Mount Gibson Iron

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ASX ANNOUNCEMENT
16 July 2025

Agreement to acquire 50% of the Central Tanami Gold Project from Northern Star

- Mount Gibson has entered into an agreement with Northern Star Resources Limited to acquire its **50% interest in the advanced Central Tanami Project Joint Venture** (**CTPJV**) and adjacent wholly-owned exploration tenements for **cash consideration of $50 million**.
- The CTPJV assets comprise **over 2,100 sqkm of mining and exploration tenements** in the Central Tanami region of the Northern Territory, hosting the following:
  - **JORC 2012* Mineral Resources** totalling **13.8 million tonnes (Mt) at 3.6 grams per tonne (g/t) gold for 1.6 million ounces (Moz) of contained gold**, the vast majority of which are located on **granted mining leases**, including the **high grade Groundrush deposit**. Strong potential exists to increase the resource base through further resource definition and exploration drilling;
  - **Historical (JORC 2004) estimates** of mineral resources totalling 11.2 Mt at 2.7 g/t gold for 1.0 Moz of contained gold. Work is currently underway to update these historical estimates to JORC 2012 standards but pursuant to the ASX Listing Rules, it should be noted that a Competent Person has not yet completed sufficient work to classify these historical estimates in accordance with the JORC Code (2012) and it is uncertain that following further evaluation and/or exploration work that the historic estimates will be able to be reported as Mineral Resources in accordance with the JORC Code (2012); and
  - **A 1.2 Mtpa carbon-in-leach processing plant** (non-operating) representing a refurbishment option for future project development, plus other existing infrastructure including **haul roads, an accommodation camp, bore field and gravel airstrip**.
- The other party in the CTPJV is Tanami Gold NL, an ASX-listed junior gold company seeking to advance the CTPJV project. Mount Gibson and Tanami Gold share a common major shareholder.
- The wholly-owned exploration tenements being acquired from Northern Star cover approximately 3,600 sqkm, providing Mount Gibson with exposure to a **dominant land position** in a **major established gold region** with a history of substantial gold production.
- The acquisition consideration equates to an **attractive resource multiple of $61/oz** for a 50% interest in the CTPJV's JORC 2012 Mineral Resources, and **$38/oz** inclusive of the historical (JORC 2004) estimate of mineral resources. Refer to Table 1 and the section entitled "*Summary of information material to understanding the reported Mineral Resource Estimates for the Central Tanami Project Joint Venture*" for further information in relation to the historical estimate.
- Key conditions to completion, to be satisfied before 31 March 2026, comprise Foreign Investment Review Board approval, further extension of existing infrastructure arrangements on one tenement by the Central Land Council, non-exercise by Tanami Gold of its right of first refusal under the CTPJV, and various other pre-completion items standard for a transaction of this nature.
- The acquisition provides Mount Gibson with an **attractive near-term development project**, building on its remote-site operating strengths, with the ability to fast track permitting and technical work to position for a potential development decision within the next 12-18 months.

*Reported in accordance with the 2012 Edition of the "Australasian Code for the Reporting of Mineral Resources and Ore Reserves", as reported by Tanami Gold NL on 14 September 2023. Refer below for Competent Persons information.

Mount Gibson will host an investor teleconference call at **11:00am AEST (9:00am AWST) today, Wednesday, 16 July 2025** which can be accessed by dialling 1800 896 323 immediately prior to the scheduled start time and entering the access code 99219906# at the prompts. A recording of the teleconference will be available on the Mount Gibson website after completion of the teleconference. In case of difficulties, operator assistance can be reached on +61 (0)2 8088 0900.

Mount Gibson Iron Limited (**Mount Gibson** or the **Company**) advises it has reached a binding conditional agreement with Australian gold producer Northern Star Resources Ltd (**Northern Star**) to acquire Northern Star's 50% interest in the Central Tanami Project Joint Venture (**CTPJV**) in the Northern Territory of Australia, along with an extensive wholly-owned exploration landholding adjacent to the CTPJV.

Central Tanami Project Joint Venture (CTPJV)

The CTPJV is an unincorporated 50:50 joint venture between Northern Star and ASX-listed gold explorer Tanami Gold NL (**Tanami Gold**) located approximately 650 kilometres northwest of Alice Springs and encompassing a total landholding of approximately 2,100 sqkm in the Tanami region of the Northern Territory (Refer *Figure 1*).

The CTPJV was initially established in 2015 as a farm-in arrangement and subsequently restructured in 2021 into the current 50:50 arrangement whereby each party is required to contribute its share of expenditure to the joint venture.

The objective of the CTPJV is to advance the Central Tanami Gold Project, a highly prospective and underexplored area that is known to be well-endowed with gold mineralisation. In excess of 2 million ounces (**Moz**) of gold have previously been extracted from the tenements, including approximately 1.3 Moz between 1995 and 2005.

The CTPJV includes a non-operating 1.2 Mtpa carbon-in-leach (**CIL**) processing plant which has been idle since 2005, and the high-grade Groundrush gold deposit which yielded production of more than 600,000oz from a single open pit for Newmont Mining Ltd between 2001 and 2005. In addition, the CTPJV infrastructure assets include existing mine haul roads, an accommodation camp, a water bore field and a 1.4km gravel airstrip.

CTPJV Mineral Resources

The CTPJV (100%) has a reported Mineral Resource estimate of 13.8 Mt grading 3.6 g/t gold for 1.6 Moz of contained gold defined in accordance with the 2012 Edition of the Joint Ore Reserves Committee's *Australasian Code for the Reporting of Mineral Resources and Ore Reserves* (**JORC 2012**). The JORC 2012 Mineral Resource estimate comprises 0.78 Moz of gold in the Measured and Indicated categories and 0.82 Moz of gold in the Inferred category. The vast majority of the Mineral Resources (JORC 2012) are located on granted mining leases.

The CTPJV also includes historical estimates of mineral resources totalling 11.2 Mt at an average grade of 2.7 g/t gold for approximately 1.0 Moz of contained gold. These historical estimates were reported by Tanami Gold and Northern Star according to the prior JORC 2004 Code. A Competent Person has not yet completed sufficient work to classify these historical estimates in accordance with JORC 2012 and it is uncertain that following further evaluation and/or exploration work that these historical estimates will be able to be reported in accordance with JORC 2012. The CTPJV parties are presently working to update a number of these historic estimates consistent with JORC 2012 standards.

Refer to *Table 1* and the section below entitled "*Summary of information material to understanding the reported Mineral Resource Estimates for the Central Tanami Project Joint Venture*" for further information.

Mount Gibson considers there is strong potential to increase the CTPJV Mineral Resource estimates through ongoing resource definition drilling and exploration activities.

Project Development

Work within the CTPJV over the last few years has focused on the Groundrush, Ripcord and Jims deposits, including technical development and scoping studies. Each of these deposits is located on existing mining leases.

The Groundrush deposit hosts currently reported Indicated and Inferred Mineral Resources of 7.7 Mt grading 4.3 g/t Au for 1.1 Moz contained gold and is central to future development options within the CTPJV. The Ripcord and Jims deposits host a combined total of 0.2 Moz of gold, with Ripcord comprising Indicated and Inferred Mineral Resources of 0.75 Mt at 2.1 g/t gold and Jims comprising Measured, Indicated and Inferred Mineral Resources totalling 1.6 Mt at 2.4 g/t gold.

Mount Gibson considers the CTPJV to have potential for accelerated development and production based on its substantial current Mineral Resource estimates located on granted mining leases, notably at Groundrush, the extensive work conducted to date on technical development studies, and the strong potential to substantially increase resources. The CTPJV parties are currently undertaking drilling at several brownfields targets with a view to further increasing gold resources in the near term.

Wholly-owned exploration tenements

The acquisition also includes approximately 3,600 sqkm of granted and pending exploration tenements in the Central Tanami region wholly owned by Northern Star, giving Mount Gibson exposure to a dominant exploration position covering approximately 5,700 sqkm in a proven gold province with a century-long record of substantial gold production.

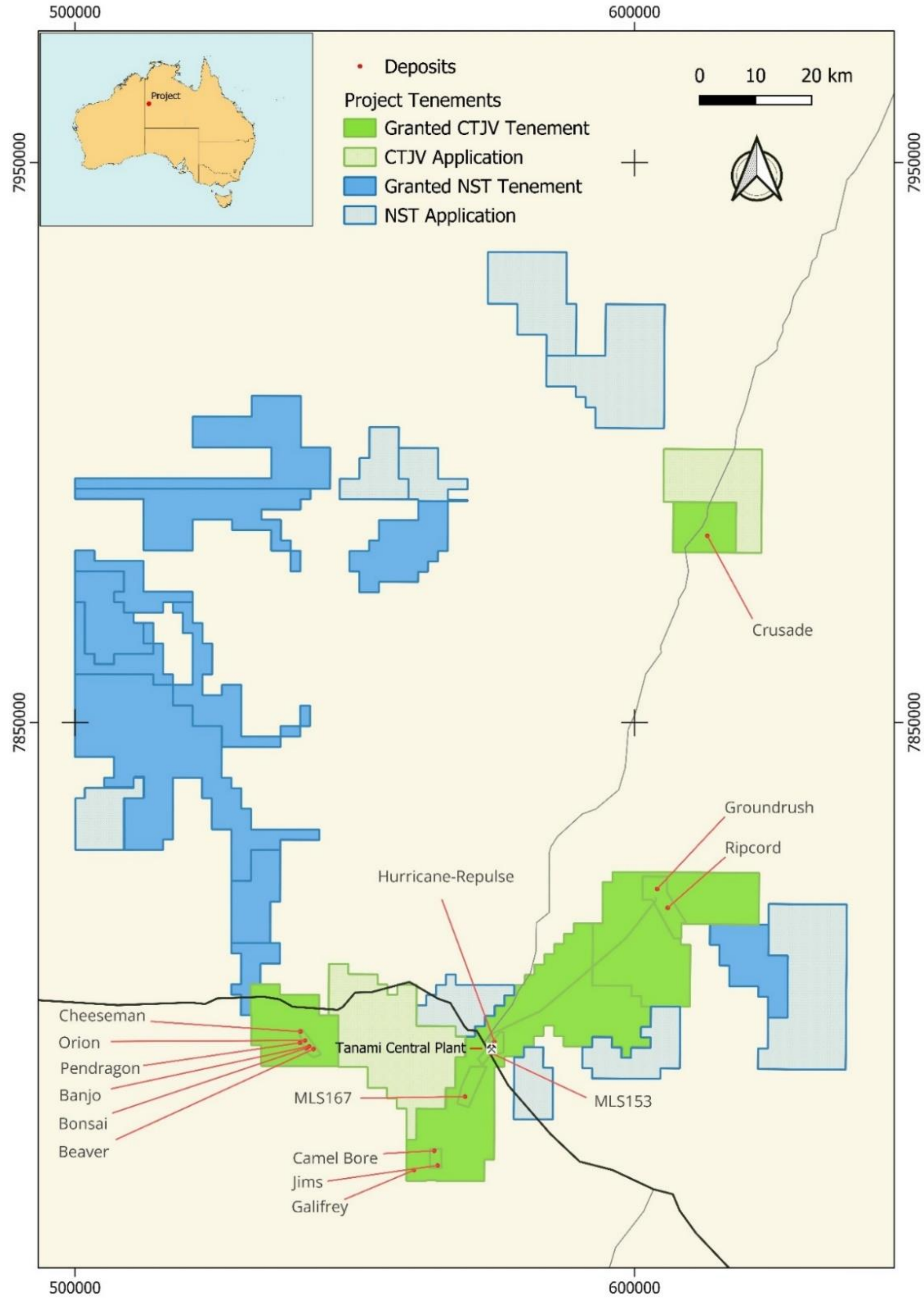

Figure 1

Central Tanami tenements subject to the transaction, comprising mining and exploration tenements within the CTPJV (shown in green) and exploration tenements owned outright by Northern Star outside of the CTPJV (shown in blue)

On completion of the transaction, Mount Gibson intends to work actively with Tanami Gold to bring the CTPJV to a development decision in the shortest possible timeframe. The Company's intention will be to progress technical update studies for the Groundrush, Ripcord and Jims deposits to feasibility level, incorporating additional drilling and other data generated by the CTPJV partners in recent periods.

In addition, an engineering review will be required to define the optimum processing and infrastructure options for re-establishing production within the CTPJV. This includes assessment of both new build and full or partial refurbishment of the existing CTPJV processing plant which has been idle since 2005.

Figure 2
CTPJV 1.2 Mtpa Processing Plant (idle since 2005, requires refurbishment)

Acquisition Terms

The acquisition will be achieved through Mount Gibson acquiring Northern Star's subsidiary entity (on a cash and debt free basis) which holds Northern Star's 50% CTPJV interest and its adjacent wholly-owned exploration tenure.

Consideration for the acquisition is $50 million cash, payable on completion, along with typical adjustments for any cash, debt and working capital within the acquired entity at completion. Mount Gibson will fund the purchase price from its internal cash reserves. Mount Gibson will also be required to replace existing bank guarantees totalling approximately $5.8 million, which it will do from its existing performance bonding facility.

Completion will occur 10 days after satisfaction or (where permitted) waiver of: (1) approval pursuant to the Foreign Investment Review Board; (2) further extension of existing infrastructure arrangements on one tenement by the Central Land Council; (3) non-exercise by Tanami Gold of its right of first refusal under the CTPJV; and (4) various other pre-completion conditions considered standard for a transaction of this nature. These conditions must be satisfied by 31 March 2026 (unless extended by agreement).

In the period prior to completion, Northern Star will conduct its business within the CTPJV on an ordinary course basis and in line with the approved CTPJV program and budget for the period. Northern Star is responsible for the first $3 million of its share of CTPJV expenditure in the pre-completion period, and Mount Gibson is responsible for expenditure incurred in excess of $3 million, with any reimbursement to occur at completion along with the other cash, debt and working capital adjustments described above. Consent protections apply in certain circumstances, and Mount Gibson will also be provided with regular reports and consultation opportunities.

The acquisition terms also involve customary representations and warranties, and an agreed liability regime.

Acquisition Rationale

The consideration for a 50% interest in the CTPJV equates to an average acquisition cost of approximately $61/oz of JORC 2012 Mineral Resources and $38/oz inclusive of historical estimates of mineral resources. This is comparable with other recent gold sector transactions and attractive for Mount Gibson given the advanced state of the CTPJV project, its substantial existing gold inventory and significant exploration upside.

Mount Gibson intends to work closely with Tanami Gold to actively pursue development studies and permitting activities to position for a potential development decision within the next 12-18 months, centered on the high grade Groundrush deposit. Mount Gibson has sufficient cash reserves to fund its share of anticipated CTPJV development costs.

The acquisition is complementary with Mount Gibson's continued pursuit of other investment opportunities to build a successful multi-commodity metals producer in Australia. The Company views the CTPJV acquisition as a first step toward building a precious metals production base.

Mount Gibson notes that its 38.4% major shareholder, Hong Kong-listed resources investment group APAC Resources Limited, also holds a direct 46.3% shareholding interest in Tanami Gold. Recently appointed (effective 17 April 2025) Mount Gibson Chairman Brett Smith is also a non-executive Director of Tanami Gold and has abstained from Mount Gibson Board decisions in relation to this transaction.

Advisers

Mount Gibson has been advised on this acquisition transaction by Azure Capital and Gilbert & Tobin, and assisted by other professional and technical advisers in specific areas.

Comment

Mount Gibson Chief Executive Officer Peter Kerr said: "This acquisition represents a compelling opportunity to enter the gold sector at an attractive price, with the sector having strong fundamentals. Involvement in the Central Tanami Gold Project provides Mount Gibson with an opportunity to leverage the success of its Koolan Island iron ore operation to establish the foundations of a gold production business.

"Given the substantial work that has been undertaken on the project by the CTPJV to date, we intend to work with Tanami Gold to fast track technical studies to position the project for a development decision within the next 12-18 months. We have already commenced the Foreign Investment Review Board application process and look forward to progressing to completion in the coming months.

"In the interim, we will remain a close observer of the work being undertaken in accordance with the joint venture schedule approved by Tanami Gold and Northern Star, including the resource definition drilling programs that are presently underway."

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Table 1: Central Tanami Project Joint Venture Total Mineral Resources and Historical Estimates (Figures are reported for 100% of the CTPJV)

Deposit	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (koz)	Tonnes (kt)	Gold (g/t)	Ounces (koz)	Tonnes (kt)	Gold (g/t)	Ounces (koz)	Tonnes (kt)	Gold (g/t)	Ounces (koz)
Mineral Resource Estimates (JORC 2012)													
GROUNDRUSH													
Groundrush Gold Deposit													
Open pit	0.70	-	-	-	2,600	3.8	320	170	5.6	30	2,800	3.9	350
Underground	1.70	-	-	-	1,400	3.9	170	3,600	4.8	550	4,900	4.6	720
Total		-	-	-	**4,000**	**3.8**	**490**	**3,700**	**4.8**	**580**	**7,700**	**4.3**	**1,100**
Ripcord Gold Deposit													
Open pit	0.60	-	-	-	640	2.1	43	110	2.2	8	750	2.1	51
Total		**-**	**-**	**-**	**640**	**2.1**	**43**	**110**	**2.2**	**8**	**750**	**2.1**	**51**
TANAMI SOUTHWEST													
Jims Gold Deposit													
Open pit	0.70	120	1.9	7	500	2.1	34	120	1.7	6	740	2.0	48
Underground	1.70	-	-	-	170	2.3	13	680	2.7	60	850	2.7	73
Total		**120**	**1.9**	**7**	**670**	**2.2**	**47**	**800**	**2.6**	**66**	**1,600**	**2.4**	**120**
TANAMI MINE CORRIDOR													
Hurricane-Repulse Gold Deposit													
OP – Oxide/Trans	0.63	-	-	-	510	2.6	42	160	2.1	11	670	2.5	53
OP - Primary	0.97	-	-	-	20	4.4	3	-	-	-	20	4.4	3
Underground	2.80	-	-	-	66	3.7	8	700	5.0	110	770	4.9	120
Total		-	-	-	**590**	**2.8**	**53**	**870**	**4.5**	**120**	**1,500**	**3.8**	**180**
TANAMI NORTHEAST													
Crusade Gold Deposit													
Open pit	0.77	-	-	-	1,200	2.2	86	38	1.7	2	1,200	2.2	88
Underground	3.00	-	-	-	49	3.7	6	-	-	-	49	3.7	6
Total		-	-	-	**1,200**	**2.3**	**92**	**38**	**1.7**	**2**	**1,300**	**2.3**	**94**
MOLECH AREA													
Beaver Gold Deposit													
Open pit	0.65	-	-	-	100	3.9	13	41	4.1	5	140	4.0	18
Underground	1.80	-	-	-	110	3.3	12	140	3.2	14	250	3.3	26
Total		-	-	-	**210**	**3.6**	**24**	**180**	**3.4**	**20**	**390**	**3.5**	**44**
Banjo Gold Deposit													
Underground	1.80	-	-	-	120	3.6	13	23	2.2	2	140	3.4	15
Total		-	-	-	**120**	**3.6**	**13**	**23**	**2.2**	**2**	**140**	**3.4**	**15**
Bonsai Gold Deposit													
Open pit	0.65	-	-	-	110	2.1	8	25	2.8	2	140	2.2	10
Underground	1.80	-	-	-	9	2.1	1	73	2.7	6	81	2.6	7
Total		-	-	-	**120**	**2.1**	**8**	**98**	**2.7**	**9**	**220**	**2.4**	**17**
Orion Gold Deposit													
Open pit	0.65	-	-	-	39	3.1	4	9	5.7	2	47	3.6	5
Underground	1.80	-	-	-	27	2.3	2	17	2.6	1	43	2.4	3
Total		-	-	-	**65**	**2.8**	**6**	**25**	**3.7**	**3**	**91**	**3.0**	**9**
Cheeseman Gold Deposit													
Open pit	0.65	-	-	-	11	4.8	2	8	2.3	1	19	3.7	2
Underground	1.80	-	-	-	-	-	-	50	3.5	6	50	3.5	6
Total		-	-	-	**11**	**4.8**	**2**	**59**	**3.4**	**6**	**69**	**3.6**	**8**
Pendragon Gold Deposit													
Open pit	0.65	-	-	-	-	-	-	24	2.2	2	24	2.2	2
Underground	1.80	-	-	-	-	-	-	17	2.3	1	17	2.3	1
Total		-	-	-	-	-	-	**41**	**2.3**	**3**	**41**	**2.3**	**3**
Total JORC 2012		**120**	**1.9**	**7**	**7,626**	**3.2**	**778**	**5,944**	**4.3**	**819**	**13,801**	**3.6**	**1,641**
Historical Estimates (JORC 2004)													
TANAMI MINE CORRIDOR & SOUTHWEST													
MLS153													
Various deposits		1,100	2.2	73	2,200	1.9	140	370	1.8	21	3,600	2.0	230
Total		**1,100**	**2.2**	**73**	**2,200**	**1.9**	**140**	**370**	**1.8**	**21**	**3,600**	**2.0**	**230**
MLS167													
Various deposits		2,700	3.4	290	2,600	2.9	240	2,000	2.9	190	7,400	3.1	730
Total		**2,700**	**3.4**	**290**	**2,600**	**2.9**	**240**	**2,000**	**2.9**	**190**	**7,400**	**3.1**	**730**
MLS168													
Camel Bore		73	2.8	7	51	1.8	3	43	2.3	3	170	2.4	13
Total		**73**	**2.8**	**7**	**51**	**1.8**	**3**	**43**	**2.3**	**3**	**170**	**2.4**	**13**
Total Historical Est.		**3,873**	**3.0**	**370**	**4,851**	**2.4**	**383**	**2,413**	**2.7**	**214**	**11,170**	**2.7**	**973**

Note regarding Historical Estimates: A Competent Person has not completed sufficient work to classify the historical estimates as Mineral Resources in accordance with the JORC 2012 Code. It is uncertain, following evaluation and/or further exploration work, that the historical estimates can be reported as Mineral Resources in accordance with JORC 2012. Refer to the following pages for further information.

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 for each deposit have been rounded to two significant figures to reflect the relative uncertainty of the estimates. Rounding may cause values in the table to appear to have computational errors. Mineral Resources are reported on a dry in-situ basis.

Mineral Resource estimates reported in accordance with the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves are Groundrush, Ripcord, Hurricane-Repulse, Jims, Crusade, Beaver, Banjo, Bonsai, Orion, Cheeseman and Pendragon. They are reported at cut-off grades ranging from 0.60 g/t gold to 0.77 g/t gold within an optimised pit shell based on a A$2,700 per ounce gold price and cut-off grades ranging from 1.70 g/t gold to 3.00 g/t gold within stope optimisation wireframes based on a A$2,700 per ounce gold price.

Mineral Resource estimates reported in accordance with the 2004 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves are MLS153, including the Southern, Miracle, Bastille, Dinky and Thrasher deposits; MLS167 including the Carbine, Phoenix, Redback Rise, Lynx, Legs, Bulldog and Dogbolter deposits; and MLS168 representing the Camel Bore deposit. They are reported at a cut-off grade of 0.7 g/t gold but have not been subjected to economic constraints. Refer to the following summary of material information relevant to understanding the historical estimates.

Competent Persons Information

The information in this announcement that relates to the reported Mineral Resources Estimates for the Central Tanami Gold Project is extracted from the Tanami Gold NL (Tanami Gold) ASX announcement entitled "Annual Mineral Resource Statement" published on 14 September 2023 and reflects the information prepared on behalf of the CTP JV by MoJoe Mining Pty Ltd (MJM), as published on the ASX by Tanami Gold NL in its releases dated 24 November 2022 entitled *"Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs"* and on 30 August 2023 entitled *"Mineral Resource Updates Completed for Gold Deposits In The Molech Area"* . These announcements are available to view at www.tanami.com.au and at www.asx.com.au. Northern Star Resources Limited (Northern Star) has also republished its attributable share of the Mineral Resources Estimates without variation in its *"Annual Mineral Resources and Ore Reserves Statement as at 31 March 2025*" published on the ASX on 15 May 2025, and which is available for viewing at www.nsrltd.com.au. Mount Gibson confirms its understanding that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed. To the extent disclosed above, the Company confirms that the form and context in which the Competent Person's findings are presented have not been materially modified from the original market announcement.

Mount Gibson engaged Mr Graeme Thompson from MJM to act as Competent Person on Mount Gibson's behalf and confirm that all material assumptions and technical parameters underpinning the Mineral Resources estimates in this market announcement continue to apply and have not materially changed. To the extent disclosed above, Mount Gibson confirms that the form and context in which the Competent Person's findings are presented have not been materially modified from the original market announcements, and Mount Gibson confirms that it is not aware of any further new information or data that materially affects the information included in the original market announcements by Tanami Gold and Northern Star referred to above. The Competent Person's information relating to these estimates are reported below.

Mineral Resource Estimates (JORC 2012)

The information in this release that relates to the Mineral Exploration Results and Mineral Resource estimates for the Groundrush, Ripcord, Jims, Hurricane-Repulse, Crusade, Beaver, Banjo, Bonsai, Orion, Cheeseman and Pendragon Gold Deposits 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 and has been engaged as a consultant to Mount Gibson to act as Competent Person on Mount Gibson's behalf. Mr 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 Thompson provided written consent approving the inclusion of the Mineral Resource estimates in this report in the form and context in which they appear. Mount Gibson confirms that it is not aware of any new information or data that materially affects the Mineral Resource estimates as reported, and the assumptions and technical parameters underpinning the Mineral Resource estimates reported continue to apply and have not materially changed.

Historical estimates (JORC 2004)

The information in this release that relates to the historical estimates of mineral resources in respect of MLS153 (Southern, Miracle, Bastille, Dinky and Thrasher Gold Deposits), MLS167 (Carbine, Phoenix, Redback Rise, Lynx, Legs, Bulldog and Dogbolter Gold Deposits) and MLS168 (Camel Bore Gold Deposit) were originally published by Tanami Gold NL on the ASX on 8 June 2011 in a release titled "Tanami Lifts Gold Resources to 2.3Moz and Unveils a 400,000oz Ore Reserve". A Competent Person has not completed sufficient work to classify the historic estimate as mineral resources in accordance with the JORC Code 2012. It is uncertain, following evaluation and/or further exploration work that the historical estimates can be reported as mineral resources in accordance with JORC 2012. Refer to the following pages for further information.

Summary of information material to understanding the reported Mineral Resource Estimates for the Central Tanami Project Joint Venture

All information in this announcement in relation to the CTP JV - including in relation to past production, mineral resources estimates, historical costs and other historical information - has been sourced from the public information reported by the Central Tanami Project Joint Venture partners Northern Star Resources Ltd and Tanami Gold NL. Whilst due diligence has been undertaken and steps have been taken to review the findings from that and the information provided, no representation or warranty, expressed or implied, is made as to the fairness, accuracy, correctness, completeness or adequacy of any information relating to the CTP JV at this time.

The following information is relevant to understanding the current Mineral Resources estimates (JORC 2012) for the Project, as depicted in Table 1. The mineral resource estimates were prepared for the CTP JV by mining consultants MoJoe Mining Pty Ltd (**MJM**), an organisation with personnel experienced in best practices in modelling and estimation methods. MJM conducted reviews of the quality and suitability of the underlying information, which includes on-site visits to the Central Tanami Project area. Furthermore, the CTPJV management regularly conducts evaluations, peer reviews, and audits of internal processes and external contractors employed in these efforts.

In order to satisfy its confidence in the veracity of the current Mineral Resources Estimates for the CTP JV, Mount Gibson engaged Mr Graeme Thompson, an employee of MJM, as a consultant to act as Competent Person on Mount Gibson's behalf (refer Competent Persons Information above). Mr Thompson has provided written consent approving the publication of the Mineral Resource Estimates in this report in the form and context in which they appear and that the assumptions and technical parameters underpinning the Mineral Resource estimates reported continue to apply and have not materially changed. No more recent estimates have been completed or provided to Mount Gibson by Northern Star or MJM. The mineral resources estimates as at 30 June 2024 (as contained in the Competent Person Report) are, so as Mount Gibson is aware, the most recent estimates approved by the CTP JV partners.

Established in 2015, the CTP JV aims to advance exploration across the 2,211 sqkm tenement area in the Tanami region. Its goal is to develop and mine the Groundrush Gold Deposit (**Groundrush**), along with any other gold deposits within the Central Tanami Project area, using existing mining infrastructure. Located 650km northwest of Alice Springs in the Northern Territory, the Central Tanami Project area encompasses highly prospective geological sequences in an area that is known to be well-endowed with gold mineralisation and where around 2.1 million ounces have historically been produced since small-scale mining activities began in the 1900s.

Throughout the period from 1 July 2022 to 30 June 2023, MJM continued a process of updating the Central Tanami Project Mineral Resource estimates as part of an ongoing initiative to align the Mineral Resource estimates for the Central Tanami Project with the reporting standards outlined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves (**2012 JORC Code**). Updates were successfully completed for 11 gold deposits, including Groundrush, Ripcord, Crusade, Jims, Hurricane-Repulse, Beaver, Banjo, Bonsai, Orion, Cheeseman, and Pendragon and these were disclosed separately to the ASX by Tanami Gold NL on:

- 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs*;
- 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*.
- 14 September 2023 – *Annual Mineral Resource Statement*

These announcements are available to view at www.tanami.com.au. Northern Star Resources Limited has also republished its attributable share of the Mineral Resources Estimates without variation in its "*Annual Mineral Resources and Ore Reserves Statement as at 31 March 2025*" published on the ASX on 15 May 2025, and which is available for viewing at www.nsrltd.com.au.

These Mineral Resource Estimates were developed using revised geological models that more accurately represent the mineralised systems. Each of these updated Mineral Resource estimates were tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades are determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free-milling processing recoveries. The updated estimates do not include results from drilling completed since 2022.

For ease of explanation, Mount Gibson has grouped these current Mineral Resource Estimates into two main groups – Main CTP JV Deposits (Groundrush, Ripcord, Jims, Hurricane-Repulse and Crusade) and the Molech Area Deposits (Beaver, Banjo, Bonsai, Orion, Cheeseman and Pendragon).

The assets of the CTPJV also include substantial additional historical estimates of mineral resources, together totalling approximately 1.0 Moz contained gold. These historical estimates for 13 gold deposits at the Central Tanami Project still follow the reporting criteria of the 2004 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves (**2004 JORC Code**).

These estimates have not been subjected to economic constraints and were first released to the ASX on:

- 8 June 2011 – *Tanami Lifts Gold Resources to 2.3Moz and unveils a 400,000oz Ore Reserve.*

The CTPJV continues to progress efforts to complete updated estimates for these deposits comprising MLS153 (the Southern, Miracle, Bastille, Dinky and Thrasher deposits), MLS 167 (the Phoenix, Redback Rise, Lynx, Legs, Bulldog, Dogbolter and Carbine deposits) and MLS168 (Camel Bore deposit) as well as the previously unreleased Galifrey deposit in the Tanami Southwest area to bring them into compliance with the 2012 JORC Code.

Mount Gibson notes that at this time, a Competent Person has not done sufficient work to classify these historical estimates in accordance with the 2012 JORC Code and it is uncertain that following further evaluation and/or further exploration work, that the historical estimates will be able to be reported in accordance with the 2012 JORC Code.

Central Tanami Project Joint Venture Current Mineral Resources (2012 JORC Code)

Main CTP JV Deposits

Groundrush Gold Deposit

Groundrush is located approximately 45km northeast of the Central Tanami Mill site and is fully encompassed by Mining Lease ML22934 (refer Figure 1). The Groundrush deposit was previously subject to open-pit mining between 2001 and 2005 when Normandy/Newmont produced 611,000 ounces of gold at a reconciled mill grade of 4.0 g/t gold.

The Groundrush Mineral Resource totals 7.7Mt grading 4.3 g/t gold for 1.1Moz (Table 1). It represents open-pit and underground material reported at cut-off grades of 0.70 g/t gold and 1.70 g/t gold, respectively. The Mineral Resource estimate has been tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free-milling processing recoveries.

The Groundrush Mineral Resources were reported to the ASX by Tanami Gold NL on 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs*. Mount Gibson is not aware of any new information or data that materially affects the Groundrush Open Pit and Underground Mineral Resources, and the assumptions and technical parameters underpinning the estimates in the 24 November 2022 report continue to apply and have not materially changed.

It is noted that results remain pending for drilling undertaken during 2024/25 and will be available for inclusion in the next iteration of the Groundrush Mineral Resource estimate.

	COG (g/t Au)	Measured			Indicated				Inferred			Total	
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Groundrush Gold Deposit													
OP	0.7	-	-	-	2,600	3.8	320	170	5.6	30	2,800	3.9	350
UG	SO @ 1.7	-	-	-	1,400	3.9	170	3,600	4.8	550	4,900	4.6	720
Total		-	-	-	**4,000**	**3.8**	**490**	**3,700**	**4.8**	**580**	**7,700**	**4.3**	**1,100**

Table 2: Groundrush Mineral Resource Estimate:

Geology and Geological Interpretation

Groundrush represents a reverse fault orogenic system, with mineralisation typically hosted in stacked vein sets with a variety of orientations, as well as sub-vertical quartz-filled shear zones, within a fractionated dolerite sill. Minor mineralisation also extends into the adjacent turbiditic sediments. Along with the various orientations of veining there also exists a variety of types, including shear, extensional and also a shear-extension hybrid style of veining. The steep dipping lodes generally strike around 340° but varied between 323° to 355° and dip about

60° to 70° west but range between 32° to 80° west. They exhibit a true thickness from 1-2 to 35 metres and plunge to the south at approximately 10° to 15°. Mineralisation has been defined over a collective strike length of 1,900 metres with the various individual lodes extending from 50 to 970 metres in length and down dip from 50 to 250 metres.

The flat lying lodes are only well established in the mined-out areas where they were defined by close spaced grade control drilling. These lodes crosscut the steep lodes and are difficult to interpret from the exploration drilling data. They are largely confined to areas of dolerite and strike between 325° to 340°, dip from 25° to 50° and plunge southwest between 15° to 24°. The strike length of these lodes varies from 50 metres 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. Volumetrically they represent about 20% of the total resource with most of that volume intersecting steep lodes. A two-stage approach to interpreting and modelling the gold mineralisation was completed. Interpretations of the geology were honoured, but the veins were characterised as either steep dipping or flat dipping lodes, with greater certainty in the steep dipping stacked lenses than the flat dipping lodes.

Drill Information and Sampling

Hole Type	In Database		In OP Resource Model		In UG Resource Model	
	Hole Number	Metres Drilled	Hole Number	Intersection Metres	Hole Number	Intersection Metres
AC	16	882.00				
DDH	258	73,553.72	225	3,345.91	218	2,058.23
RC_GC	41,183	321,743.00	14,784	68,744.79	15,897	43,634.88
RAB	611	32,125.00				
RC	397	39,961.00	280	5,044.00	269	3,585.00
RC_DDH	79	30,259.75	76	709.00	75	393.40
TR	8	11.00				
VC	897	8,283.00				
Total	**43,449**	**506,818.47**	**15,365**	**77,843.70**	**16,459**	**49,671.51**

Table 3: Groundrush Gold Deposit drilling summary

The various mineralised lodes at Groundrush were sampled using surface diamond drill holes (**DDH**), reverse circulation (**RC**) drill holes, reverse circulation grade control (**RC_GC**) drill holes, aircore (**AC**), vacuum (**VC**) and rotary air blast (**RAB**) drill holes. Drilling has been completed by Normandy, Newmont, Tanami Gold and Northern Star since 1998.

Normandy: RC drilling was sampled on 1m intervals through mineralised zones and 2m intervals within pre-collars. Entire samples were collected using a cyclone then split using a riffle splitter down to approximately 2kg. Diamond holes were half-cut lengthways and sampled on 0.5m intervals, with the right half retained in the core trays for future reference. All core samples were crushed to 25 mm on site with a barren quartz wash between each sample.

Newmont: RC drillholes were logged and sampled over 1m intervals and riffle split to obtain 2-5 kg samples. Wet intervals were grab sampled. Diamond core was half core sampled where mineralisation was deemed likely on a 1.0 m interval, which was adjusted where necessary to conform to lithological boundaries.

Tanami Gold: RC samples were taken at 1m intervals and split using a cone splitter. Two minor fractions were collected for sampling, with the bulk remaining fraction either stored in plastic green bags or dumped on the ground, dependent on the nature of drilling. A permanent record of each RC hole is kept by storing a small fraction of each 1m interval in chip trays, which were then logged by Tanami geologists. Recovery is recorded within the database and averages 98% for all drillholes. Diamond drilling is completed as either HQ/HQ3 or NQ2 core. All holes are meter marked and oriented using either a Reflex ACT or EZY MARK. Core recovery, RQD and fracture frequency are all measured on metre intervals. Core is sampled and analysed for gold on intervals of between 0.2m and 1.2m. All samples are half cored and are cut using an Almonte Diamond Pty Ltd automated belt driven core saw.

Northern Star: Stage 1 RC drilling – all bulk material collected on a 1m basis directly from cyclone in pre-labelled green plastic mining bags. Primary analysis determined using 4m speared composite samples at 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). Stage 2 RC drilling – single metre (1m) samples 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. As in stage 1 drilling 4m composite speared samples were taken for primary analysis, followed up by using before mentioned 1m samples. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray. For diamond drilling, selective samples were taken at the rig geologists' discretion focusing on areas with veins, sulphides and alteration. The core was cut in half using an Almonte Core saw with the orientation line and geologists marking retained on the left-hand side of the core

Sample Preparation and Analysis

Normandy: Normandy completed sample preparation in Alice Springs before analysis by 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.

Newmont: All Newmont samples were sent to ALS in Alice Springs for 50g fire assay (method Au-AA26). Sample preparation included jaw crushing all of the interval then pulverisation by a LM5. Barren quartz flushes were inserted between each sample to minimise sample cross contamination. No record has been located outlining where the Newmont RC grade control samples were assayed, but it is assumed that it would have been in an onsite laboratory.

Tanami Gold: Samples collected 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). 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. Analysis for gold is 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).

Northern Star: Samples collected by Northern Star were sent to ALS in Malaga, Perth and ALS in Adelaide. Gold (Au) was determined by conventional fire assay with a 50g charge and AAS finish. Multi-elements done by ICP. This was common to both DDH and RC samples.

QAQC

Historical QAQC data from the Normandy and Newmont drilling has not been located but reports reference that QAQC was carried out. It is likely that the drilling was of good quality as the area was mined by open pit methods. Since that time extensive exploration has been out by Tanami Gold and Northern Star and they have confirmed the existence of significant mineralisation below the Groundrush open pit. Programs of QAQC have been carried out by Tanami Gold and Northern Star. Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. A recognised laboratory has been used for analysis of samples. Overall, the QAQC data does not indicate any bias and supports the assay data used in the Mineral Resource.

Estimation Methodology and Classification

The open pit and underground wireframes were constructed in Leapfrog software after importing all the geology files and used a low grade assay cut-off of 0.5 g/t gold and 1.2 g/t gold for the open pit and underground, respectively. Some lower grade intercepts were included for the sake of continuity.

The cut-off levels used were based upon preliminary economic studies that suggested that these values were just below the break-even grade for an operating mine. No natural breaks in the gold populations could be established. The gold envelopes were modelled into a total of 54 individual domains or lodes for the open pit resource and 53 for the underground resource.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as most of the sampling was at 1m intervals. The open pit model was composited using a 1 metre fixed length method ±10% while the underground model used a 1 metre best fit method ±50% to account for the narrower samples taken in drill core. The composites were checked for spatial correlation within the objects, the location of the rejected composites, and zero composite values.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for some lodes. For the open pit resource, a high grade cut of between 10 g/t gold and 50 g/t gold was applied to some of the lodes for gold. This resulted in a total of 911 composites being cut or 1.27% of the data. For the underground resource a high grade cut of between 20 g/t gold and 50 g/t gold was applied to some of the lodes. The dilution skins were top cut between 4 g/t and 30 g/t gold. This resulted in a total of 723 composites being cut or 1.37% of the data.

Mineralisation continuity was examined via variography. A two-structured nested spherical model was found to model the experimental variograms reasonably well. The down-hole variogram provides the best estimate of the true nugget value, which varied from 0.19 to 0.73 for the open pit resource and 0.03 to 0.66 for the underground resource.

Two block models were created using Surpac software to encompass the full extent of the deposit. The open pit 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 while the underground block model used sub-blocking of 2.5m by 0.625m by 0.625m. 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.

Ordinary Kriging (**OK**) grade interpolation was used to estimate gold values in the block models with the search ellipse oriented to the variogram axes. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Where significant negative weights were encountered, and the model was outside of 10% versus the naïve and de-clustered means some domains were re-estimated using an octant search.

A first pass of radius 20-80m with a minimum number of samples of 3-6 samples and a second pass of radius 40-160m with a minimum number of 3-6 samples were used for Groundrush. A third pass of search radius 80-320m was used with 3-4 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 fourth pass using nearest neighbour estimation.

Bulk density was applied through oxidation state and rock type. Values for Groundrush were derived from drilling completed by the CTPJV in 2015 and 2016.

			Rock Type			
Oxidation State	**Sediments**	**Groundrush Dolerite**	**Groundrush Quartz Dolerite**	**Western Dolerite**	**Tombstone Dolerite**	**Back Fill/ Waste Dump**
Oxide	2.53	2.55	2.40	2.55	2.55	
Transitional	2.62	2.90	2.76	2.90	2.90	2.20
Fresh	2.74	2.94	2.86	2.99	2.94	

Table 4 - Groundrush Bulk Density

The Mineral Resource was classified in accordance with the 2012 JORC Code 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 25m by 25m, and where the continuity and predictability of the lode positions was good, and the estimation reconciled with the input data. 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.

Both the open pit and underground Mineral Resources stated have been derived from Whittle optimisations for the open pit resource and Deswik stope optimisations for the underground resource based on a A$2,700 per ounce gold price, benchmark costs and processing recoveries.

Mining, Metallurgy and Other Modifying Factors

It is assumed the Groundrush deposit will be mined by open pit (**OP**) and underground (**UG**) methods when a new mining operation can be established. The following mining factors and costs were used for the Whittle optimisation of the open pit resource as at 30 June 2023:

- OP Mining Recovery 98%
- OP Mining Dilution 10%
- Oxide Processing Recovery 95%
- Trans Processing Recovery 95%
- Oxide and backfill slope 45°
- Trans and Fresh slope 39°
- Backfill or Waste Dump Mining Cost $2.75/t
- Mining Cost $4.40/t
- Incremental Ore Mining Cost $4.95/t
- Open Pit Grade Control Cost $0.88/t
- Mill Opex Cost (2.0 Mtpa) $34.01/t
- ROM to mill transport distance 44km
- ROM to Mill cost $4.84/t
- Admin (G&A) cost $4.95/t
- Au Royalty 5%
- Au Price AU$2700/tr oz
- Deswik software was used for the underground resource stope optimisation.
- Stope Optimiser Assumptions
- HW planned dilution skin 0.5 m
- FW planned dilution skin 0.25 m
- Minimum Mining width 3 metres not including dilution skins
- Stope optimisation length 20 m along strike
- Sub level interval 25 m
- Optimise grade
- Stope optimisation -20 degrees
- Sub Stope Shapes enabled
- Smoothing fast
- UG mining unplanned recovery 5%
- UG mining unplanned dilution 5%
- Processing Recovery 95%
- UG Stoping cost $70 per tonne ore
- UG Opex Fixed Cost $5 per tonne ore
- Mill Opex Cost (2Mtpa) $30.92 per tonne
- ROM to mill transport $4.40 per tonne
- Admin $4.50 per tonne
- NT Factor $11.48 per tonne
- Au Royalty 5%
- Au Price A$2,700 troy ounce

Metallurgy Composite samples were collected for Extraction Test Work that included a total of 18 individual samples from within the Groundrush model. The samples were derived from quarter NQ2 diamond core. The metallurgical data shows good gold recoveries ranging from 86.7% - 99.3% with an average of 94.3%.

Ripcord Gold Deposit

Ripcord is located on Mining Lease ML22934, approximately 3 kilometres southeast of the Groundrush deposit and about 40 km northeast of the Central Tanami Mill site (refer Figure 1). The Ripcord Mineral Resource totals 0.75 Mt grading 2.1 g/t gold for 51 kozs (Table 1). It represents open-pit material reported at a cut-off grade of 0.60 g/t gold. The Mineral Resource estimate has been tightly constrained by a Whittle Optimisation. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free-milling processing recoveries. The Ripcord Mineral Resources were reported to the ASX by Tanami Gold NL on 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs*. The Company confirms that it is not aware of any new information or data that materially affects the Ripcord Open Pit Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 24 November 2022 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Ripcord Gold Deposit													
OP	0.6	-	-	-	640	2.1	43	110	2.2	8	750	2.1	51
Total		-	-	-	**640**	**2.1**	**43**	**110**	**2.2**	**8**	**750**	**2.1**	**51**

Table 5: Ripcord Mineral Resource Estimate

Geology and Geological Interpretation

The geology and deposit style at Ripcord appears to have similarities to the nearby Groundrush deposit, although it is yet to be fully determined if the host dolerite body is the same as that which hosts gold mineralisation at Groundrush. The host dolerite unit at Ripcord shows similar fractionation textures as observed at Groundrush, with fractionated quartz dolerite bounded on both sides by transitional quartz dolerite zones. Gold mineralisation is primarily hosted within the larger main dolerite body, with minor mineralisation extending in to the turbiditic sediments on the footwall contact. The main mineralised lodes consist of 1 - 6m wide zones of quartz veining that trend north to northwest and dip at 80° to the southwest.

The strike of the mineralised zone is about 1200 metres and the known down dip extent from drill data is about 150 metres. The width of the zone of primary mineralisation is in the order of 40 metres. There are 3 styles of mineralisation:

- supergene or flat lying lodes;
- dolerite hosted; and
- turbiditic sediment hosted.

The supergene or flat lying mineralisation dip shallowly to the west and are separated into north and southerly plunging bodies. They consist of narrow zones of quartz veining (1-3m) but with similar vein and alteration assemblages as the main steep lode system. Many of the mineralised veins also consist of carbonate and chlorite plus blebby pyrite and minor arsenopyrite. Alteration minerals related to mineralisation include silica, hematite and sericite. The supergene or flat lying zones have strikes of up to 150 metres and dip extents of up 100 metres and true thickness of 1-3 metres.

The dolerite and turbiditic sediment hosted mineralisation display similar strikes between 320° to 330° and dip about 60° west. There is a difference in the overall dimensions of the mineralised quartz lenses. The thickness of the mineralisation varies between 1 to 6 metres for both types however the dolerite hosted mineralisation is up to 150 metres in strike and 120 metres down dip while the sediment hosted mineralisation is up to 100 metres and 25 metres down dip.

Pyrite is the dominant sulphide present with accessory pyrrhotite, arsenopyrite, chalcopyrite and sphalerite. Dolerite and sediments both contain fine disseminated pyrite within the rock mass or on joint surfaces, generally in trace amounts. Proximal to, and within the mineralised zones there is up to 5% pyrite in blebby, stringer and disseminated forms. Arsenopyrite can form local accumulations of up to 2%.

Drill Information and Sampling

The various mineralised lodes at Ripcord were sampled using DDH, RC , AC, VC and RAB drill holes. Twenty-five RC drill holes for 207 intersection metres in the resource were drilled by Normandy during the period from 2001 to 2002. This represents 19.8% of the total intersection metres but has been backed up drilling completed by Tanami Gold who drilled the remaining 84 holes.

Hole Type	In Database		In Resource Model	
	Hole Number	Metres Drilled	Hole Number	Intersection Metres
AC	41	2,042.00		
DDH	6	1,087.60	5	72.79
RAB	256	12,806.00		
RC	167	23,001.00	104	971.00
VC	83	776.40		
Total	**553**	**39,713.00**	**109**	**1,043.79**

Table 6: Summary of Ripcord Drilling.

RC drill samples were collected at 1m intervals. Samples were collected at the rig, representing cutting's coarse fraction. For Tanami Gold drillholes, all samples are taken as 1 metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site. For Normandy, RC holes analytical samples were collected at 1 metre intervals using a rig mounted splitter with the bulk sample being bagged in green plastic bags.

Sample Preparation and Analysis

Normandy assays were sent to ALS–Chemex in Perth for Aqua Regia (PM203). Any samples that came back with an Aqua Regia result greater than 2ppm were automatically sent for A & B split Fire Assay (PM209), and those that assayed over 7ppm were sent for Screen Fire Assay. Tanami Gold sent samples from RPRC0001 to RPRC0037 to SGS lab in Perth for analysis by 50g Fire assay with atomic absorption finish (FAA505). Samples from RPRC0038 to RPRC0111 were submitted to Genalysis lab in Alice Springs for analysis by 50g Fire Assay with atomic absorption finish (FA50/AA).

QAQC

Detailed QAQC analysis for all drilling completed prior to 2013 has been reported by Tanami Gold. Standards, Blanks and Duplicates were inserted in the Normandy drilling sample stream. However, the frequency of these insertions is highly variable and expected values for the standards are not available, limiting the value of this data. QC samples were inserted routinely for all Tanami Gold drillholes. Standard samples were inserted every 25 metres, blank samples every 20 samples and duplicate samples were collected every 12m.

Programs of QAQC have been carried out by Tanami Gold. Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Coarse crush duplicates show repeatable although variable results. A recognised laboratory has been used for analysis of samples. Overall, the QAQC data does not indicate any bias and supports the assay data used in the Mineral Resource.

Estimation Methodology and Classification

Mineralisation interpretations were prepared by Tanami Gold and Northern Star. They were provided to MJM in Vulcan format files and converted to Surpac format by MJM. The supplied wireframes were snapped to drill holes supplied by Northern Star. Minor modifications were made to the wireframes to ensure that they represented the mineralisation. A low grade cut off of 0.5 g/t gold appears to be the driver, but some lower values have been incorporated in the wireframes to enhance the continuity.

The gold envelopes were modelled into a total of 58 individual domains or lodes. The mineralisation was classified into 3 categories, supergene or flat lying, dolerite and sediment hosted. The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for some lodes. A high grade cut of between 15 g/t gold and 20 g/t gold was applied to some of the lodes for gold. This resulted in a total of 12 composites being cut or 1.1% of the data.

Mineralisation continuity was examined via variography. The one metre composite data was transformed into a normal distribution using a normal scores transformation to help identify the main directions of mineralisation continuity from skewed data. A two-structured nested spherical model was found to model the experimental variograms reasonably well. The down-hole variogram provides the best estimate of the true nugget value, which was 0.32, 0.64 and 0.58 for Domains 27, 59, and 104 respectively. A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the block models with the search ellipse oriented to the variogram axes. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation.

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.

A first pass of radius 40m with a minimum number of samples of 2-8 samples and a second pass of radius 80m with a minimum number of 2-6 samples were used for Ripcord. A third pass of search radius 160m 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 after 3 passes were left without grade as a reflection of the paucity of samples in the lode.

Bulk density was applied through oxidation state and rock type and were derived from the Groundrush deposit.

	Rock Type	
Oxidation State	**Dolerite**	**Turbiditic Sediment**
Oxide	2.4	2.32
Transitional	2.7	2.58
Fresh	2.85	2.70

Table 7: Ripcord Bulk Density

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 20m by 25m, and where the continuity and predictability of the lode positions was 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. Over 700 metres of strike to the northwest did not meet this criterion as the area has been drilled on 100 metre sections. For reporting purposes only material that resides within an AU$2700 pit shell is included in the reported resource.

Mining, Metallurgy and Other Modifying Factors

Mining: It is assumed the Ripcord deposit will be mined by conventional open pit methods when a new mining operation can be established.

The Whittle Optimisation Assumptions used were:
- OP Mining Recovery 98%
- Op Mining Dilution 10%
- Processing Recovery 95%
- Mining Cost $4.40 per tonne ore
- Incremental Ore Mining Cost $0
- Open Pit Grade Control $0.88 per tonne
- Mill Opex cost (2.0Mtpa) $34.02 per tonne ore
- ROM to Mill cost $4.84/t
- Admin (G&A) cost $4.95/t
- Au Royalty 5%
- Au Price A$2,700/troy oz

Metallurgy: Tanami Gold submitted composite RC samples for metallurgical testing in 2013. These samples were ground to 150 microns and tested for recovery of gold. Recovery data was collated by oxidation state (weathering) and the average level was assigned to the Ripcord block model. Recovery in oxide samples ranged from 95.8% to 98.9% at an average of 97.2%, recovery in transitional samples returned a single value of 90.1% and in fresh samples recovery ranged from 82.5% to 94.3% at an average of 89.9%.

Jims Gold Deposit

Jims is located on Mineral Lease (Southern) MLS168, approximately 23 kilometres southwest of the Central Tanami Mill site (refer Figure 1). Mining at Jims was previously carried out between 1998 and 2001, with open pits established over the Main and Central deposits. Between 30 January 1998 and 25 June 2001, an estimated 1.66 million tonnes were mined at an average grade of 2.34 g/t gold resulting in 125,000 ounces of gold. The Central deposit produced 3,069 tonnes grading 2.67 g/t gold during the period from 10 June 1998 to 1 April 1999, yielding 263 ounces. The latter pit was abandoned due to poor reconciliation with the resource model.

The Jims Mineral Resource totals 1.5 Mt grading 2.3 g/t gold for 120 kozs (Table 1). It represents open pit material and underground material reported at cut-off grades of 0.70 g/t gold and 1.70 g/t gold, respectively (Table 1). The Mineral Resource estimates has been tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries.

The Jims Mineral Resources were reported to the ASX by Tanami Gold NL on 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs.* The Company confirms that it is not aware of any new information or data that materially affects the Jims Open Pit and Underground Mineral Resources, and the assumptions and technical parameters underpinning the estimates in the 24 November 2022 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Jims Gold Deposit													
OP	0.7	120	1.9	7	500	2.1	34	120	1.7	6	740	2.0	48
UG	SO @ 1.9	-	-	-	170	2.3	13	680	2.7	60	850	2.7	73
Total		**120**	**1.9**	**7**	**670**	**2.2**	**47**	**800**	**2.6**	**66**	**1,600**	**2.4**	**120**

Table 8: Jim's Mineral Resource Estimate

Geology and Mineralisation Interpretation

The Jims deposit is a Palaeoproterozoic, basalt and sediment-hosted vein-mineralised 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 series of vein and breccia lodes developed near basalt sediment contacts.

The mineralised trend at Jims Main strikes north-south, dipping moderately to steeply west in the upper extent but changes to a steep to east dipping below the 320m RL. 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 mineralisation has been interpreted down to 250 metres below the surface. 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.

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 Jims West area has previously not been mined.

Drill Information and Sampling

The various mineralised lodes at the Jims prospect were sampled using DDH, RC, AC, ditch witch (**DW**), VC, water bore (**WB**) and RAB drill holes. Drilling was completed by various owners since 1993, the majority conducted between 1993 and 2001.

Hole Type	In Database		In Resource Model	
	Hole Number	Metres Drilled	Hole Number	Intersection Metres
AC	57	4,581.00		
DDH	57	8,832.31	57	8,832.31
DW	1,944	95,327.00		
RAB	1,048	47,675.00		
RC	3,212	77,993.10	320	33,775.50
VC	58	63.00		
WB	35	4,582.00		
Total	**6,411**	**239,053.00**	**377**	**42,607.81**

Table 9 – Jims Drilling Summary

RC drill samples were collected at 1m intervals. Samples were collected at the rig, representing cutting's coarse fraction. Samples are taken at 1 metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site.

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. The use of booster air systems since mid-1998 overcame this problem.

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 on-site core yard located at the old exploration camp, approximately 5km to the south of the Tanami mill site.

Sample Preparation and Analysis

During mining operations 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 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 use a 50ml aliquot for all AAS readings. CTPJV drill samples were analysed by ALS Perth by Fire Assay.

QAQC

Programs of QAQC have been routinely carried out. Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Coarse crush duplicates show repeatable although variable results. A recognised laboratory has been used for analysis of samples. QAQC programs from drilling data from the 1990s to 2001 was carried out but that data has not been located. Significant mining was carried during that time period. Drilling by CTPJV appears to confirm the results, but no definite conclusion can be made about the quality of the earlier period of data collection. It is assumed to be representative. Overall, the QAQC data does not indicate any bias and supports the assay data used in the Mineral Resource.

Estimation Methodology and Classification

Mineralisation interpretations were prepared by Tanami Gold and Northern Star. They were provided to MJM in Vulcan format files and converted to Surpac format by MJM.

The supplied wireframes were snapped to drill holes supplied by Northern Star. Minor modifications were made to the wireframes to ensure that they represented the mineralisation. A low grade cut off appears to be in the order of 0.5 g/t gold, but some lower values have been incorporated in the wireframes to enhance the continuity. The gold envelopes were modelled into a total of 80 individual domains or lodes and mineralisation was classified into Jims Main, Jims Central and Jims West.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as most of the sampling was at 1m intervals.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 29 lodes. A high grade cut of 6-12g/t gold was applied to some of the lodes for gold. This resulted in a total of 136 composites from 29 lodes being cut or 2.6% of the data.

Mineralisation continuity was examined via variography. The one metre composite data was transformed into a normal distribution using a normal scores transformation to help identify the main directions of mineralisation continuity from skewed data. A two-structured nested spherical model was found to model the experimental variograms reasonably well.

A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the block models with the search ellipse oriented to the variogram axes. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Where high number of negative kriging weights were encountered those domains were rerun with an "octant" search.

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 Jims. A third pass of search radius 80-160m 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 4-28 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass.

Bulk density was applied through oxidation state and rock type.

	Material Type	
Oxidation State	**Basalt**	**Waste Dump**
Oxide	2.6	
Transitional	2.7	2.5
Fresh	2.8	

Table 10: Jims Bulk Density

The Mineral Resource was 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 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. 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.

Both the open pit and underground Mineral Resources stated have been derived from Whittle optimisations for the open pit resource and Deswik stope optimisations for the underground resource based on a A$2,700 per ounce gold price, benchmark costs and processing recoveries.

Mining, Metallurgy and Other Modifying Factors

Mining: It is assumed the Jims deposit will be mined by conventional 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.

Whittle Assumptions:

- Open Pit Mining Recovery 98%
- Open Pit dilution 10%
- Processing Recovery 85%
- Mining Cost $4.40 per tonne rock
- Oxide and Backfill slope 45 degrees
- Trans and fresh Slope 39 degrees
- Backfill cost 2.75 per tonne backfill
- Incremental Ore Mining Cost $0
- Open Pit grade Control Cost $0.88 per tonne ore
- Mill Opex cost (2.0Mtpa) $34.01 per tonne
- ROM to mill transport distance (Current Mill location) 28 km
- ROM to mill transport $2.83 per tonne
- Admin (G&A) $4.95 per tonne
- Au Royalty 5%
- Au Price AU$2700 per troy ounce

Metallurgy: Metallurgical testing was carried out in 1993 to test the metallurgical properties of Jims. The sighter test work campaign was carried out on a 12 metre intersection in JRC043 that had an average grade of 2.94 g/t gold. 25% of the of the gold was recovery by gravity concentration but it was noted that it was locked up in heavy particles. It was noted that leach kinetics were good and gold recovery was of the order of 96% with much of the extraction in the first 8 hours. Further 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%. A conservative 85% processing recovery was applied to the Whittle shell constraining the Mineral Resource estimate.

Hurricane-Repulse Gold Deposit

Hurricane-Repulse is located adjacent to the Central Tanami Mill site, with the deposit fully encompassed by MLS153 and MLS125 to MLS129 (refer Figure 1). Mining at Hurricane-Repulse was previously undertaken during the mid to late 1980s.

The Hurricane-Repulse Mineral Resource totals 1.5 Mt grading 3.8 g/t gold for 180 kozs (Table 1). It represents open-pit oxide and transition mineralisation, open-pit primary mineralisation, and underground primary mineralisation that is reported at cut-off grades of 0.63 g/t gold, 0.97 g/t gold, and 2.80 g/t gold, respectively. The Mineral Resource estimate has been tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free-milling processing recoveries. The Hurricane-Repulse Mineral Resources were reported to the ASX by Tanami Gold NL on 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs*. The Company confirms that it is not aware of any new information or data that materially affects the Hurricane-Repulse Open Pit and Underground Mineral Resources, and the assumptions and technical parameters underpinning the estimates in the 24 November 2022 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Hurricane-Repulse Gold Deposit													
OP Ox/Trans	0.63				510	2.6	42	160	2.1	11	670	2.5	53
OP Fresh	0.97				20	4.4	3	-	-	-	20	4.4	3
UG	SO @ 2.8				66	3.7	8	700	5.0	110	770	4.9	120
Total					**590**	**2.8**	**53**	**870**	**4.5**	**120**	**1,500**	**3.8**	**180**

Table 11: Hurricane-Repulse Mineral Resource Estimate

Geology and Geological Interpretation

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 either:

- 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 first vein style and in chalcopyrite in the second vein style.

The overall strike length of the known gold mineralisation on the Hurricane-Repulse trend is of the order of 1,750 metres and has a variable down dip extent of about 180 metres. 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 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 a several cross-cutting structures that vary in strike from 040° to 075° close to the basalt / sediment contact. This pattern continues into the basalt.

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 in length.

Drill Information and Sampling

The various mineralised lodes at the Hurricane-Repulse prospect have been sampled using DDH, RAB, RC, blast holes (**BH**) and WB drill holes. Drilling was completed by various owners since the 1980s.

Hole Type	In Database		In Resource Model	
	Hole Number	Metres Drilled	Hole Number	Intersection Metres
DDH	75	13,940.19	56	745.77
RAB	1,352	48,429.00		
RC	1,076	81,490.47	642	6,500.00
TR	6	945.00		
BH	242,607	705,655.00		
WB	5	404.00		
Total	**244,121**	**850,863.66**	**698**	**7,245.77**

Table 12: Summary of Hurricane-Repulse Drilling

For the drill holes drilled by Zapopan NL during the early 1990s, samples were collected 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. Most assaying was conducted at the onsite laboratory with a minor amount of assaying by outside laboratories during busy mine periods or for laboratory checking purposes. RC samples were assayed for gold only by fire assay with a 0.01 ppm detection limit. Once results were received for the 3 metre composites, mineralised zones for each RC hole were re-sampled at 1 metre intervals using a riffle splitter. 2kg samples were assayed for gold at the onsite laboratory. Further Zapopan NL geologically logged all RC drill holes on a 1 metre basis, noting lithology, colour, weathering, alteration, quartz veining, iron oxides and sulphides.

For drill holes completed by Otter Gold Mines, RC drill samples were collected at 1m intervals. Samples were collected at the rig, representing cutting's coarse fraction. For CTP drillholes, all samples are taken at 1 metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site.

For 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. 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. 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 TMJV core yard located at the old exploration camp, approximately 5km to the south of the Tanami mill.

For 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. The sample dryness varied from wet to dry. There were no wet samples used in the estimation. Tanami Gold personnel went to great pains to ensure that the cyclone was cleaned on a regular basis and that the splitter was also periodically cleaned. All the samples 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.

Diamond holes produced NQ2 sized diamond drill core. The diamond core was half core sampled down the length of the core. Samples from HRDD0001 to HRDD0004 were submitted to Genalysis Laboratories, Holes HRDD0005 to HRDD0013 were submitted to SGS Laboratories in Perth. Both set of samples were assayed using a 50g fire assay charge for gold with an atomic spectrometer finish with a 0.01ppm detection limit.

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. Primary analysis determined using 4m speared composite samples at geologist discretion. Composite samples with a grade above 0.5g/t had single metre bulk samples riffle split (using a 3-tier riffle splitter).

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. As in stage 1 drilling 4m composite speared samples were taken for primary analysis, followed up by using before mentioned 1m samples. All RC chips were logged using wet sieving technique retaining a sample in a plastic chip tray.

For Northern Star diamond drilling, selective samples were taken at the rig at the geologist's discretion focusing on areas with veins, sulphides, and alteration. The core was cut in half using an Almonte Core saw with the orientation line and geologists marking retained on the left-hand side of the core

Sample Preparation and Analysis

During the period 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 limited at the onsite laboratory.

During mining operations (mid 1990s to 2001) under the Tanami Gold Joint Venture 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.

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. The method detection was 0.01 ppm Au. Sample weights were generally around 3kg. Samples from diamond holes HRDD0001 to HRDD0004 were submitted to Genalysis Laboratories, Holes HRDD0005 to HRDD0013 were submitted to SGS Laboratories in Perth. Both set of samples were assayed using a 50g fire assay charge for gold with an atomic spectrometer finish with a 0.01ppm detection limit.

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.

QAQC

No conclusions can be derived about the data quality collected prior to Tanami Gold taking control of the Hurricane-Repulse project. The only comment that can be made is that the gold assays from drilling appear to be repeatable. The area was originally mined by Zapopan NL from the mid to late 1980s to March 1994. Most assays were completed in an onsite laboratory. Records may still exist at the Tanami Gold Mine so a literature search in the archive sheds is warranted as it is likely that the laboratory would have run some form of QAQC.

QAQC programs from drilling data from the mid-1990s to 2001 was carried out but that data has not been located. Significant mining was also carried during that time period. Drilling by CTP appears to confirm the results, but no definite conclusion can be made about the quality of the earlier period of data collection. It is assumed to be representative.

Programs of QAQC have been carried out by Tanami Gold and CTP. Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Coarse crush duplicates show repeatable although variable results. This may be due to the heterogeneity of the mineralisation. A recognised laboratory has been used for analysis of samples.

Overall, the QAQC data does not indicate any bias and supports the assay data used in the Mineral Resource.

Estimation Methodology and Classification

Mineralisation interpretations were prepared by RPM Global in Leapfrog software. Other files were provided to MJM in Vulcan format files and converted to Surpac format by MJM.

The supplied wireframes were snapped to drill holes supplied by RPM Global in Leapfrog. A low grade cut off of 0.5 g/t gold was used but some lower values have been incorporated in the wireframes to enhance the continuity. The wireframes were based upon previous interpretations, structural measurements of veins and grade control drilling.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. 6,432 out of a total of 6,762 samples were 1 metre lengths. All holes were composited to 1m as most of the sampling was at 1m intervals.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 42 out of 84 lodes. A high grade cut of 5-15 g/t gold was applied to some of the lodes for gold. This resulted in a total of 526 composites from 42 lodes being cut or 7.5% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Bulk densities were applied to the model by rock type and oxidation state. These values were derived from data collected by Tanami Gold in 2011 from diamond drill holes HRDD0005 to HRDD0013.

		Material Type		
Oxidation State	**Basalt**	**Sediments**	**Waste Dump**	**Backfill**
Oxide	2.29	2.51		
Transitional	2.60	2.65	2.2	2.2
Fresh	2.84	2.87		

Table 13: Hurricane-Repulse Bulk Density

Mineralisation continuity was examined via variography. The one metre composite data was transformed into a normal distribution using a normal scores transformation to help identify the main directions of mineralisation continuity from skewed data. A two-structured nested spherical model was found to model the experimental variograms reasonably well. The down-hole variogram provides the best estimate of the true nugget value.

A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the block models with the search ellipse oriented to the variogram axes. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. A first pass of radius 25-60m with a minimum number of samples of 3-6 samples and a second pass of radius 50-120m with a minimum number of 3-6 samples were used for Hurricane Repulse. A third pass of search radius 100-240m was used with a minimum number of 3-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4- 38 depending on the number of samples in the domain. Blocks that did not fill were given a fourth pass using nearest neighbour estimation.

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 25m by 12 to 25m, 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.

For reporting purposes only material that resides within an AU$2700 pit shell and underground optimisation are listed in the reported resource.

Mining, Metallurgy and Other Modifying Factors

Mining: It is assumed the Hurricane Repulse deposit will be mined by conventional 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.

The following mining factors and costs were used for the Whittle optimisation of the open pit resource:

- OP Mining Recovery 98%
- OP Mining Dilution 10%
- Oxide Processing Recovery %
- Trans Processing Recovery %
- Oxide and backfill slope 45°
- Trans and Fresh slope 39°
- Backfill or Waste Dump Mining Cost $2.75/t
- Mining Cost $4.40/t
- Incremental Ore Mining Cost $4.95/t
- Open Pit Grade Control Cost $0.88/t
- Mill Opex Cost (2.0 Mtpa) $34.01/t
- ROM to mill transport distance 2km
- ROM to Mill cost $4.84/t
- Admin (G&A) cost $4.95/t
- Au Royalty 5%
- Au Price AU$2700/tr oz
- Deswik software was used for the underground resource stope optimisation.
- Stope Optimiser Assumptions
- UG Mining Unplanned Recovery 5%
- UG Mining Unplanned Dilution 5%
- Processing Recovery 55%
- HW planned dilution skin 0.5 m
- FW planned dilution skin 0.25 m
- Minimum Mining width 3 metres not including dilution skins
- Stope optimisation length 20 m along strike
- Sub level interval 20 m
- Optimise metal
- UG Stoping cost $70 per tonne ore
- UG Backfill Cost $10 per tonne ore
- UG Opex Fixed Cost $5 per tonne ore
- Mill Opex Cost (2Mtpa) $30.92 per tonne
- ROM to mill transport $4.00 per tonne

- Admin $4.50 per tonne
- NT Factor $12.06 per tonne
- Capex $0
- Au Royalty 5%
- Au Price A$2,700 troy ounce

Metallurgy: 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. A conservative 55% processing recovery was applied to the Whittle and Stope optimisations constraining the Mineral Resource estimate.

Crusade Gold Deposit

Crusade is located on EL28282 and is located approximately 100km northeast of the Central Tanami Mill site. The Crusade Mineral Resource totals 1.3 Mt grading 2.3 g/t gold for 94kozs (Table 1). It represents open pit and underground material that is reported at cut-off grades of 0.77 g/t gold and 3.00 g/t gold, respectively. The Mineral Resource estimates has been tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries. The Crusade Mineral Resources were reported to the ASX by Tanami Gold NL on 24 November 2022 – *Mineral Resource Updates Completed For Five Gold Deposits On The Central Tanami Project Joint Venture Yields 1.5Mozs.* The Company confirms that it is not aware of any new information or data that materially affects the Crusade Open Pit and Underground Mineral Resources, and the assumptions and technical parameters underpinning the estimates in the 24 November 2022 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Crusade Gold Deposit													
OP	0.77				1,200	2.2	86	38	1.7	2	1,200	2.2	88
UG	SO @ 3.0				49	3.7	6				49	3.7	6
Total					**1,200**	**2.3**	**92**	**38**	**1.7**	**2**	**1,300**	**2.3**	**94**

Table 14: Crusade Mineral Resource Estimate

Geology and Mineralisation Interpretation

Crusade lies on the northerly striking and westerly dipping contact between biotite dacite and mafic volcanics. The contact dips between 60° to 70° west and strikes at about 020°.

The biotite dacite has been described as being porphyritic but also includes some lithic crystal tuffs, whilst the mafic volcanics are described as mainly pyroxene porphyritic units that are probably interpreted as flows. The dacite can be interpreted from airborne magnetic data and occurs as a magnetic low with an apparent thickness of 250 to 500 metres. The mafic volcanic unit can be seen clearly in the airborne magnetic data as a high that is striking at 020° and has an apparent thickness of about 100 metres.

Primary mineralisation is 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 overall strike of the economically significant mineralisation is about 680 metres and is made up of 9 lodes with 2 high grade subdomains. The mineralisation is striking at 020° and dips vary between 40° to 60° 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.

Drill Information and Sampling

The various mineralised lodes at the Crusade prospect were sampled using DDH, RC and RAB drill holes. Drilling has been completed at Crusade by various owners since 1994 through to 2019.

Hole Type	In Database		In Resource Model	
	Hole Number	Metres Drilled	Hole Number	Intersection Metres
DDH	20	3,466.70	19	429.14
RAB	294	7,352.00		
RC	98	10,121.00	38	497.61
Total	**412**	**20,940.00**	**57**	**926.75**

Table 15: Summary of Crusade Drilling.

RC drill samples were collected at 1m intervals. Samples were collected at the rig, representing cutting's coarse fraction. For CTP drillholes, all samples are taken at 1 metre intervals directly from the cone splitter, with the bulk sample collected in green bags and left on site. For RC holes drilled in the 1990s 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 pile were taken.

Sample Preparation and Analysis

A Tanami Gold historic report states that samples collected during the 1990's were submitted to the onsite laboratory. Analysis 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. All onsite analysis was performed with a 20ml aliquot.

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 sample collection.

QAQC

Programs of QAQC have been carried out by CTP Joint venture. Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Field duplicates show some degree of variability but are considered acceptable. A recognised laboratory has been used for analysis of samples. Overall, the QAQC data does not indicate any bias and supports the assay data used in the Mineral Resource.

Estimation Methodology and Classification

Mineralisation interpretations were prepared by Tanami Gold and Northern Star. They were provided to MJM in Vulcan format files and converted to Surpac format by MJM. Gold envelopes were modelled into a total of 9 individual domains or lodes and 2 subdomains. Subdomains or sub-lodes 21 – 22 are high grade lodes of lodes 1 – 2. The mineralisation was classified into 2 categories, mafic volcanic or biotite dacite hosted.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as most of the sampling was at 1m intervals.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 2 lodes. A high grade cut of 12g/t gold was applied to some of the lodes for gold. This resulted in a total of 5 composites being cut or 0.5% of the data. The high grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography. The one metre composite data was transformed into a normal distribution using a normal scores transformation to help identify the main directions of mineralisation continuity from skewed data. A two-structured nested spherical model was found to model the experimental variograms reasonably well.

A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the block models with the search ellipse oriented to the variogram axes. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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.

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 were given a fourth pass.

Bulk density was applied through oxidation state and rock type. These values were derived from average values for dacite and basalt and adjusted for oxidation.

	Rock Type	
Oxidation State	**Basalt**	**Biotite Dacite**
Oxide	2.29	2.51
Transitional	2.60	2.65
Fresh	2.84	2.87

Table 16: Crusade Bulk Density

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 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. 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. For reporting purposes only material that resides within an AU$2700 pit shell and underground optimisation are listed in the reported resource.

Mining, Metallurgy and Other Modifying Factors

Mining: It is It is assumed the Crusade deposit will be mined by conventional open pit methods when a new mining operation can be established.

Metallurgy: 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 however the fresh rock was not.

Further test work in 1996 was carried out by Normet Pty Ltd ("Normet") on CDH007 from 53 to 83 metres in a zone that was considered to represent saprolite and weathered bedrock. 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 in oxide and transitional material.

Further testing was carried out in 2020 by the Northern Star exploration department on ore grade material from drill holes SJRC0005-6 using a 500g Leachwell assay 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.

A conservative 94% for the oxide and transitional material and 40% for the fresh rock for the processing recovery was applied to the Whittle shell and Stope Optimisation constraining the Mineral Resource estimate.

Molech Area Deposits

Tanami Gold NL reported revised Mineral Resource estimates for the Beaver, Banjo, Bonsai, Orion, Cheeseman and Pendragon deposits five gold deposits in the Molech area of the Central Tanami Project Joint Venture (refer Figure 1), approximately 36km west of the Central Tanami mill site, in its ASX release dated 30 August 2023 entitled: *Mineral Resource Updates Completed for Gold Deposits in the Molech Area*.

The updates were part of an ongoing transition of the Central Tanami Project Mineral Resource estimates to allow these estimates to be reported in accordance with the 2012 JORC Code.

The estimates were compiled by mining consultants MJM using revised geological models that better reflect the mineralised systems. The reported Mineral Resources have been tightly constrained by Whittle and Stope Optimisations with deposit specific cut-off grades based on a A$2,700 per ounce gold price, haulage to the existing Central Tanami mill site, benchmark operating costs and free milling processing recoveries. The Company confirms that it is not aware of any new information or data that materially affects the Molech area Mineral Resources, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

Beaver Gold Deposit

Beaver is located on Mineral Lease Southern MLS180, approximately 36 kilometres west of the Central Tanami Mill site. Beaver was previously subject to open-pit mining by Otter between June 1999 and April 2001, during which 540 kt were mined at a reconciled grade of 3.3 g/t gold, resulting in 57 kozs of gold. Mining was halted due to geotechnical concerns arising from wall failures in the west and south walls, as well as water inflows.

The Beaver Mineral Resource represents open pit material and underground material reported at cutoff grades of 0.65 g/t gold and 1.80 g/t gold, respectively (Table 1). The Mineral Resource has been tightly constrained by Whittle and Stope Optimisations. Deposit specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries.

The Beaver Mineral Resources were reported to the ASX by Tanami Gold NL on 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area* (refer Table 1 and Table 17)*.* At this time, it represented a 1% increase in grade and a 6% decrease in tonnes and ounces when compared to the historic resource estimate. The Company confirms that it is not aware of any new information or data that materially affects the Beaver Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred				Total	
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Beaver Gold Deposit													
OP	0.65	-	-	-	100	3.9	13	41	4.1	5	140	4.0	18
UG	SO @ 1.80	-	-	-	110	3.3	12	140	3.2	14	250	3.3	26
Total		-	-	-	**210**	**3.6**	**24**	**180**	**3.4**	**20**	**390**	**3.5**	**44**

Table 17: Beaver Mineral Resource estimate

Geology and Geological Interpretation

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

Open-pit mapping shows the lithology as a thick sequence of mudstone to siltstone that strikes 315° and dips 70° south. Numerous faults transect the local geology.

Historic production came from two dominant 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 greater than 2 metres in width and consists of quartz veining and quartz stockworks within a 20-metre-wide altered shear, with visible gold noted in the quartz veining. The strike length is of the order of 210 metres. The northern vein is described 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 has been noted. The northern vein of the East Lode is only 0.5 metre wide within a 2 to 3 wide shear.

The best gold grades were found in the volcanoclastic sediments, and it was noted by Otter that widths and grades dropped off in the basalt, which are interpreted to crosscut the mineralisation.

Drilling Techniques

The various prospects on the Molech tenement, MLS180 were sampled using surface diamond core drill holes (**DDH**), reverse circulation (**RC**), air-core (**AC**), rotary air blast (**RB**) and water bore (**WB**) drill holes, as well as ditch witch trench lines (**DW**), with drilling completed by various owners from 1994 through to 2014.

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

Table 18: Summary of Molech (MLS180) Drilling and in the Beaver Resource Model

A total of 349 RC and 5 DDH drill holes were used to interpret and model the Beaver deposit (Table 18).

Sampling and Sub-Sampling Techniques

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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cut-off of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity. Geology interpretations were prepared from drill hole and aeromagnetic data.

Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations. The Beaver mineralisation wireframes were validated in Surpac to form three-dimensional models (**3DM**). The gold envelopes were modelled into a total of 31 individual domains or lodes, with mineralisation classified into 2 categories, sediment / basalt or laterite hosted.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 9 lodes. A high grade cut of 10 to 21 g/t gold was applied to some of the lodes for gold. This resulted in a total of 105 composites being cut or 3.9% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography to determine the appropriate kriging parameters for estimation. All variography was completed using Supervisor software. A block model was created using Surpac software to encompass the full extent of the deposit.

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.

OK grade interpolation was used to estimate gold values in the Beaver block model using the nugget, sill values and ranges determined from the variogram models. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Search ellipses and the minimum and maximum number of samples were lode dependent.

A first pass search radius of 25 to 40 metres with a minimum number of samples of 2 to 6 samples and a second pass radius of 50 to 80 metres with a minimum number of 2 to 6 samples were used. A third pass search radius of 100-160m was used with 2 to 4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4 to 26 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing		340° to 88°	
Dip		0° to 90°	
Plunge		-55° to 90°	
Major-Semi Major Ratio		1.0 to 3.0	
Major-Minor Ratio		3.0 to 5.0	
Search Radius	25-40	50-80	100-160
Minimum Samples	2 to 6	2 to 6	2 to 4
Maximum Samples	4 to 26	4 to 26	4 to 26
Max. Sam. per Hole	3 to 4	3 to 4	3
Block Discretisation		1 X by 2 Y by 1 Z	
Percentage Blocks Filled	81%	17%	2%

Table 19: Beaver Interpolation Parameters

Bulk density values were applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material.

Rock type	RL		Bulk Density
	From	**To**	Tonnes m^3
TR	surface	380	2.20
Sedimentary Felsic Volcanic	440	360	2.20
	360	340	2.30
	340	330	2.40
	330	320	2.50
	320	300	2.60
	300	290	2.70
	290	180	2.80
Basalt	surface	400	2.10
	400	380	2.20
	380	360	2.30
	360	340	2.40
	340	320	2.50
	320	300	2.60
	300	290	2.70
	290	180	2.80

Table 20: Bulk Density Values

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 21. The Whittle have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory.

Assumptions	Units	Values
OP Mining Recovery	%	98%
OP Mining Dilution	%	10%
Oxide and Trans Processing Recovery	%	90.0%
Fresh Processing Recovery	%	90.0%
Oxide and backfill slope	degrees	45
Trans and fresh slope	degrees	39
Backfill or Waste Dump Mining cost *	$/t rock	2.50
Mining cost *	$/t rock	4.00
Incremental Ore Mining cost *	$/t ore	0.00
Open Pit Grade Control cost *	$/t ore	0.80
Mill Opex cost (2.0Mtpa) *	$/t ore	30.92
ROM to mill transport distance (Current Mill location)	km	37.4
ROM to mill transport cost *	$/t ore	3.74
Admin (G&A) cost *	$/t ore	4.50
Northern Territory/Contingency Cost Factor (10%)	$/t ore	4.45
Capex	$	0
Au Royalty	%	5%
Au Price	AU$/tr.oz	2,700

Table 21: Molech Open Pit 2023 Whittle Assumptions

A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 7. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory (refer Table 22).

Assumptions	Units	Values
UG Mining Unplanned Recovery	%	5%
UG Mining Unplanned Dilution	%	5%
Processing Recovery	%	90%
HW planned dilution skin	m	0
FW planned dilution skin	m	0
Min Mining Width	m	2.0
Stope optimisation length (along strike)	m	10
Sub level interval	m	10
Optimise grade or metal		grade
U/G Stoping Costs	$/t ore	70.00
U/G Backfill Cost	$/t ore	0
U/G Opex Fixed Cost	$/t ore	5.00
Mill Opex cost (2.0Mtpa)	$/t ore	30.92
ROM to mill transport (Current Mill location)	$/t ore	3.74
Admin (G&A)	$/t ore	4.50
NT Factor (10%)	$/t ore	11.42
Capex	$	0
Au Royalty	%	5%
Au Price	AU$/tr.oz	2,700

Table 22: Stope Optimiser Assumptions.

Classification Criteria

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.

Mining, Metallurgy and Other Modifying Factors

Mining

Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 21 and 22.

Metallurgy

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 the material is derived from the Main Lode in the Beaver open-pit. The metallurgical test work undertaken on samples from Beaver may not be fully representative of the entire Molech area. The assumed Molech recovery rate of 90% for processing fresh rock is based on the test work from PGRC080. Further metallurgical test work is scheduled as a part of future drilling campaigns.

ample	**Grind**	**Gold Head Grade (g/t)**		**Residue (g/t)**		**Extraction**	**WEATHERING**
	µm	**Assay**	**Calc**	**@ 24 hrs**	**@ 48 hrs**	**%**	
PGRC075	95	6.44	6.66	0.64		90.4	OXIDE
	95	6.44	6.88	0.57		91.7	45m deep & mined
PGRC080	98	3.58	3.18	0.49		84.6	FRESH \ Transitional
	98	3.58	3.91	0.39		90	110m deep & insitu
	150	3.58	3.48		0.29	91.7	Just below Fresh boundary
	106	3.58	3.48		0.19	94.5	
	75	3.58	3.37		0.15	95.5	
	53	3.58	4.01		0.12	97	
PGRC081	100	2.9	2.52	0.37		85.3	OXIDE
	100	2.9	2.66	0.36		86.5	30 metre deep
	75	2.9	2.9		0.06	97.9	MINED
	53	2.9	2.81		0.05	98.2	

Table 23: Beaver Metallurgical Summary

Banjo Gold Deposit

Banjo is located on Mineral Lease Southern MLS180, approximately 36 kilometres west of the Central Tanami Mill site. Banjo underwent open-pit mining by Otter between June 1999 and April 2001, during which 100 kt were mined at a reconciled grade of 2.5 g/t gold, resulting in 8.3 kozs of gold.

The Banjo Mineral Resource represents underground material reported at a cut-off grade of 1.80 g/t gold (Table 1 and Table 24).

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Banjo Gold Deposit													
UG	SO @ 1.80	-	-	-	120	3.6	13	23	2.2	2	140	3.4	15
Total		-	-	-	**120**	**3.6**	**13**	**23**	**2.2**	**2**	**140**	**3.4**	**15**

Table 24: Mineral Resource estimates for the Banjo Gold Deposit

The Mineral Resource has been tightly constrained by a Stope Optimisation. Deposit specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries. The Banjo Mineral Resources were reported to the ASX by Tanami Gold NL on 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*. The Company confirms that it is not aware of any new information or data that materially affects the Banjo Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

Geology and Geological Interpretation

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

Geological interpretations 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 approximately 40 metres wide.

Gold mineralisation at the Banjo deposit is hosted by the 340° trending shear that occurs near the contact of the basalt and sedimentary units. These units are striking at 272° and dipping -80° south. Mineralisation has been described as being 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.

The overall strike of the gold mineralisation within the model is of the order of 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.

Drilling Techniques

The various prospects on the Molech tenement, MLS180 were sampled using DDH, RC, AC, RB and WB drill holes, as well as DW trench lines, with drilling completed by various owners from 1994 through to 2014. 136 RC and 2 DDH drill holes were used to interpret and model the Banjo deposit.

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

Table 25: Summary of Molech (MLS180) Drilling and in the Banjo Resource Model.

Sampling and Sub-Sampling Techniques

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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cutoff of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity. Geology interpretations were prepared from drill hole and aeromagnetic data. Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations.

The Banjo mineralisation wireframes were validated in Surpac to form 3DM. The gold envelopes were modelled into a total of 31 individual domains or lodes, with mineralisation classified into 2 categories, sediment / basalt or laterite hosted.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 4 lodes. A high grade cut of 7.5 to 25 g/t gold was applied to some of the lodes for gold. This resulted in a total of 29 composites being cut or 2.1% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography to determine the appropriate kriging parameters for estimation. All variography was completed using Supervisor software.

A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the Banjo block model using the nugget, sill values and ranges determined from the variogram models. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Search ellipses and the minimum and maximum number of samples were lode dependent.

A first pass search radius of 25 to 30 metres with a minimum number of samples of 6 samples and a second pass radius of 50 to 60 metres with a minimum number of 6 samples were used. A third pass search radius of 100-120m was used with 4 samples to ensure all blocks within the mineralised lodes were estimated.

The maximum number of samples ranged from 10 to 18 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing	315° to 350°		
Dip	0° to 90°		
Plunge	-49.7° to 15°		
Major-Semi Major Ratio	1.0 to 3.0		
Major-Minor Ratio	3.0 to 5.0		
Search Radius	25-30	50-60	100-120
Minimum Samples	6	6	4
Maximum Samples	10 to 18	10 to 18	10 to 18
Max. Sam. per Hole	4	4	3
Block Discretisation	1 X by 2 Y by 1 Z		
Percentage Blocks Filled	60%	35%	5%

Table 26: Banjo Interpolation Parameters

Bulk density was applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material. A listing of bulk density levels applied is provided in Table 20.

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 20. The Whittle assumptions have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory. A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 21. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

Classification Criteria

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.

Mining, Metallurgy and Other Modifying Factors

Mining - Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 21 and 22.

Metallurgy - The process recovery levels obtained from test work undertaken on samples from Beaver have been applied to all deposits in the Molech area, noting that they may not be fully representative. Refer to Table 23.

Bonsai Gold Deposit

Bonsai is located on Mineral Lease Southern MLS180, approximately 36 kilometres west of the Central Tanami Mill site. Bonsai underwent open-pit mining by Otter between June 1999 and June 2001, during which 160 kt were mined at a reconciled grade of 2.0 g/t gold, resulting in 10 kozs of gold. The Bonsai Mineral Resource represents open pit material and underground material reported at cutoff grades of 0.65 g/t gold and 1.80 g/t gold, respectively (Table 1).

The Mineral Resource has been tightly constrained by Whittle and Stope Optimisations. Deposit specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries. The Bonsai Mineral Resources were reported to the ASX by Tanami Gold NL on 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*. The Company confirms that it is not aware of any new information or data that materially affects the Bonsai Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Bonsai Gold Deposit													
OP	0.65	-	-	-	110	2.1	8	25	2.8	2	140	2.2	10
UG	SO @ 1.80	-	-	-	9	2.1	1	73	2.7	6	81	2.6	7
Total		-	-	-	**120**	**2.1**	**8**	**98**	**2.7**	**9**	**220**	**2.4**	**17**

Table 27: Bonsai Mineral Resources

Geology and Geological Interpretation

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 suggest that the basalt and sediments are striking about 280° to 335°, dipping steeply and display several fault offsets. Near surface mineralisation at Bonsai consists 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 the 290° trending shear zone, with better grades associated with silicification, quartz stockwork and veins.

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.

Drilling Techniques

The various prospects on the Molech tenement, MLS180 were sampled using DDH, RC, AC, RB and WB drill holes, as well as DW trench lines, with drilling completed by various owners from 1994 through to 2014. 199 RC and 4 DDH drill holes were used to interpret and model the Bonsai deposit.

In Molech Database			In Bonsai Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	158	13753		
DDH	17	2639.37	4	13.6
DW	3113	118578.9		
RB	1112	56955		
RC	1947	124001.2	199	1516.9
SL_RC	12	320		
WB	24	1764		
Grand Total	**6383**	**318,011**	**203**	**1530.5**

Table 28: Summary of Molech (MLS180) Drilling and in the Bonsai Resource Model

Sampling and Sub-Sampling Techniques

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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cutoff of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity. Geology interpretations were prepared from drill hole and aeromagnetic data. Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations.

The Bonsai mineralisation wireframes were validated in Surpac to form 3DM. The gold envelopes were modelled into a total of 31 individual domains or lodes, with mineralisation classified into 2 categories, sediment / basalt or laterite hosted.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 10 lodes. A high grade cut of 6 to 14 g/t gold was applied to some of the lodes for gold. This resulted in a total of

39 composites being cut or 2.5% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography to determine the appropriate kriging parameters for estimation. All variography was completed using Supervisor software. A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the Bonsai block model using the nugget, sill values and ranges determined from the variogram models. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Search ellipses and the minimum and maximum number of samples were lode dependent.

A first pass search radius of 25 to 50 metres with a minimum number of samples of 2 to 6 samples and a second pass radius of 50 to 100 metres with a minimum number of 2 to 6 samples were used. A third pass search radius of 100-200m was used with 2 to 4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 3 to 24 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing	303.6° to 150°		
Dip	0° to 90°		
Plunge	-15° to 15°		
Major-Semi Major Ratio	1.3 to 3.0		
Major-Minor Ratio	3.0 to 10.0		
Search Radius	25-50	50-100	100-200
Minimum Samples	2 to 6	2 to 6	2 to 4
Maximum Samples	3 to 24	3 to 24	3 to 24
Max. Sam. per Hole	3 to 4	3 to 4	3
Block Discretisation	1 X by 2 Y by 1 Z		
Percentage Blocks Filled	46%	42%	11%

Table 29: Bonsai Interpolation Parameters

Bulk density was applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material. A listing of bulk density levels applied is provided in Table 20.

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 21. The Whittle assumptions have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 22. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

Classification Criteria

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.

Mining, Metallurgy and Other Modifying Factors

Mining Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 21 and 22. Metallurgy The process recovery levels obtained from test work undertaken on samples from Beaver have been applied to all deposits in the Molech area, noting that they may not be fully representative. Refer to Table 23.

Orion Gold Deposit

Orion is located on Mineral Lease Southern MLS180, approximately 36 kilometres west of the Central Tanami Mill site. Orion underwent open-pit mining by Otter between June 1999 and June 2001. During this period, 100 kt were mined from the larger northern open pit at a reconciled grade of 2.7 g/t gold, resulting in 8.6 kozs of gold. An additional 17 kt were mined from the southern open pit at a reconciled grade of 3.7 g/t gold, yielding 2.0 kozs of gold. The Orion Mineral Resource represents open pit material and underground material reported at cut-off grades of 0.65 g/t gold and 1.80 g/t gold, respectively (Table 1).

The Mineral Resource has been tightly constrained by Whittle and Stope Optimisations. Deposit-specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs, and free milling processing recoveries. The Orion Mineral Resources were reported by Tanami Gold NL to the ASX on 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*.

At this time, it represented a 12% increase in grade and a 56% decrease in tonnes, as well as a 51% decrease in ounces when compared to the historic resource estimate. The Company confirms that it is not aware of any new information or data that materially affects the Orion Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred				Total	
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Orion Gold Deposit													
OP	0.65	-	-	-	39	3.1	4	9	5.7	2	47	3.6	5
UG	SO @ 1.80	-	-	-	27	2.3	2	17	2.6	1	43	2.4	3
Total		**-**	**-**	**-**	**65**	**2.8**	**6**	**25**	**3.7**	**3**	**91**	**3.0**	**9**

Table 30: Orion Mineral Resource

Geology and Geological Interpretation

The Orion North and South deposits are hosted by a 40 metre wide regional shear near the contact of basalt and sedimentary units, which 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 outside of the shear strikes at about 330° and has an 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.

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 are 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. Orion South gold mineralisation is hosted by sedimentary units.

Drilling Techniques

The various prospects on the Molech tenement, MLS180 were sampled using DDH, RC, AC, RB and WB drill holes, as well as DW trench lines, with drilling completed by various owners from 1994 through to 2014. 137 RC drill holes were used to interpret and model the Orion deposit.

In Molech Database			**In Orion Resource Model**	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	158	13753		
DD	17	2639.37		
DW	3113	118578.9		
RB	1112	56955		
RC	1947	124001.2	137	909
SL_RC	12	320		
WB	24	1764		
Grand Total	**6383**	**318,011**	**137**	**909**

Table 31: Summary of Molech (MLS180) Drilling and in the Orion Resource Model

Sampling and Sub-Sampling Techniques

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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cutoff of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity. Geology interpretations were prepared from drill hole and aeromagnetic data. Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations.

The Orion mineralisation wireframes were validated in Surpac to form 3DM. The gold envelopes were modelled into a total of 27 individual domains or lodes, with mineralisation classified into 2 categories, sediment / basalt or laterite hosted.

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains.

All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 9 lodes. A high grade cut of 8 to 22 g/t gold was applied to some of the lodes for gold. This resulted in a total of 47 composites being cut or 5.4% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography to determine the appropriate kriging parameters for estimation. All variography was completed using Supervisor software. A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the Orion block model using the nugget, sill values and ranges determined from the variogram models. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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. Search ellipses and the minimum and maximum number of samples were lode dependent.

A first pass search radius of 20 to 30 metres with a minimum number of samples of 2 to 6 samples and a second pass radius of 40 to 60 metres with a minimum number of 2 to 6 samples were used. A third pass search radius of 80-120m was used with 2 to 4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 6 to 30 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing	320° to 345°		
Dip	0° to 90°		
Plunge	-4.98° to 30°		
Major-Semi Major Ratio	1.2 to 3.30		
Major-Minor Ratio	1.7 to 5.00		
Search Radius	20-30	40-60	80-120
Minimum Samples	2 to 6	2 to 6	2 to 4
Maximum Samples	6 to 30	6 to 30	6 to 30
Max. Sam. per Hole	3 to 4	3 to 4	3
Block Discretisation	1 X by 2 Y by 1 Z		
Percentage Blocks Filled	32%	47%	19%

Table 32: Orion Interpolation Parameters

Bulk density was applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material. A listing of bulk density levels applied is provided in Table 20.

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 21. The Whittle assumptions have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory. A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 22. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

Classification Criteria

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.

Mining, Metallurgy and Other Modifying Factors

Mining - Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 20 and 21.

Metallurgy - The process recovery levels obtained from test work undertaken on samples from Beaver have been applied to all deposits in the Molech area, noting that they may not be fully representative. Refer to Table 23.

Cheeseman Gold Deposit

Cheeseman is located on Mineral Lease Southern MLS180, approximately 36 kilometres west of the Central Tanami Mill site. Cheeseman underwent open-pit mining by Otter between June 1999 and June 2001. During this period, 59 kt were mined at a reconciled grade of 3.9 g/t gold, resulting in 7.4 kozs of gold. Most of the production ore was extracted from an 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. The Cheeseman Mineral Resource represents open pit material and underground material reported at cut-off grades of 0.65 g/t gold and 1.80 g/t gold, respectively (Table 1).

The Mineral Resource has been tightly constrained by Whittle and Stope Optimisations. Deposit specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs and free milling processing recoveries. The Cheeseman Mineral Resources were reported by Tanami Gold NL to the ASX on the 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*. At this time, it represented a 6% increase in grade and a 49% decrease in tonnes and 46% decrease in ounces when compared to the historic resource estimate. The Company confirms that it is not aware of any new information or data that materially affects the Cheeseman Mineral Resource and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Cheeseman Gold Deposit													
OP	0.65	-	-	-	11	4.8	2	8	2.3	1	19	3.7	2
UG	SO @ 1.80	-	-	-	-	-	-	50	3.5	6	50	3.5	6
Total		-	-	-	**11**	**4.8**	**2**	**59**	**3.4**	**6**	**69**	**3.6**	**8**

Table 33: Cheeseman Mineral Resource

Geology and Geological Interpretation

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. 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

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°.

Drilling Techniques

The various prospects on the Molech tenement, MLS180 were sampled using DDH, RC, AC, RB and WB drill holes, as well as DW trench lines, with drilling completed by various owners from 1994 through to 2014. 79 RC drill holes were used to interpret and model the Cheeseman deposit.

In Molech Database			In Cheeseman Resource Model	
Hole Type	**No. Holes**	**Metres drilled**	**No. Holes**	**Intersection Metres**
AC	158	13753		
DD	17	2639.37		
DW	3113	118578.9		
RB	1112	56955		
RC	1947	124001.2	79	572
SL_RC	12	320		
WB	24	1764		
Grand Total	**6383**	**318,011**	**79**	**572**

Table 34: Summary of Molech (MLS180) Drilling and in the Cheeseman Resource Model

Sampling and Sub-Sampling Techniques

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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cutoff of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity. Geology interpretations were prepared from drill hole and aeromagnetic data. Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations. The Cheeseman mineralisation wireframes were validated in Surpac to form 3DM. The gold envelopes were modelled into a total of 19 individual domains or lodes, with mineralisation classified into 2 categories, sediment / basalt or laterite hosted. The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Surpac software was then used to extract down hole composites within the different resource domains.

All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined. Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were required for 9 lodes. A high grade cut of 5 to 25 g/t gold was applied to some of the lodes for gold. This resulted in a total of 27 composites being cut or 4.7% of the data. The high-grade cuts were applied to the composite data prior to grade estimation.

Mineralisation continuity was examined via variography to determine the appropriate kriging parameters for estimation. All variography was completed using Supervisor software. A block model was created using Surpac software to encompass the full extent of the deposit. 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.

OK grade interpolation was used to estimate gold values in the Cheeseman block model using the nugget, sill values and ranges determined from the variogram models. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode. 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. Search ellipses and the minimum and maximum number of samples were lode dependent.

A first pass search radius of 20 to 30 metres with a minimum number of samples of 3 to 6 samples and a second pass radius of 40 to 60 metres with a minimum number of 3 to 6 samples were used. A third pass search radius of 80-120m was used with 3 to 4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 8 to 24 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing	315° to 345.9°		
Dip	0° to 90°		
Plunge	0° to 30°		
Major-Semi Major Ratio	1.5 to 4.69		
Major-Minor Ratio	3.0 to 10.00		
Search Radius	20-30	40-60	80-120
Minimum Samples	3 to 6	3 to 6	3 to 4
Maximum Samples	8 to 24	8 to 24	8 to 24
Max. Sam. per Hole	3 to 4	3 to 4	3
Block Discretisation	1 X by 2 Y by 1 Z		
Percentage Blocks Filled	40%	42%	17%

Table 35: Cheeseman Interpolation Parameters

Bulk density was applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material. A listing of bulk density levels applied is provided in Table 20.

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 21. The Whittle assumptions have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory. A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 22. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

Classification Criteria

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. –

Mining, Metallurgy and Other Modifying Factors

Mining Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 21 and 22.

Pendragon Gold Deposit

Pendragon is located on Exploration Licence 26925, around 36 kilometres west of the Central Tanami Mill site. The Mineral Resource represents open pit material and underground material that is reported at cut-off grades of 0.65 g/t gold and 1.80 g/t gold, respectively (Table 1). The Mineral Resource has been tightly constrained by Whittle and Stope Optimisations. Deposit specific cut-off grades were determined based on an A$2,700 per ounce gold price, haulage to the existing Central Tanami Mill Site, benchmark operating costs and free milling processing recoveries.

The Pendragon Mineral Resources were reported by Tanami Gold NL to the ASX on 30 August 2023 – *Mineral Resource Updates Completed for Gold Deposits In The Molech Area*. This report represented the maiden Mineral Resource estimate for this deposit. The Company confirms that it is not aware of any new information or data that materially affects the Pendragon Mineral Resource, and the assumptions and technical parameters underpinning the estimates in the 30 August 2023 report continue to apply and have not materially changed.

	COG (g/t Au)	Measured			Indicated			Inferred			Total		
		Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)	Tonnes (kt)	Gold (g/t)	Ounces (kozs)
Pendragon Gold Deposit													
OP	0.65	-	-	-	-	-	-	24	2.2	2	24	2.2	2
UG	SO @ 1.80	-	-	-	-	-	-	17	2.3	1	17	2.3	1
Total		-	-	-	-	-	-	**41**	**2.3**	**3**	**41**	**2.3**	**3**

Table 36: Pendragon Mineral Resource

Geology and Geological Interpretation

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. Geological interpretations suggest that the basalt and sediments are striking between 300° to 330°. Gold mineralisation at Pendragon is hosted within a 300° trending shear. 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.

Drilling Techniques

The various prospects on the Pendragon lease EL26925 were sampled using RC, AC, RB and WB drill holes and DW trench lines ("DW"). 108 RC holes were drilled from 1994 to 2001 at Pendragon by Otter. Tanami Gold drilled 10 RC holes in 2012 to explore for mineralisation intersected in earlier RB and AC holes. 11 RC drill holes were used to interpret and model the Pendragon deposit.

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**	**99182**	**11**	**85**

Table 37: Summary of EL26925 Drilling and in the Pendragon Resource Model

Sampling and Sub-Sampling Techniques

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. The use of booster air systems since mid-1998 overcame this problem. For Tanami Gold drillholes, all samples are 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.

Sample Analysis Method

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

Geology and mineralisation interpretations were prepared by MJM in Surpac software. A low grade cut-off of 0.5 g/t gold was used for gold mineralisation wireframes, but some lower values have been incorporated to enhance the continuity.

Geology interpretations were prepared from drill hole and aeromagnetic data. Major structural features were interpreted from the geophysics to guide the geology and mineralisation interpretations. The Pendragon mineralisation wireframes were validated in Surpac to form 3DM. The gold envelopes were modelled into a total of 17 individual domains or lodes, but only 1 to 10 were used in the model. The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections.

Surpac software was then used to extract down hole composites within the different resource domains. All holes were composited to 1m as 99% of the sampling was at 1m intervals. Individual composite files were created for each of the domains in the wireframe models and summary statistics determined.

Analysis of statistics and histogram plots for all lodes suggested that high grade cuts were not necessary, due to the low number of samples in the domains. There was insufficient data to calculate variograms for Pendragon.

A block model was created using Surpac software to encompass the full extent of the deposit. 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.

Inverse Distance Squared ("ID2") grade interpolation was used to estimate gold values in the Pendragon block model. For all zones in the block model, the wireframe interpretations were used as hard boundaries in the interpolation. That is, only grades inside each lode were used to interpolate the blocks inside the lode.

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.

Search ellipses and the minimum and maximum number of samples were lode dependent. A first pass search radius of 20 to 30 metres with a minimum number of samples of 4 to 6 samples and a second pass radius of 40 to 60 metres with a minimum number of 4 to 6 samples were used. A third pass search radius of 80-120m was used with 3 to 4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 6 to 30 depending on the number of samples in the domain.

Parameter	Pass 1	Pass 2	Pass 3
Search Type	Ellipsoid	Ellipsoid	Ellipsoid
Bearing	300° to 335°		
Dip	0° to 90°		
Plunge	0° to 10°		
Major-Semi Major Ratio	1.5 to 2.00		
Major-Minor Ratio	2.0 to 5.00		
Search Radius	20-30	40-60	80-120
Minimum Samples	4 to 6	4 to 6	3 to 4
Maximum Samples	7 to 24	6 to 30	6 to 30
Max. Sam. per Hole	3 to 7	3 to 7	3 to 7
Block Discretisation	1 X by 2 Y by 1 Z		
Percentage Blocks Filled	24%	29%	9%

Table 38: Pendragon Interpolation Parameters

Bulk density was applied by RL over 20 metre increments using average bulk density measurements for basalt and sediment and then compensating for oxide and transitional material. A listing of bulk density levels applied is provided in Table 20.

Cut-off Grades

The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65 g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste. The 0.65 g/t gold breakeven in-situ cut-off for the open-pit Mineral Resource estimate was based on the Whittle assumptions provided in Table 21.

The Whittle assumptions have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory. A 1.8 g/t gold breakeven in-situ cut-off for the underground Mineral Resource estimate was based on the Optimiser Assumptions provided in Table 22. These have been derived from Northern Star benchmarks and studies, factored to reflect costs in the Northern Territory.

Classification Criteria

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 wide spaced drilling. 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

Mining - Whittle and Stope Optimisations were carried out as part of the Mineral Resource estimate process. The Whittle and Stope Optimisation assumptions have been derived from Northern Star benchmarks and studies and factored to reflect costs in the Northern Territory. These are provided in Tables 20 and 21.

Metallurgy - The process recovery levels obtained from test work undertaken on samples from Beaver have been applied to all deposits in the Molech area, noting that they may not be fully representative. Refer to Table 23.

Historical Estimates (JORC 2004)

The CTP JV assets include substantial historical estimates for mineral resources at 13 gold deposits, together totalling approximately 1.0 Moz contained gold.

These estimates have not been subjected to economic constraints and were first released to the ASX by Tanami Gold on:

- 8 June 2011 – *Tanami Lifts Gold Resources to 2.3Moz and unveils a 400,000oz Ore Reserve*.

Mount Gibson notes that at this time, a Competent Person has not done sufficient work to classify these historical estimates in accordance with the JORC Code (2012) and it is uncertain that following further evaluation and/or further exploration work, that the historical estimates will be able to be reported in accordance with the JORC Code (2012).

The historical estimates for mineral resources on MLS153, MLS167 and MLS168 were prepared in accordance with the 2004 JORC Code and were completed using MineMap, Vulcan and Micromine software packages comprising a combination of ellipsoidal inverse distance and ordinary kriging grade interpolation methods:

- Grade estimation was constrained to material within >0.5g/t mineralisation outlines.
- Variable gold assay top cuts were applied based on geostatistical parameters and historical production reconciliation.
- Resources were reported above 0.7g/t block model grade, and above 2.5g/t block grade for mineralisation at the Carbine deposit, within MLS167, occurring below the southern plunge extent of a design pit shell optimised at A$1350 per ounce gold price.
- Stockpile figures were estimated from those previously reported in Otter Gold Mines NL 2001 Mineral Resource estimate less recorded treatment by Newmont Asia Pacific.
- Tonnes and ounces rounded to the nearest thousand and grade rounded to 0.1g/t. Rounding may affect tallies.

No more recent estimates have been completed or provided to Mount Gibson by Northern Star or MJM. The mineral resources estimates as at 30 June 2023 (as contained in the Competent Person Report) are, so as Mount Gibson is aware, the most recent estimates approved by the CTP JV partners.

Mount Gibson considers the historical estimates, which were prepared under the previous 2004 edition of the JORC Code, to be reliable as they have been continually reported by the CTP JV partners since the joint venture was established in 2015. MJM, on behalf of the CTP JV, has been undertaking a systematic process to review and bring the historic estimates into compliance with the JORC Code 2012.

Work is ongoing to complete updated estimates for MLS153, representing the Southern, Miracle, Bastille, Dinky, Thrasher deposits, and MLS 167 representing the Phoenix, Redback Rise, Lynx, Legs, Bulldog, Dogbolter, and Carbine deposits in the Tanami Mine Corridor area, along with Camel Bore and the previously unreleased Galifrey deposit in the Tanami Southwest area to bring them into compliance with the JORC Code (2012).

Work required to achieve this includes reviewing the veracity of the source data on which the estimates are based, potentially undertaking additional drilling and assaying if necessary, and applying updated economic and design parameters (including contemporary open pit and stope optimisations) to the estimates. Mount Gibson expects the work to be undertaken progressively over coming periods and reported as required as new estimates are completed.

The historical estimates are considered material to the CTP JV representing approximately 37% of total estimated contained ounces of gold. As Mount Gibson is seeking to acquire a 50% interest in the CTP JV from Northern Star, the Company accordingly considers these historical estimates to be material given its intention, through the proposed acquisition, to seek to progress the CTP JV towards production in the shortest possible timeframe.

MLS153

The Southern, Miracle, Bastille, Dinky, and Thrasher deposits are collectively reported under MLS153. They are located adjacent to the Central Tanami Mill site, with the deposits fully encompassed by MLS153 (Refer Figure 1). The MLS153 historical estimates represent material that is reported at a cut-off grade of 0.70 g/t gold (Table 1). No economic constraints have been applied to these historical estimates. The Company confirms that it is not aware of any new information or data that materially affects the MLS153 estimates, and the assumptions and technical parameters underpinning the estimates in the 8 June 2011 report continue to apply and have not materially changed.

MLS167

The Carbine, Phoenix, Redback Rise, Lynx, Legs, Bulldog, and Dogbolter deposits are collectively reported under MLS167. They are located about 5 km southwest of the Central Tanami Mill site, with the deposits fully encompassed by MLS167. The MLS167 historical estimates represent material that is reported at a cut-off grade of 0.70 g/t gold (Table 1). No economic constraints have been applied to these historical estimates. The Company confirms that it is not aware of any new information or data that materially affects the MLS167 estimates, and the assumptions and technical parameters underpinning the estimates in the 8 June 2011 report continue to apply and have not materially changed.

MLS168

The Camel Bore deposit is reported under MLS168. It is located approximately 23 kilometres southwest of the Central Tanami Mill site, with the deposit fully encompassed by MLS168. The MLS168 historical estimate represents material that is reported at a cut-off grade of 0.70 g/t gold (Table 1). No economic constraints have been applied to this historical estimate. The Company confirms that it is not aware of any new information or data that materially affects the MLS168 estimates, and the assumptions and technical parameters underpinning the estimates in the 8 June 2011 report continue to apply and have not materially changed.

Appendix 1 - JORC Table 1
Groundrush Gold Deposit

Section 1 - Sampling Techniques and Data

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

Criteria	Commentary
Sampling techniques	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. 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. 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 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	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	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 are considered consistent. 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. 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. 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	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. 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 entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	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. 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. Sample preparation was completed at various labs depending on the drilling campaign and are deemed appropriate. Normandy completed sample preparation in Alice Springs 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. 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. 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. 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	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. MPAES instrument finish was used to measure gold levels. The methodology used measures total gold. In 2012 samples were sent to Intertek Genalysis (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. No geophysical tools were used to determine any element concentrations. 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 stated 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	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. No twinned holes were drilled for this data set. 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. 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	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 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. Collar coordinates were recorded in MGA94 Zone 52. 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	Drillhole spacing at Groundrush varies, although minimum 25m spacing was targeted during the design and drilling phases. 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. 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	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends. No sampling bias is considered to have been introduced by the drilling orientation.
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	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	Commentary
Mineral tenement and land tenure status	The Groundrush Gold Deposit is located in the Tanami Region in the Northern Territory on Mining Licence (Groundrush) ML22934, approximately 45km northeast of the Central Tanami Mill site. ML22934 covers an area of 39.5 sqkm and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,211 sqkm 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 ten Exploration Licences, eight of which are granted, and two applications, nineteen Mineral Leases, and one Mining Licence. 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. ML22934 is granted and in good standing.
Exploration done by other parties	The Groundrush 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. Drilling reported with this release is contiguous with the Groundrush 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	Rocks of the Killi Killi Formation host the Groundrush deposit exposed in a narrow N- to NNW-trending corridor flanked by lobes of the younger Frankenia Dome granite. Groundrush 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. 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.
Drill hole information	This release pertains to the reporting of Mineral Resources. Exploration results have previously been reported to the ASX by the Joint Venture parties. Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information.
Data aggregation methods	This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	The reported drillholes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends. The exact 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. When exploration results were previously disclosed, only downhole lengths were reported. True widths are not known.
Diagrams	Appropriate diagrams have previously been reported to the ASX by the Joint Venture parties.

Balanced Reporting	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.
	Both high and low grades have been reported accurately, clearly identified with the drill-hole attributes and 'From' and 'To' depths. All intercepts have been reported regardless of grade previously by the Joint Venture parties..
Other substantive exploration data	Exploration results have previously been reported to the ASX by the Joint Venture parties.
Further work	Drilling is planned to infill and expand the current resource envelope. Appropriate diagrams accompany this release.

Section 3 - Estimation and Reporting of Mineral Resources

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

Criteria	Commentary
Database integrity	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: 1. Down-hole survey depths did not exceed the hole depth as reported in the collar table. 2. Visual inspection of drill hole collars and traces in Surpac. 3. Assay values did not extend beyond the hole depth quoted in the collar table. 4. 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	A number of site visits have been conducted by the CP, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	The confidence in the geological interpretation is moderate as there is no 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 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. 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. 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	Gold mineralisation 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. Further, relative metal proportions were estimated to be approximately 70% in the steep shear hosted lodes, 20% in the east dipping lodes and the remaining 10% in the west dipping lodes. The known strike length of the gold mineralisation is about 1.9 km. The down dip extent is about 450 metres with a number of lodes dipping variably. Individual lenses down dip extent varies from 20 to 250 metres. True thickness varies from a few metres to over 20 metres. Overall, the 32 steep and 22 flatter dipping lodes were modelled for the open pit wireframe with one less flatter dipping lode for the underground wireframe. The steep lodes make up 80% of the resource and confidence in these lodes is high. The flatter dipping lodes intersect the steep lodes in many areas. The steep lodes are generally striking around 340o but varied from 323° degrees to 355o and have a total strike length of 1900 metres. They dip about -60° to -70° west but range from -32° to -80° west. These lodes are mostly plunging to the south at about 10°. The strike length of the lodes varies from 50 to 970 metres, and they extend down dip from 50 to a maximum of 250 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 flat lying lodes are only well established in the mined-out areas where they were defined by closed spaced grade control drilling. These lodes crosscut the steep lodes and are difficult to interpret from the exploration drilling data. They are largely confined to areas of dolerite and strike between 325° to 340°, dip from 25° to 50° and plunge southwest between 15° to 24°. 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. Volumetrically they represent about 20% of the total resource with most of that volume intersecting steep lodes.
Estimation and modelling techniques	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 multi-variate and bi-variate 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 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 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. Where significant negative weights were encountered, and the model was outside of 10% versus the naïve and declustered means some domains were re-estimated using an octant search. A first pass of radius 20-80m with a minimum number of samples of 3-6 samples and a second pass of radius 40-160m with a minimum number of 3-6 samples were used for Groundrush. A third pass of search radius 80-320m was used with 3-4 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 3-28 depending on the number of samples in the domain. Blocks that did not fill after 3 passes 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 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	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	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 and a AU$2700 optimisation pit shell for open pit. The underground resource is reported by a AU$2700 stope optimisation that includes dilution. These figures were based upon financial studies by MoJoe Mining Pty Ltd.
Mining factors or assumptions	It is assumed the Groundrush 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 Whittle optimisation of the open pit resource: - OP Mining Recovery 98% - OP Mining Dilution 10% - Oxide Processing Recovery 95% - Trans Processing Recovery 95% - Oxide and backfill slope 45° - Trans and Fresh slope 39° - Backfill or Waste Dump Mining Cost $2.75/t - Mining Cost $4.40/t - Incremental Ore Mining Cost $4.95/t - Open Pit Grade Control Cost $0.88/t - Mill Opex Cost (2.0 Mtpa) $34.01/t - ROM to mill transport distance 44km - ROM to Mill cost $4.84/t - Admin (G&A) cost $4.95/t - Au Royalty 5% - Au Price AU$2700/tr oz - Deswik software was used for the underground resource stope optimisation. - Stope Optimiser Assumptions - HW planned dilution skin 0.5 m - FW planned dilution skin 0.25 m - Minimum Mining width 3 metres not including dilution skins - Stope optimisation length 20 m along strike - Sub level interval 25 m - Optimise grade - Stope optimisation -20 degrees - Sub Stope Shapes enabled - Smoothing fast - UG mining unplanned recovery 5% - UG mining unplanned dilution 5% - Processing Recovery 95% - UG Stoping cost $70 per tonne ore - UG Opex Fixed Cost $5 per tonne ore - Mill Opex Cost (2Mtpa) $30.92 per tonne - ROM to mill transport $4.40 per tonne - Admin $4.50 per tonne - NT Factor $11.48 per tonne - Au Royalty 5% - Au Price AU$2700 troy ounce

Metallurgical factors or assumptions	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%
Environmental factors or assumptions	No assumptions have been made regarding environmental factors.
Bulk density	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 predetermined 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.

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

Classification	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 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.
Audits or reviews	Internal reviews have been conducted by Northern Star Resources resource geologists.

Discussion of relative accuracy/ confidence	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. 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 (4.2Mt @ 4.5 g/t) via open pit mining. The current model at a low grade cut off of 1 g/t Au produces 4.06 million tonnes @ 4.4 g/t Au for about 574k ounces. The current model slightly under calls the amount of gold produced. This is due to some of the minor lodes not being modelled. The result is within acceptable limits of what was mined. The Mineral Resource statement relates to global estimates of tonnes and grade.

Appendix 2 - JORC Table 1

Ripcord Gold Deposit

Section 1 - Sampling Techniques and Data

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

Criteria	Commentary
Sampling techniques	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. 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. 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 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	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	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. 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. 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. 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	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. 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 entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	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. 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. Sample preparation for early drill programs (Normandy and Tanami Gold) were completed to a high standard. For Northern Star 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 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. 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. 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 DD core sampling. 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	Normandy assays were sent to ALS–Chemex in Perth for Aqua Regia (PM203). Any samples that came back with an Aqua Regia result greater than 2ppm were automatically sent for A & B split Fire Assay (PM209), and those that assayed over 7ppm were sent for Screen Fire Assay. Tanami Gold sent samples from RPRC0001 to RPRC0037 to SGS lab in Perth for analysis by 50g Fire assay with Atomic absorption finish (FAA505). Samples from RPRC0038 to RPRC0111 were submitted to Genalysis lab in Alice Springs for analysis by 50g Fire Assay with Atomic Absorption finish (FA50/AA). For Northern Star drilling programs, gold concentration was determined 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. No geophysical tools were used to determine any element concentrations. Standards, Blanks and Duplicates were inserted in the Normandy drilling sample stream. However, the frequency of these insertions is highly variable and expected values for the standards are not available, limiting the value of this data. QC samples were inserted routinely for all Tanami Gold drillholes. Standard samples were inserted every 25 metres, blank samples every 20 samples and duplicate samples were collected every 12m. Tanami Gold QAQC includes: Certified reference standards were inserted at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Coarse crush duplicates show repeatable although variable results. A recognised laboratory has been used for analysis of samples. 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. • 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	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. No twinned holes were drilled for this data set. 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. 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	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. Collar coordinates were recorded in MGA94 Zone 52. 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	Drill-hole spacing across the area varies; the indicated mineral resource was defined within areas of RC drilling of 20m by 25m, and where the continuity and predictability of the lode positions was good. Drill spacing of 40 x 40m was targeted during the design and drilling phases for recent drilling campaigns. The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be 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	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends. No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	Geologists and geotechnicians managed the chain of custody of samples. 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	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.)

Mineral tenement and land tenure status	The Ripcord prospect is located in the Tanami Region in the Northern Territory in the Northern Territory on Mining Licence (Groundrush) ML22934, approximately 45km northeast of the Central Tanami Mill site. ML22934 covers an area of 3,950ha and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,211 sqkm 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 ten Exploration Licences, eight of which are granted, and two applications, nineteen Mineral Leases and one Mining Licence. 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. ML22934 is granted and in good standing.

Exploration done 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. Recent exploration in the area has been completed by the Joint Venture partners, Tanami Gold NL and Northern Star Limited.
Geology	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 dolerite and sediment, in either quartz vein or shear hosted, respectively.
Drill hole information	Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information.
Data aggregation methods	This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	The reported drillholes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends. Mineralisation structures are vertical to sub-vertical. Only downhole lengths have been reported. True widths are not known.
Diagrams	Appropriate plans and sections have been included.
Balanced Reporting	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. 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	Exploration results have previously been reported to the ASX by the Joint Venture parties.
Further work	Drilling is planned to infill and expand the current resource envelope. Appropriate Diagrams accompany this release.

Section 3 – Estimation and Reporting of Mineral Resources

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

Criteria	Commentary
Database integrity	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	The competent person Graeme Thompson, Principal resource Geologist, has made a number of site visits.

Geological interpretation	The confidence in the geological interpretation is moderate as there is no 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 Ripcord deposit is hosted within the Killi Killi Formation of the Tanami Group (1838+/-6Ma), a turbiditic siltstone and sandstone (arkose and greywacke) unit up to 4 km thick. 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 associated with the contact of the dolerite and the turbiditic sediments. The dolerite is striking at about 340º and dipping 70º west. Mineralisation is primarily hosted within the larger main dolerite body, and minor mineralisation extending in to turbiditic sediments. The main mineralised lodes consist of 1 - 6m wide zones of quartz veining that trend north to northwest (290° to 340°), dip at 80° to the southwest. The strike of the mineralised zone is about 1200 metres and the known down dip extent from drill data is about 150 metres. The width of the zone of primary mineralisation is of the order of 40 metres. There are 3 styles of mineralisation: • Supergene or flat lying lodes • Dolerite hosted • Turbiditic sediment hosted The supergene or flat lying mineralisation dip shallowly to the west and are separated into north and southerly plunging bodies. They consist of narrow zones of quartz veining (1-3m). The supergene or flat lying zones have strikes of up to 150 metres and dip extents of up 100 metres and true thickness of 1-3 metres. The dolerite and turbiditic sediment hosted mineralisation display similar strikes between 320° to 330° and dip about 60° west. There is a difference in the overall dimensions of the mineralised quartz lenses. The thickness of the mineralisation varies between 1 to 6 metres for both types however the dolerite hosted mineralisation is up to 150 metres in strike and 120 metres down dip while the sediment hosted mineralisation is up to 100 metres and 25 metres down dip. 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 strike of the mineralised zone is about 1200 metres and the known down dip extent from drill data is about 150 metres. The width of the zone of primary mineralisation is of the order of 40 metres. There are multiple gold bearing quartz lenses of various dimensions within the zone. The mineralisation starts at a depth of 20 metres below the surface.
Estimation and modelling techniques	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 multi-variate and bi-variate statistics) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Ripcord 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 (40, 80 and 160 metres). A first pass of radius 40m with a minimum number of samples of 2-8 samples and a second pass of radius 80m with a minimum number of 2-6 samples were used for Ripcord. A third pass of search radius 160m 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 after 3 passes were left without grade as a reflection of the paucity of samples in the lode. 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	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	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 underground material.
Mining factors or assumptions	It is assumed the Ripcord deposit will be mined by open pit methods when a new mining operation can be established. The Whittle Optimisation Assumptions used were: - OP Mining Recovery 98% - Op Mining Dilution 10% - Processing Recovery 95% - Mining Cost $4.40 per tonne ore - Incremental Ore Mining Cost $0 - Open Pit Grade Control $0.88 per tonne - Mill Opex cost (2.0Mtpa) $34.02 per tonne ore - ROM to mill transport distance 44 km - ROM to mill transport $4.84 per tonne - Admin (G & A) $4.95 per tonne ore - Au Royalty 5% - Au Price AU$2700
Metallurgical factors or assumptions	Tanami Gold NL submitted 9 composite RC samples for metallurgical testing in 2013. 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	No assumptions have been made regarding environmental factors.

Bulk density	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. <table><tr><td></td><td colspan="2">Rock Type</td></tr><tr><td>Oxidation State</td><td>Dolerite</td><td>Turbiditic sediment</td></tr><tr><td>Oxide</td><td>2.4</td><td>2.32</td></tr><tr><td>Transitional</td><td>2.7</td><td>2.58</td></tr><tr><td>Fresh</td><td>2.85</td><td>2.7</td></tr></table> At this stage of the project, it is assumed that these values will be close to the real values
Classification	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 20m by 25m, and where the continuity and predictability of the lode positions was 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.
Audits or reviews	No audits or reviews of this estimate have been conducted.
Discussion of relative accuracy/ confidence	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 3 - JORC Table 1
Jims Gold Deposits
Section 1 - Sampling Techniques and Data
(Criteria in this section apply to all succeeding sections.)

Criteria	Commentary
Sampling techniques	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. 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. 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	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	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 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. 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. 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	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. 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 entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	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. 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. 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. During mining operations drill samples were prepped either at onsite or at ALS in Alice Springs to industry standards. 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. 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. 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. 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	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. No geophysical tools were used to determine any element concentrations. 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	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. No twinned holes were drilled for this data set. 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. 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	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. Collar coordinates were recorded in MGA94 Zone 52. 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	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. The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be 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	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends No sampling bias is considered to have been introduced by the drilling orientation.
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	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	Commentary
Mineral tenement and land tenure status	Jims Gold Deposit is 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 2,211 sqkm 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 ten Exploration Licences, eight of which are granted, and two applications, nineteen Mineral Lease (Southern) and one Mining Licence. 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. MLS168 is granted and in good standing.
Exploration done by other parties	The Jims 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	The Jims deposit is a Palaeoproterozoic, basalt and sediment-hosted vein-mineralised 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 series of vein and breccia lodes developed near basalt sediment contacts.
Drill hole information	Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information.

Data aggregation methods	This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends Mineralisation are sub-vertical to vertical. Only downhole lengths have been reported. True widths are not known.
Diagrams	Appropriate plans and sections have previously been reported to the ASX by the Joint Venture parties.
Balanced Reporting	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. 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	Exploration results have previously been reported to the ASX by the Joint Venture parties.
Further work	Upon receipt of all results, a review of the drilling completed is required before further work is planned.

Section 3 - Estimation and Reporting of Mineral Resources

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

Criteria	**Commentary**
Database integrity	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	A number of site visits have been conducted by the Competent Person, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	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 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.

Dimensions	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 250 metres below the surface or 300 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 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 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.
Estimation and modelling techniques	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 multi-variate and bi-variate 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 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 (20-40, 40-80 and 80-160 metres). 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 4-6 samples were used for Jims. A third pass of search radius 80-160m 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. 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	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.67g/t gold cut-off grade for open pit material within a $AU2700 pit shell.

Mining factors or assumptions

It is assumed the Jims 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 although it can be used for a preliminary assessment of underground potential.

Whittle Assumptions:

- Open Pit Mining Recovery 98%
- Open Pit dilution 10%
- Processing Recovery 85%
- Mining Cost $4.40 per tonne rock
- Oxide and Backfill slope 45 degrees
- Trans and fresh Slope 39 degrees
- Backfill cost 2.75 per tonne backfill
- Incremental Ore Mining Cost $0
- Open Pit grade Control Cost $0.88 per tonne ore
- Mill Opex cost (2.0Mtpa) $34.01 per tonne
- ROM to mill transport distance (Current Mill location) 28 km
- ROM to mill transport $2.83 per tonne
- Admin (G&A) $4.95 per tonne
- Au Royalty 5%
- Au Price AU$2700 per troy ounce

Metallurgical factors or assumptions

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

Environmental factors or assumptions

No assumptions have been made regarding environmental factors.

Bulk density

No bulk density data from the Jims prospects could be located. Given that there were three open pits in the area it is assumed that bulk density data would have been taken. Density values were taken from the previous estimate.

	Material Type	
Oxidation State	**Basalt**	**Waste Dump**
Oxide	2.6	
Transitional	2.7	2.5
Fresh	2.8	

Classification	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 Measured Mineral Resource is located below Jims Main Open Pit and has already been grade controlled drilled in part. 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. 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.
Audits or reviews	No audits or reviews of this estimate have been conducted.
Discussion of relative accuracy/ confidence	The Jims Mineral Resource Estimate has 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. 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. 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_apr2022.mdl results in a resource that was mined of 1.66 million tonnes @ 2.33 g/t Au for 125,195 ounces. Once mining factors are applied this figure is within acceptable limits of the mined reserve. 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 1.0 g/t Au from ctp_jims_apr2022.mdl produces 2,993 tonnes @ 2.04 g/t Au for 157 ounces. This zone is just below the depletion zone and variable results can be expected. The Mineral Resource statement relates to global estimates of tonnes and grade.

Appendix 4 - JORC Table 1
Hurricane-Repulse Gold Deposit
Section 1 - Sampling Techniques and Data
(Criteria in this section apply to all succeeding sections.)

Criteria	Commentary
Sampling techniques	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. 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. 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	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	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 campaign was considered consistent. Diamond drill core recoveries are recorded as a percentage calculated from measured core versus drilled intervals length. 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. 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	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. 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 DDH logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed. The entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	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. Depending on the drilling campaign, 1m RC samples were collected 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 and then split using a riffle splitter down to approximately 2kg. Samples collected from RC drill holes drilled by Tanami Gold between 2010 to 2011 were selected on a one metre basis through a 75:25% riffle splitter. Primary analysis on some RC drilling was determined using 4m or 3m 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. 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, Tanami Gold Joint Venture sent samples to ALS in Alice Springs for sample preparation, alternatively, samples were prepared at the onsite laboratory. Tanami Gold sent RC samples to SGS in Perth from 2010 to 2011 for sample preparation. For Northern Star, sample preparation for DD drilling 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. For Northern Star, 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 4kg are crushed to <6mm and cone split to nominal mass before pulverisation. 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. 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. 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. 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	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. During mining operations (the mid-1990s to 2001) under the Tanami Gold Joint Venture, analysis (both on and offsite) was done by AAS with selective FA checks. 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. 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. QAQC programs from drilling data from the mid-1980s to 2001 were carried out, but that data has yet to be located. Significant mining was also carried out during that period. Drilling by CTP appears to confirm the results, but no definite conclusion can be made about the quality of the earlier data collection period. It is assumed to be representative. QAQC programs completed by Tanami Gold included insertion of certified reference material at regular intervals and results have, in the main, accurately reflected the original assays and expected values. Coarse crush duplicates show repeatable although variable results. This may be due to the heterogeneity of the mineralisation. A recognised laboratory has been used for analysis of samples. 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. • 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	All significant intersections were verified by a Northern Star Senior Geologist on-site during the drill-hole validation process and later signed off by a Competent person, as defined by JORC. No twinned holes were drilled for this data set. 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. 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	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. Collar coordinates were recorded in MGA94 Zone 52. 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	The Indicated Mineral Resource was defined within areas of RC and diamond drilling of 25m by 12 to 25m. 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. 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	Drillholes were drilled at an angle approximately perpendicular to the orientation of the mineralised trends. No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	Geologists and geotechnicians managed the chain of custody of samples. 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	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	Commentary
Mineral tenement and land tenure status	The Hurricane/Repulse deposit is located in the Tanami Region in the Northern Territory in the Northern Territory fully encompassed by mineral leases MLS153 and MLS125 to MLS129., approximately 600km northwest of Alice Springs, NT. Hurricane/Repulse forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,211 sqkm 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 ten Exploration Licences, eight of which are granted, and two applications, nineteen Mineral Leases, and one Mining Licence. 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. MLS153 and MLS125 to MLS129 are granted and in good standing.
Exploration done by other parties	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	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 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.
Drill hole information	Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information. Exploration results have previously been 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.
Data aggregation methods	This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. No metal equivalent values were used to report previous exploration results. The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralised trends.
Relationship between mineralisation widths and intercept lengths	The exact orientation of the Hurricane/Repulse mineralised system is generally well understood. The geometry of the mineralisation to drill hole intercepts generally at a high angle, often nearing perpendicular. There is enough historic exploration and production data at Hurricane/Repulse to infer geological continuity in mineralisation reported. Only downhole lengths have been reported. True widths are not known. Appropriate plans and sections have been included.
Diagrams	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.
Balanced Reporting	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. Exploration results have previously been reported to the ASX by the Joint Venture parties.
Other substantive exploration data	A review of the drilling completed is required before further work is planned.
Further work	To be determined

Section 3 - Estimation and Reporting of Mineral Resources

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

Criteria	Commentary
Database integrity	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	A number of site visits have been conducted by the CP, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.
Geological interpretation	The confidence in the geological interpretation is moderate to good as there are exposures and it is based upon RC and diamond drill holes and geological mapping. 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 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
Dimensions	The overall strike length of the known gold mineralisation on the Hurricane Repulse trend is of the order of 1750 metres and has a 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.
Estimation and modelling techniques	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 RPM Global in Leapfrog software 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) using Supervisor software. MJM has not made assumptions regarding recovery of by-products from the mining and processing of ore at the Hurricane Repulse 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 (25-60, 50-120 and 100-240 metres). A first pass of radius 25-60m with a minimum number of samples of 3-6 samples and a second pass of radius 50120m with a minimum number of 3-6 samples were used for Hurricane Repulse. A third pass of search radius 100-240m was used with a minimum of 3-6 samples to ensure all blocks within the mineralised lodes were estimated. The maximum number of samples ranged from 4-38 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 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	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.67g/t gold cut-off grade in oxide and transitional and 0.97 g/t gold in fresh for open pit material within a $AU2700 pit shell.
Mining factors or assumptions	It is assumed the Hurricane Repulse 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 although it can be used for a preliminary assessment of underground potential. The following mining factors and costs were used for the Whittle optimisation of the open pit resource: - OP Mining Recovery 98% - OP Mining Dilution 10% - Oxide Processing Recovery % - Trans Processing Recovery % - Oxide and backfill slope 45° - Trans and Fresh slope 39° - Backfill or Waste Dump Mining Cost $2.75/t - Mining Cost $4.40/t - Incremental Ore Mining Cost $4.95/t - Open Pit Grade Control Cost $0.88/t - Mill Opex Cost (2.0 Mtpa) $34.01/t - ROM to mill transport distance 2km - ROM to Mill cost $4.84/t - Admin (G&A) cost $4.95/t - Au Royalty 5% - Au Price AU$2700/tr oz - Deswick software was used for the underground resource stope optimisation. - Stope Optimiser Assumptions - UG Mining Unplanned Recovery 5% - UG Mining Unplanned Dilution 5% - Processing Recovery 55% - HW planned dilution skin 0.5 m - FW planned dilution skin 0.25 m - Minimum Mining width 3 metres not including dilution skins - Stope optimisation length 20 m along strike - Sub level interval 20 m - Optimise metal - UG Stoping cost $70 per tonne ore - UG Backfill Cost $10 per tonne ore - UG Opex Fixed Cost $5 per tonne ore - Mill Opex Cost (2Mtpa) $30.92 per tonne - ROM to mill transport $4.00 per tonne - Admin $4.50 per tonne - NT Factor $12.06 per tonne - Capex $0 - Au Royalty 5% - Au Price AU$2700 troy ounce

Metallurgical factors or assumptions	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.
Environmental factors or assumptions	No assumptions have been made regarding environmental factors.
Bulk density	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. Densities of 2.2 were applied to backfill and waste dumps. **Oxidation State** / **Material Type**: **Basalt**, **Sediments** Oxide: 2.29, 2.51 Transitional: 2.6, 2.65 Fresh: 2.84, 2.87
Classification	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 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 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.
Audits or reviews	No audits or reviews of this estimate have been conducted.
Discussion of relative accuracy/ confidence	The Hurricane Repulse 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. Production The Mineral Resource statement relates to global estimates of tonnes and grade. 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. The current resource of 0.69 million tonnes @ 2.53 g/t Au has the same tenure of grade as previous production.

Appendix 5 - JORC Table 1
Crusade Gold Deposit

Section 1 - Sampling Techniques and Data

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

Criteria	Commentary
Sampling techniques	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. 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. 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	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 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	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. 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. 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. 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	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. 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 entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	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. 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. 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. 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. 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. 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	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. No geophysical tools were used to determine any element concentrations. 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 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	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. Two twin holes were completed in the 2019 Northern Star drilling campaign, SJRC0005 twinned CDH007 and SJRC0006 twinned CDH008. 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. 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	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. Collar coordinates were recorded in MGA94 Zone 52. 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	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. The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be 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	Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends No sampling bias is considered to have been introduced by the drilling orientation.
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	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	Commentary
Mineral tenement and land tenure status	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 101.07 sqkm and forms part of the Central Tanami Project, a 50/50 Joint Venture between Tanami Gold NL and Northern Star Limited. The 2,211 sqkm 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 ten Exploration Licences, eight of which are granted, and two applications, nineteen Mineral Leases, and one Mining Licence. 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. EL28282 is granted and in good standing.

Exploration done 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	The Crusade deposit is a Palaeoproterozoic, 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	Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information.
Data aggregation methods	This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. This release pertains to the reporting of Mineral Resources. Exploration results have previously been 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. No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralized. Mineralisation structures are vertical to sub-vertical. Only downhole lengths have been reported. True widths are not known.
Diagrams	Appropriate plans and sections have previously been reported to the ASX by the Joint Venture parties.
Balanced Reporting	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. 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	Exploration results have previously been reported to the ASX by the Joint Venture parties.
Further work	A review of the drilling completed is required before further work is planned.

Section 3 - Estimation and Reporting of Mineral Resources

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

Criteria	Commentary
Database integrity	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	A number of site visits have been conducted by the CP, Mr Graeme Thompson, Principal Resource Geologist of MoJoe Mining Pty Ltd.

Geological interpretation	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. 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 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 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	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 multi-variate and bi-variate 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. 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 4^{th} 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. 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 (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

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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.</td></tr>
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<tr><td>Metallurgical factors or assumptions</td><td>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 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.</td></tr>
<tr><td>Environmental factors or assumptions</td><td>No assumptions have been made regarding environmental factors.</td></tr>
<tr><td>Bulk density</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 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>No audits or reviews of this estimate have been conducted.</td></tr>
</table>

Discussion of relative accuracy/ confidence	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.

Appendix 6 - JORC Table 1
Molech Gold Deposits

Section 1 - Sampling Techniques and Data

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

Criteria	Commentary
Sampling techniques	Sampling was completed using reverse circulation (RC) and diamond core (DDH) drilling. Sampling of RC chips was completed on RC drillholes, and half core sampling on diamond drillholes was completed. 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. DDH core is reconstructed into continuous runs, measured by tape and compared to down hole core blocks consistent with industry practice. All drill core is geologically and geotechnically logged, photographed, with sampling of DDH drillholes was completed using a diamond core saw. Half core was sampled on intervals mostly in 1.0m intervals. 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. Core from DDH drilling was collected with a standard tube and orientated where possible using the bottom dead centre technique. Hole deviation surveys were completed on all drill holes but details on the instrument used has not been located. 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.
Drilling techniques	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.
Drill sample recovery	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. Experienced RC and DDH drilling contractors were engaged to complete the drilling campaigns. Drilling contractors are supervised and routinely monitored by site geologists. 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	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 DDH 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. 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 DDH logging was quantitative where possible and qualitative elsewhere. All diamond drill core was photographed. The entire length of each RC and diamond core hole was logged.

Sub-sampling techniques and sample preparation	Diamond **dril**l core was cut in half using a diamond core saw. Half core was sampled mostly on 1.0m intervals. 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. 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. Sample preparation was completed at either external offsite laboratories ALS or Genalysis or at the onsite laboratory facility. The sample preparation methods employed have not been detailed in historic records, but it is assumed that the methods employed by the laboratories, in particular the known external facilities that the sample preparation process employed would have followed industry standards at that time. Selective Fire Assay checks were carried out on samples from both the onsite and external laboratory facilities. 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. No data has been located for the QAQC samples submitted during the RC drilling campaigns completed by Tanami Gold. Historical records indicate that QAQC processes were carried out, details of the frequency of the inclusion of QAQC samples is not known. 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	Analysis pre-2010 (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. 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). The analytical methods employed yielded total gold. No geophysical tools were used to determine any element concentrations. Selective Fire Assay checks were carried out on samples from both the onsite and external laboratory facilities. 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. No data has been located for the QAQC samples submitted during the RC drilling campaigns completed by Tanami Gold.
Verification of sampling and assaying	All significant intersections were verified by on-site geologists during the drill-hole validation process and later signed off by a Competent Person. No twinned holes were drilled. Historical details of the original importation of primary data is not known. Primary data is now either entered directly or imported into a SQL acQuire database using semiautomated 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. 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	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. 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. 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 and distribution	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.. The data spacing and distribution is sufficient to establish geological and/or grade continuity appropriate for the Mineral Resource and classifications to be applied. No sample compositing was applied. Sample compositing was only undertaken as part of the Mineral Resource estimation process. Drillholes were drilled at an angle that is approximately perpendicular to the orientation of the mineralised trends
Orientation of data in relation to geological structure	No sampling bias is considered to have been introduced by the drilling orientation.
Sample security	No historic record has been identified that outlines the measures taken to ensure sample security.
Audits or reviews	No historic record has been identified 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	**Commentary**
Mineral tenement and land tenure status	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 36km west of the Central Tanami Mill site. MLS180 covers an area of 803.6ha and EL26925 60 blocks (190.01 sqkm) and are registered to Northern Star (Tanami) Pty Ltd and Tanami (NT) Pty Ltd. They form part of the 2,211 sqkm 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. MLS180 and EL26925 are granted and in good standing.
Exploration done 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	**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.
Drill hole information	Drill hole information previously reported to the ASX by the Joint Venture parties is summarised in the summary of material information.
Data aggregation methods	This report pertains to the reporting of Mineral Resources. Exploration results have previously been reported to the ASX by the various previous explorers. No metal equivalent values were used to report previous exploration results.
Relationship between mineralisation widths and intercept lengths	The reported drill holes have been drilled approximately perpendicular to the orientation of the targeted mineralized. Mineralisation structures are vertical to sub-vertical. When exploration results were previously disclosed, only downhole lengths were reported. True widths are not known.
Diagrams	Appropriate diagrams accompany this report.
Balanced Reporting	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	Exploration results have previously been reported to the ASX by the previous various explorers.
Further work	To be determined.

Section 3 - Estimation and Reporting of Mineral Resources

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

Criteria	**Commentary**
Database integrity	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	The competent person, Mr Graeme Thompson, Principal Resource Geologist, has made a number of visits to the Tanami JV area.
Geological interpretation	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** 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** 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. 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** 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. 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** 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**

	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 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** 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	**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 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 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	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 topcuts to the data. These cut values were determined through statistical analysis (histograms, log probability plots, CVs, and summary multi-variate and bi-variate 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 due 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 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	Tonnages and grades were estimated on a dry in situ basis.
Cut-off parameters	The Mineral Resource estimate has been constrained by the wireframed mineralised envelopes, is undiluted by external waste and reported above a 0.65g/t gold cut-off grade for open pit material within a $AU2700 pit shell. The underground resource is reported within a $AU2700 optimised stope and is undiluted by waste.

Mining factors or assumptions	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. - OP Mining Recovery 98% - OP Mining Dilution 10% - Oxide Processing Recovery 90% - Trans Processing Recovery 90% - Oxide and backfill slope 45° - Trans and Fresh slope 39° - Backfill or Waste Dump Mining Cost $2.50/t - Mining Cost $4.00/t - Incremental Ore Mining Cost $4.50/t - Open Pit Grade Control Cost $0.80/t - Mill Opex Cost (2.0 Mtpa) $30.92/t - ROM to mill transport distance 37.4 km - ROM to Mill cost $3.74/t - Admin (G&A) cost $4.50/t - Au Royalty 5% - Au Price AU$2700/tr oz - Deswik software was used for the underground resource stope optimisation. - Stope Optimiser Assumptions - HW planned dilution skin 0.5 m - FW planned dilution skin 0.25 m - Minimum Mining width 1.8 metres not including dilution skins - Stope optimisation length 20 m along strike - Sub level interval 20 m - Optimise grade - Stope optimisation -20 degrees - Sub Stope Shapes enabled - Smoothing fast - UG mining unplanned recovery 5% - UG mining unplanned dilution 5% - Processing Recovery 90% - UG Stoping cost $70 per tonne ore - UG Opex Fixed Cost $5 per tonne ore - Mill Opex Cost (2Mtpa) $30.92 per tonne - ROM to mill transport $3.74 per tonne - Admin $4.50 per tonne - NT Factor $11.48 per tonne - Au Royalty 5% - Au Price AU$2700 troy ounce
Metallurgical factors or assumptions	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.
Environmental factors or assumptions	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.

Bulk density	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. At this stage of the project, it is assumed that these values will be close to the real values.
Classification	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
Audits or reviews	Reviews of this estimate have been conducted by Northern Star Resources and Tanami Gold NL geologists.
Discussion of relative accuracy/ confidence	There are a number of assumptions used in the modelling the Molech & Pendragon MRE that affects the confidence of the MRE. 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