Bonsai mineralisation. Gold mineralisation appears to be smeared along this structure and was noted to be erratic in grade and weakens with distance from the main lode. The southern part of the main lode is described as greater than 2 metres and consisting of quartz veining and quartz stockworks within a 20-metre-wide altered shear. The strike length is of the order of 210 metres. Visible gold was noted in the quartz veining. The northern vein as a massive quartz vein up to 1.5 metres true thickness within a 5 to 8 metre wide shear. This lode has a strike of about 110 metres.' The east lode has a strike length of about 160 metres in the south and 100 metres in the north. The southern vein has been described as greater than 1 metre true thickness of massive quartz vein and stockworks within a 5 to 8 metre wide shear zone. Visible gold was also noted. The northern vein of the East lode is only 0.5 metre wide within a 2 to 3 wide shear. The best gold grade was found in the volcanoclastic sediment. **Bonsai Deposit** The Bonsai deposit is hosted within a 290° trending shear zone that transect basalt and interbedded siltstone and sandstone. Geological interpretations of drill logging and aeromagnetic data suggest that the basalt and sediments are striking about 280° to 335°, dipping steeply and display several fault offsets. Near surface mineralisation at Bonsai consisted of laterite or supergene gold mineralisation. Very few descriptions of this material have survived, and it has been largely mined. The strike of the supergene was about 200 metres with a width of 20 metres and a true thickness of up to 3 metres. The bedrock mineralisation at the Bonsai deposit is hosted by sheared basalt and to a lesser degree within the margins of the sheared basalt / sediment contact within a major 290° trending shear zone. The better grades are associated with silicification with quartz stockwork and veins.

Criteria	JORC Code Explanation	Commentary
		Modelling of the gold at a 0.5 g/t Au low grade cut off shows that mineralisation has been defined over a strike length of 810 metres in a zone that is up to 120 metres wide. The mineralisation is discontinuous and many of the lodes have an apparent southeast plunge. Strike lengths vary between 20 to 300 metres, The steeply dipping lodes have limited down dip extents and vary from 20 to 130 metres with true thickness of 1-2 metres to 7-8 metres. **Cheeseman Deposit** The Cheeseman deposit is hosted by regional shear that generally trends at about 340°. In the Cheeseman area there is an apparent inflection in this zone where the shear changes from about 330° to 320°. The host rocks consist of basalt and siltstone and sandstone. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Interpreted basalt outside of the shear has an apparent strike of between 1° to 20° and is steeply dipping. Within the shear the basalt has an apparent strike that is parallel to the shear zone. An intact regolith profile was encountered in the Cheeseman deposit and the near surface mineralisation consisted of laterite or supergene hosted gold. The mineralisation in the supergene is flat lying and has a strike length of up to 75 metres and a width between 5 to 45 metres and a true thickness of 1 to 8 metres. The strike of the southernmost supergene lodes is between 335° to 340° whilst the northern supergene lodes strike between 315° to 330° and are much smaller in size. The total production from the Cheeseman open pit was 59,136 tonnes @ 3.91 g/t Au for 7,486 ounces of gold. Most of the production ore was derived from enriched laterite cap directly above a high-grade quartz vein. The pit was mined down to a bleached, silty – talc material with high grade gold associated with ferruginous nodules. Primary gold mineralisation consists of south plunging auriferous quartz veins hosted by sandstones and siltstones. There is a spatial relationship between the mineralisation and the contacts between the sediment and basalt. The strike of the veins ranges from 30 to 100 metres whilst the dip extent ranges from 25 to 80 metres. The true thickness of the veins is generally 1 to 2 metres but can be up to 8 metres. The quartz veins south of 7794850mN strike at about 330° whilst the veins north of this strike between 315° and 325°. **Orion Deposit** The Orion North and South deposits are hosted by a regional shear that generally trends between 325° to 340° and is interpreted to be the same structure that hosts Banjo in the south and Cheeseman in the north. The local

Criteria	JORC Code Explanation	Commentary
		geology consists of siltstone, sandstone, and basalt with minor felsic units. Basalt noted in the drill hole logging were used as marker units to interpret the geology. Basalt outside of the shear strikes at about 330° and has apparent steep dip and is 50 to 60 metres thick. Basalt within the shear is discontinuous and is up to 15 to 20 metres thick and steeply dipping. Gold mineralisation at the Orion deposit is hosted by a 40 metre wide 325° to 340° trending shear near the contact of basalt and sedimentary units. Mineralisation at Orion North strikes at between 325° to 335° and dips -80° East but smaller lodes do vary. Individual lodes vary in strike length from 10 to 170 metres and have true thickness from 1-2 to 10 metres. The down dip extent varies from 10 to 100 metres. There is minor supergene mineralisation. The Orion North mineralisation occurs in an inflexion in the shear. High grade gold zones were associated with increasing quartz veins and stockworks within a bleached and silicified basalt. Gold mineralisation at Orion South strikes at about 330°, dips steeply and has an apparent plunge to the south. Strike lengths vary between 10 and 80 metres and have true thickness from 1-2 to 9 metres. The down dip extent varies from 10 to 60 metres. During mining the open pit did not correspond well with model and high grade intercepts received in the RC drilling were not reproduced. **Pendragon Deposit** The Pendragon deposit is hosted within a 300° trending shear zone that transect basalt and interbedded siltstone and sandstone from the Mt Charles formation. This appears to be a similar structure to the shear that hosts the Bonsai deposit. Gold mineralisation at Pendragon is hosted within a 300° trending shear. The interpretation used all available drilling to interpret the mineralisation as the drill spacing was broad. The lithology that hosts the mineralisation is mostly sandstones and siltstone with a spatial association with basalt. Strike lengths vary between 50 to 200 metres with true thickness varying from 1-2 to 8 metres and down dip extents from 20 to 90 metres. The strike of individual lenses varies between 300° to 320° and dips are near vertical.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	**Banjo** The overall strike of the gold mineralisation within the model is 860 metres. Bedrock mineralisation is steeply dipping with many lenses vertical. There are four small supergene flat lying lenses. The steep dipping lenses have strike lengths and down dip extents of up to 100 metres but typically they are shorter. True thickness varies from 1-2 metres to 7-8 metres. The open pit has been mined down

Criteria	JORC Code Explanation	Commentary
		to about 70 metres below the surface with mineralisation in the base of the pit. Mineralisation has been interpreted down to 150 metres below the surface. **Beaver** The southern part of the main lode at Beaver has a strike of about 210 metres and varies from 2 to 8 metres wide and extends down dip about 220 metres. The northern part of the main lode has a strike of 110 metres and has a true thickness of at least 1.5 metres to several metres and extends down dip 150 metres. The east lode has a strike length of about 160 metres in the south and 100 metres in the north. The southern vein has been described as greater than 1 metre true thickness of massive quartz vein and stockworks. The northern vein of the East lode is only 0.5 metre wide. The down dip extent is about 150 metres. The Beaver Open Pit has been mined down to 110 metres below the surface. Mineralisation is exposed in the base of the pit and a small lens crops out near surface. The mineralisation has been interpreted down to 220 metres below the surface. **Bonsai** The strike of the supergene was about 200 metres with a width of 20 metres and a true thickness of up to 3 metres. Modelling of the steep dipping gold at a 0.5 g/t Au low grade cut off shows that mineralisation has been defined over a strike length of 810 metres in a zone that is up to 120 metres wide. The mineralisation is discontinuous and many of the lodes have an apparent southeast plunge. Strike lengths vary between 20 to 300 metres, The steeply dipping lodes have limited down dip extents and vary from 20 to 130 metres with true thickness of 1-2 metres to 7-8 metres. The open pit was mined down to 50 metres below the surface. Mineralisation is exposed in the base of the pit. Many of the lodes are near surface and the deepest mineralisation has been extended to 150 metres below the surface. **Cheeseman** Supergene mineralisation is flat lying and has a strike length of up to 75 metres and a width between 5 to 45 metres and a true thickness of 1 to 8 metres. The strike of the southernmost supergene lodes is between 335° to 340° whilst the northern supergene lodes strike between 315° to 330° and are much smaller in size. Primary gold mineralisation consists of south plunging auriferous quartz veins. The strike of the veins ranges from 30 to 100 metres whilst the dip extent ranges from 25 to 80

Criteria	JORC Code Explanation	Commentary
		metres. The true thickness of the veins is generally 1 to 2 metres but can be up to 8 metres. The open pit was mined down to 25 metres below the surface. The pit is now backfilled. Mineralisation is near surface and has been interpreted down to 150 metres below the surface. **Orion** Gold mineralisation at the Orion deposit is hosted by a 40 metre wide 325° to 340° trending shear. Mineralisation at Orion North strikes at between 325° to 335° and dips -80° East but smaller lodes do vary. Individual lodes vary in strike length from 10 to 170 metres and have true thickness from 1-2 to 10 metres. The down dip extent varies from 10 to 100 metres. Gold mineralisation at Orion South strikes at about 330°, dips steeply and has an apparent plunge to the south. Strike lengths vary between 10 and 80 metres and have true thickness from 1-2 to 9 metres. The down dip extent varies from 10 to 60 metres. The Orion North Pit was mined down to 55 metres below the surface while the Orion South Pit was mined down to 35 metres. The Orion South Pit is now backfilled. Mineralisation is exposed in the base of the Orion North Pit and has been interpreted down to 120 metres below the original surface. The top of insitu mineralisation is interpreted to be 7 to 20 metres below the current surface. **Pendragon** Gold mineralisation at Pendragon is hosted within a 300° trending shear. Strike lengths vary between 50 to 200 metres with true thickness varying from 1-2 to 8 metres and down dip extents from 20 to 90 metres. The strike of individual lenses varies between 300° to 320° and dips are near vertical. The top of the mineralisation is about 7 metres below the surface and has been interpreted down to 100 metres.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CVs, and

Criteria	JORC Code Explanation	Commentary
	estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	summary statistics) using Supervisor software. Top cuts were done on a lode basis and prior to estimation. Reconciliations with the production from each open pit was also attempted. The mining cut-off grade used by OGM was 1.1 g/t gold and it appears that the total production from the area was about 0.97 Mt @ 3.01 g/t gold for 94.2 kOz. The current models at 1.1 g/t Au give 0.76 Mt @ 3.69 g/t Au for 90.1 kOz. Estimated ounces from the current models are within 2.2% of the claimed production and if dilution was around 10-15% then the estimations reconcile well, however there are too many assumptions to be confident. The largest error was in the Beaver open pit where claimed production was 57,381 ounces of gold versus 52,619 ounces of gold from the current model. The difference can be accounted for in the southern end of open pit where the mineralisation was interpreted to be striking about 310 in the Bonsai shear zone. This area was bulk mined. The current model used all available data to interpret the mineralisation and it suggested numerous NE trending short strike length, southwest plunging discontinuous lenses with waste separating them within the Bonsai shear zone. Not all lenses could be wireframed dur to the uncertainty in the geometries. The current estimate in in line with the Tanami Gold NL April 2010 MRE that was constrained by economic parameters. The Molech April 2010 MRE was 1.0 Mt @ 3.25 g/t Au for 109 kOz. This does not include the Pendragon resource. The Tanami Gold NL Molech January 2011 MRE is 1.68 Mt @ 2.78 g/t Au but this is unconstrained by economics. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Molech & Pendragon deposits. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction. QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization

Criteria	JORC Code Explanation	Commentary
		An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Search ellipses and the minimum and maximum number of samples were lode dependent and varied considerably. A first pass search radius of 25 to 50 metres with a minimum number of samples of 2-6 samples and a second pass radius of 50 to 100 metres with a minimum number of 2-6 samples were used. A third pass search radius of 100-200m was used with 2-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 3-26 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass using nearest neighbour estimation. Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the lodes. This analysis was completed for northings and elevations across each deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.6g/t gold cut-off grade for open pit material within a $AU3500 pit shell. The underground resource is reported within a $AU3500 optimised stope and is diluted by waste.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the*	It is assumed the Molech and Pendragon deposits will be mined by open pit and underground methods when a new mining operation can be established. This model is only suitable for open pit purposes although it can be used for a preliminary assessment of underground potential. Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10% • Pit Stopes – Oxide 39°

Criteria	JORC Code Explanation	Commentary
	assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Pit Slopes – Other 45° • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10% • Mine ROM to Mill ROM Haulages $0.10/t per km ore • Mill Opex cost $35.46 per tonne • Admin (G&A) $4.50 per tonne • CIL Processing Recovery 88% oxide, 88% transitional, 90% fresh • Processing cost contingency 10% • Au Price AU$3500 per troy ounce • Au Royalty 5.5% • Discount Rate 8% • Mining Rate 20 Mtpa rock • Banjo haulage 36.3km • Beaver haulage 36.3km • Bonsai haulage 36.3km • Cheeseman haulage 39.9km • Orion haulage 38.2km • Pendragon haulage 38.2km Deswik Underground Stope Optimiser Assumptions • *Minimum Mining Width 2.4 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 20 metres • Optimise grade or metal: grade • Stope Strike ±40 degrees • Stope Dip – Minimum 40 degrees • Sub Stope Shapes 2 U / 2 V • Smoothing None • *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • CIL Processing recovery 90% • UG Stoping Costs $75/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $35.46/tonne ore • ROM to Mill transport $3.63/tonne ore • Admin (G&A) $4.50/tonne ore • NT Factor (10%) $12.36/tonne ore • Au Royalty 5.5% • Au Price AU$3500/troy ounce

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td>Metallurgical factors or assumptions</td><td>The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.</td><td>Metallurgical testing was carried out in August 1998 on samples from 3 RC holes PGRC075, PGRC080 and PGRC081. The extract depth downhole of the samples is unknown but comments are provided of the type of material that was tested. The location of the test work is the main lode of the Beaver open pit. The metallurgical test work may not be representative of the entire Molech area. The recoveries ranged from 84.6 to 97%.

There has been 6 open pits mined in the Molech area, and the recoveries are thought to be in this range however no mill recovery data has been located.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.</td><td>No assumptions have been made regarding environmental factors. The area has been previously mined during the late 1990s through to 2001. Haul roads, open pit and waste dumps still exist from this period.</td></tr>
<tr><td>Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc),</td><td>There is no bulk density data from the Molech Gold deposits. The only bulk density data is from the open pit closure reports where average bulk density for the entire pit was back calculated.

Density values were taken from average densities for sedimentary rocks and basalts and adjusted for RL to account for oxidation. These values may not be correct. It is recommended that diamond holes are completed to gather representative bulk density measurements.
<table><tr><th rowspan="2">Rock type</th><th colspan="2">RL</th><th>Bulk Density</th></tr><tr><th>From</th><th>To</th><th>Tonnes m^3</th></tr></table></td></tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td>moisture and differences between rock and alteration zones within the deposit.
• Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td><table>
<tr><td>TR</td><td>surface</td><td>380</td><td>2.20</td></tr>
<tr><td rowspan="7">Sedimentary Felsic Volcanic</td><td>440</td><td>360</td><td>2.20</td></tr>
<tr><td>360</td><td>340</td><td>2.30</td></tr>
<tr><td>340</td><td>330</td><td>2.40</td></tr>
<tr><td>330</td><td>320</td><td>2.50</td></tr>
<tr><td>320</td><td>300</td><td>2.60</td></tr>
<tr><td>300</td><td>290</td><td>2.70</td></tr>
<tr><td>290</td><td>180</td><td>2.80</td></tr>
<tr><td rowspan="8">Basalt</td><td>surface</td><td>400</td><td>2.10</td></tr>
<tr><td>400</td><td>380</td><td>2.20</td></tr>
<tr><td>380</td><td>360</td><td>2.30</td></tr>
<tr><td>360</td><td>340</td><td>2.40</td></tr>
<tr><td>340</td><td>320</td><td>2.50</td></tr>
<tr><td>320</td><td>300</td><td>2.60</td></tr>
<tr><td>300</td><td>290</td><td>2.70</td></tr>
<tr><td>290</td><td>180</td><td>2.80</td></tr>
</table>At this stage of the project, it is assumed that these values will be close to the real values.</td></tr>
<tr><td>Classification</td><td>• The basis for the classification of the Mineral Resources into varying confidence categories.
• Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
• Whether the result appropriately reflects the Competent Person's view of the deposit.</td><td>The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC). The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. No Measured Resource was categorised due to no actual bulk density data, no QAQC data, open pits finishing early due to geotechnical issues, water ingress and loss of the detailed mining history. The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25 by 25 metre, where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined. Validation of the block model shows good correlation of the input data to the estimated grades.
The result reflects the competent person's view that the classification is Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>• The results of any audits or reviews of Mineral Resource estimates.</td><td>Reviews of this estimate have been conducted by Northern Star Resources and Tanami Gold NL geologists.</td></tr>
<tr><td>Discussion of relative</td><td>• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource</td><td>There are a number of assumptions used in the modelling the Molech & Pendragon MRE that affects the confidence of the MRE.</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
accuracy/ confidence	*estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	No QAQC data is available to assess the quality of the sampling and assaying. The data should be used with caution. Comments during mining stated that grade control with ditch witch trenching could not reproduce the grades obtained by RC drilling. Grades estimated in modelling by OGM were not achieved when mining and only Beaver and Cheeseman Open Pits made a profit. No bulk density measurements have been taken at the Molech deposits. The bulk density values applied were from averages for basalt and sediment and adjusted for oxidation by RL. These may not be correct. Metallurgical testing has only been carried out on 3 RC holes from the main lode at the Beaver deposit and may not be reflective of the rest of the mineralisation. No geotechnical studies have been completed to determine the mining parameters. The area is known for open pit wall failures and water inflows. Some of the resources may not be mineable. The classification of transitional material as an underground resource is high risk as ground conditions may be too unstable to mine the material using underground methods. The Beaver and Bonsai open pits were abandoned due to pit wall failures and water ingress in the early 2000s. Ore was left in the base of the Open Pits. It may not be possible to safely mine any of the Molech area by open pit methods without laying down the walls and increasing the strip ratio. During mining of the Banjo, Beaver and Bonsai deposits by OGM, tonnages of ore from the models versus production was reasonable however the grade recovery was 75%, 91% and 88% respectively. This reflected in the ounces produced. Mining of Cheeseman and Orion North deposits returned far more tonnes at a lower grade than the models predicted but the ounces were as predicted. Mining of Orion South returned on 64% of the tonnages predicted and 60% of the ounces.

Appendix 7 - JORC Table 1
EL28282

Section 1 Sampling Techniques and Data
(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Sampling was completed using reverse circulation (RC) and diamond (DD) core drilling. Sampling of RC chips was completed on RC drillholes, and half core sampling on diamond drillholes was completed.
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	RC metres intervals are defined by paint markings on the rig. The larger split or sample reject is left at the sample pad to indicate metres drilled. Diamond drilling used a combination of HQ and NQ2-sized core. HQ core was drilled until competent ground was intersected, then NQ2 core was drilled. Drill core was oriented, aligned, and half-cut using metre intervals and geologically determined intervals (max 1.2 metres and min 0.3 metres), with geologically determined intervals taking precedence.
	• *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	1m RC samples were collected from a cone splitter on the rig, in a calico bag. The sample/bulk ratio was 12.5/87.5. Sample weights ranged between 1kg and 4kg, although sample weight/size are ideally uniform, at least within a drillhole. For RC holes drilled in the 1990s samples were taken at 1 metre intervals from the cyclone and manually fed through a riffle splitter Sampling of DD drillholes was completed using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. Sample weights are typically between 0.5kg and 3kg, mostly dependent on length, however sometimes dependent on lithology.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	RC Drilling was completed using a 5.25" face sampling hammer drill bit. Diamond core was completed using a combination of HQ and NQ2 size drill bits and oriented where possible using the bottom dead centre technique. Deviation surveys were completed on all drillholes using Boart Longyear TruCore, or Axis Champ Ori equipment, or similar. Single Shot Surveys were completed at 30m intervals during drilling, and a continuous in/out survey was completed at the end of the hole.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.*	Approximate RC recoveries are sometimes recorded as percentage ranges based on a visual and/or weight estimate of the sample.

Criteria	JORC Code Explanation	Commentary
		RC recovery in the completed campaigns were considered consistent. DD core was reconstructed into continuous runs with depths checked against core blocks. Core recoveries were recorded as a percentage and calculated from measured core versus drilled intervals by geologists.
	• *Measures taken to maximise sample recovery and ensure representative nature of the samples.*	Experienced RC drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by geologists. The diamond drill contractors adjusted their drilling rate and method if recovery issues arose. All recovery was recorded by the drillers on core blocks. This was checked and compared to the core measurements by the geological team. Any issues were communicated back to the drilling contractor, and necessary adjustments were made.
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	No relationship was noted between RC sample recovery and grade. The consistency of the mineralised intervals suggests sampling bias due to material loss or gain is not an issue. No relationship was noted between core recovery and grade. The consistency of the mineralised intervals suggests that sampling bias due to material loss or gain is not an issue.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All RC holes were logged by geologists at the drill rig to a high level of detail to support resource estimation, mining studies and metallurgical studies. RC logging is undertaken on a metre-by-metre basis at the time of drilling. Geologists log DD core to industry standards. All relevant features such as lithology, structure, texture, grain size, alteration, oxidation state, vein style and veining percentage per interval, and mineralisation were recorded in the geological logs.
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.*	RC samples were logged for lithology, alteration, mineralisation. Logging was a mix of qualitative and quantitative observations. Visual estimates were made of sulphide, quartz, and alteration as percentages. RC samples were not photographed. All DD logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed.
	• *The total length and percentage of the relevant intersections logged.*	The entire length of each RC and diamond core hole was logged.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*	Diamond drill core was cut in half using a diamond core saw. Half core was sampled on intervals between 0.3-1.2m in length honouring lithological boundaries. The right-hand side of the core was bagged as the primary sample for analyses. The remaining half of the core was archived and stored for reference.
	• *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.*	RC drillholes were sampled either using a cyclone rotary splitter mounted on the RC drill rig, from an approximate 12.5% split off the bulk reject, or samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.

Criteria	JORC Code Explanation	Commentary
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate. Northern Star drilling samples were prepared at ALS Perth, commencing with sorting, checking, and drying at less than 110°C to prevent sulphide breakdown. Samples were jaw crushed to a nominal -6mm particle size. If the sample is greater than 3kg, a Boyd crusher with a rotary splitter is used to reduce the sample size to less than 3kg at a nominal <3mm particle size. The entire crushed sample (if less than 3kg) or sub-sample is then pulverized to 90% passing 75µm, using a Labtechnics LM5 bowl pulveriser. 300g Pulp subsamples are then taken with an aluminium scoop and stored in labelled pulp packets. An informal analysis suggests that the sampling protocol currently in use is appropriate to the mineralisation encountered and should provide representative results.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	Grind checks are performed at both the crushing stage (3mm) and pulverising stage (75µm), requiring 90% of the material to pass through the relevant size.
	• *Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.*	The sample preparation is considered appropriate. Field duplicates for RC drilling are routinely analysed at a rate of 1 in 20 samples. No Field duplicates were submitted for diamond core sampling.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	Sample sizes are considered appropriate to represent the style of mineralisation, the thickness and consistency of the intersections, the sampling methodology and assay value ranges for gold.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples collected during the 1990s were analysed by AAS with selective FA checks with a 20ml aliquot. It is unknown where the samples were analysed. Samples collected by Northern Star were sent to ALS in Malaga, Perth. Gold (Au) concentration was determined by ICP-AAS (Atomic Adsorption Spectrometry), after conventional Lead Button Fusion and HCl/HNO3 digestion of a 50g charge sample, with at least 170g of litharge-based flux at the ALS Malaga facility. This was common to both Diamond Core and RC Chip sample collection.
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc..*	No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	The laboratory procedure during the 90s is unknown. The Northern Star QAQC protocols used include the following for all drill samples: • Field QAQC protocols used for all drill samples include commercially prepared certified reference materials (CRM) inserted at an incidence of 1 in 20 samples. The CRM used is not identifiable to the laboratory with QAQC

Criteria	JORC Code Explanation	Commentary
		data is assessed on import to the database and reported monthly, quarterly and yearly. • NSR RC Resource definition drilling routinely inserts field blanks and monitor their performance. • Laboratory QAQC protocols used for all drill samples include repeat analysis of pulp samples occurs at an incidence of 1 in 20 samples and screen tests (percentage of pulverised sample passing a 75µm mesh) are undertaken on 1 in 40 samples. • The laboratories' own standards are loaded into the database and the laboratory reports its own QAQC data monthly. • Blanks were routinely inserted into the sample sequence at a rate of 1 per 25 samples and again specifically after potential or existing high-grade mineralisation to test for contamination. Failures of blanks above 0.2g/t were followed up, and re-assayed. New pulps were prepared if failures continued. • Failed standards are generally followed up by re-assaying a second 30g pulp sample of all samples in the fire above 0.1ppm by the same method at the primary laboratory. The accuracy component (CRM's and third-party checks) and the precision component (duplicates and repeats) of the QAQC protocols are thought to demonstrate acceptable levels of accuracy and precision.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.*	All significant intersections were verified by Geologists on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC.
	• *The use of twinned holes.*	Two twin holes were completed in the 2019 Northern Star drilling campaign, SJRC0005 twinned CDH007 and SJRC0006 twinned CDH008.
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	Primary data is either entered directly or imported into a SQL acQuire database using semi-automated or automated data entry; hard copies of core assays and surveys are stored at site. Assay files are received in .csv format and loaded directly into the SQL acQuire database by geologists or database administrators. Hardcopy and electronic copies of the data is stored for future reference. Visual checks occur as a result of regular use of the data.
	• *Discuss any adjustment to assay data.*	The first (primary) gold assay is almost always utilised for any resource estimation, except where evidence from re-analysis or check analysis dictates. A systematic procedure utilising several re-assays and/or check assays are employed to determine if/when the first (primary) gold assay is changed for the final assay.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Planned drillholes were sited either with a handheld global positioning system (GPS) or a differential global positioning system (DGPS), and the initial drillhole pickup is usually with a handheld GPS, as well, with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm. During drilling, single-shot surveys were taken every 30m to ensure the hole remains close to the design. Down-hole surveys were performed using Boart Longyear TruCore, Axis Champ Ori, or

Criteria	JORC Code Explanation	Commentary
		similar equipment., recording the down-hole dip and magnetic azimuth. These results were then uploaded into the database.
	• *Specification of the grid system used.*	Collar coordinates were recorded in MGA94 Zone 52.
	• *Quality and adequacy of topographic control.*	Topographic control was established through detailed aerial and ground survey control from airborne survey acquisition, or a DGPS elevation with an accuracy of ± 10mm is used.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	Drillhole spacing across the area varies; The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 40m by 40m (with some 25 by 25 metre infill and twinning), where the continuity and predictability of the lode positions was good, and the estimation had reasonable slopes of regression.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied.
	• *Whether sample compositing has been applied.*	No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	The chain of custody of samples was managed by geologists and geotechnicians. Geologists or geotechnicians transport core and RC samples to the admin/mine site; the drill core is logged, cut, and sampled at the on-site core shed. Samples were bagged in tied numbered calico bags, grouped in larger tied polyweave plastic bags, and placed in large bulka bags with sample submission sheets. The bulka bags were sent by road freight to the laboratory. Field personnel involvement ceased at this stage. The results of analyses were returned via email or uploaded to an FTP site. Sample pulp splits are stored for a time at the laboratory. Retained pulp packets are returned to the Central Tanami Mine for storage.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	Geologists have undertaken internal reviews of applied sampling techniques and data. The completed reviews raised no issues.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Crusade deposit is located in the Tanami Region in the Northern Territory in the Northern Territory on Exploration Licence (Supplejack) EL28282, approximately 100km northeast of the Central Tanami Mill site. EL28282 covers an area of 101.07km2 and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,108km2 tenement area in the Tanami Region held by the CTPJV are registered jointly in Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. The CTPJV comprises six Exploration Licences, four of which are granted, and two applications, three Mineral Lease (Southern) and two Mineral Leases. Mineral Leases have a 25-year life and are renewable for 25 years. The Central Tanami project area lies on Aboriginal land within the Central Desert Aboriginal Land Trust and the Mt Frederick Aboriginal Land Trust, both administered by the Central Land Council.
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	EL28282 is granted and in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	The Crusade area has been explored since the mid 1990's. Several companies, including Newmont (Asia Pacific) and Tanami Gold NL have been active in the area. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The Crusade deposit is a Paleoproterozoic, mafic-hosted vein-mineralized deposit that is part of the Granites-Tanami Inlier. Mineralisation occurs within quartz veins which are parallel to the basalt/dacite contact. Primary mineralisation is associated with hydrothermal veins and vein brecciation dominated by quartz enclosing lesser amounts of pyrite, illite/sericite, and tourmaline.
Drill hole information	• *A summary of all information material to the under-standing of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length*	See attached Appendix for a table of results.
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the*	Not applicable to this report.

Criteria	JORC Code Explanation	Commentary
	Competent Person should clearly explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. In the reporting of exploration results, results are reported as weighted averages using a nominal 0.5 g/t gold cut-off and up to 2 metres of internal dilution. No high-grade cuts were applied.
	• *Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.*	This release pertains to the reporting of Mineral Resources. Exploration results have previously been regularly reported to the ASX by the Joint Venture parties. Any high-grade zones above 15g/t gold within a reported intercept are also reported as included intervals.
	• *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.*	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralized.
	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	Mineralisation structures are vertical to sub-vertical.
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	Only downhole lengths have been reported. True widths are not known.
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Appropriate plans and sections have been included.
Balanced Reporting	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Planned drillholes are sited with a handheld global positioning system (GPS), and the initial drillhole pickup is usually with a handheld GPS, as well; with accuracy between ± 0.3 to 1m. After program completion, differential GPS (DGPS) is used for the final collar pickup with an accuracy of ± 5mm.
	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths. All intercepts for all holes have been reported regardless of grade.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk*	Exploration results have previously been regularly reported to the ASX by the Joint Venture parties.

Criteria	JORC Code Explanation	Commentary
	samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large- scale step-out drilling).*	A review of the drilling completed is required before further work is planned.
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Appropriate Diagrams accompany this release.

Section 3 Estimation and Reporting of Mineral Resources – Crusade

Criteria	JORC Code Explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	The drill hole database is managed by Northern Star Resources in Acquire. MJM completed systematic data validation steps after receiving the database. Checks completed by MJM included verifying that: • Down-hole survey depths did not exceed the hole depth as reported in the collar table. • Visual inspection of drill hole collars and traces in Surpac. • Assay values did not extend beyond the hole depth quoted in the collar table. • Assay and survey information was checked for duplicate records. There are some minor overlap errors in the RC and diamond drill holes where 4 metre samples overlapped later 1 metre samples, but the occurrence was not significant
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	The competent person, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining has made a number of visits to the Tanami JV area
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	The confidence in the geological interpretation is moderate to good as there are exposures and it is based upon RC and diamond drill holes. Mineralisation was based upon sectional interpretations that were assumed to be continuous between sections. At this stage of the project no alternative geological interpretations have been considered. The Crusade deposit lies at the contact between the Nany Goat Volcanics and the Killi Killi Formation along a regional fault structure. Specifically, the deposit lies on the northerly striking and westerly dipping contact between biotite dacite and mafic volcanics. The contact dips between 60 to 70 degrees west and strikes at about 020 degrees.

Criteria	JORC Code Explanation	Commentary
		The biotite dacite has been described by Moore (1996) as being porphyritic but also includes some lithic crystal tuffs. Further, Moore describes the mafic volcanics as mainly pyroxene porphyritic units that are probably interpreted as flows. The dacite can be interpreted from TMI shown and occurs as a magnetic low has an apparent thickness of 250 to 500 metres. The mafic volcanic unit can be seen clearly in the TMI as a high that is striking at 020 degrees and has an apparent thickness of about 100 metres. Moore (1996) describes the primary mineralisation being associated with hydrothermal veining and vein brecciation that are dominated by quartz enclosing lesser amounts of pyrite, illite/sericite and tourmaline. Accessory ore minerals associated with higher gold values include chalcopyrite, galena and sphalerite. The mineralisation appears to be thickest highest grade at the intersection of the regional fault and the dacite / basalt contact.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	The overall strike of economically significant mineralisation is about 680 metres and is made up of 9 lodes. The mineralisation is striking at 020 degrees and dips vary between 40 to 60 degrees west. Individual lenses of mineralisation vary in strike length from 25 metres to 650 metres. Down dip lengths vary from 25 to 200 metres while true thickness can be from 2 to 25 metres. The best thickness of mineralisation occurs where a fault interpreted from the TMI data intersects the dacite/mafic volcanic contact. The mineralisation starts at the surface.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*	Ordinary Kriging (OK) interpolation with an oriented 'ellipsoid' search was used for the estimate. Surpac software was used for the estimations. Three dimensional mineralised wireframes (interpreted by CTPJV and checked by MJM) were used to domain the gold data. Sample data was composited to 1m down hole lengths using the 'fixed length' method. Intervals with no assays were excluded from the estimates. The influence of extreme grade values was addressed by reducing high outlier values by applying top-cuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CVs, and summary statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Crusade deposit. All modelling was completed in Surpac Geovia software. No estimation of deleterious elements was carried out. Only gold was interpolated into the block model. The block model used a primary block size of 10m NS by 5m EW by 5m RL with sub-blocking to 2.5m by 1.25m by 1.25m. The parent block size was selected based on approximately half the average drill spacing of RC drilling in the well drilled areas, while dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip direction.

Criteria	JORC Code Explanation	Commentary
	• *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	QKNA was completed in Supervisor software to justify the block size, number of samples, search ellipses and discretization An orientated 'ellipsoid' search was used to select data and was based on the observed lode geometry. The search ellipsoid was orientated to the average strike, plunge, and dip of the main lodes. Three expanding passes were used in the estimation (40-60, 80-120 and 160-240 metres). A first pass of radius 40m with a minimum number of samples of 4-6 samples and a second pass of radius 80-120m with a minimum number of 4-6 samples were used for Crusade. A third pass of search radius 160-240m was used with 3-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8-24 depending on the number of samples in the domain. Blocks that did not fill after 3 passes were given a 4th pass. Selective mining units were not modelled. The block size used in the resource model was based on drill sample spacing and lode orientation. To validate the model, a qualitative assessment was completed by slicing sections through the block model in positions coincident with drilling. A quantitative assessment of the estimate was completed by comparing the average gold grades of the composite file input against the gold block model output for all the resource objects. A trend analysis was completed by comparing the interpolated blocks to the sample composite data within the main lodes. This analysis was completed for eastings and elevations across the deposit. Validation plots showed good correlation between the composite grades and the block model grades.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Open Pit Mineral Resource Estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.7g/t gold cut-off grade in Oxide and Transitional and 0.8 g/t gold cut-off in Fresh for open pit material within a $AU3,500 pit shell. The underground Mineral resource Estimate reports all material with a AU$3,500 stope optimisation including planned dilution.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	It is assumed the Crusade deposit will be mined by open pit and underground methods when a new mining operation can be established. The following mining factors and costs were used for the Deswik optimisation of the open pit and underground resource: Concentrate assumptions • Concentrate transport $370.04 per dry metric tonne of concentrate • Concentrate treatment and refining costs $147.58 per dry metric tonne of concentrate • Gold in concentrate payability 93% Deswik Open Pit Assumptions: • Mining Ore Loss 2% • Open Pit dilution 10%

Criteria	JORC Code Explanation	Commentary
		• Pit Slopes – Oxide 39° • Pit Slopes – Other 45° • Mining Cost Insitu Rock $4.50 per tonne rock • Mining Cost Loose Rock $2.60 per tonne rock • Mining Fixed and Grade Control Costs $5.30 per tonne of ore • Mining Cost Contingency 10% • Mine ROM to Mill ROM Haulages $0.10/t per km ore • Mill CIL Opex cost $35.46 per tonne • Mil floatation Opex cost additional $3.94 per tonne • Admin (G&A) $4.50 per tonne • CIL processing recovery 90% oxide, 86% transitional, 10.1% of total in fresh tailings • Flotation processing recovery 85.1% of total gold in concentrate • Processing cost contingency 10% • Au Price AU$3500 per troy ounce • Au Royalty 5.5% • Discount Rate 8% • Mining Rate 20 Mtpa rock • Crusade haulage 105 km Deswik Underground Stope Optimiser Assumptions • *Minimum Mining Width 2.4 metres • Minimum Pillar Width 5 metres • Stope Strike Length 20 metres • Sub-level Interval 20 metres • Optimise grade or metal: grade • Stope Strike ±40 degrees • Stope Dip – Minimum 40 degrees • Sub Stope Shapes 2 U / 2 V • Smoothing None *Minimum Mining Width includes allocation for HW and FW dilution • UG mining unplanned recovery 5% • UG mining unplanned dilution 5% • Processing recovery 95.2% • UG Stoping Costs $75/tonne ore • UG Opex Fixed Cost $5/tonne ore • Mill Opex cost $39.40/tonne ore • ROM to Mill transport $10.50/tonne ore • Admin (G&A) $4.50/tonne ore • NT Factor (10%) $13.44/tonne ore • Au Royalty 5.5% • Au Price AU$3500/troy ounce
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining*	Metallurgical testing was carried out in 1996 by Oretest Pty Ltd (Oretest) to test whether the Crusade prospect was amenable to heap leach extraction of gold. Oretest concluded that the saprolite and weathered bedrock was amenable to heap leach

Criteria	JORC Code Explanation	Commentary
	reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	however the fresh rock was not. Further test work in 1996 was carried out by Normet on CDH007 from 53 to 83 metres in a zone that was considered to represent saprolite and weathered bedrock. Percolations tests were also carried out on -12 mm crushed ore showed that with the addition of 0.5% cement good percolation rates were achieved. A composite column test of –50mm ore was carried out, consisting of pre-screened plus 12.5mm ore being combined with agglomerated –12.5mm ore. Normet concluded that a recovery of 80% at a solution rate of 2.5m3/t could be expected from a heap leach extraction method. Although this testing is not directly applicable to recoveries in a CIL plant it is a reasonable assumption that the gold is cyanide extractable recoveries of around 90% could be expected. Further testing was carried out in 2020 by the NST exploration department on ore grade material from drill holes SJRC0005-6 using 500g Leachwell analysis and fire assay of the residual material. Arsenic and sulphur values were compared with the recovery data along with the oxidation state of the sample. There appears to be a correlation between As, S and Au recovery. Preliminary metallurgical testing MineScope Services Pty Ltd on historical diamond drill core at Central Tanami completed in 2025 suggests that using floatation, concentrate export and leaching of the flotation tails produces a total gold recovery of 95.7% in fresh rock. Further testing is still required.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	No assumptions have been made regarding environmental factors.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td>Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>There is no bulk density data from the Crusade prospect. Density values were taken from average densities for dacitic rocks and basalts and adjusted for oxidation. These values may not be correct. It is recommended that diamond holes are drill to gather representative bulk density measurements.
<table>
<tr><th></th><th colspan="2">Rock Type</th></tr>
<tr><th>Oxidation State</th><th>Basalt</th><th>Biotite Dacite</th></tr>
<tr><td>Oxide</td><td>2.5</td><td>2.4</td></tr>
<tr><td>Transitional</td><td>2.6</td><td>2.5</td></tr>
<tr><td>Fresh</td><td>2.77</td><td>2.65</td></tr>
</table>
At this stage of the project, it is assumed that these values will be close to the real values.</td></tr>
<tr><td>Classification</td><td>The basis for the classification of the Mineral Resources into varying confidence categories.
Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).
Whether the result appropriately reflects the Competent Person's view of the deposit.</td><td>The Mineral Resource estimate is reported here in compliance with the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' by the Joint Ore Reserves Committee (JORC). The Mineral Resource was classified as Indicated and Inferred Mineral Resource based on data quality, sample spacing, and lode continuity. The Indicated Mineral Resource was defined within areas of RC drilling of 40m by 40m (with some infill), where the continuity and predictability of the lode positions was good and the estimation had reasonable slopes of regression. The Inferred Mineral Resource was assigned to areas where support for the continuity of mineralisation was limited by wider spaced drilling or insufficient drilling in smaller lodes. The minimum requirement for an inferred resource is 3 drill holes spaced apart so that strike and dip can be determined.

Validation of the block model shows good correlation of the input data to the estimated grades.

The result reflects the competent person's view that the classification is Indicated and Inferred.</td></tr>
<tr><td>Audits or reviews</td><td>The results of any audits or reviews of Mineral Resource estimates.</td><td>The Northern Star Resources resource geologists have reviewed the Mineral Resource Estimate.</td></tr>
<tr><td>Discussion of relative accuracy/ confidence</td><td>Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.
The statement should specify</td><td>The Crusade Mineral Resource Estimate has been reported with a moderate degree of confidence.

The Indicated Mineral Resource is based upon 40 by 40 metre (with some infill) RC and diamond drilling of acceptable quality. It is assumed that the mineralisation in this area is continuous between drill sections. The project is in area of no previous mining.

The Mineral Resource statement relates to global estimates of tonnes and grade.</td></tr>
</table>

	whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	