Colosseum Bankable Feasibility Study Demonstrates a Technically Simple Restart with Strong Economics

Highlights

- **US$1.08B** undiscounted pre-tax free cashflow
  - Increases to US$1.357B using spot price
- **US$785M NPV$_{5\%}$** (pre-tax)
  - Increases to US$999M using spot price
- **49.5% IRR** (pre-tax) at base model gold price (US$4,200/oz),
  - Increases to 59.5% using spot price (US$4,700/oz)
- **US$249M** of start-up capital (including US$16M of capitalised mining) plus US$25M contingency.
- **75koz** average annual gold production of over first 6 years.
- **573koz** total gold production over 10.4 year mine life
  - **102koz** peak gold sales of in year 6.
- **~$55M** increase in undiscounted pre-tax free cashflow for every $100/oz increase in gold price
- **US$1,825/oz** All-in Sustaining Cost (AISC) based on current industry costs within ±15%.
- Low **3:1** strip ratio highlights Colosseum's strong mining efficiency, with reduced waste movement supporting lower operating costs.
- **55koz** of Inferred Mineral Resources within the pit shell that have not been included in the Ore Reserve.
- **Additional** underground potential in the northeast of the North Pit, that is open at depth and subject to ongoing drilling not included in Ore Reserve, including recently drilled holes.

Dateline Resources Limited (ASX: DTR, OTCQB: DTREF, FSE: YE1) (**Dateline** or **the Company**) is pleased to present the results of the Bankable Feasibility Study (**BFS**) for the 100%-owned Colosseum Gold and Rare Earth Element (**REE**) Project in San Bernardino County, California. The BFS demonstrates a robust gold development, generating significant margins.

Dateline's Managing Director, Stephen Baghdadi, commented:

"Since acquiring Colosseum in 2021, we have recognised the significant potential of the project. The near vertical nature of mineralisation associated with the breccia pipes demonstrates excellent continuity that continues with depth. Since the original Scoping Study was completed in October 2024, we have continued to see strength in the gold sector, with the project forecast to generate operating margins of greater than $2,500 per ounce.

"With the BFS complete and the Front-End Engineering Studies (FEED) well underway, our engagement with project financiers is advancing as we look to secure the funding required to commence production as soon as possible."

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Financial Metrics

Operating and capital cost estimates in the Study are based on current industry costs and are considered to be accurate within ±15%.

Spot Gold Price (US$4,700/oz)

- Undiscounted free cash flow of $1,357M pre-tax and $978M post-tax.
- $NPV_{5\%}$ of $999M pre-tax and $704M post-tax.
- IRR of 59.5% pre-tax and 46.2% post-tax.
- All-in Sustaining Cost (AISC) of $1,838/oz.

Base Gold Price (US$4,200/oz)

- Undiscounted free cash flow of $1,082M pre-tax and $779M post-tax.
- $NPV_{5\%}$ of $785M pre-tax and $551M post-tax.
- IRR of 49.5% pre-tax and 38.6% post-tax.
- All-in Sustaining Cost (AISC) of $1,825/oz.

Gold Price Sensitivity

- Every $100/oz increase in gold price, increases undiscounted pre-tax free cash flow by ~$55M.

Gold Production Profile

- Total gold production of 573koz.
- Peak annual gold production of 102koz in year 6.
- Average annual gold production of 75koz over first 6 years, before transitioning to stockpile processing.

Mining and Processing – Mining is front loaded in first six years

- 2Mtpa carbon-in-leach (CIL) processing plant
- Metallurgical recovery averages 91% over life of mine
- 100% of ore (20.6 tonnes of ore at 0.95g/t Au) mined in first six years
- 11.5Mt at 1.34g/t Au processed during first six years, producing 497koz
- Stockpiled material, already mined, will be processed over the next 4.4 years, producing a further 133koz

Table 1:Key Study Metrics

	Unit	May 2025 Scoping Study	2026 Bankable Feasibility Study	
			Base Case	Spot Price
Gold Price	$/oz	2,900	4,200	4,700
PRODUCTION TARGET				
Life of Mine	Years	8.3	10.4	10.4
Total Ore Mined	M Tonnes	16.6	20.6	20.6
Total Waste Mined	M Tonnes	56.8	62.2	62.2
Total Material Movement	M Tonnes	73.3	82.9	82.9
Strip Ratio	x:x	3.4:1	3.0:1	3.0:1
Total Tonnes Milled	M Tonnes	16.6	20.6	20.6
Average Plant Throughput	Mtpa	1.8	2.0	2.0
Average Head Grade	g/t Au	1.3	0.95	0.95
Average Recovery	%	92	91	91
Total Net Gold Produced	koz	635	573	573
Ave Annual Gold Production (first 6 years)	Koz pa	71	75.4	75.4
FINANCIALS				
Total Operating Costs	$M	751	942	942
Total Capital Costs	$M	195	313	313
Pre-Production Capex	$M	138	249	249
Total Pre-production capital expenditure and working capital requirements	$M	153	275	275
Net Revenue	$M	827	779	978
Total Sales Revenue (including royalties)	$M	1,773	2,337	2,612
Discount Rate	%	6.5	5	5
Pre-Tax Discounted Cashflow – NPV	$M	550	785	999
Pre-Tax Internal Rate of Return (IRR)	%	61	49.5	59.5%
UNIT COSTS				
Unit Operating Costs (C1)	$/oz	1,182	1,651	1,663
All-in Sustaining Costs	$/oz	1,490	1,825	1,838

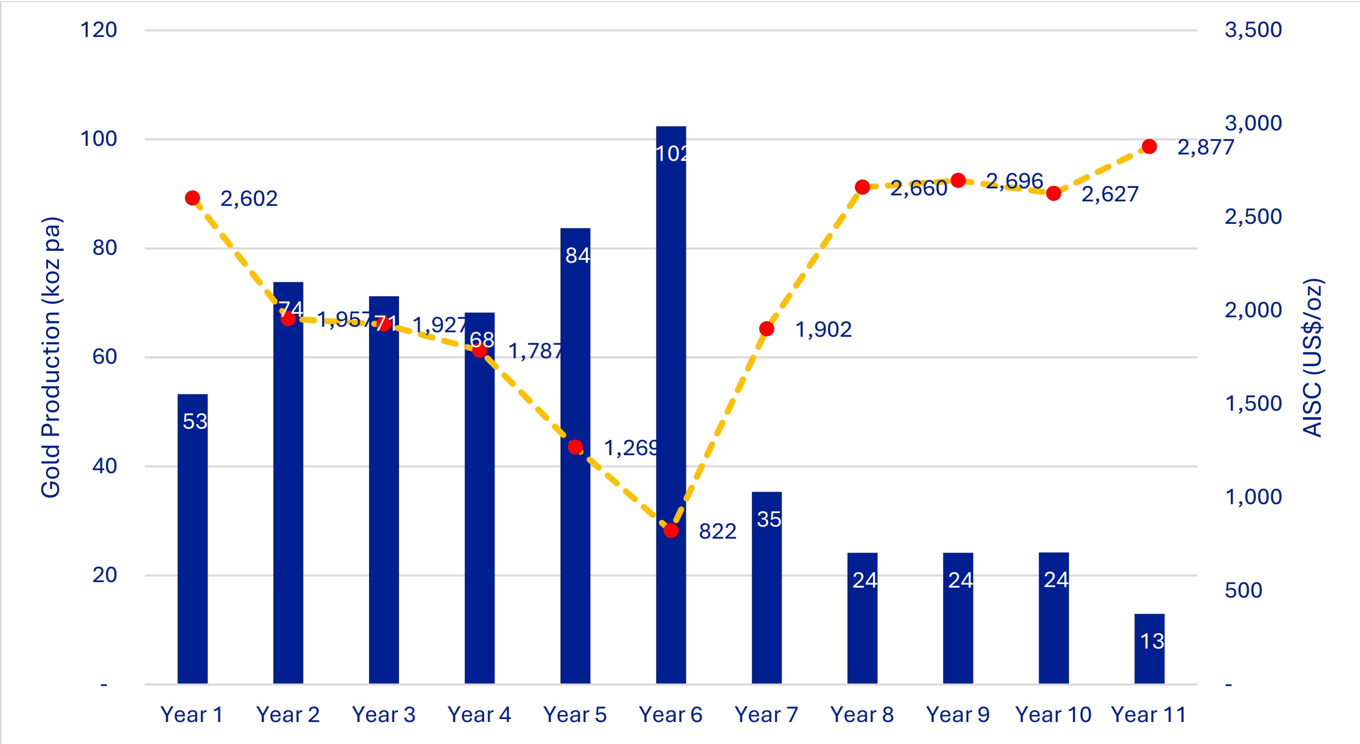

Figure 1: Annual Gold Production and AISC (@USD4,200/oz)

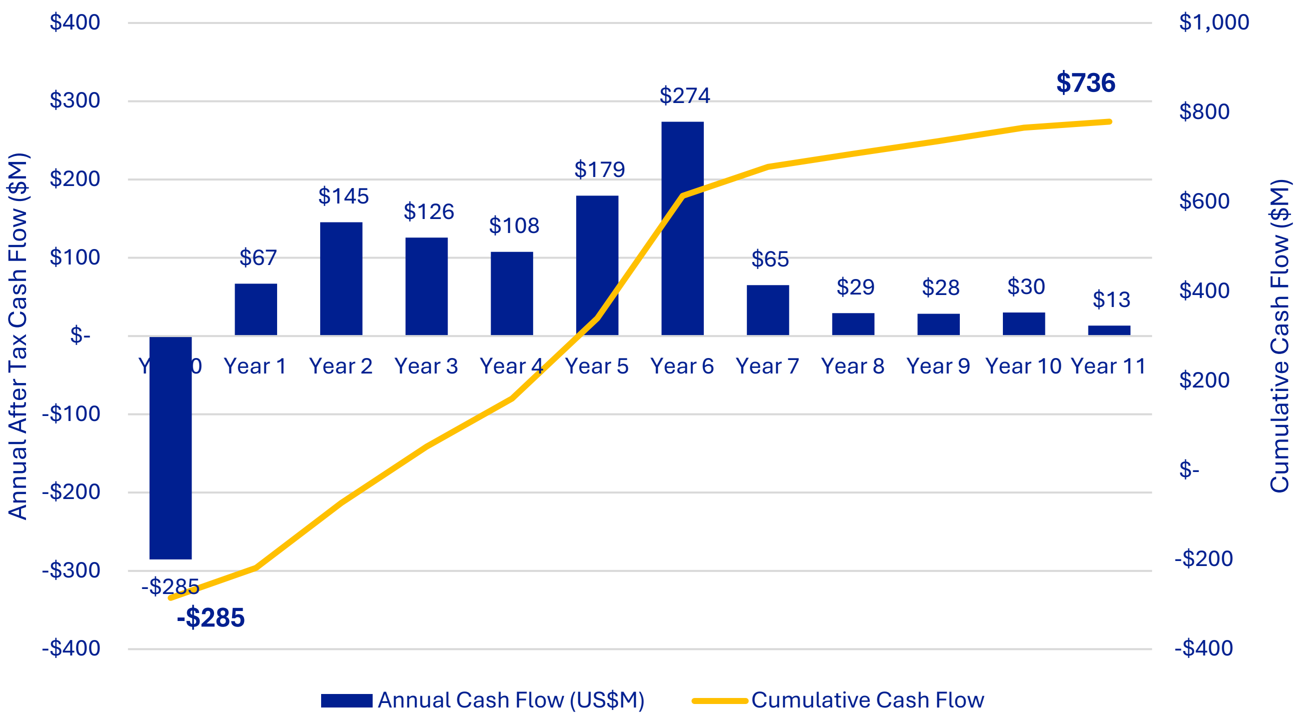

Figure 2: Annual and Cumulative Post-Tax Cash Flow (@$4,200/oz)

Production Confidence

- The Study focusses on the higher confidence Measured and Indicated Mineral Resource, which makes up 100% of the production target.
- The Proved Ore Reserve is 14.1Mt at 1.06g/t Au and the Probable Ore Reserve is 6.5Mt @ 0.72g/t Au. Total Ore Reserve ounces are 630koz at an average grade of 0.95g/t Au.

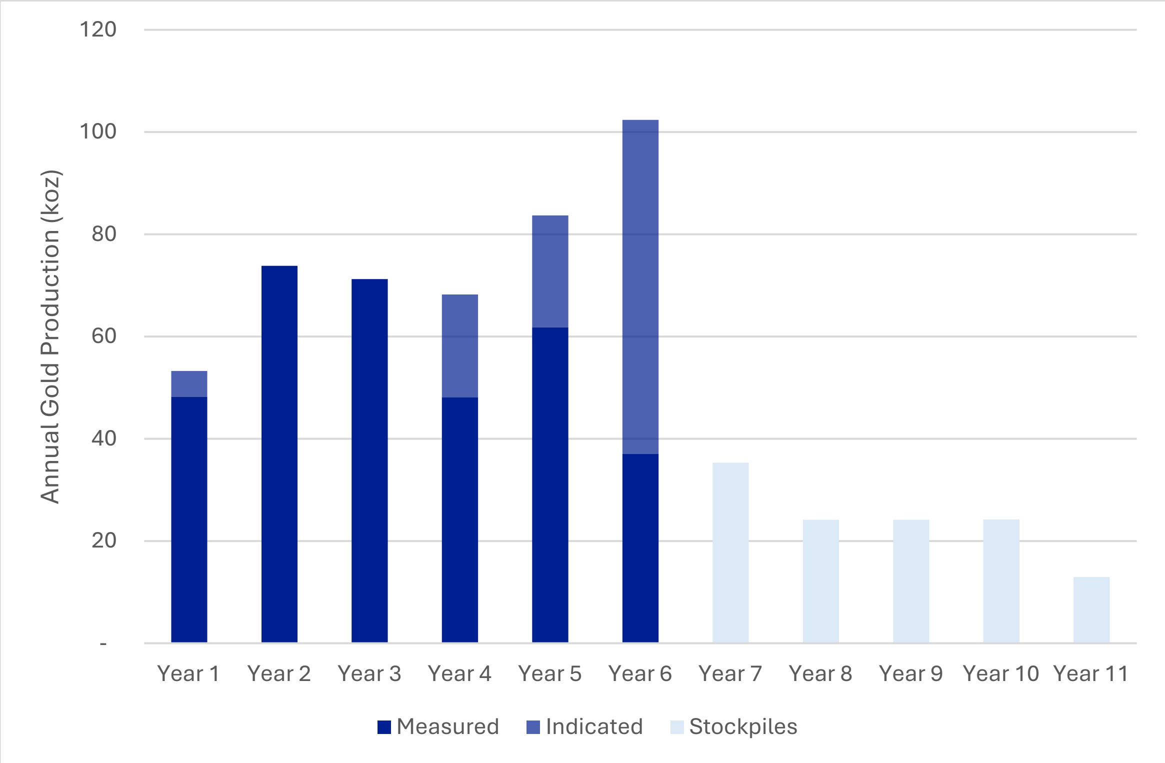

Figure 3: Processing schedule by Mineral Resource classification (57% Measured and 43% Indicated)

Production Strategy and Detailed Schedule

The production strategy involves prioritising the highest margin accessible material through the processing plant. Key points regarding the mill feed schedule include:

- Process plant commissioning and ramp-up occurs in year 1.
- Development of the Colosseum north and south open pits is staged to limit capital draw down while maintaining sufficient ore stocks to feed the mill.
- Metallurgical recovery averages 91%.
- Mining ceases at the end of year 6. A stockpile (11.3Mt) of low grade is processed through to the middle of year 11. Future extensional and regional exploration is expected to extend the period of mining, either via the open pits or an underground development.

Table 2: Colosseum Mine and Processing Production Schedule

	Units	Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Mining														
North Pit	Kt	**13,027.6**	188.9	3,992.1	4,152.6	3,470.6	1,222.4	-	-	-	-	-	-	-
	g/t	**0.84**	0.72	0.87	0.87	0.78	0.83	-	-	-	-	-	-	-
	Koz	**351.5**	4.4	111.6	115.5	87.4	32.5	-	-	-	-	-	-	-
South Pit	Kt	**7,603.3**	-	50.4	471.5	976.8	2.227.6	2.950.1	926.9	-	-	-	-	-
	g/t	**1.14**	-	0.47	0.59	0.69	0.85	1.28	2.19	-	-	-	-	-
	Koz	**278.7**	-	0.7	9.0	21.8	60.7	121.1	65.4	-	-	-	-	-
Total	Kt	**20.6**	188.9	4,042.5	4,624.1	4,447.4	3,450.0	2.950.1	926.9	-	-	-	-	-
	g/t	**0.95**	0.72	0.86	0.84	0.76	0.84	1.28	2.19	-	-	-	-	-
	Koz	**630.2**	4.4	112.4	124.5	109.2	93.2	121.1	65.4	-	-	-	-	-
Processing														
Tonnes	Kt	**20,630.9**	0.0	1,542	2,005	2,000	2,000	2,000	2,005	2,000	2,000	2,000	2,005	1,072
Grade	g/t	0.95	0.00	1.18	1.26	1.22	1.17	1.43	1.74	0.60	0.41	0.41	0.41	0.41
Milled Oz	Koz	**630.2**	0.0	58.5	81.1	78.3	75.0	92.0	112.5	38.8	26.5	26.5	26.6	14.2
Recovered Oz	Koz	**573.4**	0.0	53.3	73.8	71.2	68.2	83.7	102.4	35.3	24.2	24.2	24.2	12.9

Sensitivity Analysis

- Cashflow analysis shows that while sensitive to fluctuations in both cost and gold price, the Project continues to deliver positive cash flows under conservative assumptions. This supports the positive financial outcome modelled under the base case scenario.
- For each $100/oz change in gold price there is a ~$55M change in pre-tax free cash flow.

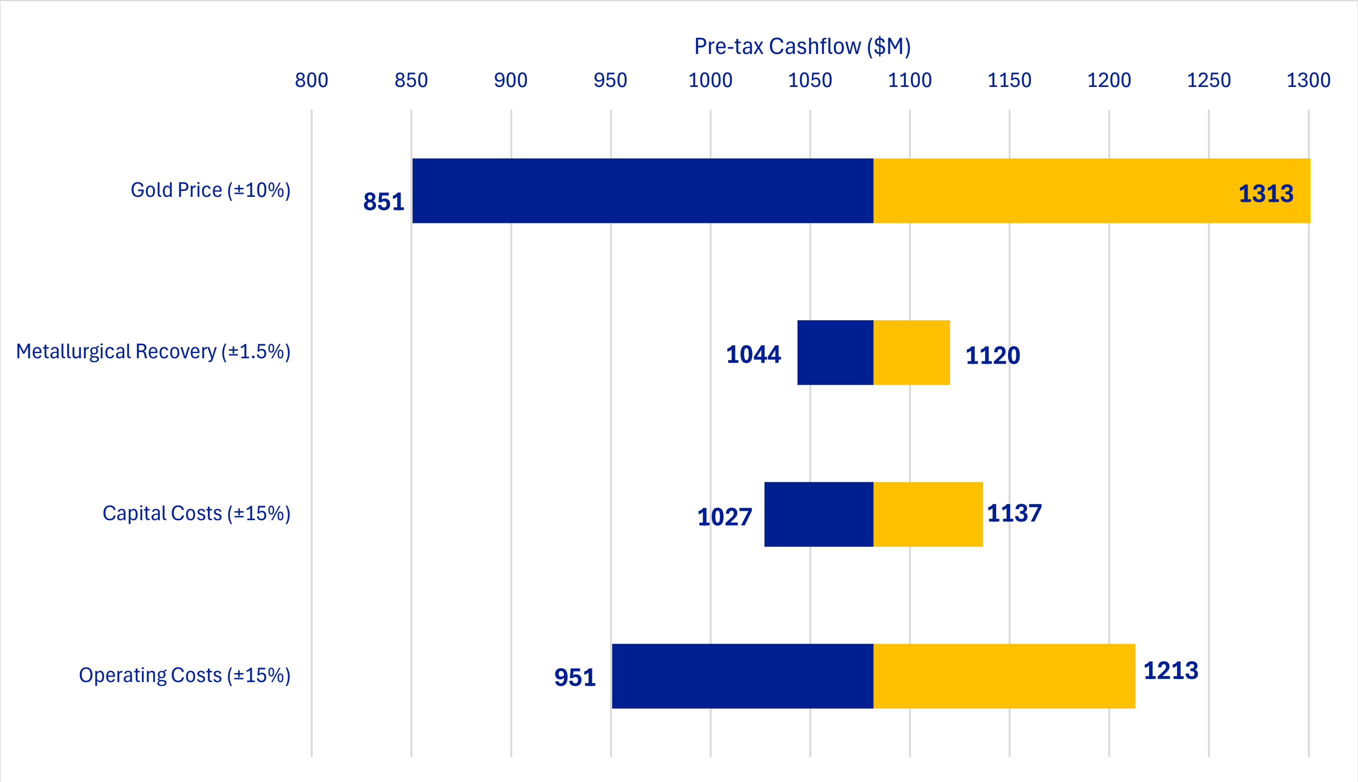

Figure 4: Undiscounted pre-tax free cash flow sensitivity (@$4,200/oz)

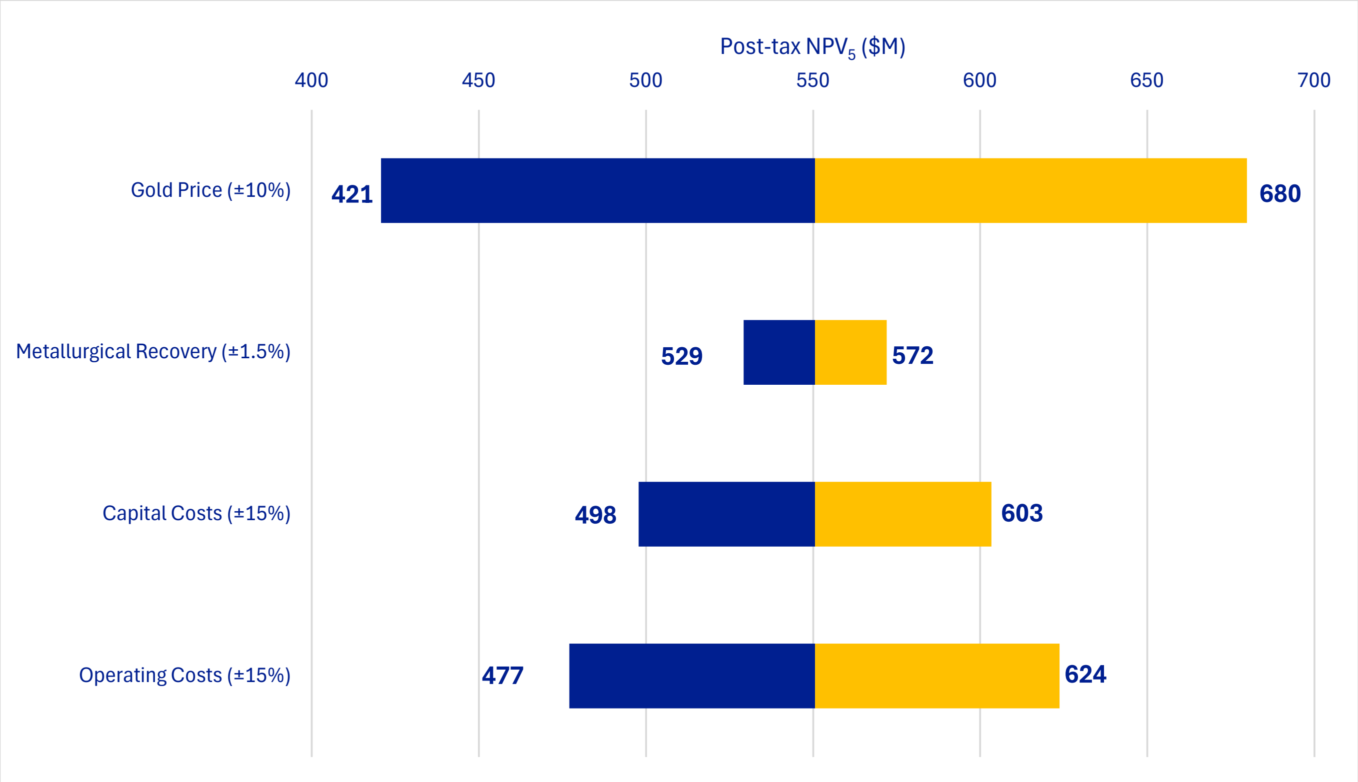

Figure 5: Post-tax free NPV_5 sensitivity (@$4,200/oz)

Funding

This Study estimates the funding required to commence production.

To achieve the range of outcomes indicated in the Study, funding of $249M, plus a contingency of $25M, is required.

The Company has cash and equivalents of A$88M as at the date of this report. The Company is seeking to fund the development of Colosseum via a mixture of project financing and cash at hand. Discussions are advanced with globally recognised financiers for the funding of the project.

Subsequent developments are assumed to be funded by positive cash flow generated from production.

Forward Work Program

Several pre-construction initiatives have commenced ahead of the Final Investment Decision (**FID**) for the Colosseum mine development.

An access road has been rehabilitated that connects the site with the Yates Well Road (sealed) and Interstate I-15.

A laydown and storage property has been optioned near the commencement of the access road on Yates Well Road. This property also includes two water bores, Colosseum #1 and #2, which will be used to supply water to the site.

The historical footings and foundations from the previous processing plant have been unearthed. Several of these areas have been assessed as being suitable for reuse in the Colosseum development.

An 'as new' SAG and Ball Mill has been agreed to be acquired for the Colosseum Project. The mills were built in 2020 but have been stored in a bonded warehouse since. They will be transported to site in the coming months.

Dateline intends to enter into a power supply agreement for the re-establishment of grid power to the Colosseum mine site. An overhead transmission line will be re-established along the approved access corridor.

GR Engineering Services (**GRES**) has been awarded the Front-End Engineering and Design contract for the Colosseum processing plant. This work commenced in parallel with the BFS.

Dateline expects to appoint an Engineering, Procurement, Construction plus Management (**EPC+M**) contract in the coming month/s. To assist with the contracting strategy as well as site construction management, Dateline has appointed experienced project management consultants Alvarez & Marsal (**A&M**). A&M have commenced their scope and will fill out the Dateline team, offering a streamlined model to quickly upskill Dateline's internal execution capability.

Mineral Resource Estimate

The Mineral Resource for the Project is 44.5Mt @ 0.76g/t Au for 1.08Moz (52% Measured and 27% Indicated). This represents less than 2% variance in total ounces from the scoping study.

Table 3: Mineral Resource (as at 25 April 2026)

	Measured			Indicated			Inferred			Total		
	Tonnes (Mt)	Grade (g/t)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t)	Ounces (Moz)	Tonnes (Mt)	Grade (g/t)	Ounces (Moz)
North Pit	11.96	0.85	0.33	9.09	0.59	0.17	7.69	0.53	0.13	28.75	0.68	0.63
South Pit	5.49	1.33	0.23	5.54	0.67	0.12	4.72	0.62	0.09	15.76	0.88	0.45
TOTAL	**17.50**	**1.00**	**0.56**	**14.60**	**0.62**	**0.29**	**12.40**	**0.57**	**0.23**	**44.50**	**0.76**	**1.08**

Notes:

1. The material assumptions and key parameters for this updated mineral resource are provided in Section 11 of the attached SK-1300 Feasibility Study Report for the Colosseum Gold Project.
2. The Mineral Resource is classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 (JORC code).
3. The North and South Pit open pit Mineral Resources are constrained within a $4,200/oz optimised pit shell and above a 0.25g/t Au cutoff grade.
4. Estimates are rounded to reflect the level of confidence in the Mineral Resources at the time of reporting.
5. JORC Table 1 is appended to this announcement.

Table 4: Previous Mineral Resource (at 0.5g/t Au cut-off, as at 6 June 2024)

	Measured			Indicated			Inferred			Total		
	Tonnes (Mt)	Grade (g/t)	Ounces (koz)	Tonnes (Mt)	Grade (g/t)	Ounces (koz)	Tonnes (Mt)	Grade (g/t)	Ounces (koz)	Tonnes (Mt)	Grade (g/t)	Ounces (koz)
North Pit	6.9	1.18	264	4.2	1.16	158	5.3	1.07	182	16.4	1.14	603
South Pit	2.7	2.23	191	3.0	1.28	124	5.0	1.13	183	10.7	1.45	498
TOTAL	**9.6**	**1.47**	**455**	**7.2**	**1.21**	**281**	**10.3**	**1.10**	**365**	**27.1**	**1.26**	**1,101**

Mineral Resource – Summary of Material Assumptions

A detailed summary of all material assumptions underpinning the Mineral Resource pursuant to ASX Listing Rule 5.8 is provided in the Bankable Feasibility Study (Section 11 of the S-K 1300 Report), further summarised below and also in the JORC Table 1 appended to this release.

ASX LR 5.8.1 - Geology and geological interpretation:

The Colosseum deposit is located at the southern end of the Sevier foreland thrust belt in the southern Basin and Range Province, SW USA. The project lies within in the Clark Mountain Mining District in the northeast portion of the Clark Mountain Range. The district includes the Mountain Pass rare earth mine seven miles south of the Colosseum Mine, numerous abandoned copper mines, and scattered fluorite, antimony, and tungsten prospects. Most gold and silver deposits in the district are within the northeast quadrant of the district north of Clark Mountain and are associated with emplacement of a felsic breccia complex into Precambrian basement rocks.

The deposit itself is associated with the emplacement of a breccia complex into Precambrian gneissic basement rocks. The complex is comprised of two felsite breccia pipes that form a northeast-southwest elongate zone, which contains mineralised zones of disseminated auriferous pyrite.

Gold at the Colosseum deposit is generally sub-microscopic and associated with sulphide mineralisation, chiefly pyrite. Gold within the deposit occurs as free gold, with minor alloyed silver. It is primarily in contact with pyrite, in fractures in the pyrite or along pyrite grain edges. It also occurs as isolated particles in quartz and other gangue minerals but spatially always close to pyrite but rarely as particles encased in euhedral pyrite.

The Colosseum deposit style is a hydrothermal breccia pipe with a combination of epithermal mineralisation at original higher levels and mesothermal mineralisation at the lower levels. Sedimentary breccia fragments with associated sulphides within the breccia may have originated from an earlier replacement deposit, not related to the breccia pipe itself.

ASX LR 5.8.1 - Sampling and sub-sampling techniques:

Sampling was predominantly on 5 feet (1.6m) intervals with a sizeable proportion at 2 feet (0.6m) intervals. Core sampling consisted of sawn half core whilst RC and rotary sampling comprising a split of the bulk sample using a free-standing riffle splitter. No compositing was undertaken on the RC samples. The sub-samples were then sent to a commercial laboratory for sample preparation and analysis.

Individual laboratory sample preparation procedures for the different historical drilling campaigns varied slightly but followed a standard analytical industry process of taking submitted samples through successive stages of reducing particle sizes and weights to obtain representative subsamples for assaying. Procedures comprised drying, crushing (jaw or rolls), splitting (riffle), pulverizing (spindle, plate, bowl), splitting (scoops) and fire assaying.

Quality Assurance/Quality Control (**QA/QC**) programs for the drilling have demonstrated that sample preparation and laboratory performance for the various drilling campaigns provided sample assays which are considered appropriate, with sufficient accuracy and precision, for the purpose of defining a Mineral Resource estimate.

There were no reports of significant numbers of wet samples for the RC drilling. Field duplicates were collected for the RC drilling at a ratio of 1 in 21 samples and indicate good precision and accuracy for the gold results.

Sampling of the 2022 to 2025 Dateline core drilling was via sawn half core completed at the commercial analytical laboratory (ALS Global in Reno Nevada). The RC sampling was as a sub-sample split via a rig-mounted rotary cyclone captured in a labelled calico bag. All samples weighed between 4-8kg.

The 2025 RC drilling encountered the water table in 40% of the holes, and within those holes the bottom third produced wet samples, approximately 12% of the total samples. The wet samples generally had lower recoveries, however twin hole drilling and analysis by Dateline indicated that there was only a minor under-reporting of the gold grade with the RC sampling.

QA/QC for the Dateline drilling comprised 189 core duplicates and 119 RC field duplicates (a second split off the rotary cyclone). The insertion rate for duplicates was 1 in 45-50. 10 laboratory duplicates (a second pulp split) were also collected. The results indicated reasonably good accuracy and precision for the gold results.

The sample preparation, sample size and analytical method are deemed appropriate.

ASX LR 5.8.1 - Drilling techniques:

A total of 658 holes for a total of 72,950 metres has been drilled in the Colosseum Mine area. The historical drilling was completed from 1972 to 1991 and includes 599 holes for a total of 55,609 metres. Most of the historical drilling was done using reverse-circulation ("RC") and conventional rotary methods. An inventory of known drilling in the area totals 5,166 metres in 262 Air Trac holes, 6,611 metres in 31 core holes, 40,288 metres in 273 RC holes and 3,543 metres in 33 rotary/percussion holes.

Between April 2022 and December 2025, Dateline drilled 27 diamond core holes (with one abandoned hole) along existing haul roads into the South and North Pits, for a total of 4,921 metres.

The majority of this drilling was aimed at confirming mineralisation grades at depth and to better define lateral margins to the deposit.

All the Colosseum drillhole data was used in developing the Mineral Resource model, with the exception of one historic drill hole, CP-2, which is an exploration hole testing an IP anomaly and is outside the area of the Mineral Resource.

ASX LR 5.8.1 - The criteria used for classification, including drill and data spacing and distribution. This includes separately identifying the drill spacing used to classify each category of mineral resources (inferred, indicated and measured) where estimates for more than one category of mineral resource are reported:

The classification of the recoverable Mineral Resources is based on the data point distribution which is a function of the drillhole spacing and the search parameters. Search Pass 1 equals Measured Resources, Search Passes 2 & 3 equal Indicated Resources and Search Passes 4 & 5 equals Inferred Resources.

Other aspects have been considered in the classification including the host geology and style of mineralisation, validation of the historic drilling, sampling methods and recoveries, the QA/QC programmes and results and comparison with previous resource estimates.

HSC believes the confidence in tonnage and grade estimates, the geological understanding, and the distribution of the data reflect the Measured, Indicated and Inferred categorisation. The estimates appropriately reflect the Competent Person's view of the deposit.

ASX LR 5.8.1 - Sample analysis method:

Historic sample analysis was by fire assay with a 30 to 60g charge using a lead collector and an AAS finish. Use of Certified Reference Materials (standards) indicated no issues with the accuracy of the reported laboratory results. There were no unusual or questionable gold assaying methods used. Copies of submittal sheets and assay certificates are available for most of the later drilling campaigns. Gold assay values were reported as ounces per short ton and were converted to grammes per tonne for the resource estimation.

The Dateline core and RC samples were assayed for gold at ALS Global in Reno, Nevada, using a fire assay method with a 30g charge and a gravimetric finish. The QA/QC procedure included standards (422 samples), blanks (386 samples), field duplicates (257 samples) and second laboratory checks (176 samples) which indicated no issues with the assay results.

The fire assay analytical methods used for Colosseum are considered as total digest techniques and appropriate for the commodity type and style of mineralisation.

ASX LR 5.8.1 - Density

No historical density data was supplied.

122 density measurements were taken by Dateline from their recent drilling. Samples consisted of single pieces of core 10-15 cm long and density was measured using an immersion in water technique i.e. the Archimedes Principle of weight in air / (weight in air minus weight in water). The average density value was 2.65 t/m^3 with a range of 2.1 to 4 t/m^3. Density values tended to show an increase with hole depth.

A default density of 2.65 t/m^3 was used for the Mineral Resources and is considered by Hellman and Scofield (**HSC**) as reasonable based on the granitic host unit and its experience with other similar deposits.

ASX LR 5.8.1 - Estimation methodology:

Recoverable Multiple Indicator Kriging (**MIK**) was used to complete the gold grade estimation using the GS3M modelling software. The geological interpretation, such as it is, block model creation and validation were completed using the Surpac mining software. HSC considers MIK on unconstrained data to be an appropriate estimation technique for the type of mineralisation and extent of data available.

The drillhole database was composited, with no constraints, to 1m intervals covering the whole of the prospect. The 1m composite interval may lead to a smoothing out of the variance but is unlikely to have a significant impact on the global estimates. A minor amount of peripheral, isolated data was removed from the composite file. A total of 62,920 composites were generated from the drillhole database, using the Surpac 'best fit' option and modelled for gold only. Two drilling domains were designed, one for the South Pit (domain 1 with 36,589 data and a coefficient of variation of 8.8) and another for the North Pit (domain 2 with 26,331 data and a coefficient of variation of 3), reflecting a difference in intensity of drilling and assay grades.

Metal variogram maps of gold for domains 1 and 2 indicated weak results which points to a relative lack of structure to the gold data. Overall grade continuity was very modest with a weak E-W trend for domain 1 coupled with a steeply west plunging feature in the XZ plane and a vertical plunge in the YZ plane. For domain 2, a WNW trend was interpreted with a subvertical plunge in both the XZ and ZY planes.

Grade interpolation was unconstrained, except by the search parameters and the variography, in acknowledgement of the gradational nature to the margins of the gold mineralisation and the abundance of buffering low grade peripheral values.

No base of oxidation was used. No cover surface was created as the mineralisation is outcropping and is exposed in many places along its ridge line and flanks and where previous open pit mining had occurred.

A fundamental concept behind MIK method is that it generally precludes the need for top cutting. However, a review of the conditional statistics for the top indicator class for both domains indicated a significant difference between the mean and the median. As a result, an averaged value for the mean and median was used as the gold grade for the top indicator class for both domains.

Block dimensions are 10 m by 10 m by 5 m (E, N, RL respectively) with no sub-blocking. The selective mining unit (**SMU**) is 5 m by 5 m by 2.5 m. The north and east dimensions were chosen as they are close to the nominal drillhole distances in the detailed drilled area of the South Pit. The vertical dimension was chosen as a compromise between the two deposits, a reflection of the sample spacing, possible mining bench heights and to allow for flexibility in potential mining scenarios after discussions with independent mining consultants, Australian Mine Design and Development (**AMDAD**).

Both domains were modelled as a combined dataset with soft boundaries but with separate sets of conditional statistics. A total of 5 search passes were employed with progressively larger radii and/or decreasing data point criteria. The initial search parameters for domain 1 were 20m by 20m by 35m with a minimum of 16 data and 4 octants increasing to a final Pass 5 search of 60m by 60m by 120m with a minimum of 8 data and 2 octants. For domain 2, the initial search was 25m by 25m by 25m with the same data requirements as for domain 1 expanding to a Pass 5 search of 70m by 70m by 70m with a minimum of 8 data and 2 octants. The slightly different search dimensions for the two domains are a function of the geometry and drill spacing of the mineralisation in each pit.

The maximum extrapolation for the Mineral Resources is the Pass 5 search axial lengths.

No other elements were modelled therefore there are no assumptions about correlation between variables. No by-products are anticipated from production (it should be noted that silver was not routinely analysed for in the historic drilling). No assessment has been made for any deleterious elements.

Drillhole spacing ranges from 10 to 20 m in the core of the two domains but at a variety of directions giving rise to localised relatively close spaced samples. Downhole sampling was generally at 5’ (and 2’) intervals.

The resource estimates are controlled by the data point distribution, the variography, block size and the search ellipses. Conventional use of wireframes to control the mineralisation was not considered necessary in this case.

The new block model was reviewed visually by HSC, and it was concluded that the block model fairly represents the grades observed in the drill holes. HSC also validated the block model using a variety of summary statistics and statistical plots. Block model validation confirmed the modelling strategy as acceptable with no significant issues.

Comparison with the 2024 resource estimates shows a significant increase in tonnes but with a corresponding drop in gold grade leading to a <1% reduction in gold ounces. The increase in size and reduction of gold grade is due to using a lower gold cut-off grade 0.2g/t cf 0.5g/t. None of this is unexpected with the 2025 infill drilling tending to trim away peripheral areas of lower grade material.

Tonnages are estimated on a dry weight basis, and moisture content has not been determined.

The historic mining operation exploited both the South and North Pits but there are no meaningful production figures available to allow for any reconciliation with the new Mineral Resources.

ASX LR 5.8.1 - Cut-off grade(s), including the basis for the selected cut-off grade(s):

The recoverable MIK resources are reported at a gold cut-off grade of 0.2g/t based on the outcome of a recently completed pit optimisation study by AMDAD. The cut-off grade at which the resource is quoted reflects the intended bulk-mining approach. Consideration of “reasonable prospects of eventual economic extraction” has utilised an optimised pit shell with a revenue factor of 1 run at a US$5000/oz gold price with estimates of mining costs and pit wall slopes.

ASX LR 5.8.1 - Mining and metallurgical methods and parameters, and other material modifying factors considered to date:

The Mineral Resources were estimated on the assumption that the material is to be mined by open pit using a bulk mining method. The proposed mining method is a conventional drill & blast, truck & excavator with extracted material sent to an on-site ROM pad with a processing plant adjacent to the planned pit(s). Minimum mining dimensions (SMU) are envisioned to be around 5m by 5m by 2.5m (X, Y, Z). Any internal dilution has been factored in with the modelling and as such is appropriate to the block size. Internal dilution has been incorporated as part of the MIK modelling, but there is no allowance for external dilution and mining losses.

The operation of the grinding mill (cyanide leach with carbon in pulp recovery) in the January 1988 through June 1993 period conclusively demonstrated the feasibility of gold recovery from the Colosseum ore. Process recoveries for gold during the operations were reported to be around 92%. For the current project a standard CIL plant is envisaged for the ore processing, similar to the process used for the initial mining campaign.

A 2022 NI43-101 report for the property stated: “There are no known environmental liabilities that are adversely impacting air, water or soil resources on the Colosseum Mine project.” The current tenement status over the project area permits the resumption of opencut mining and ore processing. Preliminary mining studies by AMDAD indicate any future mining operations eg tailings and waste dumps, can be contained within the unpatented mine leases. There are no reports of acid mine drainage from the stockpiles or the waste dumps. All waste and process residues will be disposed of in a responsible manner and in accordance with the mining license conditions.

The area comprises modestly rugged terrain with alluvial fans, basalt flows, hills, and low mountains and is generally sparsely vegetated. The climate is typical of a high desert environment with high temperatures in excess of 100°F during the summer and low temperatures slightly below freezing in the winter. Annual precipitation is approximately 8 inches.

Ore Reserve Estimate

The Ore Reserve Estimate for the Project is 20.6Mt @ 1.0g/t Au for 630 koz. The Ore Reserve was completed for the BFS and is the maiden Ore Reserve Estimate for the Project.

The Ore Reserve is a subset of the Measured and Indicated Mineral Resource that is assessed as economically minable following the application of appropriate modifying factors. The Ore Reserve was compiled in March 2026.

Table 5: March 2026 Ore Reserve Estimate

Location	Cut-off	Proved			Probable			Total		
	Grade (g/t)	Tonnes ('000t)	Grade (g/t)	Ounces ('000 oz)	Tonnes ('000t)	Grade (g/t)	Ounces ('000 oz)	Tonnes ('000t)	Grade (g/t)	Ounces ('000 oz)
North Pit	0.25	9,800	0.88	280	3,300	0.70	70	13,000	0.83	350
South Pit	0.25	4,400	1.45	200	3,200	0.74	80	7,600	1.15	280
Total	**0.25**	**14,100**	**1.06**	**480**	**6,500**	**0.72**	**150**	**20,600**	**0.95**	**630**

Notes:

1. The Ore Reserve is classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 Edition (JORC code 2012).
2. The open pit Ore Reserve cut-off grades were estimated using a $4,200/oz gold price.
3. Estimates are rounded to reflect the level of confidence in the Ore Reserve at the time of reporting.
4. Gold ounces shown are contained gold prior to application of process recovery.
5. JORC Table 1 is appended to this announcement.

Ore Reserve – Summary of Material Assumptions

A detailed summary of all material assumptions underpinning the Ore Reserve pursuant to ASX Listing Rule 5.9 is provided in the Bankable Feasibility Study (Section 11 of the S-K 1300 Report), further summarised below and also in the JORC Table 1 appended to this release.

> *ASX LR 5.9.1 - The material assumptions and the outcomes from the preliminary feasibility study or the feasibility study (as the case may be). If the economic assumptions are commercially sensitive to the mining entity, an explanation of the methodology used to determine the assumptions rather than the actual figure can be reported:*

Ore Reserves are based on a Bankable Feasibility Study (**BFS**) completed in May 2026.

The project consists of an open pit mine delivering 2 million tonnes of ore per year to a carbon in leach processing plant to produce gold doŕe on site. Mining will be conducted over six years followed by a further four and a half years processing low-grade ore stockpiles.

The mine plan is based on a pit optimisation run with the following key inputs:

- Gold price of US$4,200/oz with current realisation costs,
- Process gold recovery of 91%,
- Measured and Indicated mineral resources only,
- Mining loss and dilution modelled to reflect orebody geometry and mining method,
- 2 Mtpa process feed rate,
- Operating cost estimates drawn from supplier pricing and detailed first principles cost estimates, and
- Physical parameters such as slope stability based on analyses conducted for the BFS.

The inputs result in an economic cut-off grade of 0.24 g/t Au. After consideration of site operational issues and maximisation of project present value the cut-off for the ore reserves was set at 0.25 g/t Au.

Practical pit designs based on the pit optimisation are used to report ore and waste tonnes and grades for the ore reserves and to drive the production schedule.

Capital and operating cost estimates are drawn from supplier pricing and detailed first principles cost estimates to a ±15% level of accuracy. The cost estimates are based on the mine plan, production schedule and process test work.

The BFS financial model uses a base gold price of US$4,200/oz.

The Study shows the Project delivers a robust financial outcome delivering free cashflow of $1,082M (pre-tax) and $779M (post-tax), net present value (**NPV$_{5\%}$**) of $785M (pre-tax) and $551M (post-tax) and an internal rate of return (**IRR**) of 49.5% (pre-tax) and 38.6% (post-tax) over the initial 10.4-year production plan using a $4,200/oz gold price.

Sensitivity analysis shows the effect of fluctuations in both cost and gold price. A ±15% change in operating costs delivers a ~$72M change in pre-tax free cash flow. For each $100/oz change in gold price there is a ~$55M change in pre-tax free cash flow.

ASX LR 5.9.1 - The criteria used for classification, including the classification of the Mineral Resources on which the Ore Reserves are based and the confidence in the modifying factors applied:

The Proved Ore Reserve is derived from the portion of the Measured Mineral Resource and the Probable Ore Reserve is derived from the portion of the Indicated Mineral Resource within the mine design that may be economically extracted and which include modelled allowances for dilution and ore loss.

Inferred Resources within the pit designs are treated as waste rock. These resources are in position which could not be readily targeted by exploration drilling but will be tested by grade control drilling during operations.

The project is a brownfields operation last operated from 1987 to 1993. Dateline has conducted extensive drilling since 2022 to confirm and extend the Mineral Resource. The BFS includes geotechnical, metallurgical and other test work to confirm and improve upon the previous operation. Capital and operating costs have been estimated to AACE Class 3 confidence. All material legal agreements and approvals are either in place or the Company is confident, based on information available, that they will be in place in a suitable timeframe to execute the Project.

For these reasons, the Competent Person for the Ore Reserves considers the modifying factors to be defined a level of confidence commensurate with Proved and Probable Ore Reserves.

ASX LR 5.9.1 - The mining method selected and other mining assumptions, including mining recovery factors and mining dilution factors:

The project will be mined as a conventional open pit using hydraulic excavators and off highway dump trucks.

A mining loss and dilution procedure is run on the MIK resource model. It applies a dilution skin to each MIK grade increment. This “onion skin” approach allows the dilution to be applied to whichever MIK grade increment the cut-off grade falls in. It includes both the diluted grade and diluted proportion of the MIK panel at the cut-off grade. An additional 1% dilution at zero grade and 2% ore loss is applied to the diluted blocks to allow for operational inefficiencies. This approach matches mining loss and dilution to the orebody geometry and gold distribution.

Mining will be conducted by a US based mining contractor with geological control and mine planning completed by Dateline.

No material changes are expected to mining conditions and ore characteristics from the previous operation. The BFS included geotechnical and hydrogeological work to confirm slope stability and ground water assumptions. There is no evidence of potential for acid mine drainage or any other contaminants from the mine waste rock.

The mining fleet is planned around up to three 120 tonne hydraulic excavators loading 55 tonne payload articulated dump trucks. Checks on sinking rate and working area for each excavator confirm this fleet is operationally capable of meeting the scheduled quantities. Final fleet configurations will be decided during the mining contract tender.

Mining includes some areas of mine waste rock left from the previous operation around the new pit crests. All other material mined is rock and will be drilled and blasted.

Mine operating costs include grade control drilling and sampling over 125% of the area of expected ore zones on each pit bench.

ASX LR 5.9.1 - The processing method selected and other processing assumptions, including the recovery factors applied and the allowances made for deleterious elements:

Gold ore processing to doré will be by the carbon in leach (**CIL**) method. Process recoveries are based on extensive grind size and leach testing conducted for the BFS, which confirmed gold recoveries from the previous operation. Current test work includes grade variability, which confirms a fixed gold recovery of 91% down to head grades of 0.5 g/t Au.

Process tailings will be de-toxified, dewatered to between 15% and 25% moisture and trucked to a dry-stack tailings cell enclosed within the waste rock dump.

ASX LR 5.9.1 - The basis of the cut-off grade(s) or quality parameters applied:

Table 6: Open Pit Cut-Off Grade Estimation

Variable	Unit	Colosseum
Gold Price	US$/oz	4,200
Government Royalty	%	nil
Doré transport, Insurance and Refining	US$/oz	50.00
Vendor Royalty	%	2.5% NSR
Metallurgical Recovery	%	91%
Mining Incremental Ore Cost	US$/t ore	0.28
Processing Costs	US$/t ore	23.86
G&A	$/t ore	3.80
Calculated Cut-off Grade	g/t Au	0.24
Applied Cut-off Grade	g/t Au	0.25

ASX LR 5.9.1 - Material modifying factors, including the status of environmental approvals, mining tenements and approvals, other governmental factors:

The tenements are in good standing. All material legal agreements and approvals are either in place or the Company is confident, based on information available, that they will be in place in a suitable timeframe to execute the Project.

Infrastructure requirements for selected mining methods and for transportation to market:

The Colosseum Mine is located 10km from Interstate I-15 and has good road access.

The project is located 88km from Las Vegas, Nevada with major airport, hospital, accommodation and residential facilities.

Personnel will live in Las Vegas and operate on a drive in- drive out basis daily. There will be no accommodation on site.

A 34.5 kV powerline is located in close proximity to the east of the mine and will be connected to the project via a 10km overhead transmission line that follows the established access road.

Two water bores that previously serviced the operation have been optioned and will be reactivated.

Dateline intends to construct the process plant and associated infrastructure on the same location as that used in the previous mining operation.

This ASX announcement has been authorised for release by the Board of Dateline Resources Limited.

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About Dateline Resources Limited

Dateline Resources Limited (ASX: DTR, OTCQB: DTREF, FSE: YE1) is an Australian company focused on mining and exploration in North America. The Company owns 100% of the Colosseum Gold-REE Project in California.

The Colosseum Gold Mine is located in the Walker Lane Trend in East San Bernardino County, California and is located 10km north of Mountain Pass rare earth mine. Drill testing the REE potential at Colosseum has commenced.

On 11 May 2026, Dateline announced that the BFS economics for the Colosseum Gold Project generated a pre-tax NPV_5 of US$785 million and a pre-tax IRR of 49.5% using a gold price of US$4,200/oz.

Dateline has also acquired the high-grade Argos Strontium Project, also located in San Bernadino County, California. Argos is reportedly the largest strontium deposit in the U.S. with previous celestite production grading 95%+ $SrSO_4$.

In March 2026, Dateline consolidated the Music Valley Heavy Rare Earth Project in Riverside and San Bernardino Counties, California. The region has known HREE mineralisation from USGS rock chip sampling, however it has not been subjected to modern exploration techniques.

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Forward-Looking Statements

This announcement may contain “forward-looking statements” concerning Dateline Resources that are subject to risks and uncertainties. Generally, the words “will”, “may”, “should”, “continue”, “believes”, “expects”, “intends”, “anticipates” or similar expressions identify forward-looking statements. These forward-looking statements involve risks and uncertainties that could cause actual results to differ materially from those expressed in the forward-looking statements. Many of these risks and uncertainties relate to factors that are beyond Dateline Resources’ ability to control or estimate precisely, such as future market conditions, changes in regulatory environment and the behaviour of other market participants. Dateline Resources cannot give any assurance that such forward-looking statements will prove to have been correct. The reader is cautioned not to place undue reliance on these forward-looking statements. Dateline Resources assumes no obligation and does not undertake any obligation to update or revise publicly any of the forward-looking statements set out herein, whether as a result of new information, future events or otherwise, except to the extent legally required.

Competent Person Statements

Sample preparation and any exploration information in this announcement is based upon work by Mr Graham Craig who is a Member of the Association of Professional Engineers and Geoscientists of Manitoba (APEGM). Mr Craig has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to quality as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting Exploration Results, Mineral Resources and Ore Reserves" (JORC Code). Mr Craig is a full time employee of Colosseum Rare Metals Inc. which is a wholly owned subsidiary of Dateline Resources Limited and consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

The data in this report that relates to Mineral Resource estimates for the Colosseum gold deposit is based on information evaluated by Mr Simon Tear who is a Member of the Australasian Institute of Mining and Metallurgy (AusIMM) and who has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the “JORC Code”). Mr Tear is a Director of H&S Consultants Pty Ltd and he consents to the inclusion in the report of the Mineral Resource Estimate in the form and context in which it appears.

The data in this report that relates to Ore Reserves estimates for the Colosseum gold deposit is based on work conducted or supervised by Mr John Wyche who is a Fellow and Chartered Professional of The Australasian Institute of Mining and Metallurgy (AusIMM) and who has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the "JORC Code"). Mr Wyche is a Director of Australian Mine Design and development Pty Ltd and he consents to the inclusion in the report of the Ore Reserves Estimate in the form and context in which it appears.

The production target which forms the basis of the Feasibility Study is based solely on Proved and Probable Ore Reserves. It is presented in terms of the life of mine production schedule and forecast financial outcomes. The scheduled ore tonnes in the production target comprise 68% Proved Ore Reserves and 32% Probable Ore Reserves. The Competent Person for the Ore Reserves, John Wyche, is satisfied that the Feasibility Study is based on, and fairly represents, information and supporting documentation prepared by him. Mr Wyche is a director of Australian Mine Design and Development Pty Ltd which is a mining engineering consultancy that is independent of Dateline Resources Limited.

SK-1300 TECHNICAL REPORT SUMMARY

Colosseum Gold Project | 2Mtpa Feasibility Study

San Bernardino County, California, U.S.A

Effective Date:	**May 11, 2026**	**Project No. 300203**
Issue Date:	**May 11, 2026**	

SK-1300 Technical Report Summary

Colosseum Gold Project | 2Mtpa Feasibility Study

San Bernardino County, California, U.S.A

Project Number: 300203

Effective Date: May 11, 2026

Report Date: May 11, 2026

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FORWARD-LOOKING STATEMENTS

This Technical Report Summary contains forward-looking statements within the meaning of the U.S. Securities Act of 1933, as amended, and U.S. Securities Exchange Act of 1934, as amended, and forward-looking information within the meaning of Canadian securities laws. All statements other than statements of historical facts included in this Technical Report Summary that address future activities, events, developments, or outcomes that we or others expect or anticipate will, may, or may not occur in the future, are forward-looking statements and forward-looking information.

Forward-looking statements and forward-looking information include, but are not limited to statements regarding such things as: estimates of Mineral Resources and Mineral Reserves; the gold price and other inputs used to estimate Project output and performance, Project design and availability of required approvals; timing or ability to complete any activity as set forth herein; annual and cumulative gold production at estimated recovery rates over the life of mine; mining methods and procedures; processing methods and procedures; projected Project economics, including but not limited to anticipated production and revenue, cash costs, royalty payments, government royalties and taxes payments, other payments that may or may not have been contemplated, after-tax NPV, IRR, non-U.S. GAAP measures and any other monetarily derived value; and other such matters are forward-looking statements and forward-looking information.

Among the material factors and assumptions used to develop the forward-looking statements and forward-looking information contained in this Technical Report Summary include: the accuracy of test work and interpretation of results used to prepare this Technical Report Summary, Mineral Resources and Mineral Reserves estimates, and exploration findings and assay results; the terms and conditions of the Company's agreements with third-parties; Dateline's approved or expressed business plans; the anticipated timing and completeness of approvals and permissions; the potential occurrence of certain threatened species of flora, vegetation, and fauna within the mine site; no change in laws that materially impact mining development or operations of a mining business; the potential occurrence and timing of a formal investment decision; the anticipated gold production at the Project; the life of any mine at the Project; all economic projections relating to the Project, including estimated cash costs, all-in sustaining costs, NPV, IRR, initial and sustaining capital requirements, reclamation and closure costs, and self-funding reclamation proceeds; and Dateline's objective to advance the Project to be a producing gold mine.

When used in this Technical Report Summary, the words and derivatives of words such as "optimistic", "potential", "indicate," "expect", "intend", "plan", "believe", "may", "will", "if", "anticipate", and similar words or expressions that reference or imply future conditions are intended to identify forward-looking statements and forward-looking information. Statements that include such words or expressions reflect known and unknown risks, uncertainties and other factors that may cause actual results, performance or achievements of Dateline to be materially different from any future results, performance or achievements expressed or implied by such statements.

Such factors include, among others, uncertainty of Mineral Resources estimates, estimates of results based on such Mineral Resources estimates and Mineral Reserves estimates; risks relating to cost increases, scope changes, and consumption requirements for capital and operating costs; risks related to the timing and the ability to obtain the necessary permits, risks of shortages and fluctuating costs of equipment or supplies; unforeseen delays; risks relating to fluctuations in the price of gold and foreign exchange rates; the inherently hazardous nature of mining-related activities; potential effects on Dateline's operations of applicable and influencing environmental and other regulations; risks due to legal proceedings; risks relating to political, social, and economic instability; as well as those factors discussed under the headings "Note Regarding Forward-Looking Statements" and "Risk Factors" in Dateline's Annual Report as filed in September 2025 and other documents filed with the Australian Securities Exchange.

Although Dateline has attempted to identify important factors that could cause actual results to differ materially from those described in forward-looking statements and forward-looking information, there may be other factors that cause results not to be as anticipated, estimated or intended. Except as required by law, Dateline assumes no obligation to publicly update any forward-looking statements or forward-looking information, whether as a result of new information, future events, or otherwise.

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1. EXECUTIVE SUMMARY

1.1 Overview

Dateline Resources Limited and its subsidiaries (collectively Dateline or the Company) retained GRES, along with AMDAD, Agapito Associates, LLC and Tundra, to prepare this Feasibility Study (FS or Technical Report Summary) for its Colosseum Gold Project (the Project) in San Bernardino County, California, United States of America. This Technical Report Summary evaluates a development scenario for a 2.0 million tonne per annum (2.0 Mtpa) processing facility.

Dateline and its subsidiary, Colosseum Rare Metals, LLC, entered into an agreement to acquire an interest in the Project located in San Bernardino County, California on February 23, 2021. The acquisition was completed on October 27, 2021, when the mineral claims comprising the Project were transferred to Dateline. Colosseum Rare Metals, LLC is the operator of the Project.

The Project contains known occurrences of gold, which have been explored and/or exploited historically. Colosseum was mined as two open pits between 1988-1992 when mining was suspended. Processing of stockpiles continued into 1993. No material exploration work was undertaken at Colosseum between 1993 and 2021. Dateline has reported the Mineral Resource estimates in accordance with the SEC's Regulation S-K subpart 229.1300 mining disclosure rules.

The information presented in this Technical Report Summary is intended to assist stakeholders and other readers of this Technical Report Summary in their understanding of the Project and in forming judgements regarding the quality of the data collected, reported, and used in this Technical Report Summary.

1.2 Property Description and Location

Colosseum Mine is located at 35°34'13"N 115°33'58"E in San Bernardino County in the state of California, USA. It is 14km from the California Nevada state boundary. The site is accessed by 16.5km of road from Interstate Route 15. The first 6.2km is sealed and the remaining 10.3km is unsealed.

The Colosseum mine is in a mining region rich in history with activity commencing in the 1860's when exploration commenced, leading to the discovery of the Colosseum mine in 1865. Various small-scale mines were operated intermittently up until the 1970s and 1980s, when large scale exploration took off. Open pit mining and processing on site occurred from 1989 to 1993, with 344koz of gold produced during the period. The mine closed due to a low gold price environment, with a majority of the defined 1.1Moz reserve remaining unmined.

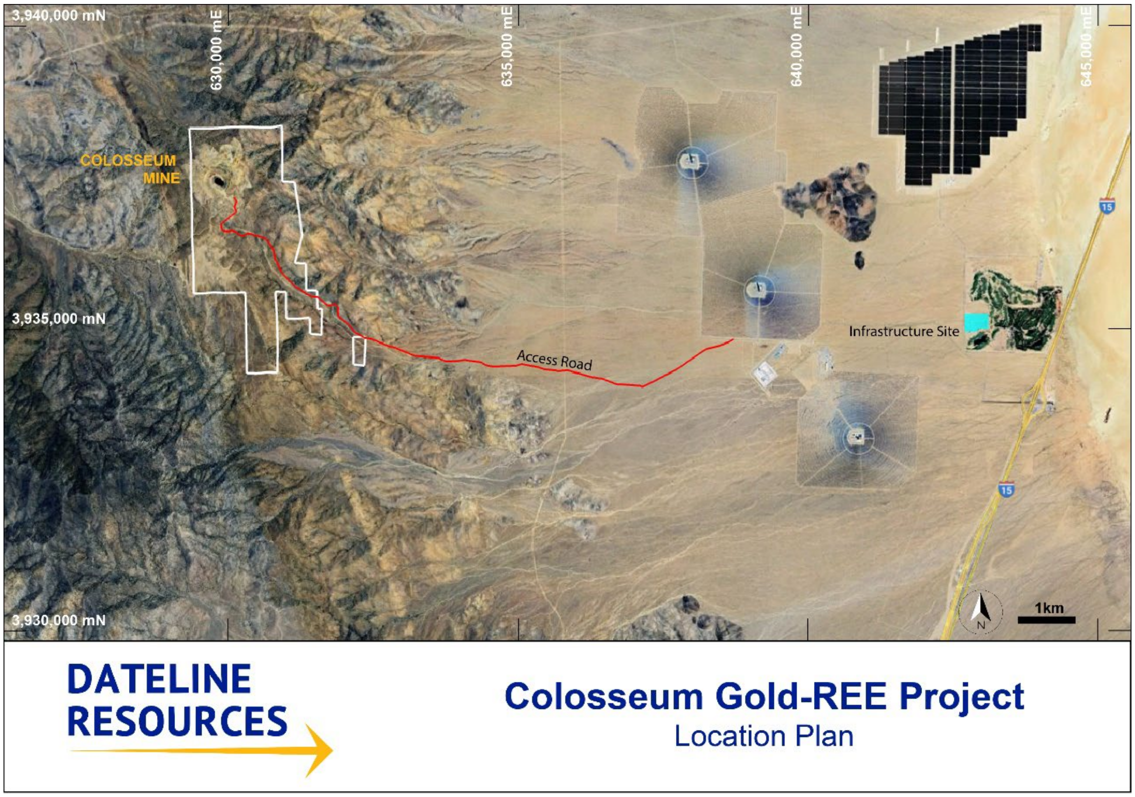
Little work occurred on site from 1994 to 2021 apart from the removal and partial remediation of the processing plant area.

Dateline acquired 100% of the Colosseum Gold Project from Barrick Gold Corporation in February 2021 and committed to the first modern exploration and development program. All available exploration and production data was compiled into a digital database and used for exploration planning and the estimation of a mineral resource estimate (MRE) to JORC-2012 standard.

As part of the acquisition of Colosseum, Dateline also acquired the vested Mining Rights (Mining Rights) and an approved Plan of Operation over the project, which allows for mining to recommence based on existing approvals. The Study has been prepared on the basis that a future development would comply with the requirements of the Mining Rights.

Source: Prepared by Dateline, 2026

Figure 1 Colosseum Location Plan



Source: Prepared by Dateline, 2026

Figure 2 Colosseum Claims Outline

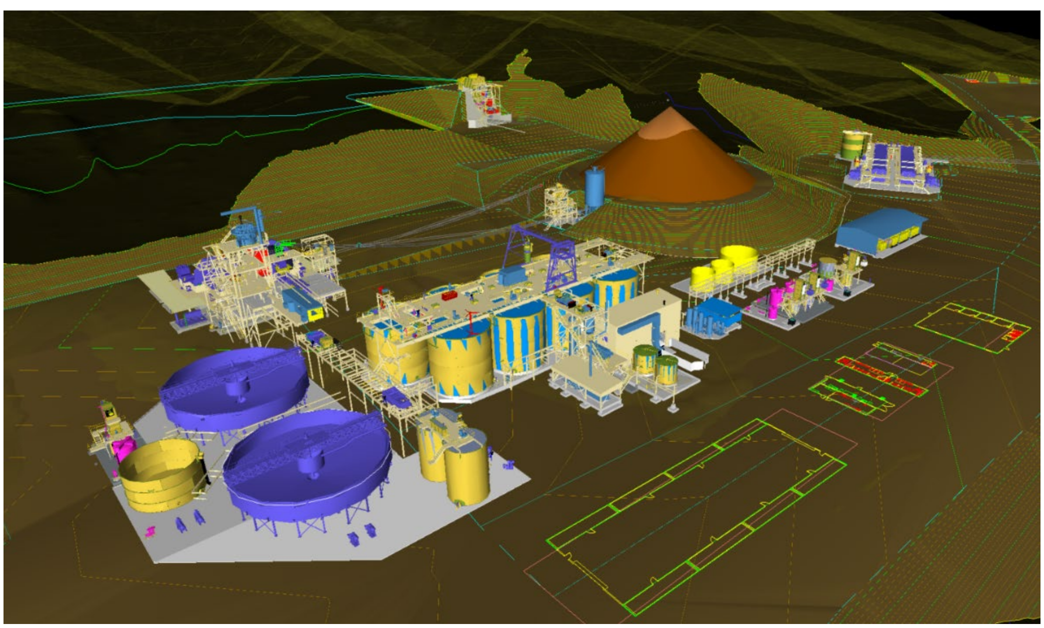


Figure 3 Overall Plant Layout for the Project

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Colosseum is located at the northern end of the Clark Mountains (see Figure 2 and Figure 4). The project area is approximately 1,000 metres north to south and 600 metres east to west with surface elevations ranging from 1,680 metres above sea level in the south to 1,810 metres in the north. The average surface elevation of 1,700 metres is 600 to 700 metres above the alluvial fans and dry lake beds east and west of the mountain range.

Topography in the mine area is steep and irregular. It is gentler to the south of the project area where the process plant and tailings storage facility were located from 1988 to 1993. All drainage over the project area is to the west.

The project is within the Mojave Desert. The climate is arid. Vegetation is sparse consisting of salt bush, pinyon pines and similar species.

1.4 History

The North and South Pipes were mined as two adjacent pits from 1998 to 1992. Operations were suspended in 1993, and the equipment and processing facility dismantled and moved from the site. The South Pit remains as a 130-metre-deep void with the bottom 26 metres flooded to the standing groundwater level. Mining in the North Pit was suspended before it went substantially below ground level, so the benches are still open out onto the western slope of the range.

Waste rock from both pits was dumped on the western side of the range, which drops away immediately west of the pit crests, and in a small area immediately north of the North Pit. The waste rock appears to have been end dumped as it now sits at angle of repose with no benching or contouring.

The tailings storage facility from the former operation remains as a broad flat area south of the mine. The containment wall, overflow spillway and downstream holding dams remain in place.

The processing plant and associated infrastructure was removed from site following the suspension of processing activities in 1993. The concrete foundations were covered over with waste material and were excavated in 2025. Some of the existing concrete foundations and structures are in excellent condition and will be reutilised in the proposed development.


Source: Dateline, 2026

Figure 4 Looking west to Clark Mountains from Colosseum

1.5 Geology and Mineralisation

The Colosseum deposit is located at the southern end of the Sevier foreland thrust belt in the southern Basin and Range Province, SW USA. The project lies within in the Clark Mountain Mining District in the northeast portion of the Clark Mountain Range. The district includes the Mountain Pass rare earth mine 10 kilometres south of the Colosseum Mine, numerous abandoned copper mines, and scattered fluorite, antimony, and tungsten prospects. Most gold and silver deposits in the district are within the northeast quadrant of the district north of Clark Mountain and are associated with emplacement of a felsic breccia complex into Precambrian basement rocks.

The deposit itself is associated with the emplacement of a breccia complex into Precambrian gneissic basement rocks. The complex is comprised of two felsite breccia pipes that form a northeast-southwest elongate zone, which contains mineralised zones of disseminated auriferous pyrite. (See Figure 5 below)

Gold at the Colosseum deposit is generally sub-microscopic and associated with sulphide mineralisation, chiefly pyrite. It occurs as free gold, with minor alloyed silver. Gold is primarily in contact with pyrite, in fractures in the pyrite or along pyrite grain edges. It also occurs as isolated particles in quartz and other gangue minerals but spatially always close to pyrite but rarely as particles encased in euhedral pyrite.

The Colosseum deposit style is a hydrothermal breccia pipe with a combination of epithermal mineralisation at original higher levels and mesothermal mineralisation at the lower levels.

Source: Supplied by Dateline, 2026

Figure 5 Felsite breccia at Colosseum

1.6 Mineral Resource Estimate

Dateline's consultants modelled the mineral resource in 2022, 2024 and 2026.

The July 2022 MRE used Ordinary Kriging (OK) and was mainly based on historic drilling (1991 and earlier) and five holes drilled by Dateline in 2022. It was included in the Australian Securities Exchange (ASX) release "813,000 ounce Mineral Resource estimate for Colosseum Gold Project" dated 6 July 2022.

The June 2024 MRE used Multiple Indicator Kriging (MIK). It added seven holes drilled in 2023 and a further two drilled in 2024. The June 2024 MRE was reported by Dateline in the ASX release "1.1 million ounces of Gold at the Colosseum" dated 6 June 2024.

The April 2026 MRE also used MIK. It added a further 7 RC and 36 diamond holes since the June 2024 estimate.

For clarity, where older geological reports refer to the East and West breccia pipes, current reports and mine plans refer to the South Pipe or Pit (formerly West Pipe) and the North Pipe or Pit (formerly East Pipe).

	Category	Cut-off (g/t Au)	Metric Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Percentage
South Pit	Measured	0.2	5.49	1.33	0.23	
	Indicated	0.2	5.54	0.67	0.12	
	Inferred	0.2	4.72	0.62	0.09	
	Sub-Total		**15.76**	**0.88**	**0.45**	
North Pit	Measured	0.2	11.96	0.85	0.33	
	Indicated	0.2	9.09	0.59	0.17	
	Inferred	0.2	7.69	0.53	0.13	
	Sub-Total		**28.75**	**0.68**	**0.63**	
Combined	Measured	0.2	17.5	1.00	0.56	52%
	Indicated	0.2	14.6	0.62	0.29	27%
	Inferred	0.2	12.4	0.57	0.23	21%
	TOTAL		**44.50**	**0.76**	**1.08**	**100%**

Notes:

(1) Measured, Indicated and Inferred Mineral Resources are inclusive of Proved and Probable Ore Reserves

(2) Mineral Resources are quoted at a 0.2g/t Au cut-off grade.

(3) Mineral Resources constrained within a US$4,200/oz gold pit shell. Pit parameters: Processing and Ore Mining Cost USD27.94/tonne processed, General and Administrative Cost USD3.80/tonne processed, Au Recovery 91%.

(4) Simon Tear of H&S Consultants Pty Ltd (HSC) is the CP responsible for the Statement of Mineral Resources for the Colosseum deposit.

(5) The effective date of the Colosseum Mineral Resource estimate is April 25, 2026.

(6) Differences in the table due to rounding are not considered material.

(7) "-" indicates no reported value.

Table 1 Summary of the 2026 Mineral Resource Estimate

1.7 Mineral Reserve Estimate

No mineral reserves have previously been announced by Dateline.

The Project is currently at the FS stage and is based on a conventional open pit, truck, and hydraulic excavator operation, feeding a nominal 2.0 Mtpa processing plant. The Mineral Reserve evaluation within this Technical Report Summary was supported by a Lerchs Grossmann open pit optimization evaluation, excluding Inferred classified material within the Mineral Resources estimate for the deposits.

The FS level mine design, mine scheduling, mining costing, and overall Project economic model evaluation confirmed positive economic outcomes for the Mineral Reserve. A gold cut-off grade of 0.24 g Au/t was adopted based on economic parameters and recoveries determined as part of this Technical Report Summary. The resulting Mineral Reserve summary includes Proved and Probable Mineral Reserves for the Colosseum deposit.

Detailed mine design, production schedule, and the mining costs have been used in the economic model. The final slope zone values reflective of a spiral ramp design were used for optimization, and the final shell selection forms the basis for subsequent phase design and production scheduling.

The resultant Mineral Reserves summary is shown in Table 2.

	Category	Cut-off (g/t Au)	Metric Tonnes (Mt)	Grade (g/t Au)	Contained Gold (k ounces)	Percentage
South Pit	Proved	0.25	4.4	1.45	200	
	Probable	0.25	3.2	0.74	80	
	Sub-Total	0.25	**7.6**	**1.15**	**280**	
North Pit	Proved	0.25	9.8	0.88	280	
	Probable	0.25	3.3	0.70	70	
	Sub-Total	0.25	**13.0**	**0.83**	**350**	
Combined	Proved	0.25	14.1	1.06	480	76%
	Probable	0.25	6.5	0.72	150	24%
	Total	0.25	**20.6**	**0.95**	**630**	**100%**

Notes:

(1) *The Mineral Reserves point of reference is the point where material is fed into the process plant.*
(2) *Colosseum deposit Mineral Reserves are reported using a 0.25 g Au/t cut-off grade and USD4,200 per ounce gold price.*
(3) *John Wyche of AMDAD is the CP responsible for the Statement of Mineral Reserves for the Colosseum Proved and Probable Mineral Reserves.*
(4) The effective date of the Colosseum Mineral Reserves is May 11, 2026.
(5) Differences in the table due to rounding are not considered material.
(6) The Mineral Reserves were estimated in accordance with subpart 229.1300 of Regulation S-K.
(7) "-" indicates no reported value.

Table 2 Project Mineral Reserves Estimate

1.8 Mining Methods

During the Scoping Study for Colosseum, various mining methods including continuous mining, bulk mining and selective mining were considered. For the size and geological properties of the orebody, selective mining was chosen as the preferred method.

This was again considered in this study, with selective mining using hydraulic backhoes selected as the opencut mining method.

Colosseum will be a small to medium scale opencut gold mine. Annual material movements would be:

- 2.0 Mtpa mill feed and an average of 10.3 Mtpa waste rock peaking at 14.7 Mtpa in Year 1.

Apart from some areas around the pit crests, which would mine through old waste rock emplacements, all material will require drilling and blasting.

Grade control will be critical to achieving maximum recovery of the resource with minimal dilution.

It is likely that higher grade ore and lower grade ore will be mined and stockpiled separately. If required, in conjunction with the Waste Rock Management Plan, different waste rock types will be mined separately and stored in the waste rock dump according to the guidelines established by the environmental and geochemical specialists.

Ore that is mined during the pit development phase prior to commissioning of the mill would be placed in a stockpile on the ROM pad or in a temporary stockpile adjacent to the ROM pad or another suitable site.

Waste rock will be co-stored with tailings in an emplacement south of the south-western side of the South Pit. Tailings from the mineral processing plant will be de-watered in a press filter and "dry-stacked" in a cell within the mine waste rock storage facility. The tailings and mine waste rock will be placed separately so the two material types will not be co-mingled. The co-storage will be undertaken accordance with the Waste Rock Management Plan and Waste Discharge Requirements.

1.9 Mineral Processing and Metallurgical Testing

Historical metallurgical test work was carried out between 1983 to 1987 for the proposed Project to support the earlier phase of production between 1988 and 1993. These earlier studies, coupled with five years of operating information from the mine, were used as the basis for a further round of test work in 2025 in support of this Technical Report Summary.

The Colosseum ore host rock is very hard and competent. Gold is fine grained (<10 µm) and associated with sulphide minerals and quartz. The historical test work has demonstrated that the ore is amenable to gold extraction by conventional cyanidation processes but requires fine grinding to achieve moderately high gold extractions.

The processing plant design is based on the treatment of 2.0 Mtpa (6 ktpd) of hard ore from the Colosseum open pits. The key process design criteria that the plant was based upon is shown in Table 3. The flowsheet will consist of a simplified crushing circuit design to limit dust creation, maintaining a single stage crushing circuit. Comminution data supports a semi autogenous grind, ball mill and pebble crushing circuit (SABC). A primary grind size of P_{80} 106µm has been adopted for the design.

A 24-hour leach time will be sufficient for the recovery of gold with additional capacity to 27 hours to assist with improved silver recovery. A CIL circuit was adopted early in the design based on the historical plant design basis. The current design test work supports either leach/CIP or a CIL design. It is noted that the ore is not considered to be preg-robbing with low organic carbon assay levels.

The design includes a provision for oxygen injection into leach tanks 1 and 2 based on historical production reports where low dissolved oxygen levels had affected the recovery of gold. The design has allowed for a 3 tpd PSA plant.

INCO detox, used previously, has been maintained in the current flowsheet. If required some of the O_2 generated by the PSA plant can be redirected to the detox system. The default will be AIR/SO_2 system using SMBS as the supply reagent for SO_2 delivery.

Cyanide will be supplied by a Isotank dissolution system via Orica Chemicals – this will remove personnel from the mixing of cyanide and the dangers associated with the reagent.

Industry standard elution, electrowinning and smelting circuits will be used to produce gold doré. The elution circuit will include carbon regeneration.

Dry stack tailings will be required for the process. Testing of both pressure and vacuum filtration have both demonstrated that a sub 20% moisture cake can be developed. The process will include 2 x 182 m^2 HBVF with vibration rollers to ensure the cake is under 20% moisture for materials handling and stacking of cake. The cake will be paddock dumped and moved by dozer over time. The arid climate in the area will allow for a high evaporation rate hence there was some relaxation on the moisture content to 20% moisture at nominal operation and a design basis of 15% moisture.

Description	Unit	6 ktpd
Annual Ore Feed Rate (ROM feed)	Mtpa	2.0
Operating Days per Year	d/a	365
Daily Ore Feed Rate (ROM feed)	tpd	6,000
ROM Feed F_{100} Size to Primary Crushing	mm	800
Primary Crusher P_{80}	mm	140
Crushing Rate (6,570 hours per year)	tph	304
Milling Rate (8,000 hours per year)	tph	250
Mill Circuit Circulating Load	%	250
Gold Head Grade (Mill Feed) – Design	g Au/t	1.20
Design Ore Specific Gravity	t/m^3	2.70

Description	Unit	6 ktpd
Design Abrasion Index	-	0.325
Design Crushing Work Index	kWh/t	11.8
SMC Drop Weight Index	kWh/m^3	8.38
Design Rod Mill Work Index	kWh/t	20.5
Design Ball Mill Work Index	kWh/t	20.1
Mill Grind P_{80} Size	µm	106
Leach System	-	CIL
Leach Slurry Density	% solids w/w	50
Total Leach and Adsorption Time - Design	H	27
Elution System	-	Split AARL
Final Tailings Cyanide Destruction Type	-	Air/SO_2
Overall Recovery (LOM Average)	% Au	91.1

Table 3 Key Process Design Criteria

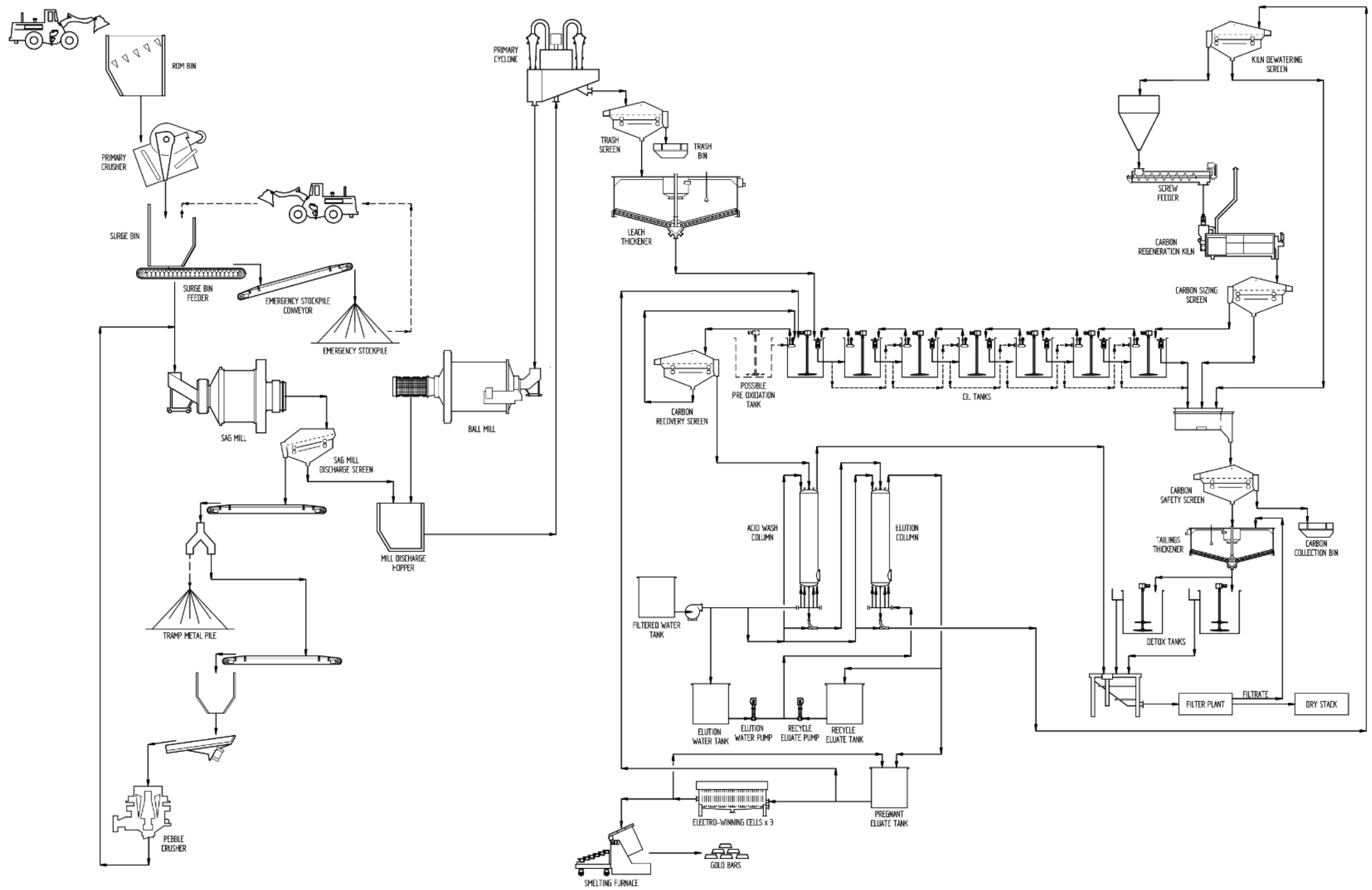


Figure 6 Process Plant Flowsheet

1.10 Project Infrastructure and Access

Access to local resources and infrastructure is considered to very good, with Interstate I-15 located within 10 miles of the mine. I-15 is a major highway that passes through Las Vegas, Nevada, located 55 miles away.

Las Vegas is a major city in Nevada with the region having a population of 2,400,000+, with schools, airport, accommodation, communications, hospitals and supplies to support a 'drive in-drive out' residential workforce.

The access road from I-15 consists of four miles of sealed road and six miles of recently upgraded unsealed road.

Grid power will be supplied to the site via a 10km 34.5kV overhead powerline that follows the existing access road to site.

The property currently does not have any reticulated power or water supplies. There are three shipping containers that are used for storage and exploration purposes.

Planned infrastructure for the site includes the following:

- Mine Infrastructure will be supplied, installed by a mining contractor and includes – (HME workshop and warehouse, maintenance support facilities, contractor laydown and storage yards, fuel and lubricant farm, explosives storage and facilities, water cart filling point (Turkeys Nest), Mine administration and personnel facilities, information and communication).
- Waste Rock Dump (WRD) (existing and future).
- Water Treatment Plant (WTP).
- Waste Rock Water dam.
- Power Supply via offsite substation and transmission line.
- Pit Dewatering system.
- Communications.
- Gatehouse.
- Emergency Services Building.
- Process Plant and Administration Building.
- Process Plant Workshop and Stores building with offices.
- Reagents Storage Facility.
- Process Plant Control Rooms.
- Sample Preparation and Laboratory, and
- Solid and liquid waste disposal facilities.

1.10.1 Water Supply

Water will be supplied from two water bores, Colosseum #1 and #2, both located at the base of the access road, near I-15. These bores were used in the earlier operation of the mine and have demonstrated sufficient capacity for the mine's requirements. Water will be piped to the mine following the route of the access road.

Dateline has secured a lease over a 27-acre parcel of land at the base of the Colosseum access road, near I-15. Initially, this will be utilised as a laydown area during construction, transitioning to a storage facility and employee car park when production commences.

1.10.2 Plant Infrastructure

The Colosseum Gold Project is being designed on the basis of 2 Mtpa (metric) gold milling and carbon in leach circuit using a conventional cyanide leaching operation. The tailings from the cyanide leach will be detoxified using the INCO – SO_2/Air process prior to filtration for dry stacking of a 15% to 20% moisture cake product.

The project will have the following key operating areas

- Primary crushing and coarse ore stockpile;
- Ore storage and reclaim;
- Grinding and classification;
- Leach feed thickening;
- Leaching and adsorption (Carbon-In-Leach);
- Elution and gold recovery;
- Tailings thickening and detoxification;
- Detoxified tailings filtration;
- Reagent mixing, storage and distribution;
- Electrical power and control systems;
- Water and air services.

1.10.3 Mine Facilities

Non process infrastructure (NPI) is allowed for in the capital estimate. This includes buildings such as the administration building, workshop and warehouse building etc. Roads internal to the processing facility are included.

The main access road from the I-15 highway to the site is upgraded to provide all weather access for light vehicles. Costs for this upgrade are considered as pre-work and are not included in the capital estimate.

A raw water supply pipeline to convey water from the wells located near to the Ivanpah substation to the site raw water tank is included in the capital estimate. A number of upgrades to existing wells and equipment are allowed for. Likewise new electrical infrastructure to service the wells and lift pumps is allowed for.

1.10.4 Tailings Storage Facilities

A dry stacked tailings system is proposed for the Colosseum project. A wet tailings dam will not be used. Tailings filtration will be undertaken by two 182 m^2 horizontal vacuum belt filters (HVBF) with vibration rollers. The filters will produce a tailings cake at a moisture of 15%-20%. Residual WAD CN to the cake limit of 18mg/l and solution limit of 1mg/l will be maintained by the detoxification circuit. Filtrate will be pumped back to the tailing's thickener for circuit water recovery.

The tailings cake will be transported by truck to be co-deposited with the waste material from the mine.

1.11 Site Services

1.11.1 Power Supply

Access to power will be from the local Southern California Edison grid (34.5 kV) The power access is located near to the SoCal Edison Ivanpah substation. Power is transmitted to the project site (at 34.5 kV) via an overhead transmission line which follows the alignment of the main access road (approx. 10km). Power is reticulated on site at 13.8 kV. The system includes the utility metering at the process plant, HV switchgear and the following reticulation of power to the process plant and the infrastructure.

1.11.2 Water Supply

Water will be sourced from the existing Colosseum #1 and #2 water bores, which were previously used when the mine originally operated. Water will be pumped along the side of the access road to the mine.

Dateline has also included a tailings filter press into the processing circuit, which will significantly increase the reuse of water in the plant, rather than being lost to evapotranspiration.

1.11.3 Fuel Supply and Storage

Fuel tanks and associated bunds and pumping infrastructure located on site will be provided by and maintained by the mining contractor as part of the mining contract. The mining contractor will be responsible for demobilisation at the end of the Project.

1.11.4 Communications

Starlink connections are to be used for internet connections.

1.11.5 *Road Access*

There is a bitumen access road from I-15 at Yates Well Road. The road continues to the west past the Ivanpah solar facilities before a turn off to the Colosseum gravel access road. The road has been upgraded as part of the pre-works and is suitable for use for construction and mine operation.

1.11.6 *Accommodation and Camp Facilities*

No camp or accommodation facilities are needed for the Project. Construction contractors are expected to be accommodated in Primm, whilst mine operations personnel will be based in Las Vegas, Nevada.

1.11.7 *Site Buildings and Support Infrastructure*

Office administration and support offices will be constructed on site. Where possible, the existing concrete foundations from the original offices will be reutilised.

1.11.8 *Waste Management*

Waste rock disposal will be on designated waste dumps surrounding the open pit to the west and south of the open pits.

Physical man-made waste from operations will be collected in designated waste bins and periodically removed from site. No man-made waste is intended to be disposed of on-site via burial or incineration.

Sewage waste will be collected in septic tanks and regularly pumped out and removed from site.

1.12 Market Studies and Contracts

The price of gold is the primary factor in determining the Project's profitability and cash flow from operations. The gold price of $3,800 per gold ounce used in the technical analysis was derived from a combination of sources, including consensus forecasts reflecting a composite of financial institutions, gold prices used in various recent technical reports completed by mining companies, developers and consulting groups, and recent historical price trends.

For the economic analysis, the financial model was generated using a gold price of $4,200 per ounce as well as the gold spot price as of 7 May 2026 of $4,700 per ounce.

Dateline has no refining or bullion sales contracts in place. Commitments to deliver gold bullion are presently limited to 3,000 ounces of gold to be sold at a discounted price over a six-year period relating to a funding agreement entered into prior to the commencement of the FS.

Dateline expects that terms contained within any refining, sales, or other contracts for delivery of gold bullion will be typical of, and consistent with, standard industry practices.

1.13 Environmental Studies, Permitting, Social and Community Impact

1.13.1 Environmental

Environmental and Social/Community impacts for Colosseum were originally addressed for the operation by a Draft Environmental Impact Statement (EIS) (Bureau of Land Management 1985a) and Final EIS (Bureau of Land Management 1985b).

Environmental and Social/Community commentary was also provided by the U.S. Environmental Protection Agency (EPA) in 1993.

Mining and processing from 1988 to 1993 were managed in accordance with the EIS and Plan of Operations and Reclamation Plan for the Colosseum Project, 21 July 1986.

Dateline notes that when mining and processing operations were suspended in 1993, the EIS and Plan of Operations remained in place and would apply to the resumption of activities at Colosseum. These documents for Dateline's minimum standard for the project.

1.13.2 Potential Emissions, Waste, and Effluents Generated by the Project

The mining operations will involve removal of waste rock, removal of ore, and ore processing by milling and tank leaching. Dust emissions from the mine haul roads will be minimized by application of chemical dust suppressants and water to the haul roads.

Particulate emissions resulting from ore crushing and handling will be controlled using a high efficiency wet scrubber. All other operations (milling and leaching) will either be fully enclosed or will be completely wet and will have negligible particulate emissions.

Gaseous emissions at the Colosseum Project will result from the operation of diesel haul trucks, from storage of diesel fuel in an on-site tank, and from the combustion of propane used in a small boiler at the mill.

Dateline intends to operate Colosseum within the limits of the existing Plan of Operations and valid Mining Rights, with regulatory reporting as required.

1.13.3 Closure and Post-Closure Stage

The mine will be progressively rehabilitated, with waste dumps contoured and seeded with local plants.

At the completion of mining and processing activities, the plant and associated infrastructure will be removed from site and the area, graded, contoured and seeded to replicate the natural landform.

The model assumes that the removal and rehabilitation costs will be fully offset by the sale/ salvage value of the plant and associated infrastructure.

1.14 Capital and Operating Cost Estimates

1.14.1 Capital Cost

Capital costs have been developed from first principles with quotes for all major equipment components. A turnkey engineering, procurement and construction model has been used as the basis for the Project construction. The Technical Report Summary contemplates a 15-month period for engineering, construction and commissioning. Contract mining at an average rate of 3.4 Mtpa (ore and waste) and grid power supplied via a transmission line that follows the access road.

The R&M costs for the processing plant include minor capital replacements. The current mine life does not warrant capital expenditure for major components of the processing plant.

Summaries of capital costs are shown in Table 4.

Capital Expenditure Item	Initial Capital Cost (USDM)	Operational Phase Capital (USDM)
Capitalised Mining	$ 16.3	$ 36.2
Process Plant	$ 95.0	-
Process Infrastructure	$ 25.9	-
Management, Engineering, EPC Services	$ 47.1	-
Preproduction Costs and Capital Spares	$ 23.9	-
Reclamation	N/A	$ 10.0
California Sales and Use Tax	$ 9.2	-
Sub-Totals: Capital Expenditures	$ 249.1	$ 46.2
Combined Engineering Growth and Contingency	$ 25.5	-
Total Capital Costs	**$ 274.6**	**$ 46.2**

Table 4 Capital Expenditures

1.14.2 Operating Cost

Mining costs have been provided by well-established U.S. contract miners. Power costs are based on a proposal from SoCal Edison to provide grid power to site via a 34.5 kV powerline along the existing access road.

Processing and G&A costs have been developed from first principles with major consumable supply component quotes and competitive U.S. labour rates. The operating costs contemplate that the workforce will be contracted on a drive-in-drive-out basis (DIDO) out of Las Vegas, Nevada.

Summaries of operating costs, before taxes and depreciation, are shown in Table 5.

Operating Cost Item	Units	Years 1-6	LOM Yr 1-11
Mining Costs	USD/t mined	$4.15	$4.63
Processing Costs	USD/t processed	$20.54	$20.56
G&A Costs	USD/t processed	$4.48	$4.45
NSR Royalty	USD/t processed	$2.73	$1.74
Refining Costs	USD/t processed	$0.32	$0.22
Total Cash Costs	USD/t processed	$62.03	$45.66

Table 5 Operating Expenditures

1.15 Economic Analysis

Project economics for the 2.0 Mtpa operation are based on inputs developed by GRES, AMDAD, Agapito Associates, LLC, Tundra Resource Analytics, EY and Dateline. Economic results presented in this Technical Report Summary suggest the following conclusions, assuming a 100% equity project, a gold price of USD4,200/oz.

- Mine Life 6 years
- Production Life 10.4 years
- Pre-Tax $NPV_{5\%}$ USD785 million, IRR: 49.5%
- After-tax $NPV_{5\%}$ USD551 million, IRR: 38.6%
- Payback (After-tax) 3 years
- NSR Royalty Paid USD35.8 million
- County Taxes Paid USD16.6 million
- California State Taxes Paid USD124.8 million
- United States Federal Taxes Paid USD206.5 million
- Cash costs (C1, including Royalties) USD1,651/oz Au
- All In Sustaining Costs (AISC) USD1,825/oz Au

Project cost estimates and economics were prepared on a monthly basis. Based upon design criteria presented in this Technical Report Summary, the level of accuracy of the estimate is considered ±10-15%.

Costs and economic results are presented in Q2 2026 U.S. dollars unless otherwise stated. No escalation has been applied to capital or operating costs. The 5% discount rate used is a gold mining industry standard in North America generally used for comparability purposes among projects; it is not intended to fully reflect consideration of cost of capital, risk adjustments, or other factors.

Technical and economic tables and figures presented in this volume require subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding, which are not considered to be material.

1.16 Interpretations and Conclusions

This Technical Report Summary shows that the mine plan is technically achievable and economically viable taking into consideration all material modifying factors. The resultant Mineral Reserves are also reasonable and achievable.

The mining operations will be executed by a tier 1 U.S. contract mining company, selected for its capability to manage medium-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 90-120 tonne class hydraulic excavators and 50-60 tonne class articulated haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts. The contractor will also provide site mining infrastructure and all personal to operate and maintain all mining equipment, while ensuring its supervision and operations management.

Reasonable mine designs, mine production schedules, and mine costs have been developed for the Project. Costs for mining and mining infrastructure have been provided based on the analysis of Technical Report Summary mine schedule completed by a tier 1 U.S. mining contractor and consider local site and California requirements, and availability of resources such as equipment and labour. These mining costs have incorporated within the overall Project financial model.

Several opportunities and risk have been identified within this Technical Report Summary, which can be managed as the project progresses its development through to execution and pre-production stages of development towards commencement of the mining operation.

Most of the capital and operating costs are within the front end of the plant. The plant has a restricted front-end layout due to the limited available land and so is restricted in the ability to expand this area of the plant due to the waste dumps, water course and other restricted areas.

When laying out this plant, the remanent concrete foundations were considered as much as possible to reduce front end capital costs. The admin plant foundations and crusher footings and tunnel will be reutilized in this plan.

1.17 Recommendations

All required work is complete for this Technical Report Summary, and no additional work is necessary for this phase. This Technical Report Summary presents a project that is ready for submission for financial and other support necessary to progress to the next phase.

The next phase of the Project is the FEED process, which leads to a financial investment decision. The following are considered part of the FEED phase and represent the normal progression of the Project from an FS towards construction.

- Secure long lead time items associated with the project.
- Commence an Early Contract Involvement process to define costs and execution strategy for the Processing Plant.
- Commence recruitment of key construction and execution personnel.
- Detail system requirements to support project development.

With regards to the Mineral Resources at Colosseum, several areas have been identified for expansion/ extension of the Mineral Resources. Some are currently being assessed with others to be reviewed closer to the commencement of mining.

For the mining area, there are two critical risks identified that are recommended to be a focus for future planned work. These are related to grade control, as well as the engagement of the mining contractor in sufficient time to be ready as per the current Project schedule.

The focus of grade control work is to design a robust system to ensure the Mineral Resource is maximised.

For the recommendation regarding the mining contractor ensuring it is engaged with sufficient time to be ready as per the current projects schedule this is easily addressed by planning to have commercial discussions and negotiations as part of a formal tender process with mining contractors at least 9 months prior to required site mobilization. This should be easily implementable based on current Project timing, and recent experiences and interactions with four established US mining contractors and other service provides as part of this Technical Report Summary, that are aware of the Project and enthusiastic to be involved.

2. INTRODUCTION

Dateline Resources operates in the gold and critical minerals industry. The Company's flagship asset is its 100% owned Colosseum Gold Project in San Bernardino County, California. The Project has valid Mining Rights, an Approved Plan of Operations and operating permits for the 2.0 Mtpa project, which was the basis of past technical studies.

Dateline was originally incorporated on February 3, 2011, as "Conto Resources Limited" under the *Corporations Act, 2001*. In December 2013, Conto Resources Limited changed its name to Dateline Resources Limited.

2.1 Purpose of the Technical Report

This Technical Report Summary was prepared in accordance with the disclosure requirements of Subpart 229.1300 of Regulation S-K 1300 Technical Report Summary for Dateline by GRES, to be attached as an exhibit to support mineral property disclosure, including Mineral Resource estimates and Mineral Reserve estimates for the Colosseum Gold Project. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in GRES's services, based on:

1. information available at the time of preparation,
2. data supplied by outside sources, and
3. the assumptions, conditions, and qualifications set forth in this Technical Report Summary.

This Technical Report Summary provides Mineral Resource and Mineral Reserves estimates, and a classification of Mineral Resources and Mineral Reserves in accordance with the definitions in subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations in Regulation S-K 1300 (S-K1300).

This Technical Report Summary is a comprehensive study of a range of options for the technical and economic viability of a mineral that has advanced to a stage where a preferred mining method and the open pit configuration is established, and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the modifying factors and the evaluation of any other relevant factors which are sufficient for a CP, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserves at the time of reporting. Modifying factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors.

This Technical Report Summary contains forward-looking statements; refer to the note regarding forward-looking statements at the front of the Technical Report Summary.

2.2 Previous Technical Report

This Technical Report Summary supersedes the previous Updated Scoping Study for the Colosseum Gold Project dated May 26, 2025.

2.3 Background Information

Dateline retained GRES, to coordinate several consultants under the supervision of Dateline to prepare this Technical Report Summary. The FS (Technical Report Summary) evaluates a development scenario of a 2.0 Mtpa processing facility.

The 2.0 Mtpa operation includes:

- Average annual gold production of 75,400 ounces during years 1-6 and 573,000 ounces over the 10.4-year life of mine.
- Average ore grade of 1.32 grams gold per tonne (“g Au/t”) over the first 6 years of operations and 0.95 g Au/t over the life of mine.
- LOM average gold recovery of 91% from single-stage crush, SABC grinding circuit, and CIL recovery circuit.
- Contract mining and grid power supply reduce capital costs and operational risks.
- Initial capital requirements of USD274.6 million, including contingency and capitalised mining.

2.4 Detailed Personal Inspections

1. The Competent Person (CP), Graham Craig, for the Geology studies of this Technical Report Summary works full-time on the Colosseum Project and is based in Las Vegas. Mr Craig maintains a comprehensive drill hole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs.
2. Various site visits were undertaken in 2025 and 2026 by GR Engineering Services professionals, including Daniel Bird, Mick Gavrilovic and Bob Dickey for the mineral processing aspects of this Technical Report. The purpose of the visits was to review the plant site location, gather relevant site data and gather historical operations data.
3. A site visit was undertaken in 9-12 December 2025 by AMDAD professionals, including John Wyche, the Competent Person for the Mineral Reserves aspect of this Technical Report. Mr Wyche visited the Colosseum North and South pits as well as the Las Vegas office. The purpose of the visit was to inspect the drill hole database and understand the layout of the project.

CPs not listed above have not visited or inspected the property. Personal inspections by these CPs are not required to complete their responsibilities.

The CPs consider that the site visits conducted prior to 2026 can be regarded as current personal inspections on the basis that the work completed on the Project since that time has been reviewed and the CPs are of the opinion that the limited work carried out on the Project since 2021 is not material. The CPs are satisfied that no unauthorised access or other work has been conducted on the property based on the site security including site access via a paved road through a locked security gate combined with the fact that the site is continuously manned by Company personnel. Finally, the CPs also reviewed publicly available information on the Company and its activities including the audited financial statements of the Company, which the CPs are satisfied do not point to any additional work being conducted on the property.

2.5 Capability and Independence

GRES is an Australian Stock Exchange (ASX) listed engineering and construction company with a global footprint and has previously been involved with feasibility studies and project delivery in Australia and other locations regionally. AMDAD provides independent mining advisory services to the global mining and finance sectors. Within its core expertise it provides independent technical reviews, resource evaluation, mining engineering and mine evaluation services to the resources and financial services industries. Agapito Associates, LLC is an American consulting and engineering services firm. Agapito provides consulting, engineering, program management, and construction management services in the areas of water, environment, infrastructure, resource management, energy, and international development. Tundra Resource Analytics is an Australian financial analysis and modelling consultancy, specialized in the mining industry.

All opinions, findings and conclusions expressed in this Technical Report Summary are those of the Competent Persons and their specialist advisors.

2.6 Reliance on Other Experts

The following individuals, by virtue of their education, experience and professional association, are considered CPs as defined in the subpart 229.1300 of Regulation S-K, for this Technical Report Summary, and are members in good standing of appropriate professional institutions.

2.7 Sources of Information and Data

The primary technical documents and files relating to the Project, including previous technical reports, research documents and historical available information on the Project, that were used in the preparation of this Technical Report Summary are listed in Section 24-References.

2.8 Effective Dates

The effective date for the estimation of Mineral Resources is May 11, 2026, the effective date for the estimation of Mineral Reserves is May 11, 2026, and the effective date for the estimation of capital and operating costs for the project is May 1, 2026.

2.9 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

The metric system has been used throughout this Technical Report Summary. Tonnes are metric of 1,000 kilograms (kg), or 2,204.6 pounds (lb). Gold is reported in troy ounces (oz), equivalent to 31.1035 grams (g). The summary capital and operating cost currency is in Q2 2026 U.S. dollars (USD) unless otherwise stated.

3. PROPERTY, DESCRIPTION AND LOCATION

3.1 Location

The Colosseum Project is located in the Clark Mountains, San Bernardino County, California, 45 miles (72.4 km) southwest of Las Vegas, Nevada (population ~ 2,400,000); 7 miles (11.3 km) northwest of Mountain Pass, CA.

3.2 Land Tenure

The property covers approximately 1,381 acres (559 ha) and is comprised of 80 unpatented lode mining claims and mill sites, and two patented mining claims – Colosseum No. 1 and No. 2.

Unpatented claims are held on federal lands and confer a possessory mineral interest subject to compliance with applicable federal and state requirements. Patented claims are private land (fee simple) and form an important part of the project footprint, including legacy mine infrastructure areas. The project has an approved Plan of Operations, with the Plan confirmed by the US Department of the Interior and the Bureau of Land Management.

Figure 7 Colosseum Project Area – South Pit

3.3 Jurisdiction and land management context

Colosseum sits within the Mojave National Preserve, a unit of the National Park System established under the California Desert Protection Act, 1994. Public commentary and agency material highlight that mining activities at Colosseum trace back to a Bureau of Land Management approved Plan of Operations dating to the 1980s, which is central to the project's grandfathered operating framework within the Preserve.

Surface management and broader land stewardship sits with the National Park Service as the Preserve manager. Mining operations and mining plan compliance rely on the existing Plan of Operations framework administered through the Department of the Interior and BLM, with inter-agency coordination and ongoing compliance obligations.

3.4 Acquisition and ownership

In March 2021, Colosseum Rare Metals, Inc., a wholly owned subsidiary of Dateline Resources entered into an agreement with LAC Minerals (USA) LLC, a wholly owned subsidiary of Barrick Gold Corporation to acquire the Colosseum Gold Mine, located in San Bernardino County, California for the 80 unpatented claims and millsites, and 2 patented claims which host the mine and its surrounding areas covering approximately 1,381 acres.

In October 2021, Dateline announced that all outstanding conditions precedent for the completion of the acquisition had been fulfilled. As part of the transaction, Dateline has provided US$770,000 in reclamation bonds to replace the Barrick bonds with the relevant authorities. Under the terms of the agreement, the Company has the right to use the property for exploration and mining purposes, provided it meets the following obligations: Dateline to pay Barrick US$1,500,000 on the earlier of:

- Completion of a bankable feasibility study; or
- Commencement of site development for the extraction of ore; or
- Sale of the properties.

Barrick was entitled to a 2.5% Net Smelter Return royalty on all future production of any metals from the Colosseum Gold Mine. It has since sold the royalty to Triple Flag Precious Metals Corporation.

4. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Access

The Colosseum Project is in east San Bernardino County, California, about 75 km southwest of Las Vegas, Nevada. It sits less than 10 km north of the Mountain Pass rare earths operation.

Regional access is via Interstate 15 (I-15), the primary highway corridor between Southern California and Las Vegas. Site access then uses a local road network linking from the I-15 corridor to the Colosseum Mine Road and internal mine haul and service roads developed during prior operations.

Dateline upgraded approximately 10 km access road from I-15 to site to support heavy haul vehicles.

The existing disturbance footprint includes legacy pits, plant areas, waste dumps, tailings facilities, and established internal tracks. These reduce early-stage access and laydown requirements relative to a greenfield project.

4.2 Climate and Physiography

The project lies in the Mojave Desert high desert environment. Mojave National Preserve climate varies strongly with elevation. Annual precipitation ranges from about 3.5 inches at lower elevations to nearly 10 inches in the mountains. Most rainfall occurs between November and April. Summer monsoon thunderstorms bring short-duration, high-intensity rainfall. Snow occurs in the mountains during winter. Winds are a prominent feature, with strong winds common in fall, late winter, and early spring.

Climate normals from nearby Mountain Pass support this high-elevation desert profile and provide a useful proxy for operational planning assumptions. Published station normals indicate average annual precipitation of about 8.9 inches.

For design and operating planning, the controlling implications are:

- High summer temperatures and high diurnal variability drive heat management and water balance sensitivity.
- Winter freezing conditions at elevation affect water lines, reagents, and maintenance routines. Thunderstorm events and wind seasons affect dust control, trafficability on unsealed segments of road, and stormwater management design.

4.3 Local Resources and Infrastructure

4.3.1 Workforce and Services

The project sits within easy reach of the large regional labour market and service base of Las Vegas. Regional towns and highway services along the I-15 corridor provide routine logistics support. The historical mine footprint supports practical staging from existing disturbed areas.

4.3.2 Water

In December 2025, Dateline executed an option agreement to secure water and land rights near the base of the Colosseum Mine Road. Dateline reported the agreement provides the right to lease about 27 acres and includes two dedicated groundwater wells, described as primary and back-up, with supply stated as sufficient to meet full-scale mining and processing water demand.

4.3.3 Power and Consumables

The Colosseum Project is a brownfield site within an established regional mining and industrial corridor that includes Mountain Pass. This supports access to common mining consumables, contract services, and freight logistics using the I-15 corridor. Power will be supplied via an overhead transmission line from an electricity grid operated by SoCal Edison or other infrastructure located in the Ivanpah Valley.

4.4 Topography, Elevation and Vegetation

The project is in the Clark Mountain Range within Mojave National Preserve. Terrain is mountainous to rugged, with steep local relief and incised drainages typical of the Preserve's higher elevation zones. The mine area is situated at about 5,500 to 6,000 ft elevation.

Vegetation communities vary with elevation. Across Mojave National Preserve, lower elevation creosote scrub and Joshua tree woodland transition upslope to pinyon-juniper woodland at higher elevations. The Colosseum site sits within the higher elevation life zone where pinyon-juniper woodland is a common community.

Fauna is typical of the Preserve's mountain and transition habitats. Desert bighorn sheep occur in the Preserve and use rocky terrain and limited water sources effectively.

4.5 Regional Operating Context

MP Materials' Mountain Pass operation is the nearest major operating mine and sits about 10 km south of Colosseum.

5. HISTORY

5.1 Exploration History

The Clark Mountain Mining District was organised in the 1860s. Gold mineralisation at Colosseum was first reported in 1865. No recorded production is documented until the 1930s. During that decade, approximately 615 troy ounces of gold (19 kg) with minor silver, copper, and lead were produced, as reported by Hewett (1956). The mine closed in 1942 following designation as a non-essential industry during World War II. Post-war activity consisted of intermittent small-scale exploration by private parties and junior companies.

Modern exploration commenced in the early 1970s. Between 1972 and 1974, Draco Mines completed detailed mapping and sampling and drilled five diamond holes totalling 7,065 ft (2,153 m) to test the breccia pipe complex. Work initially targeted molybdenum indications, with focus shifting to gold during the mid-1970s as geological understanding of the breccia-hosted system improved.

During 1975 and 1976, Placer Amex drilled 18 diamond holes under a joint venture with Draco. Fifteen holes tested the West (South) Pipe, and three holes tested the East (North) Pipe, totalling 8,253 ft (2,516 m). Placer exited the joint venture in late 1976. In 1980, Draco completed 29 rotary holes in the West Pipe totalling 11,583 ft (3,530 m).

In 1982, Amselco Exploration Inc., a wholly owned subsidiary of British Petroleum Company, leased the property from Draco and advanced the project through extensive drilling and feasibility-level studies between 1982 and 1984. Amselco completed three phases of reverse circulation drilling between February 1983 and September 1984, totalling 163 RC holes for 95,463 ft (29,097 m). Amselco also completed six diamond drill holes totalling 3,738 ft (1,139 m) for geological, engineering, and metallurgical investigations. Amselco initiated permitting in 1983, culminating in approval of a Final EIR and EIS in July 1985. This work delineated mineable reserves based on a 0.026 oz/st cut-off of 12,146,000 short tons at 0.064 oz/st gold, as reported by Amselco (December 1984).

In October 1985, Grants Patch Mining Limited and Regent Mining Limited acquired Amselco's interest and entered an agreement in December 1985 to purchase Draco's remaining interest. The Draco transaction closed in June 1986, consolidating ownership through a US subsidiary of Grants Patch and Regent. In October 1986, Grants Patch and Regent sold Colosseum California Inc. to Dallhold Resources due to inability to raise development capital. The property was conveyed to Colosseum Gold, Inc., an indirectly owned US subsidiary of Dallhold, for mine development and operation.

Dallhold became part of Bond International Gold, controlled by Alan Bond. Royal Resources acquired a 25 percent interest in November 1986. Construction of a 3,400 short ton per day carbon-in-pulp cyanide plant commenced in November 1986 and was completed in September 1987 (McClure and Schull, 1988). Pre-production mining commenced in May 1987. Mining infrastructure was developed across a combination of unpatented federal land administered by the Bureau of Land Management and patented claims in the South Pit area. Operations were conducted from two open pits, the South Pit and the North Pit.

Pre-production mining operations commenced in May 1987. Mineable reserves were estimated to be 9,560,831Mt (10,539,000St), with an average grade of 2.1 g/t (0.062oz/st) (Beatty, 1989a and 1989b). The mining facilities occupied 284 acres with another 3,316 acres held as private land and unpatented mining claims. Mining was conducted in two open pits, the South Pit, and the North Pit. Most of the mining facility occupies unpatented Federal land under the jurisdiction of the Bureau of Land Management (BLM); two patented claims are in the South Pit area.

Figure 8 Colosseum Gold Processing Plant in 1988

Between 1988 and 1989, LAC Minerals Ltd acquired control of the mine through acquisition of Bond International Gold assets. LAC's subsidiary operated the mine until mining ceased on 10 July 1992. Milling continued through May 1993, processing stockpiles and remaining material. Dateline reports total historical production of approximately 344,000 ounces of gold during the 1988 to 1993 operating period. Gold prices at closure were reported around the mid US$300s per ounce. Barrick acquired LAC in 1994, after which the site remained in closure and reclamation under applicable federal and state requirements.

5.2 Historical Process Description

The historic process plant, since disassembled and removed, consisted of a 3,400 short ton per day carbon-in-pulp cyanide plant. The plant was completed in September 1987 and operated through to May 1993, following the cessation of mining in July 1992.

5.3 Dateline Resources Exploration and Development 2021-2026

Dateline Resources entered the ownership chain in 2021. On 15 March 2021, Dateline announced a binding agreement with LAC Minerals (USA) LLC, a wholly owned subsidiary of Barrick, to acquire 100 percent of the Colosseum Gold Mine and associated permits. Completion of the acquisition was announced on 27 October 2021. Dateline reported replacement of required reclamation bonds totalling US$770,000 and compilation of historical drilling and assay records into a modern database, including 386 drillholes and 35,352 assays.

Following acquisition, Dateline progressed technical studies and project enhancement work intended to support redevelopment. A Scoping Study was released on 23 October 2024. It modelled mining of 16.6 million tonnes of ore and 56.8 million tonnes of waste over an initial 8.4 year mine life, for production of approximately 635,000 ounces of gold. The study used a gold price assumption of US$2,200 per ounce.

Dateline commenced a Bankable Feasibility Study on 22 April 2025 following completion of a Project Selection Stage and appointment of lead advisors, including AMDAD for mining and GR Engineering Services (GRES) for processing plant and infrastructure in August 2025. In May 2025, Dateline released an updated Scoping Study applying a revised gold price assumption of $2,900/oz, while maintaining the study's scoping-level confidence and associated cautionary statements. The project generated an NPV (@6.5%) of $550 million and an IRR of 61%.

During 2025, Dateline reported a material increase in the gold price relative to the October 2024 study basis, including references to spot prices exceeding US$5,000 per ounce during the BFS period. Dateline also reported technical progress on BFS inputs and design work.

In parallel with gold development, Dateline advanced rare earth (REE) exploration across the Colosseum tenure based on proximity and interpreted geological continuity with the Mountain Pass district. Dateline reported extensive surface mapping and large-scale rock chip and soil sampling across felsite, fenite, and breccia units on the project area.

In July 2025, Dateline reported magneto-telluric survey results identifying a high-resistivity feature beneath mapped REE-bearing fenite and trachyte dykes and linked this to an exploration model involving carbonatite or alkalic intrusive bodies adjacent to the known gold breccia pipes.

In December 2025, Dateline contracted KLM Geosciences to complete an induced polarisation survey across the claim boundary, comprising five east to west lines of approximately 2.3 to 2.8 km using a 100 m dipole-dipole configuration. These datasets are intended to support drill targeting for both gold extensions and REE-related targets within the Colosseum system.

In January 2026, Dateline contracted MWH Geo-Surveys International Inc. to undertake ground gravity and UAV magnetics across the claim boundary comprising 6 lines run NW-SE spaced approximately 0.5 km apart.

6. GEOLOGICAL SETTING AND MINERALISATION

6.1 Regional Geological Setting

The Colosseum Gold Deposit is in eastern San Bernardino County, California, within the southern Basin and Range Province and along the eastern margin of the Mojave Desert. The deposit lies within a structurally complex corridor influenced by the Clark Mountain fault system, a long-lived crustal structure documented by the United States Geological Survey as a major control on magmatism, fluid migration, and mineralisation in the region.

6.2 Local Geology

The Colosseum deposit is hosted by a multiphase hydrothermal breccia complex emplaced into Proterozoic crystalline basement rocks. Mineralisation is centred on two principal breccia bodies that form a northeast to southwest trending elongate complex. These breccia bodies have historically been described as felsite or rhyolite felsite breccia pipes and intrude gneissic and granitoid basement lithologies. The breccias are characterised by intense fragmentation, variable clast populations, hydrothermal cementation, and pervasive sulphide development.

6.3 Mineralisation

Gold mineralisation occurs predominantly as disseminated auriferous pyrite within the breccia matrix and along clast margins. Higher grade zones correspond to areas of increased brecciation intensity, enhanced permeability, repeated hydrothermal fluid ingress, and high sulphide replacement of carbonaceous matrix/clasts within sedimentary breccia units. Dateline drilling has confirmed strong vertical continuity of mineralisation over several hundred metres, with gold grades remaining largely confined to the breccia footprint and structurally related zones.

Peripheral mineralisation occurs within the adjacent Proterozoic basement and within faulted remnants of Palaeozoic sedimentary units, including Tapeats Quartzite and Bright Angel Shale. These units formerly overlaid the breccia complex prior to extensional faulting and gravity-driven displacement. They are preserved as fault blocks and as clasts entrained within the breccia system. Gold grades within these sedimentary units are subordinate to those within the breccia bodies and display limited continuity, though they provide important constraints on the vertical extent and evolution of the hydrothermal system.

Deposit specific geochronological constraints place gold mineralisation in the early Late Cretaceous. Mineralisation post-dates Sevier compressional deformation and predates significant Basin and Range extensional tectonism. The timing is consistent with shallow intrusive activity associated with the northeastern outliers of the Teutonia Batholith. Regional geochronology and structural relationships support a genetic link between breccia formation, hydrothermal fluid circulation, and calc-alkaline magmatism during this period.

Gold is present primarily as sub-microscopic particles closely associated with pyrite. Petrographic studies described in historical technical literature identify gold occurring along pyrite grain boundaries, within microfractures, and as inclusions adjacent to sulphide growth zones. Minor free gold has been reported, with limited silver alloying. Secondary occurrences include isolated gold particles within quartz and gangue minerals, though spatial association with pyrite remains consistent across all observed settings.

The Colosseum deposit is best described as a hydrothermal breccia pipe gold system formed under transitional epithermal to mesothermal conditions. Upper structural levels preserve features consistent with epithermal environments, including open-space brecciation and low-temperature alteration assemblages. Deeper levels show increased sulphide abundance, reduced vein textures, and pressure temperature conditions consistent with mesothermal regimes. Earlier interpretations of replacement-style mineralisation reflect the presence of sedimentary clasts containing pre-existing sulphides, now interpreted as material incorporated during breccia emplacement rather than evidence of a separate mineralising event.

Oxidation is limited to near-surface zones, with primary sulphide mineralisation dominating the system at depth. This geological framework underpins the current Mineral Resource estimate and provides the basis for mine design, scheduling, and economic evaluation presented elsewhere in this Feasibility Study.

7. EXPLORATION

7.1 Exploration database and historical drilling

As of December 13, 2025, Colosseum Rare Metals Inc. completed 7,713.2 metres of drilling with 8 diamond core holes and 51 Reverse Circulation drillholes.

As per this report, the Colosseum gold Mineral Resource Estimate (MRE) reported under JORC 2012 totals 44.50 Mt at 0.76 g/t Au for 1,080 koz at a 0.20 g/t Au cut-off. This MRE is split into 17.5 Mt Measured at 1.00 g/t (561 koz), 14.6 Mt Indicated at 0.62 g/t (294 koz), and 12.4 Mt Inferred at 0.57 g/t (230 koz).

The Colosseum mineral resource database is underpinned by extensive historical drilling, completed between 1972 and 1991 by prior operators, supplemented by Dateline drilling and modern verification work completed after Dateline's acquisition of the project in 2021. Dateline's initial work program focused on reconstructing and validating the historical drillhole dataset in modern 3D software, confirming breccia unit geometry and the distribution of higher-grade mineralisation within the breccia pipe complex.

Historical drilling intersects the West (South) breccia pipe to substantial depth. Dateline reported historical drilling continuing to intersect the West (South) pipe at roughly 1,000 m vertical depth below the starting surface, reinforcing the depth continuity potential of the breccia pipe system beyond the historical open pit and the earlier resource shell.

7.2 Dateline drilling and gold exploration since 2021

7.2.1 2022 diamond drilling, database validation and down-plunge step-out

Dateline's first drilling after acquisition comprised five diamond drill holes completed in 2022 (CM22-01 to CM22-05) for a reported total of 605.38 m.

This programme had two core objectives:

- Validate the historical drilling database and lithological interpretation.
- Test mineralisation below the historical resource model and below the previously modelled mineralized envelope.

Dateline reported the drilling confirmed the lithological interpretation of historic drill data and assessed both densely drilled areas and the potential below the historic resource model. Reported highlights include 10.67 m at 13.71 g/t Au from 18.29 m in CM22-04 and 19.81 m at 5.19 g/t Au from 79.24 m in CM22-05.

Dateline further reported that two of these five holes were drilled approximately 100 m below the historical mineral resource shell and intersected high gold values, including CM22-05 intersecting over 100 m averaging 4.16 g/t Au.

7.2.2 2023 diamond drilling, extensional drilling and down-plunge targeting

In 2023, Dateline advanced extensional drilling to increase drill density in lower density portions of the higher-grade domain and to test below the then-current Mineral Resource envelope.

Dateline reported, for the then-current MRE framework, the existing open pit elevation at 1,621 mRL, the MRE extending to 1,493 mRL, and the 2023 program targeting down to 1,417 mRL, described as approximately 75 m below the base of the MRE.

Dateline described the 2023 program as eight diamond drillholes targeting areas of high-grade sedimentary breccia within the mineral resource model, with the targeted zone having low drill density and extending below the existing model.

Reported 2023 drilling results include 81.35 m at 2.57 g/t Au (including 36 m at 3.97 g/t Au) in CM23-11a, drilled inside and below the Mineral Resource envelope.

Dateline also reported ongoing drilling in CM23-09 remaining within breccia at over 306 m downhole, supporting an interpretation of a wider breccia pipe geometry than previously modelled.

7.2.3 2024 infill and extensional drilling, and the June 2024 MRE update

In early 2024 Dateline reported further diamond drilling results consistent with continuity of the mineralized system and expansion of the interpreted higher grade breccia unit geometry. Dateline reported 70.1 m at 6.53 g/t Au (including 25.9 m at 15.31 g/t Au) in CM23-14 and described follow-up drill testing targeting lateral and down-dip extents of the higher-grade zones.

Dateline reported additional drilling results in 2024 demonstrating gold mineralisation outside the previously emphasized high grade sedimentary breccia, including significant mineralisation within the felsite breccia unit. Reported highlights include 88 m at 4.18 g/t Au (including 22.8 m at 8.17 g/t Au) in CM24-16 and 19.8 m at 4.19 g/t Au in CM24-15, with Dateline noting the felsite breccia as the most voluminous unit at Colosseum and a key driver for potential mineralized volume expansion.

Dateline also introduced reverse circulation drilling in 2024, with the RC program focused on infill drilling and testing margins of mineralisation. Reported RC results include 74.7 m at 4.27 g/t Au from 77.72 m in RC24-008 and 25.9 m at 1.91 g/t Au from 57.91 m in RC24-003.

For the June 2024 MRE update, Dateline reported engagement of H&S Consultants Pty Ltd to update the MRE and stated the updated MRE would include Dateline diamond drilling completed since acquisition, with certain RC results received after the MRE cut-off date flagged for later incorporation.

7.2.4 2025 BFS Inferred Resource Upgrade and Exploration Drilling Programs

Dateline's BFS Inferred Resource Upgrade Program RC and core drilling program returned results of 132.58 m @ 0.94 g/t Au in RC25-004, 117.34 m @ 1.01 g/t Au in RC25-020, 62.48 m @ 2.52 g/t Au in RC25-002, 65.53 m @ 1.21 g/t Au in RC25-001, and 70.10 m @ 0.80 g/t Au in RC25-005 from inside both the North and South Pits.

Dateline's reported drilling results in 2025 demonstrated gold mineralisation outside the previously defined North Pit resource shape. The drilling reached depths up to approximately 300 m without reaching the Precambrian Granite contact. This indicates a potential extension to the mineralisation to the northeast of the North Pit. Reported highlights include 295.64 m @ 1.04 g/t Au in RC25-035, 297.17 m @ 0.68 g/t Au in RC25-034, 205.73 m @ 0.88 g/t Au in RC25-039, and 149.65 m @ 1.39 g/t Au (including a high-grade zone of 55.2 m @ 2.83 g/t Au) in CM25-41. Dateline also reported that the mineralisation remained open at depth.

For the June 2024 MRE update, Dateline reported engagement of H&S Consultants Pty Ltd to update the MRE and stated the updated MRE would include Dateline diamond drilling completed since acquisition, with certain RC results received after the MRE cut-off date flagged for later incorporation.

7.3 Geophysical surveys, target generation, and implications for gold exploration

7.3.1 Geophysics-led targeting rationale

Gold mineralisation at Colosseum is hosted within a breccia pipe complex. Dateline's post-acquisition exploration approach progressively integrated drilling, geologic modelling, and geophysics to define and rank additional breccia pipe targets beyond the two historically mined pipes and beyond the existing gold MRE footprint.

The Colosseum system is a large, zoned breccia pipe complex where the mined pipes represent exposed upper portions of a broader system, with additional pipe structures preserved nearby.

7.3.2 Magneto-telluric (MT) survey, gravity integration, and definition of new gold targets

Dateline reported completion of a comprehensive MT survey in June 2025. The program deployed 167 MT stations on 14 parallel east-west lines spaced 200 m apart, with readings collected at approximately 150 m intervals, generating a 3D resistivity model to depths of several kilometres for subsurface interpretation.

Dateline's interpretation framework links the known gold-bearing breccia pipes with coincident gravity-low (low density) and high conductivity responses and then applied this signature to define new targets in surrounding ground. Dateline reported that the 3D MT inversion showed the known breccia pipes remain open at depth for at least an additional 300 m below the deepest historical drilling, supporting a down-plunge mineralized system continuation concept.

Dateline reported that the MT inversion results were integrated with gravity and mapping datasets to generate a revised set of gold targets, identifying six high priority breccia pipe targets within 1.5 km of the existing MRE, with four of six targets having comparable or larger areal dimensions than the geophysical response over the existing mineral resource.

7.3.3 *Induced polarization survey, sulphide-associated targeting, and 2026 refinement*

Dateline reported completion of an induced polarization (IP) survey at Colosseum with five east-west lines approximately 2.3 to 2.8 km long, using a 100 m electrode configuration.

The IP survey is positioned as a refinement tool to assist drill targeting by delineating chargeability and related conductivity anomalies, complementing MT resistivity and gravity-derived target definitions and improving confidence in the location of sulphide-associated mineralized envelopes within or adjacent to breccia pipe structures.

7.3.4 *Gold drilling and target testing from late 2025 into 2026*

Dateline reported commencement in early October 2025 of an approximately 10,000 m drilling campaign combining BFS drilling and broader exploration drilling targeting new gold and REE discoveries generated from gravity, MT, and geochemical datasets.

Dateline described BFS drilling inputs including infill and extensional holes around the existing 1.1 Moz Mineral Resource, geotechnical drilling for pit wall stability, and RC drilling of historical low-grade stockpiles.

Dateline also reported wide and open gold intercepts from drilling, including intercepts outside the existing mineral resource envelope. Reported examples include 295.64 m at 1.04 g/t Au from surface in RC25-035, 300.21 m at 0.66 g/t Au from surface in RC25-036, and 67.36 m at 1.01 g/t Au from surface in CM25-18.

These broad intercepts support a working interpretation of extensive mineralized halos and potential for mineralized continuity beyond the current mineral resource shell, subject to geologic domain validation, grade continuity analysis, and incorporation into future MRE updates under the applicable reporting code and estimation methodology.

7.3.5 *Data handling, use of drilling in modelling, and mined-out considerations*

Dateline's approach to reconciling historic drilling with modern modelling, includes accounting for mined-out volumes and the post-mining topographic surface when using historical composites and block model estimation, and using targeted confirmation drilling to validate historic grades, lithologic contacts, and the geometry of higher-grade breccia units. Dateline's post-acquisition drilling was explicitly used to confirm the presence of the higher-grade breccia unit and to validate continuation of mineralisation below prior interpreted limits.

Where holes or intervals fall within mined-out areas, the appropriate treatment in estimation workflows includes mined depletion solids, topographic constraints, and limitations on the use of mined-out intervals for grade estimation, while retaining relevant data for variography and geologic interpretation where appropriate.

8. SAMPLE PREPARATION, ANALYSES, AND SECURITY

8.1 Overview

Sampling, preparation, assay, and data security at Colosseum span multiple campaigns from the 1970s through Dateline's post-acquisition drilling. Historic programs used conventional industry methods. Dateline campaigns adopted modern chain-of-custody, commercial laboratory workflows, and routine QA/QC insertions, suitable for Mineral Resource estimation and feasibility-level study inputs.

8.2 Sample collection and handling

8.2.1 Core drilling

Core is cut along the long axis. Half-core is submitted for assay. The remaining half is retained in labelled core boxes for future reference and verification.

Sample interval boundaries follow geological controls including lithology, alteration and mineralisation observed during logging.

8.2.2 Reverse circulation (RC) drilling

RC sampling uses a cone splitter to collect a representative sub-sample from each interval. All sample weights are recorded to monitor recovery. Certified Resource Materials and verified blank materials are inserted in rotation every 10 samples. Field duplicates are collected at approximately 1 in every 40 samples but prioritized to mineralized zones.

8.2.3 Sample security and chain of custody

Samples are continuously under the custody of site personnel until they are dispatched and delivered to the laboratory by a licensed transport provider. Dateline also reports strict chain-of-custody protocols for drill samples.

8.3 Laboratory selection and analytical workflow

8.3.1 Historic drilling and mine-era campaigns

Historic sample analysis is reported as fire assay using a 30g charge with a lead collector and AAS finish. The method was appropriate and reports no unusual or questionable gold assaying methods across those campaigns, where documentation is available. Submittal sheets and assay certificates exist for all recent drilling and most of the historic drilling campaigns, supporting traceability of results used in modelling.

8.3.2 *Dateline drilling campaigns*

Assay work was completed at ALS Global or Paragon Geochemical in Reno, Nevada, with “umpire” checks sent to Bureau Veritas in reported programs.

Preparation steps for drill samples include drying, weighing, crushing to a target of 70% passing 2mm, and riffle splitting off a 250g sub-sample, followed by ring-and-puck pulverization to produce 85 percent passing 75 microns. Pulps are blended and packaged, then a 30g charge is taken for fire assay. Gold and silver are analysed using industry standard fire assay methods. Over-limit gold is re-analysed using gravimetric finish when greater than 10ppm Au.

8.4 Quality Assurance and Quality Control

8.4.1 *Historic QA/QC context*

Dateline’s Mineral Resource disclosure states QA/QC programs across drilling campaigns demonstrate laboratory performance and sample preparation suitable for Mineral Resource definition, with sufficient accuracy and precision.

Dateline also reports RC field duplicates collected at a ratio of approximately 1 in 20 samples in the context of prior resource work, with results indicating acceptable precision and accuracy for gold. Certified reference materials are reported as indicating no issues with accuracy in historical datasets where standards were used and documented.

8.4.2 *Dateline QA/QC design and execution*

For reported Dateline drilling, routine QA/QC samples were inserted into sample runs at a disclosed rate of 10 percent, comprising certified reference materials supplied by CDN Resource Laboratories Ltd and verified blank granitic material.

Dateline also discloses additional QA/QC insertion practices in program-level reporting, including duplicates placed in mineralized zones and regular insertion of QA/QC samples into RC batches. Dateline reports that QA/QC procedures have been developed and reviewed by named technical personnel for the 2025 program and that the QA/QC program is designed to detect errors.

8.4.3 *Database compilation*

All historical and modern information has been consolidated into a single drillhole database to support Mineral Resource estimation and study work. This included the digitisation of historical mine records and direct loading of modern assay certificates into the database structure.

8.4.4 *Creation and verification procedures*

GeoGRAFX Consulting LLC created working versions of collar, survey, lithology, and assay tables using original digital analytical certificates where available and checked collar and downhole survey records against original digital records for historical drilling.

Verification procedures described by Dateline include conversion to structured templates with embedded checking routines and auditing of the digital database against original historical paper records for collar coordinates, hole orientations, and analytical information.

Dateline reporting also notes review of historical work by GeoGRAFX in 2022 and states that earlier third-party work is non-JORC compliant, with Dateline's validation intended to support compliant reporting moving forward.

8.5 Competent Person level conclusion

Based on the disclosed preparation methods, laboratory selection, chain-of-custody, QA/QC insertion, and the described database verification workflow, the sample preparation, analytical methods, and data security practices reported by Dateline support use of the drillhole assay database for Mineral Resource estimation and feasibility study inputs, subject to the project-wide data verification and reconciliation steps described elsewhere in the report.

9. DATA VERIFICATION

9.1 Drill Core and Geologic Logs

The Senior Geologist (Mr Graham Craig) and Competent Person for Dateline Resources is based on site; no site visit was performed by the Mineral Resource estimation CP (Mr Simon Tear of H&S Consultants Pty Ltd). Dateline Resources' Senior Geologist reviewed the comprehensive drill hole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs. All data was readily available for inspection and verification. In addition, most of the subsequent companies or their consultants that have examined the Project have completed checks of the data and assay results. The author reviewed drill core, drill core logs and assay certificates and found a minimal number of errors (i.e. mislabelled intervals, number transpositions), which were corrected prior to development of the Mineral Resource estimation. It is the opinion of the CP responsible for this section that the databases and associated data were of a high quality in nature and valid for use in the Mineral Resource estimation.

The competent person responsible for this section found no significant discrepancies with the existing drill hole geologic logs and is satisfied that the geologic logging, as provided for the development of the three-dimensional geologic models, fairly represents both the geologic and mineralogic conditions of each of the deposits that comprise the Project.

9.2 Topography, Elevation and Vegetation

The topographic map of the Project area, created by drone photogrammetry, was verified using co-ordinates provided by Lochsa Engineering's survey data and delivered electronically in a Surpac compatible format. The native co-ordinate system of the topography is UTM Zone 11N, WGS84, and for the Mineral Resource estimation and as the Project goes forward UTM Zone 11N, WGS84 will be the co-ordinate system used.

It is the opinion of the competent person for this section that the current topographic map is accurate and accurately represents the topography of the Project area. In addition, it is suitable for the development of the geologic models, Mineral Resource estimates, and Mineral Reserve estimates.

9.3 Analytical Data Verification

For the 2025 Mineral Resource estimates, a detailed data verification procedure was undertaken by Dateline Resources, which focused on the 2022-2025 drill programs. This verification was accomplished by reviewing the assay database and comparing results with laboratory certificates and field QA/QC samples.

To verify the accuracy of historical data, and individual lab accuracy, 5% of the samples from the 2022-2025 drill programs were selected in from the North and South pits to cover a range of grades, particularly the high grades, for check analyses at a second laboratory. The remaining pulverized material from the original ALS Global sub-sampling was sent to Bureau Veritas where it was split, homogenized, and analysed for gold using a fire assay method (with a 30g charge) with an AAS finish along with a gravimetric overlimit trigger for any assay greater than 10 ppm Au. This method is very similar to the ALS Global method.

9.4 Historical Data Verification - 2022 and Prior

A comprehensive independent verification was undertaken by an outside auditor (GeoGRAFX GIS Services, 2022) for the Mineral Resource Estimate (MRE) conducted in 2022. This study consisted of an extensive data compilation for the project. The author, and CP of the 2022 MRE, reviewed the historical project data, audited surface data, verified the historical drill hole database, reviewed historical QA/QC procedures, and performed a site visit in April 2022. The verification of the historical database consisted of geochemical surveys, geologic mapping, mining, production, metallurgy, drilling, lithology logging and assays of over 599 drill holes. The report states there were no unusual or questionable gold assaying methods used, and individual laboratory sample preparation procedures followed a standard analytical process that were considered appropriate for a Quality Assurance/Quality Control (QA/QC) program. There were no significant risks or uncertainties identified that could be identified by the author of the report that could be expected to affect the reliability of the information included in the 2022 Mineral Resource estimate.

9.5 2023-2025 Verification

As part of the 2022 to 2025 exploration programs, an exercise to verify both historical drilling and further define mineralisation extents for the updated Mineral Resource estimate was undertaken. This consisted of a review of historical drilling results previously signed off in the 2022 study and an extensive Reverse Circulation (RC) drill program along with three diamond core twin holes. These twin holes were for the purposes of confirming the accuracy and reliability of results from the RC drill program. This program also consisted of three holes being analysed for four-acid trace multi-element digestion to correlate with gold intercepts and previous multi-element work completed within the deposit.

The RC drilling was completed to industry standards by Major Drilling using a 5 1/4" face sampling bit. Bulk and two smaller samples were collected from the rotary cone splitter. One small sample was used as the primary sample for the drill run and the other was used as a duplicate, where applicable, or kept along with the bulk sample. QA/QC consisted of field duplicates (secondary sample bag collected from rotary cone splitter) and the insertion of Certified Reference Materials (CRMs or standards) and coarse blanks. Three diamond core twin holes were completed to provide a measure of representativity of the RC sampling along with collecting other information for geotechnical and mine planning purposes. To check the homogeneity of the laboratory sub-sample preparation (and the accuracy of the analysis), laboratory duplicates (2nd pulps) were also inserted into the sample suite. Approximately 5% of the 2022-2025 samples were sent to a second laboratory for check analyses.

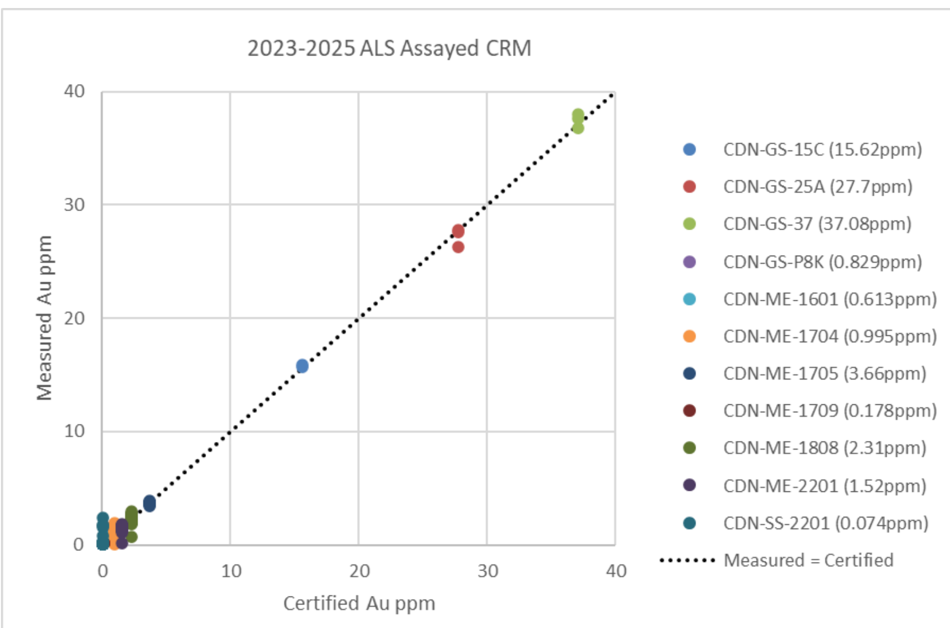
9.6 Quality Control Samples

Certified reference materials and blanks were inserted into the sample sequence at a minimum ratio of 1 in 20 for each type. Within suspected mineralised zones, duplicates were added to verify mineralisation the sub-sample preparation and analytical results. Certified Reference Materials were sourced from CDN Resource Laboratories with gold ranges from 0.074 to 37.08 ppm. Coarse blank material was sourced from a granite quarry in Gunnison, CO that was previously verified as null from multiple independent laboratories.

The coarse blank material was used to monitor cross-contamination within the laboratories from the crushing and pulverizing stages during sample preparation. Following any suspected high-grade zones, additional coarse blank samples were added to check for contamination.

9.6.1 Certified Reference Materials (CRM)

A total of 422 CRMs were used in the 2023-2025 drilling program (Figure 9). The CRMs were chosen for a range of grades and were based on matrix compatibility with the host rocks at Colosseum. The assayed values of the CRMs compared well with their certified values. Including all CRMs assayed, less than 24% of CRMs assayed outside 3-standard deviations from their certified values. One CRM identified (CDN-SS-2201) had majority of the errors and was therefore taken out of rotation and replaced with a new low-grade CRM. With CDN-SS-2201 CRM removed, the failures drop to less than 9%. A re-assay of these failures and surrounding samples was performed to verify no laboratory mistakes were present in the samples surrounding the failing CRM.

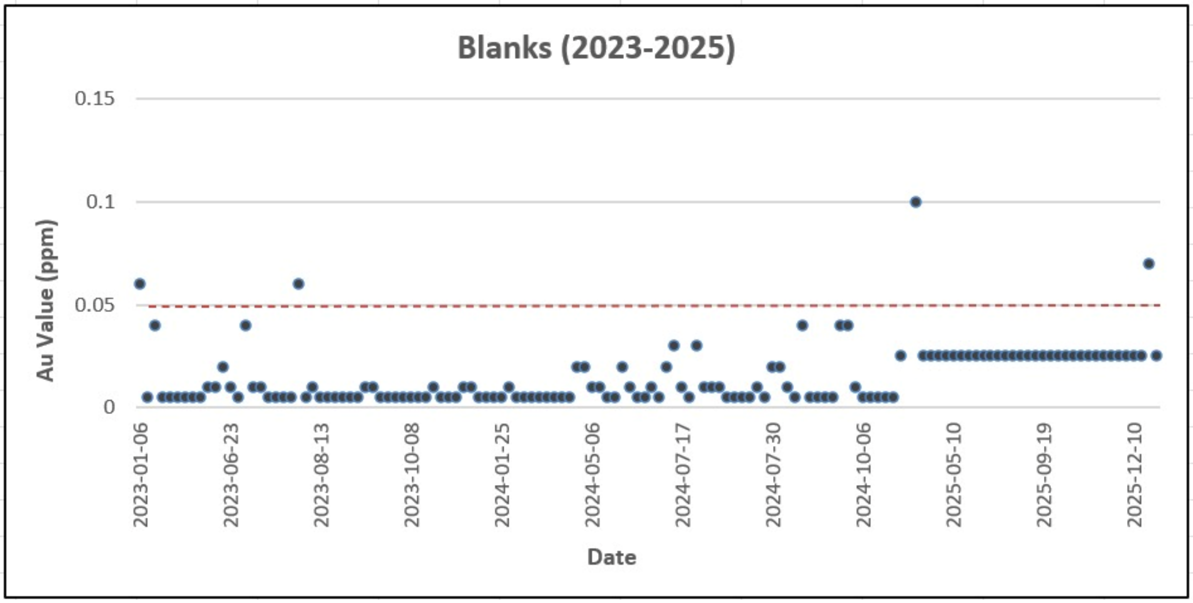


Source: Prepared by Dateline Resources, 2026

Figure 9 ALS Assayed CRM versus CDN Certified Values Au ppm

9.6.2 Blanks

386 blanks, shown in the figure below, were used between 2023-2025. These blanks consisted of both coarse gravel blanks and pulp blanks (CDN-BL-10) that were used on a rotational basis as a means of checking laboratory preparation and analytical processes. Blank failures that were suspected of contamination were sent in for re-assay. The procedures and checks in place were reviewed and found to be acceptable within industry standards (**Error! Reference source not found.**). Note: analytical methodology and lower detection limit changed for samples for 2025 samples.

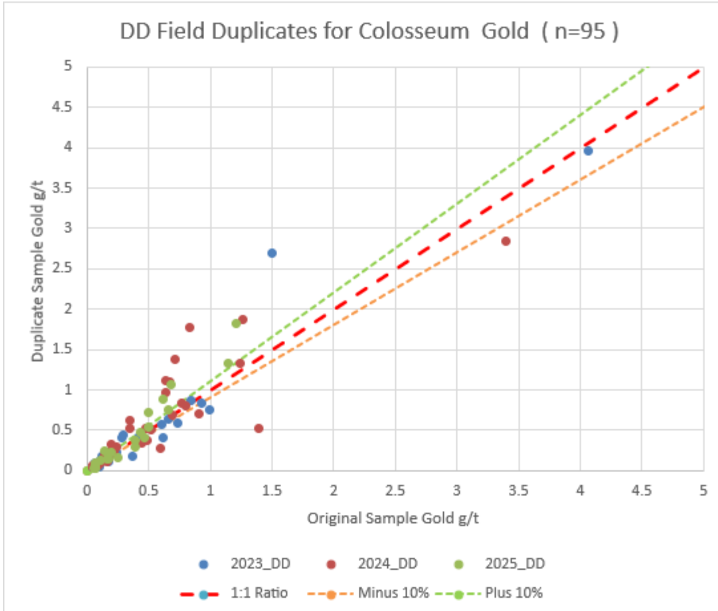


Source: Prepared by Dateline Resources, 2026

Figure 10 Blanks Analyses, 2023-2025

9.6.3 Field Duplicates

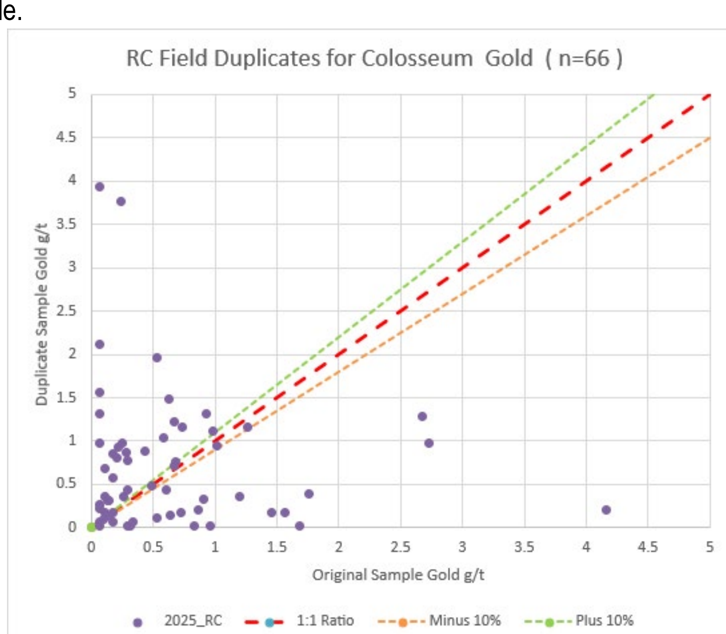
257 Field Duplicates were collected between 2023-2025. Core duplicates were collected in mineralised zones at a ratio of approximately 1 in 30. RC duplicates were collected in mineralised zones at a ratio of approximately 1 in 40. RC duplicates were created using the second sample bag collected from the rig-mounted rotary cone splitter and relabelled with a unique identifier from the parent sample. Field Duplicates were overall comparable, but variability increased in low-grade parent samples due to the higher lower detection limit used in the fire-assay methodology utilised by ALS. The duplicates in the diamond drilling indicate moderate repeatability with a small but not significant bias towards the duplicate sample (Figure 11). Note in Figure 11, assays were cut to 5 g/t Au to better view the data, which has affected the removal of several samples.



Source: Prepared by Dateline Resources, 2026

Figure 11 Diamond core drilling duplicate sample plot

The RC program duplicates showed poor repeatability likely due to cone splitter sampling, mislabelled samples, or possible sample mix-up at the field site or laboratory. The RC drilling however does not show any clear bias between the parent and duplicate samples (Figure 12). The secondary lab checks show no issues regarding accuracy and therefore, poor precision therefore results are considered accurate and acceptable.

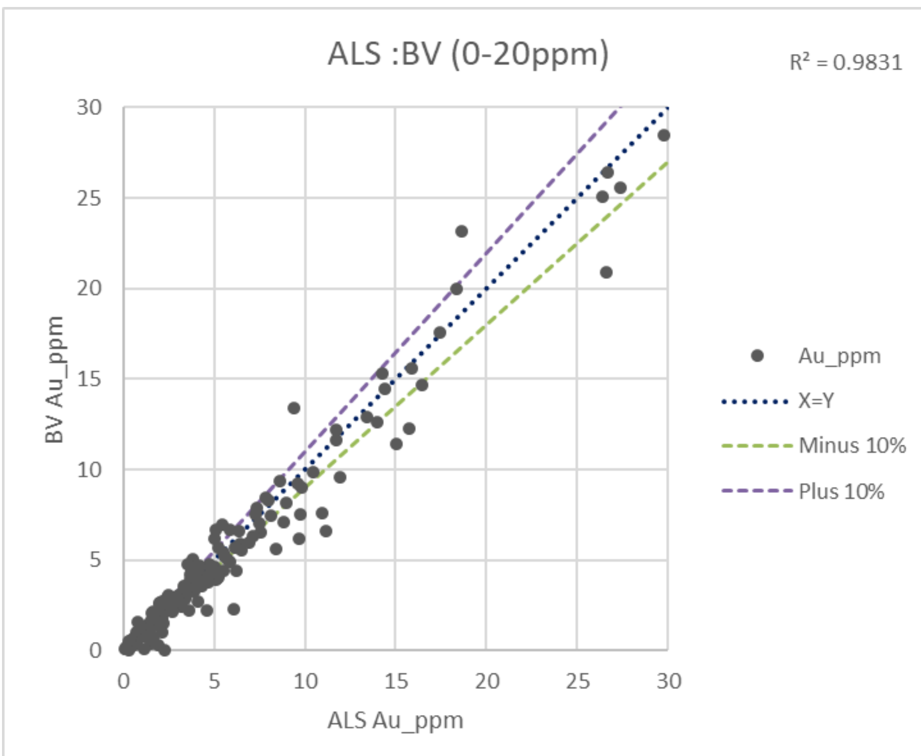


Source: Prepared by Dateline Resources, 2026

Figure 12 RC field duplicate comparison plot

9.7 Independent Assay Verification Results

176 pulp residue samples covering a range of gold grades from 2023-2025 were sent for check analysis to Bureau Veritas (BV) in March 2026. The samples covered a range of grades with some emphasis on the higher grades. The BV method was similar to ALS Global (fire assay with a 30g charge and an AAS finish). There was a gravimetric overlimit trigger for any assay greater than 10 ppm Au. Standards and blanks were included in the sample suite. Results from the second laboratory check assays by Bureau Veritas indicate a small positive bias with the original ALS assays but is not considered significant. The variance is greatly reduced with higher grade samples. 95% of the data falls within the limits of agreement otherwise where variability is noted in the higher-grade samples (>10ppm), while the lower-grade values are reasonably interchangeable in terms of the method used. These results indicate good correlation where over 90% of the gold grades are under 5 ppm within the deposit. There are a few minor outlier samples, primarily in the lower ppm samples, but there is no evidence of a major bias present between the two lab comparisons (Figure 13).



Source: Prepared by Dateline Resources, 2026

Figure 13 Check assay results ALS (X) BV (Y) with 10% lines

Results from the second laboratory checks indicate the two laboratories and their analytical methods (ALS vs BV) are reasonably consistent with no significant bias.

9.8 Competent Person Opinion

The Competent Person has reviewed all data involved in this report including collar surveys, downhole surveys, logging and sampling, and analytical procedures including QA/QC procedures and data; which concludes that there are no significant issues with sample locations, sample recoveries, sample representativity and analytical outcomes. The data is suitable for Mineral Resource estimation.

10. MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 Introduction

The section reports on the historical and current test work completed to develop the understanding of the metallurgical characteristics of the Colosseum North and South deposits. The understanding contributes to the design of a technically effective and economically efficient gold recovery operation.

10.2 Historical Plant

The previous operation at Colosseum operated from late1987 through to closure in mid-1993, during which time it processed 5.82 mt with a processing availability of 89.7%. Feed grades varied from 1.23 g/t Au to 2.6 g/t Au with an operations average of 1.97 g/t Au. A total of 344,691 oz Au was recovered with an overall recovery of 90.98% Au.

The operation consisted of a conventional SAB circuit, which included a pebble crushing circuit (SABC) after subsequent upgrade. The leach circuit consisted of hybrid leach/ carbon in leach (CIL) using a 24-hour leach residence time. Gold recovery was completed using AARL elution circuit/ electrowinning/ smelt process for doŕe gold onsite. Tailings from the plant was detoxed using the INCO Air/SO_2 process to the required WAD CN target 0.5 ppm in the tailing's solution assay.

10.3 Historic Metallurgic Test Programs

The feasibility study level test work program was conducted in 1984 at Hazen Research Labs (Hazen) on behalf of Amselco Minerals Incorporated (Amselco). The metallurgical testing followed on from the previous Amselco pre-feasibility study test work which investigated heap leaching, bulk sulphide flotation followed by cyanidation of the flotation concentrate, and a whole of ore cyanidation as potential gold recovery methods. The author wasn't able to find the previous study; hence the following high-level summary is taken form the Amselco feasibility study report:

- Column leach testing produced good results on oxide ores with gold recoveries of up to 90%, but recoveries on sulphide ore were low at 53%.
- Bulk flotation with cyanidation of concentrate worked well on sulphide ores, recovering 85% to 90% of the gold, but was unsuccessful on oxide ores producing only 70% recovery.
- Whole ore cyanidation produced high recoveries on both sulphide and oxide ores. Gold leached rapidly from the ores, but excessive leach times caused high cyanide consumption and dissolution of deleterious impurity metals.

On the basis of the results the pre-feasibility study proposed heap leaching to process oxide ores and cyanidation milling to process sulphide ores.

Prior to the Feasibility test program Hazen conducted testing on “Adit Samples” which were reported in a November 4, 1983, Report HRI Project 5760 for Amselco. This test program included Column Leach Tests, Bottle Roll Cyanidation, Agitation Cyanidation and Sulphide Flotation Testing.

In addition, Hazen conducted tests that were reported in a Letter to Dallhold Resources Inc – Dec 3, 1987, in reference to HRI Project 6551 for the Minproc design.

10.3.1 Hazen HRI Project 5760 Report -November 1983

Details on the testing are reported in the GR DFS study, with summary data report only in this section.

Head assays from the three composites are shown in Table 6.

	Composite 1 Sulphide	Composite 2 Mixed	Composite 3 Oxide
Au , g/t (Au oz/ton)	7.30 (0.213)	12.75 (0.372)	1.33 (0.039)
Ag, g/t	13.71 (0.4)	14.06 (0.41)	1.37 (0.04)
Cu %	0.12	0.15	0.035
Total S%	1.8	7.2	7.0

Table 6 1983 Audit Sample Head Assay

10.3.1.1 Bottle Roll Cyanidation

Bottle roll cyanidation tests were conducted on both crushed ¼ inch (6.35 mm), minus 10 mesh (2 mm), and ground ore to 200 mesh (75 µm). Testing used variations in cyanide concentrations 0.25, 0.5 and 1 g NaCN/ litre. Tests were conducted over 48 hours.

- Minus 6.35 mm produced 45 to 58% extraction on sulphide and average 85% on the oxide material. Transitional material average 53%. At cyanide consumptions of 1.8 kg/t.
- Minus 2 mm produced 42 to 53% extraction on sulphide and average >98% on the oxide material.
- Ground to 75-micron leached material produced 79 to 86% extraction on sulphide and average 88 to 95% on the oxide material at 1 g/l NaCN leach concentration.

10.3.1.2 Agitation Cyanidation

Mechanically agitated tests were undertaken on the composites with pre-wash and pre-oxidation with air. The tests were run for 48-hour with ground material passing 200 mesh (75 µm). The washing was done by filtering the ground solids and completing three displacements using deionized water. Filter solids were then repulped to 50% solids and oxidised with air sparging for 18 hours before leaching at 1 g/l NaCN.

Test 1580	Comp	Grind % minus 200 mesh	Au % Ext	NaCN Consumption kg/t	$Ca(OH)_2$ Consumption kg/t	
31	1 (Sulphide)	97	89.4	4.05	1.85	Prewash and pre-oxidation
32	2 (Mixed)	95	93.9	4.60	4.10	Prewash and pre-oxidation
33	3 (Oxide)	97	94.5	1.50	2.20	Prewash and pre-oxidation
34	1 (Sulphide)	97	87.0	0.20	1.40	Cyanidation of Flotation Concentrates
35	2 (Mixed)	95	85.3	4.10	3.35	Cyanidation of Flotation Concentrates

Table 7 Agitation Tests

Comparing the prewash and pre-oxidation against the bottle roll tests at 1 g/l NaCN, showed a substantial reduction in cyanide consumption. Test 11 used 5.55 kg/t, against Test 31 consumption of 4.05 kg/t, the same applied for the transitional-mix ore with a reduction from Test 13 at 8.75 kg/t to Test 32 at 4.60 kg/t.

10.3.1.3 Flotation Tests

A series of sulphide flotation testing was conducted at varies grind sizes passing 200 mesh (75 µm). The sulphide composite achieved excellent rougher gold recoveries >97% Au. The final tails assays were less than 0.010 oz Au/ton (0.34 g Au/t). The flotation test data sheets indicated that most of the mass recovery is in the first 2 ½ minutes with a fast flotation kinetic response.

10.3.2 Hazen HRI Project 5790 –Colossuem Bottle Rolls – Jan 1984

Amselco provided additional samples to Hazen for bottle rolls, to assess the leach time and kinetics based on sampling at 4, 8, 24 and 48 hours. Details on the testing is reported in the GR DFS study report.

The data gathered was sufficient to show that the samples have a very fast leach rate. Sufficient savings in cyanide consumption can be made by limiting the overall leach time. The final 48-hour leach recovery is based on assayed residuals from the leach test. A leach kinetic curve for the average of the data set and the 85th percentile is illustrated in Figure 14.

- Leach extractions range from 77 to 94% Au
- Head grades varied significantly from 1.06 g/t up to 8.98 g/t.

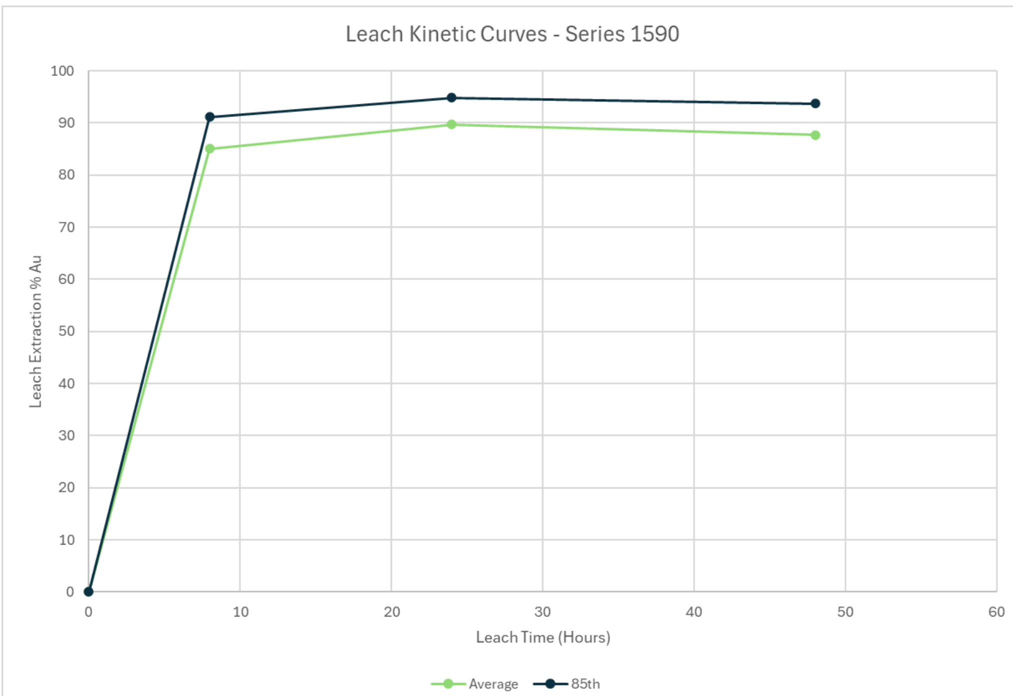


Figure 14 Series 1590 Kinetic Curve

10.3.3 Amselco – Feasibility Study 1984

The metallurgical test program undertaken at Hazen Research Laboratory included heap leach testing and bench scale cyanidation leach tests.

10.3.3.1 Samples

Hazen received approximately 5,805 kg of ore for testing from Amselco. The material represented rejects from previous drilling programs, new drill core from the ongoing program and a run of mine oxide ore sample.

The master composite represented a blend of the yearly composites for the associated years 1 through to 7. The master composite assayed 3.29 g Au/t. The head analysis is tabulated in Table 8.

Yearly Composites	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Fe %	S % (Total Sulphur)
1	1.65	6.51	0.016	0.037	0.12	4.06	0.37
2	4.46	1.03	0.016	0.008	0.11	3.86	1.60
3	3.39	0.69	0.140	0.011	0.13	3.43	0.87
4	2.19	0.34	0.021	0.022	0.12	5.23	2.64
5	3.29	1.71	0.015	0.036	0.16	3.75	1.57
6	7.03	2.06	0.044	0.02	0.11	4.37	3.09
7	2.88	3.09	0.017	0.052	0.05	5.36	3.95
8	2.57	3.43	0.010	0.113	0.12	1.25	0.25
9	2.54	2.74	0.011	0.066	0.11	1.32	0.21
10	1.51	2.74	0.008	0.088	0.07	1.06	0.11
11	1.75	3.09	0.008	0.064	0.07	1.27	0.28
Master	3.29	2.40	0.017	0.025	0.11	4.18	1.94

Table 8 1984 Amselco Feasibility - Key Samples

10.3.3.2 *Master Composite – Grind Size Determination*

The master composite was leached at five grind sizes ranging from 45 % to 67 % passing 75 µm screen size. The results indicated that within the grind size tested that the gold recovery was not grind sensitive, with the grind size having no significant effect on the gold metallurgy on the 24-hour leach basis. Results of the grind tests are tabulated in Table 9.

Grind Size % Passing 75 µm	% Gold Dissolution			Gold Tail	Solution	NaCN
	Grind	12 hr	24 hr	Au g/t	Au mg/l	kg/t
45	14.3	90.3	90.0	0.31	1.72	1.25
46	14.6	88.9	92.2	0.27	1.93	1.00
57	16.7	92.7	92.8	0.24	1.86	1.05
62	19.6	90.4	92.2	0.24	1.76	1.05
67	22.6	94.4	92.9	0.21	1.64	1.05

Table 9 Grind Size Determination

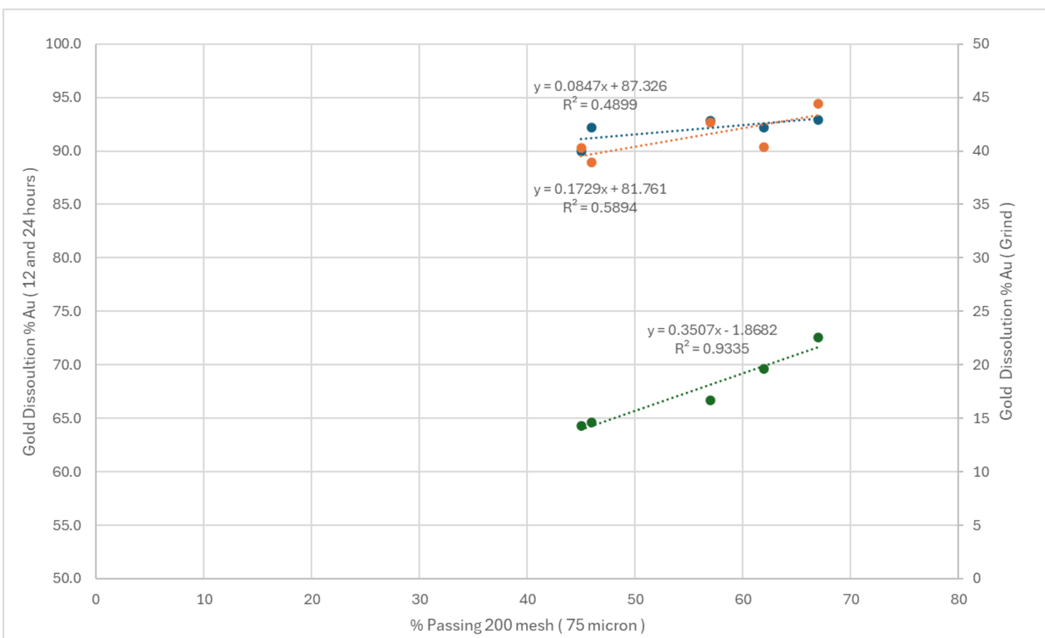


Figure 15 Grind Size vs Recovery (Grind, 12h, 24h)

The noted increase in gold dissolution during the grinding stage can be associated with longer grinding times required for the finer grind.

10.3.3.3 *Yearly and Master Cyanidation Leach*

The prepared yearly composites and the master composite were leached for 24 hours, followed by an 8-hour contact with activated carbon (CIP) circuit design. Test conditions were maintained as the same including the grind time, independent of ore hardness.

Results were summarised in the Amselco Feasibility Study but were missing from the copy of the test work report. The limited results from the FS summary are tabulated in Table 10.

Composite (Yearly)	Number of Test	% Au Recovery	NaCN consumption kg/t
1	6	91.7	1.10
2	6	87.0	0.90
3	6	93.2	0.55
4	6	93.5	0.75
5	6	88.8	0.75
6	6	93.2	0.85
7	6	92.6	0.60
8	4	92.3	0.60
9	4	92.1	0.50
10	4	90.6	0.45
11	4	89.6	0.40
Average	58	91.4	0.65
Master	16	91.5	0.80

Table 10 Yearly and Master (CIP) Recovery Results

The FS report (1984) noted that the cyanide consumption was lower than the previous (Pre-Feasibility) because the samples had been milled with lime. Results from the bottle rolls on the master composite indicated that cyanide consumption may be minimised by controlling the cyanide dosage. Stage additions of 0.15 kg/t to 0.2 kg/t results in gold recoveries of 92% and a CN- consumption 0.4 kg/t after 24 hours.

Additional testing for carbon equilibrium was conducted on the composites, using a 24-hour leach and a 24-hour CIP test. The results however are missing from the report, but the commentary with the report suggested that a barren liquor of 0.02 mg Au /L was possible.

10.3.3.4 Cyanidation of Drill Core – Bond Work Index

A series of cyanide bottle roll leach tests were conducted on 38 on WDDH-4 and 5 and six of the composites from EDDH-6. The ball mill used for grinding was calibrated to permit the calculation of relative Bond Work Index. A filtered set of data for grind size analysis was charted as Figure 16. This indicated a trend towards higher gold extractions with a finer grind P_{80}, it also showed scatter effect with some of drill core composites achieving high gold extractions at a coarser primary grind.

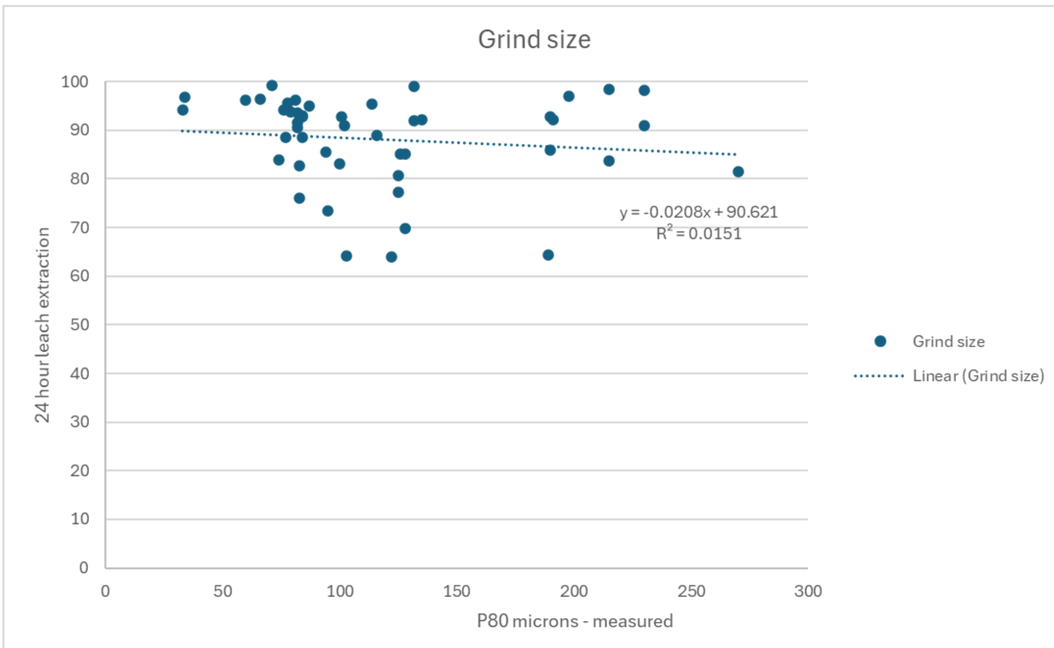


Figure 16 Grind vs 24-hour Au Extractions – Drill Core Composites

Most of the test data reflects high extractions >85% Au. The calculated bond data also indicated a significant variation in ore hardness over the three drill cores.

Bond WI recorded ranged from 13 to 22 kWh/t.

10.3.3.5 *Thickening Results*

Tailings samples from the continuous CIP circuit tests were used. Results indicated that slurry underflow density of 70% could be achieved. Unit area results ranged with around 2 ft^2/ton/day being achieved with good flocculation. The result indicates sizes of 90 to 100ft in diameter or 27 to 34m.

	Flocculant			% Solids		Unit Area		
Test 1653-	Type	Amount lb/t	Amount kg/t	Feed	Underflow	ft^2/ton/day	$t/m^2.h$	Comments
132	SF 1128	0.19	0.10	41.9	62.4	1.9	0.214	No rakes
133	SF 1201	0.18	0.09	41.0	63.0	1.9	0.214	No rakes
136	SF 1202	0.10	0.05	41.4	66.0	2.1	0.193	
134	SF 1202	0.16	0.08	40.2	61.7	1.5	0.271	No rakes
137	SF 1202	0.17	0.09	41.5	66.8	1.0	0.407	
135	SF 1204	0.10	0.05	40.8	63.4	1.9	0.214	No rakes
144	N 7871	0.44	0.22	39.1	62.9	4.6	0.088	
142	N 7872	0.44	0.22	39.2	68.6	2.3	0.177	
139	N 7873	0.11	0.06	39.7	68.3	2.6	0.156	
140	N 7874	0.22	0.11	39.5	65.3	2.3	0.177	
145	N 7877	0.05	0.03	40.9	70.7	6.5	0.063	
143	N 7877	0.11	0.06	40.1	68.7	2.8	0.145	
146	N 7877	0.21	0.11	40.7	69.3	1.7	0.239	

Table 11 Thickening Results

10.3.3.6 Cyanide Destruction - INCO

Six tests were conducted using sodium metabisulphite as the SO_2 reagent with 65.5% SO_2 equivalent. The tests were conducted in a Gallagher flotation cell using air with pH maintained at 8.4 to 8.8 using hydrated lime. Air was sparged at 1.6 lt/min per litre of pulp.

The feed solution to the detox contained:

Feed Solution mg/l					
Cl^-	CNO^-	$CN_{(total)}$	CN^-	SCN^-	CN^-
(1)	(2)	(3)	(4)	(5)	(6)
18.9	24	162	149	27	132

(8) Total Chlorides
(9) Cyanate
(10) Total Cyanide
(11) Cyanide amenable to chlorination
(12) Thiocyanate
(13) "Free" cyanide – selective ion electrode

Table 12 Detox Feed Solution conditions

Test No	Feed pulp	Treated Soln mg/l		kg SO_2 / kg Total Cyanide	Treatment	$Ca(OH)_2$	Copper Equivalent mg/l
1662-	% Solids	$CN_{(total)}$	CN^-		time	kg/t	
		(8)	(11)		hr		
5-1	36.8	106	74	1.13	0.5	0	0
5-3	37	78	74	2.36	0.5	0.35	0
5-5	36.5	76.5	66	1.18	2	0	0
5-7	37.7	1.2	1.8	6.62	1	1.9	60
5-9	38	0.45	1.8	5.35	1	1.65	60
5-11	40.4	0.85	2.9	4.67	1	1.05	60

Table 13 Detox Test Results.

Test 5-7 to 5-9 achieved >2ppm free CN- using 5.35 to 6.62 kg SO_2 to kg Total CN, while the hydrated lime required was 1.66 to 1.9 kg/t of ore. The result indicates that INCO SO_2 can be used for cyanide detoxification on the Colosseum ore.

10.3.4 Hazen HRI Project 6551- December 1987

A series of grind and leach tests including carbon were conducted for Dallhold Resources Inc as part of the design requirements for Minproc. The test work focused on the conditions required to achieve >90% gold extraction from sulphide ore, ideally with conditions close to the Minproc design criteria (1987). Plant design conditions listed by the lab at the time were for a P_{90} of 150µm, (PDC - P_{80} 130µm), with 24 hours overall – 9 hours direct leach followed by 15 hours of CIL at 53% solids and using up to 20g/l of carbon. Cyanide concentration was less than 400 ppm average NaCN in the leach.

Test results for the baseline conditions are tabulated in Table 14. These ranged from very good at 92% to very poor at 36%. While the grind was almost 100% passing the 100 mesh (150 µm) screen size, the low level of NaCN maintained at the ~400 ppm average was the most likely effect on the recovery. Subsequent testing targeted a much finer grind size target.

Grind size test analysis was conducted and tabulated in Table 16. The grind analysis shows an increasing gold recovery with reducing grind size (Figure 17), the cyanide concentrations were also increased and ranges from 575 to 910 ppm NaCN against a head grade range of 8.98 to 12.96 g Au/t.

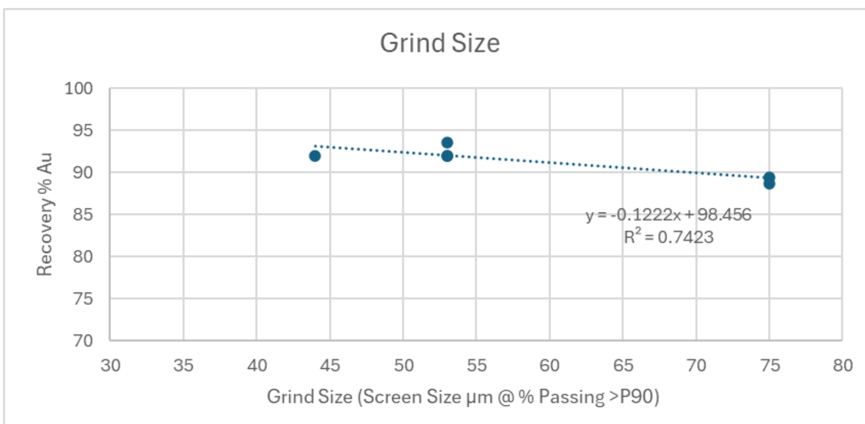


Figure 17 Leach Recovery vs Grind Size

The effect of cyanide concentration was examined in Table 15 at a fixed screen size target of P_{90} 75 µm and a fixed solids density of 53% solids in Series A, with a variation in solids density to 40% solids in Series B. The cyanide concentrations seem too low and reported poor recoveries 40-50% except for 1836-71, which reported 70% Au recovery. All the tests were 24-hour CIL basis. In series B, the lower density at 40% using a leach/CIL circuit option also failed in test 1850-53, with 35% Au recovery but then achieved 89.8% in test 180-54.

It was also noted in the report that the cause of poor cyanidation may be related to the formation of zinc and copper cyanides, with 0.1% zinc and 0.2% copper recorded on the carbon assays.

The report also noted that the main core sample F-16 has shown signs of coarse free gold. The letter from "*Mr R. B Coleman, Hazen VP*", specifically noted the following:

'Amalgamation of the sulphide composite sampled followed by leaching of the amalgam tails showed an overall recovery 92.5% (43.2% from the amalgamation and 49.3% from leaching). The sulphide sample contained 0.294 oz Au /t (10.08 g Au/t), and the amalgam tails were leached at 90% passing 100 mesh (150µm), 53% solids, 20 g/l carbon and 334 ppm NaCN concentration. The result indicated that if free gold can be recovered by gravity concentration, the remaining gold will be subject to CIL treatment under plant design conditions.' – *Note the final design conditions in the Minproc PDC varied from this report.*

The effect of a straight CIL versus a hybrid leach/CIL was examined in Table 17, which did not show any key difference between the two methods. Cyanide concentrations varied significantly with average concentrations from 290- 590 ppm NaCN, as displayed in Figure 18, increasing concentration trend with both head grade and recovery.

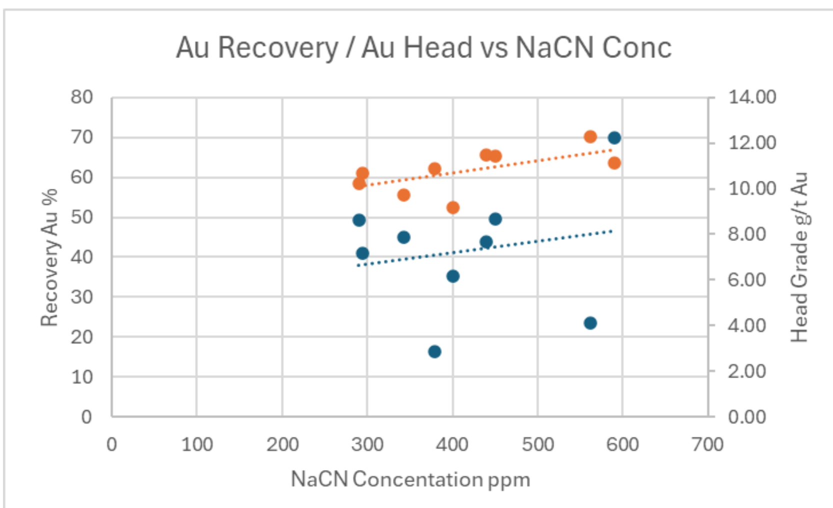


Figure 18 ***Gold Recovery and Head Grade against NaCN concentration.***

Carbon concentration on the recovery of gold was examined and reported in Table 18 for CIL and Table 19 for Leach/ CIL circuit options. The results varied with cyanide concentrations and density while the grind was maintained at 75 µm. The best result was at 50 g/l carbon with higher cyanide concentration 696 ppm NaCN achieving 93.2% Au recovery. Refer to (Test 1836-42, -48 and 115) for variations.

Baseline data				% Passing Micron			Ave NaCN		Tails	Au % Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
1836-10	20.78	0.59	CN/CIL	97	150	53	368	20	1.51	92.8	94.1	20.78	25.89	4.25	1.05
1836-11	2.67		CN/CIL	82	150	53	388	20	0.34	87.2	89.0	2.67	3.15	5.65	0.85
1836-12	14.13	12.2	CN/CIL	99	150	53	434	20	6.31	44.5	51.5	14.13	12.99	1.30	0.70
1836-13	9.81	12.4	CN/CIL	99	150	53	395	20	7.89	19.7	36.4	9.81	12.38	1.85	1.00

Table 14 Baseline Tests for Leach + CIL circuit

Effect of grind size and NaCN conc				% Passing Micron			Ave NaCN			Au % Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
1836-115	10.08	11.5	CIL	91	75	40	575	50	0.96	90.5	89.4	10.08	8.98	1.50	2.10
1836-42	12.07		CIL	90	53	40	696	50	0.82	93.2	92	12.07	11.83	1.00	2.45
1836-48	12.17		CIL	92	75	40	910	50	1.41	88.4	88.7	12.17	11.49	1.30	2.60
1836-43	12.07		CIL	96	53	40	641	50	0.75	93.8	93.6	12.07	11.69	1.10	3.20
1836-44	12.07		CIL	98	53	40	687	50	1.03	91.4	92	12.07	12.96	1.35	3.10
1836-15	11.86		CIL	95	44	40	786	50	0.89	92.5	91.9	11.86	10.94	3.30	3.10

Table 15 Effect on Recovery via Grind size and NaCN concentrations

Effect of NaCN conc at approx. P_{90} 75 micron				% Passing Micron			Ave NaCN		Tails	Au % Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
Series A															
1836-73	12.17		CIL	95	75	53	290	20	5.83	49.2	42.7	12.17	10.25	1.85	0.80
1836-72	12.17		CIL	93	75	53	295	20	6.58	40.9	38.7	12.17	10.70	1.35	0.70
1836-49	12.17		CIL	90	75	53	440	20	6.79	43.9	40.6	12.17	11.49	1.15	1.15
1836-71	12.17		CIL	91	75	53	590	20	3.57	70	66.3	12.17	11.11	1.45	1.45
Series B															
1850-53	10.08	11.5	CN/CIL	89	75	40	400	20	6.51	35.4	28.9	10.08	9.15	1.85	NR
1850-54	10.08	11.5	CN/CIL	89	75	40	860	20	1.03	89.8	89.7	10.08	9.94	1.7	1.3

Table 16 Effect on Recovery at P_{90} 75 µm and NaCN concentrations

Effect of CN/CIL vs CIL				% Passing Micron			Ave NaCN		Tails	Au Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
1836-50	12.17		CN/CIL	89	75	53	451	20	6.10	49.7	46.6	12.17	11.45	1.20	1.20
1836-45	12.17		CN/CIL	99	53	53	562	20	9.12	23.6	25.1	12.17	12.31	1.85	0.90
1850-53	10.08	11.5	CN/CIL	89	75	40	400	20	6.51	35.4	28.9	10.08	9.15	1.85	NR
1836-49	12.17		CIL	90	75	53	440	20	6.79	43.9	40.6	12.17	11.49	1.15	1.15
1836-71	12.17		CIL	91	75	53	590	20	3.57	70.0	66.3	12.17	11.11	1.45	1.45
1836-72	12.17		CIL	93	75	53	295	20	6.58	40.9	38.7	12.17	10.70	1.35	0.70
1836-73	12.17		CIL	95	75	53	290	20	5.83	49.2	42.7	12.17	10.25	1.85	0.80
1836-119	10.08	11.5	CIL	92	75	40	343	20	5.55	44.9	43.3	10.08	9.74	2.1	0.85
1850-52	10.08	11.5	CIL	89	75	40	379	20	8.43	16.3	22.8	10.08	10.90	1.8	NR

Table 17 Effect on Recovery for Leach/CIL vs CIL

Effect of Carbon Conc for CIL @ P_{90} 75µm				% Passing Micron			Ave NaCN		Tails	Au % Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
1836-42	12.07		CIL	90	75	40	696	50	0.82	93.2	92	12.07	11.83	1.00	2.45
1836-48	12.17		CIL	92	75	40	910	50	1.41	88.4	88.7	12.17	11.49	1.30	2.60
1836-115	10.08	11.5	CIL	91	75	40	575	50	0.96	90.5	89.4	10.08	8.98	1.50	2.10
1836-49	12.17		CIL	90	75	53	440	20	6.79	43.9	40.6	12.17	11.49	1.15	1.15
1836-71	12.17		CIL	91	75	53	590	20	3.57	70	66.3	12.17	11.11	1.45	1.45
1836-119	10.08	11.5	CIL	92	75	40	343	20	5.55	44.9	43.3	10.08	9.74	2.10	0.85
1850-52	10.08	11.5	CIL	89	75	40	379	20	8.43	16.3	22.8	10.08	10.90	1.80	NR

Table 18 Effect on Recovery at P_{90} 75µm with carbon variation in CIL

Effect of carbon conc for CN/CIL @ P_{90} 75µm				% Passing Micron			Ave NaCN		Tails	Au % Rec		Assay Head	Calc Head	Consumption	
HRI Test No.	Au g/t	S %	Type	Grind %	micron	% solids	Conc ppm	Carbon g/l	Au g/t	(1)	(2)	g Au/t	g Au/t	CaO kg/t	NaCN kg/t
1836-50	12.17		CN/CIL	89	75	53	451	20	6.10	49.7	46.6	12.17	11.45	1.20	1.20
1850-53	10.08	11.5	CN/CIL	89	75	40	400	20	6.51	35.4	28.9	10.08	9.15	1.85	0.00
1836-118	10.08	11.5	CN/CIL	93	75	53	685	20	2.26	77.6	75.9	10.08	9.33	2.10	1.80

Table 19 Effect on Recovery at P90 75µm with carbon variation in Leach/CIL

10.4 2025 Metallurgical Testwork

10.4.1 Composite Background

Six new composites were generated from existing and newly sourced drill core from the North and South Pit zones. The composites are listed by lithology type to examine the differences in ore hardness as well as response to the proposed CIL leach circuit.

The overall original project was going to examine 16 composites and a Master; however, this was reduced from that detailed in Table 20 to target lithology and reduce the overall number of required tests. Further, three low grade samples were added late in the programme to look at pit cut-off grade optimisation. The planned grades where to be less the 1 g/t Au to assess the final cut-off grade against metallurgical recovery.

Composite 1 – Sedimentary Breccia **(South Pit)**

CM23-08 (1) 1.15 g/t Au, CM23-11a (6) 1.58 g/t Au and CM23-16 (10) 1.30 g/t Au

Composite 2 – Granite **(South Pit)**

CM24-16 (8) 0.39 g/t Au and CM24-17 (11) 1.81 g/t Au

Composite 3 – Felsite **(South Pit)**

CM23-09 (2) 1.21 g/t Au

Composite 4 – Felsite **(South Pit)**

CM23-11a (4) 0.57 g/t Au and CM24-16 (9) 4.08 g/t Au

Composite 5 – Felsite **(North Pit)**

CM24-19a (15) 1.52 g/t Au and CM24-19a (16) 2.23 g/t Au

Composite 6 – Felsite Breccia **(North Pit)**

CM23-09 (3) 0.54 g/t Au, CM24-18 (12) 1.08 g/t Au CM23-11a (13) 1.29 g/t Au and CM24-19a (14) 0.49 g/t Au.

Low Grade # 1 – Hole CM24-18 weighted average 0.33 g/t Au

Low Grade # 2 – Hole CM24-18 weighted average 0.51 g/t Au

Low Grade # 3 – Hole CM24-18 weighted average 0.74 g/t Au.

Details on the composite makeup relate to Table 20. (e.g. Comp 5 CM24-19a (15), refers to the original composite 15 ID). A location map showing drill holes on both the North and South pits is illustrated in Figure 19.

ID	Hole_ID	From m	To m	Interval m	Weight kg	Au g/t	Pit	Primary Lithology	Oxidation State	Oxidation Intensity	Sulphidation
1	CM23-08	95.7	107.9	12.19	25.6	1.15	South	Sedimentary Breccia	OX	M	S
2	CM23-09	21.3	50.6	29.26	61.4	1.21	South	Felsite	OX	P	M
3	CM23-09	129.2	139.3	10.04	21.1	0.54	South	Felsite Breccia	OX	M	W-M
4	CM23-11a	35.3	47.9	12.60	26.5	0.57	South	Felsite	OX	S	W
5	CM23-11a	127.7	142.2	14.57	30.6	6.58	South	Felsite	OX	M	M
6	CM23-11a	148.9	163.8	14.93	31.4	1.58	South	Sedimentary Breccia	OX	W	M
7	CM23-11a	246.7	253.4	6.71	14.1	0.03	South	Granite			W
8	CM24-16	67.6	77.2	9.57	20.1	0.39	South	Granite	OX	M	W
9	CM24-16	77.2	87.4	10.24	21.5	4.08	South	Felsite	OX	W	M
10	CM24-16	130.1	140.8	10.67	22.4	1.30	South	Sedimentary Breccia	OX	M	W-M
11	CM24-17	0.0	21.3	21.34	44.8	1.21	South	Granite Breccia			W
12	CM24-18	0.0	25.9	25.91	54.4	1.08	North	Felsite Breccia			W-M
13	CM24-18	60.9	83.3	22.39	47.0	1.29	North	Felsite Breccia			W-M
14	CM24-19a	1.5	16.7	15.24	32.0	0.49	North	Felsite Breccia			W
15	CM24-19a	53.3	74.6	21.34	44.8	1.52	North	Felsite			M
16	CM24-19a	71.6	92.9	21.33	44.8	2.23	North	Felsite			W-M

Table 20 Original Composite Core Table

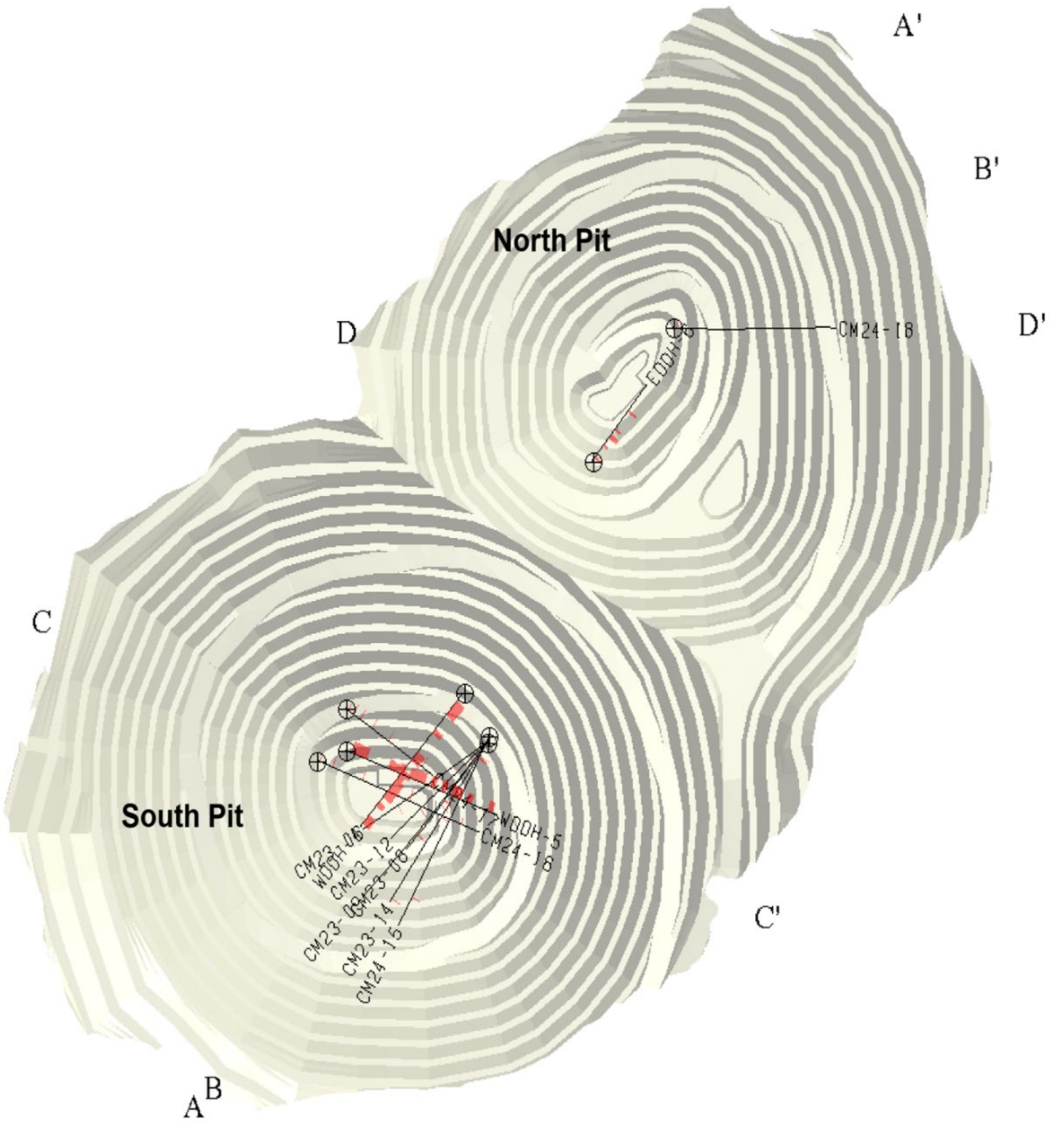


Figure 19 Plan View of Drill Hole Locations

10.4.2 Composite Head Assays

All six main composites were subject to XRD, QEMSCAN and combined density and multi-element chemical composition deterred by Gas Pycnometry, Fire Assay with AAS, Leco Combustion and Leach, Leco Combustion, ICP-OES, ICP- MS and direct Mercury Analysis. Results are detailed in Table 21.

Sample	Density g/cm3	Au g/t	Ag g/t	Total C	Carb C	Org C	Total S	Sulphide S
	Gas Pycno	FA/ASS		Leco Combustion & Leach , wt%				
Comp 1	2.77	**2.12**	**2.4**	0.415	0.39	0.03	2.88	2.50
Comp 2	2.79	**0.90**	**7.1**	0.498	0.47	0.03	0.95	0.73
Comp 3	2.75	**0.84**	**0.7**	0.075	0.07	0.07	0.69	0.63
Comp 4	2.81	**1.82**	**3.4**	0.135	0.12	0.12	2.39	2.15
Comp 5	2.73	**1.22**	**2.6**	0.027	0.02	0.02	0.39	0.30
Comp 6	2.72	**1.17**	**1.1**	0.247	0.24	0.01	0.46	0.42

Sample	Cu	Zn	As	Sb	Te	Pb	Bi	Hg
	ICP-OES and ICP-MS , ppm							DMA, ppb
Comp 1	153	167	94	<10	<10	157	<30	180
Comp 2	203	2540	50	<10	<10	136	<30	42
Comp 3	80	450	60	<10	<10	100	<30	116
Comp 4	273	892	42	<10	<10	175	<30	121
Comp 5	63	697	30	<10	<10	836	<30	210
Comp 6	50	940	40	<10	<10	670	<30	100

Sample	Li	Mg	Ti	V	Mn	Sr	Ce	Th
	ICP-OES, wt%							
Comp 1	0.007	0.39	0.079	0.0016	0.041	0.010	0.004	0.004
Comp 2	0.0073	0.65	0.349	0.0068	0.110	0.021	0.010	0.004
Comp 3	<0.006	0.22	0.061	<0.001	0.117	0.006	0.005	0.002
Comp 4	0.0063	0.25	0.065	0.0012	0.126	0.005	<0.003	0.003
Comp 5	<0.006	0.19	0.031	<0.001	0.032	0.005	<0.003	<0.002
Comp 6	<0.006	0.29	0.038	<0.001	0.071	0.002	<0.002	<0.002

Table 21 Composite Head Grades

10.4.3 Mineralogy

A mineralogical analysis was completed using XRD, QEMSCAN and optical microscopy on the six composites.

The gold (Au) grade of the six composites varies between 0.8 and 2.1 g/t Au, while silver (Ag) varied from 0.7 to 7.1 g/t. Average across the six composites was 1.35 g/t Au and 2.9 g/t Ag. Optical reflected light microscopy of composite 4, detected a cluster of gold grains averaging 10 μm in size and associated with quartz-goethite as illustrated in Figure 20.

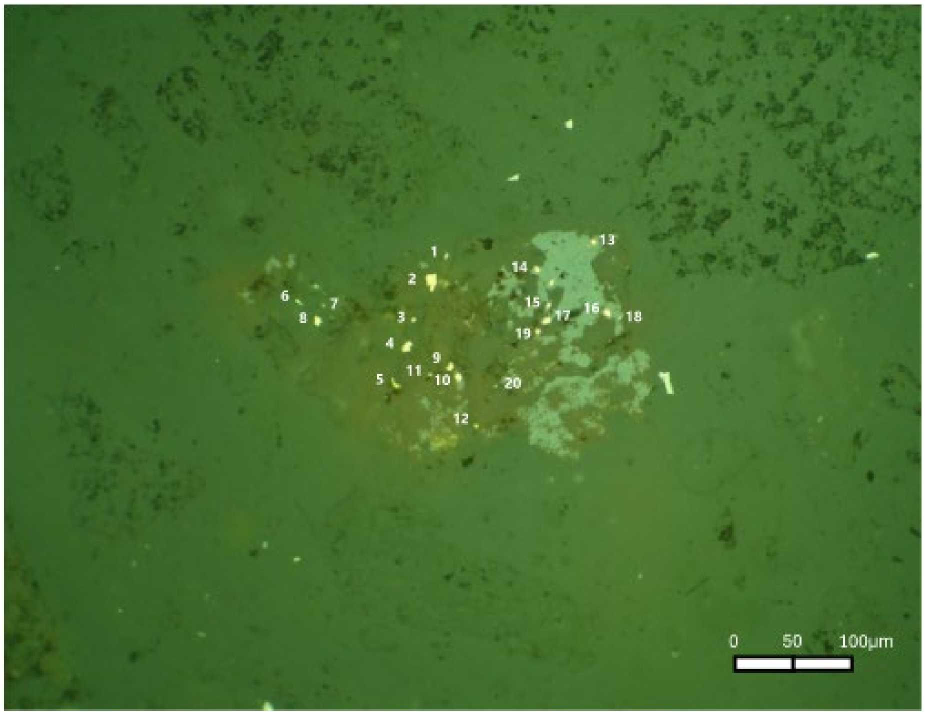


Figure 20 Gold Photo

All composites are defined by silica dominated matrix, mainly quartz (25 to 48%), feldspar (20 to 46%) and mica's (14 to 25%). The composites contain relatively low total sulphur with the main sulphide species being pyrite with minor sphalerite and chalcopyrite. Pyrite hosts 88% to 98% of the total sulphur as sulphides (Refer Table 23). Composites 1 and 4 host the highest sulphide content with pyrite containing 98.8 and 97.3% of the total sulphur. Composites 5 and 6 from the North pit have 6.9% and 5.8% respectively of the sulphur as sphalerite, however the sulphur weight % is only 0.4% and 0.5% respectively. The composites 1, 2 and 4 also contain minor copper varying from 50 to 275 ppm Cu. Most of the copper in the three composites is hosted in chalcopyrite, varying between 59 and 77% (Refer Table 24). Copper hosted in the chalcopyrite will have a very low potency to be a cyanide consumer. The remaining copper in the samples is associated with chalcocite, covellite and copper oxide phases which are cyanide soluble.

Composite 5 and 6 from the North pit hosted copper mainly in secondary copper sulphide and oxides minerals, however the copper content is very low at 63 and 50 ppm respectively.

Zinc content of the composites is higher than the copper content with values ranging from 160 to 2540 ppm, with composite 2 noted as being the highest 2450 ppm Zn. The zinc in the composite 2 was predominantly hosted as iron rich smithsonite $(Zn,Fe)CO_3$ which will partly react with cyanide and will be a cyanide consumer. The zinc concentration in the other samples is lower 160-950ppm and is contained in both sphalerite and Fe-rich smithsonite.

The composites appear to have a very low oxygen consuming potential based on the relatively low sulphide sulphur content and the dominate sulphide being pyrite. Minor sphalerite may act as a cyanide and oxygen consumer.

The composites display low levels arsenic (30 to 95 ppm), antimony (<10 ppm), tellurium (<10 ppm), bismuth (<30ppm) and mercury (<200 ppb). The composite indicate that the gold is not likely refractory and should be free milling. In addition, the organic carbon content was very low <0.03%, hence the potential for preg- robbing by organic carbon is very low.

Bulk mineralogical composition by mass is detailed in Table 22 and Figure 21.

Mineral	Formula	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6
Quartz	SiO_2	44.3	34.1	58.6	61.8	60.7	57.7
k-Feldspar	$KAlSi_3O_8$	39.7	22.7	16.7	11.0	20.1	20.0
Plagioclase	$NaALSi_3O_8$	nd	25.1	nd	nd	nd	nd
Mica (Muscovite)	$Kal_2(Si_3Al)O_{10}(OH,F)_2$	10.1	12.2	23.5	23.0	18.8	20.3
Clinochlore	$(Mg,Fe)_5Al(Si_3Al)O_{10}(OH)_8$	nd	2.5	nd	nd	nd	Nd
Pyrite	FeS_2	3.9	1.3	1.2	3.7	0.4	1.1
Dolomite	$Ca(Fe,Mg)(CO_3)_2$	1.5	tr	tr	0.1	tr	0.9
Siderite	$FeCO_3$	0.5	1.6	nd	0.1	nd	Nd
Calcite	$CaCO_3$	tr	0.5	tr	0.3	tr	tr

Table 22 % Mass of dominate mineral phases

Sulphide Mineral	Formula	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6
Pyrite	FeS_2	5.82	1.63	1.51	4.43	0.60	0.95
Sphalerite	ZnS	0.02	0.01	0.01	0.04	0.08	0.10
Chalcopyrite	$CuFeS_2$	0.04	0.04	0.01	0.04	0.00	0.00
Bornite	Cu_5FeS_4	-	-	-	-	-	-
Chalcocite	Cu_2S	-	-	-	-	-	-
Covellite	CuS	-	-	-	0.01	-	1.1
% Total		5.89	1.69	1.53	4.52	0.69	1.05

Table 23 QEMSCAN Bulk Model Analysis (Sulphides) %Mass out of 100%

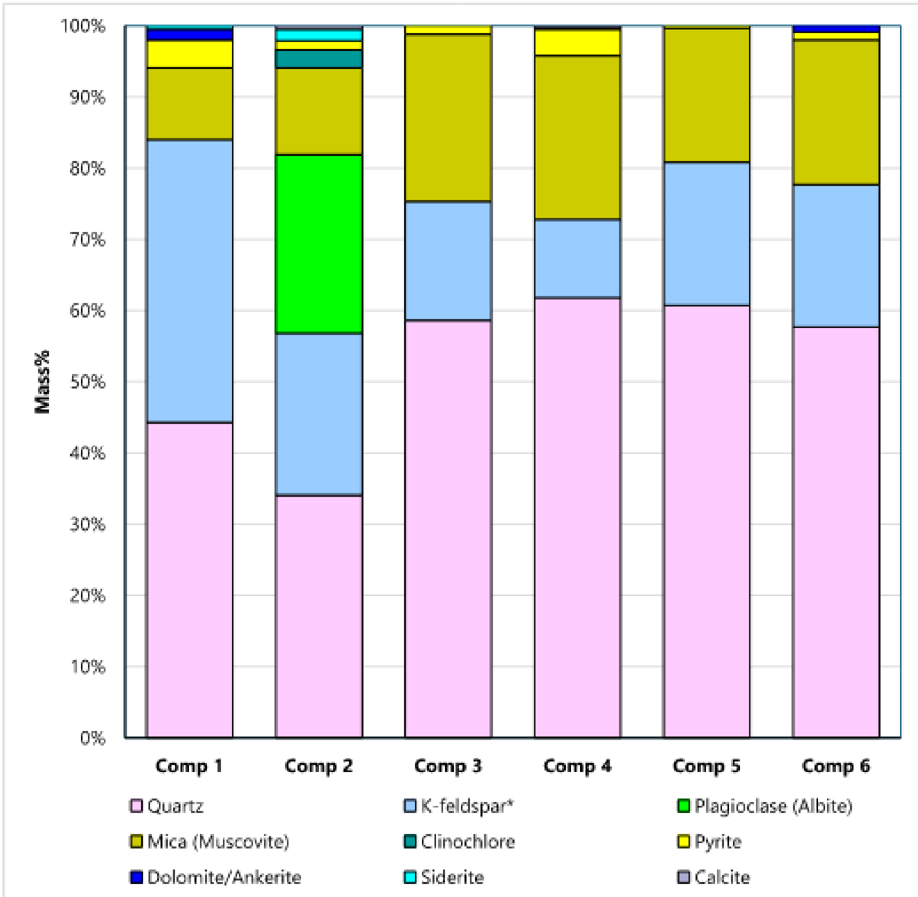


Figure 21 Bulk Mineralogical Composition

Sulphide Mineral	Formula	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6
S Grade (wt %)		2.88	0.95	0.69	2.39	0.39	0.46
Pyrite	FeS_2	98.80	96.38	97.32	98.07	87.88	90.76
Sphalerite	ZnS	0.18	0.33	0.37	0.49	6.91	5.79
Chalcopyrite	$CuFeS_2$	0.48	1.55	0.26	0.64	0.38	0.17
Bornite	Cu_5FeS_4	0.01	0.03	0.00	0.02	0.00	0.00
Chalcocite	Cu_2S	0.03	0.01	0.00	0.01	0.01	0.01
Covellite	CuS	0.04	0.16	0.14	0.11	0.35	0.23
Gypsum	$CaSO_4.2H_20$	0.00	0.04	0.90	0.01	0.01	0.03
Others		0.46	1.51	1.00	0.65	4.46	3.02
% Total		100	100	100	100	100	100

Table 24 Indicative Sulphur Deportment

Copper Mineral	Formula	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6
Cu Grade (ppm)		153	203	80	273	63	50
Chalcopyrite	$CuFeS_2$	67.07	77.05	37.94	58.40	33.18	21.66
Bornite	Cu_5FeS_4	2.83	3.12	-	4.15	0.00	0.00
Chalcocite	Cu_2S	14.70	2.91	2.43	5.22	3.49	3.54
Covellite	CuS	11.30	15.55	41.83	19.19	60.15	58.53
Others		4.09	1.36	17.80	13.04	3.18	16.27
% Total		100	100	100	100	100	100

Table 25 Indicative Copper Deportment

10.4.4 Comminution Testing

The program looked to provide updated SMC/ JK and Bond comminution data for the project design criteria. Crushing work indices were limited to composites 3 (CM23-09 core) and 6 (CM24-18 core). This test work was conducted by FLS and caused some delays in finalising the composites as the rejects needed to be return to Hazen.

Hazen conducted Bond Rod Mill, Ball Mill, Ai tests followed by SMC testing for the Morrell DWi, Mia, Mih and Mic indices. Results from the SMC test were then sent to JK Tech for the generation of JK – Axb, and the data from the modified SMC test.

Sample	Density (SG)	CWi (kWh/st)	CWi (kWh/t)	Classification
CM23-09	2.67	10.9	12	Medium
CM24-18	2.59	10.6	11.6	Medium

Table 26 Bond Crusher Work Index (CWi)

Comp	RWi (kWh/t)	BWi (kWh/t) (Closing Screen 150µm)	F_{80} (µm)	P_{80} (µm)	Gbp	Ai (g)
1	18.2	20.8	2659	119	1.04	0.5500
2	14.7	16.7	2592	124	1.41	0.1697
3	19.0	20.1	2661	117	1.07	0.3290
4	18.4	19.8	2829	117	1.08	0.2640
5	20.8	22.5	2708	118	0.94	0.2741
6	20.9	22.8	2737	116	0.91	0.3628

Table 27 Bond Rod, Ball Mill Work Indices and Ai data

Comp	DWi	SMC Mi Parameters (kWh/t)				JK Parameters					
	kWh/m³	Mia	Mih	Mic	*Mib(12)*	A	b	Axb	ta	SCSE (kWh/t)	SG
1	7.46	21.5	16.2	8.4	*26.92*	60.4	0.59	35.64	0.35	10.34	2.66
2	5.64	17.3	12.4	6.4	*20.32*	60.2	0.78	46.96	0.46	9.11	2.65
3	8.98	24.9	19.5	10.1	*25.94*	81.8	0.36	29.45	0.29	11.33	2.66
4	7.44	21.4	16.1	8.3	*25.39*	52.7	0.68	35.84	0.35	10.33	2.67
5	9.89	27	21.6	11.2	*29.55*	54.8	0.49	26.85	0.26	11.84	2.65
6	10.89	29	23.7	12.2	*30.54*	87.1	0.28	24.39	0.24	12.49	2.67

Table 28 SMC and Derived JK Parameters

(14) The Mib value is calculated using the bond Ball Mill Test data. $$M_{ib} = \frac{18.18}{P_1^{0.295}(\mathrm{Gbp})(p_{80}^{f(p_{80})} - f_{80}^{f(f_{80})})}$$

The results indicate that the ore can be treated by a conventional SAB or SABC circuit. The RWi/BWi was not > 1.2, which is an indication for the potency to generate pebbles. The circuit design still retains the pebble crusher based on the historical issues that the Bond Gold Operation incurred.

10.4.5 Grind Size Evaluation

All six composites were subjected to a standard cyanide leach at four grind sizes P_{80} of 150, 120, 106 and 75 µm. The leach conditions consisted of a bottle run at 50% solids, pH 10.5-11 with hydrated lime, NaCN at 1 g/l for 48 hours.

The leach analysis indicates an improving leach extraction with a finer grind size. This was to be expected with increased liberation. The target grind size for the design is set for 106 µm, indicated by the green band on Figure 22.

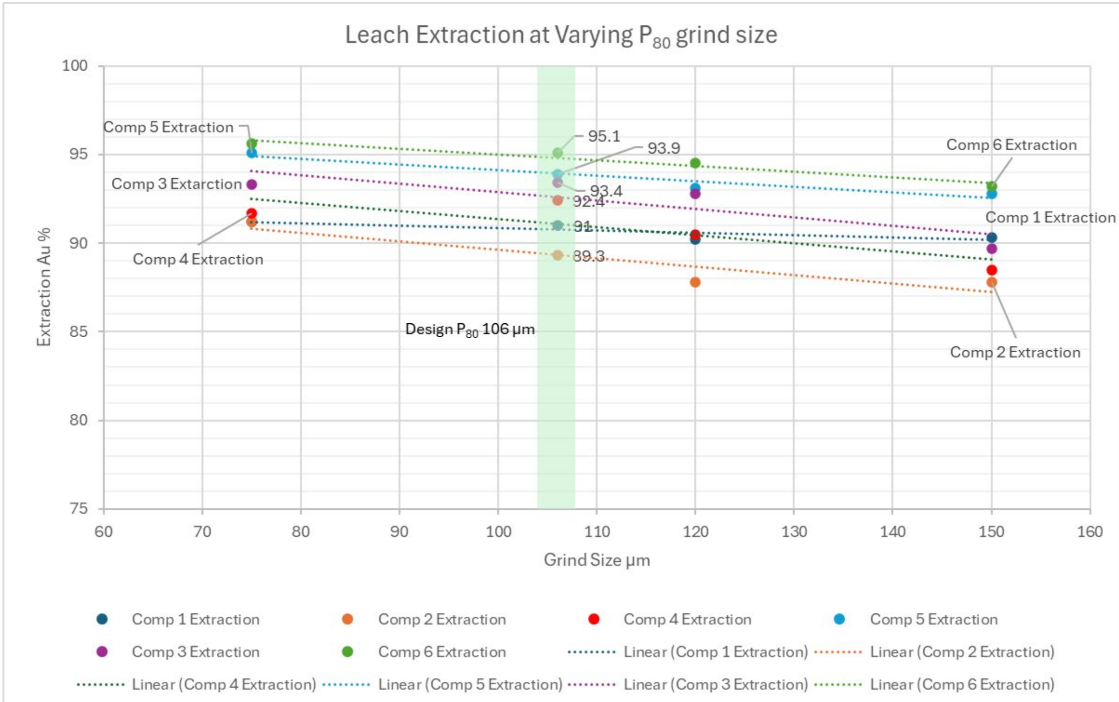


Figure 22 Leach Extraction a varying grind size

Composite 1 had an only a minor increase in extraction from 91% to 91.2 % from 106 µm to 75 µm. Composite 2 gave the best results jumping from 89.3% to 91.2%, with a 1.9% extraction improvement from 106 µm to 75 µm. The best performance was noted with composite 5 from the North pit. These figures need to also be assessed in terms of the head grade of the leach tests, noting that Composite 5 increased from 1.47 g/t (106 µm) to 1.81 g/t (75 µm), while the residues from both were identical at 0.089 g/t.

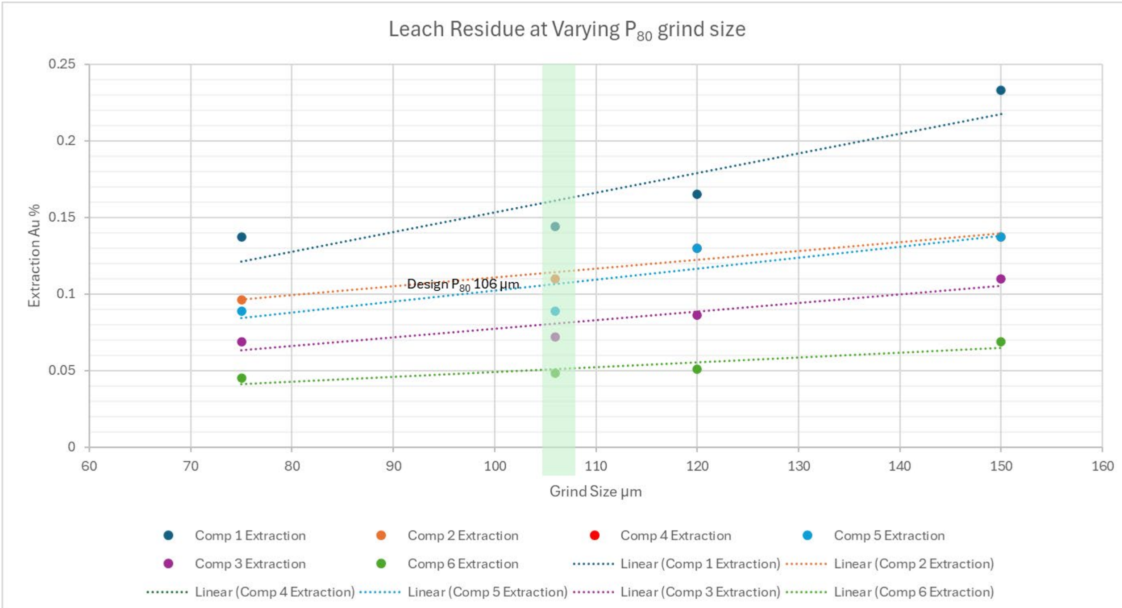


Figure 23 Au g/t Residue at varying grind P_{80}

Residual tails indicate that grinding to 75µm would be similar to the 106µm figures, hence for a fixed head grade, the effect of the additional grinding would limit the Au extraction % in each case.

10.4.6 Leach Kinetics

The grind analysis work was carried out with samples taken at 2, 4, 8, 24 and 48 hours at the 1g/l NaCN concentration and 50% solids.

Composite 1 – South Pit indicated that the leach is mainly complete at around 24 hours, with a minor increase in Au extraction at 48 hours. The gold leach was rapid, while additional time in the leach circuit benefited the Ag extraction figures.

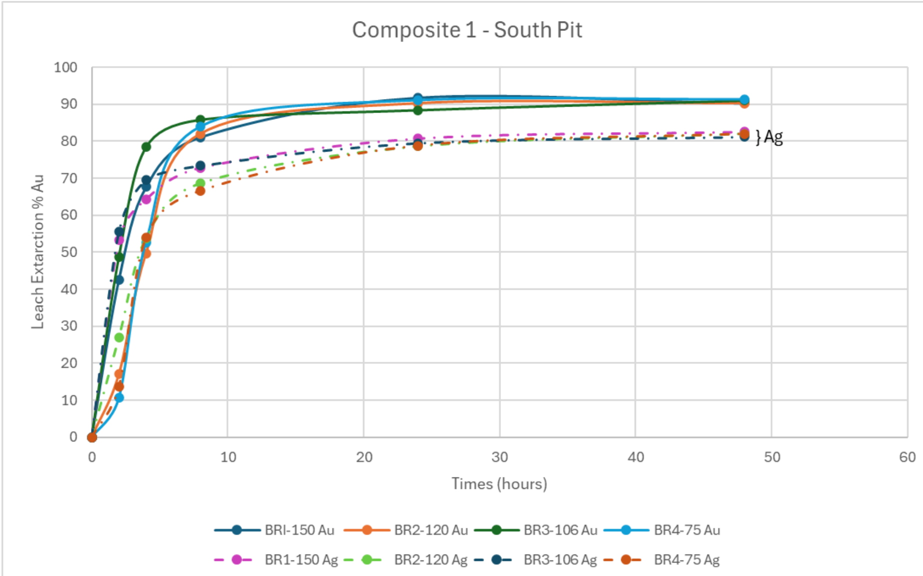


Figure 24 Composite 1 – South Pit Leach Kinetics

Composite 2 – South Pit was like composite 1 with slightly better silver kinetics at 48 hours. The coarsest grind at 150 µm yielded 89.7% Ag extraction, mainly due to the elevated head grade of 2.44 g/t Ag at 0.5 to 1.2 g/t higher than the other grinds in the series.

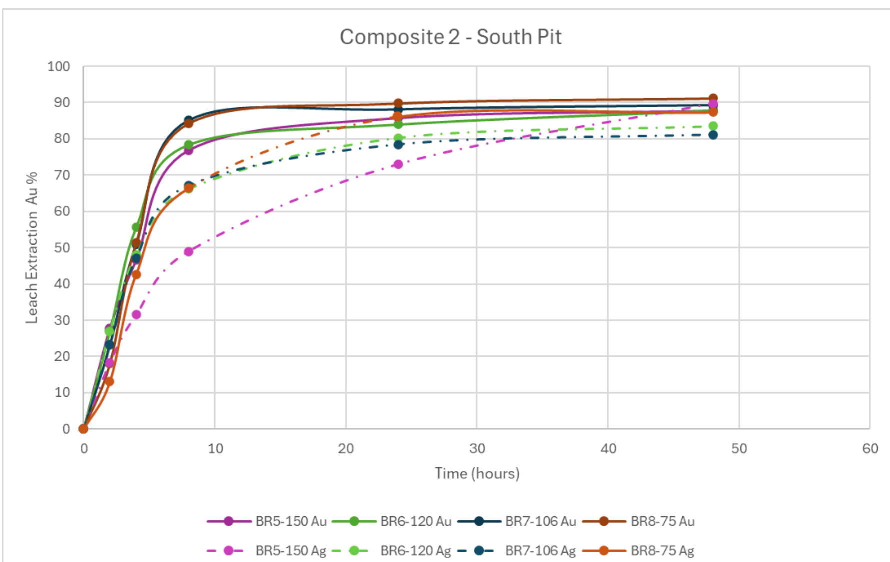


Figure 25 Composite 2 – South Pit Leach Kinetics

Composite 3 -South Pit leach results were slightly slower than comps 1 and 2, but overall, at 48 hours were similar. Silver kinetics were notably slower than previous composites and performed poorly in terms of leach extraction with the grind size variations all terminating around 60% Ag extraction.

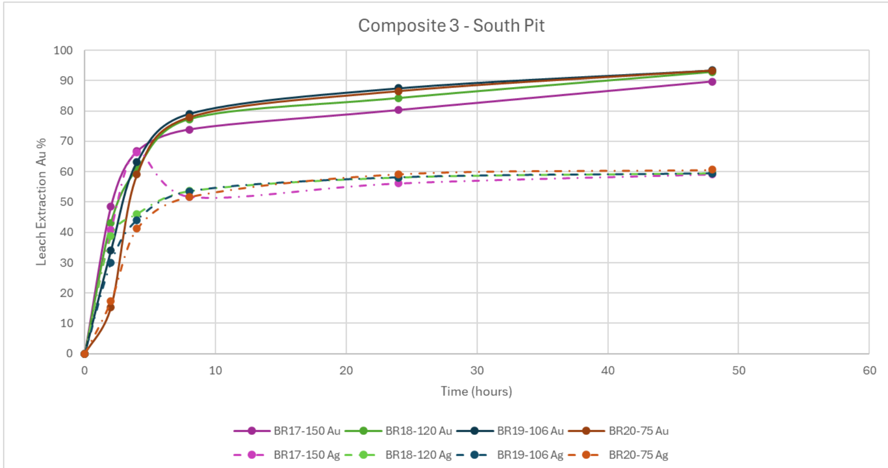


Figure 26 Composite 3 – South Pit Leach Kinetics

Composite 4 – South Pit showed late increases to silver extraction on the 48-hour basis, while gold remained static with no real increase in leach extraction witnessed after 24 hours.

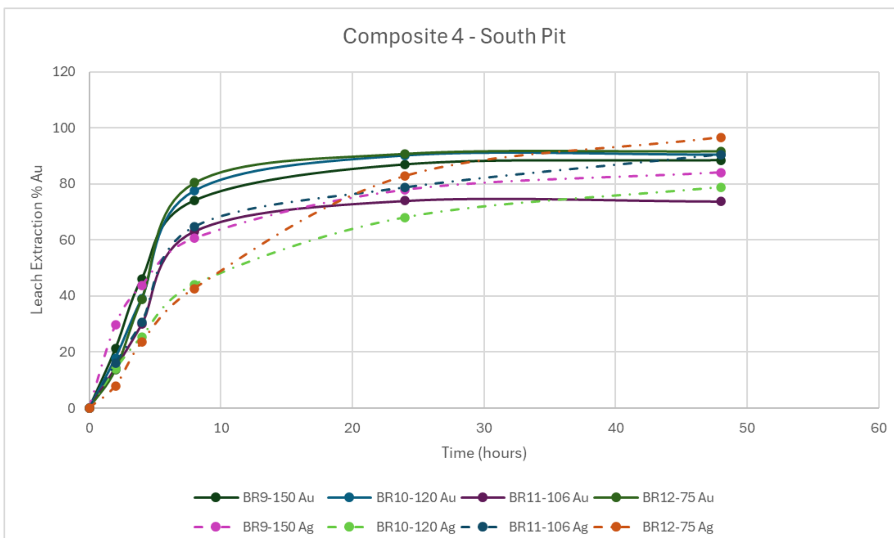


Figure 27 Composite 4 – South Pit Leach Kinetics

Composite 5 – Noth Pit indicated that the leach is mainly complete at around 24 hours, with a minor increase in Au extraction at 48 hours. The gold leach was rapid, while additional time in the leach circuit benefited the Ag extraction figures. Results for test (BR16-75), Ag based on residual tails against the calculated head grade confirm a final recovery of 70.9% Ag, the curve is odd as pregnant solution dropped from 24 to 48 hours and may be due to assaying errors in the earlier samples (8 and 24 hours). Cyanide concentration was maintained at 0.96 g/l at 48 hours.

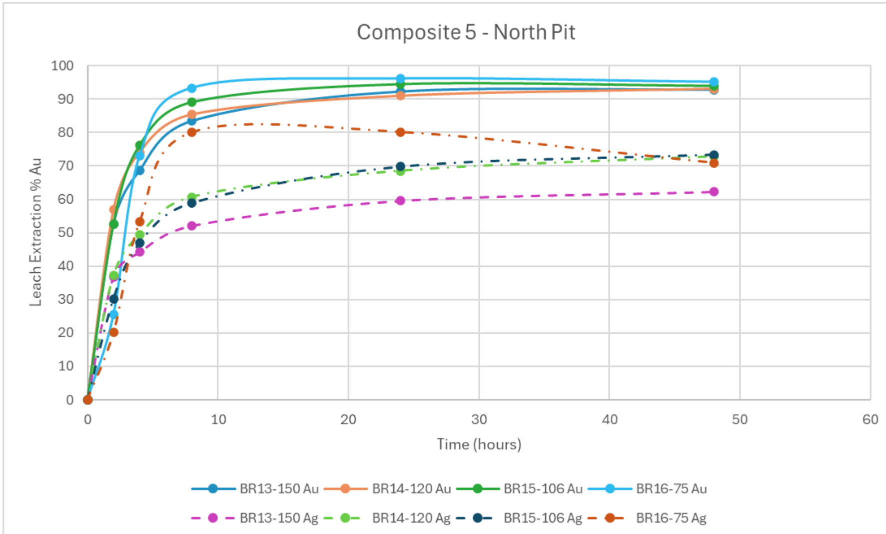


Figure 28 Composite 5 – North Pit Kinetics

Composite 6 – North Pit indicated that the leach is mainly complete at around 24 hours, with a minor increase in Au extraction at 48 hours. The gold leach was rapid, while additional time in the leach circuit benefited the Ag extraction figures. Cyanide concentration was maintained at 0.92- 0.96 g/l at 48 hours.

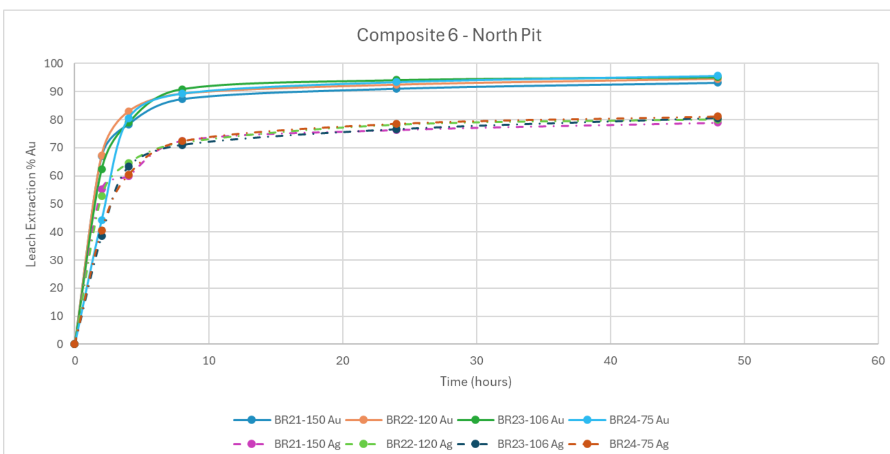


Figure 29 Composite 6 – North Pit Kinetics

10.4.7 Low Grade Composites – Bottle rolls

Three low grade composites were developed aimed at producing a head grade of 0.35-0.4 g Au/t, 0.40-0.75 g Au/t and 0.75-1.0 g Au/t Au range. The reason for this was to examine the leach extraction at the lower grades as part of the pit optimisation process for a pit cut-off grade point. The weighted average figures were in the predicted range, however the back calculated head grades from the bottle roles varied significantly. The tests were all run for 30 hours at 0.75 g/l NaCN and at the P_{80} 106 µm grind size. Results are table in Table 29.

Sample	Au g/t (weighted ave)	Leach Au g/t Head Grade	Leach Residual Tail Au g/t	Au Ext%	Leach Ag g/t Head Grade	Leach Residual Tail Ag g/t	Ag Ext %
LG 1	0.33	0.54	0.06	88.6	1.87	1	46.5
LG 2	0.51	0.51	0.03	93.3	0.66	<0.5	62.1
LG 3	0.74	1.13	0.33	70.9	3.58	1.8	49.7

Table 29 Low Head Grade Results

LG 1 and 2 both proved that a lower head grade can still be leached to a high extraction rate based on 30 hours of leach retention time. LG3 had a poor result with a high silver head grade associated. This sample could have Au-Ag amalgam as both the gold and silver extractions were poor with high residual tails.

10.4.8 Detoxification

Detoxification testing was undertaken at ACZ laboratory on bulk samples of both North and South pit composites. Overall test results show significant reduction in all forms of cyanide from before to after detox using the INCO process. WAD cyanide was reduced to under 1 ppm, while total cyanide was between 1 ppm to 2.5 ppm.

Composite	Parameter	Results	Units
North Pit – Prior to Detox	Free Cyanide Total Cyanide WAD Cyanide	147 117 79.1	mg/L mg/L mg/L
North Pit – Detoxed	Free Cyanide Total Cyanide WAD Cyanide	0.0509 1.17 0.572	mg/L mg/L mg/L
South Pit – Prior to Detox	Free Cyanide Total Cyanide WAD Cyanide	184 147 88.8	mg/L mg/L mg/L
South Pit – Detoxed	Free Cyanide Total Cyanide WAD Cyanide	0.178 2.4 0.963	mg/L mg/L mg/L

Table 30 Detoxification Results

10.4.9 *Initial Static Cylinder Testing*

A series of static cylinder tests were conducted at RDi using a finer grind P_{80} 75 µm. Samples were flocculated with 6 ml of 0.10% strength flocculant in 1000 ml test cylinders.

Tabulated results are detailed in Table 31.

	Flux t/m².h	Unit Area m²/ mt/ day	U/F density	Thickener size (m)
Comp 1	0.801	0.052	50%	19.9
Comp 2	0.621	0.067	50%	22.6
Comp 3	0.555 0.180	0.075 0.231	45% 50%	23.9 42.0
Comp 4	0.967 0.336	0.043 0.124	45% 50%	18.1 30.8
Comp 5	1.126 0.448	0.037 0.093	45% 50%	16.8 26.7
Comp 6	0.389 0.178	0.107 0.233	45% 50%	28.6 42.2

Table 31 Initial Static Cylinder Settling Test Results

10.4.10 *Initial Filter Testing*

Three composites (5, 2 and 4) from the initial grind size evaluation at P_{80} 75 µm were subjected to lab scale pressure and vacuum filtration to assess filtration capacity for initial sizing of equipment. Subsequent filtration work was competed on a North and South pit composite at the P_{80} 106 µm grind size by filter vendors to provide a more robust assessment of filtration rates for plate and frame (P&F) type filters and vacuum filtration at the target moisture of 15%.

Comp	Type	Feed % solids	Pressure kPa	Cake Moisture %	Filtration rate kg.DS/m².h	Filtrate Rate lt/m².h
Comp 5, Detoxed	Pressure	53	550	17.4	527	357
Comp 2, Detoxed	Pressure	50	550	16.6	706	565
Comp 4, Detoxed	Pressure	53	550	17.1	360	247
Comp 5, Detoxed	Vacuum	51.5	-57	23.5	146	110
Comp 2, Detoxed	Vacuum	50	-57	24.5	302	202
Comp 4, Detoxed	Vacuum	53.5	-57	23.5	146	110

Table 32 Average data of test results for pressure and vacuum lab filtration tests

Initial results showed that vacuum filtration technology will produce a ~23% filter cake, with this expected to be lower with the coarser plant design P_{80} 106µm. Discussions with vacuum belt supplier, Jord, indicate that new Viper™ technology units (vibration of cake to improve percolation), could reach 15-17% moisture targets with existing HVBF technology.

The pressure filtration results (360-706 kg DS/m^2.h) at low feed pressures compared to industrial machines, indicated that good filtration rates and moistures should be achievable. Ground ore samples at P_{80} 106µm were also sent to several plate and frame pressure filter suppliers for lab scale testing and machine sizing. The filtration rates achieved in the lab do not include any technical time – open/close and fill time associated with P&F filters and only considered form time and dry time on the batch samples.

10.4.10.1 Historical Colosseum Tailings

Two composites were generated from geotechnical drilling program of the existing historical tailings. Composite 1 had a P_{80} 126 µm with 38% of the material sized passing 38 µm. Composite 2 had a P_{80} 103 µm with 30% of the material sized passing 38 µm.

The two composites were tested at Jord's SA lab using a HVBF lab scale filter equipped with Viper™ technology units.

Test No.	1	2	3	4	5	6	7	8
Sample	Colosseum, Sample 1				Colosseum, Sample 2			
Feed Solids (% w/w) (exc. flocculant)	56%	56%	56%	56%	55%	55%	55%	55%
Filter Cloth	230 CFM				230 CFM			
Solids in Filtrate (% w/w solids)	~1% (TBC)							
Cake Thickness (mm)	8.0	8.0	8.0	8.0	8.0	10.0	8.0	10.0
Floc Addition [] - (g/metric t [solids])	25.0	25.0	35.0	35.0	18.0	25.0	18.0	25.0
Vacuum Pressure (kPag)	-62	-62	-62	-62	-62	-62	-62	-62
Form Time (s)	38	38	30	30	27	35	27	35
Vibration (Y/N, if Y [stages])	Y [3]	Y [3]	Y [3]	Y [3]	Y [3]	Y [3]	Y [2]	Y [2]
Drying, including vibration (s)	45	40	45	40	40	40	30	30
Total Time (s)	83	78	75	70	67	75	57	65
Total Time (min)	1.38	1.30	1.25	1.17	1.12	1.25	0.95	1.08
Cake Moisture (%w/w)	14.8%	15.0%	15.2%	15.5%	14.3%	14.6%	15.1%	15.5%
Cake SG (Wet)	2.09	2.09	2.08	2.07	2.11	2.10	2.08	2.07
Filtration Rate (kg DS/m^2.h)	618.6	654.9	677.7	720.6	776.1	860.1	893.9	970.1

Table 33 Jord testing results on Historical Tailings (P_{80} 126 and 103 µm)

10.4.11 Thickening (Pocock)

Preliminary settling and thickening tests were conducted at RDi (section 10.4.9) above and used as a guide for the dynamic thickener testing. SNF Flocculant AN905SH was used for the test work. The results are tabled in Table 34.

Material	Tested Feed Solids	Floc	Overflow TSS (mg/l)	Settling rate	Unit Area	U/F Density	Net Feed Loading
		Dose (g/MT)	(mg/L)	(t/m²‧h)	(m²/MTPD)	% Solids	(m³/m²‧h)
North LR	24.7 %	65	150-250	0.71	0.058	59.9	2.44
South LR	19.6 %	44	150-150	0.54	0.078	59.1	2.38

Table 34 Pocock Dynamic Thickening

Sample	Predicted Thickener Underflow Density					
	Max Achieved		Fully Sheared Yield Value			Recommended Maximum Design
	Static Tests	Dynamic Tests	20 Pa	30 Pa	40 Pa	
Thickened North LR	71.9%	64.5%	65.3%	66.6%	67.5%	66.5%
Thickened South LR	69.5%	70.5%	58.4%	60.4%	61.7%	60.4%

Table 35 Predicted U/F Density

The recommended hydraulic loading rate for the North Pit material was 2.4 m^3/m^2.h at 25% feed solids % using approximately 70 g/t SNF AN905SH. Under these conditions, overflow suspended solids would likely remain below 250 mg/l.

The recommended hydraulic loading rate for the South Pit material was 2.4 m^3/m^2.h at 25% feed solids % using approximately 45 g/t SNF AN905SH. Under these conditions, overflow suspended solids would likely remain below 250 mg/l.

10.4.12 Slurry Rheology (Pocock)

Testing of slurry rheology was undertaken using a Fann (Model 35A) Viscometer. Viscosity tests studied the flow behaviour of thickened slurries across a specific range of shear rates. The data include the relationships between apparent viscosity (Pa·sec) and shear rate (sec^{-1}), and the yield stress (Pa) at varied solids concentrations, all recorded at fixed operating temperatures, grind sizes, solid concentrations, residual flocculant levels, and pH.

Expected yield valves at the design density of 55% solids will be less than 13 N/m^2.

Material	Rheological Characteristics based on Viscosity Testing											
	Solids Conc. (%)	Yield Value (Pascals or N/m^2)	Coefficient of Rigidity (Pa)	Apparent Viscosity (Pa•sec) @ the following Shear Rates:								
				5 Sec^{-1}	10 Sec^{-1}	25 Sec^{-1}	50 Sec^{-1}	100 Sec^{-1}	150 Sec^{-1}	300 Sec^{-1}	600 Sec^{-1}	1,000 Sec^{-1}
Thickened North LR	68.5	59.8	0.138	7.526	4.734	2.566	1.614	1.015	0.774	0.487	0.306	0.218
	67	33.7	0.082	4.932	3.004	0.950	0.711	0.578	0.433	0.352	0,300	0.264
	65.5	18.6	0.054	3.217	1.906	0.565	0.416	0.335	0.247	0.198	0.168	0.146
	63.7	11.2	0.039	2.255	1.309	0.371	0.270	0.215	0.157	0.125	0.105	0.091
	62.2	8.1	0.028	1.613	0.937	0.265	0.193	0.154	0.112	0.089	0.075	0.065
	60.2	5.5	0.022	1.045	0.630	0.195	0.145	0.117	0.087	0.071	0.060	0.053
Thickened South LR	65.2	74.0	0.056	7.585	4.630	2.411	1.472	0.898	0.673	0.411	0.251	0.174
	62.3	52.2	0.044	5.800	3.486	1.069	0.794	0.643	0.477	0.386	0.328	0.287
	60.7	37.3	0.036	4.575	2.688	0.782	0.573	0.460	0.337	0.270	0.228	0.198
	60	24.9	0.031	3.467	1.994	0.552	0.399	0.317	0.23	0.183	0.153	0.132
	56.8	13.4	0.021	2.217	1.241	0.322	0.230	0.18	0.128	0.101	0.084	0.072
	53.9	6.7	0.014	1,216	0.685	0.181	0.129	0.102	0.073	0.057	0.048	0.041

Table 36 Rheology Testing

10.4.13 Filtration (Pocock)

Pressure filtration testing was conducted on the thickened sampled from section (10.4.11). Pressure filtration tests evaluated how cake thickness and air-dry time affect production rate and filter cake moisture for the thickened leach residue materials.

Two operational scenarios were tested during the campaign: air blow only and light membrane squeeze during air blow, followed by a full-pressure membrane squeeze. An 80-psig driving force was used for all filling and air blow operations throughout these procedures. For conditions involving a squeeze during air blow, a 100-psig squeeze was applied until the last 30 seconds of air blow, when the squeeze pressure was increased to 232-psig to complete the cycle. The table below summarizes the pressure filtration performance data for the tests performed.

Material	Design Conditions	Feed Solids	Dry Bulk Density MT/m^3	Chamber Cale Thickness (mm)	Sizing Basis (m^3/MT)	Design Cake Moisture	Total Cycle Time (min)	Vol Production Rate (MTPD/m^3)	Area Basis Production Rate (dry $kg/m^2.h$)
North Pit	Air Blow	65.7%	1842	40 / 40	0.679	14.4%	14.10	125.4	102.63
	Air Blow with membrane Squeeze	65.7%	1865	40 / 39.5	0.670	13.9%	14.30	125.3	101.19
South Pit	Air Blow	60.0%	1746	40 / 40	0.716	15.4%	13.52	124.1	102.24

	Air Blow with membrane Squeeze	60.0%	1849	40 / 37.8	0.676	14.5%	13.59	130.6	101.74

(1) Filter press sizing basis in cubic meters per metric ton of dry solids includes a 1.25 scale-up factor.

(2) The cake moistures chosen for the design provided acceptable discharge and stacking properties at reasonable dry times. Cake moistures are based solely on air drying. Squeeze testing was not performed on these materials.

(3) The filter press cycle time is based on a 20-hour operating day and includes an assumed 6.0-minute dead time for opening and closing the press.

(4) Volumetric production rate in metric tons per day of ore processed per cubic meter of available filtration volume. These values are specific to the sizing basis, cake thickness, and total cycle time indicated.

(5) Predicted production rate in kilograms per hour per square meter of available filtration area, based on the volume production rates, chamber sizes, and areas specified in the individual summary sheets. These values can be directly compared to the results of vacuum filtration.

Table 37 Pocock Filtration Tests

Both the North and South pit material formed dischargeable cakes within reasonable form and dry times. The form time was heavily influenced by the cake and chamber thicknesses. A 40 mm chamber was selected for the design simulation, as it provided a good trade-off between reducing form time, achieving lower cake moisture, and maximizing production rates. A thicker chamber could increase production rates at the expense of increased moisture in the cakes. The cake moisture chosen for the design provided good discharge and stacking properties within acceptable drying periods.

10.4.14 2026 – RDI Ground Sample for Vendor Tails Filter Test

A bulk sample made up from reserves of composite 1 to 6 was developed as a Master sample for Vendor filtration testing. This sample was milled under the same conditions as the leach test work to develop a bulk sample at P_{80} 106 µm for the Vendors to use for their filtration tests. The sample was Not Leached and was only milled with pH adjusted to pH 10 to simulate a possible tail without the leaching stage.

10.4.14.1 Jord Vacuum Filtration (HBVF)

Jord ran a series of tests and reported tests 4 through to 32 using there standard buchner funnel with the addition of vibration. They used ES-0262 cloth, which is a high permeability mono-mono (monofilament) satin weave. Results are detailed in Table 38.

The vendor ran some of the tests at higher feed densities than the proposed design figure of 55% and at vacuum pressure of 300 mbar abs (30 kpa_{abs}) relates to a gauge vacuum pressure of -70.9 kPa relative to standard atmosphere of 101.325 kPa. This is high vacuum figure, and at 5000 ft (1524 m) the atmospheric pressure is 84.31 kpa, which would drop the effective gauge vacuum of-54 kPa.

Test were conducted with the standard test (no vibration) and with vibration to simulate the Viper™ units used in the full-scale machine. In addition, the vendors were requested to test for a range of target moistures from 15 to 23% Moisture cakes and assess the thixotropic behaviour of the samples. Filter rates varied from 324 kg D.S/m^2.h (test 6) to 1572 kg D.S/m^2.h (test 32). The test runs 21 to 24 did not use any vibration, had high start solids at 60% and utilised M5250 flocculant. These runs yielded high moistures 24.3 to 27% with rates from 944 to 1269 kg D.S/m^2.h.

The vibration test also used a variation of high and low vibration. The assessment from the test which ranged in both moisture and filtration rate showed that the moisture can be reduced by using the Viper type system with the likely operating window for the plant design sitting at 18 to 19% moisture. While the cake samples did not show thixotropic conditions from the Vendor test runs, samples from other tests still indicate that the sample can act as a thixotropic material and has the ability to release water and partially liquefy.

Figure 30 illustrates the Vendor's filtration results. The target window of 18-19% Moisture developed a range from 720 to 550 kg D.S/m^2.h. From the graph at 18.5% rate would be equivalent 637 kg D.S/m^2.h using 1 stage of vibration to reach the target.

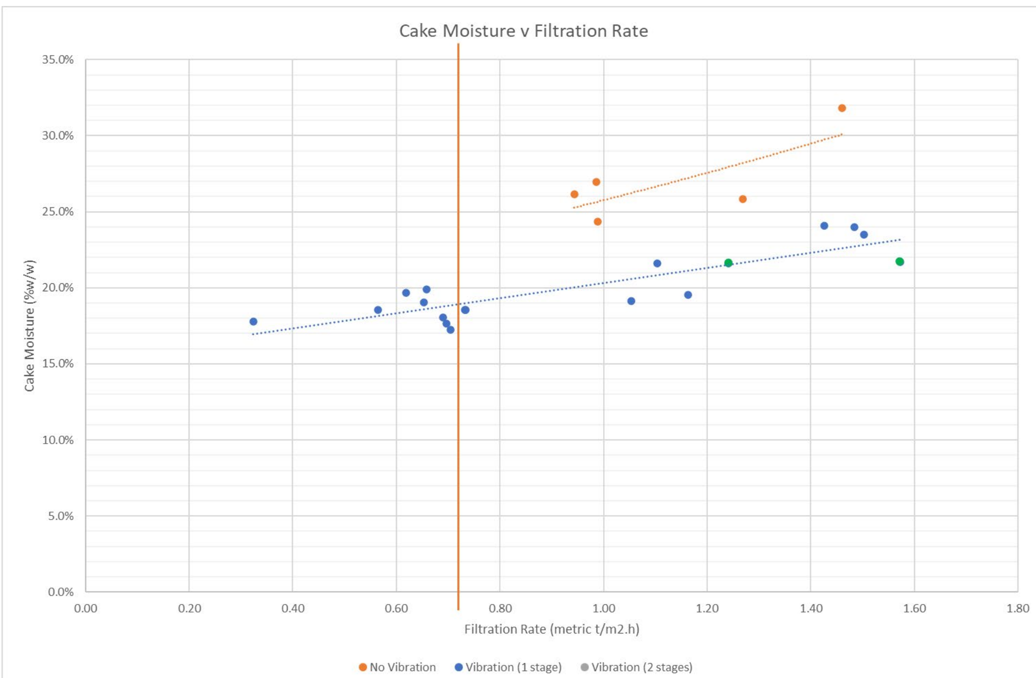


Figure 30 Jord Filter Results

Test No.	4	5	6	7	12	13	15	16	17	18	19	20
Sample												
Feed Solids % w/w (exc. flocculant)	64%	64%	55%	55%	55%	55%	55%	55%	55%	55%	55%	55%
Filter Cloth	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262
Solids in Filtrate (% w/w solids)	3.90%											
Cake Thickness (mm)	8.0	12.0	8.0	9.0	10.0	12.0	8.0	8.0	8.0	8.0	8.0	8.0
Floc Addition [M5250] - (g/metric t [solids])		20		12							30	30
Floc Addition [M10] - (g/metric t [solids])					24	36	44	59	59	59		
Vacuum Pressure (mbar abs)	300	300	300	300	300	300	300	300	300	300	300	300
Form / Pre-Vibration Time (s)	40	30	120	65	60	60	50	35	35	35	40	40
Vibration (Y/N, if Y [intensity L/H])	Y [1H]	Y [1H]	Y [1H]	Y [1H]	Y [1H]	Y [1H]	Y [1H]	Y [1L]	Y [1H]	Y [1L]	Y [1L]	Y [1H]
Drying, including vibration (s)	30	30	30	30	30	0	20	30	40	30	30	30
Total Time (s)	70	60	150	95	90	60	70	65	75	65	70	70
Total Time (min)	1.17	1.00	2.50	1.58	1.50	1.00	1.17	1.08	1.25	1.08	1.17	1.17
Cake Moisture (% w/w metallurgical)	17.7%	19.5%	17.8%	18.6%	19.1%	21.6%	17.2%	18.6%	19.7%	18.6%	18.1%	19.9%
Cake S.G.	2.06	2.01	2.05	2.03	2.02	1.96	2.07	2.03	2.00	2.03	2.05	2.00
Filtration Rate (metric t [solids]/m2.h)	0.70	1.16	0.32	0.56	0.65	1.10	0.70	0.73	0.62	0.73	0.69	0.66
Single Filter Throughput (metric t/h), 182m2 filter	127	212	59	103	119	201	128	133	113	133	125	120
Filtration rate kg D.S/m2.h	696	1163	324	564	654	1104	704	733	618	733	690	658

Test No.	21	22	23	24	25	26	28	29	30	31	32
Sample											
Feed Solids % w/w (exc. flocculant)	60%	60%	60%	60%	60%	60%	60%	55%	55%	55%	55%
Filter Cloth	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262	E30262
Solids in Filtrate (% w/w solids)	1.21%										
Cake Thickness (mm)	14.0	16.0	12.0	13.0	11.0	10.0	9.0	9.5	10.0	12.5	10.0
Floc Addition [M5250] - (g/metric t [solids])	27	39	40	36	48	48	48	44	52	52	52
Floc Addition [M10] - (g/metric t [solids])											
Vacuum Pressure (mbar abs)	300	300	300	300	300	300	300	300	300	300	300
Form / Pre-Vibration Time (s)	55	63	30	50	20	20	20	38	15	15	15
Vibration (Y/N, if Y [intensity L/H])	N	N	N	N	Y [1LL]	Y [1L]	Y [2L-H]	Y [1L]	Y [1L]	N	Y [2L-H]
Drying, including vibration (s)	15	12	15	15	20	15	20	15	20	20	20
Total Time (s)	70	75	45	65	40	35	40	53	35	35	35
Total Time (min)	1.17	1.25	0.75	1.08	0.67	0.58	0.67	0.88	0.58	0.58	0.58
Cake Moisture (% w/w metallurgical)	24.3%	27.0%	25.8%	26.2%	24.1%	23.5%	21.6%	19.1%	24.0%	31.8%	21.7%
Cake S.G.	1.81	1.76	1.78	1.78	1.90	1.91	1.95	2.02	1.90	1.66	1.95
Filtration Rate (metric t [solids]/m2.h)	0.99	0.99	1.27	0.94	1.43	1.50	1.24	1.05	1.48	1.46	1.57
Single Filter Throughput (metric t/h), 182m2 filter	180	180	231	172	260	273	226	192	270	266	286
Filtration rate kg D.S/m2.h	988	986	1269	944	1426	1503	1241	1053	1484	1460	1572

Table 38 Jord Filtration Results (HVBF)

10.4.14.2 *Metso – Pressure Filtration*

Metso conducted a series of pressure filtration tests simulating a FFP style of pressure filter. The feed density used by the vendor varied between 51.45 and 52.65 % solids. It was noted in discussion on the test work that they did struggle to get the density higher than the 52.65% solids. Measurement of the PSD was undertaken by both Jord and Metso with almost an identical result, so it is not clear as to why the density of 55% solids was not achievable. Lime was used to adjust pH in line with leach tails expectations.

Pressure Filtration							
Description	Unit	T1	T2	T3	T4	T5	T6
Filter		Mito S610	Mito S610	Mito S610	Mito S610	Mito S610	Mito S610
Test Unit		Labox 100	Labox 100	Labox 100	Labox 100	Labox 100	Labox 100
Feed Density	% w/w	51.45	51.45	51.45	52.62	52.62	52.62
Chamber Depth	mm	45	45	45	45	45	45
Cycle Time	Min	12	10.5	9	10.5	11	10
Pumping Time	min	4.5	4.5	4.5	2.5	3	2.5
Pressing time	min	0.5	0.5	0.5	1	1	1
Air drying time	min	3	1.5	0	3	3	2.5
Pumping Pressure	Bar	6	6	6	6	6	6
Air Drying	Bar	10	10	10	10	9	10
Cake Thickness	mm	44.6	43.6	44.7	37.8	39.8	44
Cake Moisture	%w/w	17.11	17.07	18.74	14.72	15.89	Awaiting
Filtration Rate D.S	kg/m^2.h	175.7	192.9	236.7	183.9	180	TBC

Table 39 Metso Pressure Filtration

Moisture versus drying time was examined by the Vendor as part of the overall cycle to get the cake moisture of 15% solids.

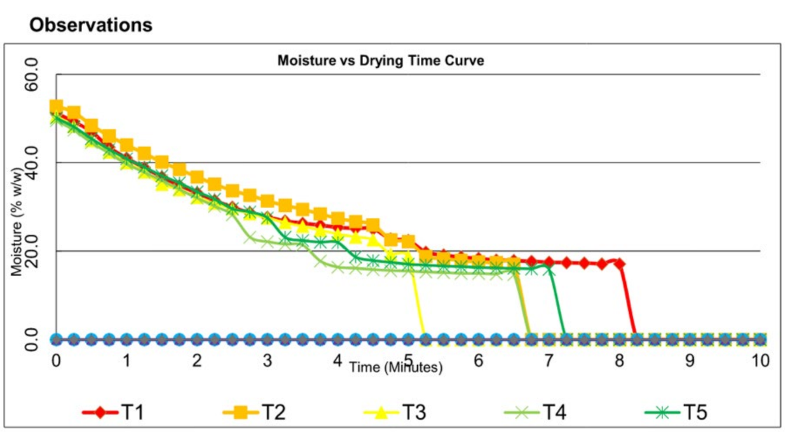


Figure 31 Moisture vs Drying Time

Cycle times for drying can be limited to 4 minutes with limited benefit from extended time past this point. This will reduce the overall cycle times required for the FFP unit. The full-scale unit should be able to operate at 6 cycles per hour (10 minutes per cycle). Test 3 at 18.74% moisture and 236 kg D.S/m^2.h relates to 1059 m^2 or required filter area while processing 250 t/h of ore. An FFP3512 62/74 M45 will provide 832 m^2 with expanded to 992 m^2 by two operating machines. Filters being sized on 85.7% filter plant time increase the duty to 292 t/h feed rate or 1228 m^2 required out of 1664 m^2 (74%) of filtering capacity.

10.4.14.3 Metso – Vacuum Filtration Results

Metso ran a similar Buchner funnel test with no vibration, flocculent 913 SH and Aluminium Hydroxide. They also ran a staged vibration test passing the cake across several vibration stages. They used a much finer cloth Maro S40 which gave clearer filtrates at the cost of both moisture and filtration rates. Filtration with no flocculant gave filter rates >300 kg D.S/m^2.h, averaging 220 kg D.S/m^2.h at 22.25% moisture. This was much lower than the Jord results, although the particle size distribution measured was almost identical.

Improvements using flocculent 913 SH increased the filtration rates in Table 39 to ~450 kg D.S/m2.h, but this suffered in terms of moisture being >25% and hence not transportable. The same fine cloth Maro S40 was used in the test work. The addition of Aluminium Hydroxide did not show any net gain over the as received samples for moisture but gave a slightly higher filtration rate over the as received.

Run	Units	1	2	3	4	5	6	7	8	9	10
Feed Solids	%w/w	52.40%	52.40%	52.40%	52.40%	52.40%	52.40%	52.40%	52.40%	52.40%	52.40%
Filter Cloth		Maro S60	Maro S40	Maro S30	Maro S40	Maro S40	Maro S40	Maro S40	Maro S40	Maro S40	Maro S40
Total Cycle Time	s	174	176	198	367	291	400	407	429	463	393
Thickness	mm	7.4	8.9	9.2	14.8	12.1	16.4	14.9	14.6	14.5	14.4
Cake Moisture	% w/w	21.7	22.3	21.1	22.4	23.5	23.5	22.7	21.4	21.4	22.5
Filtration rate	kg D.S/m^2.h	269	292	257	218	227	206	196	179	167	201

Table 40 As Received Filtration Tests

$Al(OH)_3$ added	Units	1	2	3	4	5
Feed Solids	%w/w	52.40%	52.40%	52.40%	52.40%	52.40%
Filter Cloth		Maro S40	Maro S40	Maro S40	Maro S40	Maro S40
Total Cycle Time	s	318	333	256	235	240
Thickness	mm	14.2	13.9	13.9	13.9	13.7
Cake Moisture	% w/w	21.5	21.4	21.5	21.5	21.5
Filtration rate	kg D.S/m2.h	243	231	303	326	323

Table 41 Aluminium Hydroxide Test Results

The vibration testing (Table 42) was done sequentially with a steady improvement in cake moisture at the cost of filtration rate.

Filter Aid + Vibration	Units	1	2	3	4	5
Feed Solids	%w/w	52.40%	52.40%	52.40%	52.40%	52.40%
Vibration		No	1st Stage	2nd Stage	3rd Stage	4th Stage
Filter Cloth		Maro S40	Maro S40	Maro S40	Maro S40	Maro S40
Total Cycle Time	s	368	465	550	625	689
Thickness	mm	12	12	12	12	12
Cake Moisture	% w/w	22.2	17.1	15.2	14.3	13.5
Filtration rate	kg D.S/m^2.h	212	168	142	125	113

Table 42 Sequential Vibration Stages

Floc 913 SH	Units	1	2	3	4
Feed Solids	%w/w	52.4	52.4	52.4	52.4
Filter Cloth		Maro S40	Maro S40	Maro S40	Maro S40
Total Cycle Time	s	171	159	95	173
Thickness	mm	14.5	15.5	17.2	17.8
Cake Moisture	% w/w	24.8	25.9	32	33.7
Filtration rate	kg D.S/m^2.h	446	473	786	430

Table 43 Flocculent 913 SH Results

10.4.14.4 Ishigaki Filter Tests

Ishigaki conducted seven (7) tests to determine a filtration rate on the same material supplied to each vendor. The tests were run at the as received sample at 37.2% as such were too low compared to the plant design. The bulk of the tests conducted used cake blow at varying cake thicknesses, based on 30, 40 and 50 mm chamber depths.

		FEED			CAKE BLOW						
Test	Chamber Thickness (mm)	Time (min:sec)	Filtrate (g)	Pressure (Mpa)	Time (min:sec)	Filtrate (g)	Blow Pressure (Mpa)	Cake Thickness (mm)	% Cake Mositure	Filtration Rate kg/m^2.h (Misc excluded)	Filtration Rate kg/m^2.h (Misc Included)
2	30	7	2650	0.7	1.50	250	0.6	30	15.7	134.5	131.29
3	40	8	3050	0.7	1.50	150	0.6	38.5	19.3	165.57	161.82
4	40	8	3200	0.7	1.50	250	0.6	42	19.1	162.18	158.7
5	50	10	4200	0.7	1.50	300	0.6	50	16.4	161.45	158.57
6	50	9	3650	0.7	1.50	250	0.6	50	17.3	177.25	173.8
7	50	6.5	2600	0.7	1.5	300	0.6	50	15.5	249.92	243.57

Table 44 Ishigaki Test Results

Results indicated that filtration rates of ~165 kg D.S/m^2.h could be achieved in the target range of 16 to 19% moisture. The low feed solids used in the test will impact on the feed time and the filtrate rates generated as the material will carry less initial water load. The filter rates are hence not directly comparable to the data generated by Metso. It is evident that the filter will be able to reach the target moisture range and was almost able to reach to original plant target for design of 15% residual cake moisture.

10.5 Process Flowsheet

The process flowsheet development has focussed on a hybrid leach / CIL flowsheet based on the historical test work in combination with the current test work verification program 2025/26 being conducted by Hazen and RDi laboratories. The following key decisions have been made on the process design.

- Simplified crushing circuit design to limit dust creation, maintaining a single stage crushing circuit for SABC comminution circuit design.
- Comminution data supports the decision to use SAB type circuit with the inclusion of a pebble crushing circuit SABC.
- Primary grind size of P_{80} 106µm has been adopted for the design.
- A 24-hour leach time will be sufficient for the recovery of gold with additional capacity to 27 hours to assist with improved silver recovery. A CIL circuit was adopted early in the design based on the historical plant design basis. The current design test work supports either leach/CIP or a CIL design. It is noted that the ore is not considered to be preg-robbing with low organic carbon assay levels.
- Provision for oxygen injection into leach tanks 1 and 2 has been allowed based on historical production reports where low dissolved oxygen levels had affected the recovery of gold. Initial bottle roll tests on composites 4 and 5 both indicated initial low DO_2 (2-3ppm) at time zero (post grind), indicating that some of the oxygen had been consumed in the laboratory milling process. This was restored in the bottle roll which included O_2 injection and measurements. The design has allowed for a 3 tpd PSA plant.
- INCO detox used previously has been maintained in the current flowsheet. If required some of the O2 generated by the PSA plant can be redirected to the detox system. The default will be AIR/SO_2 system using SMBS as the supply reagent for SO_2 delivery.
- Cyanide will be supplied by a Isotank dissolution system via Orica Chemicals – this will remove personnel from the mixing of cyanide and the dangers associated with the reagent.

11. MINERAL RESOURCE ESTIMATES

11.1 Introduction

The nature, grade and distribution of the drilling data lend itself to modelling unconstrained 1m composited gold grades via a non-linear grade interpolation technique. Multiple Indicator Kriging (MIK) is HSC's choice of modelling for Colosseum using the GS3M software to produce a recoverable block model that is loaded into a Surpac block model for block model validation and recoverable Mineral Resource reporting.

Since the early-1980s, MIK has become a frequently applied non-linear geostatistical estimation method. It is especially widely applied in Australia by both consulting companies and in-house resource estimation teams.

MIK differs from Ordinary Kriging (OK) in that it specifically models continuity of grade at different grade thresholds. The block model products are recoverable resources (the portion of the in-situ resource that can be selected as ore during mining) and E-Type estimates (the latter more akin with an OK modelling scheme with average block grades).

Completing the grade interpolation using a linear modelling method like Ordinary Kriging is considered a higher risk. The GS3M software was developed by Neil Schofield (Stanford University) in the late 1990s, specifically targeting resource estimation for open pit gold deposits. Its sister software MP3, using the same non-linear process and algorithms is commercially sold to mining companies to assist with grade control modelling.

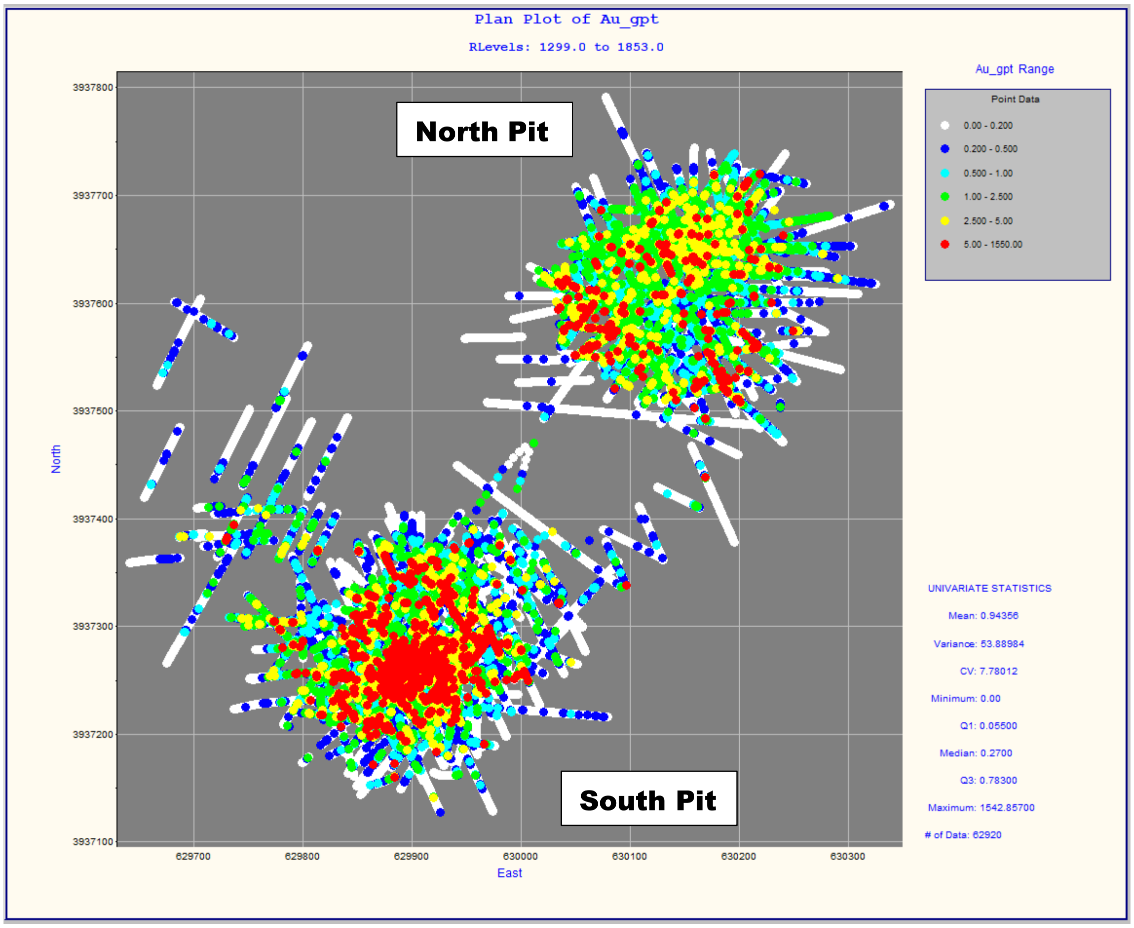
11.2 Geologic Modelling of the Colosseum Deposit

The gold mineralisation comprises disseminated auriferous pyrite hosted by a combination of felsite dyke intrusion, felsite breccias, sedimentary breccias and altered granite. Mineralisation is diffuse and not hosted exclusively by a particular rock type. The gold is associated with pyrite grain margins and fractures and is not refractory. There is no obvious visible lithological or structural control to the gold mineralisation, save for a broad NE/SW-striking enriched zone, presumably some form of structural corridor related to the felsite intrusions.

No geological interpretation per se for the mineralisation has been completed as the gold grades define the gold mineralisation in the various host rocks. Any wireframe for the gold mineralisation would ultimately be a simple grade shell with no geological control.

Lithological units were delineated for the felsite/felsite breccia, sedimentary breccia and granite to provide a measure of the gold grade distribution within the various lithologies. No oxidation surface was created due to a lack of logging data.

Figure 32 provides a plan view of the gold composite grade and distribution from the drilling for the South Pit and the North Pit.



Source: Prepared by H&S Consultants, 2026

Figure 32 Colosseum Gold Composite Distribution Plan View

11.3 Block Model Parameters

The Mineral Resources have an 800m by 800m surface extent comprising two separate bodies roughly 200m by 200m. The mineralisation is exposed at surface, and the Mineral Resources continue to a depth of approximately 300m below surface at an RL of approximately 1410m.

The lower limit to the Mineral Resource is an arbitrary one being the result of a supplied revenue factor 1 pit shell run at US$5000/oz from a pit optimisation study. The mineralisation is open at depth and laterally to the southeast, beyond the North Pit zone.

Block dimensions are 10m by 10m by 5m (E, N, RL respectively) with no sub-blocking. The selective mining unit (SMU) is 5m by 5m by 2.5m. The north and east dimensions were chosen as they are close to the nominal drillhole distances in the detailed drilled area of the South Pit. The vertical dimension was chosen as a compromise between the two deposits, a reflection of the sample spacing, possible mining bench heights and to allow for flexibility in potential mining scenarios after discussions with independent mining consultants Australian Mine Design and Development (AMDAD).

Table 43 contains spatial details of the block model.

Block Model: col_mik_v4cm_working_170326.mdl			
Colosseum MIK Model Mean_Median_compromise			
Type	X	Y	Z
Minimum Co-ordinates	629500	3936900	1300
Maximum Co-ordinates	630500	3938000	1900
User Block Size	10	10	5
Min. Block Size	10	10	5
Rotation	0	0	0

Table 45 Block Model Details

11.4 Estimation

Recoverable Multiple Indicator Kriging (MIK) was used to complete the gold grade estimation using the GS3M modelling software. The geological interpretation, such as it is, block model creation and validation were completed using the Surpac mining software. HSC considers MIK to be an appropriate estimation technique for the type of mineralisation and extent of data available.

The drillhole database was composited, with no constraints, to 1m intervals covering the whole of the prospect. The 1m composite interval may lead to a smoothing out of the variance but is unlikely to have a significant impact on the global estimates. A minor amount of peripheral, isolated data was removed from the composite file. A total of 62,920 composites were generated from the drillhole database, using the Surpac 'best fit' option and modelled for gold only. Two drilling domains were employed, one for the South Pit (domain 1 with 36,589 data and a coefficient of variation of 8.8) and another for the North Pit (domain 2 with 26,331 data and a coefficient of variation of 3), reflecting a difference in intensity of drilling and assay grades.

Metal variogram maps of gold for domains 1 and 2 indicated weak results which points to a lack of structure to the gold data. Overall grade continuity was very modest with a weak E-W trend for domain 1 coupled with a steeply west plunging feature in the XZ plane and a vertical plunge in the YZ plane. For domain 2 a WNW trend was interpreted with a subvertical plunge in both the XZ and ZY planes.

Grade interpolation was unconstrained, except by the search parameters and the variography, in acknowledgement of the gradational nature to the margins of the gold mineralisation and the abundance of buffering low grade peripheral values.

No base of oxidation was used due to limited data and the impact of previous open pit mining. No cover surface was created as the mineralisation is outcropping and is exposed in many places along its ridge line and flanks and where previous open pit mining had occurred.

A fundamental concept behind MIK method is that it generally precludes the need for top cutting. However, a review of the conditional statistics for the top indicator class for both domains indicated a significant difference between the mean and the median. As a result, an average value for the mean and median was used as the gold grade for the top indicator class for both domains.

Both domains were modelled as a combined dataset with soft boundaries and separate conditional statistics. A total of 5 search passes were employed with progressively larger radii and/or decreasing data point criteria. The initial search parameters for domain 1 were 20m by 20m by 35m with a minimum of 16 data and 4 octants increasing to a final Pass 5 search of 60m by 60m by 120m with a minimum of 8 data and 2 octants. For domain 2 the initial search was 25m by 25m by 25m with the same domain 1 data requirements expanding to a Pass 5 search of 70m by 70m by 70m with a minimum of 8 data and 2 octants. The slightly different search dimensions are a function of the drill spacing and mineralisation in each pit.

The maximum extrapolation for the Mineral Resources is the Pass 5 search.

No other elements were modelled therefore there are no assumptions about correlation between variables. No by-products are anticipated from production (silver was historically not assayed for) and no assessment has been made for any deleterious elements.

Drillhole spacing ranges from 10 to 20m in the core of the two domains but at a variety of directions giving rise locally to relatively close spaced samples. Downhole sampling was generally at 5' (and 2') intervals.

The resource estimates are controlled by the data point distribution, the variography, block size and the search ellipse parameters. Conventional use of wireframes to control the mineralisation was not considered necessary in this case.

The new block model was reviewed visually by HSC, and it was concluded that the block model fairly represents the grades observed in the drill holes. HSC also validated the block model using a variety of summary statistics and statistical plots. Validation confirmed the modelling strategy as acceptable with no significant issues.

Tonnages are estimated on a dry weight basis, and moisture content has not been determined. A default density of 2.65t/m^3 based on 122 core samples was used to estimate tonnages.

The historic mining operation exploited both the South and North Pits but there are no meaningful production figures available to allow for any reconciliation with the new Mineral Resources.

11.5 Mineral Resources

The classification of the Mineral Resources is based on the pass number derived from the grade interpolation with qualitative consideration of other aspects including drill spacing, variography, density measurements, sampling method & sample recovery, QAQC data and the geological model. Pass 1 is converted to Measured Resource, Passes 2 and 3 are converted to Indicated Resource and Passes 4 and 5 are converted to Inferred Resource

Positives for the resource classification:

- Close spaced drilling/sampling
- Appropriate sampling and assaying methods
- Relatively simple geological scenario and straightforward geological logging
- Sample recoveries for the drilling indicate no issues.
- Previous mining indicates good gold recoveries are achievable.
- DD Twin holes confirming 2025 RC drilling
- Multiple Indicator Kriging grade interpolation technique,

Negatives for the resource classification

- Lack of density data, although the density range is relatively small.
- Possible sample representivity with wet RC samples, slight under-reporting from the RC samples
- Uncertainty with the historical depletion preventing reconciliation.

Mineral Resources are reported as recoverable estimates for a 0.2 g/t gold cut off within the planned pit shell developed by AMDAD (Table 44). In comparison with the 2024 estimates, there is a significant increase in tonnes and a corresponding drop in gold grade leading to a <1% reduction in gold ounces. Increase in size and reduction of gold grade is due to using a lower gold cut-off grade i.e. 0.2 g/t cf 0.5 g/t. A small subset of peripheral high grade block values with very limited drill support have been excluded from the estimates.

	Category	Mt	Au g/t	Au Moz
South Pit	Measured	5.49	1.33	0.23
	Indicated	5.54	0.67	0.12
	Inferred	4.72	0.62	0.09
Sub Total		**15.76**	**0.88**	**0.45**
North Pit	Measured	11.96	0.85	0.33
	Indicated	9.09	0.59	0.17
	Inferred	7.69	0.53	0.13
Sub Total		**28.75**	**0.68**	**0.63**
Total		**44.50**	**0.76**	**1.08**
Combined	**Category**	**Mt**	**Au g/t**	**Au Moz**
	Measured	17.5	1.00	0.56
	Indicated	14.6	0.62	0.29
	Total	**32.1**	**0.83**	**0.85**
	Inferred	12.4	0.57	0.23

Table 46 Recoverable Mineral Resources at a 0.2g/t Au Cut-off

A graphic representation of the E-type Mineral Resources for Colosseum is shown in Figure 33 (graphic representation of recoverable resources is not easily done, hence the E-type results are used).

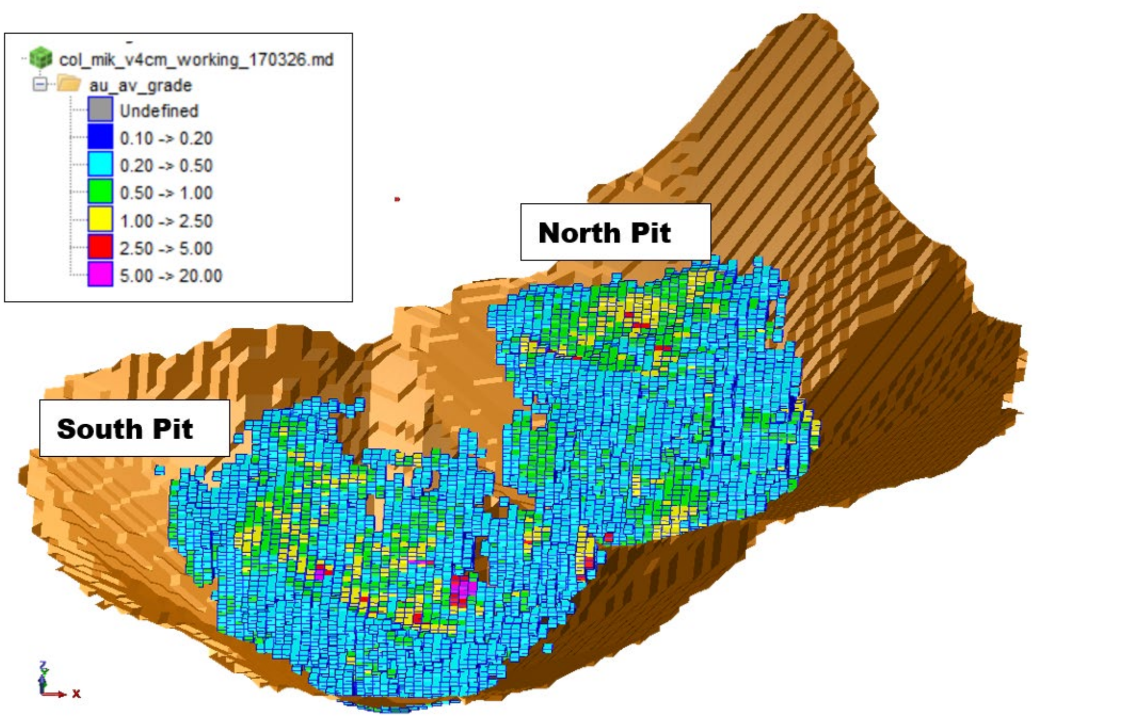


Figure 33 Colosseum Mineral Resources Oblique View

A graphic representation of the gold grade tonnage data for the E-type estimates within the pit shell is included as Figure 34.

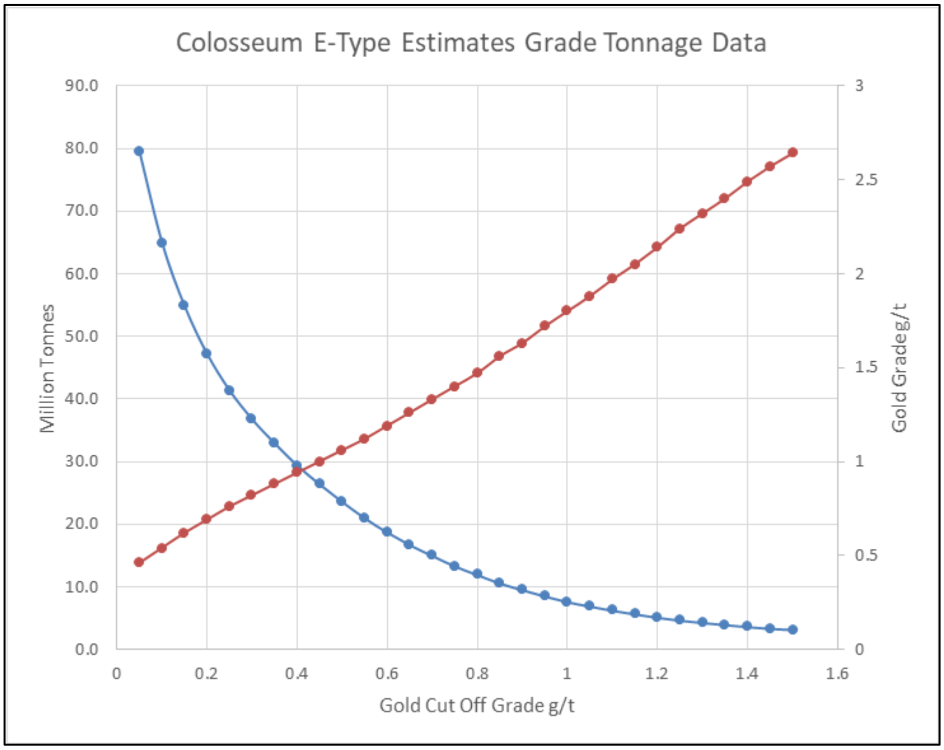


Figure 34 Gold E-type Grade Tonnage Data

11.6 Relevant Factors Affecting Resource Estimates

11.6.1 Multiple Indicator Kriging ("MIK")

Coefficients of variation (CV = standard deviation/mean) for the gold composites are relatively high for the deposit, 8.8 for Domain 1 and 3 for Domain 2, which indicates potentially skewed data and/or extreme values. This would generally imply non-linear data and therefore a non-linear, more sophisticated grade interpolation technique, like MIK, would be more appropriate. The success of the method relies on a large composite dataset, the operator's ability to handle multiple indicator variograms, the block size and the search ellipse parameters. Non-linear modelling provides increased confidence in the estimates.

11.6.2 Geological Interpretation

Conventional resource modelling would invoke the use of mineral wireframes for constraining the grade interpolation. However, the lack of lithological control, the lack of any structural control and the diffuse nature of the gold mineralisation would mean any wireframe at a nominal gold cut off would simply be a grade shell with not necessarily any geological context. The large dataset with extensive low-grade data buffering the margins of the deposit means, in HSC's experience that unconstrained modelling of the composite data is appropriate and allows the data to control the grade interpolation. The idea of unconstrained models is not unusual to HSC. Allowing the data to control block grade distribution without the bias effect of wireframes provides increased confidence in the estimates.

11.6.3 Drillhole Spacing and Direction

The drilling comprises vertical and steeply angled holes with various orientations that to some extent criss-cross the deposit giving a nominal 20 by 20m horizontal drill spacing. This has created a relatively unbiased drilling dataset that is well suited to MIK. The varied orientations of the drilling and the potential for a lack of bias provides increased confidence in the estimates.

11.6.4 Density Data

The number of density samples is limited to 122 however the range is relatively small, and the host rock variations are relatively small. Hence the use of a default density i.e. 2.65t/m^3 is appropriate and compatible with a felsic/granitic host unit with minor amount of pyrite mineralisation. The lack of density data has only a very limited impact of the confidence in the estimates.

11.6.5 Relatively Small Block Size

Ideally the maximum block size for an MIK model is the drillhole spacing i.e. in this case 20m. However, with the relatively localised close spaced drilling and the large composite data set a slightly smaller block size 10m by 10m by 5m was used with an SMU of 5m by 5m by 2.5m. It should be noted that often a small block size can lead to over-smoothing of grades and thus an over-statement of grade for the Mineral Resource. The selected block size for Colosseum is a compromise of competing factors listed above but has only a very limited impact of the confidence in the estimates.

11.6.6 Use of Top Cuts for Gold

This issue promotes substantial debate, but HSC would like to point out that there is no scientific or mathematic basis for selecting a top cut. One of the key components of MIK modelling is that there is generally no need for top cuts to the composite data and thus the original data is left untouched. To counter potential overstatement of grade the mean and median of the top indicator class (from the conditional statistics) for each mineral domain is reviewed. Where there is a significant difference between the two it is prudent to consider using a compromise grade value for the top indicator class in the MIK modelling. This compromise in this case is the average value for the mean (the most optimistic value) and the median (potentially too pessimistic a value). Use of the compromise averaged value of the mean and median for the top indicator class is effectively a top cut without affecting the original data and is considered a prudent action and thereby increasing the confidence in the estimates.

11.6.7 Minimum Number of Data

The use of 16 minimum number of composite data for the grade interpolation in conjunction with a minimum of 4 octants ensures at least two drillholes are used in the block grade assignment. HSC has kept the minimum number of data for the bigger search passes relatively high at 8 (and a minimum of 2 octants), this reduces the potential of overstatement of gold block grades on the periphery (Inferred Resource) part of the deposit. Using a lower number of minimum data i.e. <8, invites an increase in risk to the interpolated grades particular at the margins of the lodes or in areas of wide drillhole spacing. The relatively high number (compared to standard industry practice) of 8 minimum number of data helps to limit the impact of peripheral higher grades and thus increase confidence in the estimates.

11.6.8 Variography

This provides a measure of the grade continuity of the composites and the weighting required when undertaking block grade interpolation. Good variography is often dependent on close spaced drilling, however in this case the variography showed very limited grade continuity, further supporting the use of MIK as a modelling method and thus adding to the confidence in the estimates.

11.6.9 Sample Recovery

The 2025 RC drilling encountered the water table in 40% of the holes, and within those holes the bottom third produced wet samples, approximately 12% of the total samples. The wet samples generally had lower recoveries and potentially could reduce the confidence and hence the classification of the Mineral Resources. However twin hole drilling and analysis by Dateline indicated that there was only a minor under-reporting of the gold grade with the RC sampling. Sample recovery for the core drilling was close to 99% (excluding top of hole lower recoveries) and therefore the wet RC samples impacted slightly on the deliberations for the resource classification. The incidence of wet samples for the recent drilling has a slight negative impact on the confidence of the estimates, this negative impact has been reduced by the positive outcomes of Dateline core hole twinning and the lack of reported wet samples in the historic drilling.

12. MINERAL RESERVE ESTIMATE

12.1 Introduction

As part of the Feasibility Study (FS), Australian Mine Design and Development (AMDAD) prepared a Mineral Reserve Estimate for open cut mining of the Colosseum gold deposit. AMDAD prepared the Mineral Reserve Estimate in accordance with The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 Edition (the 'JORC Code 2012'). The Mineral Reserve Estimate also accords with the SME Guide 2017. The Competent Person signing off on the Mineral Reserve Estimate is John Wyche.

The reference point for the Mineral Reserve Estimate is for ore delivered to the run of mine (ROM) crusher and stockpile.

The methodology, key assumptions, cutoff grade and reported Mineral Reserve Estimate are provided below.

12.2 Methodology

Key steps completed in the Mineral Reserve estimation are outlined below:

- Pit optimisation was first undertaken by AMDAD based on the Measured and Indicated Resources of the updated Mineral Resource Estimate prepared by H&S Consultants (HSC). Using a mining block model prepared from the Mineral Resource Estimate, the optimisation defined a simple computer-generated open pit shell that maximises the discounted operating cashflow based on the key project mining, processing and economic assumptions.

- An important aspect of the optimisation block model is that it preserves the fundamental elements of the multiple indicator kriged (MIK) mineral resource model. The optimisation block model also incorporates dilution adjustment on a block-by-block basis.

- Guided by the results of the pit optimisation, AMDAD then prepared a practical open cut design, with ramps and berms. The practical open cut design is a staged design incorporating a northern starter pit and pushback to the final pit.

- A waste rock and tailings co-disposal facility was also designed, to accommodate the volume of waste rock generated from mining the open cut and dewatered tailings output from the processing plant.

- Monthly life of mine scheduling was then completed by AMDAD, defining the key mining quantities as the basis for mining costs as well as ore production for processing costs, G&A costs and revenue.

- Financial modelling was completed using the life of mine schedule, operating and capital costs, to confirm the project economic viability.

12.3 Mining Block Model

12.3.1 Resource Block Model

The Mineral Resource Estimate (MRE) block model that underpins the mining block model and FS mine plan was prepared by HSC, as described in Part 11 - Mineral Resource Estimates. HSC used recoverable Multiple Indicator Kriging (MIK) to complete the gold grade estimation. Silver is not recorded in most of the drill hole assays and is not included in the mineral resource model, pit optimisation and Mineral Reserve Estimate. Block dimensions are 10m by 10m by 5m (E, N, RL respectively) with no sub-blocking and no rotation of blocks. The MIK modelling assumes a selective mining unit (SMU) is 5m by 5m by 2.5m.

The key fields of the MIK block model for the mining model, optimisation and Mineral Reserve Estimate are:

- Block proportion above cutoff, “au_p_{cutoff}”, for cutoffs in g/t gold of 0.10, 0.15, 0.20, 0.25, 0.30, 0.35, 0.40, 0.45, 0.50, 0.60, 0.70, 0.80, 0.90, 1.00 and 1.25, with the proportion as a decimal value.
- Block grade above cutoff, “au_g_{cutoff}”, in g/t gold.
- Panel average specific gravity, “ave_sg”
- Resource Classification, “resclass”, where 1 = Measured, 2 = Indicated, 3 = Inferred

The FS mine plan only considers the Measured and Indicated components of the MRE.

12.3.2 Topography surface

As well as the surveyed current ground surface, the mining block model and FS mine plan uses pre-mining topography for the site, as shown in the following figure, to adjust for rock fill placed onto the original ground surface during the previous mining operations.

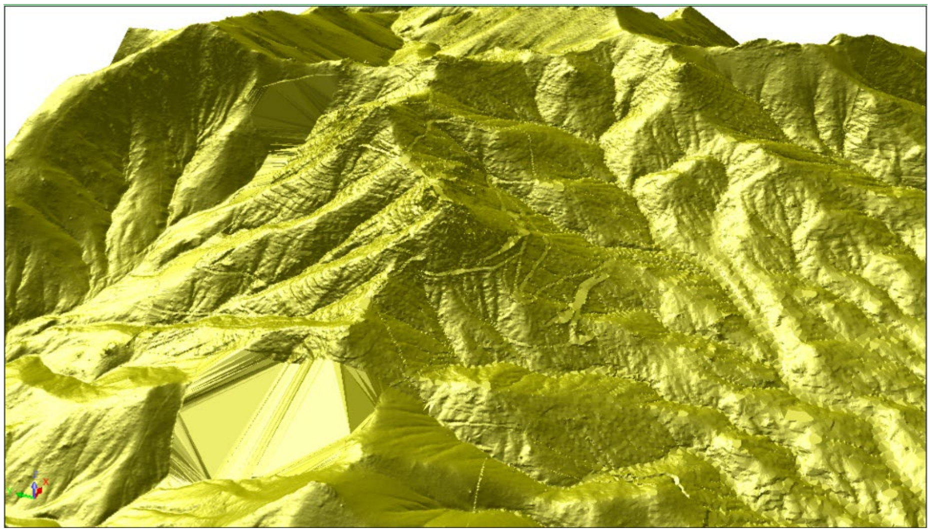


Figure 35 Pre-mining topography surface, topo_utm1983.dtm

The figure below shows the current topographic surface, including waste rock fill, and indicates the depth of previous mining and height of waste rock fill placement.

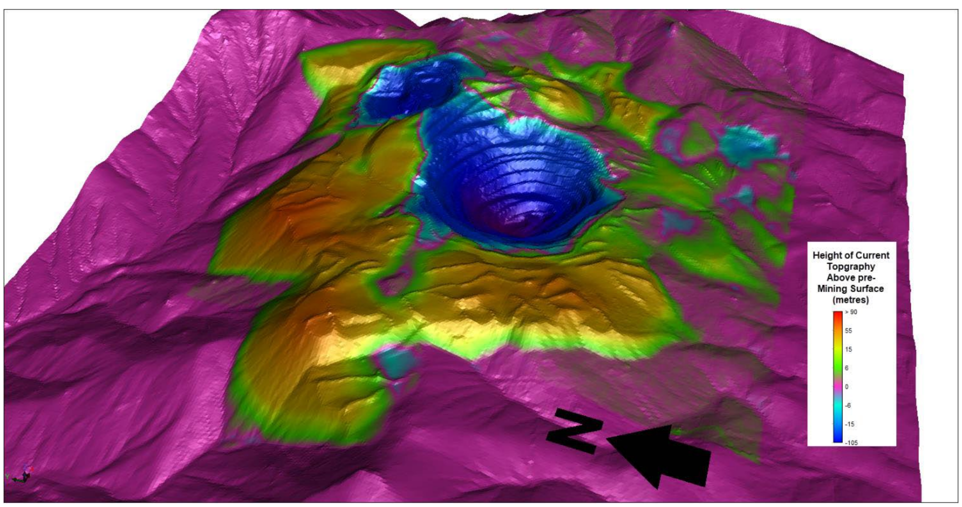


Figure 36 Current surface coloured by height above pre-mining surface

12.4 Mining Dilution and Loss

Dilution from mixing of ore and waste rock at the ore boundaries is incorporated within the mining block model (AMDAD, 2026) by adjustment of the MIK grades and proportions. The concept/model for this adjustment is a 0.5m wide surrounding skin of dilution that is added to that part of the block to be mined as ore. Key assumptions and logic for the adjusting algorithm are:

- The material above a selected MIK cut-off in a block is a notional single square sub-block within the parent block,

- The dimensions of that notional sub-block and the amount of dilution represented by the dilution skin are then geometrically determined by the MIK proportion for that cutoff, and

- The dilution skin for that notional sub-block is assumed to have the average gold grade of the next MIK increment below that of the sub-block. If there is insufficient tonnage in the increment below, then the next increment down is added until the required dilution tonnage is achieved. The resulting tonnage factor is capped so that the block tonnage will be conserved. Each set of MIK grades and proportions is adjusted in turn.

This "onion skin" dilution concept is depicted schematically in the following figure.

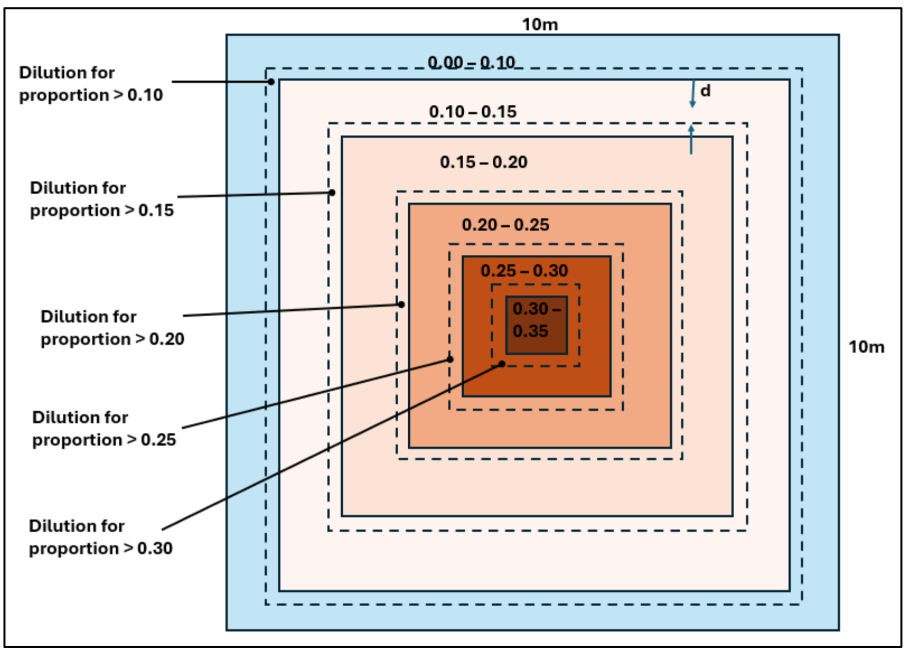


Figure 37 "Onion skin" dilution concept for MIK model

Once the dilution algorithm is run, the diluted MIK proportions and grade are used to construct an optimisation block model that contains this data incrementally. This allows selection of ore components optimised according to the cutoff grade, based on nominated processing and economic assumptions. Lower grade portions of a block will be diluted to a sub-economic grade and will not be selected as ore. In this way the dilution method also results in loss of resource that would have been economic prior to dilution.

In addition to the dilution process described above, AMDAD applied the following percentage adjustments to the mining tonnes and grade:

- 1% additional dilution factor to allow for miscellaneous barren waste material being added to ore from sources such as road sheeting, wall fall-off, over-digging the floor of the ROM pad when rehandling from stockpile, and mistakes in grade control and equipment control.

- 2% mining loss, to allow for excessive movement of blasted ore outside the defined excavation boundaries, as well as mistakes in grade control and equipment control.

12.5 Economic and Processing Assumptions

As part of the pit optimisation, mine plan evaluation and ore selection processes, AMDAD applied a processing recovery and preliminary economic assumptions, and generated indicative cashflows for the project. These assumptions are described and tabulated below. All costs, prices and revenues are in US$.

12.5.1 Processing Recovery

A processing recovery of 91% was applied for the pit optimisation, mine plan evaluation and cutoff grade. This recovery is supported by extensive bottle roll tests across a range of head grades and mineralogies conducted as part of the FS and by historical production from 1987 to 1993. Current leach tests were run at head grades down to 0.5 g/t Au with no apparent reduction in recovery suggesting no impact of fixed tail grades at the design grind size.

12.5.2 Operating and Selling Costs

Operating and selling costs applied in the pit optimisation process are summarised in the table below.

Mining operating costs are based on U.S. mining contractor budget prices. These costs incorporate mining equipment ownership and running costs, fuel costs, consumables, contractor personnel, overheads and profit margin, grade control, Dateline's mining management and technical team and mine area environmental management. The mining operating costs vary by depth and are assigned accordingly within the mining block model. The average mining cost applied for the pit optimisation was USD4.55/t mined.

The final FS cost estimates, including capital costs, are provided in Part 18 - Capital and Operating Costs. Those final cost estimates are applied in the financial modelling (Part 19 – Economic Analysis) that confirms the economic viability of the Mineral Reserves and the project.

Cost Item		Assumed Value	Source
Operating Costs	Mining cost	Variable by bench	AMDAD
	Additional ore cost	USD0.28/t ore	AMDAD
	Crusher feed cost	USD0.51/t ore	AMDAD
	Processing cost	USD19.75/t ore	GRES
	Tailings management	USD3.60/t ore	AMDAD

Cost Item		Assumed Value	Source
	Site G&A cost	USD3.80/t ore	GRES/Dateline
	Subtotal processing/ore costs	USD27.94/t ore	
Selling cost	Refining	USD2.50/oz gold	CostMine US database examples
	Payable percentage	99%	
	Dore transport & Insurance	USD1.00/oz gold	AMDAD
	Royalty	2.5%	Dateline
	Subtotal selling costs	USD149.36/oz gold	
DCF	Discount rate	6.5%	Dateline

Table 47 Preliminary economic and processing inputs for the FS mine plan

Mining costs are defined in the mining block model as ore mining costs and waste mining costs. For the pit optimisation process the ore mining cost premium above the waste mining cost (additional ore cost) is included in the processing/ore cost and is thereby considered in the cut-off grade and ore selection. The additional ore mining cost is assigned in the mining block model as positional processing cost adjustment factors (pcafs).

12.5.3 Gold Price

AMDAD applied a gold price of USD4,200/oz, nominated by Dateline. This is approximately a 15% discount to the average spot price through first quarter of 2026.

12.6 Open Pit Wall Slopes

Open cut geotechnical wall slope designs were modelled by Agapito Associates (2025) based on historical geotechnical reports, structural mapping of the currently exposed pit walls, geotechnical logging and rock strength testing for six core holes in the 2025 drill program and numerical modelling of the data gathered. Agapito defined 12 geotechnical domains shown in the following figure – N1 to N5 in the North Pit and S1 to S7 in the South Pit. For each domain, Agapito proposed wall face angles and bench (berm) width and corresponding Inter Ramp Angles (IRAs) to achieve a minimum factor of safety of 1.2. These slope geometries are listed in the following table. For the overall wall slopes to be used in the pit optimisation AMDAD flattened off the IRAs to take account of in-wall haul roads (ramps).

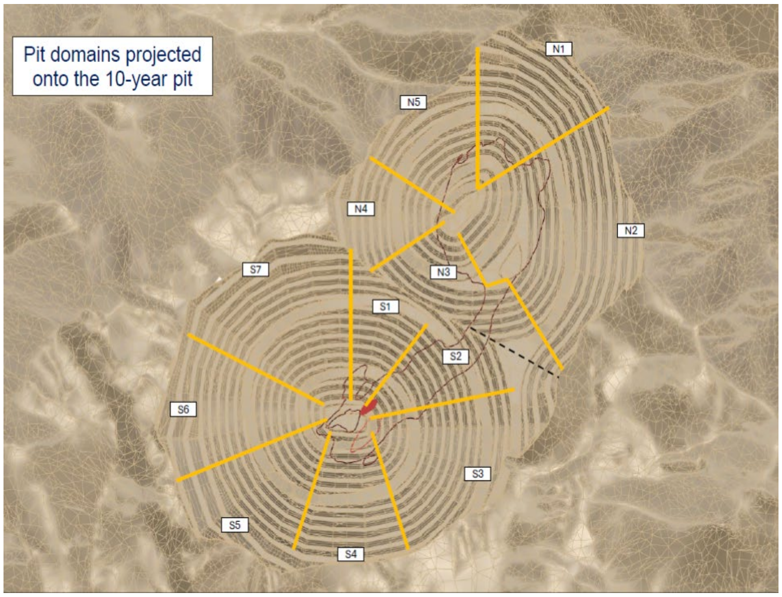


Figure 38 Plan view showing geotechnical domains defined by Agapito

Domain	Critical Failure Mode	Bench Face Angle (°)	Maximum Inter-ramp Angle (°)
N1	Wedge	57.0	43.2
N2	Planar	50.0	38.5
N3	Both	59.0	44.5
N4	Wedge	62.0	56.5
N5	Both	60.0	45.2
S1	Wedge	60.0	45.2
S2	Wedge	57.0	43.2
S3	Planar	50.0	38.5
S4	Wedge	54.0	41.2
S5	Both	60.0	45.2
S6	Wedge	62.0	46.5
S7	Both	64.0	47.9

Table 48 Geotechnical wall design parameters

AMDAD assigned the geotechnical domains in the mining block model to facilitate definition of slopes in the pit optimisation and for the practical pit design process.

12.7 Rock Type

Rock types are defined in the mining block model for the pit optimisation and scheduling:

- In addition to barren waste rock, existing broken fill material from the previous mining operation is defined with an in situ dry bulk density of 2.04 t/m^3.
- Mineralised blocks are defined as MN1, MN2, MN3 according to the resource category (Measured, Indicated and Inferred respectively).
- MN5 is assigned to a zone of low-confidence resource defined by a wireframe surface provided by HSC.

12.8 Cut-Off Grade

12.8.1 Marginal Economic Cutoff Grade

The assumptions described in 12.5 Economic and Processing Assumptions are the basis for calculating the cutoff grade.

AMDAD used the marginal economic cutoff grade for the life of mine schedule and Mineral Reserve estimation. The cutoff grade is applied to the diluted block model grades.

The marginal economic cutoff grade is defined as the grade above which the net revenue per tonne of ore will exceed the cost per tonne to handle and process the ore. i.e.:

recovered net revenue \$/t ore > ore cost \$/t ore

(grade g/t Au) x (process recovery) x (net price \$/g Au) > ore cost \$/t ore

grade g/t Au > ore cost \$/t / (process recovery x net price \$/g Au)

where

ore cost = process cost + G&A cost + additional ore mining cost

Therefore:

$$\text{cutoff grade (g/t Au)} = \frac{\text{ore cost (\$/t)}}{\text{process recovery x net price (\$/g Au)}}$$

The only mining cost included in the ore costs is the difference, if any, between mining material as ore versus mining material as waste. An example would be the difference between haul costs to the ROM area versus waste rock dump. This is referred to as the additional ore mining cost in the formulae above.

The formula above calculates the cut-off head grade for mined, diluted ore delivered to the process plant. The adjusted cutoff grade to apply to the undiluted resource is calculated as follows:

resource cutoff grade = (1+dil) x (cut-off head grade)

where “dil” is the additional dilution adjustment, as a fraction.

For the USD4,200/oz gold price, the nominated assumptions give a cutoff grade of 0.24 g/t Au for the diluted mining model or approximately 0.25 g/t Au for the undiluted mineral resource model.

12.8.2 Adjusted Cut-off Grade

There is limited available area within the claims for the pits, waste rock dump, tailings cell, process plant and long-term low-grade stockpile. The scheduling strategy used to maximise value of the project is to mine as quickly as the fleet will practically allow and process the highest-grade ore available while stockpiling lower grade material for processing at the end of the mine life. This strategy accumulates low grade stockpiles of over 10 million tonnes. The maximum storage capacity that could be designed in the available area is reached at diluted cut-off grade of 0.25 g/t Au. For this reason, the marginal economic cut-off grade for the Mineral Reserve and production schedule was set at 0.25 g/t Au.

Raising the cut-off grade from 0.24 to 0.25 g/t Au has minimal impact on project value because the material rejected is at the lowest end of economic grades with only a minor amount of contained gold and it would have been processed at the end of the project life.

12.9 Pit Optimization

From the mining block model, the optimisation block model is exported in required format for the Whittle™ optimisation program. The pit optimisation process is then completed in the following steps.

12.9.1 Optimised shell generation

Optimised pit shells were generated using the Lerchs-Grossmann (LG) algorithm. This creates a nest of shells by varying the “revenue factor” (applied to price) from 0.3 to 1.5. Each shell is cash-optimal for the nominated revenue factor and other project assumptions.

The pit shells generated by the optimisation are displayed in “block” format in the following figure. These shells were generated using the Measured and Indicated resources after applying the project assumptions described above. Each shell number in the drawing is defined by a different colour, and the crest of the

Revenue Factor "1.0" shell is shown. The pit shells range from small, high-value-per-tonne shells shown in blue to large low-value-per-tonne shells shown in red. The smaller shells are considered as possible intermediate stages. The larger shells are considered for the final pit and used for sensitivity analysis for upside conditions.

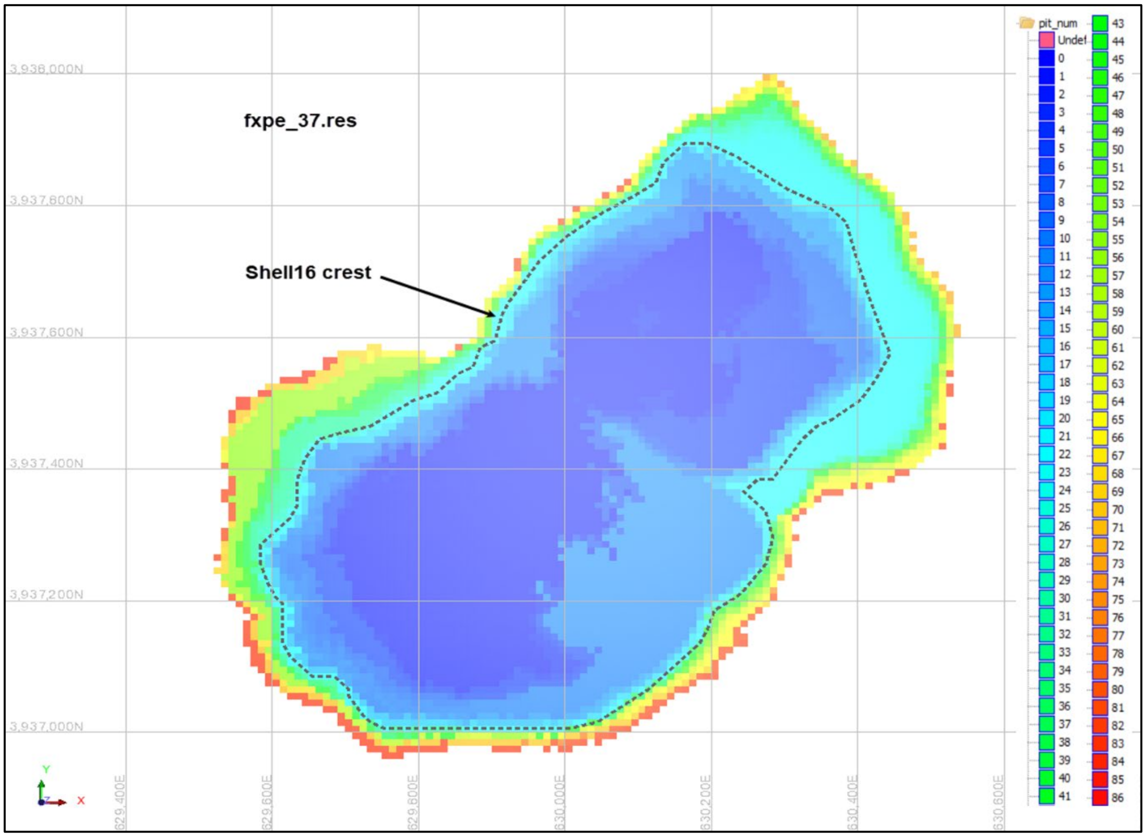


Figure 39 Plan view of optimised pit shells, coloured by shell number corresponding to revenue factor

12.9.2 Scenario Evaluation

Simple schedules were run against the pit shells to generate undiscounted and discounted cashflows (DCF) to assist with the selection of the preferred optimal shell. A processing limit of 2Mtpa was applied in the pit-by-pit analysis for pit selection.

The "Worst Case" schedule for a particular pit shell involves mining each bench completely before starting on the next bench, with no staging. This schedule gives the worst possible DCF.

The "Specified Case" schedule used a specifically selected shell as a starter pit to generate a more realistic mining scenario which defers waste in the upper benches of larger shell in the pushback between the starter and final pits.

The initial pit optimisation run on the inputs listed above produced a final shell selected on the highest Specified Case DCF with 23.5 Mt of mill feed at a gold grade which produced over 630 koz of gold in dore.
However, this shell was so large when trial designs were prepared for the waste rock and tailings storage there was not enough available area in the claims to fit them.

Rather than just selecting a smaller shell form the results list, a new set of pit optimisations were run using a range of elevated cut-off grades to force the optimisation software to define shells that could be scheduled to produce the maximum value. Excluding grades between the 0.25 g/t cut off and selected elevated cut-off reduces the shell size by removing volumes with marginal value driven by the lower grades.

The elevated cut-off grade that defined a shell with waste rock and tailings volumes that can be placed in the available area is 0.4 g/t Au. The following discussion relates to this modified pit optimisation.

In reviewing the pit optimisation results it is noted that:

- The mill feed tonnes and grade, gold production and values shown are on the basis of a 0.4 g/t Au cut-off. The Mineral Reserve and production schedule still include ore down to the 0.25 g/t Au cut off. The scheduling strategy to maintain the values indicated by the pit optimisation is to stockpile lower grades for processing after mining is completed.
- Except where otherwise stated, values referred to in relation to the pit optimisation are for operating cashflows and DCFs only. The pit optimisation does not consider any capital costs for the mining equipment and establishment, project infrastructure and processing plant.
- "Mill" or "Mill feed" material represents above-cutoff resource within an optimised pit shell, with dilution and loss adjustment. The quoted tonnes do not constitute a Mineral Reserve. Caution must be exercised in considering the cashflows as:
    - They are based on optimised pit shells and not on practical pit designs with appropriate adjustment to take account of practical mining widths, smooth walls, and incorporating berms and access ramps.
    - Completion of a practical open cut design with these adjustments typically increases the amount of waste rock to be mined and may also result in a slight reduction in mill feed. This is likely to reduce the DCF compared to the "raw" results for the unadjusted optimised shells.

The pit-by-pit results for the pit optimisation are summarised in the following table.

Pit shell	Mine life (Yr)	Cashflow ($M)	DCF Case ($M)		Strip ratio	Mined (Mt)	Waste (Mt)	Mill Feed		Prod. Au (koz)	
			Specified	Worst				(Mt)	Grade		
1	2.3	504	452	452	0.89	9	10	6	1.33	228	
2	4.3	785	653	653	1.04	18	12	7	1.26	261	Starter
3	4.6	825	680	678	1.02	19	15	8	1.25	296	
4	5.4	924	748	739	1.17	23	16	9	1.23	314	
5	5.8	971	779	765	1.17	25	26	10	1.26	374	
6	7.0	1,146	887	858	1.63	37	36	12	1.26	434	
7	8.1	1,298	968	928	1.98	48	37	12	1.26	443	
8	8.3	1,316	978	933	1.99	50	40	13	1.25	460	
9	8.7	1,359	995	941	2.04	53	41	13	1.24	464	
10	8.8	1,370	999	943	2.03	54	44	13	1.24	481	
11	9.1	1,395	1,010	953	2.11	56	46	14	1.24	489	
12	9.4	1,429	1,022	958	2.17	60	47	14	1.23	496	
13	9.6	1,444	1,030	959	2.18	61	48	14	1.23	500	
14	9.7	1,452	1,033	960	2.20	62	50	14	1.23	507	
15	9.9	1,467	1,037	961	2.24	64	51	14	1.23	508	
16	9.9	1,471	1,038	962	2.26	65	51	14	1.23	511	
17	10.0	1,476	1,039	963	2.29	66	52	14	1.23	513	
18	10.0	1,479	1,041	963	2.30	66	52	14	1.22	513	
19	10.2	1,486	1,043	963	2.32	67	54	15	1.22	519	
20	10.4	1,500	1,045	961	2.38	70	56	15	1.22	525	
21	10.4	1,501	1,045	961	2.38	71	56	15	1.22	525	
22	11.0	1,526	1,044	949	2.50	77	56	15	1.22	525	
23	11.7	1,569	1,047	925	2.79	89	56	15	1.22	525	
24	11.8	1,571	1,046	925	2.79	89	56	15	1.21	525	
25	11.9	1,576	1,045	922	2.81	91	56	15	1.22	526	
26	12.1	1,579	1,045	920	2.84	92	56	15	1.21	526	Final
27	12.2	1,584	1,044	918	2.88	94	70	16	1.18	567	
28	12.3	1,585	1,043	917	2.88	95	72	17	1.18	573	
29	12.3	1,587	1,042	915	2.90	96	73	17	1.18	574	
30	12.4	1,588	1,042	914	2.91	96	73	17	1.18	575	
31	12.6	1,592	1,038	910	2.98	99	76	17	1.17	581	
32	12.7	1,594	1,036	907	3.00	101	76	17	1.17	582	
33	12.8	1,594	1,036	906	3.00	101	77	17	1.17	584	
34	12.8	1,594	1,034	904	3.02	102	78	17	1.17	586	
35	12.9	1,595	1,033	902	3.04	103	78	17	1.17	587	
36	13.0	1,596	1,030	899	3.09	105	79	17	1.17	587	
37	13.1	1,597	1,028	894	3.12	106	80	17	1.17	590	
38	13.3	1,597	1,025	890	3.19	109	80	17	1.17	591	
39	13.3	1,597	1,024	890	3.19	110	81	17	1.16	593	
40	13.4	1,597	1,023	889	3.19	110	83	18	1.16	596	
41	13.5	1,596	1,019	884	3.26	113	83	18	1.16	596	
42	13.8	1,595	1,011	875	3.32	116	83	18	1.16	596	
43	13.8	1,594	1,008	871	3.34	117	84	18	1.16	598	
44	13.9	1,594	1,007	871	3.35	117	86	18	1.16	601	
45	13.9	1,594	1,005	868	3.36	118	87	18	1.16	602	

Table 49 Pit optimisation "pit-by-pit" results

The results indicate that:

1. Shell 44, the “revenue factor 1.00” shell, generates the highest undiscounted cashflow of $1,594M, in a 117Mt pit with a total mill feed of 18Mt at 1.16 g/t Au. Each shell increment up to this shell will add value on an undiscounted cash basis. Stepping out to larger shells will progressively lose value on an undiscounted cash basis.

2. When cashflows are discounted at 6.5%, Shell 19 generates the highest discounted cashflow (DCF) for a Worst Case (no stages) schedule of $963M, in a 67Mt pit with a total mill feed of 15Mt at 1.22g/t Au.

3. With cashflows discounted at 6.5%, Shell 26 generates the highest DCF for a “specified case” with Shell 2 as a starter pit. This gives an operating DCF of $1,045M, in a 92Mt pit with a total mill feed of 15Mt at 1.21g/t Au.

Based on these results, Shell 26 was selected as the optimal pit shell to guide the practical open cut stage design. Shell 2 was selected to guide the North starter pit design.

12.10 Mine Design

12.10.1 Open Cut Design Parameters

The open cut stage designs were prepared using the Surpac™ program, guided by the selected optimised pit shells. The general open cut mine design parameters, in addition to the geotechnical wall design parameters, are summarised in the table below.

Design element		Assumption
Batter height between berms		60 ft (18.29 m)
Berm width		25 ft (7.62 m)
Batter Angles	In rock	Variable depending on slope zone
		50° minimum, 64° maximum. Mostly 57° to 60°
	In fill	35°
Ramps	Gradient	10% (1 in 10), 12.5% (1 in 8) near pit base
	Width dual lane	60ft – to suit 55t payload articulated dump trucks
	Width single lane (near pit base)	46ft
	Ramp safety bund	Minimum of half wheel height of largest vehicle.
Minimum bench width		115ft minimum, mostly 200 to 300ft.

Table 50 General Mine Design Parameters

12.10.2 Open Cut Design

The final stage open cut design is shown in the following figure and is described in Part 13 – Mining Methods.

Figure 40 Final Open Pit Design

12.11 Mineral Reserve Classification

The critical mining, metallurgical, infrastructure, cost, revenue, environmental, social and permitting assumptions defined as part of the Colosseum Feasibility Study are considered by the contributing experts to be at a sufficiently high level of confidence for estimation of Proved Mineral Reserves. The confidence category applied to the Mineral Reserves therefore corresponds with the category of the Mineral Resources.

The estimated Proved Mineral Reserves are the economically mineable part of the Measured Mineral Resources, and the estimated Probable Mineral Reserves are the economically mineable part of the Indicated Mineral Resources. No portion of the Probable Mineral Reserves has been derived from the Measured Mineral Resource. No Inferred Mineral Resources are included in the Mineral Reserve.

12.12 Mineral Reserve Estimate

The Mineral Reserve was reported inside the final pit design at the economic cut off 0.25 g/t Au. The Mineral Reserve estimate is summarised in the table below.

Category and Area	Million Tonne	Au g/t	Au koz
Proved Reserves			
North Pit	9.8	0.88	280
South Pit	4.4	1.45	200
Total Proved Reserves	**14.1**	**1.06**	**480**
Probable Reserves			
North Pit	3.3	0.70	70
South Pit	3.2	0.74	80
Total Probable Reserves	**6.5**	**0.72**	**150**
Proved and Probable Reserves			
North Pit	13.0	0.83	350
South Pit	7.6	1.15	280
Total Proved and Probable Reserves	**20.6**	**0.95**	**630**

Notes:

(6) The tonnes and grades shown are stated rounded to a number of significant figures reflecting the order of relative accuracy of the estimate. The table may nevertheless show apparent inconsistencies between the sum of components and the corresponding rounded totals.

(7) Gold ounces are contained gold prior to application of process recovery.

Table 51 Mineral Reserve Estimate

The Mineral Reserve estimate was reported in accordance with The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 Edition (the 'JORC Code 2012'). The Ore Reserve Statement, including Table 1 of the JORC Code which corresponds to the SME Table 1 Checklist of Assessment Criteria, is included with Dateline's public release of the FS.

12.13 Comparison with Optimised Pit Shell

The following tables show how the pit design compares to the optimised pit shell.

		Shell 26 @ 0.40 g/t Au Cut Off	Pit Design @ 0.25 g/t Au Cut Off	Comparison
Ore	Mt	14.8	20.6	139%
	Au g/t	1.21	0.95	78%
	Au koz	526	630	120%
Waste Rock	Mt	56.1	62.3	111%
Total Pit	Mt	70.9	82.9	117%
Waste: Ore		3.8	3.0	80%

Table 52 Comparison with Optimised Pit Shell at 0.40 g/t Au Cut Off

		Shell 26 @ 0.25 g/t Au Cut Off	Pit Design @ 0.25 g/t Au Cut Off	Comparison
Ore	Mt	20.3	20.6	101%
	Au g/t	0.97	0.95	98%
	Au koz	633	630	100%
Waste Rock	Mt	50.6	62.3	123%
Total Pit	Mt	70.9	82.9	117%
Waste: Ore		2.5	3.0	121%

Table 53 Comparison with Optimised Pit Shell at 0.25 g/t Au Cut Off

The pit design has 17% more total tonnes than the optimised pit shell. This is due to:

- Matching practical mining shapes to the irregular optimised shell while maintaining as much of the contained gold as possible, and
- Cutting into the ridge on the east side of the North and South Pits to establish truck access into the North Pit.

The pit optimisation was run at an elevated cut-off grade of 0.40 g/t Au to make it focus on high value parts of the deposit. When Shell 26 is reported at the Mineral Reserve cut-off of 0.25 g/t Au it contains an additional 107 koz of contained gold. Almost all this gold was captured in the final pit design.

12.14 Mineral Reserve Estimate Risks and Opportunities

The project is considered to have low technical and commercial risk.

- The Mineral Resource is well defined by drilling.
- It is in an arid environment not subject to extremes of weather.

- It is close to a skilled workforce and major city for supply of services.
- The FS has confirmed adequate water and power supplies.
- The mine plan addresses requirements of the existing Environmental Impact Statement, Plan of Operations and Reclamation Plan and includes measures to ensure all water coming into contact with mine affected areas is retained on site.
- Mining and gold ore processing are well understood.
- Economic analysis shows the project to be financially robust with regard to operating and capital costs. It is most sensitive to variations in the gold price.

The following risk assessment is as recommended in the checklist of the SME Guide 2017. None of these are considered a significant risk to project failure.

Risk Factor	Impact	Mitigation
Gold price overestimated	Selected cutoff grade and optimised shell may not be optimal. Potential reduction in Mineral Reserve tonnes, increase in grade and reduction in contained gold.	Gold price used for the Mineral Reserve is at 15% discount to average spot price in three months prior to FS. Sensitivity conducted in economic analysis shows project to be viable at gold prices 42% lower than used in the FS.
Gold process recovery lower than forecast	Selected cutoff grade and optimised shell may not be optimal. Potential decrease in Mineral Reserve tonnes.	Process recovery set at lower than historical recoveries. FS test work supports 91% recovery over the range of grades and mineralogies expected.
Mining and / or processing costs higher than expected.	Increase to cut off grade which reduces gold production and project value. Selected optimised shell may not be optimal.	Sensitivity conducted in economic analysis shows project to be viable at operating costs 107% higher than used in the FS.
Ore selectivity is inadequate; dilution and mining loss are underestimated	Adverse impacts on life of mine ore production. Potential reduction in Mineral Reserve tonnes, grade and contained gold.	Careful attention to mining control, grade control, reconciliation, etc.

Table 54 Mineral Reserve Risks

The following opportunity assessment is as recommended in the checklist of the SME Guide 2017.

Opportunity	Impact	Recommendation
Additional ore within the pit design	Additional revenue from material currently classed as waste rock.	The pit design includes 2.8 Mt of Inferred Mineral Resource. If grade control during mining converts any of this to ore it has potential to add up to 55 koz of product gold. Recommend grade control and pit mapping of Inferred zones during operation.
Underground mining potential	Extend mine life and add value from material outside the pit design.	Underground optimisation analyses during the FS identified a sublevel caving (SLC) target below the eastern wall of the final North Pit and long hole stoping targets below the final South Pit. On the currently defined Mineral Resource the targets have potential to add over 50 koz of gold production. The North Pit SLC target is being drilled and is open at depth and north along strike. Recommend ongoing exploration drilling and underground mining evaluation of the North SLC target.

Table 55 Mineral Reserve Opportunities

13. MINING METHODS

13.1 Introduction

Australian Mine Design and Development (AMDAD) was engaged by Dateline Resources (Dateline) to compile the mining section of the Colosseum Feasibility Study (FS). AMDAD prepared the FS mine plan and Mineral Reserve Estimate using the March 2026 Mineral Resource Estimate (MRE) prepared by H & S Consultants (HSC).

13.2 Project Setting

Features of the site relevant to selection of the mining method and development of the mine plan are described below:

- Topography in the mine area is moderately steep and irregular, the climate is arid and vegetation is sparse.
- Colosseum mine was initially developed in the 1930's as an underground mine and more recently as an open cut operation. The North and South Pipes were mined as two adjacent pits from 1988 to 1993. All facilities were removed from the project after completion of mining and processing in 1994. The South Pit remains as a 130-metre-deep void with the bottom 26 metres flooded to the standing groundwater level. Mining was suspended before the North Pit went substantially below ground level, so the benches still open out onto the western slope of the range.

Figure 41 Clark Mountains looking north to Colosseum

Figure 42 Looking southwest to Clark Mountain

- Waste rock from both pits was dumped on the western side of the range, which drops away immediately west of the existing pit crests, and in a small area immediately north of the North Pit. The waste rock appears to have been end dumped as it now sits at angle of repose with no benching or contouring.

13.3 Supplementary Studies to the Mine Plan

The following studies were conducted to provide inputs to the mine plan.

13.3.1 Open cut geotechnical assessment

Open cut geotechnical slope analysis was conducted for the FS by Agapito Associates LLC. The wall design parameters are described in Part 12 – Mineral Reserve Estimates. Agapito's full report, "Geotechnical Assessment for the Feasibility Study of the Colosseum Open Pit Mine" February 2026, is provided as an Appendix to this report.

13.3.2 Open cut hydrological and hydrogeological assessment

The surface water management system was developed for the FS by Agapito Associates, LLC.

Ground water assessments covering the water supply from the Ivanpah Valley and the ground water regime at the mine site were conducted for the FS by HRS Water Consultants Inc.

13.4 Mining method selection

Open cut mining was selected as the mining method following the Project Definition Study (PDS) that was announced by Dateline in October 2024 (ASX release 23 October 2024). The PDS examined two mining cases:

Case 1 – Opencut mining of North Pipe and underground mining of South Pipe to provide earlier access to higher grade ore below the existing South Pit. Sublevel caving was selected for underground mining on the bases of low cost, higher resource recovery and higher value compared to other underground methods. Processing would be at 1.2 Mtpa to match former operation.

Case 2 – Both the North and South Pipes would be mined by opencut. Processing would be at 2.0 Mtpa to achieve better utilisation of the fleet at a realistic mining rate for the size and shape of the deposit.

The opencut cases were defined using pit optimisation. Practical staged pit designs were prepared based on the optimisation results.

For Case 1 underground mining, sublevel cave stopes were defined using stope optimisation and a detailed development design was prepared to support scheduling and cost estimation.

At a Scoping Study level of confidence, the PDS indicated that both cases could be technically and commercially viable. Case 2 with all opencut mining and processing at 2.0 Mtpa was considered likely to deliver significantly higher value than Case 1 with underground and opencut mining at 1.2 Mtpa.

Following the PDS, Dateline opted for open cut only for the BFS considering the following factors:

- The tonnage of potential ore remaining beneath the South Pit may not be sufficient to justify the underground development cost.
- Underground mining targets ore at a higher cutoff grade than open cut mining. This means that underground mining would leave behind a significant tonnage of resource that would have been extracted as ore in an open cut.
- Open cut only mining is consistent with existing approvals and the most recent operation.
- Open cut mining offers greater confidence for estimation of ore tonnes and grade than underground mining.

Since the PDS was released, Dateline has conducted extensive drilling programs to support the Mineral Reserve Estimate for the FS. In addition to adding confidence to the open pit mine plan, the drilling has delineated a significant underground mining target below the east wall of the North Pit. There has been insufficient time to assess underground mining of this target at a FS level of confidence, and it is still open at depth and along strike to the north. Dateline is continuing to drill this area to assess its potential to add life and value to the project.

13.5 Open Pit Mining Method Description

Open pit mining will be conducted by a U.S. based mining contractor with mine planning and geological control by Dateline personnel. Activities covered by the mining fleet will include:

- Mining of ore and waste from the pits and all associated activities for this such as grade control, blasting, surface preparation and haul road maintenance.
- Haulage and spreading of dry stack tailings into a cell within the waste rock dump,
- Crusher feed including general maintenance of the run of mine (ROM) and reclaim of ore form short- and long-term stockpiles to the crusher.
- Establishment and maintenance of the site water management system to ensure the project operates with zero discharge of water that has come into contact with mining or processing activities.

A summary of the main mining activities is provided below. Further detail is provided in the following sections.

13.5.1 Land Clearing, Vegetation and Topsoil management

Any existing vegetation and topsoil will be removed from the footprint of areas to be mined or areas on which rock piles will be placed. This activity will be in accordance with the approved Plan of Operations and Reclamation Plan.

For the purpose of initial planning, topsoil is defined as the top 150mm (6") of material. This depth will be assessed as clear and grubbing and topsoil harvesting proceeds to ensure all viable topsoil is collected for use in progressive rehabilitation over the mine life.

Care will be taken to retain as much vegetation as possible to promote viability of the topsoil for future revegetation.

Topsoil will be stored in low height stockpiles around the northern western rim of the North Pit and southeast of the waste rock dump.

Topsoil will be reclaimed from the stockpiles and spread over areas where mining activities have been completed. This will happen progressively over the mine life to ensure areas are rehabilitated as soon as possible.

A nursery will be established early in the mine life to develop a stock of plants collected from the area to be planted on the progressive rehabilitation areas.

13.5.2 Construction Earthworks

The mining contractor will be directed to perform any necessary earthworks required to commence the mining operations, including:

- Ex-pit haul roads. Approximately 800 metres (2,600ft) of ex-pit haul roads are required to connect the pits to the waste rock dump and dry stack tailings cell, the dry stack tailings stockpile laydown areas, pads, haul roads, culverts and water management structures. Only minor volumes of cut are required for the ex-pit haul roads. They will be formed mostly with fill from mine waste rock.

- Earthworks required for establishment of the site water management system including a clean water cut off drain and three retention ponds. The site water management system also includes installation of impermeable HDPE liners and new monitoring wells as described later in this section.

13.5.3 Pioneering Access and Bench Establishment

Prior to productive mining within the open cut footprint, the mining fleet will:

- establish haul roads into the uppermost open cut benches
- establish level benches by a combination of dozing of the initial uneven surface, drilling and blasting of material too hard for dozing, and excavation of material down to the first level bench elevation.

13.5.4 Ore Definition and Grade Control

Prior to blasting, each mining bench will be drilled and sampled to model the detailed gold distribution which will be used to define ore and waste. The most appropriate grade control system will be investigated and defined by the Dateline geological team. For the purpose of the FS, a dedicated reverse circulation (RC) drill rig is assumed for grade control.

The grade control system will be optimised once mining commences. The following system is assumed for the FS:

- Drill holes angled at approximately 60°, with conceptual drill hole line spacing of 30ft x 30ft pattern.
- 70ft hole depth to cover three 20ft benches.
- Samples analysed in a laboratory at site and the assayed grades applied to the grade control block model to define ore zones for mark out.

The FS allows for grade control drilling and sampling across 125% of the area of expected ore zones on each bench as predicted by the mineral resource block model.

Grade control drilling will be supplemented by bench mapping by the Dateline mine geologists.

Ore zones will be defined by geostatistical modelling of the grade control data guided by pit mapping and interpretation. The grade control model will be regularly compared against the mineral resource block model, diluted mine model and ore tonnes mined and milled. Regular mine reconciliations will facilitate adjustments to the models and mining methods to ensure Mineral Reserve is extracted as efficiently and completely as possible.

13.5.5 Drill and Blast

It is anticipated that all materials mined in the open cut, except areas where the pits mine through existing waste rock emplacements, would require drilling and blasting for productive excavation.

Drill and blast operations will entail the following:

- Drill and blast will generally be conducted on 20ft high benches.
- 6" diameter blastholes will be drilled using track-mounted down the hole hammer (DTHH) drills.
- Blast patterns and delay timings will be designed to optimise fragmentation for mining with minimal movement of ore. Care will be taken to minimise blast vibration, fly rock, noise and dust.
- It is anticipated that the arid environment and minimal influence of ground water will allow ANFO explosive to be used for most of the blasting.
- Explosive primers and detonators will be stored in compliant container style magazines southeast of the waste rock dump. The magazines will be bunded and fitted with lightning protection. The magazine site will be in a secure site with restricted access.
- Stemming material will be obtained by crushing and screening of competent waste rock from within the open cut footprint.

Drill traces in the current pit walls indicate the previous mining operation used pre-split blasting to promote stable pit walls free of loose rock. The mine will assess effectiveness of pre-split blasting. If it shows positive benefits for safety and productivity it will be adopted in sections of the pit walls where suitable.

13.5.6 Loading and Hauling

Waste rock and ore will be loaded and hauled by a main mining fleet comprising a 120-tonne class hydraulic excavator, loading 60-tonne to-100 tonne payload mining trucks.

The excavator will be in backhoe configuration, as this will provide a higher degree of selectivity than digging in face shovel configuration. After each 20ft high blast, the approximately 24ft high shot would be mined in three 8ft or four 6ft high flitches, depending on the level of selectivity required and stability of the blasted dig faces.

Ore will be hauled either directly to the crusher, or to short term stockpiles adjacent to the crusher or to a long-term stockpile, most of which will be reclaimed for processing following the completion of mining.

13.5.7 Selective Ore Mining

Forecast gold prices, gold process recovery and project operating costs allow adoption of a much lower cut-off grade than the 1987 to 1993 operation. Examination of grade continuity in exploration drill holes and the mineral resource block model relatively broad ore zones on the pit benches.

Selective mining will be a priority with emphasis on good mining practices such as:

- Minimising and tracking blast movement,
- Timely preparation of grade control model and ore mark ups in the pit,
- Good communication between Dateline and the mining contractor and their operators,
- Dateline geological control when mining ore zones, and
- Regular mine reconciliations.

13.5.8 Waste Rock Storage

Waste rock characterisation prior to 1987 and in 1994 has not identified potential for acid drainage or leaching of deleterious chemicals. All waste rock will be hauled to the waste rock dump and placed without segregation by material type.

The waste rock dump will enclose and eventually fully encapsulate a cell of de-watered tailings.

13.5.9 ROM Stockpiling and Crusher Feed

Ore is scheduled to be mined in three grade bins:

- High Grade >=0.75 g/t Au
- Low Grade 0.55 to 0.75 g/t Au
- Marginal Grade 0.25 to 0.55 g/t Au

These bins are set against the planned dig rate of the mining fleet, so the majority of ore processed during the mine life is High Grade, supplemented as necessary by Low Grade with the remaining Low Grade and Marginal Grade stockpiled for processing after the completion of mining.

13.5.10 Tailings Haulage and Placement

Tailings from the process plant will be dewatered to less than 20% moisture. Test work for the FS shows it will be handleable at this moisture. A dedicated front-end loader will load trucks shared with the mine fleet, which will haul the tailings to a cell within the waste rock dump where they will be spread and compacted by bulldozers prior to the next lift being placed on top of the compacted layer.

13.5.11 Site Water Management

The entire mining and processing operation is designed as a closed loop system for water entering the site. Any rainfall or process water which comes into contact with any of the processing or mining activities will be captured and retained on site for use as process water, dust suppression or evaporated in lined retention ponds.

The system will consist of:

- A cut off drain south of the waste rock dump to divert clean rainfall run off before it enters the mining area,

- Three lined retention ponds sized to hold rainfall run off over the entire mining and process areas from a 1 in 100 year 24-hour rainfall event,
- Formation of the waste rock dump and other facilities to direct drainage to the retention ponds, and
- Impermeable HDPE liners under the waste rock dump, dry stack tailings cell and long term stockpiles.

Site water management is not entirely a mining function, but installation and maintenance of the systems will be conducted by the mining contractor under direction of Dateline.

13.5.12 Ancillary Operations

As well as the primary excavators, trucks and blast hole drills the mining contractor's fleet will include:

- Tracked bulldozers to maintain level, clean floors on the mining benches and to spread and compact waste rock and dewatered tailings,
- Graders to maintain running surfaces on pit floors, haul roads and stockpile areas,
- Water carts to provide dust suppression in all areas where trucks operate,
- Standpipe fill points for the water carts,
- Large wheeled front-end loaders maintain the crusher area, reclaim for short term stockpiles to the crusher, load trucks reclaiming from the long-term stockpile and load dewatered tailings into trucks,
- Explosives truck to deliver bulk explosives down each blast hole,
- Service vehicles for field servicing and refuelling of the mobile fleet,
- Explosives truck to deliver bulk explosives down each blast hole,
- A mobile crane for bucket changes, component changes and positioning pumps,
- A contract crushing and screening plant to provide aggregate for road sheeting, blasthole stemming and general construction,
- A small wheel loader or integrated tool carrier to feed the crushing and screening plant, load stemming into blastholes, and facilitate tyre handling,
- Diesel powered dewatering pumps to manage any inflows to the pits and to move water from the retention ponds to the raw water storage as required,
- Diesel powered lighting plants to allow night shift operation,
- Light vehicles for movement of personnel around the site, and
- Buses to ferry mine personnel from a staging point adjacent to the I15 highway to and from the mine.

13.5.13 Mine Rehabilitation and Closure

Mining and processing at Colosseum are still subject to and will be run in compliance with the existing Environmental Impact Statement (1985) and approved Reclamation Plan and Plan of Operations (1984). Mine rehabilitation and closure will be conducted in accordance with these documents.

The project will adopt progressive rehabilitation so that permanent drainage and vegetation can be established as soon as practical in each area where mining and processing activities have been completed.

13.6 Mine Design

The opencut pits will mine two mineralised pipes referred to as the South and North Pipes. Most of the ore mined from 1987 to 1993 came from the South Pipe. The original South Pit crest was at an average elevation of 5740ft (1745m) above sea level (asl) and it was mined to 5310ft (1618m). The proposed new South Pit will widen and deepen the pit to 5020ft (1530m) asl.

Only limited mining was conducted in the original North Pit so the benches are still at close to original ground level at 5700ft (1737m) asl. The proposed North Pit will be mined to 5240ft (1597m) asl. The practical open cut design was guided by a pit optimisation process and based on design parameters outlined in Part 12 Mineral Reserve Estimates. The staged design, haul roads and WRD are shown in the following figures and described in the subsections below.

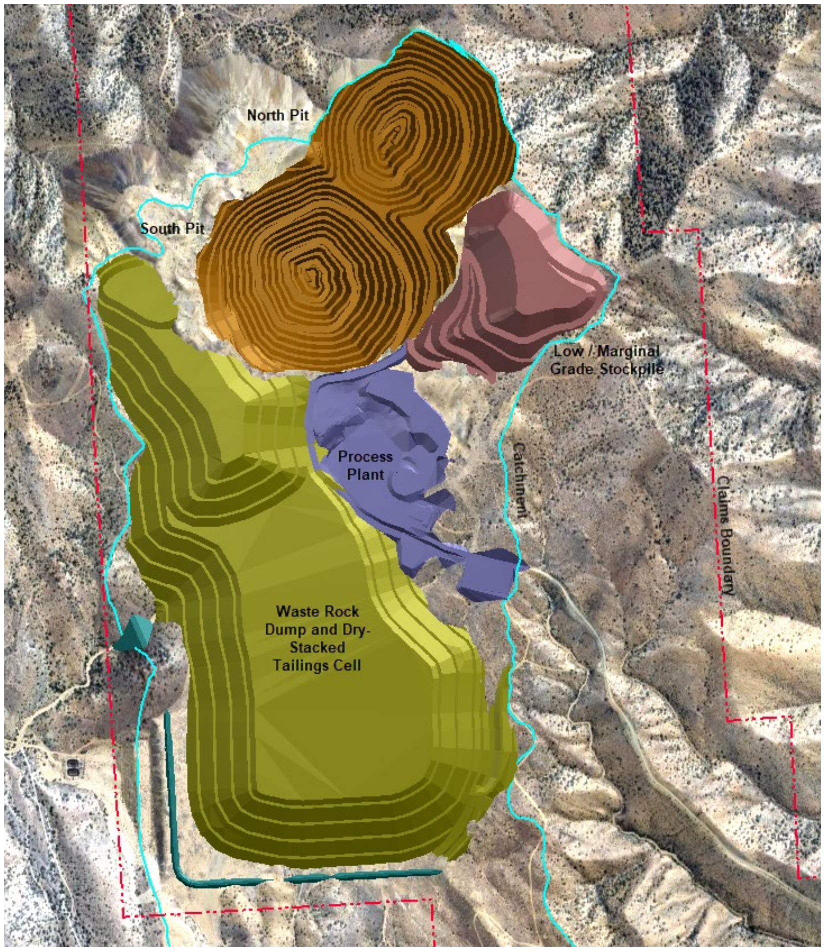


Figure 43 Mine design – plan view of final open cut, northern end of waste rock and tailings emplacement, and site infrastructure

13.6.1 Starter Pit

The North Starter Pit, shown in the following figure, is located northern side of the deposit, with pit base at 5420 ft asl and upper bench top at 6044 ft asl. It spans approximately 470 m east west from crest to crest and 410 m south north from crest to crest.

The ramp exits the pit at 5720 ft asl and cuts into the ridge southwards to the ROM area. This section of ex-pit ramp is shared with the North Final Pit pushback. The exit to the ROM is at 5634 ft asl.

The North Starter pit includes a ramp in the east wall up from 5720 ft asl to provide access to the North Final pushback benches above 5720 ft asl.

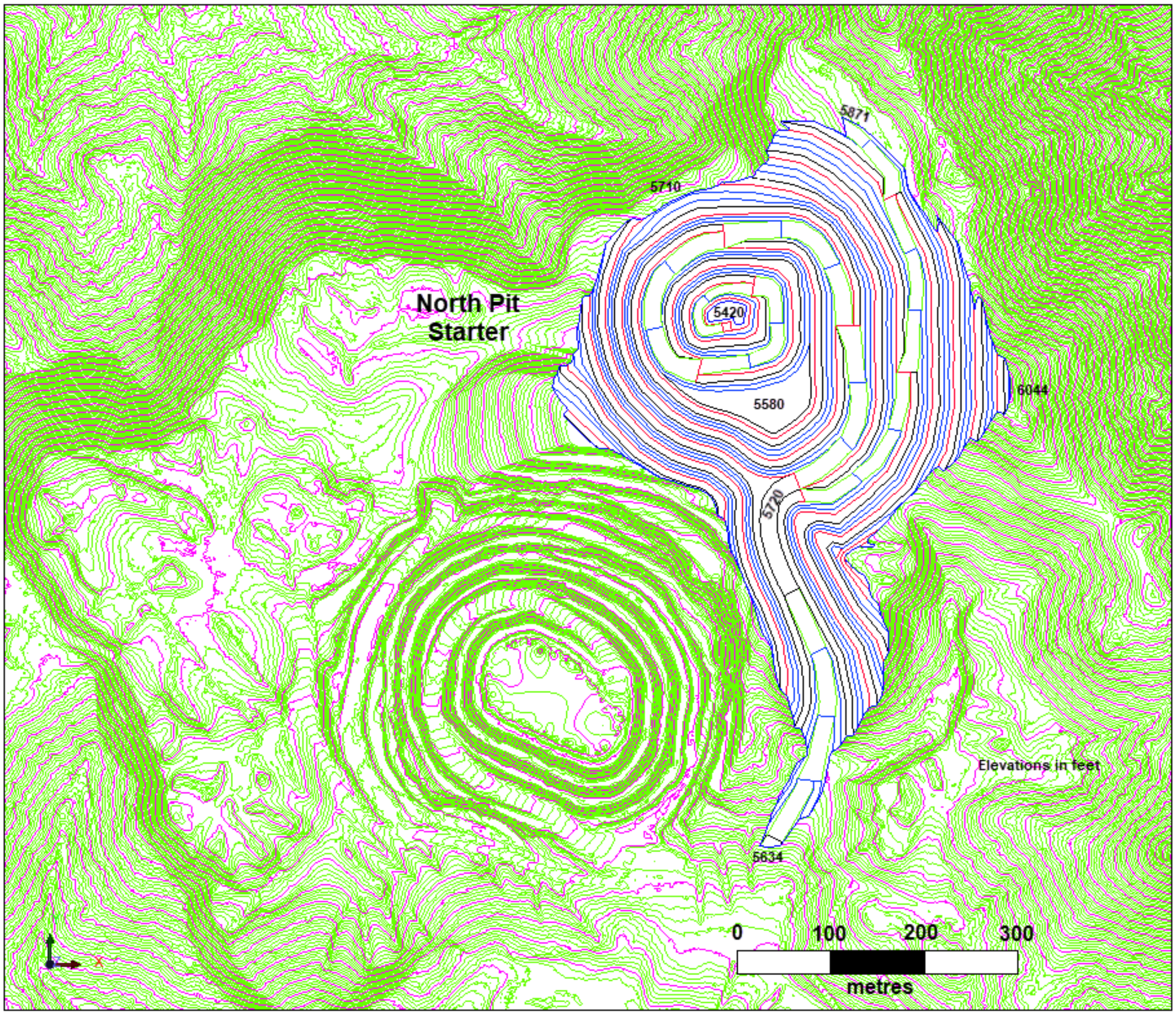


Figure 44 Northern Starter Pit design – plan view

13.6.2 North Final Pit

The North Final Pit has a pit base at 5240 ft asl and upper bench top at 6013 ft asl. It spans approximately 544 m east west from crest to crest and 500 m south north from crest to crest. It shares the same ramp exit to the ROM area as the North Starter Pit.

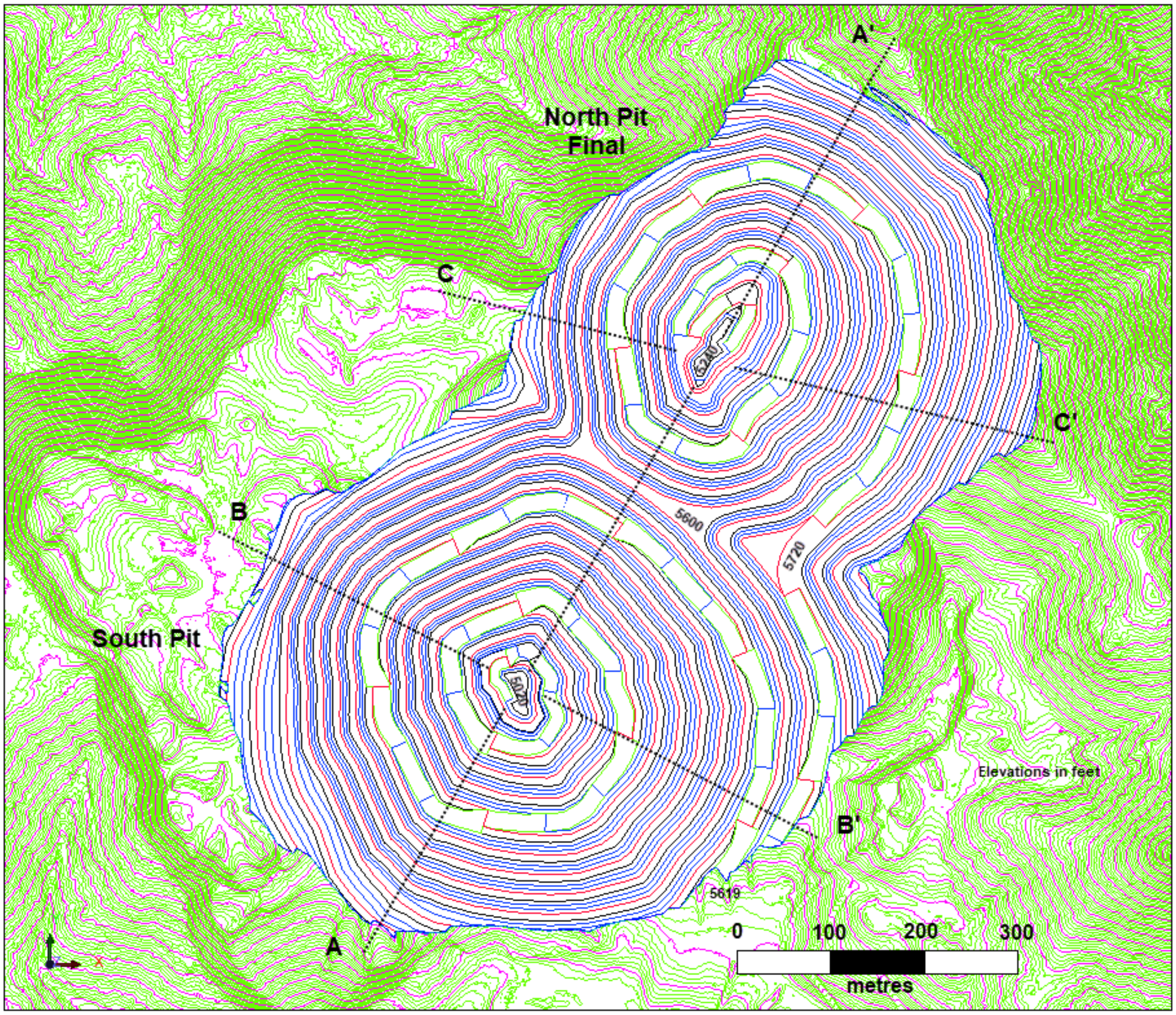


Figure 45 Mine design – plan view of final open cut

13.6.3 South Pit

The South Pit will be mined in a single pushback from the existing pit to the final wall.

The South Pit has a pit base at 5020 ft asl and upper bench top at 5840 ft asl. It spans approximately 685 m east west from crest to crest and 520 m south north from crest to crest.

The South Pit ramp exit to the ROM area is adjacent to the North Pit exit.

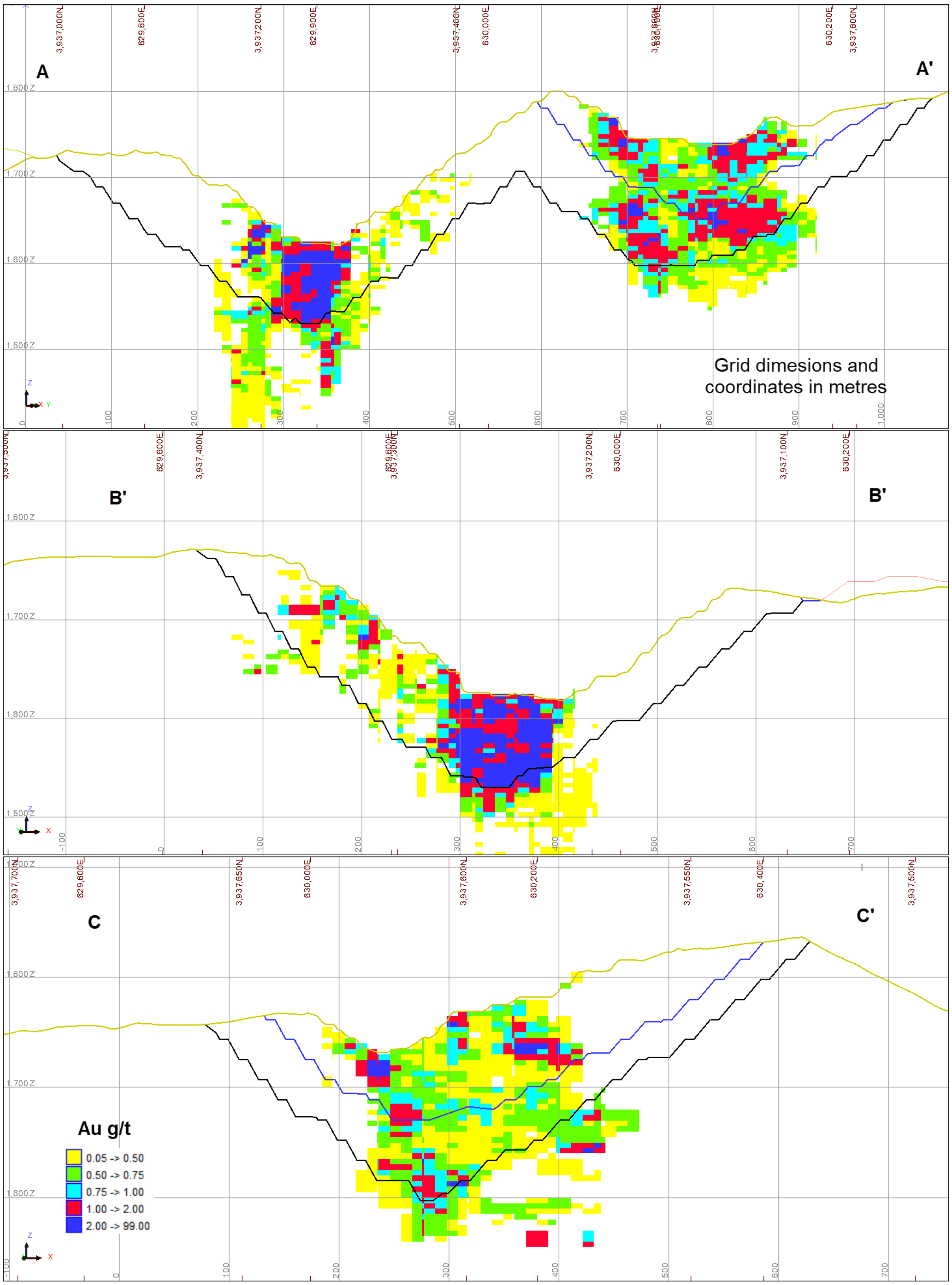


Figure 46 ***Pit Cross Sections***

13.6.4 Waste Rock and Tailings Co-Storage

13.6.4.1 Overview

Tailings from the mineral processing plant will be de-watered via the filter plant and "dry-stacked" in a cell within the mine waste rock storage facility. The tailings and mine waste rock will be placed separately so the two material types will not be co-mingled. The waste rock and tailings and tailings storage emplacement is designed to meet the following objectives:

- The storage emplacement will be stable and will not suffer failures that could result in solid materials spilling outside the claims area,
- Moisture from the tailings is prevented from escaping to external surface or ground water systems,
- The tailings will always be contained to mitigate the risk of water or wind erosion, and
- The mine waste rock placement sequence is designed to always contain and eventually encapsulate the dry stacked tailings. The tailings will be fully encapsulated in a final landform suitable for rehabilitation in accordance with the Reclamation Plan.

13.6.4.2 Waste Rock Description

Approximately 62 Mtonnes (68 Mtons) of waste rock is expected to be mined in the open pits. Almost all the mine waste rock consists of unmineralized gneiss, granite and rhyolite. Only minor sulphides have been observed in mine waste rock away from the gold bearing mineralised zones. Mine waste rock was treated as Class C material ("mining wastes from which any discharge would be in compliance with the applicable water quality control plan, including water quality objectives, other than turbidity") during operations from 1987 to 1993.

No evidence of acidic or other deleterious seepage from the exposed waste rock dumps has been reported since mining was halted in 1993.

The last waste rock mined from the South Pit in 1993 is considered representative of the deepest material that will be mined in the proposed re-commenced operation.

13.6.4.3 Waste Rock Emplacement

All the waste rock will be stored in an emplacement on the western side of the claims area south and southwest of the South Pit. This area covers most of the existing tailings dam.

Additional capacity, should it be required, is available on the hillside east of the North and South Pits. Up to 4.6 Mtonnes (5.1 Mtons) of waste may also be back dumped into the North Pit once mining is completed in that pit.

13.6.4.4 Waste Rock Design Parameters

The waste rock emplacement will be built from bottom up in 50ft high lifts with 1 in 2.5 slopes on the final faces separated by 25ft wide berms to give an overall final slope of 1 in 3.

Haul roads will be constructed within the emplacement as required with 60ft width and maximum 10% grade.

13.6.4.5 Waste Rock Emplacement Sequence

Waste rock will initially be placed in the southern end of the emplacement to form a wall around the dry stack tails cell. The first year of placement in this part of the dump will also include construction of the clean water cut off drain and the primary site water retention pond across the south end of the dump. A second retention pond will be formed in the valley where the current tailings dam spillway discharges. Formation of the cut off drain and two initial retention ponds at commencement of the works will allow site drainage to be managed so that clean water run-off from the south never contacts the mine workings and mine affected water can always be retained in lined containments.

Over the first three years, the waste rock wall enclosing the dry stack tailings will be built up to a final height 5ft above the final maximum height of the tailings. In this period, a third retention pond will be formed on the east side of the waste rock dump to supplement and act as a buffer to the primary retention pond. The rate of waste rock placement ensures the waste rock wall will always be above the height of the tailings as they are placed.

Once the southern end of the waste rock dump has been placed to fully enclose the dry-stack tails cell, waste rock will be placed in the northern end of the dump east of the South Pit. At completion of mining in the North Final Pit in Year 4, waste rock will be placed in the North Pit void for six months to allow time for tailings at the north end of the tails cell to be stacked to full height. This end of the tailings can then be over-stacked with waste rock while the remaining southern end of the tails cell is filled.

During the final year of mining in Year 6, waste rock will be surcharged above the northern end of the tails cell. At completion of ore processing in Year 11 when the last tailings are placed in the tails cell, the surcharged waste rock will be rehandled to cover the tailings to a minimum depth of 20ft.

13.6.4.6 Final Landform

The waste rock placement sequence allows for progressive rehabilitation of the final surfaces so that by the end of the project life the final tailings capping at the south end of the emplacement and the area from which waste rock has been rehandled for the capping will be the last areas to commence rehabilitation including re-planting of endemic species.

The final dump shapes are designed for stability, ability to promote re-growth of vegetation and conformity with the surrounding natural terrain.

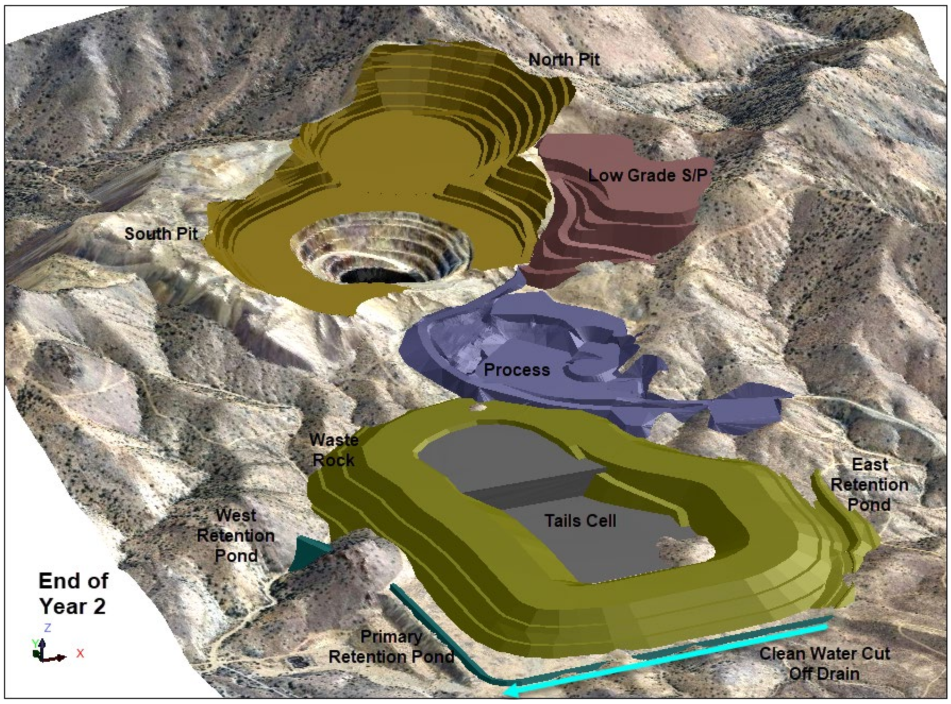


Figure 47 Mine Progress End of Year 2

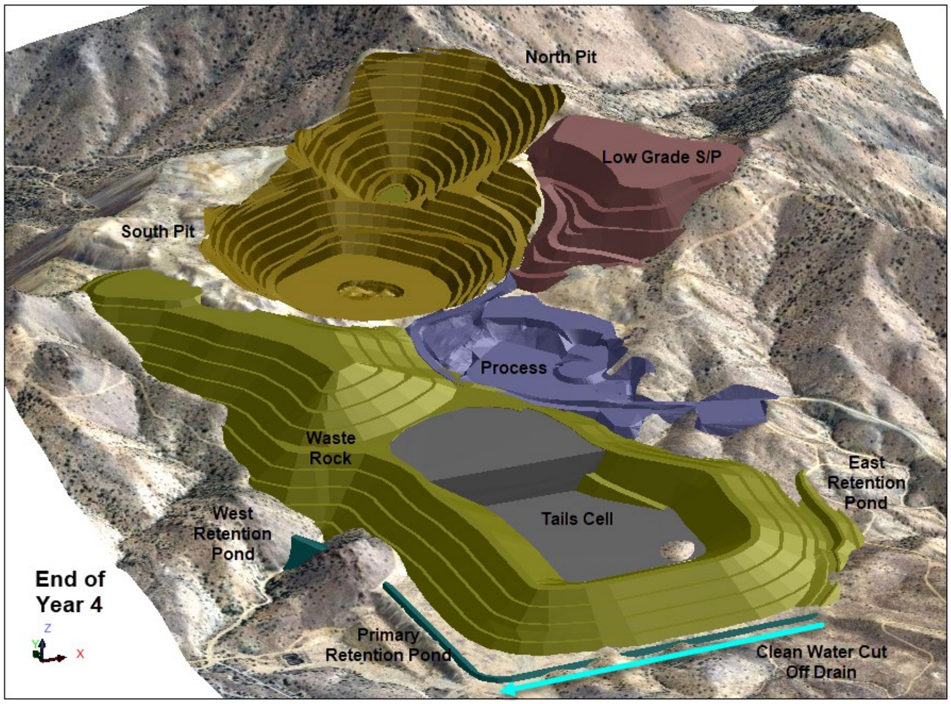


Figure 48 Mine Progress End of Year 4

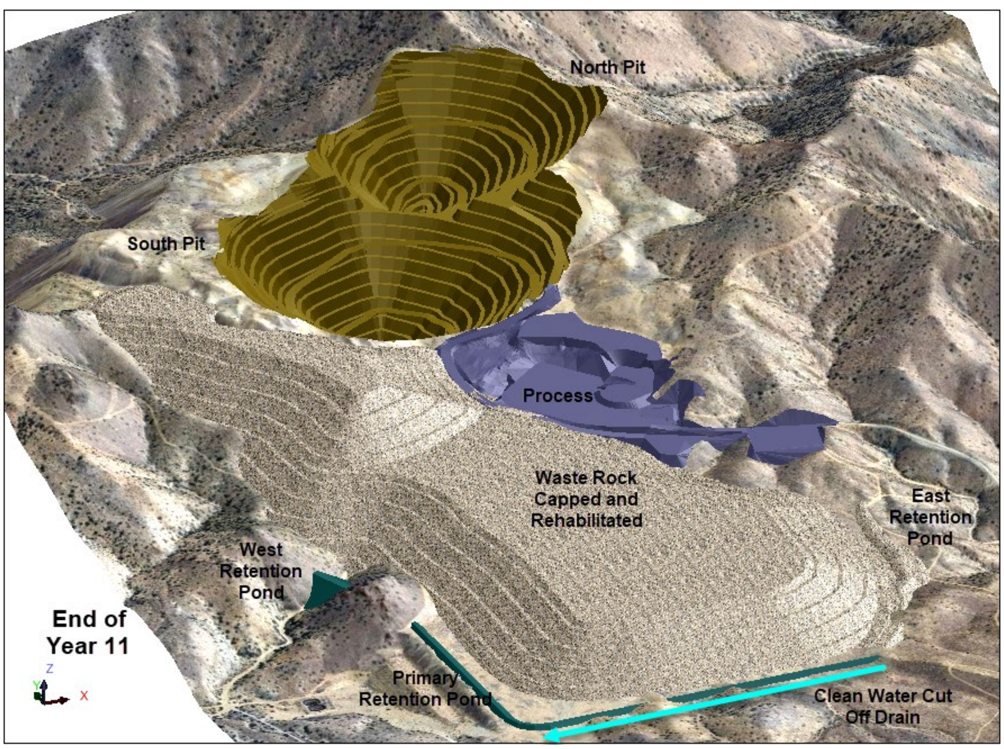


Figure 49 Mine Progress End of Year 11

13.6.4.7 *Dry Stack Tailings Storage*

The dry stack tailings will leave the process plant at a minimum solids density of 80% to 85%, backhauled to the waste rock emplacement by the mine haul trucks, and be placed in a confined cell, which will eventually be fully encapsulated. Both the waste rock and dry stack tailings cell base will be on an impermeable HDPE membrane liner. The dry stack tailings will be de-toxified to extremely low levels of free and WAD cyanide prior to leaving the process plant. Tailings will be hauled to the storage cell by trucks and spread by bulldozers in 3 to 6 feet thick layers. This will promote further drying and compaction.

13.6.4.8 *Dry Stack Tailings Placement Sequence*

Initially tailings will be placed over the whole available area of the tailings cell. Using the whole area allows the confining waste rock dump to rise faster than the top of tailings.

During the second year of operations, the tailings cell will be divided in two from north to south for ongoing placement. The north section will be placed to full height first, then the south section. This sequence allows areas of the tailings cell to be completed so they can be over-stacked with waste rock as the mine progresses.

13.6.4.9 *Waste Storage Stability*

Stability of the proposed waste rock emplacement, tailings cell and underlying existing tailings storage was analysed by Agapito Associates as part of the FS. The report concluded that:

- The existing tailings, tailings dam wall and underlying bedrock provide a stable foundation for the new waste rock dump and tailings cell,
- Potential for liquefaction of existing tailings or the new dry stack tailings is very low, and
- Stability evaluations demonstrate factors of safety that meet or exceed accepted industry criteria for long term or seismic loading conditions.

13.6.5 *Mine Road Design*

Permanent and semi-permanent mine haul roads will comprise:

- In pit ramps for the North and South Pits,
- Pit exit to ROM stockpile / crusher area,
- Pit exit to waste rock dump,
- De-watered tailings stockpile to dry stack tails cell, and
- Ramps on the waste rock dump and long-term ROM stockpile.

Haul road design for the FS is limited to alignment, widths, gradients and turning radii plus allowances for safety bunds and drains where required. Cost estimates for the FS make adequate allowance for earthworks and running surface formation. Detailed designs including cambers and intersections will be conducted during mine development.

13.6.6 ROM Stockpiles and Crusher Feed

Modelling of life of mine production profiles identified the highest value strategy of running the mining fleet at maximum capacity to deliver the highest-grade ore possible to the mill early in the mine life, while stockpiling lower grades for processing after completion of mining. Three grade bins based on the diluted Mineral Reserve were defined to facilitate this strategy:

- High Grade (HG) >= 0.75 g/t Au
- Low Grade (LG) 0.55 to 0.75 g/t Au
- Marginal Grade (MG) 0.25 to 0.55 g/t Au

Run of mine (ROM) ore stockpiles will be formed as:

- Short term stockpiles adjacent to the crusher. These will mainly comprise HG. They will be close enough to the crusher to be reclaimed by front end loader. The short-term stockpiles will often be close to zero tonnes but may reach up to 300,000 tonnes as the mine works through high grade areas.
- Long term stockpiles will be located in a large storage site in a valley on the ridge southeast of the pit exit. This site currently has low grade ore left from the previous operation. The combine LG and MG stockpiles peak at 10.3 Mt. Ore reclamation from the long-term stockpile will be by a front-end loader loading mine trucks for direct tipping to the crusher.

Most of the ore feed over the mine life will be direct tipped by trucks into the crusher hopper. However, the front-end loader will be used to rehandle ore off the short-term stockpiles in periods when the mine delivers less than the target tonnage of HG for the period, or an assumed minimum of 20% of the crusher feed tonnes for the period, whichever is larger. The 20% factor is to allow for mismatches between the relatively continuous mill operation and the discontinuous delivery of ore from the mine.

13.6.7 Mining Infrastructure and Facilities

Mining infrastructure and facilities will mostly be provided by the mining contractor. It will include:

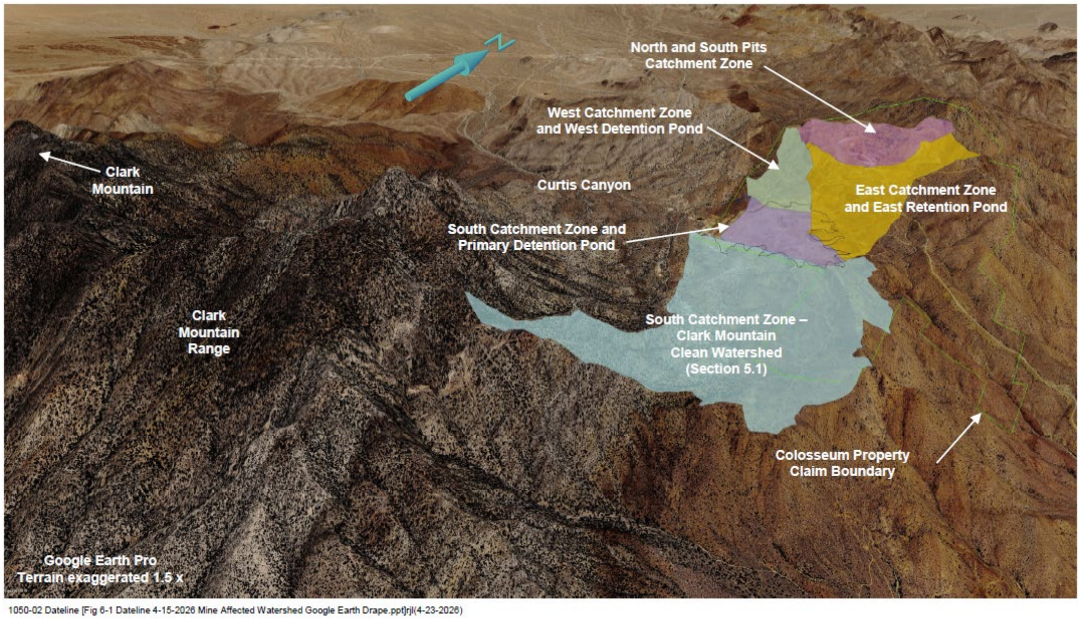
- Workshops,
- Vehicle washdown facility encompassing oil-water separation,
- Offices,
- Fuel storage,
- Explosives magazine and AN storage,
- Water truck fill points.

Earthworks for the mine contractor's area will be done by Dateline. Dateline will supply services including electricity, service water and fire water to the edge of the contractor's area. Costs for these items are included in the site infrastructure estimate.

13.6.7.1 Site Water Management

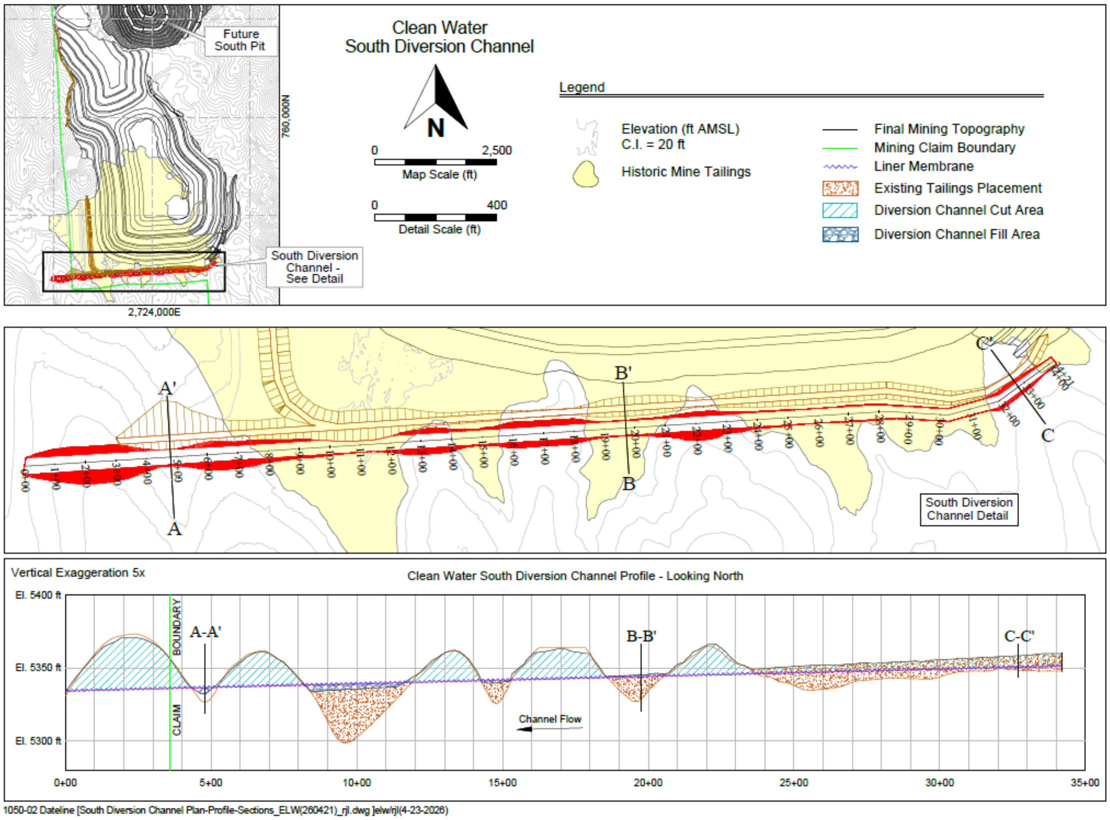
Drainage across the entire site will be managed to create a closed loop system with zero discharge of any water that has come into contact with mining or processing activities. The system comprises four main elements:

- A cut off drain placed south of the waste rock dump will intercept rainfall run off from areas in the catchment south of the mine and direct it west into the Shadow Valley. This re-establishes pre-1987 water flows for almost half of the catchment area.
- Lined retention ponds will be formed east and south of the waste rock dump and in a valley west of the waste rock dump where the spillway from the current tailings dam discharges. The position and size of these ponds are designed to hold rainfall run off over the entire mine and process plant site in a 1 in 100 year 24-hour rainfall event.
- Final faces of the waste rock dump are designed to direct rainfall run off to the retention ponds. The final face configurations will be formed as the waste rock is placed so the drainage will work throughout and beyond the mine life. All other areas of the mine and process plant drain naturally to the east and south retention ponds.
- The entire area under the waste rock dump, tailings cell within the waste rock dump, and a long-term low-grade stockpile will be lined with an impermeable membrane. All hydrogeological analyses since the 1980s, including work done for the DFS, show the granite/gneiss rock at Colosseum has extremely low permeability with the only potential for groundwater movement being through isolated fractures. Nonetheless, installation of the liners prevents any migration of water from mine affected areas into potential groundwater below or adjacent to the project. Any water which may percolate through the waste rock or tailings will run along the liners into the retention ponds where it will be collected and used as process water.



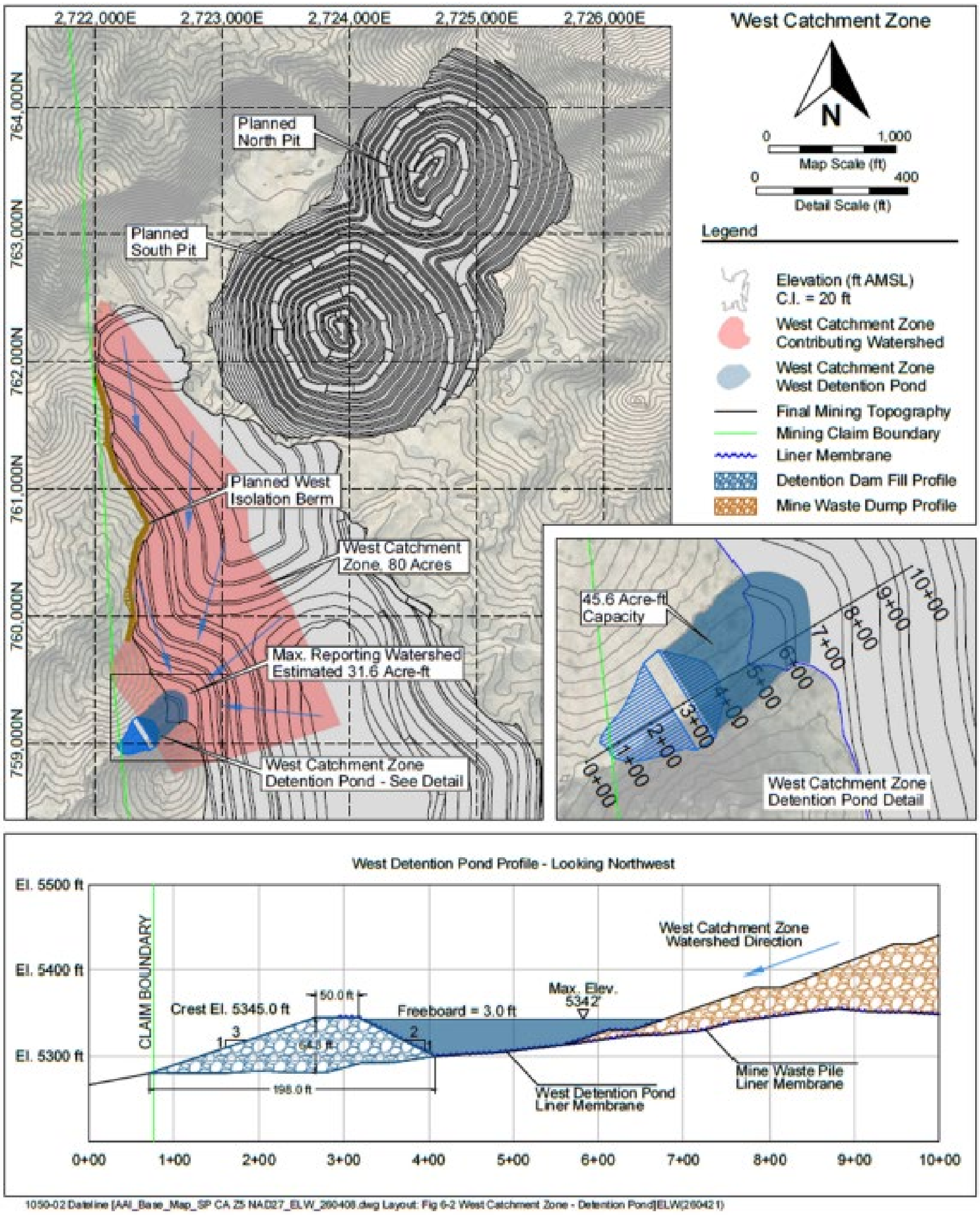
Source: Prepared by Agapito Associates, 2026

Figure 50 Catchment Zones (Agapito Associates 2026)



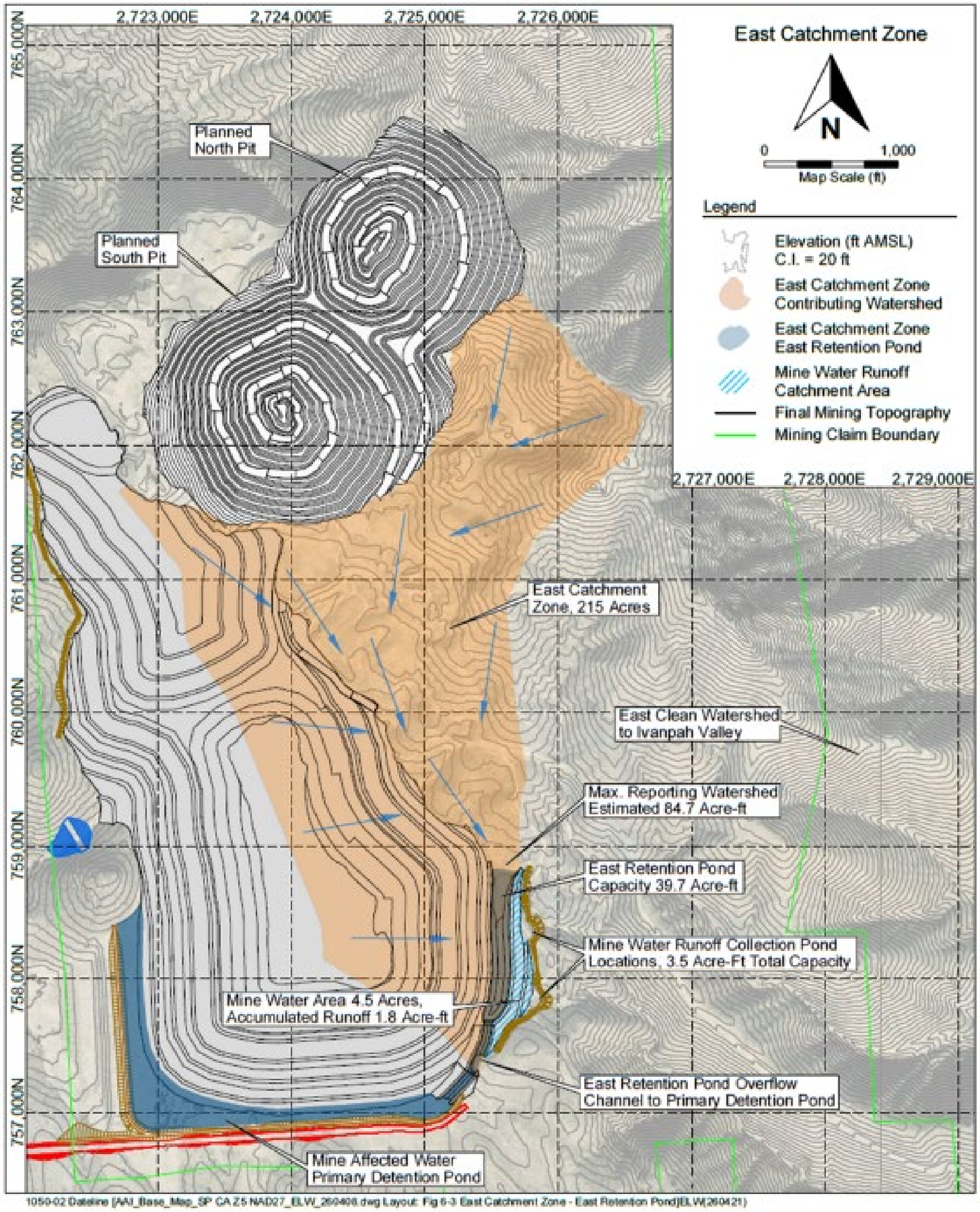
Source: Prepared by Agapito Associates, 2026

Figure 51 Clean Water Diversion Channel



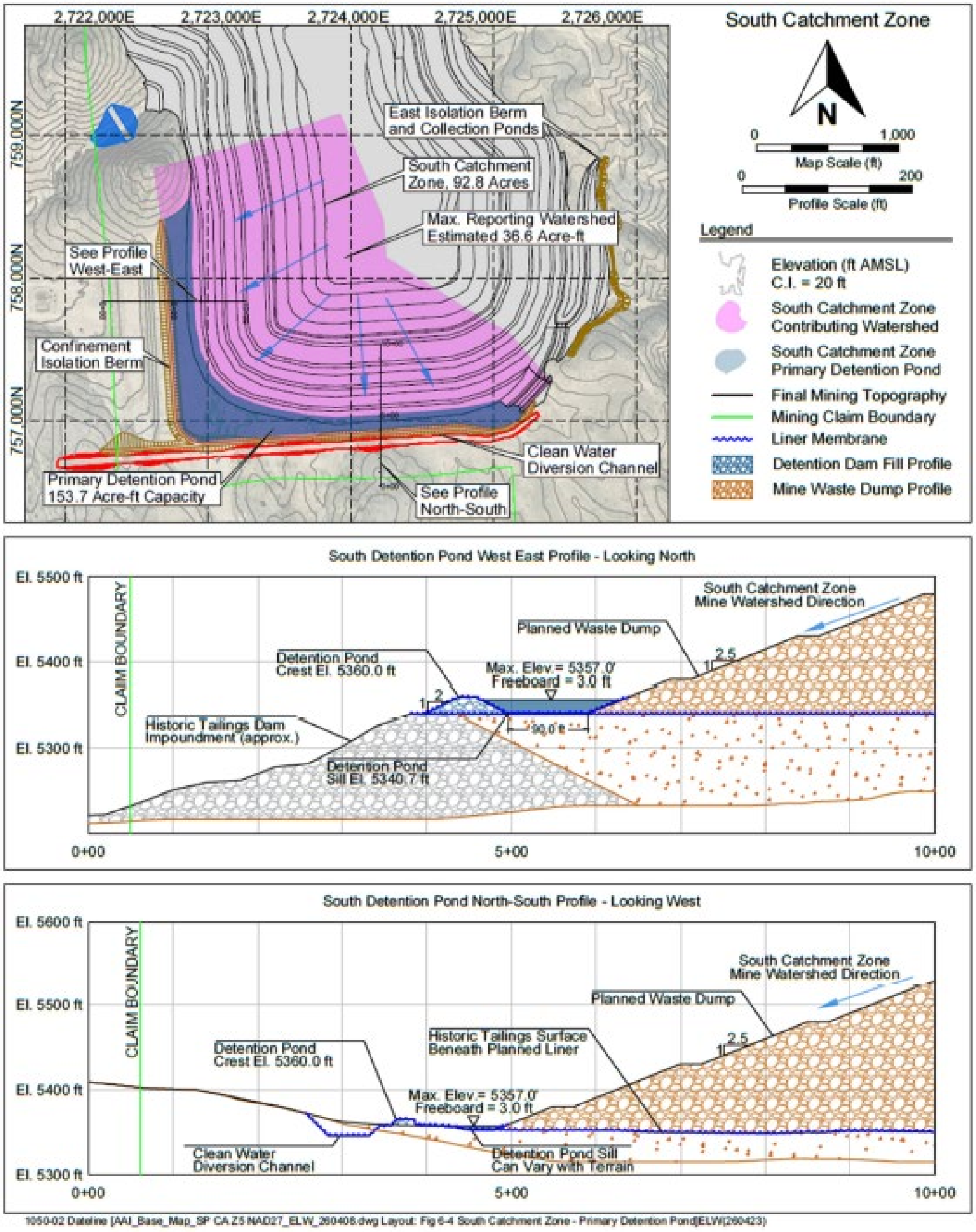
Source: Prepared by Agapito Associates, 2026

Figure 52 West Retention Pond



Source: Prepared by Agapito Associates, 2026

Figure 53 East Retention Pond



Source: Prepared by Agapito Associates, 2026

Figure 54 Primary Retention Pond

13.6.7.2 Explosives and AN Storage

The explosives storage facilities will be located on levelled pads in an area southeast of the existing tailings dam and new waste rock dump. The site is approximately 700 metres from the nearest infrastructure at the mine contractor's area and over 1,300 metres from the pits. It is only accessible from the mine contractor's area so that any traffic must pass through the site entry security. It is visually in line with the mine contractor's office.

The explosives magazine facility will include the following:

- Bunded container style magazine storage
  - High explosives e.g. boosters and presplit cartridges
  - Detonators
  - Parking for the mobile mixing unit (MMU)
  - Secure fencing and bunding
- Ammonium nitrate (AN) storage shed
  - A separate compound will contain a storage shed for AN prill and an auger for loading the prill into the MMU.

The magazine and AN storage will comply with relevant legislation including container size and materials, spacing, separating bund heights, lightning protection and security.

Bulk ANFO will be mixed by the explosive delivery truck (commonly termed a "mobile mixing unit" or MMU) and placed directly into the blast holes, from the onboard AN prill and diesel storage, rather than being manufactured at a dedicated facility onsite and then transported to the pit.

13.7 Mining Schedule

13.7.1 Production Strategy and Sequence

The production schedule strategy runs the mining fleet at maximum capacity to deliver the highest-grade ore possible to the mill early in the mine life while stockpiling lower grades for processing after completion of mining. Three grade bins based on the diluted Mineral Reserve were defined to facilitate this strategy:

- High Grade (HG) >= 0.75 g/t Au
- Low Grade (LG) 0.55 to 0.75 g/t Au
- Marginal Grade (MG) 0.25 to 0.55 g/t Au

This strategy maximises present value and reduces project risk by bringing forward gold production.

Mining rates are based on the estimated capacity for 120 tonne class hydraulic excavators loading 55 tonne payload truck. Practicality of the dig rates was checked against the sinking rate (benches per period) in each pit stage. Almost all benches in the schedule advance at less than 1 x 20ft bench per month.

A consequence of the strategy is that a large LG / MG stockpile is accumulated peaking at 10.3 Mt. This is reclaimed for processing at the end of mining. Loader, truck and support fleet requirements were modelled to demonstrate that gold recovered from the low-grade stockpile pays for the storage and reclamation cost as additions to the processing and G&A costs.

The general mining sequence is:

- Mining commences in the North Starter Pit in Month 7 of Year 0, six months in advance of first ore through the mill. This is necessary to mine out the barren benches at the top of the pit stage and establish starting stockpiles.
- The pushback to the North Final Pit commences in Month 1 of Year 1 to ensure continuity of ore supply when the starter pit is completed in Month 11 of Year 2.
- The South pit, which is a pushback of the existing pit, starts in Month 9 of Year 1. While there is some ore in the western wall of the South Pit, most of the ore is below the existing pit floor so it is necessary to start the South Pit relatively early to ensure continuity of ore supply when the North Final Pit is completed in Month 5 of Year 4. Mining continues in the South Pit until it is depleted in Month 3 of Year 6.

13.7.2 Production Schedule

AMDAD prepared the detailed life of mine schedule using the Geovia MineSched program. Mining and ore processing were scheduled on a monthly basis to assess sequencing issues, bench advance rates and run of mine stockpile size.

Ore delivery rates target 2.0 Mtpa. Dig rates are based on multiples of the asses dig rate for a single 120 tonne class hydraulic excavator loading 55 tonne payload trucks. The peak dig rate of approximately 1.6 Mt per month is achieved with three excavators from Month 3 of Year 1 to Month 5 of Year 4. Checks were done on the average bench area per machine over this period. These are in the range of 35,000 to 40,000 m^2 per machine, which is considered adequate for the machines to operate efficiently.

Total scheduled mining quantities for are 20.6 Mtonnes at 0.95 g/t Au and 62.3 Mtonnes of waste rock.

Processing was scheduled at 2.0 Mtonnes (2.2 Mtons) per year commencing in Month 1 of Year 1. A three month ramp up period to the full mill feed rate was allowed in Year 1. Gold production over the first six years would averages over 70 koz per year before falling to 24 koz per year for the remaining 4.5 years of low-grade stockpile reclaim. Total life of mine saleable gold production is 573 koz.

Yearly schedule totals are shown in the following table.

Project Year		0	1	2	3	4	5	6	7	8	9	10	11	Total
Opencut Mine														
Mill Feed	kt	190	4,042	4,624	4,447	3,450	2,950	927	0	0	0	0	0	**20,631**
Gold grade	g/t	0.72	0.86	0.84	0.76	0.84	1.28	2.19	0.00	0.00	0.00	0.00	0.00	**0.95**
Waste tonnes	kt	3,722	14,753	14,564	13,010	10,843	5,076	297	0	0	0	0	0	**62,264**
Total tonnes	kt	3,912	18,795	19,188	17,457	14,293	8,026	1,224	0	0	0	0	0	**82,895**
Waste: Ore Ratio	t:t	19.6	3.6	3.1	2.9	3.1	1.7	0.3	0.0	0.0	0.0	0.0	0.0	**3.0**
Stockpile Reclaim	kt	0	460	546	618	538	500	1,610	2,000	2,000	2,000	2,005	738	**13,014**
End of year ROM Stockpile	kt	190	2,356	4,974	7,422	8,872	9,822	8,743	6,743	4,743	2,743	738	0	
Process Plant Feed	kt	0	1,877	2,005	2,000	2,000	2,000	2,005	2,000	2,000	2,000	2,005	738	**20,631**
Gold grade	g/t	0.00	1.20	1.25	1.22	1.18	1.64	1.42	0.57	0.41	0.41	0.41	0.41	**0.95**
GOLD PRODUCED	**koz**	**0**	**66**	**73**	**71**	**69**	**96**	**83**	**33**	**24**	**24**	**24**	**9**	**573**

Table 56 Annual Mining Schedule Quantities

13.7.3 Production Schedule by Mineral Resource Category

The production and Mineral Reserve Estimate do not include Inferred Mineral Resources, which have too low a geological confidence to be included in the FS. Over the mine life, 68% of the tonnes mined are Proved and 32% are Probable Mineral Reserves.

13.8 Mining Equipment

13.8.1 Drilling

Blastholes will be drilled using crawler-mounted down the hole hammer (DTHH) drill rigs.

Drill and blast will generally be conducted on 20’ (6.1m) high benches, drilling 6” (152mm) holes. Assumed drilling productivities are based on similar operations and include move and set up between holes.

13.8.2 Blasting

The mining contractor will supply suitable explosives trucks capable of mixing and delivering ANFO, heavy ANFO, and emulsion into the blastholes.

13.8.3 Loading

The preferred configuration of the loading fleet is as follows:

- 120 tonne class primary excavator
  - Diesel-hydraulic
  - Utilised for excavation of waste and ore zones
  - Capable of selective mining
  - Also capable of digging large sumps
- Large wheel loader
  - Loading the crusher from short term ROM stockpiles, rehandling ore into trucks from the long-term ROM stockpile and loading dewatered tailings into trucks for haulage to the dry stacked tails cell
  - Capable of loading the same trucks as the primary excavator

13.8.4 Hauling

The FS schedule and mining costs are based on 55 tonne payload articulated dump selected on the following criteria:

- Matches well with 120 tonne primary excavators,
- Capable of being loaded by the secondary excavator and ROM wheel loader,
- Allows 60ft ramp widths with two-way haulage, and
- Works well on narrow upper benches and tight final benches at the pit base.

An example is the 55-tonne capacity Volvo A60H articulated dump truck.

13.8.5 Support Fleet

The primary fleet is supported by the machines listed below. The fleet size and cost estimate is based on operating hours, operator numbers and operating costs for all the major equipment items and applies contingency factors to allow for the smaller intermittently used items.

- Water truck
- Motor Grader
- Bulldozers - pit, waste rock dump and tails cell
- Crushing and screening plant
- Service truck
- Fuel truck
- Integrated tool carrier / small loader / tyre handler
- Cranes
- Light vehicles
- Lighting plants
- Pumps

13.8.6 Fleet Schedule

Haulage of ore and waste rock is typically the largest component of the mining cost because it requires the largest number of units, the largest number of operators and maintenance crew. Truck fleets were modelled by defining individual haul routes:

- Ore and waste rock haul routes from each mining bench to the ROM, long term stockpile and waste rock dump,
- Dewatered tailings hauled from stockpile to the tails cell, and
- Reclaimed ore hauled from the long-term stockpile to the crusher

Using the haul modelling tools in MineSched software, speeds and truck hours through each period over the mine life were estimated. The haul model tracks growth of the waste rock dump, tails cell and long-term stockpile as well as changing mine face positions to provide a realistic assessment of required truck hours, which in turn is used to estimate truck fleet numbers.

Another key to reducing mining cost is to maximise fleet utilisation. The machines have fixed costs associated with ownership and labour. If the machine is not working, the fixed costs are still incurred even though it is not contributing to revenue generating work. The fleet schedule mitigates fixed ownership costs by selecting a loader / truck combination that matches the required production rate as closely as possible.

Labour fixed costs are based on a four-panel roster to achieve 2 x 12 hour shifts per day, 7 days per week. For example, a single truck requires four operators to cover the roster.

Project Year	0	1	2	3	4	5	6	7	8	9	10	11
Excavators 120t	2	3	3	3	2	1	0	0	0	0	0	0
Front End Loaders	1	2	2	2	2	2	2	2	2	2	2	2
Trucks 55t	15	23	22	18	13	9	2	2	2	2	2	2
Blast Hole Drills	2	3	3	3	2	1	0	0	0	0	0	0
Bulldozers	4	7	7	6	5	4	1	1	1	1	1	1
Motor Graders	2	3	3	3	2	2	1	1	1	1	1	1
Water Carts	2	3	3	3	2	2	1	1	1	1	1	1

Table 57 Life of Mine major Equipment Fleet schedule

13.9 Mining Workforce

Mining workforce numbers are estimated as follows:

- Machine operators built up against the mining fleet on a 4 panel 2 x 12-hour shift, 7 days per week roster,
- Maintenance crews based on 60% of the operator workforce,
- Non -operator roles such as the blast crew, and

- Management, technical and supervisory roles.

Project Year	0	1	2	3	4	5	6	7	8	9	10	11
Mine Workforce												
Operators' workforce	112	176	172	152	116	88	28	28	28	28	28	28
Maintenance workforce	69	111	108	97	73	56	19	19	19	19	19	19
Explosives Crew	10	10	10	10	10	10	0	0	0	0	0	0
Management / Technical / Supervision	32	32	32	32	32	32	10	10	10	10	10	10
Total Mine Workforce	**223**	**329**	**322**	**291**	**231**	**186**	**57**	**57**	**57**	**57**	**57**	**57**

Table 58 Personnel schedule – Owner's Team (Mine Management and Technical Services)

The mining contractor accounts for most of the estimated mining workforce. The owner's team consists of 22 personnel covering mine management, mining engineering, geology, surveying and clerical. The other 10 management roles are for the mining contractor's operations and maintenance management and fleet supervision. The costs of the contractor's staff are included in a fixed monthly charge.

13.10 Diesel Fuel Consumption

At full production, diesel fuel consumption in the mine and tailings haulage is expected to average 196,000 gallons per month.

Project Year	0	1	2	3	4	5	6	7	8	9	10	11
Diesel Consumption												
klitres	1,860	9,106	8,912	8,127	6,605	4,642	2,360	1,918	1,895	1,869	1,843	696
kgallons	491	2,406	2,355	2,147	1,745	1,226	624	507	501	494	487	184

Table 59 Mine Fleet Annual Disel Fuel Consumption

13.11 Future Opportunities

13.11.1 Inferred Mineral Resources

The FS production schedule and Mineral Reserve are based entirely on Measured and Indicated Mineral Resources, which converted to Proved and Probable Mineral Reserves. The pit design includes 2.8 Mt of Inferred Mineral Resource. If grade control during mining converts any of this to ore, it has potential to add up to 55 koz of saleable gold.

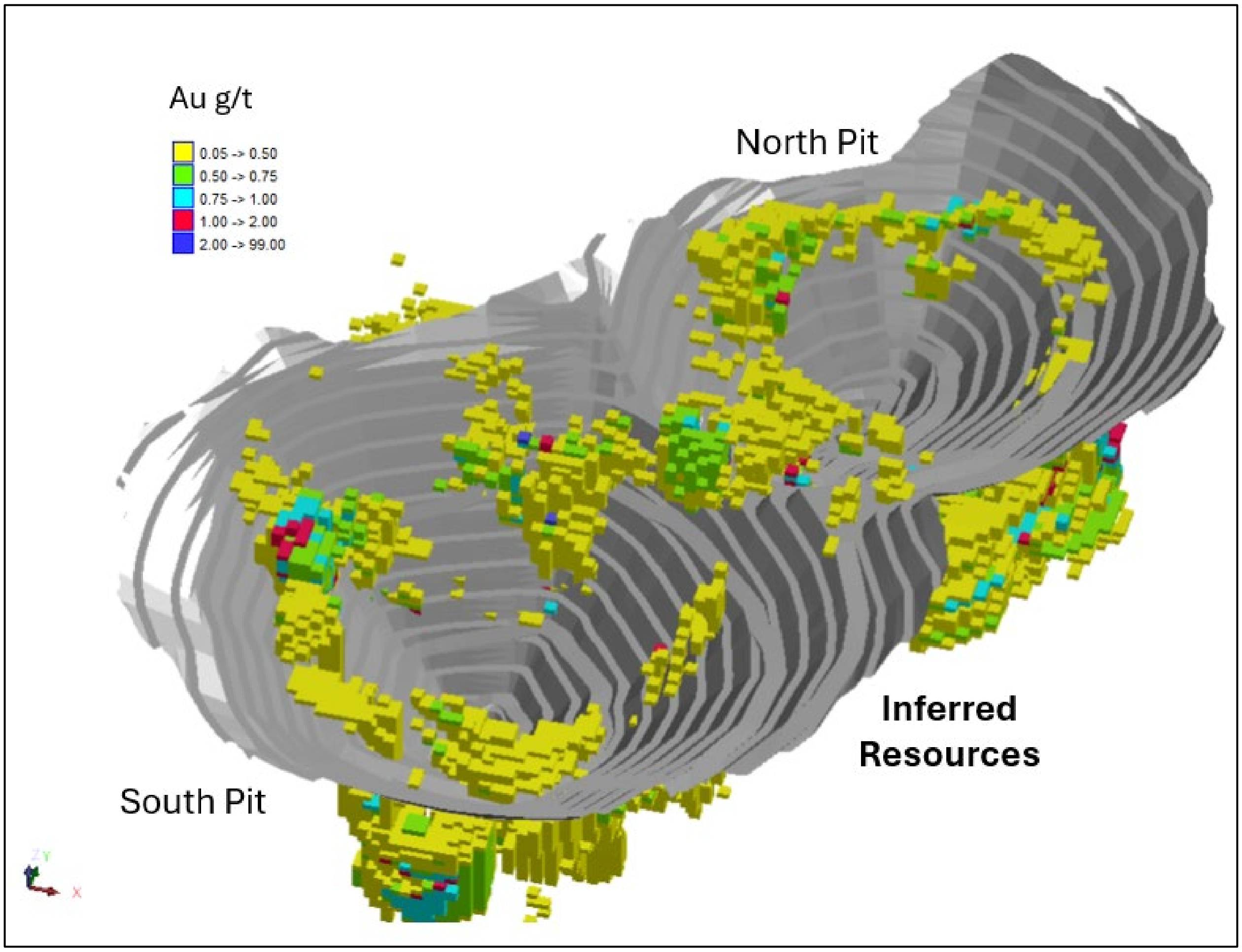


Figure 55 Inferred Mineral Resources contained within the designed pit shells

13.11.1.1 Underground Mining

Underground optimisation analyses during the FS identified a sublevel caving (SLC) target below the eastern wall of the final North Pit and long hole stoping targets below the final South Pit. On the currently defined Mineral Resource the targets have potential to add over 50 koz of gold production. The North Pit SLC target is being currently drilled and is open at depth and north along strike.

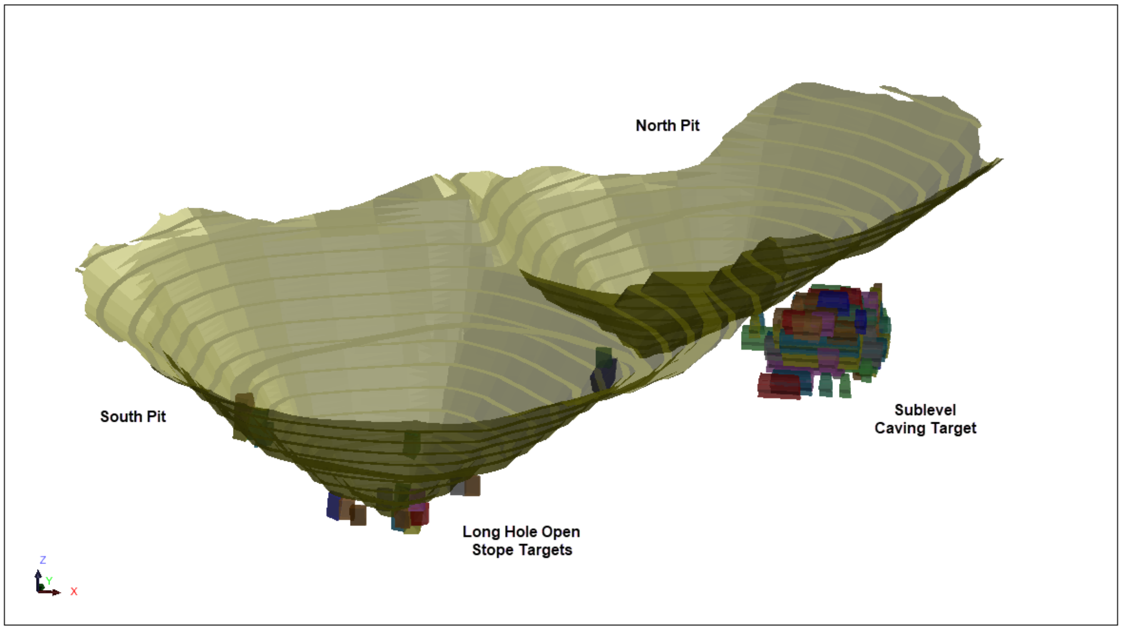


Figure 56 Underground Mining Targets

14. RECOVERY METHODS

14.1 Introduction

The Colosseum Gold Project has been designed on the basis of 2 Mtpa (metric) gold milling and carbon in leach circuit using a conventional cyanide leaching operation. The tailings from the cyanide leach will be detoxified using the INCO – SO_2/ Air process prior to filtration for dry stacking of 15 to 20% moisture cake product.

The processing plant has been designed to operate at a nominal treatment rate of 250 tph based on 8,000 operating hours per year or a grinding circuit utilization rate of 91.3% (based on 365 available operating days per year) The crushing circuit has been designed at a higher nominal treatment rate of 304 tph based on 6,570 operating hours per year or an annual utilization rate of 76% (based on 365 available operating days per year).

The project will have the following key operating areas

- Primary crushing and coarse ore stockpile;
- Ore storage and reclaim;
- Grinding and classification;
- Leach feed thickening;
- Leaching and adsorption (Carbon-In-Leach);
- Elution and gold recovery;
- Tailings thickening and detoxification;
- Detoxified tailings filtration;
- Reagent mixing, storage and distribution;
- Electrical power and control systems;
- Water and air services.

14.2 Process Design Criteria

A detailed design criteria has been developed for the 6000 tpd throughput configuration. The process plant has used the following key design criteria listed in Table 58.

Description	Unit	6 ktpd
Annual Ore Feed Rate (ROM feed)	Mtpa	2.0
Operating Days per Year	d/a	365
Daily Ore Feed Rate (ROM feed)	tpd	6,000
ROM Feed F_{100} Size to Primary Crushing	mm	800
Primary Crusher P_{80}	mm	140
Crushing Rate (6,570 hours per year)	tph	304
Milling Rate (8,000 hours per year)	tph	250
Mill Circuit Circulating Load	%	250
Gold Head Grade (Mill Feed) – Design	g Au/t	1.20
Design Ore Specific Gravity	t/m^3	2.70
Design Abrasion Index	-	0.325
Design Crushing Work Index	kWh/t	11.8
SMC Drop Weight Index	kWh/m^3	8.38
Design Rod Mill Work Index	kWh/t	20.5
Design Ball Mill Work Index	kWh/t	20.1
Mill Grind P_{80} Size	µm	106
Leach System	-	CIL
Leach Slurry Density	% solids w/w	50
Total Leach and Adsorption Time - Design	H	27
Elution System	-	Split AARL
Final Tailings Cyanide Destruction Type	-	Air/SO_2
Overall Recovery (LOM Average)	% Au	89.3

Table 60 Key Process Design Criteria

The historical test work results from the 1984 and 1987 campaigns, along with additional metallurgical and process testing conducted in 2025, were used to develop the process design criteria, process flowsheet and mass balance.

14.3 Process Flow Diagram Development

The test work programme has followed a similar approach to the previous work that has been conducted and to a greater degree the design basis of the previous operation. To keep dust to a minimum, a SABC circuit was selected and confirmed based on commination data for the ore type as being suitable.

A conventional cyanide leach using a carbon in leach (CIL) circuit was examined in the new test work and confirmed from historical work. The process will utilise seven (7) CIL stages with the option of adding a pre-oxidation tank at the start of the circuit later if additional pre-oxidation is required.

The desorption circuit will use a split AARL acid and elution system, as the raw water quality was excellent and can be used by the AARL process. The system has been sized for a 5-tonne column with associated gold room.

Cyanide detoxification via the INCO process has been applied in the design and will meet the target requirement of 1 ppm WAD CN target.

Dry stack tailings will be required for the process. Testing of both pressure and vacuum filtration have both demonstrated that a sub 20% moisture cake can be developed. The process will include 2 x 182M^2 HBVF with vibration rollers to ensure the cake is under 20% moisture for materials handling and stacking of cake. The cake will be paddock dumped and moved by dozer over time. The arid climate in the area will allow for a high evaporation rate hence there was some relaxation on the moisture content to 20% moisture at nominal operation and a design basis of 15% moisture.

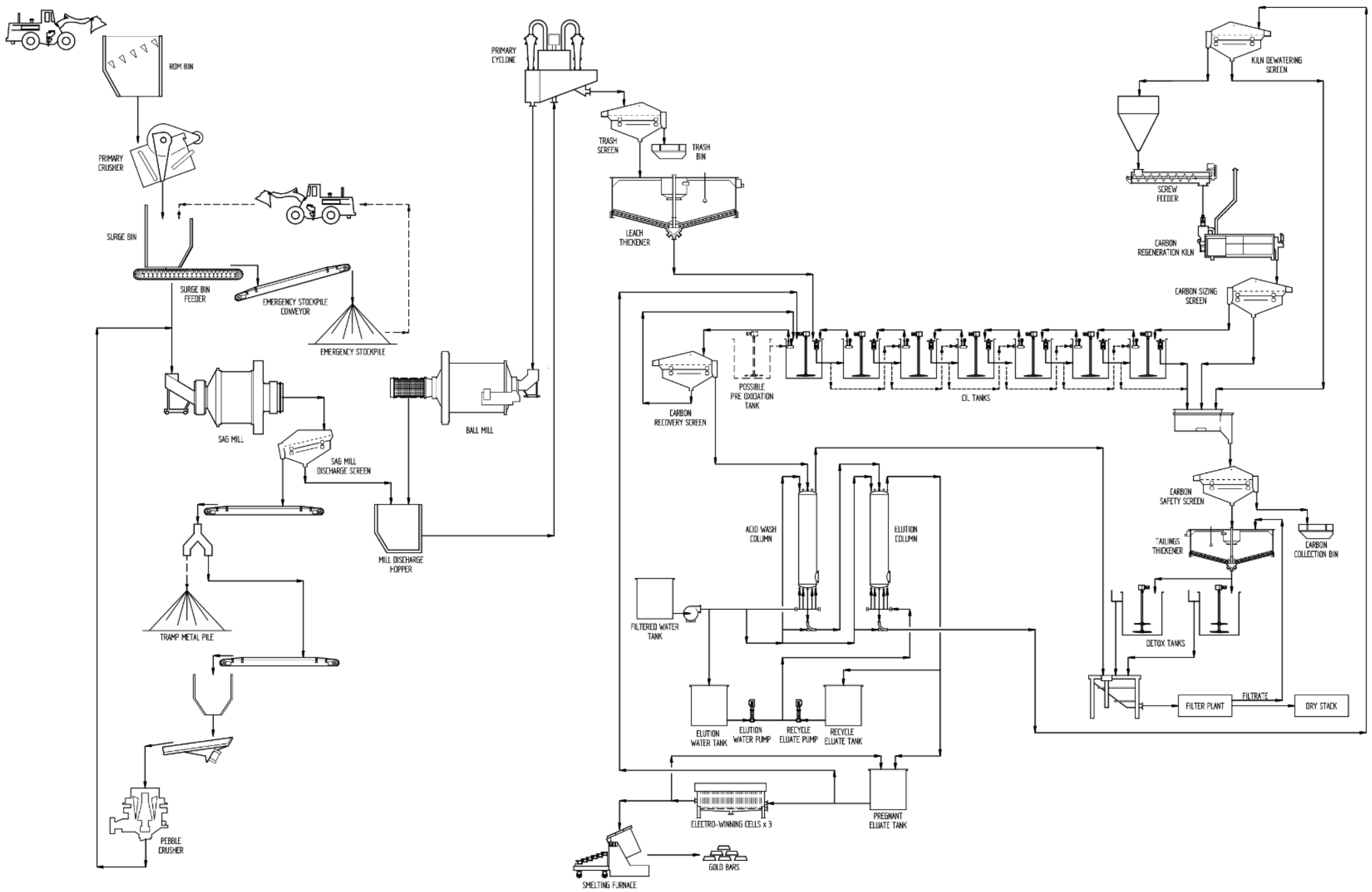


Figure 57 Process Plant Flowsheet

14.4 Description of Process Areas

The Colosseum Gold Project has been designed on the basis of 2 Mtpa (metric) gold milling and carbon in leach circuit using a conventional cyanide leaching operation. The tailings from the cyanide leach will be detoxified using the INCO – SO_2/ Air process prior to filtration for dry stacking of a 15 to 20% moisture cake product.

The project will have the following key operating areas

- Primary crushing and coarse ore stockpile;
- Ore storage and reclaim;
- Grinding and classification;
- Leach feed thickening;
- Leaching and adsorption (Carbon-In-Leach);
- Elution and gold recovery;
- Tailings thickening and detoxification;
- Detoxified tailings filtration;
- Reagent mixing, storage and distribution;
- Electrical power and control systems;
- Water and air services.

14.4.1 Primary Crushing

The crushing circuit will be a conventional single stage jaw crusher operating in open circuit. Product from the crushing circuit will be conveyed to a coarse ore stockpile. The circuit will crush 350 dry tonnes per hour of open pit ore to a product size P_{80} of 140 mm.

The Run-of-Mine (ROM) ore will be loaded into the crushing circuit feed bin (ROM bin) by a front-end loader. A static grizzly with 800 mm square apertures will be fitted to the ROM. The ROM ore will be drawn from the ROM bin at a controlled rate by a variable speed apron feeder and discharged into the jaw crusher. The crusher product will be discharged onto the crusher discharge conveyor. A dust collector will be positioned at the end of the crusher discharge for dust management.

The crusher discharge conveyor will feed to the stockpile feed conveyor. The stockpile feed conveyor will discharge onto the coarse ore stockpile. The stockpile will have a live storage capacity of approximately 6,000 dry tonnes equivalent to 24 hours of milling time.

14.4.2 *Coarse Ore Storage and Handling*

Crushed ore will be reclaimed from the coarse ore stockpile via a tunnel under the stockpile. The tunnel will be fitted with two apron feeders to reclaim the ore and feed it onto the mill feed conveyor. The mill feed conveyor will feed the grinding circuit.

A 100-tonne lime silo, fitted with a variable speed rotary valve and screw feeder will dose lime onto the mill feed conveyor to provide protective pH control in the leaching and adsorption circuit. Lime will be pneumatically transferred into the silo from a delivery truck.

The reclaim area and the lime silo area will each be serviced by a dedicated vertical spindle centrifugal slurry pump for clean-up purposes.

14.4.3 *Grinding and Classification*

The grinding circuit will consist of a conventional two stage milling circuit. The first stage will be a grate discharge semi-autogenous (SAG) mill in open circuit with a pebble crusher, and the second stage will be an overflow discharge ball mill in closed circuit with a cyclone cluster. The circuit will grind 250 dry tonnes per hour of open pit ore to a product size P_{80} of 106 µm.

The primary SAG mill will be an 8.23 m diameter (IS) by 3.35 m long (EGL) variable speed mill fitted with a 4,400 kW motor. SAG mill discharge will be screened on a horizontal, wet vibrating screen with apertures of 10 mm by 40 mm. The screen oversize will be conveyed to a pebble crushing circuit. The pebble crushing circuit will consist of a feed bin and vibrating feeder feeding a short head cone crusher. The pebble crusher discharge will be returned to the mill feed conveyor.

The SAG mill discharge screen undersize will combine with the ball mill discharge pulp in the mill discharge hopper. Centrifugal slurry pumps, arranged in a duty/standby configuration, will pump the combined pulp to a cyclone cluster for classification. The cyclone cluster will consist of twelve 400 mm diameter cyclones (eight duty cyclones and two standby cyclones). Cyclone overflow will gravitate to the trash screen, while cyclone underflow will report to the ball mill.

The ball mill will be a 5.5 m diameter (IS) by 8.70 m long (EGL) overflow discharge with a 4,400 kW motor. The ball mill will discharge onto a trommel screen with 10 mm x 15 mm apertures. Trommel screen oversize will discharge into the ball mill scats bunker whilst trommel screen undersize will discharge into the mill discharge hopper. The grinding area will be serviced by a drive-in sump and two vertical spindle centrifugal slurry pumps for clean-up.

14.4.4 *Leaching and Adsorption*

After screening to remove trash, the cyclone overflow from the grinding circuit will be thickened and then leached with cyanide in a seven-stage conventional carbon in leach (CIL) circuit.

The cyclone overflow from the grinding circuit will gravitate to horizontal, wet vibrating trash screen. The screen aperture will be 0.8 mm. Trash screen oversize will discharge into the trash bin for periodical removal. Trash screen undersize will gravitate to the leach thickener.

Leach thickener feed will be dosed with flocculant and thickened in the 26 m diameter Hi-rate thickener to 50% solids (w/w). The thickener underflow will be pumped by one of two centrifugal slurry pumps, arranged in a duty/standby configuration, to the CIL tanks. The thickener overflow will gravitate to the process water pond via a sedimentation pond.

The CIL circuit will consist of six 1,309 m^3 CIL tanks with a nominal pulp residence time of 27 hours. Cyanide will be stage dosed into the first three CIL tanks as required. Pipe spargers will be installed in all CIL tanks to enable the injection of additional oxygen.

Each CIL tank will be fitted with a 9 m^2 mechanically wiped, intertank screen with 1.0 mm aperture stainless steel wedge wire to retain carbon. Carbon will be advanced through the CIL circuit counter current to the pulp, on a batch basis, by recessed impeller pumps. Loaded carbon from the first stage of the CIL will be pumped to the loaded carbon screen. The loaded carbon screen will be a horizontal, wet vibrating screen with 0.8 mm apertures. Loaded carbon from the loaded carbon screen will gravitate into the acid wash column.

The design advance rate for the circuit will be 5 t/d and six strips per week based on a head grade of 1.2g/t Au. Once the elution is complete the carbon will be transferred to a regeneration kiln. Regenerated carbon from the kiln (or barren carbon directly from the elution column) will be returned to the circuit via the barren carbon screen. The barren carbon screen will be a horizontal, wet vibrating screen. The leach thickener area will be serviced by one vertical spindle centrifugal slurry pump, and the CIL area will be serviced by two vertical spindle centrifugal slurry pumps for clean-up.

14.4.5 Gold Recovery

The batch of pregnant eluate will be discharged from the elution column, via the eluate filters and recovery heat exchanger to the pregnant eluate tank. The solution will then be pumped to three 800 mm by 800 mm electrowinning cells operating in parallel. The electrowinning cell discharge solution will gravitate back to the pregnant eluate tank. Barren solution will be pumped to the leach and adsorption circuit.

The gold recovered by electrowinning will be pressure washed from the electrowinning cell cathodes, filtered and dried prior to smelting to produce gold doré. A safe and a vault will be provided in the gold room to store the valuable products.

14.4.6 *Tailings Thickening and Detoxification*

Leach circuit tails will pass over a carbon safety screen prior to gravitating to a tailing thickener for the recovery of cyanide solution to the process water tank. Process water from both the leach feed thickener and the tailings thickener will report to the same common process water tank. Underflow from the tailing thickener at a nominal 55% solids will be pumped to the first of two detoxification tanks. An INCO tailing detoxification system will dose liquid sodium metabisulphite (SMBS) and copper sulphate solutions to the first detox tank. A mixture of SO_2 and air in the presence of a soluble copper catalyst oxidizes free cyanide and WAD metal complexes in aqueous solution to the less toxic compound cyanate (OCN-).

$CN^- + SO_2 + O_2 + H_2O \rightarrow CNO^- + H_2SO_4$

The optimum pH for the reaction is 9, but it will proceed to a significant extent between pH 7.5 and 9.5. Normally the pH is maintained between pH 8.0 to 9.0 with the addition of lime.

Sulphuric acid generated during cyanide oxidation and other oxidation reactions is neutralised with lime in the reactor to form gypsum, as follows:

$H_2SO_4 + Ca(OH)_2 \rightarrow CaSO_4\ 2H_2O$

Detoxed tailings will be pumped to the filter feed tanks.

14.4.7 *Tailings Filtration*

Tailings filtration will be undertaken by two 182 m^2 horizontal vacuum belt filters (HVBF) with vibration rollers. The filters will produce a tailings cake at a moisture of 15 to 20%. Residual WAD CN to the cake limit of 18 mg/l and solution limit of 1 mg/l will be maintained by the detoxification circuit. Filtrate will be pumped back to the tailing's thickener for circuit water recovery.

14.4.8 Grinding Media and Reagent Management

The following process additives will be necessary to operate the processing facilities:

Process Additive	Packaging	Mixing	Storage	Dosing
Steel balls	Bulk	-	Bunkers	Emergency feeder (SAG mill); Kibble and 2t hoist (ball mill)
Quicklime	Bulk	-	100 t	Variable speed drive rotary valves and fixed speed drive screw feeders
Flocculant	750 kg bags	Automated batch system	30.0 m³	Variable speed drive dosing pumps
Sodium cyanide	2x 20t solid to solution in isotainers	-100 m³	100 m³	Circulating pumps and dosing valves
Oxygen	-	-3tpd	-	Generated on-site by PSA plant
Sodium hydroxide	22.5 t liquid solution	-	30 m³	Circulating pumps and dosing valves
Hydrochloric acid	22.5 t liquid solution	-	30 m³	Dosing pumps
Sodium Metabisulphite	Bulk Bags 1 tonne	10m³	20m³	Dosing pumps
Copper Sulphate	Bulk Bags 1 tonne	7m³	12m³	Dosing pumps
Hydrated lime	Bulk Bags 1 tonnes	5m³	20m³	Circulating Pump
Liquid petroleum gas	Bulk	-	66 m³	Gas Control Valves
Smelting fluxes	25 kg bags on a pallet	-	4 t	Manual mixing

Table 61 Reagent Mixing Systems

14.4.9 Process Control System

The process plant control system will be a programmable logic controller (PLC) based system. The human machine interface (HMI) will utilize standard personal computers running Citect SCADA software to facilitate control. The process facility will be controlled from the centrally located main control room in the plant area.

4-20 mA analogue I/O signals will predominantly be associated with process instrumentation and control, including flow, pressure, density and the control of modulating valves and actuators, and variable speed drives.

Digital I/O will generally be based on 24 Visual Display Cabinets (VDC) hardwired signals, typically associated with the status and control of drives, valves and actuators and mechanical plant.

In each area, the I/O associated with the Motor Control Centres (MCC) will be installed in one or more tiers of the MCC and will be hard wired to the starter modules within the MCC. The digital and analogue I/O associated with the process instrumentation will be wired to Process Control Cubicles (PCCs).

Two Visual Display Units (VDUs) will be installed within the control room to provide operators with HMI. These units will present graphical process information in the form of trends, mimic pages, alarm summaries, logs and reports. This HMI will also enable the operator to start and stop equipment remotely, control variable speed drives and alter process set points.

Controller parameters will be adjusted from the controller faceplate, and this adjustment can be password protected to prevent unauthorised changes. Display screens will be configured for the trending of individual or related parameters, and alarm pages will be developed to facilitate the setting of alarm points specific to various parameters. All analogue input signals, including outputs from flow, pressure, temperature and weighing instruments will be displayed appropriately on mimic pages. A short-term trend plot for each input and output from the system can be provided where required on the mimic pages.

The analogue and digital I/O associated with the plant instrumentation will be cabled to one or more PCC within the plant areas. These units will be located within the area switch rooms and house the PLC racks, instrumentation power supplies and communication hardware. Communication between these units and the control system HMI will be via Ethernet, using fibre optic or copper cable as appropriate.

External and emergency communication will be available in the control room.

14.5 Main Process Plant Equipment

The main processing equipment will be:

• Primary C-130 Jaw Crusher	185 kW
• SAG Mill – 8.5m (IS) Ø x 3.35m EGL	4,400 kW
• Ball Mill – 5.5m (IS) Ø x 8.9m EGL	4,440 kW
• Pebble Crusher HP200	150 kW
• Warman Cavex 400CVX10 Cyclones	
• Leach Feed Thickener – 26m Ø	11 kW
• 7 Leach Circuit Tanks and Agitators – 1309 m^3	55 kW
• 5 tonne Elution Circuit and Goldroom	
• Cyanide Recovery Thickener – 26m Ø	11 kW
• Detoxification Circuit Tanks – 300 m^3	55 kW
• Tails Filters – 2 x J405-15V3-182m2 with 2 x Viper™	620 kW

15. PROJECT INFRASTRUCTURE

15.1 Mine

Mining infrastructure and facilities will mostly be provided by the mining contractor. It will include:

- Workshops,
- Vehicle washdown facility,
- Offices,
- Fuel storage,
- Explosives magazine and ammonium nitrate storage,
- Water truck fill points.

Earthworks for the mine contractor's area will be done by Dateline. Dateline will supply services including electricity and fire water to the edge of the contractor's area. Costs for these items are included in the site infrastructure estimate.

The mining contractor will be responsible for mobilising the equipment to site, construction, deconstruction and demobilisation of the mining infrastructure.

15.2 Project Services

15.2.1 Water Supply

Raw water supply for the project is derived from two existing wells located near to the Ivanpah substation. The water quality is extremely high (very low TDS and near neutral pH) and is suitable for use in the process plant without further treatment. The water is also suitable for potable purposes following ultra filtration and sterilisation. Well #2 is used as the primary water source, whilst well #1 is used for back-up (standby). Pump tests performed on well #2 showed the well to be capable of providing water to the project in the necessary quantities. Two (one duty, one standby) new high pressure lift pumps are provided to transfer raw water from the well site to the process site raw water tank. A pipeline is installed to convey the water from the pumps to the site raw water tank. The pipeline is routed beside the site access road generally buried in the safety berm located at the edge of the road. Electrical supply for the well and lift pumps is existing, drawing from the Southern California Edison 34.5 kV supply. New electrical infrastructure is provided to power and control the raw water pumps.

15.2.2 Power Supply

Access to power will be from the local Southern California Edison grid (34.5 kV) The power access is located near to the SoCal Edison Ivanpah substation. Power is transmitted to the project site (at 34.5 kV) via an overhead transmission line which follows the alignment of the main access road (approx. 10km). Power is reticulated on site at 13.8 kV. The system includes the utility metering at the process plant, HV switchgear and the following reticulation of power to the process plant and the infrastructure.

15.2.3 Communications

On site communications are facilitated using fibre optic cable connections. Such communication includes process control and administrative functions. Site connection to the internet is Starlink connections are proposed for connection to the internet.

15.2.4 Tailings Storage Facilities

A horizontal vacuum belt filter press has been incorporated into the process plant design, with tailings material dewatered and reduced to a moisture level of <20%. The tailings filter cake material will then be trucked and placed in a tailings cell within the waste rock dump. The tailings cell will be enclosed by mine waste rock at all times and will eventually be fully encapsulated.

The existing tailings storage facility will be covered by waste rock dumps in the current mine plan. All tailings for the new project will be stored in the tailings cell described above.

15.2.5 Waste Disposal

Mine waste rock storage will be in a waste rock dump located west and southwest of the South Pit. The waste rock dump will incorporate the dry stack tailings cell.

Physical man-made waste from operations will be collected in designated waste bins and periodically removed from site. No man-made waste is intended to be disposed of on-site via burial or incineration.

Sewage waste will be collected in septic tanks and regularly pumped out and removed from site.

15.3 Project Infrastructure

The Colosseum project is serviced by the following infrastructure elements.

- Administration and plant operations building;
- Gatehouse;
- Workshop / warehouse building;
- First aid building;
- Laboratory building;
- Potable water reticulation;
- Fire water reticulation;
- Power reticulation;
- Communications reticulation;

- Sewage system;

15.4 Permanent Accommodation

Site permanent accommodation is not proposed for the Colosseum project. Construction and operations personnel are accommodated locally in the nearby township of Primm.

15.5 Site Establishment and Early Works

Dateline has upgraded the access road from the bitumen road (Colosseum Mine Road) to the mine site. The access road is suitable for use during construction and operation.

The water bores, Colosseum #1 and #2 have been investigated and flow tested to ensure their suitability for the Project. Pipework connecting the bores to the site will follow the access road to site.

Front End Engineering & Design (FEED) studies have commenced for the Colosseum Project, led by GRES. The early commencement ensures timely delivery of the project is possible.

15.6 Management, Engineering, EPCM Services

Process plant engineering, procurement and construction management services are provided by an engineering house widely experienced in the design and construction of minerals processing plants generally but more particularly gold processing plants.

15.7 Preproduction Costs

Ore and waste mining commence six months prior to first ore through the process plant. This advance work to build initial run of mine (ROM) stockpiles and develop adequate bench areas in the pits is to ensure that once production commences the mine is able to continuously deliver ore to the process plant at target tonnes and grade.

15.8 Electric Power

Electric power is sourced from the SoCal Edison grid, as explained by section 15.2.2.

16. MARKET STUDIES AND CONTRACTS

16.1 Gold Market

Gold is a critical element with unique properties that enhance portfolio diversification, serve as a store of value, and provide a hedge against systemic risk. Gold price rises through 2024 and 2025 have continued in the first half of 2026, peaking at $5,550 per ounce in January.

Gold prices are heavily influenced by global economic conditions, including inflation rates and geopolitical tensions. Despite the U.S. Federal Reserve's tightened monetary policy, gold has continued to rise, indicating strong underlying demand and investor confidence in gold as a safe haven.

Historically, gold has responded to various global events, including financial crises and changes in monetary policy. The current trend reflects past periods where gold has strengthened amidst global uncertainties, indicating a recurring pattern of investor behaviour during economic stress.

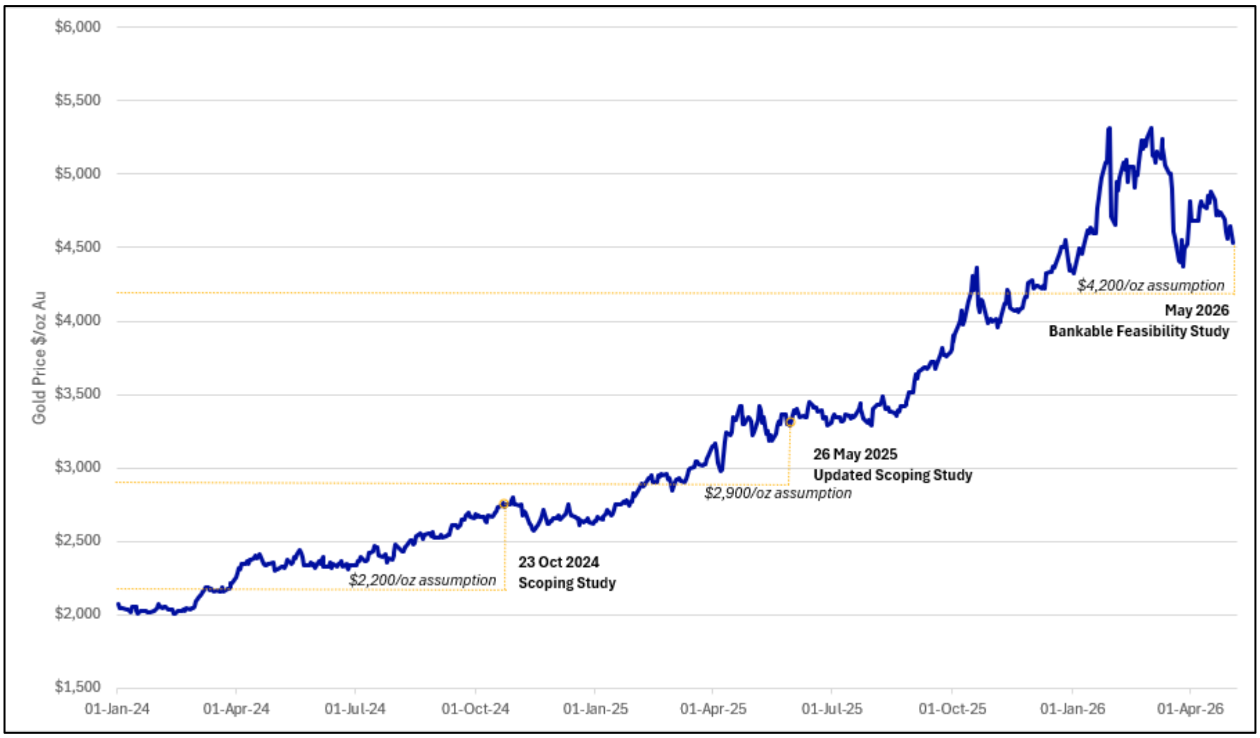


Figure 58 Gold Price Chart – January 2024 to May 2026

As at the date of this report, the gold spot price is $4,700/oz, 12% above the gold price assumption of $4,200/oz used for optimising the pits and developing the schedule.

16.2 Gold Contracts

The Company has entered into a small gold sales contract associated with an earlier financing transaction. Under the deal, Dateline agrees to sell 500 ounces per annum for six years at a discounted price to the buyer.

If the Company does not enter into further sales contracts for gold produced before production commences, it intends to produce a gold doré on site and sell it on the spot market via a gold smelter.

17. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

17.1 Environmental Studies

Environmental review for modern mining at Colosseum originates from the federal and county NEPA and CEQA process completed for the Amselco development in the mid-1980s. The Draft Environmental Impact Report and Draft Environmental Impact Statement for the New Colosseum Mine were issued in March 1985 (Bureau of Land Management 1985a). The Final Environmental Impact Report and Final Environmental Impact Statement for the AMSELCO Colosseum Project were approved in July 1985 by the US Department of the Interior Bureau of Land Management and San Bernardino County (Bureau of Land Management 1985b).

Mining and processing during the 1988 to 1993 operating period proceeded under the approved environmental documentation and the approved operating and reclamation framework established for the project during that approval cycle. Dateline treats these approvals, and the associated environmental safeguards and reclamation protocols embedded within them, as the minimum standard governing the project's redevelopment scope.

Regulatory commentary from the US Environmental Protection Agency exists for Colosseum in the form of EPA site visit material associated with mine oversight and reclamation review. EPA records reference the Draft EIR and Draft EIS and the Final EIR and Final EIS documentation for the Colosseum Mine.

Dateline notes that when mining and processing operations were suspended in 1993, the EIS and Plan of Operations remained in place and would apply to the resumption of activities at Colosseum.

17.1.1 Flora and Vegetation

Several defined plant communities occur in the study region. These are described according to classifications developed by Prigge (1975) in his study of Clark Mountain flora, supplemented with information, particularly common names, from Munz (1974). The vegetation in the immediate project area consists principally of a Pinyan-Juniper Woodland association. This community occurs from the 1550-m (5100 ft) elevation to the summit of Clark Mountain. Its indicator species are pinyon pine (Pinus monoohvlla, Utah juniper (Juniperus osteosperma), mountain mahogany (Cercocarpus intricatus), Mormon tea (Ephedra viridis), service-berry (Amelanchier covillei), and sagebrush (Artemisia nova).

To the west a Joshua Tree Woodland plant community occurs between the elevations of 1110 and 1500 m (3500-4900 ft), with indicator species of Joshua tree (Yucca brevifolia var. jaegeriana), Mojave yucca (Y. schidigera), goldenhead (Acamptopappus shocklevi), paperflower (Psilostroohe cooperi), and Haplooaoous linearifolius. Another major plant community in the study area is Blackbush Scrub, indicated by the dominance of blackbush (Coleogvne ramosissima), spiny menodora (Menodora spinescens), Mojave yucca, fleshy-fruited yucca (Y. baccata), Mormon tea (Ephedra nevadensis), and turpentine broom (Thamnosma montana).

The effect that the operations will have on vegetation was assessed in the Plan of Operations as First and Second Modified. Previous analyses concluded that there a were no species of protected vegetation in the area which may be affected by mining operations.

17.1.2 Fauna

Faunal resources recognised in the region include bighorn sheep (~canadensis nelsoni), mule deer (Odocoileus hemionus), cottontail rabbits (Sylvilagus auduboni) and jackrabbits (Lepus californicus), wood rat (Neotoma lepida), desert tortoise (Gopherus agassizi), and quail (Loohortvx gambeli), among others (Lerch, 1985).

The effect that the operations will have on wildlife was assessed in the Plan of Operations as First and Second Modified. Previous analyses concluded that there are no threatened or endangered species of wildlife which may be affected by the program.

17.1.3 National Heritages Places

The archaeological surveys undertaken for the original mine identified twenty sites of potential archaeological value within the claims area, with three identified as being within the planned mining area. A plan was initiated for the preservation of data and archaeological values contained in sites SBr-4889, -5300, and -5303, which resulted in the Bureau of Land Management, in consultation with the State Historic Preservation Officer and the Advisory Council on Historical Preservation, issuing a determination of ‘no adverse effect’ for the undertaking (Lerch and Wilke, 1985).

17.2 Waste and Tailings Disposal, Water Management

The original mining episode, between 1988-1993 involved the mining of ore and waste at Colosseum. Waste was deposited on waste dumps, whilst ore was stockpiled and/ or proceed through the processing plant. The resulting tailings material was deposited on a tailings facility to the southwest of the plant where water was captured via gravity or lost via evapotranspiration.

For this Technical Study, a Jord filter press has been incorporated into the design. Tailings material will be pressed through the filter, reducing the moisture level to <20%, with water to be captured and reused in the processing circuit. The filtered tailings material will then be trucked and co-deposited with the waste material, negating the need for a tailings facility and reducing the amount of water drawn from the Colosseum borefield.

17.2.1 *Waste Rock Disposal*

Waste rock will be mined and deposited on waste dumps adjacent to the open pits. As noted above, dried tailings material will be co-deposited with the waste rock material.

17.2.2 *Tailings Disposal*

Tailings will be filtered using a Jord filter press before being trucked and co-deposited with the waste material. No standalone tailings storage facility will be constructed, however the waste dump areas where the tailings are to be co-deposited will be lined to avoid the possibility of any leakage.

17.2.3 *Site Monitoring*

The Colosseum site will have an environmental and safety team that will be responsible for monitoring and statutory reporting with regards to the approved activities at Colosseum.

17.2.4 *Water Management*

In late 2025, Dateline entered into an option agreement to secure water and land rights supporting the Colosseum development. The agreement covers two groundwater wells (Colosseum #1 and #2) and 27 acres of real estate near the base of the Colosseum Mine Road. Dateline positions this as addressing key infrastructure requirements for development and logistics.

17.3 Permitting and Authorisations

Dateline has an existing Plan of Operations for the resumption of activities at Colosseum. It also has valid Mining Rights for the Project. The Plan of Operations and Mining Rights have been confirmed as valid by the U.S. Department of the Interior, Bureau of Land Management, National Parks Service and the San Bernardino County.

Dateline's stated redevelopment approach is to design mining, waste placement, and processing infrastructure to fit within the approved Plan of Operations and the scope of the valid rights position. This reduces permitting uncertainty relative to a greenfield development, while still requiring active compliance, agency coordination, and performance against reclamation and environmental safeguards.

17.4 Social and Community Relations

Colosseum is a brownfield site in a sparsely populated portion of the Mojave National Preserve, with the nearest large labour and services market in Las Vegas. A workforce model based on commuting from an established urban centre reduces the need for a remote village or permanent camp and shifts most social impacts toward transport management, shift scheduling, and regional procurement.

From an impact perspective, the main social and community themes for Colosseum are practical and measurable:

- Employment and contracting opportunities sourced from Las Vegas and the I-15 corridor.
- Traffic and road safety management along the upgraded access route and local road network.
- Local procurement for freight, maintenance, earthworks, and specialist technical services.
- Workforce roster design and fatigue management due to commute distance.
- Ongoing engagement with agencies managing the Preserve, and adjacent stakeholders, focused on disturbance limits, heritage protection, rehabilitation outcomes, and water stewardship.

Dateline's secured infrastructure site near the base of the Colosseum Mine Road supports logistics staging and workforce management options linked to the I-15 corridor. The detailed operating model, includes personnel parking at the infrastructure site and travelling onward via company transport, aligning with the Company's logistics plan and the project's safety management system.

17.5 Potential Emissions, Waste and Effluents Generated by the Project

The mining operations will involve removal of waste rock, removal of ore, and ore processing by milling and tank leaching. Dust emissions from the mine haul roads will be minimized by application of chemical dust suppressants and water to the haul roads.

Particulate emissions resulting from ore crushing and handling will be controlled by the use of a high efficiency wet scrubber. All other operations (milling and leaching) will either be fully enclosed or will be completely wet and will have negligible particulate emissions.

Gaseous emissions at the Colosseum Project will result from the operation of diesel haul trucks, from storage of diesel fuel in an on-site tank, and from the combustion of propane used in a small boiler at the mill.

Dateline intends to operate Colosseum within the limits of the existing Plan of Operations and valid Mining Rights, with regulatory reporting as required.

17.6 Closure and Abandonment Stage

A reclamation and closure review of the Project was developed by Dateline in support of the Technical Report Summary for resuming mining operations at Colosseum. The environmental and closure commitments have been established in the 1985 California Environmental Impact Report (EIR) and the Federal Impact Statement (EIS).

Agapito (2026) developed an Environmental and Closure Obligations Framework compiles and organizes the environmental and closure commitments established in the 1985 EIR/EIS record for the Colosseum Mine and summarizes how those commitments were implemented during the 1993 - 1995 closure period using the 1995 WESTEC closure documentation as the principal closure era reference. The framework is structured to support recommencement planning by providing a traceable register of impact findings, mitigation measures, and monitoring commitments by resource category, documenting historical closure implementation where it informs baseline conditions and prior agency expectations, and identifying recommencement alignment actions that should be carried into the updated mine plan and mine closure plan to remain aligned with prior approvals and commitments.

Where historical requirements describe performance objectives rather than prescriptive design details, the report emphasizes underlying intent so that recommencement designs can demonstrate equivalency. The report also supports feasibility and permitting by making historical governing commitments explicit and traceable, highlighting potential alignment sensitivities early, and providing a defensible linkage between the recommencement concept and the historic environmental and closure record.

Across the resource categories evaluated, the alignment actions collectively emphasize a limited set of recurring, cross-cutting themes that should be treated as core closure planning deliverables for recommencement, including: (1) defining stable post closure landforms and engineered features (and supporting analyses where applicable); (2) implementing sitewide drainage, erosion, and sediment controls through reclamation and into the post closure condition; (3) managing water and groundwater protection through design controls, monitoring networks, and defined response actions; (4) conserving and managing growth media and revegetation measures to support long-term stabilization and habitat recovery; and (5) maintaining public safety, access controls, and contingency planning appropriate to a remote mine setting during operations, closure earthworks, and the post closure period.

17.6.1 Closure and Reclamation Plan

Based on a preliminary regional groundwater flow model that included enlargement of the Colosseum pits and post-mining recovery of the groundwater system, a terminal-sink pit lake is anticipated to result during the post-closure phase, making active dewatering and treatment of pit water unnecessary following closure. All water inflow to the pit lake, including precipitation, storm-water runoff and groundwater, will leave the pit lake only via evaporation. No surface water or groundwater drainage from the pit lake is expected to occur.

An access restriction berm (also termed “bund”) will be constructed around the perimeter of the Colosseum pits to impede human access and reduce the inflow of surface water to the pit.

Waste rock dumps will be contoured to more stable slopes and integrated with surrounding landforms. They will be seeded to resemble the natural existing landform of the area.

The existing tailings storage facility will be covered by waste rock dumps in this phase of the development and therefore will be treated as a waste rock dump for rehabilitation purposes.

The Company does not intend to construct a new tailings storage facility for the Project. Tailings will be dewatered using a Jord filter press and then co-deposited with the waste rock material. To ensure no leakage and to increase the capture of surface run-off, all the waste dumps are planned to be lined with rubber liner.

Post-closure, the process plant and all associated infrastructure will be removed from site and sold for ongoing use or salvage value. The area of infrastructure will be graded, covered with topsoil and reseeded, as per what occurred with the original mine operations.

17.7 Closure Cost Estimate

The operating cost plan for the Colosseum mine includes provisions for progressive rehabilitation of the waste dump areas, with no additional budget post closure apart from monitoring.

It is estimated that the removal and rehabilitation cost of the above surface infrastructure will be $10 million. This is estimated to be fully offset by the sale and/ or salvage of the plant and equipment for $10 million.

17.8 Summary of Main Environmental Topics for the Project

Dateline intends to operate Colosseum under the existing Plan of Operations and valid Mining Rights. Dateline intends to defer to the requirements under the existing statutory approvals for the operation with regards to management of the environmental topics for the Project.

17.8.1 Topography

Mitigation measures described in the EIS focus on designing permanent landforms for long-term stability and reducing the visual and erosional effects of disturbance. Pit walls will be designed for stable slope configurations to minimize potential failures during operations and after closure. Roads will be regraded and bermed at closure to restore more natural drainage pathways and reduce erosion. Waste rock dumps will be contoured to more stable slopes and integrated with surrounding landforms. Slopes will be monitored throughout operations so that any emerging stability issues could be addressed promptly.

The EIS includes mitigation measures for the tailings storage facility, noting that it should be backfilled, graded and revegetated. These measures occurred post the original phase of mining. For the planned development, the existing tailings area will become a waste rock dump. Tailings will be dewatered using a Jord filter press and the resulting tailings cake will be co-deposited with waste rock material.

The incorporation of the belt filter into the processing circuit is expected to have the greatest impact on the environmental aspects of the project. By incorporating the filter press, less water will need to be drawn from the Colosseum #1 and #2 water bores, with less water extraction forecasted than for the original mine operation, despite an increase in plant throughput rate. The incorporation of the filter press also removes the need for a tailings storage facility completely, with benign tailings material to be co-deposited with the waste material.

17.8.2 Geology/ Seismology

The EIS evaluated potential geologic and seismologic impacts associated with regional seismicity, including earthquake ground shaking and related stability concerns, and identified soil contamination as a potential geologic related impact consideration. Overall, the EIS characterized seismic and liquefaction hazards as low, citing limited historical seismic activity, shallow poorly sorted soils, and limited groundwater.

Mitigation measures in the EIS incorporate engineering and operational safeguards is in full compliance with state and federal requirements.

17.8.3 Erosion/ Drainage

The EIS identified erosion and drainage impacts associated with disturbance of soils and vegetation during grading, construction of roads and waste facilities, potential uncontrolled releases from pipeline ruptures, and changes in runoff patterns around the tailings impoundment during operations and reclamation.

Mitigation measures employ engineering controls and reclamation strategies. Phased revegetation will be used to stabilize disturbed soils, with catchments and berms constructed to minimize erosion. Waste rock dumps, haul roads, and millsite areas would be graded for revegetation, with swales as alternatives to sediment basins.

17.8.4 Hydrology

The EIS evaluates hydrologic impacts associated with groundwater withdrawal and surface water management. Dewatering and process water demands were projected to reduce groundwater availability in Shadow and Ivanpah Valleys, with potential drawdown of local wells and corresponding effects on agricultural and domestic supplies in this arid region.

Mitigation measures emphasize monitoring and adaptive management. Project design optimized the use of reclaimed water to minimize the necessary withdrawal of groundwater resources from the Shadow and Ivanpah Valleys. Continuous monitoring of Shadow and Ivanpah Valley wells was proposed to track water level changes.

Dateline intends to employ a closed loop water management model for Colosseum. Waste rock dumps will have a rubber liner to capture surface water run-off was well as avoid leakage. The tailings material will be dewatered using a filter press, with the water to be reused in the processing circuit, reducing the requirements for drawdown from the Colosseum #1 and # bores.

17.8.5 Groundwater Quality

The EIS identifies potential groundwater quality impacts from releases of contaminants associated with mining activity. Untreated tailings were expected to contain low concentrations of heavy metals and cyanide compounds, creating a risk of leaching to underlying aquifers if containment systems failed.

To manage these risks, the EIS specifies a suite of design, operational, and monitoring measures. These included regular monitoring of the tailings impoundment and limiting wastewater discharges to designated disposal locations. Three monitoring bores were established to monitor groundwater quality.

The planned Project includes filtering of the tailing material and reuse of the water in the processing circuit, minimizing the potential for groundwater leakage. The waste rock dumps will be lined, further reducing the potential for groundwater contamination.

17.8.6 Air Quality

Air quality impacts from dust generation and vehicle emissions during the mining operation (approximately 46-50 one-way trips) were also considered. In the arid environment with limited dispersion, prevailing winds could carry dust to sensitive areas, affecting vegetation and habitats, while emissions may temporarily elevate nitrogen oxides and carbon monoxide.

Mitigation measures to control dust and emissions include graveling roads, watering them if necessary, or otherwise managing them to reduce dust. Working areas and storage pile surfaces will also be watered or otherwise controlled for dust.

Dateline has secured access to a 27-acre parcel of land close to the I-15 that it intends to use as a staging area during construction and then as a personnel parking area during operations. Personnel will be bussed from there to site, reducing the potential for dust emissions.

17.8.7 Vegetation

The potential for significant vegetation impacts was considered in the EIS, including removal of 240 acres of blackbush scrub in the impoundment area, displacement of small sensitive plant populations, and clearing of pinyon-juniper woodland along access roads. Two sensitive plant species, which would be disturbed by the activity at the mine area and along access roads, were specifically mentioned in the EIS:

- Striped horsebush (Tetradymia argyrea): as of September 2025, the California Native Plant Society (CNPS) lists this species with a CA Rare Plant Rank of 4.3 (Uncommon in California: Plants of limited distribution, a watch list, not very threatened in California).
- Pygmy agave (Agave utahensis var. nevadensis): as of September 2025, the California Native Plant Society (CNPS) lists this species with a CA Rare Plant Rank of 4.2 (Uncommon in California: Plants of limited distribution, a watch list, moderately threatened in California).

Another rare plant species, which existed along access roads, is specifically mentioned in the EIS:

- Clark Mountain buckwheat (Eriogonum heermannii var. floccosum): as of September 2025, the California Native Plant Society (CNPS) lists this species with a CA Rare Plant Rank of 4.3 (Uncommon in California: Plants of limited distribution, a watch list, not very threatened in California)

Mitigation measures prioritized reclamation and restoration. Salvaged topsoil will be stockpiled for revegetation with native species, including transplants from the impoundment area and nitrogen fixers for soil stability (i.e., lotus rigidus, L. scoparius, Acacia greggii, and Astragalus minthorniae). The final seed mixture to be utilized will be determined by regulatory authorities at the time of reclamation.

17.8.8 Wildlife

The EIS evaluated wildlife impacts associated with habitat loss, disturbance, and exposure to project facilities. Identified concerns include removal of habitat and water sources, disturbance of bighorn sheep movements, potential effects on species such as Gila monster and desert tortoise, and the risk that wildlife could use contaminated water (e.g., tailings effluent) as a drinking source.

Mitigation measures focus on limiting the affected footprint and reducing wildlife exposure to hazards. Concentrating project facilities within a relatively compact mine and mill complex is intended to minimize the overall area of habitat loss. Fencing around the detoxified tailings pond and other hazardous facilities will restrict wildlife access to process solutions. Reduced vehicle speeds on access roads will be enforced to limit wildlife-vehicle collisions, and reclamation measures would be implemented to encourage habitat recolonization over time.

The risk of wildlife using tailings effluent as a drinking source is minimised through the use of the filtered tailings system, with no tailings impoundment facility to be developed. Water captured from the filter process will be stored within a fenced and lined pond for reuse in the process circuit.

17.8.9 Socioeconomics

The 1985 EIS notes socioeconomic concerns primarily related to the isolation of the site and the resulting demands on emergency response capabilities. The remote location could place additional burdens on local emergency services responding to incidents associated with the project.

Mitigation measures include maintaining equipment and trained personnel on site, implementing a site-specific contingency plan for emergency response, and seeking to establish mutual assistance agreements with nearby developments, such as Mountain Pass.

17.8.10 Access/ Transportation

The EIS identifies access and transportation impacts associated with constructing and improving access roads. These activities would remove vegetation and wildlife habitat, modify landforms along cut and fill sections, and introduce project traffic to previously less travelled routes, although overall traffic volumes were expected to remain modest.

Mitigation measures include bussing employees to and from the site. Dateline intends to use the existing mine access road. Post mining, the road would be regraded and bermed to discourage access post reclamation with revegetation efforts initiated. Lower speeds will be enforced on low visibility road segments.

17.8.11 Cultural Resources

The EIS considers impacts on archaeological and historical resources, including Native American sites and historic mining features, that could be disturbed or destroyed by excavation, road construction, and facility development in the pit, tailings, and infrastructure areas. The EIS also recognizes the potential to uncover previously undocumented sites during construction.

Mitigation is based on avoidance as the preferred strategy. Where multiple route or facility alternatives exist, the EIS indicates that options producing the least disturbance to identified cultural sites should be selected, and that alignments may be locally redesigned to avoid sensitive resources. Where avoidance is not feasible, the EIS calls for implementation of a data recovery program, providing for systematic excavation, documentation, and curation of affected resources prior to disturbance.

Three sites were identified in the original mine development, covering the same area as the planned development. The data from the sites was recovered in alignment with this program.

17.8.12 Land Use: Grazing Land

The EIS estimates that proposed mine development would remove approximately 600 acres of grazing land, equivalent to roughly 15 - 17 animal unit months (AUMs) historically used by local ranchers. Loss of forage and modifications to water availability associated with mine pits, the tailings impoundment, and related infrastructure could adversely affect livestock operations.

Mitigation measures include relocating grazing use to alternative lands and providing replacement water sources. The original mine developers committed to supplying water for grazing to offset loss of the existing pool at Green's Well and to installing additional wildlife guzzlers and a permanent alternative drinking water source for bighorn sheep, sited away from mining affected areas and locations heavily used by cattle.

The EIS notes that residual impacts would consist primarily of a long-term reduction in grazing land during the operational phase, though these would be mitigated through relocation and water furnishing, with effects expected to diminish post reclamation.

17.8.13 *Visual Resources*

The EIS recognizes that the open pits, tailings facility, waste rock piles, and support infrastructure would substantially alter the visual character of the area, introducing strong geometric forms and colour contrasts that would be visible from portions of the surrounding public lands and roadways.

To reduce visual impacts during operations, the mine plan includes the following measures during active mining:

- colour buildings and other infrastructure to blend with the surrounding landscape
- stockpile salvageable topsoil for later use in reclamation
- where practicable, placing water pipelines within or adjacent to existing roadbeds and trails to avoid creating new linear surface contrasts

Post mining measures include:

- use of seed mixes composed of local species
- shape reclaimed slopes and escarpments to conform to natural landforms
- remove all buildings at the time of abandonment
- treat or partially remove concrete structures so they appear unobtrusive
- cover the tailings impoundment with waste rock and/or salvaged topsoil (if available) followed by reseeding
- dismantle and remove the water pipeline once it is no longer needed for reclamation or other uses

17.8.14 *Recreation*

The EIS evaluates potential effects on recreation, particularly on adjacent public lands. Loss of undeveloped land and changes to the natural setting from exploration, mining, and access road construction were expected to reduce opportunities for nature study and resource-oriented activities in localized areas, especially along the eastern flank of Clark Mountain. Overall recreational use was expected to remain limited due to the remote location.

To mitigate recreational impacts and address public safety, the Company will install highly visible warning signs to keep the public away from hazardous areas near the mining facilities during operations and after reclamation. Fencing around the open pits is planned, both to prevent inadvertent public access and to enhance safety following closure. In addition, long term interpretive programs are planned to explain the reclamation plan and educate the public regarding mine development and closure.

17.8.15 *Wilderness*

The EIS evaluated impacts on Wilderness Study Areas (WSAs) associated with the proposed Colosseum Mine project, focusing on access roads and power line routes. The EIS investigated the various environmental impacts based on several alternative access routes to site.

Dateline intends to utilize the existing access road that was established during the original mining operation, minimizing disturbance from a new access road. Dateline intends to route the water piping from the Colosseum #1 and #2 bores along the access road corridor. The planned overhead powerline will also use the same access road corridor.

The proposed water and powerline routing along the access road is intended to minimize both cost and environmental impacts by using only one corridor.

17.8.16 *Public Health and Safety*

The EIS addresses public health and safety concerns related primarily to dust and noise, fire hazards, and potential exposure to lead and cyanide.

Dust will be generated by vehicular traffic on unpaved access roads and by material handling, but modelled ambient concentrations are expected to remain within applicable National Ambient Air Quality Standards (NAAQS). Traffic related air quality impacts are considered minimal along the existing access road.

Noise from mine and mill operations would attenuate with distance and was predicted to be within acceptable levels at the nearest residences (e.g., approximately 30–40 dB at a distance of about seven miles).

Fire risk was assessed as low due to sparse vegetation and the implementation of protective measures such as brush removal around project facilities.

Public concern regarding potential toxic effects from lead inhalation, atmospheric deposition of lead, and hydrogen cyanide (HCN) releases was acknowledged, although lead and cyanide management will be subject to OSHA and MSHA standards. Through the implementation of a filtered tailings process in the circuit and lined water reuse ponds, the planned project reduces the potential for lead and/ or cyanide deposition.

18. CAPITAL AND OPERATING COSTS

18.1 Capital Cost

18.1.1 Mining

18.1.1.1 General

Mining capital costs for the Colosseum definitive feasibility study are developed to meet the recommendations of the AACE International document 47R-11, class 3 as described in the AusIMM Cost Estimation Handbook, Second Edition Monograph 27. The estimate has an expected accuracy range of ±10 to ±15%. The mine plan is based on Proved and Probable Ore Reserves with geotechnical, hydrogeological and environmental inputs defined, the mine design defined and optimised and the production schedule defined and optimised against the mining fleet.

The mine plan is at 60% to 70% maturity over the life of the project. This is adequate to estimate material movement quantities, mining fleet and workforce, mining consumables and mine related facilities to support the mine capital cost estimate at class 3 accuracy.

The mining capital cost estimate is based on operation of the mine by a mining contractor. Quotes were called from multiple US based mining contractors on a preliminary mine plan in July 2025. Capital costs are estimated either directly from the mining contractor quotations or using equipment and supplies databases to generate estimates which are calibrated against the contractor quotes.

The estimate is inclusive of contingency. It covers both initial capital costs up to first ore through the mill and ongoing costs over the project life and mine closure.

Several work areas not directly involving mining but using the mining fleet and facilities are included in the mining capital costs estimate. The full scope of the mining capital cost estimate includes:

- Pre-production Mining
- Contractor Mobilisation / Demobilisation
- Haul Roads
- Topsoil Management
- Site Drainage
- Waste and Stockpile Underliners
- Mine Closure Earthworks

Mine capital costs are separated into initial costs up until first ore feed to the process plant and deferred over the rest of the project life.

Area		Initial	Deferred	Total
Pre-production Mining	USDM	$16.28	$0.00	$16.28
Contractor Mobilisation / Demobilisation	USDM	$3.45	$0.00	$3.45
Haul Roads	USDM	$0.07	$0.00	$0.07
Topsoil Management	USDM	$1.30	$0.71	$2.01
Site Drainage	USDM	$3.66	$0.52	$4.18
Waste and Stockpile Underliners	USDM	$11.97	$17.47	$29.44
Mine Closure Earthworks	USDM	$0.20	$3.08	$3.28
Total Mine Capital Cost	**USDM**	**$36.93**	**$21.78**	**$58.71**

Table 62 Mining Capital Cost Estimate Summary

18.1.1.2 Estimate Currency and Base Date

The estimate is expressed in US dollars based. All costs are derived from US dollar sources.

The base date for the estimate is first quarter 2026.

18.1.1.3 Pre-production Mining

Ore and waste mining commence six months prior to first ore through the process plant. This advance work to build initial run of mine (ROM) stockpiles and develop adequate bench areas in the pits is to ensure that once production commences the mine can continuously deliver ore to the process plant at target tonnes and grade.

18.1.1.4 Contractor Mobilisation / Demobilisation

Mining contractor mobilisation includes:

- Costs quoted by US contractors to establish facilities on site including workshops, offices, explosives magazine and fuel storage. Dateline supplies services (mainly power and fire water) up to the edge of the contractor's area. Mobilisation costs also include recruitment and onboarding of the contractor's workforce.
- Transport and commissioning costs for each item of mobile mining equipment delivered to or removed from the site.

18.1.1.5 Topsoil Management

Topsoil management includes:

- Clearing and grubbing of vegetation,
- Topsoil harvesting to topsoil stockpiles, and
- Reclamation of harvested topsoil to be spread over areas where mining activities have been completed.

Topsoil management is planned on new pit areas outside the current pit crests, the entire waste rock dump area which includes the base of the dry stacked tailings cell and the base of long-term low-grade ROM stockpile.

Clearing, grubbing and topsoil harvesting is timed to be just in advance of mining to minimise surface disturbance.

Topsoil harvesting and spreading is conducted progressively over the project life in areas where mining activities are completed.

18.1.1.6 Site Drainage

The entire site is planned as a closed loop for rainfall so that no rainfall on areas affected by mining can leave the site. Key elements of this system include a cut-off drain south of the waste rock dump to divert run off west into the Shadow Valley before it comes into contact with mining activities and three retention ponds to collect rainfall run off from mining areas before it can leave the site. Capital costs for site drainage include excavation cut to fill, placement and compaction of fill material and lining with impermeable HDPE liners.

18.1.1.7 Waste Rock and Stockpile Underliners

The closed loop drainage system includes impermeable HDPE liners under the entire waste rock dump area, which includes the base of the dry stacked tailings cell and the base of long-term low-grade ROM stockpile. This removes the possibility of any water that may percolate through the waste rock, tailings or stockpiled ore from coming into contact with the underlying granite / gneiss bedrock. Capital costs are based on 80 mil (2mm) HDPE liner placed, secured and welded. All areas to be lined are also included in the topsoil management activity.

Installation of the underliners is timed to match disturbance by mining activities and to ensure than any water intercepted is directed to the retention ponds.

18.1.1.8 Mine Closure Earthworks

Mine closure earthworks cover the same activities carried out when the operation was paused in 1993 and 1994. It includes ripping of all compacted areas and grading and shaping to manage surface drainage, minimise erosion and promote revegetation.

Mining rehabilitation is planned progressively over the project life as part of waste rock dump final face formation and topsoil management. Some of the activities, such as revegetation, are included as operating costs. The mine closure earthworks capital cost estimate covers the areas remaining at completion of mining and processing to prepare the site for final closure. It is planned in two phases:

- At completion of mining to cover areas of the mine and waste rock dump not already rehabilitated during operations,
- At completion of processing of the low grades stockpiles when waste rock is rehandled to cap the final area of dry stack tailings.

Costs include machine "wet" hire costs and owner's costs for planning and supervision of the work.

18.1.2 Process

18.1.2.1 General

The capital cost estimate for the Colosseum definitive feasibility study is developed to meet the recommendations of the AACE International document 47R-11, class 3. The estimate has an expected accuracy in the range -10% to -20% on the low side and +10% to +30% on the high side and is considered suitable for funding authorisation. The class three methodology includes "Semi-detailed unit costs with assembly level line items". The maturity level of engineering deliverables is stipulated as being in the range of 10% - 40%

The above requirements have been met by the methodology employed in the development of the Colosseum definitive feasibility study capital estimate.

The Project Capital Cost Estimate developed for the Study is based on an approach assuming engineering and procurement is performed under a fixed price agreement by an engineering organisation whilst construction is performed by Dateline.

The estimate is inclusive of contractor's contingency but excludes future costs and owner's costs.

The estimate includes all others costs associated with process engineering, design engineering, drafting, procurement, construction and commissioning of the process facility and associated infrastructure, first fills of plant reagents and consumables, and commissioning spares.

The estimate is based upon preliminary engineering and quantity take-offs from the design, and budget price quotations for bulk commodities and major equipment.

The estimate pricing was obtained predominantly during the fourth quarter of 2025. Where possible pricing is received quoted in USD. Where pricing is received in a foreign currency, it is converted to USD at the foreign exchange rates set at 1Q26.

The total capital cost for the Colosseum processing facility is estimated to be USD $208.6M. This estimate includes contingency and engineering and procurement costs. Additionally, infrastructure capital costs are estimated to be USD57.6M. Infrastructure capital costs include the following:

- Raw water supply pipeline

- Initial fills (reagents, lubricants)
- Insurance spares
- Communication equipment
- Power line and metering infrastructure
- Import Tariffs

The majority of the infrastructure costs are ascribed to the powerline and metering infrastructure (USD 10.4M). This cost includes the overhead transmission line from the existing infrastructure, transformers and switchgear. Contingency is generally not applied to the power station cost (only to transformer supply costs). Contingency is applied to the pipeline and initial fills costs.

Table 61 shows the capital cost estimate summary for the Colosseum processing facility.

Description	Total Installed Cost (USDM)
Direct Costs	
Earthworks	-
Civil Works	$16.75
Mechanical Equipment	$48.86
Platework	$4.59
Tankage	$5.62
Structural Steel	$12.44
Electrical Installations	$51.45
Piping	$9.94
Buildings	$4.47
Construction Equipment	$15.43
Sub-total: Direct Costs	**$169.54**
Indirect Costs	
Temporary Construction Facilities	$5.25
Supervision and Construction management	$8.50
Project and Procurement Management	$6.99
Engineering Design	$14.17
Vendor Commissioning	$3.02
Commissioning	$1.46
EPC Indirect Costs	$1.67
Sub-total: Indirect Costs	**$41.04**
Infrastructure Costs	
Pipeline	$5.68
Camp	-
Initial Fill	$1.50

Insurance Spares	$3.49
Communications Equipment	$0.36
Power Station	$10.40
Tariffs	$3.61
Sub-total: Infrastructure Costs	**$25.05**
Grand Total	**$235.63**

Table 63 Processing Capital Cost Estimate Summary

18.1.2.2 Estimate Currency and Base Date

The estimate is expressed in U.S dollars based on the exchange rates listed below:

- 1.00 A$ = 0.65 US$

The base date for the estimate is first quarter 2026.

18.1.2.3 Accommodation and Messing

A construction camp / village will not be built to service the construction of the Colosseum processing plant. The capital estimate assumes all construction personnel are housed locally (Primm). Accommodation and messing costs are therefore not carried by the capital estimate. However, subsistence (per diem) payments to construction personnel are allowed for in the capital estimate. It is expected that the construction crew will be housed at the Primm Valley Casino Resort.

18.1.2.4 Bulk Earthworks

Bulk earthworks are carried out as part of pre-works. The capital cost estimate therefore does not include costs for bulk earthworks. Likewise, the upgrade to the site access road is carried out as part of pre-works. Costs for the access road upgrade are therefore not included in the capital estimate.

18.1.2.5 Concrete

Concrete quantities are developed by performing material take-offs from the plant 3D model. Several local concrete contractors were approached to provide all-inclusive budget rates for supply and placement of various types of concrete e.g. ground slabs, footings, suspended slabs etc. The concrete rates do not allow for the establishment and operation of a site batch plant. Concrete is batched in Primm and delivered to site by transit mixer.

The duration to complete concrete works was determined by the quantities and final estimated man-hours to complete the scope. Rates are inclusive of equipment, labour, fuel, consumables, materials, and indirect costs.

18.1.2.6 *Structural Steel*

Structural steel quantities (including floor grating, handrailing and stairs) are developed by performing material take-offs from the 3D model. Enquiries for the supply and fabrication of structural steel were placed with several contractors.

The rates used in the capital estimate are arrived at after consideration of the offers received. The structural steel is thereby estimated on a quantity by rate basis.

18.1.2.7 *Process Equipment*

The process design criteria are used to develop a mechanical equipment list, which defines the requirements and sizes of all mechanical equipment, platework and tankage items. Specifications and data sheets are developed for all major equipment for presentation to equipment vendors.

Written budget quotations, accompanied by engineering specifications and data sheets or equivalent, are requested from recognised suppliers of major equipment items in the plant.

The inquiries issued to vendors request budget quotations to an accuracy of +/- 10%. Generally U.S based vendors are selected for the supply of major process equipment items. Where equipment cannot be sourced from a U.S vendor then overseas vendors are selected.

Minor items of process equipment are priced based on historical prices received for recent similar projects.

18.1.2.8 *Equipment Refurbishment*

All equipment and facilities are assumed to be supplied new, excepting the grinding mills, whereby second hand (not previously installed or used) mills have are included. The two mills (SAG and Ball) were procured new in 2021 for a project which was subsequently cancelled. The two mills have been in storage (generally undercover) since this time. The two mills are entirely suitable for the duty required for the Colosseum project. Two new main drive motors will be purchased to drive the mills. Allowances are included to convert the mill ancillaries (e.g. lube systems) from 50 Hz power to 60 Hz power.

The existing (from the original plant) reclaim chamber and associated tunnel are assumed to be reused, having new reclaim feeders and steelwork installed.

18.1.2.9 *Platework*

The process design criteria are used to develop the fabricated plate work component list that defines the requirements and sizes of all tanks, bins, chutes and launders. Platework quantities are established by performing material take-offs of typical like items (e.g. conveyor head chutes) or by calculation of quantities for items specific to the project e.g. tankage. Platework cost estimates are thereby developed based on a quantity by rates approach.

18.1.2.10 Piping

Piping costs are derived from major pipeline and valve take-offs, taken from the preliminary piping and instrument diagrams. Rates for piping materials and fittings are applied to major process piping elements. Minor piping costs are estimated based on actual costs experienced on similar projects.

Costs for the raw water supply pipeline are established considering the quantities of pipe and fittings required, to which unit cost rates are applied. The provision of installation equipment and labour are included.

18.1.2.11 Electrical and Instrumentation

Electrical, instrumentation and control quantities are compiled from single line diagrams, P&ID's (piping and instrument diagrams), layouts, equipment lists (mechanical and electrical) and electrical load list.

The capital estimate for electrics is developed using GRES estimating software. This software uses a library of components and installation labour hours and rates to arrive at costs. The library is regularly updated based on market changes.

Budget pricing for major electrical equipment items (transformers, switchboards and switch rooms, variable speed drives etc.) are sought from vendors. Likewise budget pricing for electrical "bulks" such as cable, cable ladder and the like are based on estimated quantities applied to quoted unit rates.

The estimate for the process control system is based on pricing to provide all required hardware and software to control the processing facility. A PLC (programable logic controller) / SCADA (supervisory control and data acquisition) based system is allowed for.

18.1.2.12 Power Station

Access to power will be from the local Southern California Edison grid (34.5 kV) The power access is located near to the SoCal Edison Ivanpah substation. Power is transmitted to the project site (at 34.5 kV) via an overhead transmission line which follows the alignment of the main access road (approx. 10km). Power is reticulated on site at 13.8 kV. The system includes the utility metering at the process plant, HV switchgear and the following reticulation of power to the process plant and the infrastructure.

18.1.2.13 Installation Labour

Estimates for installation labour are based on estimated man-hours associated with each discipline; civil, mechanical, electrical and piping.

The estimated hours for installation reflect the labour force productivity for U.S. minerals industry construction sites and the application of industry standard labour rates for the type of work involved. The rates are developed with due consideration to the rates achieved on recent similar minerals processing projects. Labour crew rates are built up to include an appropriate mix of supervision, skilled and unskilled personnel. Each crew rate includes the costs of mandatory meetings and breaks, small tools, statutory labour costs, personal protective equipment, clothing and supervision.

18.1.2.14 Productivity Factors

A basic productivity factor is applied to installation man hours to cover time required for pre-start meetings, toolbox talks and the like.

The construction labour rates for structural, mechanical and piping (SMP) and electrical, instrumentation and control (E&IC) are based on rates provided by local contractors.

18.1.2.15 Cranage and Equipment Costs

Estimates for cranes and equipment costs are based on the estimated hours of utilisation for major cranes and equipment items associated with installation in each area of the plant. The duration required for each type of crane, forklift, truck etc. are established from the execution schedule developed for the project.

18.1.2.16 Communications

The provision of communication and security systems is allowed for in the capital estimate. Starlink connections are to be used for internet communications.

18.1.2.17 Project Spares

Commissioning and capital insurance spares are included in the capital estimate. These costs are developed based on a combination of vendor quoted costs and factors applied to capital costs of the associated items. Costs associated with wearing components such as slurry pump impellers and volutes, and mill liners are not included in the capital spares list. These items are assumed to be operating costs and are therefore carried in the operating cost estimate.

18.1.2.18 Initial Fills

First fills of reagent, lubricants, grinding media and carbon are included in the capital estimate. Pricing for reagents and grinding media are provided by vendors. These rates are applied to estimated quantities required for plant start-up and initial operating hours (300 hours) only. Costs associated with refills following initial start-up and operations are carried by the operations cost estimate.

18.1.2.19 Transport

Process equipment inquiries request pricing to CIP (Inco terms) Los Angeles port. CIP (Carriage and Insurance Paid) covers the delivery and insurance of the item from the place of manufacture to the nominated port, paid by the seller. Delivery from the port of delivery to the destination (Colosseum) is covered by the buyer. It should be noted that payment of tariffs and import duties are paid by the receiver, not by the sender. Equipment sourced ex USA is generally quoted EXW (ex-works). Some tenders were received quoting FCA, generally out of Asian ports. In this case the vendor is approached to provide pricing for delivery CIP Los Angeles. Costs for delivery of U.S. sourced equipment are allowed to transport the item from the manufacturer to the Colosseum site.

Budget quotes for logistics are sought for delivery of equipment from Los Angeles port to site.

Packing charges are included to transport all imported materials and equipment from place of manufacture to Colosseum site. These costs are included in the supply cost of the item.

18.1.2.20 Indirect On-Site Costs

The capital cost estimate assumes that on-site construction facilities including the following will be provided by the construction contractors:

- Offices.
- Stores.
- Workshops.
- Communications.
- Ablution facilities.
- Crib facilities.

The capital estimate is based on construction labour being housed locally at the Primm Valley Casino Resort.

18.1.2.21 Inclement Weather

The capital cost estimate does not include allowance for inclement weather. Rainfall at the project site is extremely low and snow unusual.

18.1.2.22 Contingency

Contingency is allowed in the capital costs estimate, the overall rate assumed being approximately 5.4%. Contingency is applied at varying rates to each line item of the estimate. The rate of contingency applied is a function of the level of maturity of the engineering performed.

It should be noted that contingency is not applied to the power station and associated items (gas offtake and metering station, OHT line) as these costs are allowances only.

18.1.2.23 Engineering Procurement and Construction Management

Costs for engineering (design), procurement and construction management are allowed in the capital cost estimate. These costs are estimated based manhours required for the delivery of services. Cost rates for resources are applied to the estimated manhours.

18.1.3 Infrastructure and Facilities

Non process infrastructure (NPI) is allowed for in the capital estimate. This includes buildings such as the administration building, workshop and warehouse building etc. Roads internal to the processing facility are included.

The main access road from the I15 highway to the site is upgraded to provide all weather access for light vehicles. Costs for this upgrade are considered as pre-work and are not included in the capital estimate.

A raw water supply pipeline to convey water from the wells located near to the Ivanpah substation to the site raw water tank is included in the capital estimate. A number of upgrades to existing wells and equipment are allowed for. Likewise new electrical infrastructure to service the wells and lift pumps is allowed for.

18.1.4 Tailings Storage Facilities

A dry stacked tailings system is proposed for the Colosseum project. A wet tailings dam will not be used. Filtered tailings will be delivered (by truck) to an emplacement located on the site of the original tailings dam. No capital costs are allowed (in the process plant capital cost estimate) for the emplacement facility. The tailings truck fleet and the associated earthmoving equipment (frontend loader, bulldozer) are assumed to be provided under the mining contract and are therefore not included in the process plant capital cost estimate.

18.1.5 Reclamation and Closure

Mine and waste dump reclamation costs are expensed and included in the mine operating cost budget.

For the closure and rehabilitation of the site, the removal of infrastructure and restoration of the process plant area is estimated at USD10 million. This is offset by a corresponding amount of USD10 million as it is assumed that the plant will be able to be sold for salvage value.

18.2 Operating Cost

18.2.1 Mining

Dateline called for contract mining quotations from US based contractors in mid-2025. The request for quotation (RFQ) and proposals received were based on a detailed monthly mine plan covering the life of the project.

Since mid-2025 the mine plan has been adjusted to match updates to the resource model, significantly increased gold prices and further definition of the waste rock, tailings and drainage management methods. Key elements of the mine plan such as haul distances and total material movements remain very similar to the plan used for the contractor proposals. However, to ensure mine operating costs are aligned to the final mine plan a first principles mining cost model was developed against the 2025 RFQ plan and calibrated to match the contractor proposals. This helps to ensure that as the specifics and timing of the mine plan evolve the mine operating costs remain representative of the contract mining proposals received.

Mine operating costs are estimated for activities related directly to mining of ore and waste rock and to activities which involve the mining fleet, but which are not directly related to mining:

- Opencut Mining - Ore
- Opencut Mining - Waste
- Stockpile Reclaim - Mine Fleet
- Tailings Haul and Place - Mine Fleet
- Crusher Feed - Mine Fleet

Project Year		1	2	3	4	5	6	7	8	9	10	11	Total
Total Costs													
Mining													
Mining Ore	USDM	17.43	20.40	20.53	16.61	16.12	4.83	0.00	0.00	0.00	0.00	0.00	95.92
Mining Waste	USDM	61.96	59.86	54.08	45.25	26.33	2.22	0.24	0.24	0.24	0.24	0.10	250.78
Total Mining	**USDM**	**79.39**	**80.26**	**74.62**	**61.86**	**42.45**	**7.05**	**0.24**	**0.24**	**0.24**	**0.24**	**0.10**	**346.70**
Stockpile Reclaim	USDM	0.24	0.00	0.00	1.36	1.31	3.88	5.84	5.72	5.58	5.41	1.99	31.33
Tailings Haul and Place	USDM	9.15	10.20	10.27	9.73	9.90	5.69	4.32	4.38	4.46	4.33	1.81	74.24
Crusher Feed	USDM	1.80	1.94	1.94	1.39	1.40	0.61	0.18	0.18	0.18	0.18	0.08	9.88
Unit Costs													
Mining													
Mining Ore	USD/t	4.31	4.41	4.62	4.81	5.46	5.21	0.00	0.00	0.00	0.00	0.00	4.69
Mining Waste	USD/t	4.20	4.11	4.16	4.17	5.19	7.48	0.00	0.00	0.00	0.00	0.00	4.28
Total Mining	**USD/t**	**4.22**	**4.18**	**4.27**	**4.33**	**5.29**	**5.76**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**4.39**
Stockpile Reclaim	USD/t	2.82	0.00	0.00	4.02	4.49	3.03	2.92	2.86	2.79	2.70	2.69	2.92
Tailings Haul and Place	USD/t	4.87	5.09	5.13	4.86	4.95	2.84	2.16	2.19	2.23	2.16	2.46	3.60
Crusher Feed	USD/t	0.96	0.97	0.97	0.70	0.70	0.31	0.09	0.09	0.09	0.09	0.11	0.48

Table 64 Mine Operating Cost by Year

18.2.1.1 Mine Operating Time Assumptions

The mine fleet is assumed to run on 2 x 12 hours shifts per day scheduled for 355 days per year. Ten days per year are assumed lost due to weather and other unplanned stoppages.

Shift delays are applied for items including shift changes and meal breaks to give 9.5 productive hours per 12-hour shift.

Mechanical availabilities and utilisation factors are applied for each machine type to derive potential productive hours per period.

18.2.1.2 *First Principles Cost Model*

The first principles operating cost model includes all the areas estimated and is built up as follows:

- Productivities for the key fleet items such as loading unit / truck matches or blast hole drills are estimated. The production schedule is driven by these productivities so that it aligns with efficient fleet usage.
- Haulage models are generated for the ore, waste rock, tailings and stockpile movements over the mine life. These models lead to truck hours per period.
- Loading unit, truck and drill hours per period lead to fleet sizes over the mine life.
- Support equipment (bulldozers, graders, water carts, lighting plants, pumps, etc) are added to match the primary fleet over the mine life.
- Hourly operating costs for repair and preventative maintenance parts, fuel, lubricants, electricity, tyres and ground engaging tools are applied to the machine hours per period.
- The operating and maintenance workforce needed to run the fleet is added and labour costs are estimated against all-inclusive annual costs from a database on mining wages in southwestern USA.
- Monthly equipment lease costs are added to simulate the contractor's ownership costs.
- Explosives usage is estimated against assumed blast patterns and explosives supply costs are assigned using a combination of the mining contractor quotes and historical monthly service charges.
- Employee numbers for mining management technical services and supervision are estimated, and annual all-inclusive salaries are applied.
- Other costs such as grade control, training, software, progressive rehabilitation and environmental monitoring are added.
- Costs for items supplied by the mining contractor are identified and a contractor margin is applied.

Wherever possible costs are assigned directly against activities. Examples include loading, haulage, grade control and drill and blast. Where costs are shared across activities they are apportioned on the most representative scheduled parameter. For example, grader and water cart costs are apportioned based on modelled truck hours for each activity.

18.2.1.3 *Mine Operating Costs*

Ore and waste mining costs include modelled loading unit and truck hours, grade control, drill and blast and a portion of shared costs.

18.2.1.4 *Stockpile Reclaim Operating Costs*

Stockpile reclaim operating costs are for loading and truck haulage of low grade and marginal ore to the crusher. High grade and other ore rehandled by front end loader from stockpiles adjacent to the crusher is covered under crusher feed operating costs. A portion of shared costs is also assigned to stockpile reclaim. Trucks used for stockpile reclaim are shared with the mine and tailings activities.

18.2.1.5 Tailings Haul and Place Operating Costs

Tailings haul and place operating costs include a front-end loader loading dewatered tailings into mine trucks which haul them to the dry stack tailings cell where they are spread by bulldozer. A portion of shared costs is also assigned to tailings haul and spread.

18.2.1.6 Crusher Feed Operating Costs

Crusher feed operating costs relate mainly to a front-end loader permanently stationed in the crusher area. Most ore is planned to be directly dumped by truck into the crusher hopper but a minimum of 20% of the feed is expected to be rehandled from short term stockpiles adjacent to the crusher. The loader is responsible for this rehandle and for keeping the area around the crusher hopper free from loose rocks and build-up of ore at the dump point.

18.2.2 Process

Operating costs have been developed using the parameters specified in the process design criteria. Annual operating costs and costs per tonne milled (processed) have been developed and are summarized in Table 63. Operating costs have been estimated to an accuracy of +/- 15%. Table 64 show figures in US dollars using AUD: USD rate of 0.65 for operating costs estimation.

The costs cover the processing of ore from the ROM pad battery limit. This includes the sections covering crushing, reclaim, milling, leaching and adsorption circuit, dewatering, cyanide detoxification, process plant site services (power, air, and water), and administration costs.

The operating cost estimate has been developed based on a process plant feed tonnage of 2,000,000 tpa.

The costs have been compiled from a variety of sources including:

- First principal estimates.
- Suppliers' budget quotations.
- GRES data base for similar operations.
- Metallurgical test work results.

The following figures summarise the split in percentages of the process plant operating costs.

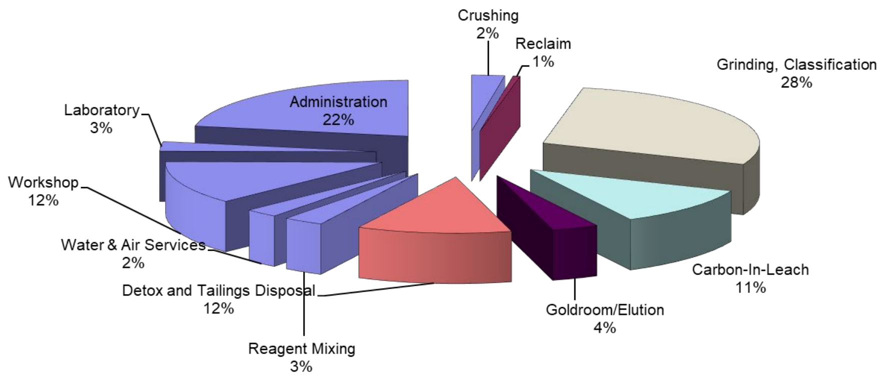


Figure 59 OPEX Item by plant area

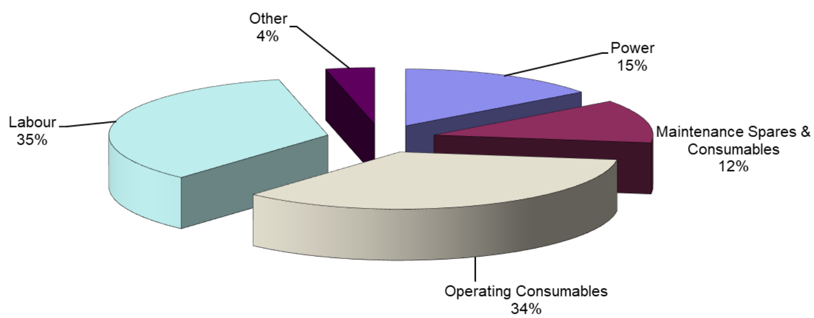


Figure 60 OPEX item by type

BY COST CENTRE	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Total
Power (USDM)	$9.104	$9.104	$9.104	$9.104	$9.104	$9.104	$9.104	$9.104	$9.104	$9.104	$3.793	**$94.834**
Maintenance Spares & Consumables (USDM)	$5.737	$5.737	$5.737	$5.737	$5.737	$5.737	$5.737	$5.737	$5.737	$5.737	$2.390	**$59.7556**
Operating Consumables (USDM)	$15.313	$16.095	$16.061	$16.061	$16.061	$16.065	$16.061	$16.061	$16.061	$16.065	$4.097	**$164.062**
Labor (USDM)	$15.282	$15.282	$15.282	$15.282	$15.282	$15.282	$15.282	$15.282	$15.282	$15.282	$6.368	**$159.190**
Other (USDM)	$3.466	$3.210	$3.220	$2.845	$2.845	$2.845	$2.845	$2.845	$2.845	$2.845	$2.060	**$31.872**
USDM Total	**$48.901**	**$49.428**	**$49.405**	**$49.030**	**$49.030**	**$49.030**	**$49.030**	**$49.030**	**$49.030**	**$49.063**	**$18.708**	**$509.714**
Power (USD/t)	$4.85	$4.54	$4.55	$4.55	$4.55	$4.54	$4.55	$4.55	$4.55	$4.54	$5.14	**$4.60**
Maintenance Spares & Consumables (USD/t)	$3.06	$2.86	$2.87	$2.87	$2.87	$2.86	$2.87	$2.87	$2.87	$2.86	$3.24	**$2.90**
Operating Consumables (USD/t)	$8.16	$8.03	$8.03	$8.03	$8.03	$8.03	$8.03	$8.03	$8.03	$8.03	$5.55	**$7.95**
Labor (USD/t)	$8.14	$7.62	$7.64	$7.64	$7.64	$7.62	$7.64	$7.64	$7.64	$7.62	$8.63	**$7.72**
Other (USD/t)	$1.85	$1.60	$1.61	$1.42	$1.42	$1.42	$1.42	$1.42	$1.42	$1.42	$2.79	**$1.54**
USD/t processed Total	**$26.06**	**$24.65**	**$24.70**	**$24.51**	**$24.51**	**$24.46**	**$24.51**	**$24.51**	**$24.51**	**$24.46**	**$25.36**	**$24.71**

Table 65 Process Plant Operating Costs Yearly Breakdown

By Area USD/t	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Total
Crushing	0.62	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.69	**0.59**
Reclaim	0.14	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.15	**0.13**
Grinding and Classification	7.55	7.24	7.25	7.25	7.25	7.24	7.25	7.25	7.25	7.24	6.84	**7.26**
Carbon In Leach	2.69	2.64	2.64	2.64	2.64	2.64	2.64	2.64	2.64	2.64	1.65	**2.61**
Goldroom/Elution	0.85	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.81	0.82	**0.82**
Detox and Tailings	3.05	2.92	2.93	2.93	2.93	2.92	2.93	2.93	2.93	2.92	2.63	**2.93**
Reagent Mixing	0.74	0.69	0.70	0.70	0.70	0.69	0.70	0.70	0.70	0.69	0.79	**0.70**
Water & Air Services	0.69	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.73	**0.65**
Workshop	3.23	3.02	3.03	3.03	3.03	3.02	3.03	3.03	3.03	3.02	3.92	**3.08**
Laboratory	0.69	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	1.10	**0.66**
Administration	5.81	5.31	5.33	5.14	5.14	5.13	5.14	5.14	5.14	5.13	6.05	**5.27**
USD/t Total	**26.06**	**24.65**	**24.70**	**24.51**	**24.51**	**24.46**	**24.51**	**24.51**	**24.51**	**24.46**	**25.36**	**24.71**

Table 66 Process plant area operating costs per tonne of ore

18.2.2.1 *Qualifications and Exclusions*

The operating costs presented are calculated from first principles and budget quotations for supply of chemicals, materials and services. Process plant operating costs are considered to have an accuracy of +/- 15%. The following items are excluded from the operating cost estimate:

- Royalties – included in financial analysis.
- All head office costs and corporate overheads.
- Exchange rate variations.
- Escalations.
- Project financing costs.
- Interest charges.
- Political Risk Insurance.
- Land compensation/landowners costs.
- Subsidies to the local community.
- Rehabilitation costs.
- Amortisation and depreciation charges.

18.2.2.2 *Basis of Estimate*

The Project operating cost development methodology is summarised in Table 65. Where there has been a deviation from the standard, the reasoning has been detailed.

All process plant operating costs are compiled in United States of America dollars (USD). All operating costs exclude any VAT (value added tax).

Item	Minimum Standard	Method Used	Comments
General			
Typical Accuracy Range	±5% to 10% for known operations; ±10% to 15% for new operations	±10% to 15%	
Contingency	Not applied	Not included	
A – Operating Cost Methodology			
A1 Staffing			
A1.1 Staffing Levels	Detailed manning schedule developed for Operations and Maintenance	Detailed manning schedule developed for Operations and Maintenance personnel.	Base on similar operational requirements in Australia and offset for specific US role requirements were necessary.

Item	Minimum Standard	Method Used	Comments
A1.2 Cost Rates	Detailed	Detailed and based on known rates and on costs for Health Insurance, Payroll tax, FUTA, SUTA, social security and Medicaid and Annual Leave	Figures for health care (ave value) are based on a split of married with children, married and singles , (1/3 rd equal basis) for site based roles.
A2 Consumables	Calculated	Detailed estimate	Based on reagent and equipment vendor submissions and recommended annual consumables or from first principles.
A3 Maintenance	Detailed estimate	See A1.1	
B – Operating Cost Basis			
B1 Labour			
B1.1 Labour Cost Rates	Detailed	Refer A1.2	
B1.2 Labour Burden Rates	Detailed	Refer A1.2	
B1.3 Labour Hours	Detailed	Not applicable	Refer to the OPEX estimate spreadsheet for details.
B1.4 Labour Overheads/ Management Costs	Detailed	Refer A1.2	On costs included in labour rate build-up
B2 Utilities and Consumables			
B2.1 Power Costs	Detailed calculation based on budget quotation	Energy costs based on the preliminary tender average price $0.091 kWh	Electrical consumption calculated from the electrical load list and typical load factors.
B2.2 Water Costs	Detailed calculation	Not calculated directly but indirectly through the equipment list, labour schedule, maintenance factors and power consumption.	Water supply from existing bores and “Red Tank” located near the golf course.
B2.3 Fuel Costs (Mobile Equipment)	Detailed calculation based on budget quotation	Fuel cost after rebate of A$1.00/L, with vehicle usage rates based on calculated run hours and typical fuel burn rates.	San Bernadino “Red Diesel for off highway cost $3.80 USD/Gallon

Item	Minimum Standard	Method Used	Comments
B2.4 Consumables	Detailed calculation based on budget quotation	Detailed calculation based on budget quotations.	Usage based on rates from vendors or calculated from first principles. Wear liners have been included here rather than in maintenance costs.
B2.5 Supplies and Reagents	Detailed calculation based on budget quotation.	Detailed calculation based on budget quotations.	Usage based on metallurgical test work or modelling.
B3 Plant Maintenance			
B3.1 Maintenance Materials	Detailed estimate	A % of the capital cost has been applied and estimates for large contract works calculated (e.g. mill reline).	This approach is consistent with a Class 3 estimate.
B4 Transport and Logistics			
B4.1 Transport and Logistics	Detailed estimate	Transport for Materials/Consumables – Costs either included in factors used or rates obtained as a delivered to site rate.	
B5 – Other Operating Costs			
B5.1 Business Systems	Preliminary estimate	Allowance included for specific systems for operations, computer equipment and communications for personnel based on manning schedule.	Admin costs have been included.
B5.2 Training	Preliminary estimate	Based on 2% of annual labour cost.	
B5.3 Auditing	Preliminary estimate	Allowance based on previous projects.	
B5.4 Bank Charges	Preliminary estimate	Allowance based on previous projects.	
B5.5 Communications	Preliminary estimate	Allowance based on previous projects.	
B5.5 First Aid	Allowance per person on site	Allowance per person on site.	

Item	Minimum Standard	Method Used	Comments
B5.6 Recruitment	Detailed estimate	Based on A$10,000 per position at 12% turnover for Year 1 and 8% thereafter.	
B5.7 Consultants	Preliminary estimate	Allowance based on previous projects.	
B5.8 Pre-operations	Preliminary estimate	Preliminary estimate	Included in Owners costs
B5.9 Insurance	Preliminary estimate	Preliminary estimate as a 0.5% of the direct capital cost.	Process plant only
B5.10 Escalation	Preliminary estimate	Not Included	
B5.11 Foreign Exchange	Identify equipment and commodities to be imported, basis, values and likely currency.	No foreign exchange components, reagents and consumables identified as at the date of the estimate.	All reagents and consumables quoted in A$ (excluding GST).

Table 67 Operation Cost Methodology

18.2.2.3 Salaries and Labor

The salaries and labour costs are derived from a combination of the 2025 Hays Salary Guide and recent costing for Black Butte Project in Montana to reflect the current labour market conditions and site location.

The process plant operations team is assumed to predominantly be regionally located drive in and out, with the site being located within proximity to Las Vegas, Nevada. This is based on the operational philosophy adopted by the historical project. It is assumed recruitment of personnel would likely be achieved within the southern California / Nevada zones and required technical personnel from within the United States.

The process plant operations personnel are detailed by position, roster type, shift and personnel headcount, refer Table 66.

Process Plant Personnel	Roster	Shift	Number
Plant Manager	5/2	Day	1
Senior Metallurgist	5/2	Day	1
Plant Metallurgist	7/7	Day	2
Production Superintendent	5/2	Day	1
Mill Trainer	5/2	Day	1
Shift Supervisor	7/7	Shift	4
Laboratory Supervisor	7/7	Day	2
Laboratory Technician	4/3	Day	4

Process Plant Personnel	Roster	Shift	Number
Control Room Technician	7/7	Shift	4
Gold Room Supervisor	5/2	Day	2
Gold Circuit Technician	7/7	Day	4
Process Technician	7/7	Shift	24
Maintenance Superintendent	5/2	Day	1
Maintenance Planner	5/2	Day	1
Mechanical Supervisor	7/7	Day	2
Electrical Supervisor	7/7	Day	2
Mechanical Trades	7/7	Day	12
Mechanical Trades	7/7	Shift	4
Electrical Trades	7/7	Day	6
Electrical Trades	7/7	Shift	4
Trades Assistant	7/7	Day	3
Total			85

Table 68 Detailed Process Plant Labor Headcount

The labour rates are annualised and inclusive of the following on-costs:

- Salary.
- Health Insurance (allowance of $15,608 per employee)
- Unemployment Insurance (UI) (3.4% of first $7,000)
- Employment Training Tax (ETT) (0.1% of first $7,000)
- FUTA (0.6% of first $7,000)
- Medicare (1.45%)
- Social Security
- 401K allowance (5%)

Departments	Personnel	Cost USD/year
Laboratory (onsite)	6	$623,629
Process Plant Management	10	$1,695,915
Process Plant Operations	34	$3,620,924
Plant Maintenance	35	$5,038,469
Administration - Non-Process	34	$4,303,284
TOTAL	**93**	**$15,282,221**

Table 69 Summary Process Plant Labour

18.2.2.4 *Mobile Vehicles and Equipment*

The allocation of mobile equipment to the processing and maintenance groups includes:

- Light vehicles (x15).
- 10 tonne all terrain truck with Hiab crane.
- Forklift (3 t capacity) (x2).
- Skid steer loader (x1).
- Tele-handler 3 tonne (x2).
- Container Lifter 30 tonne WLL (x1).
- Integrated Tool Carrier (x1)
- 25 t mobile crane (Franna or equivalent) (x1).
- Elevated work platform (boom lift) (x3).

For the equipment listed, the annual run hours have been estimated based on an average usage per day.

18.2.2.5 *Power*

Grid power will be supplied by SoCal Edison. Power will be transmitted from the power station at 34.5 kV, delivered to the project site via an overhead transmission line which follows the alignment of the site access road.

The power summary for the process plant and administration is detailed in Table 68.

Area	Power		Annual Usage
	Installed kW	Consumed kW	kWh
Processing			
Crushing and Screening	275	192	1,203,190
Reclaim/Ore Sorting	153	105	844,283
Grinding & Classification	10,478	8318	61,873,327
Carbon in Leach	940	627	4,017,199
Goldroom/Elution	183	128	344,310
Detox and Tailings Filtering	3604	2883	23,234,351
Reagents Mixing	138	111	244,963
Water & Air Services	1328	1063	6,358,988
Workshop	72	50	406,184
Laboratory	61	48	427,488
Administration	219	176	1,414,551
Totals	**17,451**	**13,701**	**100,368,833**

Table 70 Power Consumption

18.2.2.6 Reagents and Consumables

Reagents and consumables include the following cost elements:

- Crusher wear liners.
- Grinding mills wear liners.
- Grinding media for the grinding mills.
- All reagents used in the process.
- Fuel for mobile equipment assigned to the processing or maintenance groups.
- Lubricants, operating tools and equipment, general and operator supplies.

Reagent addition rates were derived from laboratory test work in combination with the historical usage figures from the previous operation. Reagent consumption rates have been calculated on a per tonne of mill feed from the steady state mass balance.

All process plant reagent and consumables are a variable cost component except fuel costs for mobile equipment. The unit costs in Table 69 are inclusive of freight to the site.

Item	Consumption			
	Rate	Basis	(USD)	Basis
Consumables				
Primary Crusher Liners	3 set per year	Allowance	$30,364 per set	Vendor data
SAG Mill Liners	2.5 sets per year	Allowance	$1,086,026 per set	Vendor estimate
Ball Mill Liners	2.0 set per year	Allowance	$301,019 per set	Vendor estimate
SAG Mill Balls	1.03 kg/t ore	Calculation	$2,200 per tonne	Vendor quotation
Ball Mill Balls	0.60 kg/t ore	Calculation	$2,240 per tonne	Vendor quotation
Reagents				
Quicklime	1.21 kg/t ore	Met testing	$270 per tonne	Vendor quotation
Sodium Cyanide	0.36 kg/t ore	Met testing	$3,390 per tonne	Vendor quotation
Carbon	30 g/t ore	Met testing	$5,930 per tonne	Vendor quotation
Sodium Hydroxide	39 g/t ore	Allowance	$580 per tonne	Vendor quotation
Hydrochloric Acid	133 g/t ore	Met testing	$650 per tonne	Vendor quotation
Oxygen	190g/t ore	Testing	$620 per tonne	Allowance

Table 71 Reagents and Consumables Costs

18.2.2.7 Laboratory

Laboratory costs include the costs for assaying of various process streams and mining grade control through the on-site laboratory.

The number of process plant assays has been calculated based on selected process streams and required frequency to monitor the process plant operation, undertake metallurgical accounting and confirm final product specifications.

The Colosseum laboratory will be onsite, and allowance has been included in the estimate at $468,195 per year for the assaying components. Labour costs $614,345 are on top of this and accounted for within Labour.

18.2.2.8 Maintenance

Maintenance costs include the cost for spare parts and maintenance materials to maintain the Process Plant, Administration and other site infrastructure. The maintenance cost has been applied as a percentage of the plant area capital cost. The overall percentage factors categorised by plant area has been summarised in Table 70 for the process plant.

Area	Base Cost USD	% Maintenance Cost	Total Maintenance Cost USD	Total Maintenance Consumables USD
Crushing and Screening	$4,459,953	6.0%	$297,033	$29,436
Reclaim	$2,297,265	6.0%	$152,998	$15,162
Grinding and Classification	$24,446,808	6.0%	$1,628,157	$161,349
Carbon in Leach	$11,415,988	5.0%	$627,879	$57,080
Goldroom/Elution	$3,303,544	3.5%	$129,499	$13,875
Detox/Tailings Disposal	$19,858,306	5.0%	$1,092,207	$99,292
Reagent Mixing	$2,119,409	5.0%	$116,568	$10,597
Water & Air Services	$5,409,491	4.0%	$238,018	$21,638
Workshop	$718,295	10%	$79,013	$7,183
Laboratory	$700,000	10%	$84,000	$14,000
Administration	$3,427,999	5.0%	$188,540	$17,140
Power	$20,454,545	4.0%	$900,000	$81,818
Piping and Infrastructure	$3,683,738	5.0%	$202,606	$18,419
Total	**$102,295,343**			**$5,736,516**

Table 72 Maintenance Costs

Maintenance costs include for contract re-lining of the crushers, grinding mills and plant shutdowns. However, the liner costs are accounted for in reagents and consumables in section.

The direct labour cost for maintenance personnel has been included in the labour cost category.

In total, maintenance materials are approximately 4.2% of the Total Equipment Cost of the process plant.

18.2.2.9 Administration

Administration costs have only covered the processing part of the plant administration, consultants, labour and administration building area maintenance. Other Administration costs have been removed to General and Administrative costs.

18.2.2.10 Freight

The freight cost for reagents and consumables has been applied either as ex works (EXW) or free carrier (FCA) as advised by the supplier.

18.2.3 Infrastructure and Mine Facilities

The mining contractor quotes include provisions for the supply and maintenance of mining infrastructure and facilities including offices and workshops. They are not detailed here as a separate line item.

18.2.4 Water Treatment Plant

Raw water is abstracted from two wells located near to the Ivanpah sub-station. The water is of high quality (low TDS and pH near to neutral) requiring no further treatment (apart from filtration) to allow it to be used for processing applications. A potable water plant produces potable quality water. Again, the raw water quality is such that very little treatment is required (filtration and sterilisation) to produce potable quality water.

18.2.5 G & A

The General and Administration (G&A) costs have been developed by Dateline based on estimates of the number of administrative staff that will be required for the project, associated ancillary costs and the lease and operation of a Las Vegas administrative office.

19. ECONOMIC ANALYSIS

Project economics for the 2 Mtpa operation are based on inputs developed by GRES, AMDAD, Agapito Associates, Tundra and Dateline. Economic results presented in the Technical Report suggest the following conclusions, assuming a 100% equity project, and a gold price of USD4,200/oz.

- Mine Life 6 years
- Production Life 10.4 years
- Pre-Tax $NPV_{5\%}$ USD785 million, IRR: 49.5%
- After-tax $NPV_{5\%}$ USD551 million, IRR: 38.6%
- Payback (After-tax) 3 years
- NSR Royalty Paid USD35.8 million
- County Taxes Paid USD16.6 million
- California State Taxes Paid USD124.8 million
- United States Federal Taxes Paid USD206.5 million
- Cash costs (C1, including Royalties) USD1,651/oz Au
- All In Sustaining Costs (AISC) USD1,825/oz Au

19.1 Methods, Assumptions and Basis

Project cost estimates and economics results are presented on an annual basis. Based upon design criteria presented in this Technical Report Summary, the level of accuracy of the estimate is considered ±10-15%.

Costs and economic results are presented in Q2 2026 U.S. dollars unless otherwise stated. No escalation has been applied to capital or operating costs. The 5% discount rate used is a goldmining industry standard in North America generally used for comparability purposes among projects; it is not intended to fully reflect consideration of cost of capital, risk adjustments, or other factors.

Technical economic tables and figures presented in this volume require subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding, which are not considered to be material.

19.2 Principal Assumptions

Parameters used in the analysis are described in Section 13 (Mining Methods), Section 16 (Market Studies and Contracts) and Section 18 (Capital and Operating Costs). These parameters are based upon current market conditions, vendor and contractor quotes and design criteria developed by Dateline and their consultants, and benchmarks against similar existing projects.

19.2.1 Taxation

The economic analysis has been undertaken on the Project at the level of Colosseum Rare Metals, Inc., the 100% owned U.S. subsidiary of Dateline. The taxes considered in the analysis include:

- U.S. Federal Income Tax
- California State Income Tax
- San Bernardino County Property Tax
- California Sales/ Use Tax
- Payroll and Associated Taxes and Levies

19.2.2 Net Smelter Royalty

The executed asset purchase agreement between Dateline and Lac Minerals (USA) LLC (LAC) included a provision for the payment of a Net Smelter Returns (NSR) Royalty for production from Colosseum.

The NSR is calculated on a quarterly basis. The NSR is calculated as receipts less allowable deductions, multiplied by the royalty percentage of 2.5%.

Allowable deductions mean the following determined without duplication:

a) all tolling charges, refining and smelting charges, treatment charges, costs of assaying analysing and sampling, representation expenses, metal deductions and losses, umpire charges, and penalties that are either paid for or incurred by Payor or its Affiliates for or in connection with mineral treatment, beneficiation or refining processes or procedures after Products leave the Mine Site (meaning Payor's mine excavations on the Mining Claims and ancillary facilities including, without limitation, leach pads, milling facilities, mineral recovery facilities, stockpiles and storage facilities on and in the vicinity of the Mining Claims owned and operated by Payor or its Affiliates);
b) all costs and expenses actually incurred with, or in connection with, the transporting, insuring, stockpiling, warehousing, shipping, and moving any of the Products produced from the Mining Claims and the delivery of such Products to a customer, refinery or other place of mineral treatment, including without limitation, all transportation costs, insurance costs and expenses, shipping and delivery costs, agency and brokerage fees and commissions, and storage charges;
c) sales, use, gross receipts, customs duties, severance, government royalties, ad valorem, VAT and other taxes and governmental charges, if any, payable with respect to the existence, severance, production, removal, sale, processing, transportation, or disposition of Products, but excluding taxes based on the net income of Payor or its Affiliates;
d) insurance, consignment, and any discounts or rebates given to customers for off specification or damaged product; and
e) marketing and other sales costs, including sales commissions or brokerage costs and fees.

On September 1st, 2022, LAC's parent company, Barrick Gold, announced that it had sold a portfolio of 22 royalties to Maverix Metals. Maverix Metals was acquired by Triple Flag Precious Metals on January 19th, 2023.

Triple Flag Precious Metals is the current holder of the 2.5% Net Smelter Returns Royalty over Colosseum.

19.2.3 Gold Price Assumption

The Technical Study Summary was calculated using a gold price assumption of USD4,200/oz Au.

19.3 General Economic Assumptions

- The financial analysis was performed on Proved and Probable Mineral Reserves. All other mined material was classified as waste and assigned no economic value.
- The Project is designed at a production rate of 6.3 ktpd. Fresh ore production will originate from the open pit mine and will be treated using conventional CIL technology. Once ore is exhausted from the pit, the Mineral Reserves in the stockpiles will then be processed.
- The Project's NPV was determined on a pre-tax and after-tax basis.
- Annual cash flows used for NPV calculations are assumed to be realized at year-end.
- All costs and sales do not consider inflation or escalation factors.
- All gold sales are assumed to occur in the same period as produced.
- Details of capital and operating costs are provided in Section 18 of this Technical Report Summary.
- Cash flows shown include payment of royalties.
- Progressive and final closure costs are included in the period incurred.
- The financial analysis includes working capital adjustments to provide for difference in the timing of sales and incurrence of obligations and the time of cash received and expended.

19.4 Economic Analysis

19.4.1 Capital Expenditures

Capital costs have been developed from first principles with quotes for all major equipment components. A turnkey engineering, procurement and construction model has been used as the basis for project construction.

The Technical Report Summary contemplates a 14-month period for engineering, construction and commissioning. Contract mining and a third-party gas-fired generating plant are included in the costs. The workforce is assumed to be based in Las Vegas, Nevada and no provision has been made for accommodation.

Summaries of capital costs are shown in Table 71. The R&M costs for the processing plant include minor capital replacements. The current mine life does not warrant capital expenditure for major components of the processing plant.

Capital Expenditure Item	Initial Capital Cost (USDM)	Operational Phase Capital (USDM)
Capitalised Mining	$ 16.3	$ 36.2
Process Plant	$ 95.0	-
Process Infrastructure	$ 25.9	-
Management, Engineering, EPC Services	$ 47.1	-
Preproduction Costs and Capital Spares	$ 23.9	-
Reclamation	N/A	$ 10.0
California Sales and Use Tax	$ 9.2	-
Sub-Totals: Capital Expenditures	$ 249.1	$ 46.2
Combined Engineering Growth and Contingency	$ 25.5	-
Total Capital Costs	**$ 274.6**	**$ 46.2**

Table 73 Capital Cost Estimate Summary

The Project also estimates closure costs of USD10 million to be spent in Year 11, to be offset by the salvage value of the processing plant at USD10 million.

19.4.2 Operating Expenditures

Mining costs have been provided by prominent U.S. contract miners. Power costs are based on connected to local grid power and the establishment of a 34.5 kV overhead powerline along the route of the access road to site.

Processing and G&A costs have been developed from first principles with major consumable supply component quotes and competitive U.S. labour rates. The operating costs account for the workforce being hired on a drive-in-drive-out basis out of Las Vegas, Nevada.

Summaries of operating costs are shown in Table 72.

Operating Cost Item	Units	Years 1-6	LOM Yr 1-11
Mining Costs	USD/t mined	$4.15	$4.63
Processing Costs	USD/t processed	$20.54	$20.56
G&A Costs	USD/t processed	$4.48	$4.45
NSR Royalty	USD/t processed	$2.73	$1.74
Refining Costs	USD/t processed	$0.32	$0.22

Total Cash Costs	**USD/t processed**	**$62.03**	**$45.66**

Table 74 Operating Expenditures

19.4.3 Revenue

At the gold price assumption of USD 4,200/oz Au, the Project generates USD2.4 billion in revenue over the life of the Project.

19.4.4 Cash Costs and All-In Sustaining Costs

Cash costs as defined in guidance from the World Gold Council include non-cash remuneration for site personnel and AISC include corporate or regional general and administrative costs, including share-based remuneration. Project cashflows, cash costs/oz and AISC/oz are presented on a site-level basis and, therefore, do not include these elements.

The average total cash cost over the life of the mine is estimated at USD1,651 per ounce of payable gold and All-in sustaining costs of USD1,825 per ounce of payable gold. Total cash cost and All-In Sustaining costs for the Project are summarized in Table 73.

Period	Cash Costs (USD/oz Au)	Sustaining Costs (USD/oz Au)	AISC (USD/oz Au
Years 1-6	1,466	189	1,655
LOM (Years 1-11)	1,651	174	1,825

Table 75 Cash Costs and All-In Sustaining Costs (USD/oz)

19.4.5 Working Capital

Working capital will vary over the mine life based on revenue, operating costs, and capital costs. Gold sales assume a customary advance payment upon shipment arrangement to be in place; therefore, 10% of the monthly value of gold produced is assumed to be in finished gold inventory and settled in the following month. The turnover rate is approximately 30 days for third-party accounts payable. Internal labour costs are assumed paid in the month incurred. All working capital is assumed to be recaptured by the end of the Project life, and the closing value of the accounts is zero. First fills of consumables and other operating supplies are included in project capital. The working capital was calculated by Dateline.

19.4.6 Cashflow Profile

A summary of the annual cash flows and the details of the cash flow model for the FS are presented in Table 74, including total for the life of the Project.

Cash Flow Summary	Units	Totals	0	1	2	3	4	5	6	7	8	9	10	11
Gold Production	**Koz**	573.4	-	53.3	73.8	71.2	68.2	83.7	102.4	35.3	24.2	24.2	24.2	12.9
Gold Price	USD	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200	4,200
Gold Sales	**USDM**	2,377	-	220.5	305.7	294.9	282.4	346.8	424.5	146.7	100.4	100.4	100.7	53.8
Cash Operating Costs		-												
Mining	USDM	381	-	62.3	76.7	71.8	60.9	42.8	20.2	10.2	10.1	10.1	10.0	5.7
Processing	USDM	424	-	32.6	40.4	41.4	40.3	41.4	40.4	41.4	40.3	41.4	40.3	24.0
G&A	USDM	92	-	7.8	8.8	8.7	8.7	8.7	8.7	8.7	8.7	8.7	8.7	5.1
NSR Royalty	USDM	36	-	2.8	4.3	4.2	4.2	6.2	8.8	2.1	0.9	0.9	1.0	0.5
Refining	USDM	5	-	0.4	0.6	0.6	0.5	0.7	0.8	0.3	0.2	0.2	0.2	0.1
Operating Taxes	USDM	45	0.1	5.3	6.1	5.8	5.2	4.7	4.0	3.7	3.3	3.1	2.7	1.5
Sub-total: Cash Operating Costs	**USDM**	982.4	0.1	111.3	137.0	132.4	119.8	104.5	82.9	66.3	63.6	64.5	63.0	36.9
Cash Operating Margin	**USDM**	1,394	-0	109	169	162	163	242	342	80	37	36	38	17
Capital Costs		-												
Initial Capex	USDM	275	274.6	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capex	USDM	38		27.7	6.0	3.4	0.9	-	-	-	-	-	-	-
Reclamation & Closure	USDM	30												10.0
Salvage	USDM	-30												-10.0
Sub-total: Capital Costs	**USDM**	312.6	274.6	27.7	6.0	3.4	0.9	-	-	-	-	-	-	-
Pre-Tax Cash Flow	**USDM**	986	-285.4	69.6	150.7	147.2	135.7	224.0	338.2	79.9	36.2	35.4	37.5	17.0
Income Taxes	USDM	302	10.7	14.6	17.2	33.3	54.2	62.9	67.8	15.4	7.6	7.6	7.7	3.7
After-Tax Cash Flow	USDM	779	-285.4	66.9	145.5	125.7	107.5	179.4	273.8	65.0	29.3	28.4	30.0	13.3
After-Tax Cumulative Cash Flow	**USDM**		-285.4	-218.5	-73.0	52.7	160.3	339.6	613.4	678.4	707.6	736.0	766.1	779.4
Payback	**Years**	3.6												
Pre-Tax NPV5%	**USDM**	785												
Pre-Tax IRR	**%**	49.5%												
After Tax NPV5%	**USDM**	551												
After Tax IRR	**%**	38.6												

Production Summary	Units	Totals	0	1	2	3	4	5	6	7	8	9	10	11
Mining														
Ore	kt	20,630.9	15.7	3,526.3	4,699.7	4,323.1	3,769.7	2,760.2	1,536.1	-	-	-	-	-
Waste	kt	62,264.2	1,743.9	14,193.1	14,669.0	13,402.4	11,150.1	6,342.0	763.7	-	-	-	-	-
Total Material Mined	**kt**	82,895.0	1,759.6	17,719.4	19,368.7	17,725.6	14,919.8	9,102.2	2,299.8	-	-	-	-	-
Stripping Ratio	Waste: Ore	3.0	111.0	4.0	3.1	3.1	3.0	2.3	0.5	-	-	-	-	-
High Grade														
Milled Ore	kt	9,318.0	-	1,365	2,005	2,000	1,571	1,317	1,060	-	-	-	-	-
Grade	g/t Au	1.51	-	1.25	1.26	1.22	1.31	1.84	2.74	-	-	-	-	-
Contained Gold	koz	451.8	-	55	81	78	66	78	93	-	-	-	-	-
Low Grade														
Milled Ore	kt	3,957.8	-	178	-	-	429	683	946	1,722	-	-	-	-
Grade	g/t Au	0.63	-	0.63	-	-	0.64	0.63	0.63	0.63	-	-	-	-
Contained Gold	koz	80.8	-	4	-	-	9	14	19	35	-	-	-	-
Marginal Grade														
Milled Ore	kt	7,355.0	-	-	-	-	-	-	-	278	2,000	2,000	2,005	1,072
Grade	g/t Au	0.41	-	-	-	-	-	-	-	0.41	0.41	0.41	0.41	0.41
Contained Gold	koz	97.6	-	-	-	-	-	-	-	3.7	26.5	26.5	26.6	14.2
Total Ore to Process Plant	kt	20,630.9	-	1,542.5	2,005.5	2,000.0	2,000.0	2,000.0	2,005.5	2,000.0	2,000.0	2,000.0	2,005.5	1,072.0
Grade	g/t Au	0.95	-	1.18	1.26	1.22	1.17	1.43	1.74	0.60	0.41	0.41	0.41	0.41
Contained Gold	koz	630.2	-	58.5	81.1	78.3	75.0	92.0	112.5	38.8	26.5	26.5	26.6	14.2
Recovery	%	91%		91%	91%	91%	91%	91%	91%	91%	91%	91%	91%	91%
Gold Production	**koz**	**567.7**	**-**	**52.7**	**73.1**	**70.5**	**67.5**	**82.9**	**101.4**	**35.0**	**23.9**	**23.9**	**24.0**	**12.8**

Unit Costs Metrics	Units	Totals	0	1	2	3	4	5	6	7	8	9	10	11
Cash Operating Costs														
Mining	$/t mined	4.59	-	3.52	3.96	4.05	4.08	4.70	8.78	-	-	-	-	-
Mining	$/t processed	18.46	-	40.41	38.27	35.89	30.44	21.39	10.07	5.09	5.07	5.04	5.00	5.31
Processing	$/t processed	20.56	-	21.15	20.14	20.71	20.15	20.72	20.14	20.71	20.15	20.72	20.12	22.43
G&A	$/t processed	4.45	-	5.05	4.41	4.37	4.37	4.37	4.36	4.37	4.37	4.37	4.36	4.76
Sub-total: Cash Operating Costs	**$/t processed**	45.66	-	70.08	65.86	63.85	57.57	48.82	36.56	32.00	31.24	31.68	30.83	33.87
Non-Operating Costs														
NSR Royalty	$/t processed	1.74	-	1.82	2.16	2.08	2.08	3.11	4.37	1.03	0.47	0.46	0.48	0.44
Refining	$/t processed	0.22	-	0.27	0.29	0.28	0.27	0.33	0.40	0.14	0.10	0.10	0.10	0.10
Sub-total: Non-Operating Costs	**$/t processed**	1.96	-	2.09	2.45	2.37	2.35	3.44	4.77	1.17	0.57	0.56	0.58	0.53
Total: Cash Costs	**$/t processed**	45.42	-	68.71	65.27	63.34	57.33	49.92	39.34	31.34	30.16	30.69	30.05	33.03
Cash Costs	$/oz	1,651	-	2,009	1,791	1,796	1,697	1,205	778	1,792	2,523	2,567	2,514	2,763
AISC	$/oz	1,825	-	2,602	1,957	1,927	1,787	1,269	822	1,902	2,660	2,696	2,627	2,877

Table 76 Annual Cash Flow Pre-Production to Year 11 in USD terms

19.4.7 NPV, IRR, Payback

Based on the Annual Cash Flow model results, the Project has an unlevered after-tax $NPV_{5\%}$ of USD551 million, and after-tax IRR of 38.6%; and a payback period of 3.6 years at a long-term gold price of USD4,200/oz. The key financial metrics of the Project are summarized in Table 75.

	Life of Project
Pre-Tax NPV (5%)	$785M
Pre-Tax IRR	49.5%
After-Tax NPV (5%)	$551M
After-Tax IRR	38.6%
Payback	3.6 Years

Table 77 Key Financial Metrics

19.5 Sensitivity Analysis

Project sensitivities are summarized in Table 76, Table 77 and Table 78 and are shown graphically in Figure 61. The Project is most sensitive to the gold price. Sensitivity on operating and capital cost are closely matched, with the Project being only slightly more sensitive to capital costs.

Gold Price Sensitivity	-15%	-10%	-5%	Base	+5%	+10%	+15%
Gold Price	$3,570	$3,780	$3,990	$4,200	$4,410	$4,620	$4,830
Pre-Tax NPV5%	$515	$605	$695	$785	$875	$965	$1,055
After Tax NPV5%	$355	$421	$486	$551	$615	$680	$744
IRR%	28.3%	31.9%	35.3%	38.6%	41.9%	45.0%	48.1%

Table 78 Gold Price Sensitivity

Capex Sensitivity	-15%	-10%	-5%	Base	+5%	+10%	+15%
Pre-Tax NPV5%	$838	$820	$803	$785	$768	$750	$732
After Tax NPV5%	$603	$586	$568	$551	$533	$515	$498
IRR%	47.1%	44.0%	41.2%	38.6%	36.3%	34.2%	32.2%

Table 79 Capex Sensitivity

Opex Sensitivity	-15%	-10%	-5%	Base	+5%	+10%	+15%
Pre-Tax NPV5%	$887	$853	$819	$785	$751	$718	$684
After Tax NPV5%	$624	$599	$575	$551	$526	$502	$477
IRR%	42.5%	41.2%	39.9%	38.6%	37.3%	36.0%	34.7%

Table 80 Operating Cost Sensitivity

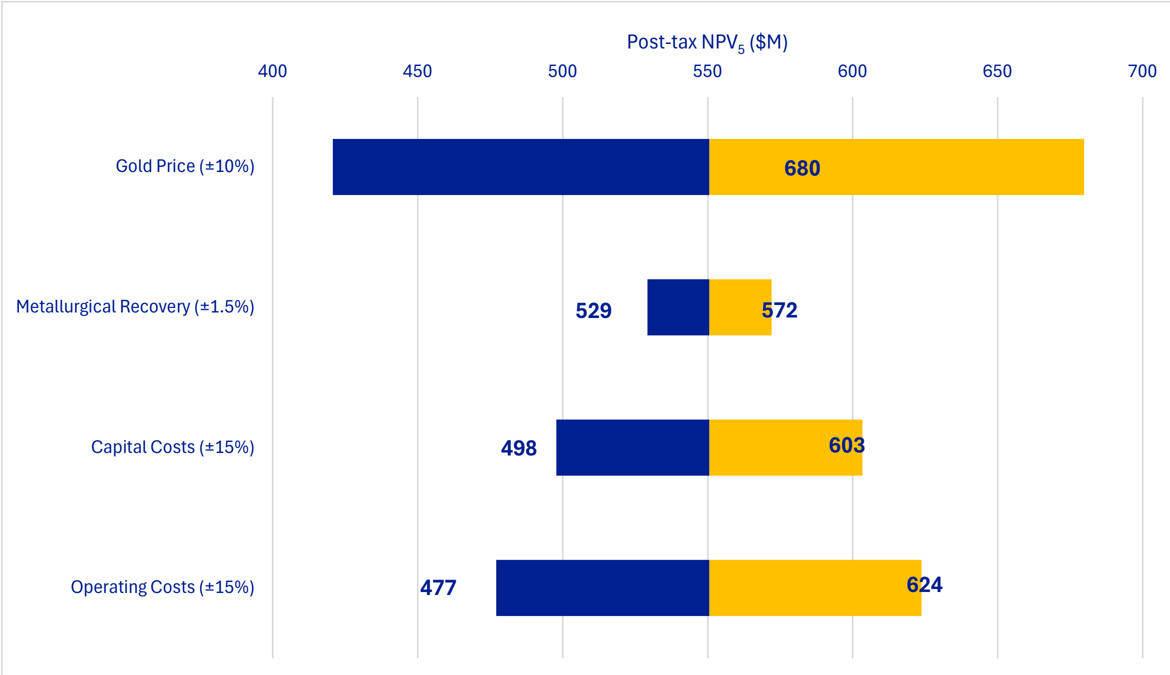


Figure 61 Project NPV5% Sensitivity Analysis

20. ADJACENT PROPERTIES

Dateline, through its 100% owned subsidiary, Colosseum Rare Metals Inc., holds 80 unpatented claims and 3 patented claims for the Colosseum Project. There are no neighbouring claims that have a contiguous boundary.

The project lies within Mojave National Preserve, established in 1994 under the California Desert Protection Act. Mining within the Preserve is regulated “subject to valid existing rights” and remains subject to laws and regulations applicable to mining in units of the National Park System. Dateline confirms that it holds valid mining rights recognised by the US Department of the Interior and the National Park Service.

20.1 Adjacent and nearby mineral properties

The nearest major operating mine is the Mountain Pass rare earths mine, operated by MP Materials. Colosseum lies about 10 km north of Mountain Pass. Mountain Pass is the only active primary rare earths operation in the United States, and it sits within the same broader alkaline igneous corridor as Colosseum.

20.2 Nearby exploration projects

To the north and east of Colosseum, several Australian listed companies including Locksley Resources, Bayan Mining and Minerals and Great Northern Minerals have commenced exploration activities in the past 18 months targeting rare earths and antimony.

21. OTHER RELEVANT INFORMATION

21.1 Project Execution

21.1.1 General

The project execution schedule developed for the Colosseum project shows a total project duration of 88 weeks. This duration includes the FEED (front end engineering and design) study through to the completion of dry commissioning. The FEED study and detailed design (following the FEED) commenced on 12th December 2025 and is scheduled for completion by June 30, 2026. Site works are scheduled to commence on June 5, 2026. Bulk earthworks have commenced and will be completed prior to June 5, 2026, to facilitate a quick ramp up into full construction. Construction continues through to the completion of dry commissioning scheduled for August 31, 2027, with practical completion being achieved one week later.

21.1.2 Engineering and Procurement

The schedule proposed for the Colosseum project includes a Front-end Engineering and Design (FEED) phase. This phase is scheduled to begin prior to the commencement of detailed design. However, there is no distinct cutoff between FEED and detailed design, with the FEED rolling straight into detailed design. The FEED is intended to shorten the period between gaining final project approval and commissioning. Finalisation of basic engineering documentation and preparation of long lead equipment procurement documentation is carried out during FEED.

Detailed design activities are largely developed following completion of the FEED phase, whilst long lead time procurement has commenced and will continue through FEED. The detail design and procurement activities associated with the process plant are largely completed by the GRES Brisbane office whilst the GRES U.S. office takes care of the non process infrastructure (NPI) aspects of the project.

Wherever possible (and appropriate) process equipment is procured from North American vendors. In some instances, where equipment is not available in North America equipment sourced in other countries (generally Australia) is used.

21.1.3 Construction Methodology

The construction of the Colosseum process plant follows conventional methodologies used for a plant of this size (2.0 Mt/a) and nature (crush, grind, CIL). Construction assumes a five-day, sixty hour working week, as per advice received from local construction contracting companies. This will allow make-up time to recover or accelerate schedule if needed.

An Early Contractor Involvement (ECI) process has commenced, to ensure realistic timeframes and construction costs are locked down prior to entering into construction contracts. Four major construction packages are proposed:

- Civil (concrete) works;
- Structural, mechanical, piping (SMP) works;
- Electrics, Instruments and Control (EI&C); and
- Specialised mechanical, including mill installation.

Various specialist subcontractors may also be engaged (e.g. mill erection). Such subcontracts are let by the prime contractor.

21.1.4 Schedule

21.1.4.1 Summary

An execution schedule for the delivery of the Colosseum project has been developed, and the near critical path is shown in Figure 62. The execution schedule shows a total project duration of 88 weeks. This duration includes Front-End Engineering and Design (FEED) through to dry commissioning and accounts for holiday breaks at Christmas / New Year. The execution schedule shows the critical path running through the following activities:

- Detail design
- Major equipment procurement
- Site access and bulk earthworks.
- Civil works (concrete)
- Tankage erection
- Mechanical installation
- Piping installation
- Electrics installation
- Commissioning

All other activities fit within the critical path. Figure 62 shows the critical path schedule for the project.

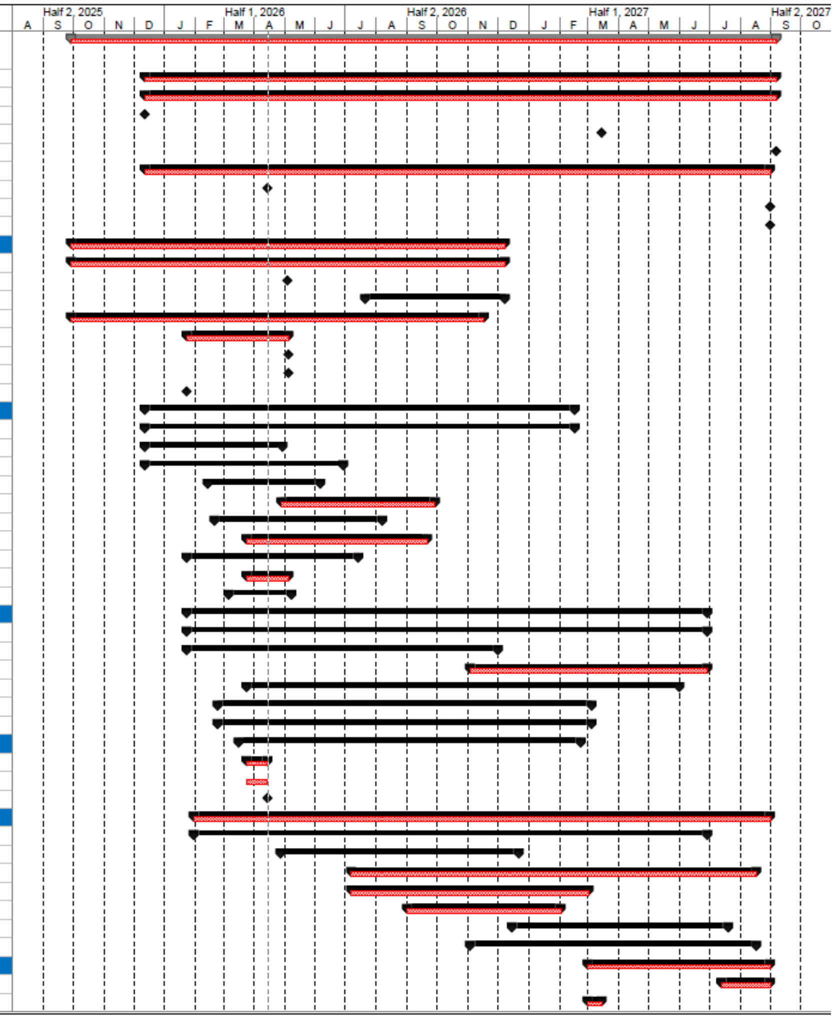
Several summary schedules have been developed from the execution schedule (level 4). These are shown in Appendix M including the following:

- Level 3 summary schedule.
- Critical path schedule.

It should be noted that these schedules are preliminary in nature, more accurate schedules will be developed during the FEED and detail design phases.

Near Critical Summary Level

ID	Task Name	Start	Finish	Duration
0	300203 - COLOSSEUM FEED PROJECT SCHEDULE - FEED & EPC/M	Mon 29/09/25	Mon 6/09/27	97 w
1	MILESTONES	Fri 12/12/25	Mon 6/09/27	86.43 w
2	Dateline Resources Responsibilities	Fri 12/12/25	Mon 6/09/27	86.43 w
3	FEED Letter of Intent issued	Fri 12/12/25	Fri 12/12/25	0 w
8	Mobilise Mining Contractor	Mon 15/03/27	Mon 15/03/27	0 w
9	Ore Required on ROM Pad	Mon 6/09/27	Mon 6/09/27	0 w
11	GRES Responsibilities	Fri 12/12/25	Tue 31/08/27	85.55 w
15	Mobilise Plant Site Bulk Earthworks Contractor	Tue 14/04/26	Tue 14/04/26	0 w
26	Project Practical Completion	Tue 31/08/27	Tue 31/08/27	0 w
27	Commence Ore Commissioning	Tue 31/08/27	Tue 31/08/27	0 w
28	DESIGN DELIVERABLES MILESTONES	Mon 29/09/25	Mon 7/12/26	60.07 w
29	Process Plant	Mon 29/09/25	Mon 7/12/26	60.07 w
32	Earthworks Drawings	Sun 3/05/26	Sun 3/05/26	0 w
34	Civil Drawings	Mon 20/07/26	Mon 7/12/26	20 w
58	Mechanical & Platework Drawings	Mon 29/09/25	Mon 16/11/26	57.07 w
71	Infrastructure & Mining	Fri 23/01/26	Tue 5/05/26	14.55 w
72	Access Roads and Bulk Earthworks	Tue 5/05/26	Tue 5/05/26	0 w
75	Mining	Tue 5/05/26	Tue 5/05/26	0 w
79	Pipeline from Red Tank	Fri 23/01/26	Fri 23/01/26	0 w
80	FEED ENGINEERING DESIGN & DRAFTING	Fri 12/12/25	Mon 15/02/27	57.5 w
81	Process Plant	Fri 12/12/25	Mon 15/02/27	57.5 w
82	Standard Drawings & Specs	Fri 12/12/25	Tue 28/04/26	17.65 w
91	Process Design & General	Fri 12/12/25	Sun 28/06/26	26.35 w
102	Earthworks Design	Fri 13/02/26	Sat 6/06/26	16.12 w
127	Area 340 - Adsorption	Mon 27/04/26	Mon 28/09/26	22 w
163	Filter Building	Fri 20/02/26	Fri 7/08/26	24 w
177	Area 370 - Electrical Design and Drafting	Tue 24/03/26	Sun 20/09/26	25.8 w
191	Infrastructure & Mining	Fri 23/01/26	Mon 13/07/26	24.5 w
196	Access Roads and Earthworks (client)	Tue 24/03/26	Tue 5/05/26	6 w
211	Raw Water System	Fri 6/03/26	Fri 8/05/26	9 w
216	MAJOR PROCUREMENT ITEMS	Fri 23/01/26	Tue 29/06/27	72.55 w
217	Process Plant	Fri 23/01/26	Tue 29/06/27	72.55 w
245	CP010 Ball Mill	Fri 23/01/26	Tue 1/12/26	44.55 w
252	CP010 Mill Lube Systems	Tue 3/11/26	Tue 29/06/27	32 w
364	CP200 Electrical Equipment	Tue 24/03/26	Tue 1/06/27	60 w
391	Infrastructure & Mining	Mon 23/02/26	Fri 5/03/27	51.57 w
418	CP015 - Pressure Filter	Mon 23/02/26	Fri 5/03/27	51.57 w
472	CONSTRUCTION CONTRACTS	Mon 16/03/26	Mon 22/02/27	47 w
473	CP502C Bulk Earthworks - Process Plant	Tue 24/03/26	Tue 14/04/26	3 w
474	Tender and Spec & Tender Earthworks Contract	Tue 24/03/26	Tue 14/04/26	3 w
475	Mobilise Earthworks Contractor to Site	Tue 14/04/26	Tue 14/04/26	0 w
496	CONSTRUCTION	Fri 30/01/26	Tue 31/08/27	80.57 w
501	Infrastructure & Mining	Fri 30/01/26	Tue 29/06/27	71.57 w
524	Bulk Earthworks	Sun 26/04/26	Mon 21/12/26	34.2 w
533	Process Plant	Mon 6/07/26	Tue 17/08/27	56.05 w
534	Civil Works	Mon 6/07/26	Mon 1/03/27	32 w
565	Tank Erection	Mon 31/08/26	Mon 1/02/27	20 w
582	Mechanical Installation	Mon 14/12/26	Tue 20/07/27	29.05 w
703	Electrical & Piping Installation Work	Mon 2/11/26	Tue 17/08/27	39.05 w
782	COMMISSIONING	Mon 1/03/27	Tue 31/08/27	26.13 w
783	Process Plant	Tue 13/07/27	Tue 31/08/27	7 w
811	Infrastructure & Mining	Mon 1/03/27	Mon 15/03/27	2 w



Page 1

Figure 62 *Colosseum Critical Path Schedule*

Two methods are used to arrive at site manning histograms (refer appendix M). The first method assumes the build-up of site personnel to be reflected by a bell curve applied to man hours and durations for each construction activity. Using this method leads to a sharp peak in site manning requirement of approximately 190 personnel. The second method assumes the build-up of site personnel to be reflected linearly across man hours and durations applied to each construction activity. Using this method shows a plateaued peak of approximately 220 site personnel. The use of the linear approach is not strictly (theoretically) correct, however the resultant plateau in site manning numbers more accurately reflects what is found in practise. Considering the above it is suggested that planning should allow for a maximum site manning of approximately 200 personnel.

It must be noted that the construction phase of the execution plan assumes accommodation to be unconstrained. This is based on there being adequate levels of accommodation available in Primm (associated with the Primm Valley Hotel and Casino). Discussions with the Primm Valley Hotel and Casino have provided confidence that there will be a surplus of rooms available to meet construction needs.

21.1.4.2 Long Lead Items

The longest lead item included in the Colosseum project is the HV and LV MCC's and switchrooms. The proposals received shows a delivery of 54 weeks to 64 weeks following placement of purchase order. The FEED process is managing the risk of delivery trough placing orders or build slots for long lead items.

The following lists the longest lead items on the project.

- MV and LV MCC's and switchrooms — 54 – 64 weeks from PO placement FCA USA / Canada 2 weeks for delivery to site
- MV VSD switchgear — 57 weeks delivered
- Mill motors — 50 weeks delivered
- VFD — 47 weeks delivered
- Tailings filters — 42 weeks delivered

21.1.4.3 Engineering and Design

The engineering and design phase of the project, including FEED is scheduled to be completed within fifty-eight weeks following FEED commencement in December 2025. Procurement of long lead items has commenced. Design of the grinding area can commence early in the FEED process due to the availability of existing mill detail drawings. Vendor drawings of the mill motors will be required early to allow design of the mill concrete to be finalised, implying early ordering of the mill motors. Procurement of process equipment items is scheduled based on the delivery of vendor drawings required for the design process. The design process is highly dependent on the delivery of certified (correct) vendor data, without which design cannot be completed and issued for construction.

Drafting commences simultaneously with engineering with the development of standard drawings and drafting documentation such as the drawing register. Drafting starts relatively slowly until the receipt of vendor data allows for ramping–up of the drafting and engineering effort. During this period the drafting team is expanded as engineering, and the availability of vendor data allows.

21.1.4.4 Erection / Installation

Mobilisation of the earthworks contractor took place in January 2026 under the direction of Dateline. The civil contractor (concrete) is scheduled to mobilise to site in mid July 2026 at which time the bulk earthworks will be complete. The SMP (structural, mechanical, piping) contractor is scheduled to commence on site approximately four months after the civil contractor, whilst the CIL tankage contractor commences a month after the civil contractor. The last contractor to commence on site (electrical) mobilises to site late in April 2027. Commissioning is scheduled to complete in August 2027, with Practical Completion after any outstanding items from commissioning are closed out and demobilisation occurs.

21.1.4.5 Construction Safety

Safety during construction shall be closely supervised. Each major construction contractor shall be responsible for the development and implementation of a safety management plan, incorporating requirements for reporting and emergency procedures. Likewise, the principal contractor shall develop and implement an overall project safety plan. Safety officers (both major contactor and principal contractor) shall be assigned to ensure compliance with the agreed and approved safety management plans. Such plans shall be submitted and approved prior to a major contractor mobilising to site. The overall project safety management plan shall be integrated into the Dateline safety management plan to provide a seamless integrated plan.

21.1.4.6 Commissioning

Precommissioning involves the checking of installed equipment including first lubricant fills and alignment checks and is scheduled to commence mid july 2027. Precommissioning is expected to take approximately thirteen weeks to complete. Dry commissioning involves bump testing of drives and the testing of electrics and controls and is scheduled to complete at the end of August taking approximately four weeks to complete. Practical competion occurs at the end of dry commissioning and is schedue to occur in last week of August 2027.

Ore commissioning and ramp-up commences following practical completion. This phase of the project is not covered by the execution plan.

21.1.5 Schedule Opportunities

The schedule currently allows for working a sixty hour five day week (twelve hour days). The construction duration may possibly be shortened by working weekends or scheduling certain activities acoss both day and night shifts. There is a cost impost associated with this considering overtime payments with these strategies.

The schedule currently shows tankage erection and completion to be on critical path. The design of the CIL tanks and associated civil works can be started early in the detail design phase (using existing designs) allowing fabrication and erection of the tanks (and subsquent activities) to come forward off the critical path.

21.2 Water Management Plan

21.2.1 Introduction

Agapito (2026) was retained to develop a site water management plan for the Colosseum mine development. Site water management for the project site will be critical in order to create a zero-discharge, closed-loop operation that maximizes the water recycling to minimize losses and required recharge quantities in this dry climate. The Agapito report focused on the following items:

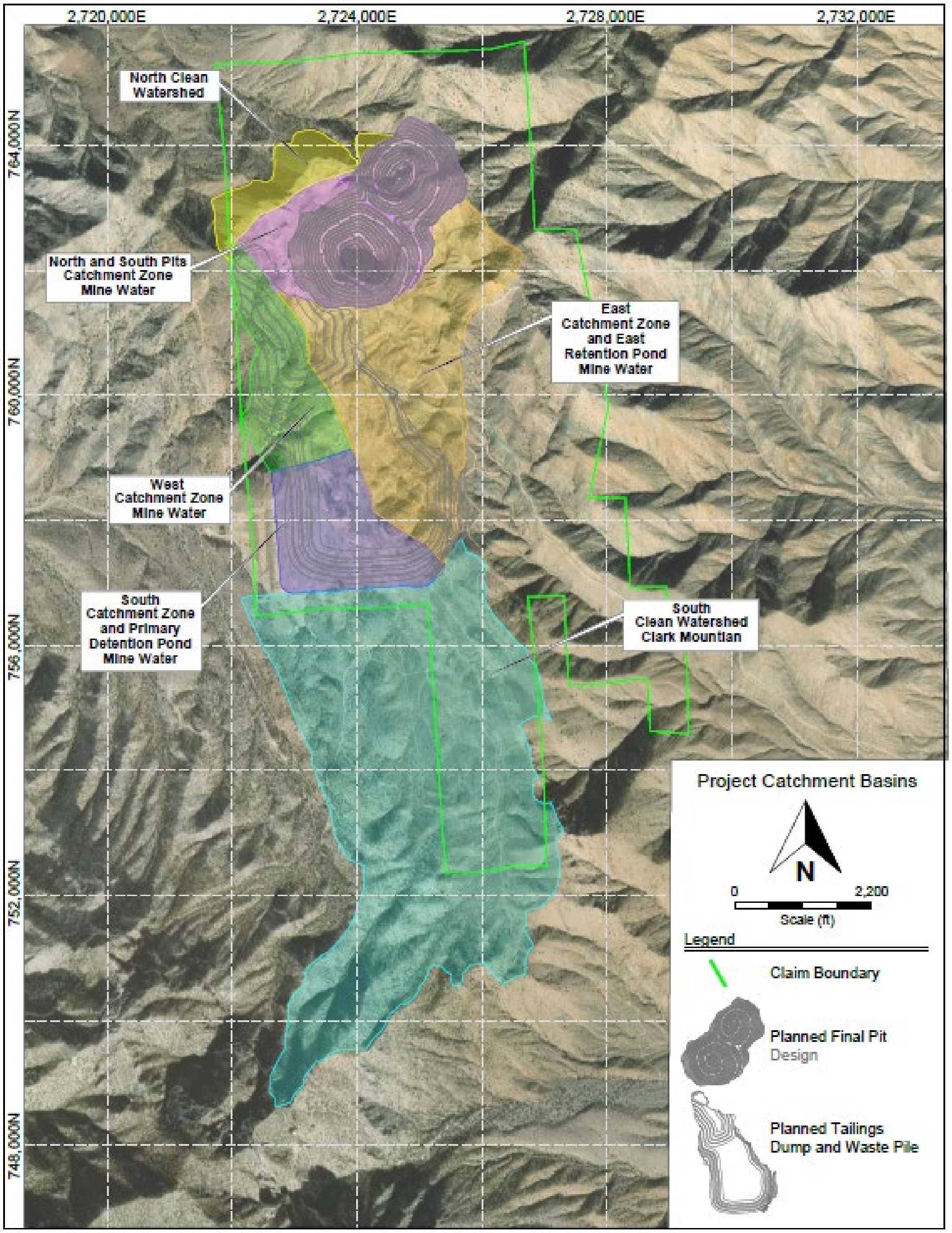
4. Describing the local climate
5. Conducting a water balance analysis
6. Creating Water Management Plans for both clean (off-site) and mine area water (on-site)

The developed Site Water Management Plans will ensure that (1) existing natural drainages are maintained or returned to the pre-1987 condition, (2) all water that comes into contact with mine and mineral processing related activities are captured and disposed of within the claims boundary, and (3) eliminate interaction between project water, groundwater, and legacy structures constructed during the historical mining on the property.

Precipitation runoff water that falls or flows onto the claims boundary can enter the Ivanpah Valley drainage system towards the east, the Shadow Valley drainage towards the west, or the Mesquite Valley drainage towards the north (WESTEC, 1995). During historical operation, all precipitation runoff on the disturbed mine property was directed to the tailings impoundment. Based on AMDAD's projected mining and dumping plans, the Colosseum Mine's claims area can be divided into the following six local surface catchment zones, which are shown on Figure 63.

1. South Catchment Zone (Clark Mountain) – Clean Water
2. North Catchment Zone (Mesquite Valley) - Clean Water
3. West Catchment Zone (Shadow Valley) – Mine Affected
4. Each Catchment Zone (Ivanpah Valley) – Mine Affected
5. South Catchment and Primary Detention Pond – Mine Affected

6. North and South Pits – Mine Affected



1050-02 Dateline (AAI_Base_Map_SP CA Z5 NAD27_rj|_RevBHs_JBW_4-10-26.dwg Layout: Fig 1-2 Catchment Basins|elw/rj(4-24-2026)

Source: Agapito (2026)

Figure 63 Colosseum Mining Project Catchment Basins

21.2.2 *Climatology, Precipitation and Pan Evaporation*

At an elevation between 5,000 and 6,000 feet (ft) above mean sea level (AMSL), the Colosseum Mine is located west of the town of Primm, Nevada (2,600 ft AMSL), along the east flanks of the Clark Mountain range, within the state of California. It is in a region characterized by an arid to semi-arid desert climate typical of the northern Mojave Desert. The regional climate is typical of an arid, mid-latitude high desert environment defined by low annual precipitation, high potential evaporation, and seasonal and daily temperature variability. Summers are generally hot and dry, while winters are cooler with occasional rain or snow. Temperatures can vary widely both seasonally and within a single day.

While average annual precipitation is relatively low, the occurrence of short-duration, high-intensity rainfall events establish peak runoff generation and, therefore, governs the design criteria for site water management features for the project site including, diversion channels, sediment control ponds, and retention basins.

The average total annual precipitation is approximately 7.80 inches as determined by Searchlight (USC00267369) NOAA weather station with comparable 100-year event characteristics, though positioned at a lower elevation. During Colosseum Mine operation, precipitation was reported to average less than seven inches per year according to rain gauge measurements at the mine (U.S. Environmental Protection Agency 1992).

Pan evaporation values are used to estimate open-water evaporation losses from sediment control ponds, retention basins, and exposed tailings surfaces. No local pan evaporation station has been operating since 2000. Therefore, regional data from Lake Mohave, the Mojave Desert, and southwestern Nevada were used to set a conservative evaporation rate of approximately 66.9 inches/year for the project area. This value is applied in water balance calculations to account for open-water and surface evaporation.

21.2.3 *Normal Water Inflows and Outflows*

Mine water management is a function of the balance of water introduced onto the disturbed mining boundary through natural precipitation, the dust suppression of the mining process, and ore processing activities. This ensures that adequate storage/retention volume is available to prevent any mine water from leaving the mining boundary regardless of its source, and to optimize the pumped water recharge rate to ensure the required quantity of water is available for the project. While the below items do not identify every minor inflow or outflow of water, they do identify the key factors that are involved in the day-to-day operation of the project.

21.2.3.1 *Water Inflows*

The total rainwater catchment basin for the Colosseum Mine is approximately 1,094 acres and includes areas within and outside of the project's boundary. Historical records show that precipitation occurs on the

Colosseum Mine area approximately 28 days per year for an average annual precipitation of 7.80 inches, resulting in an average of approximately 711 acrefeet (ac-ft) of potential runoff per year.

The project is forecasted to require approximately 600,000 gal of recharge water per day to account for water lost during gold processing, dust suppression, and other minor water uses. This daily water recharge will be provided to the project from two water wells in the Ivanpah Valley, which are located approximately 8.3 miles east of the project site. Historical site water demands, 1987-1992, from the Ivanpah Valley exceeded 700,000 gpd primarily due to higher water content within tailings slurry.

The North and South pits were mined between 1987 and 1993, with groundwater inflow currently present in the South Pit due to historic mining below the groundwater table. It is expected that groundwater inflow into the North Pit will occur once mining progresses below the potentiometric surface elevation anticipated at 5,350 ft (WESTEC, 1995). Pump tests indicate groundwater flow through the fractures in the rock mass is very low, with inflow rates of between 16.6 and 21.5 gallons per minute (gpm). Future inflow could be expected to increase as the pits are deepened, as groundwater quantities are a function of exposed excavation surfaces.

21.2.3.2 Water Outflows

The ore processing plant water system is projected to consume approximately 382,146 gpd through the water that cannot be extracted from the dry-stacked tailings material prior to disposal on site. This projection is based on a 20% moisture content by weight for the filtered tailings. Due to the regions dry climate, some of this water contained in the dry stacked tailings should evaporate during material rehandle and movement before its final deposition.

The mining process will require dust suppression in the form of applying water to the haul roads, which is expected to consume approximately 170,000 gpd. All of which will be lost due to evaporation. Common road dust control additives such as calcium chloride, magnesium chloride, or organic polymers would reduce water consumption however, these controls have not been included to reduce water outflow balance at this time.

Other site water outflow demands are estimated to require 6,000 gpd. These include potable water for site restrooms and shower facilities as well as maintenance equipment washing.

Site water management controls require all mine affected water to remain within property's claim boundaries however evaporation outflow is expected. An average 0.18 inch per day evaporation rate is estimated for water outflow of each water management detention or retention pond.

- West Detention Pond area is approximately 2.14 acres and has the capacity to retain 45.671 ac-ft, while maintaining 3 ft freeboard allowance. Evaporation outflow, if water is present, averages 10,400 gpd.

- East Retention Pond area is approximately 3.51 acs and has the capacity to retain 39.749 ac-ft prior to utilizing a spillway channel to convey capacity overage to the Primary Detention Pond. Evaporation outflow, if water is present, averages 17,200 gpd.
- The Primary Detention Pond area is approximately 14.94 ac and has the capacity to retain 153.732 ac-ft while maintaining 3 ft freeboard allowance. Evaporation outflow, if water is present, averages 73,000 gpd.
- All retention ponds will serve primarily as storm water runoff collections for mine affected water and secondarily as site surge capacity. Priority will be taken to utilize collected pond water for mining and milling operations as an alternative to supply from Ivanpah Valley water wells.

21.2.3.3 Net Water Inflow

During typical mining and processing operations, the total daily site inflow is estimated to be between 500,000 and 600,000 gpd sourced from the Ivanpah Valley water wells. Nearly 69% of the inflow will be dedicated for mineral processing and approximately 30% necessary for dust suppression. Approximately 1% of daily water usage has been assumed for sanitary purposes, potable water, and equipment washing for maintenance services.

21.2.4 Rainfall Events for Water Management Design Plan

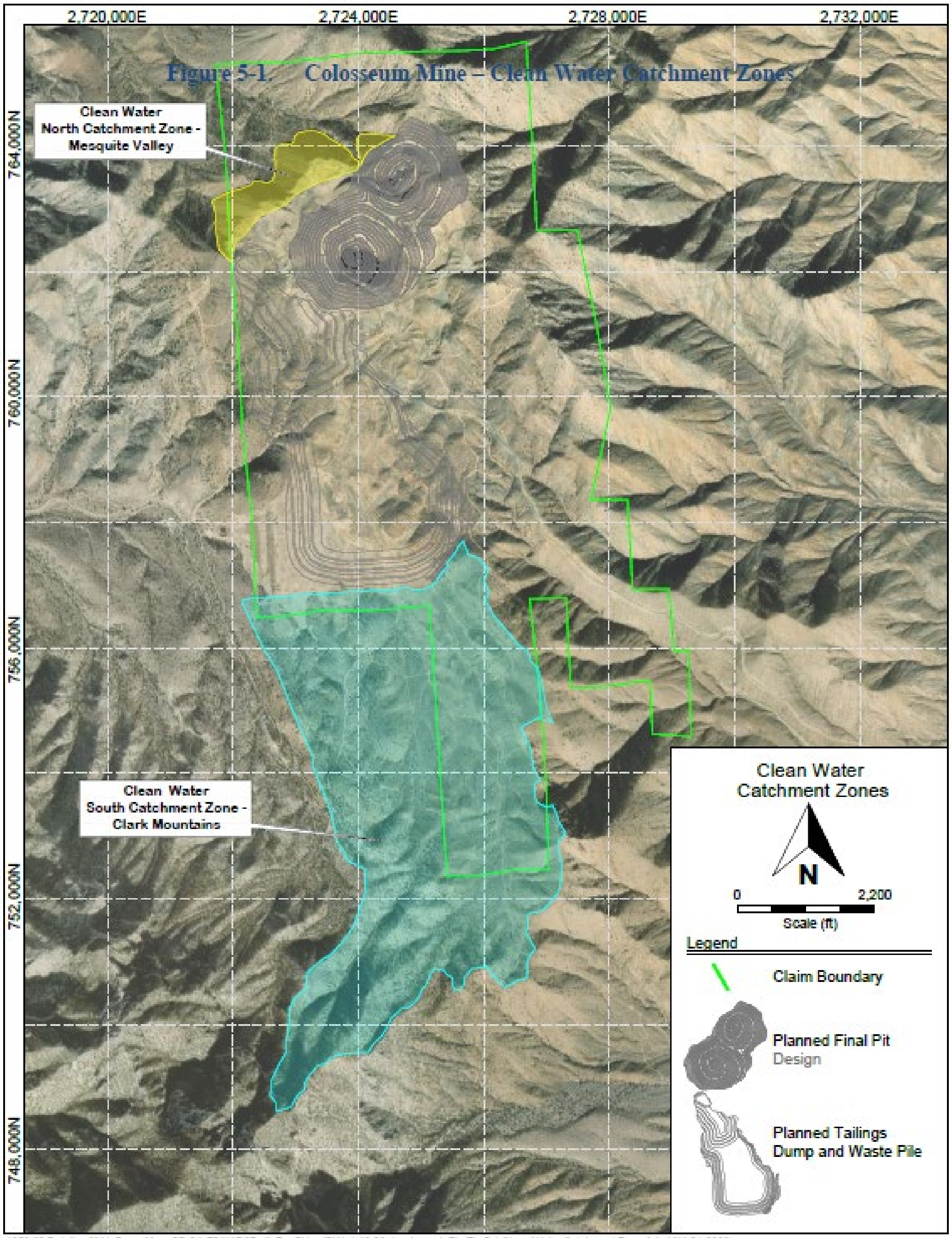
Rainfall runoff management is based on the maximum precipitation intensity for a 100-year, 60-minute, and 24-hour duration events for all impoundment capacities and water management controls/structures. The National Oceanic and Atmospheric Administration (NOAA) precipitation frequency estimates for the Colosseum Mine site, and the total rain for the two benchmark events are estimated to produce 3.02 in. and 4.73 in., respectively. With the project's catch basin approximately 1,094 acres, this results in a total precipitation volume on the project site of 275.3 ac-ft for the 60-minute event and 431.2 ac-ft for the 24-hour event.

21.2.5 Clean Water Management Plan

The clean water management plan will intercept the flow of precipitation runoff water before it is allowed to come into contact with the disturbed areas within the project boundary. The Colosseum Mine is positioned at the apex of three drainage basins, so water within the project boundary can only flow off the property towards the west, north, and east. The only source of clean water runoff that can flow onto the property is from Clark Mountain from the south. There are four main clean water catchment zones within the claim's boundary and are shown in Figure 64.

21.2.5.1 South Catchment Zone – Clark Mountain

The South Catchment Zone encompasses an estimated total watershed area of 570 acres bounded to the south, east, and west by undisturbed terrain of the Clark Mountain (Figure 64). To contain and redirect the clean water runoff from entering the disturbed area of the project boundary, a series of berms and a west flowing drainage channel, along the southern side of the existing tailings impoundment, must be constructed to divert the runoff to the Shadow Valley.



Source: Agapito, 2026

Figure 64 Colosseum Mine – Clean Water Catchment Zones

21.2.5.2 North Catchment Zone – Mesquite Valley

The waste rock from the historic and planned mining is classified as Group C waste and was historically dumped around the existing North and South pits excavation limit. The waste dump piles on the northwest side of the North Pit have sat undisturbed for more than 30-years without causing any reported issues to the surrounding surface area or groundwater.

Dateline is planning to not disturb any of the dump faces on the downstream side that could affect the stability of these historic dumps, while promoting water runoff through water management controls such as grading, to flow away from their crests towards the southeast into the open pit. Additionally, the construction of downstream water management controls within the valley would require the disturbance of previously undisturbed land.

Because of these reasons, it is assumed that no water management controls will be installed into the north catchment zone that flows into the Mesquite Valley, by way of the upper end of Green's Canyon. Therefore, Dateline will maintain this area and the surrounding areas to minimize any additional effect from future mining.

21.2.6 Mine Affected Water Management Plan

Mine affected water is any water which has come into contact with mine waste materials, disturbed geologic formations, or mining process chemicals. Water management of the mine affected water will impound any potentially hazardous water that is collected within the mining area from discharging off the property and/or reaching the groundwater to adversely affect the waters of the state. This can be completed through engineered designed waste placement and the construction of water impoundments.

To prevent mine affected water from reaching the groundwater, Dataline is planning to install continuous, impermeable liners that cover all potential discharge areas (waste rock piles, dry stacked tailings, mine affected water impoundments, etc.) where both solid and liquid waste is to be discharged within the project boundary to ensure its isolation. Lining consists of installing a long-term impenetrable layer or barrier on top of the existing ground surface to contain what is placed on top of the liner and prevent interaction, connection, erosion, or contamination with what is below the liner. Liners can be constructed using high-strength geomembranes and/or installing a natural or geosynthetic clay layer. These systems are proven technologies that are implemented globally and used for a wide range of containment applications. Current estimates of the lined area are approximately 283 acres and include portions of the historic project's waste rock dumps and the majority of the tailings impoundment.

The proposed restart of the Colosseum Mine will utilize a zero-discharge, closed-loop water management strategy. This strategy requires new detention ponds to be constructed of inert earthen fill with upstream contact surfaces lined by HDPE membrane for the catchment zones. During a rare storm event, pooled water in the detention ponds will be pumped into the existing water system that feeds the plant and dust suppression activities to reduce pumped recharge water requirements being pulled from the Ivanpah Valley wells.

All rainfall runoff that flows into the North and South pits from the surrounding areas will collect at the bottom of the active mining pits. Additional water from groundwater inflows will also accumulate within the mining pits. When measurable water accumulates in the bottom of either pit, it will be pumped into the existing water system that feeds the processing plant and dust suppression activities to reduce the recharge quantity needed from the Ivanpah wells, or to a retention pond for evaporation. Upon completion of mining and reclamation requirements, pit dewatering will cease and water will gradually fill the pits until equilibrium is reached.

22. INTERPRETATION AND CONCLUSIONS

22.1 Project Risk

Material risks and uncertainties that could reasonably affect the reliability or confidence in the Project outcome are provided in Table 79.

The Project is an advanced-staged development project that has undergone engineering for several years and previously operated as a fully permitted operation from 1988-1993. To manage cost and schedule risk, Dateline retained GR Engineering Services of Brisbane, Australia and Tucson, Arizona to undertake a benchmarking study to assess the appropriateness of capital and operating cost estimates, construction and ramp-up schedules, owner's costs and key components of the Project. As such, the development risks that are within the control of Dateline are considered low to moderate.

Risk	*Description*	*Probability*	*Severity*
Gold Price	The Project economics are sensitive to gold price. Sustained downward gold price trends could render the project uneconomic.	Medium	High
Foreign Exchange	The Project capital and operating costs and revenue are all in US dollars, with the resulting economics in US dollars The only foreign exchange risk is when the resultant financials are converted to Australian dollars for reporting.	Low	Low
Political Setting	The Trump Administration in the United States has been extremely supportive of the development of Colosseum. Changes in Government may have a negative impact on the Project.	Medium	Medium-High
Permitting & Regulatory Approvals	The Project operates under the existing Plan of Operation and valid Mining Rights	Medium	High
Property Holdings	The Project is located on both patented and unpatented claims in California, with the claims held on land administered by the Bureau of Land Management	Low	Medium
Infrastructure	The Project relies on the use of existing and local infrastructure, the condition of which is well known and functional. Significant deficiencies would result in increased capital expense.	Low	Medium
Understanding of Mineral Resource	The Project viability relies upon historical drilling as well as recent drilling to develop and assess the Mineral Resource model. Future drill results could adversely affect the interpretation of parts of the deposit, with impacts to Mineral Resources and production estimates.	Low	Low
Capital Costs	Some areas are well defined and others not as much and there are unknowns that can affect the price, including equipment and input price movements.	Medium	Medium
Reagents & Consumables	The process operating costs are sensitive to global changes in reagents and consumables pricing.	Medium	Medium

Fuel	The Project operating costs are sensitive to global changes in prices for diesel.	Medium	Medium
Mobile Equipment Capital	Mobile equipment prices are an important part of the Project capital. Significant increases could impact the Project economics.	Low	Low-Medium
Process Technology	Extensive testing has been completed to identify the most suitable technology and equipment in the process. The performance of the selected equipment could negatively impact Project economics.	Low	Low
Climatic Events	Day to day mining operations could be significantly impacted by extreme hot or cold weather events	Low	Low
Groundwater	Day to day mining operations could be impacted by groundwater inflow.	Low	Low
Tailings Disposal	The Project includes a filter plant, which will allow the tailings to be co-deposited with waste rock. Performance of the filter plant could impact the operation.	Low	Low-Medium
Reclamation & Closure	There is potential for reclamation activities to extend beyond the active planned closure period and therefore generate greater sustaining costs. Additional risk lies should the closure design not perform as intended.	Low	Medium

Table 81 Project Risks

22.1.1 Mineral Resource Estimate Risks

Risks to be considered for the Mineral Resource estimation include the mineralisation consisting of felsite breccia, which could vary during the mining and recovery processes. Changes in factors, such as metal prices, recovery, and costs may affect the cut-off grade, which would alter the reported Mineral Resource numbers. Geotechnical parameters could also have an impact on the pit shell used to report the Mineral Resource.

22.1.2 Mining and Mineral Reserve Risks

There are unknown risks and uncertainties, which could have a material adverse effect. This should not be considered an exhaustive list, but the key risks are included.

For the mining area, there are two critical risks identified that need to be a focus for future planned work. These are related to grade control and reconciliation, as well as the engagement of the mining contractor in sufficient time to be ready as per the current Project schedule.

The risk related to grade control could be a positive or negative risk dependent on the reconciliation of the actual contained grade when compared against the Mineral Resource model. Within the Technical Report Dateline has included personnel and resources to develop a robust grade control regime, including ongoing RC drilling in advance of open cut mining to inform mine planning and feed into the Process Plant. The intent will be to develop robust systems to predict gold grades in each mining block, thereby maximising the value of the Mineral Resource through the prioritisation of high value mining blocks.

Regarding the risk of ensuring the mining contractor is engaged with sufficient time to be ready as per the current Projects schedule this is easily addressed by planning to have commercial discussions and negotiations as part of a formal tender process with mining contractors at least 9 months prior to required site mobilization. This should be easily achieved based on current Project timing, and recent experiences and interactions with four of the United States' leading Tier 1 mining contractors and other service provides as part of this Technical Report Summary, that are aware of the Project and enthusiastic to be involved.

The are other identified risks with controls and actions for the execution works outlined in detail as part of this Technical Report Summary. Also, there is planned future work, particularly to be completed as part of the planned execution and pre-production stage, will be outlined. Other items related to the mining area for future work, particularly to be addressed in the planned execution and pre-production, to address identified risks are outlined below:

- Blasting Parameters – Post the completion of initial execution work involving rock mass data for the geotechnical and hydrogeological studies, this information will be utilized in conjunction with the Mineral Resource model to update drill and blast designs and assumptions, including overall pattern designs, assumptions for percentage of trim blasts required, use of emulsion assumption for wet holes, etc.

- Stockpile design and operations – Mine scheduling focused on the use of stockpiles to best deliver feed to the processing plant to maximize metal recovery and overall value. Within the detailed design execution works, further simulation of the stockpile operation and management will be required to ensure practical and safe operation of pre-crusher stockpiles to obtain mine planning requirements.

- Geotechnical - The risk related to geotechnical design parameters for pit walls could be seen as a risk or a potential opportunity if pit walls could steepen and is part of the geotechnical consultant's recommendations from their BFS geotechnical assessment. Refinement of the design will be completed via on site staff prior to the commencement of mining and is likely to include drill and blast parameters, such as pre-splitting.

- Operational Readiness – Forward works in this area focused on consideration of the broader project approach required for attraction and retention of people, and strategic procurement of key consumables and equipment. This is addressed through implementing an engagement strategy, procurement and tender plan for the mining contractor, that specialize is maintained both of their equipment procurement and people recruitment and attraction pipelines.

- Mineral Resource Estimation and Orebody Definition – Geological modelling underpins resource estimation and mine planning. Inaccuracies in the interpretation of lithology, grade distribution, structural controls, or mineral continuity can result in misclassification of ore and waste, leading to suboptimal mine design and financial underperformance. Mitigation requires rigorous geological data collection, validation, and continuous model refinement throughout the LOM.

- Cost Overruns and Budget Blowouts – Unexpected increases in construction, labour, fuel, tariffs or equipment costs can lead to budget overruns. Early engagement with potential suppliers, service providers and partners that has already occurred within this Technical Report Summary assists in addressing this risk.

- Workforce Availability and Retention – Attracting and retaining skilled labour is a persistent challenge. Competition from other mining operations and lifestyle factors can lead to high turnover and increased training costs. As outlined by the Dateline team members during this Technical Report Summary, they will continue to engage with both local – Las Vegas, and regional – California & Nevada communities to identify locals to employ and have part of the operations team which will assist with retention of people. This has also been discussed and is part of the mining contractor's operations philosophy and strategy.

- Environmental and Climate Risks – The region is subject to extreme weather events, including extreme heat and cold, which may disrupt operations, damage infrastructure, and delay production schedules. This is considered as if, for example, pit access is limited due to snowfall, there is sufficient stockpiles on the ROM pad to ensure continued mill feed for over a month. Environmental compliance, particularly around water management and biodiversity, is also a critical operational consideration which is part of this broader Technical Report Summary.

- Health, Safety, and Equipment Reliability – Mining operations carry inherent risks related to worker safety, equipment failure, and operational hazards. Maintaining high safety standards and ensuring equipment reliability are essential to avoid costly downtime and reputational damage. For example, safety in design has been considered as part of the haul road design and road layout in this Technical Report Summary and will also require a detailed traffic management plan to be developed in execution works.

- Community Engagement – The Project and Dateline must continue to navigate complex stakeholder relationships, including continued engagement with local communities, Government agencies and landholders. Failure to manage these relationships effectively can result in delays, legal challenges, or loss of social license to operate.

22.2 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resources

The exploration of Colosseum has largely been undertaken using diamond drilling, resulting in the collection of high-quality core material. This allows a higher degree of structural information to be gathered about the deposit when compared to other drilling methods such as reverse circulation.

Drill samples have been subject to industry standard QA/QC protocols with regards to collection, transport and analysis, providing confidence in the resulting data collected. Since 2022, the same geological team has collected, analysed and reported results from Colosseum, providing a good continuity of processes and systems.

The near sub-vertical nature of mineralisation in the breccia pipe means that drilling cannot be simply conducted on 'fences', with a range of drilling azimuths and dips required to test the limits of the orebodies. The drilling by Dateline has adequately defined the margins of the orebody by utilising different drilling angles.

22.3 Metallurgical Testwork

Colosseum has the benefit of metallurgical testwork prior to the original mining phase, followed by five years of operating data on ore from the mine. This operational knowledge provided confidence that a high recovery of 91-92% could be achieved from the orebody.

Subsequent testwork as part of this Technical Study confirmed that a recovery of 91-92% is again achievable using a grind size of P_{80} passing 106µm, followed by a conventional CIL circuit.

22.4 Mineral Resource Estimates

The Colosseum deposit is located at the southern end of the Sevier foreland thrust belt in the southern Basin and Range Province, SW USA. The project lies within in the Clark Mountain Mining District in the northeast portion of the Clark Mountain Range.

The deposit itself is associated with the emplacement of a breccia complex into Precambrian gneissic basement rocks. The complex is comprised of two felsite breccia pipes that form a northeast-southwest elongate zone, which contains mineralised zones of disseminated auriferous pyrite.

The Colosseum deposit comprises two felsite breccia pipes, the North Pipe and the South Pipe.

The North Pipe (also referred to as the North Pit) is defined by approximately 0.5 Moz of gold within 21.05 Mt of Measured and Indicated Mineral Resources at an average grade of 1.44 g/t Au and at a cut-off grade of 0.2 g/t Au and 0.13 Moz of gold with 7.69 Mt of Inferred Mineral Resources at an average grade of 0.53 g/t Au.

The South Pipe (also referred to as the South Pit) is defined by approximately 0.35 Moz of gold within 11.03 Mt of Measured and Indicated Mineral Resources at an average grade of 2.00 g/t Au and at a cut-off grade of 0.2 g/t Au and 0.09 Moz of gold with 4.72 Mt of Inferred Mineral Resources at an average grade of 0.62 g/t Au.

22.5 Mineral Reserve Estimates

The Project is at a BFS stage based on a conventional open pit, truck and hydraulic excavator operation feeding a nominal 2.0 Mtpa processing plant. The Mineral Reserve is supported by this Technical Report Summary.

This Technical Report Summary presented to support the Mineral Reserve that is developed to a Bankable Feasibility Study level. This includes a mine plan that is technically achievable and economically viable. Mine optimization, mine design, mine schedule, and mining costing were undertaken by AMDAD.

This Technical Report Summary was undertaken by a team of industry professionals involving numerous consultants and professionals focused on technical areas including infrastructure, approvals, environmental, governance, community, local considerations, operations readiness, geochemistry, hydrogeology, geotechnical engineering, metallurgical, and BFS discounted cashflow models

A large number of mine schedule runs have been run at both the strategic optimization and detailed BFS mine schedule level to ensure the pit development is economically robust and viable, and practical to mine with consideration of the bounds of this Technical Report Summary.

Mine production constraints were imposed to ensure that mining was not overly aggressive with respect to the equipment anticipated for use at Project and pit geometry considerations such as sink rates, while ensuring adequate mine production to allow blending to ensure quality feed and maximize value through the processing plant.

The schedule has been produced using mill targets and stockpiling strategies to enhance the Project economics, while also considering the ramp-up and commissioning requirements at of both the processing plant and mine mobile equipment. The constraints and limits are reasonable to support the Project economics which are used to justify the statement of Mineral Reserves.

In summary, the final result incorporates a Starter and Final Pit for the North Pit and a single stage cut-back of the South Pit. This Technical Report Summary shows that the mine plan is technically achievable and economically viable taking into consideration all material Modifying Factors

Mineral Reserves are reported using the following guidelines as defined by Subpart 229.1300 of Regulation S-K 1300 and are based on open pit mining methods. The Mineral Reserves are forward-looking information and actual results may vary. The risks regarding Mineral Reserves are summarized in this Technical Report Summary and in the risk Section 12.13.

Areas of uncertainty that may materially impact the Mineral Reserve estimates include: changes to long-term metal price assumptions; changes to include operating, and capital assumptions used, including changes to input cost assumptions such as consumables, labour costs, royalty and taxation rates; variations in geotechnical, mining, dilution, and processing recovery assumptions; including changes to pit phase designs as a result of changes to geotechnical, hydrogeological, and engineering data used; and changes to environmental, permitting and social license assumptions.

Agapito was engaged by Dateline to conduct the geotechnical assessment of the Colosseum pit slopes which forms part of this Technical Report Summary. Open pit slope recommendations were provided to guide pit optimization and mine design by the AMDAD team.

22.6 Mining Methods

Mineral Reserves were estimated for the Project assuming open pit mining methods with conventional equipment for drilling, blasting, loading and haulage appropriate for the proposed production rates. This Technical Report Summary assumes a contract miner model for mining operations, involving the provision of a full mining operations service and construction and operation of the mining infrastructure required to support the mining fleet.

A 20' (6.1m) bench height was used aligning with the Mineral Resource estimate block height and based on the required production rate and appropriately sized equipment. Drilling will involve 6' (152mm) diameter holes for all production shots, with modified blasting required to meet geotechnical wall control recommendations. Furthermore, the use of backhoes in final wall areas will assist with final wall quality outcomes.

The mining operations will be executed by a tier 1 U.S. contract mining company, selected for its capability to manage large-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 90-120 tonne class hydraulic excavators and 50-60 tonne class articulated haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts.

Reasonable mine designs, mine production schedules, and mine costs have been developed for the Project. Costs for mining and mining infrastructure have been provided based on the analysis of this Technical Report Summary mine schedule completed by a tier 1 U.S. mining contractor and consider local site and U.S. requirements, and availability of Mineral Resources such as equipment and labour.

The mine production schedule is based on commencement of mining in Month 6 of Year 1, with mill feed to commence in Month 9 of Year 1. A mining ramp-up is included in the schedule considering mobilization and commissioning of the mining contractor to site, with first two months of mining at 50% of production, which then lifts to 100% in Month 8 of year 1.

Prior to the commencement of mining, all mining infrastructure will be constructed by the mining contractor in year -1, to facilitate efficient mine start-up. However, any delays in the commencement in the Project may delay the start date of mining in which case, the dates provided in the schedule and this Technical Report Summary are indicative only and cannot be relied upon.

The WRD required for the mine are substantial, with detailed work on the waste rock dump design and developed completed by AMDAD in this Technical Study.

22.7 Recovery Methods

The proposed comminution circuit is considered suitable for treatment of the hard ore, however the overall circuit complexity and large number of equipment items to be commissioned and optimized increases risk of an extended ramp-up period to reach design capacity and the target metallurgical performance. The main areas of risk for the Project during ramp-up period are in the materials handling, crushing and grinding areas, specifically due to the number of unit operations and conveyors, transfer points, and wear areas located in the crushing and SAG/ball mill circuits.

The plant will use a CIL circuit, which is well known and common within the global mining industry. Risk is mainly associated with liberation of the fine gold within the Colosseum ore with gold particle size being <10 µm in size. A target grind P_{80} size of 106 µm was selected for the Project. A finer grind to further increase liberation would incur additional regrinding capital and operational costs, as well as increasing the slurry viscosity with an increasing finer grind particle size distribution without delivering a meaningful increase in recovery.

The risk of not achieving the recoveries noted in this Technical Report will result in lower revenues and economic indicators.

22.8 Project Infrastructure

There is no existing infrastructure at the Colosseum site, with all infrastructure from the original mining operation removed.

As part of pre-works, Dateline has upgraded the access road, which is suitable for construction and operational purposes.

Dateline intends to construct a 2.0 Mtpa processing plant and associated infrastructure at Colosseum. Power and water will be sourced locally from facilities that were used in the original mine development.

Given the project's proximity to Las Vegas, there are no plans to build an accommodation camp or airstrip for Colosseum, with personnel to 'drive in-drive out' from the site and be based in Las Vegas.

22.9 Tailings Storage Facilities

Due to the inclusion of a belt filter in the processing plant circuit, tailings material will be dried and moisture reduced to <20%. The filtered material will then be trucked and deposited within a cell contained in the waste rock material.

As such, no tailings storage facility is planned for the Colosseum Project.

22.10 Water Management

Dateline has developed a 'closed loop' water model for the Colosseum Project, with the use of the belt filter significantly reducing the amount of water that will need to be drawn from the Colosseum #1 and #2 water bores. Despite increasing the plant throughput over the original mine development, Dateline intends to pump less water on an annual basis.

22.11 Environmental and Social Considerations

Environmental and Social/Community impacts for Colosseum were originally addressed for the operation by a Draft Environmental Impact Statement (EIS) (Bureau of Land Management 1985a) and Final EIS (Bureau of Land Management 1985b).

Environmental and Social/Community commentary was also provided by the U.S. Environmental Protection Agency (EPA) in 1993.

Mining and processing from 1988 to 1993 were managed in accordance with the EIS and Plan of Operations and Reclamation Plan for the Colosseum Project, 21 July 1986.

Dateline notes that when mining and processing operations were suspended in 1993, the EIS and Plan of Operations remained in place and would apply to the resumption of activities at Colosseum. These documents detail Dateline's minimum standard and conditions that the project operates under.

22.12 Social and Community Impact

Dateline intends to operate Colosseum as a 'drive in-drive out' residential operation with personnel to be based in Las Vegas, Nevada, located 55 miles from the project along I-15.

From a community and social impact, the Company believes that having personnel based in the wider Las Vegas region (population +2.4 million) provides a better work-life balance for families compared to a mining village structure.

The city of Las Vegas contains a large choice of school, housing, sporting, socialising, transport and entertainment options for Colosseum personnel. The Company believes that these choices will be a key attraction for recruiting personnel for the operation.

Dateline has acquired the rights to 27 acres of real estate infrastructure at the base of the Colosseum access road, near I-15. It is anticipated that the majority of personnel will drive to this location and be bussed the remaining eight miles to the mine site for each shift.

22.13 Permitting

Dateline has an existing Plan of Operations for the resumption of activities at Colosseum. It also has vested Mining Rights for the Project. The Plan of Operations and Mining Rights have been confirmed as valid by the U.S. Department of the Interior, Bureau of Land Management, National Parks Service and the San Bernardino County.

23. RECOMMENDATIONS

All required work is complete for this Technical Report Summary, and no additional work is necessary for this phase. This Technical Report Summary presents a project that is ready for submission for financial and other support necessary to progress to the next phase.

The next phase of the Project is the FEED process, which leads to a financial investment decision. The recommendations that follow are considered part of the detailed design phase and represent the normal progression of the Project from a BFS to construction.

23.1 Mineral Resource and Exploration

The Geology and Mineral Resources of the North and South Pipes at Colosseum are well understood by the Competent Person. Accordingly, the following section provides recommendations intended to identify potential opportunities for further study or advancement. No budget has been assigned to these recommendations at this time.

- The North Pipe deposit potentially extends at depth towards the northeast, as indicated by drilling in Q4 2025.
- The mineral resources are constrained by the depth capacity of the drill rigs used to test the deposit.
- Additional deep infill drilling should be used to help define potential deep mineralisation not detected by earlier drill holes.
- Additional drilling should also target the area which has been identified as an underground mining target to the northeast of the North Pit to increase the confidence of the mineralisation and bring that material into the Mining Reserve.
- Additional drilling exploring the claims, following up on high priority geophysical anomalies.

23.2 Mining

All required work is complete for this Technical Report Summary, and no additional work is necessary for this phase. This Technical Report Summary presents a Project that is ready to progress to the next phase. The next phase of the Project is detailed design, to be completed as part of the planned execution and pre-production stage.

For the mining area, there are two critical risks identified that are recommended to be a focus for future planned work. These are related to grade control, as well as the engagement of the mining contractor in sufficient time to be ready as per the current Project schedule.

The focus of grade control work is to design a robust system to ensure the Mineral Resource is maximised.

For the recommendation regarding the mining contractor ensuring it is engaged with sufficient time to be ready as per the current projects schedule this is easily addressed by planning to have commercial discussions and negotiations as part of a formal tender process with mining contractors at least 9 months prior to required site mobilization. This should be easily implementable based on current Project timing, and recent experiences and interactions with four established US mining contractors and other service provides as part of this Technical Report Summary, that are aware of the Project and enthusiastic to be involved.

The recommendations that follow are considered part of the FEED phase and represent the normal progression of the Project from an FS towards construction. A specific budget has not been compiled for these tasks.

- Secure long lead time items associated with the project.
- Commence an Early Contract Involvement process to define costs and execution strategy for the Processing Plant.
- Commence recruitment of key construction and execution personnel.
- Detail system requirements to support project development.

Several opportunities and risks have been identified within this Technical Report Summary, which can be managed considering the recommendations outlined in this section as the Project progresses. Since mining aspects of the Project are considered to be well understood by the Competent Person. No budget has been assigned to these recommendations at this time.

24. REFERENCES

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25. RELIANCE ON INFORMATION PROVIDED BY THE COMPANY

This Technical Report Summary has been prepared by GRES and the CP Technical Report Summary authors for Dateline. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GRES and the CP Technical Report Summary authors at the time of preparation of this Technical Report Summary;
- Assumptions, conditions, and qualifications as set forth in this Technical Report Summary; and
- Data, reports, and other information supplied by Dateline and other third-party sources.

GRES and the CP Technical Report Summary authors have not researched property title or mineral rights for Dateline as we consider it reasonable to rely on Dateline's legal counsel who is responsible for maintaining this information.

GRES and the CP Technical Report Summary authors have relied upon Dateline and its management to prepare the market analysis, owner costs, closure and reclamation security bond, and the applicable taxes and royalties used in the economic analysis (Section 19) and in various subsections throughout the Technical Report Summary. GRES and the CP believe the data presented by Dateline appear in line with the current market conditions.

The CP Technical Report Summary authors have taken all appropriate steps, in their professional opinion, to ensure that the above information from Dateline is sound.

Except for the purposes legislated under state, federal, and provincial securities laws, any use of this Technical Report Summary by any third party is at that party's sole risk.

Competent Persons who relied on information provided by others and Dateline are not responsible for the content, accuracy, or adequacy of such information, pursuant to the requirements of S-K 1300.

26. COMPETENT PERSONS AND OTHER SPECIALISTS

26.1 Competent Persons

The S-K 1300 Summary Technical Report "Colosseum Gold Project" has been prepared with input and guidance of Competent Persons in the following areas:

Sample Preparation and Exploration Information

Sample preparation and any exploration information in the summary technical report is based upon work conducted or supervised by Mr Graham Craig of Las Vegas, Nevada who is a Senior Exploration Geologist. His contribution relates to the Summary Technical Report "Colosseum Gold Project" effective date 8th May 2026.

Mr Craig's qualifications include a BASc (Geosciences) and a BComm. He has 13 years of experience in mine and exploration geology of precious metals. He is a member of the APEGM and qualifies as a "Competent Person" as defined by the SME Guide. He has visited the Colosseum Gold Project site regularly since May 2021 with most recent visits in May 2026.

Mr Craig is an employee of Colosseum Rare Metals Inc which is a wholly owned subsidiary of Dateline Resources Limited. He has worked on exploration of the Colosseum Gold Project since May 2021.

He has read the SME Guide and confirms the sections of the summary technical report for which he is responsible have been prepared in compliance with the SME Guide. Mr Craig confirms that, at the effective date of the summary technical report, to the best of the his knowledge, information, and belief, the portions of the summary technical report that he is responsible for, contain all scientific and technical information that is required to be disclosed to make the summary technical report not misleading.

Mineral Resource Estimation

Mineral Resource Estimation information in the summary technical report is based upon work conducted by Mr Simon Tear of Brisbane, Australia who is a Resource Geology Consultant. His contribution relates to the Summary Technical Report "Colosseum Gold Project" effective date 8th May 2026.

Mr Tear's qualifications include a BSc (Hons) in Mining Geology. He has over 40 years of experience in mine and exploration geology and Mineral Resource Estimation for precious metals, base metals and industrail minerals. He is a a PGEO with the IGI and a member of the European Federation of Geologists,and qualifies as a "Competent Person" as defined by the SME Guide. He has not visited the Colosseum Gold Project site but relies on close communication with Dateline Resources personnel managing the exploration, sample preparation and drillhole data quality assurance.

Mr Tear is an employee of HS Consultants Pty Ltd which is an independent geological consultancy. He has worked on Mineral Resource Estimation of the Colosseum Gold Project since early 2024.

He has read the SME Guide and confirms the sections of the summary technical report for which he is responsible have been prepared in compliance with the SME Guide. Mr Tear confirms that, at the effective date of the summary technical report, to the best of the his knowledge, information, and belief, the portions of the summary technical report that he is responsible for, contain all scientific and technical information that is required to be disclosed to make the summary technical report not misleading.

Mineral Reserves Estimation

Mineral Reserve Estimation and mine planning information in the summary technical report is based upon work conducted or supervised by Mr John Wyche of Brisbane, Australia who is a Mining Engineering Consultant. His contribution relates to the Summary Technical Report "Colosseum Gold Project" effective date 8th May 2026.

Mr Wyche's qualifications include a BE (Hons) and a BComm. He has over 35 years of experience in mine planning and Mineral Reserve Estimation for precious and base metals. He is a Fellow and Chartered Professional of the AusIMM and qualifies as a "Competent Person" as defined by the SME Guide. He has visited the Colosseum Gold Project site from 9th to 12th December 2025.

Mr Wyche is an employee of Australian Mine Design and Development Pty Ltd which is an independent mining engineering consultancy. He has worked on mine planning for the Colosseum Gold Project since early 2021.

He has read the SME Guide and confirms the sections of the summary technical report for which he is responsible have been prepared in compliance with the SME Guide. Mr Wyche confirms that, at the effective date of the summary technical report, to the best of the his knowledge, information, and belief, the portions of the summary technical report that he is responsible for, contain all scientific and technical information that is required to be disclosed to make the summary technical report not misleading.

Mineral Processing

Mineral Processing information in the summary technical report is based upon work conducted, supervised or reviewed by Dr Deepak Malhotra of Lakewood, Colorado who is a Metallurgical Engineering Consultant. His contribution relates to the Summary Technical Report "Colosseum Gold Project" effective date 8th May 2026.

Dr Malhotra's qualifications include a BSc (Metallurgical Engineering), and MSc (Metallurgical Engineering) and a PhD (Mineral Economics). He has over 50 years of experience in metallurgy and process enmgineering for precious and base metals. He is a registered member of the SME and qualifies as a "Competent Person" as defined by the SME Guide.

Dr Malhotra is the President of DM Consulting, LLC, which is an independent mining and mineral processing consultancy. He has worked on metallurgy and process engineering for the Colosseum Gold Project since September 2025.

He has read the SME Guide and confirms the sections of the summary technical report for which he is responsible have been prepared in compliance with the SME Guide. Dr Malhotra confirms that, at the effective date of the summary technical report, to the best of the his knowledge, information, and belief, the portions of the summary technical report that he is responsible for, contain all scientific and technical information that is required to be disclosed to make the summary technical report not misleading.

Consent certificates for the Competent Persons have been provided to Dateline Resources Limited.

26.2 Other Specialists

The Competent Persons are supported by advice from the following specialists:

GR Engineering Services Pty Ltd

- Process engineering design and cost estimation.
- Infrastructure design and cost estimation.

Agapito Associates LLC

- Openct mine geoetchnical assessment.
- Civil geotechnical assesment for the process plant.
- Stability assessment fo rthe mine waste rock and dry stacked tailings.
- Site water management plan.
- Mine closure requirements.

HRS Water Consultants, Inc.

- Groundwater modelling.

Tundra Resource Analytics

- Finacial modelling.

Dateline Resources Limited (and legal advisors)

- Project description and history.
- Gold price forecast.
- Permitting.

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

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

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td>Sampling techniques</td>
<td> Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.
Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.
Aspects of the determination of mineralisation that are Material to the Public Report.
In cases where ‘industry standard’ work has been done this would be relatively simple (eg ‘reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay’). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information. </td>
<td> As of 1 March 2026, the resource database includes data from 687 holes, for a total of 72,950.74 metres (239,351.38 feet), that were drilled by Dateline and various historical operators in the Colosseum Mine area.
Historic Drilling
The historical drilling was completed from 1972 to 1991 and includes 599 holes for a total of 182,444 feet (55,609 meters) of drilling. Most of the historical drilling was done using reverse-circulation (“RC”) and conventional rotary methods eg percussion drilling. An inventory of known drilling in the area totals 16,948 feet (5,166 meters) in 262 Air Trac holes, 21,691 feet (6,611 metres) in 31 diamond (“DD”) core holes, 132,180 feet (40,288 metres) in 273 reverse circulation holes and 11,625 feet (3,543 metres) in 33 rotary/percussion holes.
A summary of the historic drilling is given below:
<table>
<tr><th>Company</th><th>Date</th><th>Series</th><th># Holes</th><th>Feet</th><th>Type</th></tr>
<tr><td>Draco Mines</td><td>1972-1974</td><td>CP</td><td>5</td><td>7,070</td><td>Core</td></tr>
<tr><td>Placer Amex</td><td>1975-1976</td><td>CP</td><td>18</td><td>8,256</td><td>Core</td></tr>
<tr><td>Draco Mines</td><td>1979-1980</td><td>CH</td><td>27</td><td>11,148</td><td>Rotary/ Percussion</td></tr>
<tr><td>Amselco</td><td>1982-1984</td><td>CM</td><td>162</td><td>95,160</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1983-1984</td><td>EDDH, WDDH</td><td>6</td><td>3,740</td><td>Core</td></tr>
<tr><td>Colosseum Gold Inc</td><td>1987</td><td>C87-1,2</td><td>2</td><td>2,625</td><td>Core</td></tr>
<tr><td></td><td></td><td>C87-3-8</td><td>6</td><td>477</td><td>Rotary/ Percussion</td></tr>
<tr><td></td><td></td><td>ATDH*</td><td>262</td><td>16,948</td><td>Air Trac</td></tr>
<tr><td>Cond Gold Colosseum Inc.</td><td>1988</td><td>C88</td><td>31</td><td>16,415</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1989</td><td>C89</td><td>2</td><td>1,330</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1990</td><td>R90</td><td>53</td><td>15,265</td><td>Reverse Circulation</td></tr>
<tr><td></td><td></td><td>DB90</td><td>6</td><td>690</td><td>Reverse Circulation</td></tr>
<tr><td>LAC Minerals – Colosseum Inc.</td><td>1991</td><td>SP91</td><td>18</td><td>3,220</td><td>Reverse Circulation</td></tr>
<tr><td>TOTAL</td><td></td><td></td><td>599</td><td>182,444</td><td></td></tr>
</table>
Dateline Drilling:
Summary of drilling (includes drilling not in the MRE eg waste dumps, stockpiles and peripheral exploration).
<table>
<tr><th>Year</th><th>No Hole</th><th>Metres</th><th>Drill type</th></tr>
<tr><td>2022</td><td>5</td><td>605</td><td>DD HQ3</td></tr>
<tr><td>2023</td><td>7</td><td>1,653</td><td>DD HQ3</td></tr>
<tr><td>2024</td><td>7</td><td>1,486</td><td>DD-HQ3</td></tr>
</table></td>
</tr>
</table>

Criteria	JORC Code explanation	Commentary
		2025 | 7 | 1175.3 | DD-HQ3 2025 | 36 | 6060.7 | RC The mineralisation is intrusion related but comprises disseminated sulphide mineralisation (mainly pyrite) that is not specific to any host lithology. The geometry of mineralisation is a vertical pipe with no obvious vertical tapering of high grade structural continuity. The preponderance of samples for all drill programs of all operators were taken at 5-foot intervals, which is customary for RC drilling, and is significantly less than the thickness of the bulk-tonnage style of mineralisation at the Colosseum mine. Each drill sample interval is therefore a fraction of the true thickness of the mineralized zones. The predominant sample length for the drill intervals in the Colosseum database is five feet (28,339 samples out of 35,836– 79%) of assays with values, with the remaining percentage of shorter or longer intervals. The difference in length reflects two-foot, and five-foot sample length for reverse circulation holes, twelve-foot sample length for air track holes, and various sample lengths for core holes based on lithology. Historic work programs are described below: **Draco Mines 1972-1974** Draco completed five core holes (CP-1 to 5) totalling 7,065 ft and submitted 654 samples of varying lengths to Cortez Met, Skyline, Rocky Mountain Geochem, and Mineral Assay laboratories for gold and silver fire assays. Multi-element analyses were completed on selected samples. There is no record of the sample preparation procedures used by the assay labs and there is no record of usage of CRMs, BLKs, and DUPs. Drill hole results and supporting assay certificates are available. **Placer Amex – 1975-1976** Placer Amex completed 18 core holes (CP-6 to 23) totalling 8,230 ft and submitted 1,608 five-foot samples to Cortez Met and Mineral Assay laboratories for gold and silver fire assays. There is no record of usage of CRMs, BLKs, and DUPs. Sample submittal sheets with drill hole results and supporting assay certificates are available. **Draco Mines – 1979-1980** Draco completed 26 rotary percussion holes (CH-24 to 52) totalling 10,777 ft and submitted 2,293 five-foot samples to Skyline and Mineral Assay laboratories for gold and silver fire assays. Multi-element analyses were completed on selected samples. There is no record of usage of CRMs, BLKs, and DUPs. Sample submittal sheets with drill hole results and supporting assay certificates are available. **Amselco – 1982 – 1984**

Criteria	JORC Code explanation	Commentary
		Amselco completed two drilling campaigns comprising reverse circulation and core holes. 1982-84 – 163 reverse circulation holes (CM series) totalling 95,436 ft with 22,763 samples submitted to Monitor and Rocky Mountain laboratories for gold fire assays. Multi-element analyses were completed on selected holes by Cone Geochemical and Amselco's own laboratory. QC monitoring comprised 10% control material of known grades, 5% silica sand blanks, and 5% repeat samples inserted with each batch of samples. In addition, 10% duplicate samples, with controls, were shipped to Amselco's own laboratory. Control materials returned most results within + 5% of the known grade with a maximum of + 10%. 1982-84 – 6 core holes totalling 3,738 ft were completed for metallurgical and engineering (Section 13, Mineral Processing). **Colosseum Gold Inc – 1987** Colosseum Gold completed two drilling campaigns comprising core and air track blast holes. 1987 – 2 core holes totalling 2,625 ft with 337 samples submitted to Monitor and Rocky Mountain laboratories for gold fire assays, and copper, zinc, and sulphur analyses. Sample record sheets, and mine assay records are available for these holes, but assay certificates are not. 1987 – 6 percussion (C87-3 to 8) holes totalling 447 ft were completed and 43 samples submitted to Chemex and American Assay for gold fire assays and multi-element analyses. Assay certificates are available for these holes. 1987 – 211 air track blast holes totalling 14,398 ft and 1,236 samples were submitted to Strobeck laboratory for gold and silver fire assays. A check assaying program was completed by Cimetta and Hunter laboratories. Discrepancies were noted for the number of holes drilled and between some assay samples and drill hole identifiers. Sample submittal sheets and assays certificates are available for some samples. **Bond Gold Colosseum Inc – 1988-1991** Bond Gold completed three campaigns of reverse circulation drilling. 1988 – 36 holes (C88 series) totalling 18,555 ft and 3,926 samples submitted to Skyline for gold and silver fire assays. Assay certificates are available. 1989 – 2 deep holes totalling 1,330 ft and 266 samples submitted to American Assay laboratory for gold fire assays, total sulphur, and CN soluble copper and zinc analyses. QC monitoring comprised 10% random

Criteria	JORC Code explanation	Commentary
		duplicate samples. Drill hole results and supporting assay certificates are available. 1990 – 67 holes (R90 and DB90 series) totalling 18,200 ft and 3,113 samples submitted to American Assays Laboratories. QC monitoring comprised 10% random duplicate samples, and selected duplicate samples were submitted to Chemex and Skyline laboratories for check assays. Job order forms and assay certificates are available. **LAC Minerals - 1991** LAC Minerals completed one campaign of reverse circulation drilling. 1991 – 18 holes (SP91 series) totalling 3,200 ft and 640 samples submitted to American Assay Laboratories for gold and silver fire assays. QC monitoring comprised 10% random duplicate samples. Job order forms and assay certificates are available. **Dateline Resources - 2022 to 2026** Dateline Resources Limited acquired Colosseum in 2021 and commenced drilling in 2022. In 2022, Dateline completed 605 metres (1,986 feet) of drilling in 5 drill holes at the Colosseum Project. All the drilling was done from the surface with HQ diamond drill core. In 2023, the Company completed 1,653.1 metres (5,423.9 feet) of drilling in 7 drill holes at the Colosseum Project. All the drilling was done from the surface with HQ diamond drill core. In 2024, Dateline drilled 1,486.5 metres (4,877.2 feet) of drilling in 7 diamond drill holes at Colosseum. All the drilling was done from the surface with HQ diamond drill core. Drilling in 2025 consisted of 7 HQ diamond drill holes for 1,175.3 metres (3,856 feet) and a further 36 reverse circulation (RC) holes for 6,060.7 metres (19,885 feet). All of the drilling above was incorporated into the 2026 Mineral Resource Estimate (MRE). As at the date of this report, the Company had drilled a further 4 diamond drill holes at Colosseum during 2026 for 1,844.7 metres (6,052.5 feet), however the results were received after the cut-off for the MRE and are not included in the updated MRE. For all the Dateline drill programs, the following protocols were observed: Industry standard core handling and sampling procedures were employed to ensure high quality samples. Core samples were collected at 5 foot intervals. RC

Criteria	JORC Code explanation	Commentary
		samples were collected at 5 foot intervals. All core was logged for rock type, RQD, and recovery and dispatched for assay with standard 5 foot long sample intervals. Logging geologist identified zones of interest, but the entire hole was measured and marked up in 5 foot intervals. Whole core was sampled. RC samples were split with a cone splitter at the drill rig and a representative sample dispatched for analysis. Core was bagged into pre-numbered bags, and taken to the FEDEX Freight office in Las Vegas, palletised by the Logging Geologist, covered in shrink wrap and handed over to the FEDEX dock personnel for Paragon Geochemical Laboratory in Sparks Nevada or ALS Global in Reno Nevada. Diamond core and RC samples were sent to ALS Global or Paragon Geochemical in Sparks, Nevada for sample preparation and assaying. Samples were dried, weighed, crushed and split to obtain 1 kg. The split samples were placed in a ring and puck mill to produce 85% minus 75 micron pulp. This material was blended on clean cloth and packaged in paper pulp bags. Using a pulp balance, a 30gm sample was weighted out for standard lead collector fire assay with an AAS finish. Overlimit values using a 5 ppm threshold were analysed via gravimetric analysis. All samples followed a strict Chain of Custody. Routine QAQC samples were inserted in the sample runs at a rate of 20%, comprising Certified Reference Materials from CDN Resource Laboratories Ltd., and verified blank granitic material. Sampling practice is appropriate to the geology and mineralisation of the deposit and complies with industry best practice.
Drilling techniques	*Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	**Historic Data**

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><th>Company</th><th>Date</th><th>Series</th><th># Holes</th><th>Feet</th><th>Type</th></tr>
<tr><td>Draco Mines</td><td>1972-1974</td><td>CP</td><td>5</td><td>7,070</td><td>Core</td></tr>
<tr><td>Placer Amex</td><td>1975-1976</td><td>CP</td><td>18</td><td>8,256</td><td>Core</td></tr>
<tr><td>Draco Mines</td><td>1979-1980</td><td>CH</td><td>27</td><td>11,148</td><td>Rotary/ Percussion</td></tr>
<tr><td>Amselco</td><td>1982-1984</td><td>CM</td><td>162</td><td>95,160</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1983-1984</td><td>EDDH, WDDH</td><td>6</td><td>3,740</td><td>Core</td></tr>
<tr><td>Colosseum Gold Inc</td><td>1987</td><td>C87-1,2</td><td>2</td><td>2,625</td><td>Core</td></tr>
<tr><td></td><td></td><td>C87-3-8</td><td>6</td><td>477</td><td>Rotary/ Percussion</td></tr>
<tr><td></td><td></td><td>ATDH*</td><td>262</td><td>16,948</td><td>Air Trac</td></tr>
<tr><td>Cond Gold Colosseum Inc.</td><td>1988</td><td>C88</td><td>31</td><td>16,415</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1989</td><td>C89</td><td>2</td><td>1,330</td><td>Reverse Circulation</td></tr>
<tr><td></td><td>1990</td><td>R90</td><td>53</td><td>15,265</td><td>Reverse Circulation</td></tr>
<tr><td></td><td></td><td>DB90</td><td>6</td><td>690</td><td>Reverse Circulation</td></tr>
<tr><td>LAC Minerals – Colosseum Inc.</td><td>1991</td><td>SP91</td><td>18</td><td>3,220</td><td>Reverse Circulation</td></tr>
<tr><td>TOTAL</td><td></td><td></td><td>599</td><td>182,444</td><td></td></tr>
</table>
Drilling type details unknown
Dateline Drilling
The drilling program utilized surface core and RC drilling.
The core drilling was conducted with an EVERDIGM ECR 18 drill or Discovery II drill with HQTT core tooling. All holes utilised triple tube to increase recoveries. The drilling was completed by an experienced diamond drilling core driller.
RC drilling, using 6.09-metre tooling, was conducted by experienced drillers with a Schramm T-455WS RC drill using a 5 1/4” face sampling bit and a rig mounted rotary cone splitter.
</td></tr>
<tr><td>Drill sample recovery</td><td>
Method of recording and assessing core and chip sample recoveries and results assessed.
Measures taken to maximise sample recovery and ensure representative nature of the samples.
Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.
</td><td>
Historic Data
Sample recoveries for historic drillholes unknown. Relationship between recovery and grade unknown.
Dateline Drilling
All core drilling recoveries have been logged based on a 3.05-metre tooling. Recovery averaged >95% with most core loss occurring at the top of hole position.
To maximize core sample recoveries, triple tube drilling and long chain polymer muds were used to increase recovery. As recoveries are very high there is no relationship between core recovery and gold grades.
For the RC holes visual observation of the quantity of chip material in the bulk sample bags was reasonably consistent and therefore on a qualitative basis recoveries are deemed good for dry samples.
40% of RC holes encountered the water table with the basal one third of those holes containing wet samples
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Criteria	JORC Code explanation	Commentary
		with an associated sample loss. Diamond hole twinning of 3 of the RC holes indicated a mineral zone match although the diamond holes contained slightly higher grades suggesting some loss of gold with the RC sampling. RC drilling recoveries were calculated using bit size diameter and sample weights. Total sample weights have been recorded for the RC drilling but no analysis of the relationship between sample recovery and gold grade has been completed (it is currently in progress).
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* *The total length and percentage of the relevant intersections logged.*	**Historic Drilling** Core and chip samples were geologically and geotechnically logged at the mine site to a level of detail to support appropriate Mineral Resource estimation (MRE), mining studies and metallurgical studies. Geological data used in the MRE is qualitative, confined to lithology only. **Dateline Drilling** RC and DD samples were geologically logged. Lithology, veining, alteration, mineralisation, and weathering are recorded in the appropriate tables of the drill hole database. The geological logging is both qualitative and quantitative in nature. Each core box was photographed dry and wet, after logging of unit and structures were notated on the core. Each RC chip tray was photographed both wet and dry. Because of the diffuse nature of the mineralisation, the style of mineralisation and the lack of lithology control, the limited use of geology in the MRE is justified.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.* *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* *Whether sample sizes are appropriate to the grain size of the material being sampled.*	**Historic Data** It is not known if whole or sub-sampled core samples were taken. Samples lengths were generally 5'. Up to 1987, samples were shipped by various trucking and courier companies from the project site to laboratories in western United States. In 1987, American Assay Laboratories established an on-site laboratory for mine production samples. Individual laboratory sample preparation procedures varied slightly but still followed a standard analytical industry process of taking submitted samples through successive stages of reducing particle sizes and weights to obtain representative subsamples for assaying. Procedures comprised drying, crushing (jaw or rolls), splitting (riffle), pulverizing (spindle, plate, bowl), splitting (scoops), and fire assaying (30-60g charge using lead collector and AAS finish). There were no unusual or questionable gold assaying methods used. Copies of submittal sheets and assay certificates are available for

Criteria	JORC Code explanation	Commentary
		most of the later drilling. **Dateline Drilling** Core sampling was completed under instruction from Dateline by ALS Laboratories in Reno, Nevada. All drill core samples were cut using a diamond saw along the core long axis, with generally 5' sample lengths or under geological control as decided by Dateline. RC drilling comprised 5' (1.62-metres) intervals with three samples collected: a bulk sample and two smaller samples, via a rig mounted rotary cone splitter. The smaller samples were collected in calico bags with one sample used for the original analysis and the second sample either stored for further use or used as a field duplicate. Each RC sample was placed in a marked up sample bag for interval widths and documented in a sample book, a sample tag with the corresponding sample number was placed in the bag with the other tag stapled to the top of the bag. Sample bags were stapled along the top. RC samples were then placed in heavy-duty sealed polyweave bags for transportation to the Laboratory. All samples were dried, weighed, crushed, and split, with a split pulverized to better than 85% passing 75 microns. Dateline nominated a series of samples for laboratory duplicates (a second pulp sub-sample). A total of 10 samples were collected; the results indicated no issue with the sample preparation. A total of 119 RC field duplicates and 189 core duplicates (approximately on a 1 in 45-50 insertion rate) were sent for analysis. Analytical results indicated reasonably good repeatability for the duplicates with no significant bias. All sample sizes are appropriate to the grain size of the material being tested.
Quality of assay data and laboratory tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	**Historic Data** 1972-1984 samples were sent to reputable laboratories that followed standard analytical procedures and QAQC procedures of the day. Amselco (BHP) 1984-1985 had rigorous security and QAQC standards that exceed current reporting requirements. Fire assays for gold were completed using industry standard fire assay methodology. External standards and blank material were inserted into routine sample stream prior to laboratory submission. 1987 Samples were sent to multiple assay labs for analysis of the same sample. 1987-1991 American Assay Laboratories on-site laboratory analysed the samples. Standards and blanks were inserted at regular intervals.

Criteria	JORC Code explanation	Commentary
		Dateline Drilling Samples were assayed by industry standard methods by ALS Global Laboratories, in Reno, Nevada All samples were analysed by fire assay using a 30g charge for gold and silver values with a gravimetric finish. The analysis is considered a total digest technique. Selected samples were analysed for trace elements using a 4-acid digest with an ICP-MS finish. Commercially available Certified Reference Materials ("CRMs") i.e. standards (389 samples) and blank standards (coarse and pulp material – 374 samples) were added to the sample submission at a 1 in 20 rate. The CRMs were supplied by a commercial organisation CDN Resource Laboratories Ltd. and verified blank granitic material. No significant issues were noted with the standards or the blanks. The field duplicates were used as a set of second laboratory checks (third party umpire checks). The check samples (210 samples) were submitted to Bureau Veritas for fire assay analysis using a 30g charge with an AAS finish. The results indicated no significant bias and confirmed the accuracy of the original ALS analyses. In general the QAQC procedures are to industry standard and show reasonably good accuracy and precision with no obvious biases.
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.* *The use of twinned holes.* *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* *Discuss any adjustment to assay data.*	**Historical Data** There is no evidence of verification of significant intersections in any documentation by independent or alternative company personnel prior to-Dateline. There has been no obvious implementation of a deliberate twin hole programme(s) by previous explorers. However such is the density of drilling that many holes come close to other holes without any major discrepancies being apparent. Computer printouts and assay certificates are available for the CP, CH and CM series holes. The Amselco CM drill hole assays were loaded onto the computer in Denver directly from the Amselco laboratory. Assay data was then broken down into specific drill hole intervals to form a final database. All assay data entered in the computer was subsequently checked against original laboratory submittal sheets to remedy any errors. The completed geological and assay information was combined with drill hole collar and down the hole surveys to form an integrated database (Amselco, 1984). There is no immediate evidence of documentation primary data or data entry procedures with other previous explorers. There are a total of 37,147 assays in the historic database. The data for holes drilled prior to Dateline's work are available as scanned copies of paper files in PDF file format. The data for assays ranges from scans of original

Criteria	JORC Code explanation	Commentary
		assay certificates and submittal forms to scanned printouts from early digital assay databases through to 1985. The computer print-out files were processed using an OCR text recognition system, the results compared against the originals and any errors found corrected. Those results were then checked against the assay certificates and any discrepancies were corrected. Subsequent assays were scanned from assay certificates and verified. The CP considers the scans of original assay certificates to be primary sources, whereas the printouts from an earlier database are secondary sources. **Dateline Drilling** Sampling, documentation, and sample submittal were under the guidance and care of Graham Craig, Senior Geologist with Dateline, GIT (Association of Professional Engineers and Geoscientists of Manitoba). No other company personnel have documented any verification of the drill intercepts. Dateline completed a 3 diamond twin hole programme to validate the RC drilling. The results showed a reasonable match in both mineral interval length and gold grade. There is possibly slight under-reporting of gold with the RC drilling. Drilling, sample, and assay data is currently stored in MX Deposit, a secured data management system through the commercial organization, Seequent. No adjustments have been made to the sampling assay data except for the replacement of below detection limit assays with half below lower detection limit.
Location of data points	*Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* *Specification of the grid system used.* *Quality and adequacy of topographic control.*	**Historic Data** Collar coordinates for historic drill holes were surveyed in their respective local mine grid coordinate system in use at the time of survey. Accuracy is unknown but anticipated to be <+/-2m in X, Y and Z. Original collar survey files were available for most of the drill collars. 1990 computer printouts were found in the Barrick data files that contained the collar coordinate information for the hole series, C87, CH, CM, CP, WDDH, ATDH, C88, EDDH in the Amselco/Bond local mine grid system. The files were processed using an OCR text recognition system, the results compared against the originals and any errors found corrected. Hole Series generated in the Amselco/Bond grid were checked against the corresponding survey files. The remaining collars were entered from the survey files and compared against collar locations on plan maps. Discrepancies were noted in the Collar table. A total of 599 drill holes were entered into the collar table within the Colosseum mine area to be used in the

Criteria	JORC Code explanation	Commentary
		resource estimate. Drill holes for exploration targets were not included in the database. Additionally, 22 holes from the ATDH series assays contained references to drill holes with no known coordinates. The Amselco/Bond local mine grid was rotated 45.6 degrees west from true north. Drill hole traces from the historic database were plotted and compared to plan maps and sections. Azimuth discrepancies were observed in some of the SP91, BD90, ATDH series angled holes when comparing the historic database to the holes plotted in plan or section. Resolution to the difference in Azimuth was noted in the collar table. Downhole deviation surveys for the azimuth and inclination of the CP and CH series holes were taken at 5 foot intervals (instrument unknown). Computer printouts are available for these holes in the Barrick Data files. Drillhole downhole deviation surveys for inclination and azimuth were obtained by Amselco at 200 foot intervals using a single shot Eastman borehole camera. It was not possible to survey certain of the holes where collars collapsed immediately below the casing or where difficult conditions were encountered during drilling. Surveys were completed for 76 of the 163 CM holes and indicated that the holes tended to steepen by 1° per 200 feet while the azimuth showed little variation. These criteria were applied to non-surveyed holes. (Amselco, 1984). Later datasets used for resource estimation or level/cross sections did not include downhole survey information. Subsequent sections showed downhole surveys only for holes CP-1, CP-2, CH-50 and CH-52. Those surveys were included in the data set for the historical data set. The non-surveyed drill-holes were evaluated on section and found to have similar locations for geologic and grade breaks as compared to the surrounding surveyed drill-holes and blast hole assay data, and therefore, are considered suitable for resource estimation. **Dateline Drilling** All drillhole collars were surveyed using a handheld GPS survey equipment. The positions are accurate to within +/-4m for X, Y & Z. The holes were surveyed in UTM WGS84 Zone11N coordinate system. Down hole surveys were done using a Devico or Reflex downhole gyro survey tools on all drill holes. Collar alignment was surveyed using Devico DeviAligner or TN14 Azi tools.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.* *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and*	The spacing and location of data is currently 5-20 metre spacing with downhole sampling nominally at 1.65m intervals. The sample spacing and distribution is sufficient to establish the degree of geological and grade continuity

Criteria	JORC Code explanation	Commentary
	classifications applied. *Whether sample compositing has been applied.*	appropriate for the Mineral Resource and classifications applied. No sample compositing was undertaken by Dateline. There is no evidence of any sample compositing with the historical drilling.
Orientation of data in relation to geological structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	Due to a perceived lack of definition of structural controls drillholes were drilled at a range of azimuths and dips from vertical to -60°. It is anticipated that there is no bias with the sampling. Despite the predominance of vertical and sub-vertical drilling in line with the pipe geometries and the density of drilling sample orientation is deemed to be representative for reporting purposes. No obvious bias is considered to have been introduced by the existing sampling orientation.
Sample security	*The measures taken to ensure sample security.*	**Historic Data** Sampling techniques were developed and reviewed by mine site personnel but there is no evidence of the procedures used. **Dateline Drilling** All samples were taken and maintained under the constant care of Colosseum Rare Metals, Inc. personnel. Samples were delivered to laboratories by a licensed transportation company.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	**Historic Data** No documents have been viewed regarding any sampling audits, and it is uncertain if any exist. **Dateline Drilling** Drillhole sampling techniques and QAQC procedures have been developed and reviewed by Dale Sketchley, M.Sc., P. Geo. of Acuity Geoscience Ltd., and Graham Craig, GIT.

Section 2 Reporting of Exploration Results

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

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	*Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	The Colosseum Mine project is located in T17N R13E Sec 10, 11, 14, 15, 22, 23 SB&M. All tenements are 100% owned by Dateline Resources Limited or a wholly owned subsidiary and there exists a 2.5% Net Smelter Royalty as previously disclosed to ASX.
Exploration done by other parties	*Acknowledgment and appraisal of exploration by other parties.*	Historical work was completed by various mining companies since 1972. • Draco Mines (1972-1974) • Placer Amex (1975-1976) • Draco Mines (1980) • Amselco (1982-1984 • Dallhold Resources/Bond Gold (1986-1989 • Lac Minerals (1989-1994) All the companies were reputable, well-known mining/exploration companies that followed the accepted industry standard protocols of the time. All historical data has undergone independent review and verification and signed off as accurate.
Geology	*Deposit type, geological setting and style of mineralisation.*	The Colosseum deposit can be classed as an Intrusion-related Gold (IRG) deposit. The Colosseum mine is hosted by a Cretaceous aged breccia-pipe associated with a felsic intrusive. The pipe contains aphanitic Cretaceous rhyolite flows, Pre-Cambrian granitic basement material, and Cambrian-Devonian dolomite clasts replaced by sulphide mineralisation. The gold mineralisation occurs mainly in brecciated felsite associated with the boundaries of pyrite grains and with sediment clasts replaced by sulphides. The mineralisation is not refractory.
Drill hole Information	*A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* *easting and northing of the drill hole collar* *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* *dip and azimuth of the hole*	No exploration results are being reported in this release. Previous drillholes have been periodically released to the ASX since 2022 on the following dates: • June 6, 2022 • June 19, 2023 • July 20, 2023 • February 13, 2024 • April 2, 2024

Criteria	JORC Code explanation	Commentary
	down hole length and interception depth *hole length.* *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	• May 16, 2024 • August 27, 2024 • November 28, 2025 • December 3, 2025 • December 15, 2025 • January 12, 2026 • February 12, 2026
Data aggregation methods	*In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	Drillhole intersections are reported above a lower exploration cut-off grade of 0.1 g/t Au and no upper cut off grade has been applied. Intercept lengths are calculated to include no more than 3 samples less than 0.1 g/t Au consecutively. No Exploration Results are reported in this release. No metal equivalents are being reported.
Relationship between mineralisation widths and intercept lengths	*These relationships are particularly important in the reporting of Exploration Results.* *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	Drillhole orientations vary throughout the program. Interception angles of the mineralised structures are estimated using core drilling intercepts and existing 3D models of the pipe orientation.
Diagrams	*Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Supporting figures have been included within the body of this release. Other figures have been periodically been released since 2022.
Balanced reporting	*Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Representative reporting of both low and high grades and/or widths have been reported. No exploration results are reported in this release.
Other substantive exploration data	*Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	All meaningful and material data has been included in a previous report; including geotechnical mapping, downhole televiewer, and multiple geophysical surveys and interpretations. 3D geophysical interpretations have recently been created from historical data. The outcomes have suggested possible additional exploration targets close to the existing set of deposits.
Further work	*The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* *Diagrams clearly highlighting the areas of possible*	Continued drilling within and outside of the existing pits is planned to expand upon the understanding of the extents of mineralisation. Interpreting and analyses of geophysical data is ongoing as new drilling information is collected. Further drilling exploration is planned to

Criteria	JORC Code explanation	Commentary
	extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	test deep gravity, IP, and MT anomalies identified by the geophysical data.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	*Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* *Data validation procedures used.*	The original Dateline drill-hole databases were directly created by GeoGRAFX (a commercial organization based in Tucson, Arizona), using original digital analytical certificates in the case of the assay tables, drill log lithologies, and checking against original digital records in the case of the collar and down-hole deviation tables. Working copies of collar coordinates, downhole survey information, assays and lithology were converted into excel templates for data verification. These templates contain data checking routines designed to prevent common data entry errors. This original mine-site drill-hole information was then subjected to various verification measures, the primary one consisting of auditing of the digital data by comparing the drill-hole collar coordinates, hole orientations, and analytical information in the database against historical paper records in the Barrick data set. Verified data was loaded into a project specific MSAccess database. This database is secure, operated by a single database administrator. The recent drilling data was supplied by Dateline to HSC as an MSAccess database (2022) plus the most recent 2023-5 drilling as CSV files. This data was re-imported into an HSC 'Mineral Resource' MSAccess database to allow for some error checking. HSC completed some independent validation of the new data to ensure the drill hole database is internally consistent. The minimum and maximum values of assays and density measurements were checked to ensure values are within expected ranges. Further checks include testing for duplicate samples and overlapping sampling or logging intervals. Dateline takes responsibility for the accuracy and reliability of the data used in the Mineral Resource estimates. HSC used the national grid system converted from the local imperial grid for all interpretation and modelling work.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why*	Database Manager Barbara Carroll (CPG) conducted a field examination of the project area on April 4, 2022, and met with consulting geologist Chris Osterman PhD.

Criteria	JORC Code explanation	Commentary
	this is the case.	The visit included field review of the property geology, current drilling, core logging and handling, confirmation of the location of a number of the historic drill holes and collection of representative core samples to verify assays results from current drilling. No site visit was completed by HSC due to US immigration constraints.
Geological interpretation	*Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* *Nature of the data used and of any assumptions made.* *The effect, if any, of alternative interpretations on Mineral Resource estimation.* *The use of geology in guiding and controlling Mineral Resource estimation.* *The factors affecting continuity both of grade and geology.*	The gold mineralisation comprises disseminated auriferous pyrite hosted by a combination of felsite dyke intrusion, felsite breccias, sedimentary breccias and altered granite. Mineralisation is diffuse and not hosted exclusively by a particular rock type. There is no obvious visible lithological or structural control to the gold mineralisation, save for a broad NE/SW-striking enriched zone, presumably a structural corridor related to the felsite intrusions. No geological interpretation per se for the mineralisation has been completed as the gold grades define the gold mineralisation in the various host rocks. Any wireframe for the gold mineralisation would ultimately be a simple grade shell. Broad based lithological units were delineated for the felsite/felsite breccia, sedimentary breccia and granite. Their use is very limited. There is insufficient data to define with confidence any specific or significant fault structure playing a role in the control of mineralisation. No oxidation surface was created due to a lack of logging data and the fact that the upper portions of the deposit have been mined.
Dimensions	*The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	The Mineral Resources have an 800m by 800m surface extent with two separate bodies the south pit (domain 1) and the north pit (domain 2) each measuring 200x200m The mineralisation is exposed at surface, and the Mineral Resources continue to a depth of approximately 300m below surface at an RL of 1410m. The lower limit to the Mineral Resource is an arbitrary one being the result of a supplied pit shell from a cursory pit optimisation study. The mineralisation is open at depth, and laterally to the southeast, beyond the North Pit zone.
Estimation and modelling techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters*	Recoverable Multiple Indicator Kriging (MIK) with two search domains was used to complete the gold grade estimation using the GS3M modelling software. The geological interpretation such as it is and block model creation and validation were completed using the Surpac mining software. HSC considers recoverable MIK to be an appropriate estimation technique for the

Criteria	JORC Code explanation	Commentary
	used. *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* *The assumptions made regarding recovery of by-products.* *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* *Any assumptions behind modelling of selective mining units.* *Any assumptions about correlation between variables.* *Description of how the geological interpretation was used to control the resource estimates.* *Discussion of basis for using or not using grade cutting or capping.* *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	type of mineralisation and extent of data available. The drillhole database was composited with no constraints to 1m intervals covering the whole of the prospect. A minor amount of peripheral, isolated data was removed from the composite file. Two drilling domains were employed, one for the South Pit (domain 1) and another for the North Pit (domain 2), reflecting a difference in intensity of drilling and assay grades. A total of 62,920 composites were generated from the drillhole database, using the Surpac 'best fit' option and modelled for gold only. 36,589 data points for domain 1 (coefficient of variation = 8.8) and 26,331 data points for domain 2 (coefficient of variation = 3). Grade interpolation was unconstrained, except by the search parameters and the variography, in acknowledgement of the gradational nature to the margins of the gold mineralisation and the abundance of buffering low grade, peripheral assays. No base of oxidation was used. No cover surface was created as the mineralisation is outcropping and is exposed in many places along its ridge line and flanks and where previous mining had occurred. A fundamental concept behind MIK is that it precludes the need for top cutting. However a review of the conditional statistics for the top indicator class for both domains highlighted a significant difference between the mean and the median. As a result an average value for the mean and median value was used for the top indicator class for both domains. Block dimensions are 10m by 10m by 5m (E, N, RL respectively) with no sub-blocking. The selective mining unit (SMU) is 5m by 5m by 2.5m. The north and east dimensions were chosen as they are a half to a third of the nominal drillhole distances in the detailed drilled area of the South Pit. The vertical dimension was chosen to reflect the sample spacing and possible mining bench heights and to allow for flexibility in potential mining scenarios. Both domains were modelled as a combined dataset with soft boundaries. 5 search passes were employed with progressively larger radii and/or decreasing data point criteria. Details of search passes are:

Dom 1	**X (m)**	**Y (m)**	**Z (m)**
Pass 1	20	20	35
Pass 2	30	30	60
Pass 3	40	40	70
Pass 4	60	60	120
Pass 5	60	60	120
Dom 2	**Dom 1**	**X (m)**	**Y (m)**

Criteria	JORC Code explanation	Commentary

Pass 1	25	25	25
Pass 2	35	35	35
Pass 3	50	50	50
Pass 4	70	70	70
Pass 5	70	70	70

Dom 1	Min Data	Max Data	Min Octants
Pass 1	16	48	4
Pass 2	16	48	4
Pass 3	16	48	4
Pass 4	16	48	4
Pass 5	8	48	2
Dom 2	**Min Data**	**Max Data**	**Min Octants**
Pass 1	16	48	4
Pass 2	16	48	4
Pass 3	16	48	4
Pass 4	16	48	4
Pass 5	8	48	2

There is no maximum extrapolation as both domains are surrounded by low grades that buffer any higher grade extrapolation. In any areas where there are no buffer low grades the maximum extrapolation would be the same as the Pass 5 search ellipse.

No other elements were modelled therefore there are no assumptions about correlation between variables. The resource estimates are controlled by the data point distribution, the variography, block size and the search ellipse. Conventional use of wireframes to control the mineralisation was not considered necessary.

The new block model was reviewed visually by HSC, and it was concluded that the block model fairly represents the grades observed in the drill holes. HSC also validated the block model using a variety of summary statistics and statistical plots. No issues were noted.

Comparison with the 2024 Mineral Resources shows a minor increase of 1.9% in tonnes for Measured and Indicated with a slightly lower gold grade (by 6%) leading to a minor reduction in contained gold ounces of 4.9%. The amount of Measured Resource has increased by 14% as a result of the new infill drilling. There has been a significant reduction in the amount of Inferred Resource as a result of the infill drilling on the margins of both domains. The difference, particularly for the Inferred Resource, is also due to a change in the reporting constraint which has become a pit shell (for 2026) rather than a fixed elevation (for 2024).

Criteria	JORC Code explanation	Commentary
		No reconciliation is possible because no mining cut off grades are available and low-grade stockpiles have no assays.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages are estimated on a dry basis. Moisture not recorded.
Cut-off parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied.*	The recoverable resources are reported at a gold cut-off of 0.2g/t based on the outcome of a recently completed pit optimisation study by independent mining consultants AMDAD of Brisbane. The cut-off grade at which the resource is quoted reflects the intended bulk-mining approach. Consideration of "reasonable prospects of eventual economic extraction" has utilised a pit shell (pit shell 48) with a revenue factor of 1 at a US$5000/oz gold price with estimates of mining costs and pit wall slopes. Shell 48 as attached which is the revenue factor 1 (highest undiscounted value) shell run at US$5000/oz
Mining factors or assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	The Mineral Resources were estimated on the assumption that the material is to be mined by open pit using a bulk mining method. The proposed mining method is a conventional drill & blast, truck & excavator with extracted material sent to an on-site ROM pad with a processing plant adjacent to the planned pit. Minimum mining dimensions are envisioned to be around 5m by 5m by 2.5m (strike, across strike, vertical respectively). Internal Dilution has been incorporated as part of the MIK modelling, but there is no allowance for external dilution and mining losses.
Metallurgical factors or assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	The operation of the grinding mill (cyanide leach with carbon in pulp recovery) in the January 1988 through June 1993 period conclusively demonstrated the feasibility of gold recovery from the Colosseum ore. Process recoveries during operations were reported to be around 92%. For the current project a standard CIL plant is envisaged for the ore processing, similar to the process used for the previous mining.
Environmen-tal factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields*	A 2022 NI43-101 report stated: "There are no known environmental liabilities that are adversely impacting air, water or soil resources on the Colosseum Mine project." The current tenement status over the project area permits the resumption of open cut mining and ore processing.

Criteria	JORC Code explanation	Commentary
	project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	Future mining operations can be contained within the unpatented mine leases. There are no reports of mine drainage for the stockpiles or the waste dumps. All waste and process residues will be disposed of in a responsible manner and in accordance with the mining license conditions. The area comprises modestly rugged terrain with alluvial fans, basalt flows, hills, and low mountains and is generally sparsely vegetated. The climate is typical of a high desert environment with high temperatures in excess of 100°F during the summer and low temperatures slightly below freezing in the winter. Annual precipitation is approximately 8 inches.
Bulk density	*Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	A default density of 2.65t/m^3 was used for the Mineral Resources. No historical density data was supplied. 121 density measurements from Dateline's recent drilling were supplied. Sample density was measured by using the weight in air / (weight in air minus weight in water) method (Archimedes Principle) on single pieces of core. The average value was 2.66t/m^3 with a range of 2.1 to 4/m^3. Density values tended to show an increase with depth depending on sulphide content. The default density value used in the resource estimates is considered reasonable and is based on the supplied limited data and the Competent Person's experience with similar types of deposits.
Classification	*The basis for the classification of the Mineral Resources into varying confidence categories.* *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* *Whether the result appropriately reflects the Competent Person's view of the deposit.*	The classification of the Mineral Resources is based on the data point distribution which is a function of the drillhole spacing and the search ellipse parameters. Pass 1 = Measured, Passes 2&3 = Indicated, Passes 4 & 5 =Inferred. Other aspects have been considered in the classification including, the style of mineralisation, the geological model, validation of the historic drilling, sampling methods and recoveries, the QAQC programmes and results and comparison with previous resource estimates. HSC believes the confidence in tonnage and grade estimates, the continuity of geology and grade, and the distribution of the data reflect Measured, Indicated and Inferred categorisation. The estimates appropriately reflect the Competent Person's view of the deposit.
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*	No audits or reviews have been completed.

Criteria	JORC Code explanation	Commentary
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	No statistical or geostatistical procedures were used to quantify the relative accuracy of the resource. The global Mineral Resource estimates of the Colosseum gold deposit are moderately sensitive to lower cut-off grades. The relative accuracy and confidence level in the Mineral Resource estimates are considered to be in line with the generally accepted accuracy and confidence of the nominated Mineral Resource categories. This has been determined on a qualitative, rather than quantitative, basis, and is based on the Competent Person's experience with similar deposits and geology. The Mineral Resource estimates are considered to be accurate globally, but there is some uncertainty in the local estimates due to a lack of geological definition in certain places eg fault zones. Mining of the deposit has taken place, but production data is unsuitable for comparison and/or reconciliation.

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td>Mineral Resource estimate for conversion to Ore Reserves</td><td>Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.

Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.</td><td>The Mineral Resource Estimate was produced in March 2026 as part of the Definitive Feasibility Study for the Colosseum Project. The Mineral Resource block model and Estimate were prepared by Mr Simon Tear of H & S Consultants Pty Ltd (HSC).

The Mineral Resources are inclusive of the Ore Reserves.</td></tr>
<tr><td>Site visits</td><td>Comment on any site visits undertaken by the Competent Person and the outcome of those visits.

If no site visits have been undertaken indicate why this is the case.</td><td>The Competent Person for the Ore Reserves Estimate is Mr John Wyche of Australian Mine Design and Development Pty Ltd (AMDAD).

Mr Wyche visited the Colosseum mine site from 9^{th} to 12^{th} December 2025. The visit included inspections of the:
Site access,
Water supply,
Existing South and North Pits,
Waste rock and tailings storage sites,
Process, ROM and stockpile areas, and
Overall site drainage.
Discussions were also held on site with potential mining contractors.</td></tr>
<tr><td>Study status</td><td>The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.

The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.</td><td>The Ore Reserve is underpinned by studies conducted to a BFS level with sufficient analyses, designs and estimation to support capital and operating costs at AACE Class 3 confidence.

Modifying factors accurate to the study level were applied based on detailed expert design analyses. The study demonstrates that the Ore Reserve and mine plan are technically achievable and economically viable.</td></tr>
<tr><td>Cut-off parameters</td><td>The basis of the cut-off grade(s) or quality parameters applied.</td><td>Cut-off grade parameters for determining open pit Ore Reserves:

<table>
<tr><td>Gold Price</td><td>US$/oz</td><td>4,200</td></tr>
<tr><td>Government Royalty</td><td>%</td><td>nil</td></tr>
<tr><td>Dore transport/Insurance/Refining</td><td>US$/oz</td><td>50.00</td></tr>
<tr><td>Vendor Royalty</td><td>%</td><td>2.5% NSR</td></tr>
<tr><td>Metallurgical Recovery</td><td>%</td><td>91%</td></tr>
<tr><td>Mining Incremental Ore Cost</td><td>US$/t ore</td><td>0.28</td></tr>
<tr><td>Processing Costs</td><td>US$/t ore</td><td>23.86</td></tr>
<tr><td>G&A</td><td>$/t ore</td><td>3.80</td></tr>
<tr><td>Calculated Cut-off Grade</td><td>g/t Au</td><td>0.24</td></tr>
<tr><td>Applied Cut-off Grade</td><td>g/t Au</td><td>0.25</td></tr>
</table></td></tr>
<tr><td>Mining factors or assumptions</td><td>The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by</td><td>Scoping studies in 2024 and 2025 comparing open pit and underground mining recommended open pit mining using hydraulic excavators and off-highway dump trucks. The analyses used Whittle™ pit optimization and the Datamine™ Mineable Shape</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	preliminary or detailed design). *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* *The mining dilution factors used.* *The mining recovery factors used.* *Any minimum mining widths used.* *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* *The infrastructure requirements of the selected mining methods.*	Optimiser (MSO). MSO identified areas with underground potential but pit optimization defined a much larger open pit target with more recoverable gold at lower cut offs and much higher value. The pit optimization defined two adjacent pits which push back and deepen the existing South and North Pits. The two new expanded pits eventually merge. Inputs for the pit optimization include: • Gold price of US$4,200/oz with current realization costs, • Process gold recovery of 91%, • Measured and Indicated resources only, • Mining loss and dilution modelled to reflect orebody geometry and mining method, • 2 Mtpa process feed rate, • Operating cost estimates drawn from supplier pricing and detailed first principles cost estimates, and • Physical parameters such as slope stability based on analyses conducted for the BFS. A mining loss and dilution procedure was run on the MIK resource model. It applied a dilution skin to each MIK grade increment. This “onion skin” approach allows the dilution to be applied to whichever MIK grade increment the cut-off grade falls in. It includes both the diluted grade and diluted proportion of the MIK panel at the cut-off grade. An additional 1% dilution at zero grade and 2% ore loss is applied to the diluted blocks to allow for operational inefficiencies. This approach matches mining loss and dilution to the orebody geometry and gold distribution. The net effect of the dilution model is similar to applying global factors of 5.8% dilution at zero grade and 2.5% mining loss to the in-situ resource blocks. The pit optimization initially defined larger pits than the final ore reserve. However, the volume of waste rock and tailings generated were too large to fit within the permitted claims area. In order to target a high value pit small enough to meet the available waste storage volume the pit optimization was re-run using an elevated cut-off grade of 0.4 g/t Au. All ore down to the 0.25g/t Au cut-off is included in the ore reserve and BFS production schedule but the waste rock and tailings are reduced to manageable volumes and the pits focus on the highest value portions of the deposit. Practical pits were designed based on the optimized pit shell with pit wall batter and berm configurations defined by geotechnical analyses run as part of the BFS. Mining benches are 20 feet high with three benches between berms. The berms are 25 feet wide. Overall slopes are varied in each geotechnical zone by varying

Criteria	JORC Code explanation	Commentary
		the batter angle. Pit ramps were designed at 10% grades with running widths 3.5 times the planned truck width plus a safety bund half the wheel height of the truck on the open side and a drain on the wall side. Overall ramp width is 60 feet. The new North Pit is mined in two stages. The new South Pit is a single stage pushback of the existing deeper pit. Bench widths between the two North Pit stages and the on the South Pit push back are generally 60 to 100 metres. One short section in the upper part of the North Pit has bench widths around 35 metres. Mining is planned using a fleet of up to three 120 tonne class hydraulic excavators loading 55 tonne payload articulated dump trucks. Analysis of sinking rates (<1 x 20ft bench per month per stage) and working area (30 to 40 m^2 per excavator) confirm the practicality of the fleet against the schedule. Apart from small volumes of mine waste rock left around the new pits from the previous operation, all material mined is competent rock which will be drilled and blasted. Hydrogeological analysis of the ground around the pits conducted for the BFS confirms that ground water will not be a problem for mining in terms of ingress or slope stability. Mining costs include grade control drilling and sampling over 125% of the area of expected ore on each mining bench. Mining will be conducted by a US based mining contractor. Proposals have been received from several contractors who will establish all infrastructure required to support their fleets and blasting requirements. A significant difference from the former operation is that mine waste rock will be placed in a shaped waste rock dump running south from the South Pit across the existing tailings dam. Much of the waste rock in the previous operation was end dumped over the slopes west of the pits and remains at angle of repose. Final faces of new waste rock dump will be formed much flatter slopes and will include berms to direct surface drainage, minimize erosion and promote revegetation. Progressive formation of the final landform over the mine life will achieve the commitments set out in the approved reclamation plan. The mine plan is defined and scheduled entirely on Measured and Indicated resources. Inferred resources are treated as waste rock, although potential remains for some or all of this Inferred to be upgraded to ore in the grade control drilling.

Criteria	JORC Code explanation	Commentary
		Adoption of a smaller pit leaves some of the underground targets defined by MSO as possible future ore, particularly east and north east of the North Pit. They are not included in the mine plan or Ore Reserves at this time.
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* *Whether the metallurgical process is well-tested technology or novel in nature.* *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.* *Any assumptions or allowances made for deleterious elements.* *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.* *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	Ore will be processed through a 2 million metric tonne per year carbon in leach (CIL) plant. This metallurgical process is well-tested and widely adopted. Process performance is based on extensive metallurgical test work conducted for the BFS and six years of operational data. The BFS uses a fixed process gold recovery of 91%. This is based on BFS test work to optimize the grind size and leaching conditions. The test work included variability testing across ore lithologies and head grades with greater than 90% recoveries at head grades down to 0.5 g/t Au. No clear evidence of a fixed tail grade was observed so the fixed recovery of 91% was adopted. This compares with an average recovery of 92% achieved from 1987 to 1993. Resource drilling and metallurgical test work since 2022 has not identified and material changes to the ore processed in the previous operation. There is a good understanding of how comminution and leaching will work. As with the previous project, no issues are expected with deleterious materials in the tailings. A significant change from the previous project is that in place of a slurry tailings dam at 55% to 60% solids the tailings will be dewatered to between 15% and 20% moisture using horizontal belt vacuum filters. The dewatered tailings will be trucked to a tailings cell within the waste rock dump. They will be fully contained within broad walls of waste rock and eventually fully encapsulated. Geotechnical analysis of the tails cell within the waste rock dump was conducted for the BFS to confirm the stability of the system where it is placed on natural surface and where it buries the existing tailings. Dry stacking the tailings within the waste rock dump produces a more stable landform and uses less area and much less water than conventional slurry tailings storage.
Environmen-tal	*The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	Mining and processing at Colosseum is still subject to and will be run in compliance with the existing Environmental Impact Statement (1985) and approved Reclamation Plan and Plan of Operations (1984). Geochemical testing of waste rock in the 1980s and 1994 confirm that the mine waste rock is not acid generating. This is supported by monitoring of bores adjacent to the property which have shown no reduction to the slightly alkaline pH over 30 years. The mine plan includes a major change to environmental management to align the project with

Criteria	JORC Code explanation	Commentary
		current legislation and world best practice. Drainage across the entire site will be managed to create a closed loop system with zero discharge of any water which has come into contact with mining or processing activities. The system comprises four main elements: • A cut off drain placed south of the waste rock dump will intercept rainfall run off from areas in the catchment south of the mine and direct it west into the Shadow Valley. This re-establishes pre-1987 water flows for more than half of the catchment area. • Lined retention ponds will be formed east and south of the waste rock dump and in a valley west of the waste rock dump where the spillway from the current tailings dam discharges. The position and size of these ponds are designed to hold rainfall run off over the entire mine and process plant site in a 1 in 100 year 24 hour rainfall event. • Final faces of the waste rock dump are designed to direct rainfall run off to the retention ponds. The final face configurations will be formed as the waste rock is placed so the drainage will work throughout and beyond the mine life. All other areas of the mine and process plant drain naturally to the east and south retention ponds. • The entire area under the waste rock dump, tailings cell within the waste rock dump and a long term low grade stockpile will be lined with an impermeable membrane. All hydrogeological analyses since the 1980s, including work done for the BFS, show the granite/gneiss rock at Colosseum has extremely low permeability with the only potential for groundwater movement being through isolated fractures. Nonetheless, installation of the liners prevents any migration of water from mine affected areas into potential groundwater below or adjacent to the project. Any water which may percolate through the waste rock or tailings will run along the liners into the retention ponds where it will be collected and used as process water.
Infrastructure	*The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities),*	The Colosseum Mine is located 10km from Interstate I-15 and has good road access.

Criteria	JORC Code explanation	Commentary
	labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	The project is located 55 miles from Las Vegas, Nevada with major airport, hospital, accommodation and residential facilities. Personnel will live in Las Vegas and operate on a drive in- drive out basis daily. There will be no accommodation on site. A 34.5kV powerline will be connected to the grid in the Ivanpah Valley. The powerline will run adjacent to the access road for approximately 10km to the mine site. Two water bores that previously serviced the operation have been acquired and reactivated. Pump testing has confirmed they can supply all water requirements for the project with minimal impact on eth very large Ivanpah Valley aquifer. Dateline intends to construct the process plant and associated infrastructure on the same location as that used in the previous mining operation.
Costs	*The derivation of, or assumptions made, regarding projected capital costs in the study.* *The methodology used to estimate operating costs.* *Allowances made for the content of deleterious elements.* *The source of exchange rates used in the study.* *Derivation of transportation charges.* *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* *The allowances made for royalties payable, both Government and private.*	Capital and operating costs for the mine, process and infrastructure are estimated at AACE Class 3 confidence (approximately ±15%). They are based on supplier pricing and detailed first principals cost estimates using local US inputs. They include direct and indirect costs, delivery charges and contingency applied on an item by item basis depending on the maturity of the estimate. Gold transport, insurance and refining charges are based on typical contracts in the USA during 2025. A third party royalty of 2.5% NSR is included in the cut off grade and financial models.
Revenue factors	*The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	The base gold price for the Ore Reserve Estimate is US$4,200 per troy ounce. This is approximately a 15% discount to the first quarter average spot price for 2026.
Market assessment	*The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* *A customer and competitor analysis along with the identification of likely market windows for the product.* *Price and volume forecasts and the basis for these forecasts.* *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply*	There is a well-established and transparent spot market for gold. Not applicable.

Criteria	JORC Code explanation	Commentary
	contract.	
Economic	*The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	Operating and capital cost estimates are considered to be accurate within ±15%. Cost estimates are drawn from supplier pricing and detailed first principals cost estimates. A discount rate of 5% has been applied. This analysis shows that while sensitive to fluctuations in both operating cost and gold price, the Project continues to deliver positive NPV under conservative assumptions.
Social	*The status of agreements with key stakeholders and matters leading to social licence to operate.*	Dateline works co-operatively with the local rancher and neighbouring power utilities to maintain safe access, respect and co-ordination of activities. Dateline has developed and maintained a good relationship with the San Bernardino County, hosting site visits and featuring in county videos on the importance of mining in the county. The Company has received positive correspondence from the US Forestry Service and Bureau of Land Management that both confirm the Company's approved Plan of Operation and valid Mining Rights. The US Department of the Interior is a vocal supporter of the Colosseum development, led by the Secretary of the Department.
Other	*To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* *Any identified material naturally occurring risks.* *The status of material legal agreements and marketing arrangements.* *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	No naturally occurring risks with sufficient likelihood and severity to impact classification of the Ore Reserves have been identified. All material legal agreements and approvals are either in place or the Company is confident, based on information available, that they will be in place in a suitable timeframe to execute the Project.
Classification	*The basis for the classification of the Ore Reserves into varying confidence categories.* *Whether the result appropriately reflects the Competent Person's view of the deposit.* *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	The Proved Ore Reserve is derived from the portion of the Measured Mineral Resource and the Probable Ore Reserve is derived from the portion of the Indicated Mineral Resource within the mine design that may be economically extracted and which include modelled allowances for dilution and ore loss. The project is a brownfields operation last operated from 1987 to 1993. DTR has conducted extensive drilling since 2022 to confirm and extend the Mineral

Criteria	JORC Code explanation	Commentary
		Resource. The BFS includes geotechnical, metallurgical and other test work to confirm and improve upon the previous operation. Capital and operating costs have been estimated to AACE Class 3 confidence. All material legal agreements and approvals are either in place or the Company is confident, based on information available, that they will be in place in a suitable timeframe to execute the Project. For these reasons the Competent Person for the Ore Reserves considers the modifying factors to be defined a level of confidence commensurate with Proved and Probable Ore Reserves.
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*	Only internal audits of the Ore Reserves have been undertaken.
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	The design, schedule, and financial evaluation on which the Ore Reserve is based is to a BFS standard, with a corresponding level of confidence. The Ore Reserve is estimated as a global estimate. The nature of the orebody, the continuity shown in the resource block model and anecdotal experience from the previous mining operation suggest that production should reconcile well with the diluted mine model. However, it is likely to take six to 12 months of mining before reconciliation and adjustment of mining practices provide enough information to know if the Ore Reserve will become a good local estimator of tonnes and grades. In the opinion of the Competent Person, cost assumptions and modifying factors applied in the process of estimating Ore Reserves are reasonable. Gold price and exchange rates are subject to market forces and present an area of uncertainty.