ASX Announcement

23 October 2024

DATELINE RESOURCES

Colosseum Scoping Study Delivers Positive Outcomes

Approved progression to Project Selection Stage ahead of BFS

Dateline Resources Limited (**ASX:DTR**)(**Dateline, DTR** or **the Company**) is pleased to announce the release of the Scoping Study (**Study**) for its flagship, 100% owned Colosseum Gold Project (**Project**), based on the June 2024 Mineral Resource Estimate (**MRE**), located in San Bernardino County, California, US. The Dateline Board has endorsed this Study and has approved progression to the Project Selection Stage (**PSS**) ahead of a Bankable Feasibility Study (**BFS**).

The Study confirms Colosseum as a robust gold mine development that delivers value for shareholders. Recent drilling at the North Pipe and depth extensions outside the MRE at the South Pipe offer opportunities for potential extensions to the mine life, based on infill and extensional drilling to convert into mineral resources.

Highlights

- Two development and production cases were assessed:
    - **Case 1** assessed an underground and open pit operation, with sub level caving followed by open pit mining at a rate of 1Mtpa
    - **Case 2** focused only on open pit mining methods, with a processing rate of 2Mtpa
- Development of an open pit only mining operation is preferred (Case 2)
- Measured and Indicated mineral resources account for 81% of plant feed in Case 2
- Scoping Study indicates Case 2 production at 75koz per annum for 8.4 years

Dateline's Managing Director, Stephen Baghdadi, commented:

"The Scoping Study indicates robust project outcomes using a US$2,200 per ounce gold price. The production and development cases assessed were benchmarked against 16 projects with sufficient published information on capital costs and 38 projects for operating costs.

"Although two cases were assessed, the Company is most likely to progress Case 2 to BFS and has commenced discussions with suitable groups in the USA that are capable of completing the BFS.

"The Company will explore ways to further enhance the project economics and capital expenditure during the BFS stage"

Contact
Level 29, 2 Chifley Square
Sydney, NSW, 2000
T +61 2 9375 2353
E info@datelineresources.com.au
W www.datelineresources.com.au

Capital Structure (ASX: DTR)

Shares on Issue	1.45B
Top 20 Shareholders	63.8%
Board & Management	33.9%

Board of Directors

Mark Johnson AO
Non-Executive Chairman

Stephen Baghdadi
Managing Director

Greg Hall
Non-Executive Director

Tony Ferguson
Non-Executive Director

Bill Lannen
Non-Executive Director

Colosseum Gold-REE Project*
(100% DTR, California, USA)
27.1Mt @ 1.26g/t Au for 1.1Moz Au
Over 67% in Measured & Indicated
Mineralisation open at depth
Mining studies underway
Rare earths potential with geology similar to nearby Mountain Pass mine
* ASX announcement 6 June 2024

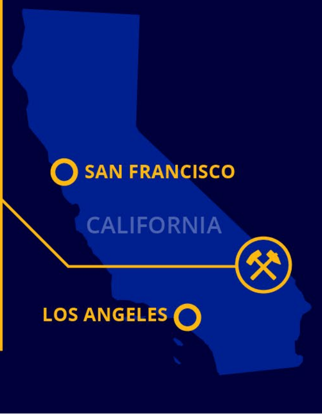

Table 1: Colosseum Gold Mine Scoping Study – Estimates of Inputs and Outcomes.

***Note:** The following table should be read in conjunction with the cautionary statement below*

PARAMETER	UNIT	PROJECT TOTAL*
Gold Price	US$/oz	2,200
Discount Rate	%	6.5
PRODUCTION TARGET		
Life of Mine	Months	100
Total Ore Mined	M Tonnes	16.6
Total Waste Mined	M Tonnes	56.8
Total Material Movement	M Tonnes	73.3
Strip Ratio	x:x	3.4:1
Total Tonnes Milled	M Tonnes	16.6
Average Plant Throughput	Mtpa	1.8
Average Head Grade	g/t Au	1.3
Average Recovery	%	92
Total Net Gold Produced	koz	635
Ave Annual Gold Production	Koz pa	71
FINANCIALS		
Total Operating Costs	US$M	751
Total Capital Costs	US$M	195
Pre-production Capex	US$M	138
Total pre-production capital expenditure and working capital requirements	US$M	153
Total Net Revenue	US$M	398
Total Sales Revenue	US$M	1,344
Discounted Cashflow (@6.5%) - $NPV_{6.5}$	US$M	235
Internal Rate of Return	%	31
UNIT COSTS		
Unit Operating Costs (C1)	US$/oz milled	1,182
All In Sustaining Costs	US$/oz	1,490

*** The results presented in Table 1 are estimates only, based on an estimated level of accuracy of +/- 35%, as per the Cautionary Statement below**

Cautionary Statement

The Study has been undertaken to assess viability of developing the Colosseum Gold Project by constructing an open cut mine ± underground mine and processing facility to produce gold doré.

It is a preliminary technical and economic study of the potential viability of the Colosseum Project. It is based on technical and economic assessments that are not sufficient to support the estimation of Ore Reserves. Further exploration and evaluation work and appropriate studies are required before Dateline will be in a position to estimate any Ore Reserves or to provide any assurance of an economic development case.

The Study is based on the material assumptions highlighted throughout this announcement. While the Company considers all the material assumptions to be based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated by the Study will be achieved.

These include assumptions about the availability of funding. To achieve the potential project development outcomes indicated in the Study, funding in the order of US$152 million is needed (DTR presently has U.S. market capitalisation of approximately US$10 million). Investors should note that there is no certainty that the Company will be able to raise funding when needed, however the Company has concluded it has a reasonable basis for providing the forward-looking statements included in this announcement and believes that it will be able to fund the development of the project. This is based on a ratio of initial capital expenditure to market capitalisation of 15 : 1.

It is also possible that such funding may only be available on terms that may be dilutive to, or otherwise affect the value of the Company's existing shares. It is also possible that the Company could pursue other strategies to provide alternative funding options. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the Study.

There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated or Measured Mineral Resources or that the production target itself will be realised.

The Study is based on the June 2024 Mineral Resource Estimate[1], is based on low-level technical and economic assessments, and is insufficient to support estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Study will be realised.

The Study has been completed to a level of accuracy of +/-35% in line with industry standard accuracy for this stage of development. The Company has reasonable grounds for disclosing a Production Target, given that in the first five years of production, 89% of the mill feed is scheduled from the Measured and Indicated Resource category, which exceeds the economic payback period for the Project by 1.75 years.

Approximately 55% of the Life of Mine Production Target is in the Measured Mineral Resource category, 26% is in the Indicated Mineral Resource category and 19% is in the Inferred Mineral Resource category. There is a lower level of geological confidence associated with Inferred Mineral Resources, and while the Company considers all the material assumptions in this Study

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[1] ASX Announcement 6 June 2024 – 1.1Moz gold for updated Colosseum Mineral Resource

to be based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated will be achieved.

The Mineral Resources underpinning the production target in the Study have been prepared by a Competent Person in accordance with the requirements of Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code (2012)). The Competent Person's Statement is found in the Mineral Resources section of the Study. For full details of the Mineral Resource Estimate, please refer to Dateline's ASX Announcement dated 6 June 2024.

Dateline confirms that it is not aware of any new information or data that materially affects the information included in that release. All material assumptions and technical parameters underpinning the estimates in that Announcement continue to apply and have not materially changed.

Note that unless otherwise stated, all currency in this Announcement is US dollars.

About Dateline Resources Limited

Dateline Resources Limited (ASX: DTR) is an Australian publicly listed 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. On 6 June 2024, the Company announced to the ASX that the Colosseum Gold mine has a JORC-2012 compliant Mineral Resource estimate of 27.1Mt @ 1.26g/t Au for 1.1Moz. Of the total Mineral Resource, 455koz @ 1.47/t Au (41%) are classified as Measured, 281koz @1.21g/t Au (26%) as Indicated and 364koz @ 1.10g/t Au (33%) as Inferred.

The Colosseum is located less than 10km north of the Mountain Rare Earth mine. Work has commenced on identifying the source of the mantle derived rocks that are associated with carbonatites and are located at Colosseum.

Scoping Study Authors & Competent Person Statements

This Scoping Study is as defined in Clause 38 of the JORC Code 2012. It refers to the Mineral Resource Estimate announced by Dateline Resources Limited (DTR) on 6 June 2024 but the Production Targets presented do not constitute Ore Reserves as defined in the JORC Code 2012.

Apart from the Mineral Resource Estimate, the PDS has been completely compiled by Australian Mine Design and Development Pty Ltd (**AMDAD**) with information supplied by Dateline, generated by AMDAD or publicly sourced.

The principal author of the report and supervisor of the work conducted by AMDAD is Mr John Wyche BE(Min Hon) BComm FAusIMM CP. Mr Wyche is a Fellow of the Australasian Institute of Mining and Metallurgy. He has 37 years of relevant experience in hard rock gold mining.

Mr Wyche does not hold shares or any other form of equity in Dateline Resources Limited.

1. Executive Summary

- The Scoping Study (**Study**) for the Open Pit Only scenario (Case 2) assesses the mining and processing of 16.55Mt of ore grading 1.30g/t Au over an initial 8.4 year mine production life (the **Production Target**)
- Mining would be via open pit methods only, with 16.55Mt of ore and 56.75Mt of waste mines at a waste:ore strip ratio of 3.4:1
- Over the mine life, the project would produce 635koz of gold
- The Scoping Study was undertaken at a gold price of US$2,200 per ounce
- The mine plan includes open pit mining from the south pipe and the north pipe at Colosseum
- An alternative 'underground and open pit' scenario was modelled using sub-level caving as the underground mining method. Whilst it also generated a positive outcome, it was not as good as the 'open pit only' scenario
- The mine plan and associated infrastructure has been modelled within the existing area where the Company holds vested Mining Rights and an approved Plan of Operation.
- The Scoping Study provides sufficient confidence to move to a Project Selection Stage (**PSS**) ahead of a Bankable Feasibility Study (**BFS**)

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 March 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.

The Case 2 mine plan for the Study involves open pit cutbacks of both the south pipe and north pipe at Colosseum, utilising standard mining methods. The mine plan includes:

- 55% of mineral resources in the Measured Category,

- 26% in the Indicated Category, and
- 19% in the Inferred category.

During the first five years of operation, Inferred mineral resources comprise <11% of the mine feed.

The open pit mine plan uses the existing June 2024 MRE. Note that this excludes the positive results of drilling in the north pipe in the September 2024 quarter due to time constraints. The mine plan uses Multiple Indicator Kriging (**MIK**) for the block model type and applies a mining recovery of 95% and a mining dilution factor of 10%. This process has resulted in a total Production Target of 16.55Mt of ore grading 1.30g/t Au containing 692koz of gold. The Production Target comprises 81% of Mineral Resources classified as Measured/Indicated and 19% as Inferred.

The proposed processing flowsheet comprises of a 2x scaled up version of the carbon in pulp (**CIP**) processing plant that successfully operated at Colosseum between 1989 and 1993. This would result in the processing of 2Mtpa of ore at Colosseum. The Company is confident that the footprint for the proposed 2Mtpa processing plant would fit within the area approved by the Plan of Operation.

When in full production, the mine is forecast for 8.4 years and produce at a rate of 75koz per annum of gold doré for direct sale.

The operations will utilise a drive in-drive out (**DIDO**) workforce with personnel recruited from nearby towns. There is no requirement for the construction of accommodation in the Study.

The Project operations are scheduled to reach full commercial production in month 13 with the estimated pre-production capital cost being US$138 million. When in full production, the average monthly operating costs for the operation are US$7.4 million. During this period, capital costs per month average US$252k, fluctuating between US$229k up to US$422k. The estimated total unit operating costs over the life of the Project are US$45/t milled with total capital costs of US$165 million over the life of mine plus a $30 million provision for mine closure.

The total unit operating costs were calculated as US$18.60 per ore tonne for mining, US$20 per tonne processed for milling (including power) with other ancillary costs being US$6.60/t. Annual power costs at commercial production average US$75.1M.

Using a US$2,200/oz gold price, the estimated Colosseum Gold Mine economic factors are:

- US$398M free cashflow generated over an 8.4 year period
- Total pre-production capital expenditure and working capital requirements of $152.7m
- All In Sustaining Cost of gold production is US$1,490/oz.
- NPV at an 6.5% discount rate for the Project is US$235M
- Internal rate of return (IRR) of 31%.

From the current defined Production Target of 16.55Mt @ 1.30 g/t Au, full operation mine production can be achieved for 8.4 years. There are exploration results that have been returned outside the MRE, which if confirmed by further drilling, have the potential to provide substantial upside and high probability of mine life extending beyond the Scoping Study projected life.

These additional Mineral Resources will be assessed and ultimately exploited, if viable, to provide an increase in the production cashflow and mine life of the Project.

Table 2: Colosseum Gold Mine Scoping Study Inputs and Outcomes

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Opportunities being explored to enhance potential returns

The Company is exploring the potential to sell the previously mined waste material as aggregate, to be used in cement and asphalt in support of the nearby increasing demand of the growing Las Vegas market. Samples have been sent to Aztech Labs for aggregate suitability analysis. If the waste material is suitable and can be sole as aggregate, this may improve project economics.

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. In 2024, gold prices reached unprecedented highs, peaking at $2,721 per ounce. This increase is partly due to significant central bank demand, particularly from emerging markets like China, India, and Turkey.

Gold prices are heavily influenced by global economic conditions, including inflation rates and geopolitical tensions. Despite the 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 1: Gold Price – 12-month spot price vs Study Price (US$2200/oz)

As can be seen in Figure 1, the recent spot price of ~US$2,650/oz is more than 20% above the price used in the Study.

2. Project 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 (Figure 2). It is 14km from the California Nevada state boundary.

Figure 2: Colosseum Gold Mine location

Colosseum is located at the northern end of the Clark Mountains (see Figure 3 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.

The North and South Pipes were mined as two adjacent pits from 1998 to 1993. Operations were suspended in 1994, 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.

Figure 3: Clark Mountains looking north to Colosseum

Figure 4: Looking south to Clark Mountains

3. Mining Scenarios

Two potential mine plans were considered in this Study:

- **CASE 1 - UNDERGOUND & OPEN PIT- Underground mining of the South Pipe and opencut mining of the North Pipe and a small zone west of the South Pipe.** Mill feed is set at 1.2 million tons per year (1.09 Mtpa) to match the actual throughput from 1988 to 1993. Underground mining of the South Pipe is used to provide high grade feed early in the mine life to boost gold production and early cash flows. The North Pipe and a small, shallow zone west of the South Pipe are mined by opencut methods. Initially the opencut is mined at a rate to make up the difference between the underground production rate and the mill feed rate. After the underground is depleted the opencut mining rate is increased to meet the full mill feed rate.
- **CASE 2 - OPEN PIT ONLY - Opencut mining of both the North and South Pipes.** Preliminary pit optimisations indicated maximum value can be achieved by opencut mining of the whole deposit. The mill feed rate was set at 2 Mtpa as this can be achieved with a moderate size mining fleet and benchmark studies indicate the initial project capital cost may be within a practical range for DTR. Amendments to existing approvals may be required to operate a 2 Mtpa mine but the process plant would still be a scaled up and modernised version of the plant that operated successfully from 1998 to 1993.

A more thorough analysis of the production rate under either scenario would be required in a future Feasibility Study. In keeping with the goals of this Study, the two cases were selected to identify and examine practical aspects of different mining methods and to assess the relative values of production at the rate of the former operation and a higher rate more aligned to the potential project scale.

Case 2, the 'Open Pit Only' scenario produced a more robust outcome from an economic and risk-based perspective and is detailed in the following sections. A summary of the 'Underground and Open Pit' scenario is included for reference in Appendix 1.

Opencut Mining

Examination of the drillhole database and mineral resource block model shows gold mineralisation in the South and North Pipes consists of higher-grade zones (>1.0 g/t Au) within broader zones of 0.3 to 1.0 g/t Au. The low costs and high selectivity achievable with opencut mining compared to underground mining make a much larger proportion of the deposit amenable to mining by opencut methods. Most of the currently defined resource is near surface which makes opencut mining a realistic alternative to underground methods.

Opencut Mining Method Selection

The main options for opencut mining consist of:

- **Continuous mining** using tools like the Wirtgen or Vermeer surface miners. These machines can be highly selective in stratified deposits with relatively low strength rock. However, in more complex steeply dipping orebodies with higher waste to ore ratios they are often too slow and too expensive and must be used in conjunction with other methods for the waste rock mining.
- **Bulk mining** using face shovels or front-end loaders on high benches to dig blasted rock is a low cost option where the ore zones are broad laterally and vertically and the target

commodity has relatively low unit value. This is typically used in large operations such as iron ore or large scale copper mines. However, in orebodies like Colosseum the costs associated with mis-classifying material as ore and sending it to the mill (increased process cost) or as waste and sending it to the waste rock emplacement (lost revenue) can be very large.

- **Selective mining** using hydraulic backhoes on low benches to dig blasted rock is the most common opencut method in small to medium scale gold mines. Benches are blasted at 5 to 6 metre heights and mined in 2 to 3 metre high 'flitches'. Hydraulic excavators in backhoe configuration provide an effective combination of selectivity and productivity and are capable of efficiently mining the waste rock as well plus additional tasks such forming final walls or drop cutting ramps.

Selective mining using hydraulic backhoes was selected as the opencut mining method for the Study.

Opencut Mining Operations

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

- 2.0 Mtpa mill feed and an average of 6.5 Mtpa waste rock peaking at 8.6 Mtpa in Years 3 to 6.

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

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

Most of the waste rock would have to be placed external to the pits. External waste rock emplacement would increase the disturbed area and would require care in design, formation and rehabilitation.

Grade control

The most appropriate grade control system needs to be investigated and defined by the DTR Geological team. For the purpose of the Study, AMDAD has assumed that grade control sampling would be undertaken by reverse circulation (**RC**) drilling in advance of the mining benches:-

- Samples would be analysed in a laboratory at Site and the assayed grades applied to the grade control block model to define ore zones for mark out.
- Drill holes angled at approximately 60°, with conceptual drillhole line spacing of 7.5m and collars spaced 5m along each line.
- 60m hole depth to cover five 10m benches.

Grade control and marking out of ore zones would commence in the half-year prior to mine production. At this time, the grade control procedures would be trialled and evaluated. As well as the methods for sampling, an assessment would be made regarding the adequacy of visual control for digging of marked out ore zones at night, using lighting plants, and whether ore mining would generally only be undertaken during daylight hours.

Optimisation of hole spacing will involve a trade-off between increase in accuracy of the orebody model and the cost of drilling, sampling and assaying. Optimisation of hole depth must consider hole deviation and the impact on effective spacing versus the benefit of gaining the data well in advance of the mining bench.

The RC grade control drilling will also define ore and waste types in conjunction with the waste rock management plan and closure plan.

Soil testing will address rehabilitation constraints (nutrient, erodibility, dispersion risks).

Waste rock classification, primarily by sulphur %, may also involve the use of a portable x-ray fluoroscope (**pXRF**).

Drill and Blast (D&B)

It is anticipated that all materials mined in the open cut, except areas where the pits mine through existing waste rock emplacements, would require some level of drilling and blasting (D&B) for productive excavation. D&B performance will be affected by faults and other fracturing, and by water.

The drilling and blasting operations would entail the following:

- Drilling and blasting will generally be conducted on 5-metre high benches but may be conducted on 10 metre benches in the broader upper benches.
- Blastholes will be drilled using a DTH hammer diesel powered drill rig.
- Geological logging, mapping and drill penetration rates from the previous bench, from the resource drilling and from a historical drill penetration rate study (Amselco 1984) will be used to predict conditions on the active bench for blast pattern design.
- Blast patterns will be sized as large as possible to minimise the frequency of blasts and would aim for a minimum of approximately 80 kt on a 5-metre bench or 200 kt on a 10-metre bench.
- Blasts will typically be fired as choke blasts to minimize disruption to the mining cycle, and to limit lateral movement of ore boundaries.
- As recommended by the 1980’s rock mechanics study (Amselco, 1984):-
    - controlled blasting will be employed with smaller diameter blast holes, closer spacing, and the use of delays to minimize the back break and damage to the final open cut wall from blasting.
    - Some experimental controlled blasting would be undertaken to determine the procedure to be used on the final bench faces.
- Blastholes will be loaded predominantly with Ammonium Nitrate/Fuel Oil (**ANFO**) mixture by a mobile explosives manufacturing unit (**MEMU**).
- In the uncommon event of wet conditions after a significant storm or due to groundwater, emulsion explosives would be used.
- Conventional non-electric initiation products would be used for most blasts.
- Blast design, including initiation, will be prepared to comply with vibration and air pressure guidelines.
- Stemming material will be obtained by crushing and screening of suitable aggregates sourced from hard rock at or near the project.

- A comprehensive blasting procedure will be implemented as part of the mine safety management plan to ensure the safety of personnel and equipment in and around the mine during blasts. This will be developed during the blast study proposed for the DFS program. A blast would be delayed or postponed if climatic or other conditions did not comply with the procedure.
- The orientation of some blasting benches relative to the project facilities, the geology and structure may pose a fly-rock risk for personnel and infrastructure. To address this risk, the mine facilities area, Run of Mine (**ROM**) crusher and critical processing infrastructure will need to be designed to lie outside a defined blast radius from the open cut. Blast procedures must specifically address this fly-rock risk with precautions in addition to pre-blast clearance procedures that would include careful logging of drilling and appropriate adjustment of charging and stemming to mitigate the risk.
- Blasts would be fired on dayshift only. The exclusion zone for personnel around the blast would depend on the specifics of the shot and the material; however, it is expected to typically be around 500 m.

A blasting study will be conducted during the Definitive Feasibility Study (**DFS**) to inform an appropriate blast design to meet acceptable standards and limits, including:-

- Confirmation of powder factors
- Preparation of typical blast designs,
- Investigation of fly-rock risk,
- Definition of the exclusion radius,
- Provision of preliminary blasting procedures
- Provision of data for blast vibration analysis
- Assessment of explosives/AN storage and transportation

Loading and Hauling

Blasted waste rock and ore will be loaded and hauled by a mining fleet likely to comprise an 90t to 120t class diesel-hydraulic excavator loading 55 t articulated dump trucks. An example would be a 107t Komatsu PC1250SE-8R excavator loading 55 t Volvo A60H ADTs.

The different ore types and waste rock types will be mined and stockpiled separately.

Mill feed will be hauled to the ROM area and either direct tipped into the primary crusher or placed in a stockpile adjacent to the ROM pad.

Waste rock will be hauled to the waste rock emplacements adjacent to the pits (see Waste Rock Emplacements section).

Water Management

The site is characterized by very low precipitation. Nevertheless, a significant rainfall event could result in considerable surface runoff and flows in the water courses within the project area. This may represent a risk to safe mining operations and to contamination of local waterways. These risks must be managed by adherence to a detailed Water Management Plan that will be developed during the DFS program. It will address the requirements for and impacts of the following facets of mining operations:-

- Management of surface water and groundwater to separate clean water and contact water, and prevent contamination of local water courses and groundwater, including:
- Diversion of clean water around the mine workings and infrastructure areas
- Control of contact-water laden with sediment or chemical leachates, using bunds, drains, channels, piping and containment ponds
- Trafficability and operability of the mining fleet on roads and benches
- Potential for submergence of working areas at the bottom of the open cut
- Impact of water on explosives charging
- Stability of open cut walls
- Stability of waste rock emplacement (**WRE**)

Ancillary Functions

Mining support, or ancillary, functions and methods are listed below:

- A bulldozer would maintain open cut bench floors, clean-up around the excavator, doze off waste rock at the WRE, maintain and profile the WRE as required.
- A grader would maintain roads, with a compactor used to construct roads,
- A water cart would be used for dust suppression,
- A smaller excavator in the range of 30 t to 50 t would be used for:
    - Batter scaling
    - Forming windrows
    - Digging small sumps and drains
    - Moving pumps
    - Secondary breakage of oversize rock after blasting using a rock breaker attachment
    - Slope maintenance and removal of material from minor wall slips will be completed by excavator, wheel loader and trucks as required,
- A mobile crane would be required for bucket changes, component changes and positioning dewatering pumps,
- A crushing and screening plant would provide aggregate for road sheeting and blasthole stemming,
- A small wheel loader or integrated tool carrier would load this plant, load stemming into blastholes, and facilitate tyre handling,
- A service truck would refuel and lube the excavators and drill rig, and
- Other ancillary equipment would include forklift, boom lift, flatbed truck, lighting plants and bus.

Waste Rock Management

The waste rock management systems approved for the former operation would be the minimum standard for the proposed resumption of mining. Specific measures to be considered for the proposed mine plan include:

- Stripping of any topsoil material if required from the WRE footprint(s) to enhance the stability of the WRE foundations and to provide soil and vegetation for rehabilitation.
- Construction of diversion systems around the WREs to minimise surface runoff water from flowing onto the WRE surfaces and or permeating the WRE material.
- Construction of containment ponds at the down-grade toes of the WREs, where appropriate, to collect sediment and water which has been in contact with eth waste rock materials.
- Construction of other drainage control measures within the WRE footprints and around the WRE perimeters including bunds, drains and drop structures to direct runoff water and seepage water towards the containment ponds.
- Waste rock types would be sampled and analysed by site geologists to allow the segregation and selective transport, handling and containment of potentially acid forming (PAF) materials with non-acid forming (**NAF**) materials.
- PAF materials would be encapsulated by NAF material and, where possible, include low permeability capping on each WRE bench.
- Low permeability capping to be achieved by compaction through traffic of haul trucks, to reduce rain infiltration and to promote stability.

Further details of the proposed WREs are included further in the report.

Waste Rock Acid Generation Potential

According to U.S. EPA (1993), the previous operator at Colosseum tested waste rock material using the acid-base accounting method for determining acid generating potential and advised that the results indicated that waste rock is not acid generating. Therefore, for the purpose of the DPS, no allowance has been made for any specific measures for management of potentially acid forming (**PAF**) waste rock. However, the DFS program would include geochemical test work to confirm that there is no PAF risk for waste rock.

Existing Waste Rock Emplacements

Waste rock previously excavated from the two open pits is stored in contiguous emplacements surrounding the South Pit and North Pit, as shown in Figure 5 below.

No liners underly the waste piles, and the dumps exist typically at their natural angle of repose except where the previous operator graded the waste rock to provide access, to maintain the waste dump surface for receiving rock, or where there has been limited rehandling and/or profiling after completion of the previous mining and processing operation.

The existence and nature of runoff controls at the waste rock piles is not known.

No underdrain or sediment collection dams were proposed for these piles, according to the Environmental Impact Statement (**EIS**). The EIS presented the following rationale for this decision (Bureau of Land Management, et al., 1985a):

- The slope of the waste rock surface will direct storm water runoff flow back toward the open pits;
- Due to low annual precipitation and high annual evaporation, percolation of storm water runoff is unlikely, and;
- In the event of storm water infiltration, any leachate created would contain the same compounds as the surrounding host rock; no sulphides were to be disposed of in the waste rock piles.

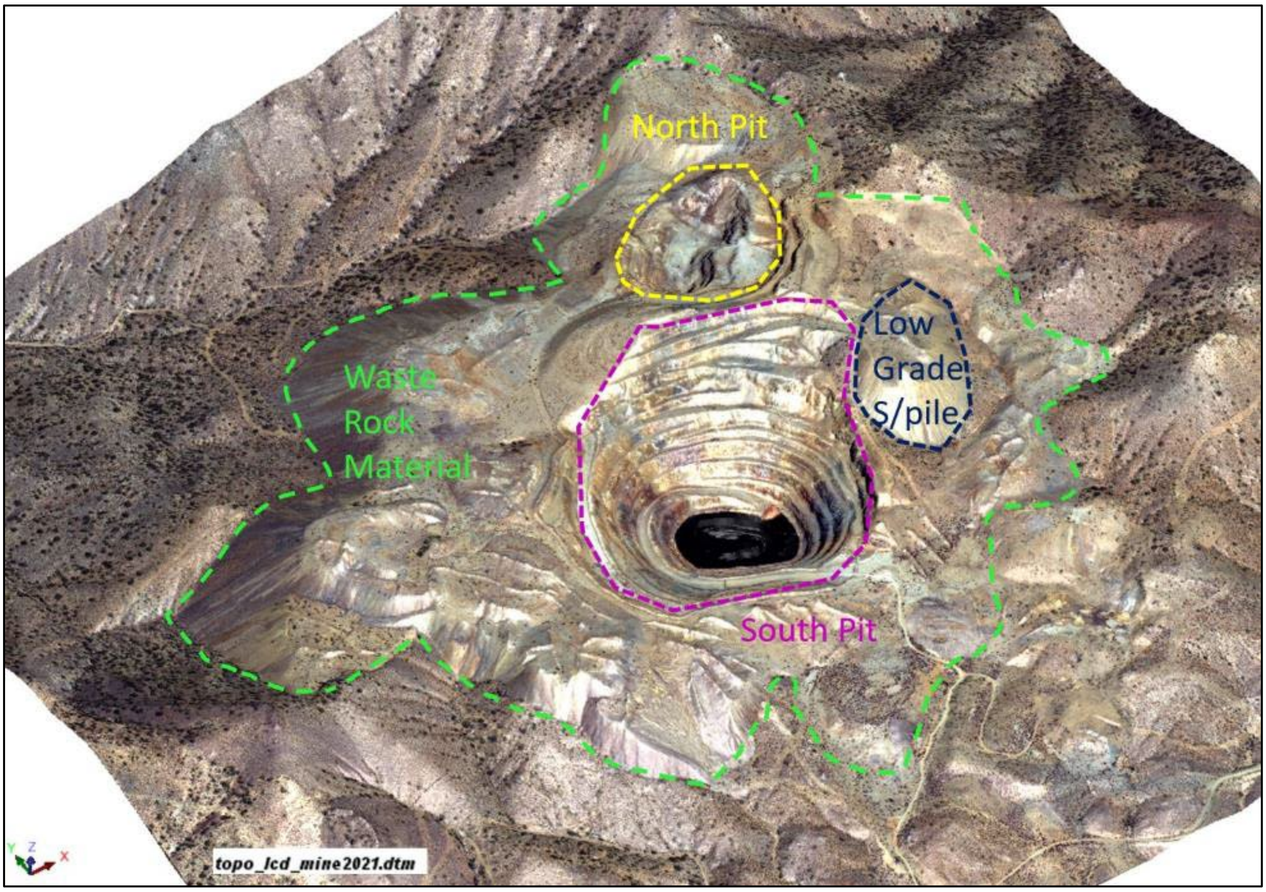

Figure 5: View looking north-east showing existing mined waste rock within green line

Open Cut Pit Optimisation

Whittle™ pit optimisation was used to define the starter and final pit shapes for the Study.

Pit Optimisation Inputs

The pit optimisation inputs are designed to provide operating, cost and revenue values across the entire zone of target mineralisation and surrounding ground which are representative of the final operating parameters but general enough to allow the software to define the shape of the highest value pit cone.

Table 3: Pit Optimisation Inputs and Estimates

Input	Units	Case 2	Comments
Block model type		MIK	See Section 5
Resource categories included		Mea, Ind, Inf	For Study
Mining recovery		95%	Assumed
Mining dilution		10%	Assumed
Mining dilution grade	g/t Au	0.00	Assumed
Overall Wall Slopes	In rock	43°	See Section 7
	In fill	26°	
Spatial constraints		Within Unpatented Claims Area	See Section 7
Maximum mining rate	tonnes pa	Unconstrained	Assumed
Mill Feed Rate	tonnes pa	2,000,000	See Section 8
Process Gold recovery		92.0%	See Section 8
Mining Costs			
Load and Haul at 1755m Bench	US$/tonne	$2.13	Preliminary cost model
Increase per metre above 1755m	US$/tonne	Variable. See chart	See Figure 7
Increase per metre above 1755m	US$/tonne	Variable. See chart	See Figure 7
Drill and blast cost	US$/tonne	$0.80	Preliminary cost model
Additional cost of mill feed tonne	US$/tonne	$1.30	Preliminary cost model
General and Administration Cost	US$/ROM tonne	$18.75	See Section 13
Process Cost	US$/ROM tonne	$6.63	See Section 13
Gold Price	US$/oz	$2,200	See Section 14
Gold Realisation Charges			See Section 14
Refining	US$/oz	$7.00	
Payable gold		99.0%	
Dore transport & Insurance	US$/oz	$1.00	
Royalties		2.5%	
Discount Rate		10%	Assumed for pit optimisation only

Mineral Resource Block Model

The mineral resource was estimated as a median indicator kriged model with grade increments of:

- 0.30 to 0.50 g/t Au
- 0.50 to 0.75 g/t Au
- 0.75 to 1.00 g/t Au
- 1.00 to 1.25 g/t Au
- 1.25 to 1.50 g/t Au
- 1.50 to 1.75 g/t Au
- 1.75 to 2.00 g/t Au
- 2.00 to 2.25 g/t Au
- 2.25 to 2.50 g/t Au
- 2.50 to 2.75 g/t Au
- 2.75 to 3.00 g/t Au
- 3.00 to 3.50 g/t Au
- 3.50 to 4.00 g/t Au
- 4.00 to 5.00 g/t Au
- 5.00 g/t Au

Grades were estimated into 10x10x5 metre panels, and each grade increment estimate included an estimate of the proportion of the panel above the bottom grade of the increment. The model also includes an e-type estimate of the average gold grade of the whole panel (a zero cut-off grade). The pit optimisation software treats each grade increment as a parcel within each panel and so works down to the grade increment containing the economic cut-off grade.

The model classes each panel as Measured, Indicated or Inferred resources but also includes estimates for panels where distance from samples or other reasons reduce estimation confidence to lower than Inferred. The pit optimisation, and Study generally, only consider panels with Measured, Indicated or Inferred resources.

Mining Loss and Dilution

In determining the marginal economic cut-off grade to see which panels contain potential ‘ore’, the pit optimisation software applies nominated mining ore loss and dilution factors. It is difficult to spatially model mining loss and dilution in MIK models. AMDAD ran some rudimentary analyses to model ‘edge’ dilution of 0.3 to 0.5 metres at the cut-off grade in each panel and the resulting adjusted grades and panel proportions were used as the basis of global allowances of 10% dilution at zero grade and 95% mining recovery.

Spatial Constraints

Two spatial constraints were applied:

- The optimised pit shell must be within the Patented and Unpatented Claims boundaries.
- In the alternative 'underground plus open pit' scenario, the pit optimisations assumes that no mineralisation remains within the estimated subsidence cone following underground mining of the South Pipe.

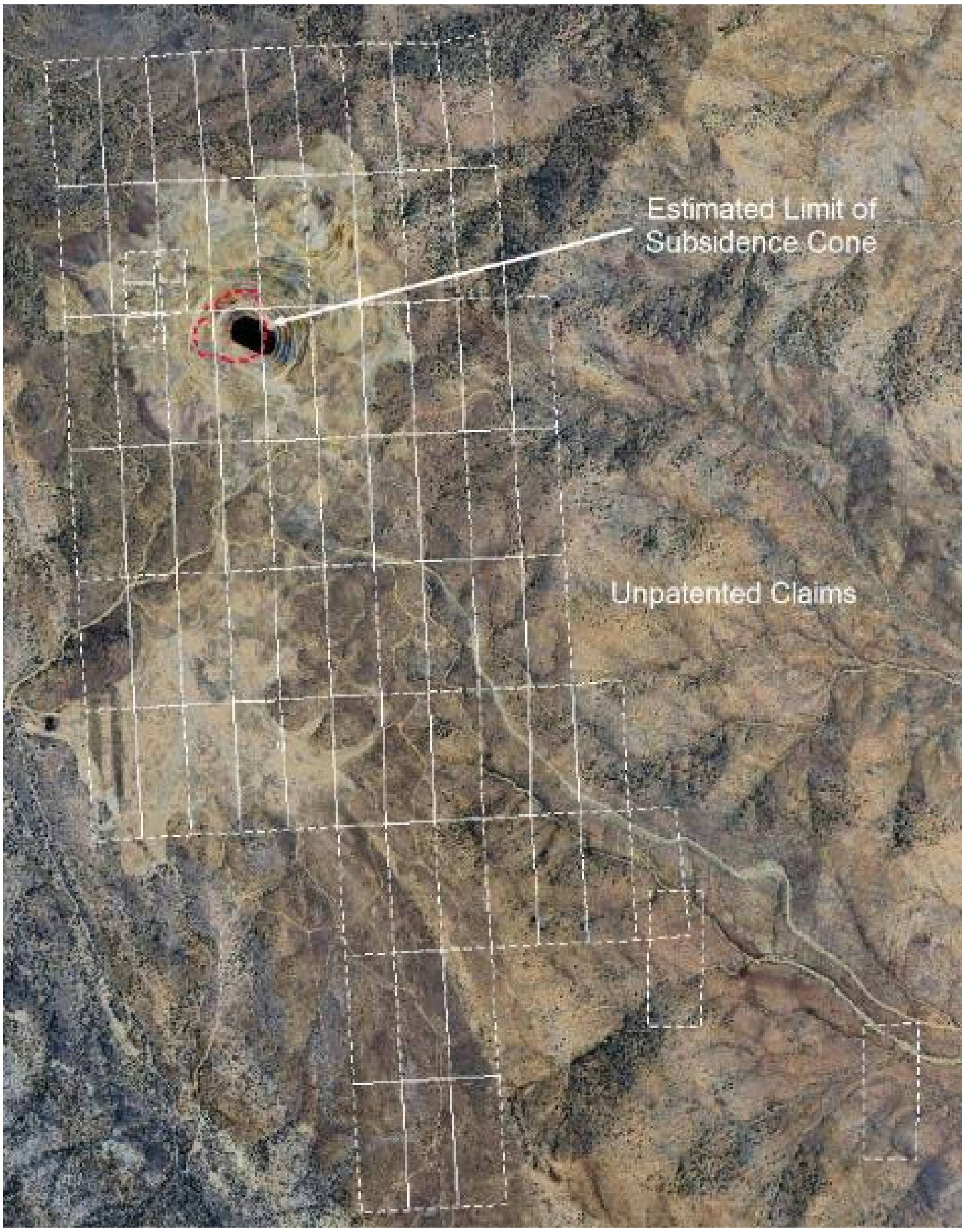

Figure 6: Spatial Constraints for Case 1 (UG & Open pit scenario)

Mill Feed Rate

The mill feed rate is set to 2,000,000 tonnes pa to provide the best economic outcome based on the level of Measured and Indicated mineral resources considered in this Study.

Opencut Mining Costs

Opencut mining costs are made up of:

- Load and haul costs. Varies with depth and position in the deposit due to changing haul lengths.
- Drill and blast costs. Applies to rock but not to fill where the pits mine through existing waste rock emplacements.
- Management, technical and supervisory (**MTS**) costs. Allocated to ore and waste on tonnes mined.
- A range of other costs such as grade control and environmental management. Some specific ore costs and some general allocations.

To capture these costs in a way that could be used by the pit optimisation throughout the deposit, AMDAD ran a trial pit optimisation using a single cost per tonne for ore and for waste. A production schedule was run for the resulting pit shells, simple haul routes were estimated, and a first principles mining cost estimate was prepared in a similar manner to the model described in Table 16.

AMDAD graphed the estimated load and haul costs for each period against the average mining bench level for that period to derive the following chart in which the dots represent the modelled ore and waste load and haul costs per tonne and the crosses were assigned as the values for use in the pit optimisation.

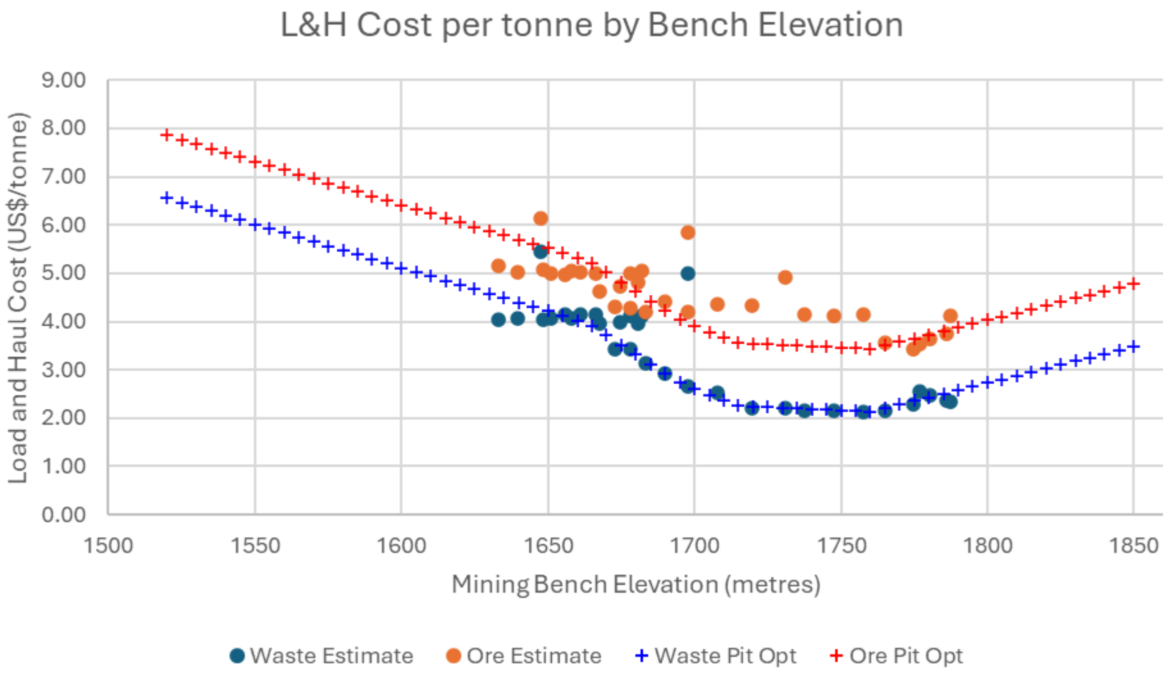

Figure 7: Mining Costs for Pit Optimisation

AMDAD added drill and blast costs to the non-fill material below the current topography surface and assigned ore and waste MTS costs to each block in the model to complete the total mining cost for each block if the software assigned it as ore or waste.

As a check that the estimated costs per tonne by bench level are representative of the first principles cost estimate, mill feed and waste tonnes in the production schedule each period were multiplied by the costs per tonne from the pit optimisation inputs for the average mining bench level in that period. The following figure shows a chart of cumulative mining costs over the mine life estimated from first principles and from the optimisation inputs. The optimisation inputs case closely follows the first principles estimate.

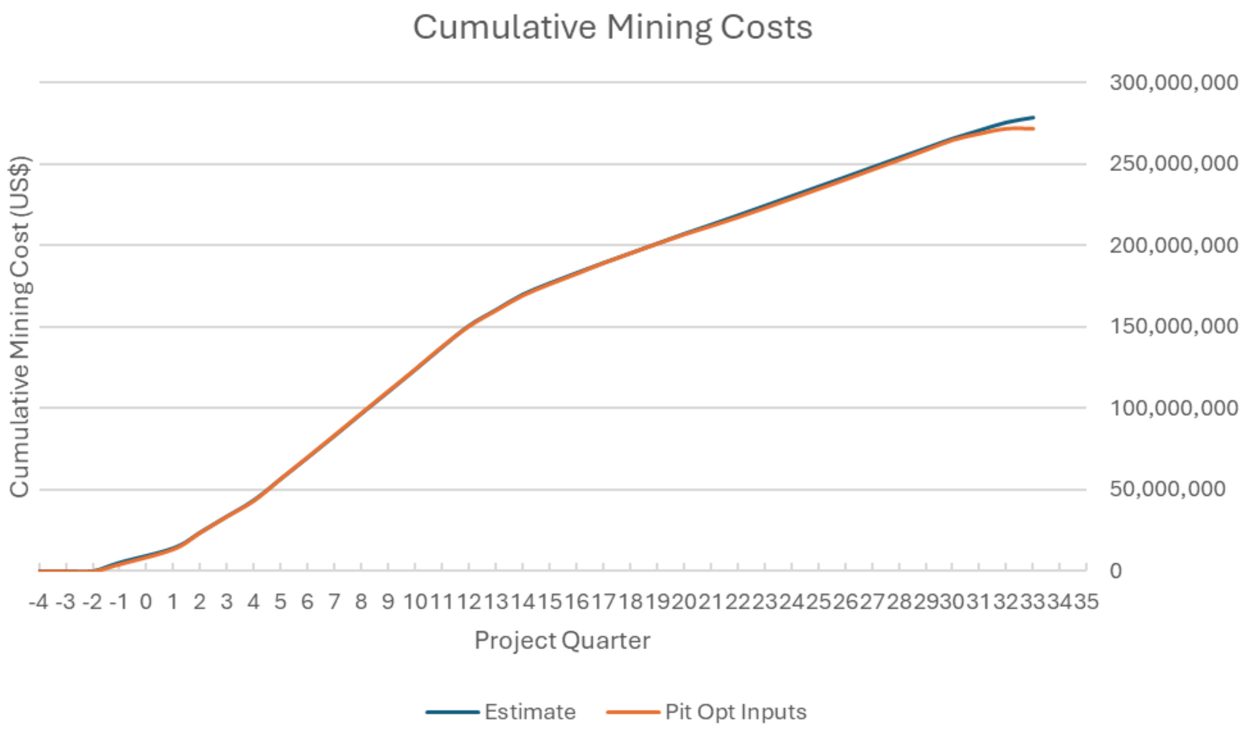

Figure 8: Check on Mining Costs for Pit Optimisation

Process and Site Costs

Process and General and Administration costs per ROM tonne were estimated from benchmarking of other gold projects. See Section 10.

Gold Price

A fixed gold price of US$2,200/oz was set based on the average forecasts from five reputable banking and commodity forecast groups. See Section 10.

Discount Rate

Whittle™ pit optimisation uses a discount rate to allow selection of pit shells on a maximum discounted cashflow basis to assess pit staging. The optimisation runs used 10%. This is higher than in the final economic analysis in Section 10.

Pit Optimisation Results

Whittle™ pit optimisation produces a series of nested shells to match a range of nominated revenue factors. A revenue factor (**RF**) is a multiplier applied to the metal price. RF = 1.00 for the base case price. RF's less than 1.00 force the optimisation to form smaller shells targeting the parts of the deposit with highest value per tonne mined. RFs greater than 1.00 produce larger shells with lower value per tonne mined.

The software reports the operating cashflow (total revenue from recovered product less total operating costs and realisation charges) for each shell produced.

Looking at the undiscounted cashflows will always show RF = 1.00 to have the highest value when values are calculated at the base case inputs. By definition RF =1.00 means the software has sought out the pit cone with the highest base case value. Smaller shells could increase value by pushing deeper and wider, as long as the incremental mining cost is less than the net value of additional recoverable product. Larger shells will always have lower undiscounted value because the incremental mining cost of each shell outweighs the net value of additional recoverable product.

The same shells can also be reported as values discounted at the nominated rate. The software does this by effectively scheduling the shell from top to bottom at whatever mining rate is required to meet the nominated mill feed rate, noting that cones are wider at the top and so have more waste per tonne of mill feed at top than at the bottom. If a maximum mining rate is nominated the software will not exceed it, even if the mill would be underfed. Selecting the highest discounted cashflow (**DCF**) shell is generally a more realistic approach to pit optimisation than using the highest undiscounted value because it recognises that higher early mining costs to meet the mill feed tonnage in early periods may reduce the present value compared with smaller shells.

Using DCF also assists selection of pit staging. Nominating a starter shell smaller than the maximum single stage DCF shell defers the cost of pushing back to the larger shells and will generally results in maximum value staged DCF final shell closer to the undiscounted case.

The maximum DCF value staged pits were selected as the bases of the mine plan scenarios.

Table 4: Pit Optimisation Results, based on modelled and estimated inputs

	Final	Start	Mine	Undisc.	DCF	DCF	Strip	Material Mined			Grade	Product
	Pit	Pit	life	Cashflow	Staged	Single	ratio	Total	Waste	Mill	Au	Au
	Shell	Shell	Yrs	US$M	US$M	US$M		Mt	Mt	Mt	g/t	oz
Undiscounted												
Open Pit Only	36		12.5	**723**	424	341	2.6	90.2	65.2	25.0	1.09	804,459
DCF Single Stage												
Open Pit Only	22		9.3	671	435	**383**	1.9	52.9	34.3	18.6	1.16	636,698
DCF Staged												
Open Pit Only	**27**	**9**	**9.9**	687	**437**	379	**2.0**	**59.0**	**39.2**	**19.8**	**1.14**	**670,534**

The values shown in Table 4 are estimates of operating cashflows excluding capital costs. They are approximate because they:

- Are derived from optimised shells rather than practical pit design,

- Use general mining costs per tonne by elevation rather than first principles cost estimation, and
- Do not account for detailed practical scheduling issues.

The main purpose of the values is to provide a relative value ranking of the shells.

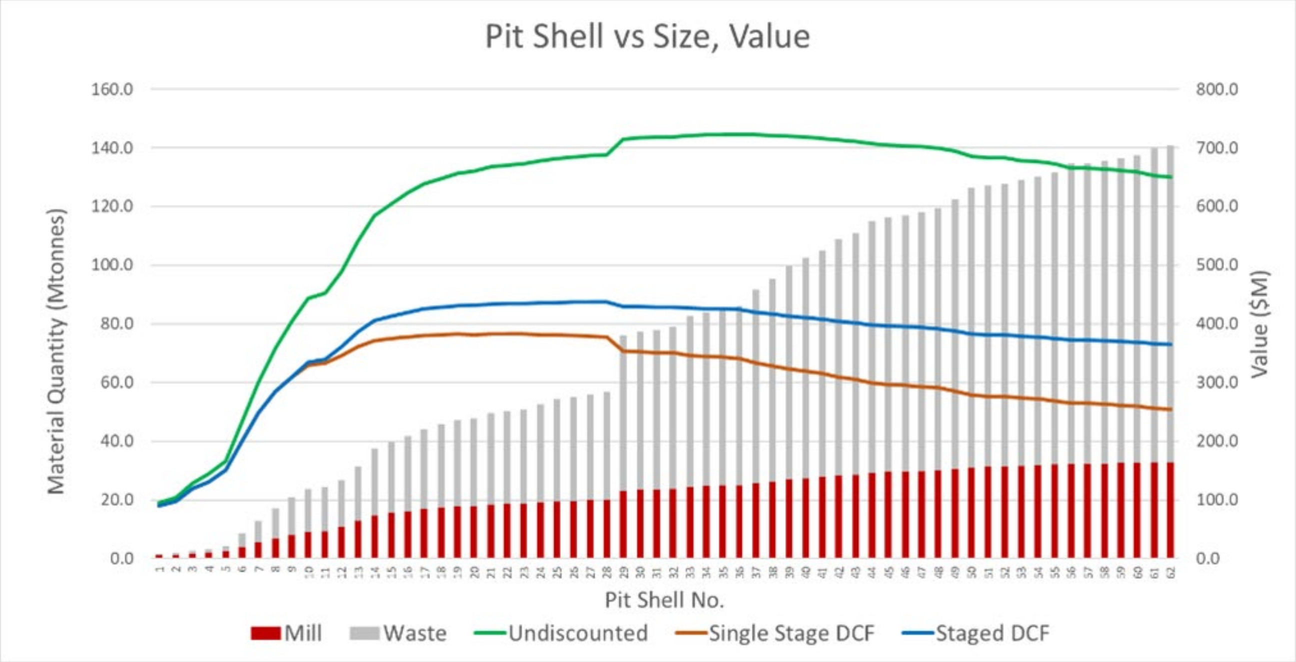

Figure 9: 'Open Pit Only' scenario Optimisation Results

Practical Opencut Design

The optimised pit shells are used to guide design of practical pits including berms and ramps and smoothing out shapes in the pit walls and floors which cannot be mined using standard commercially viable methods.

Pit Wall Design

The open cut slope design in the Feasibility Study report by Amselco (1984) is significantly steeper than the walls that were actually mined. Until a geotechnical assessment is conducted for the DFS, the Study adopts a flatter design based on the as-mined slopes which have stood without major failures for over 30 years. The wall slopes are shown in the following figure and table.

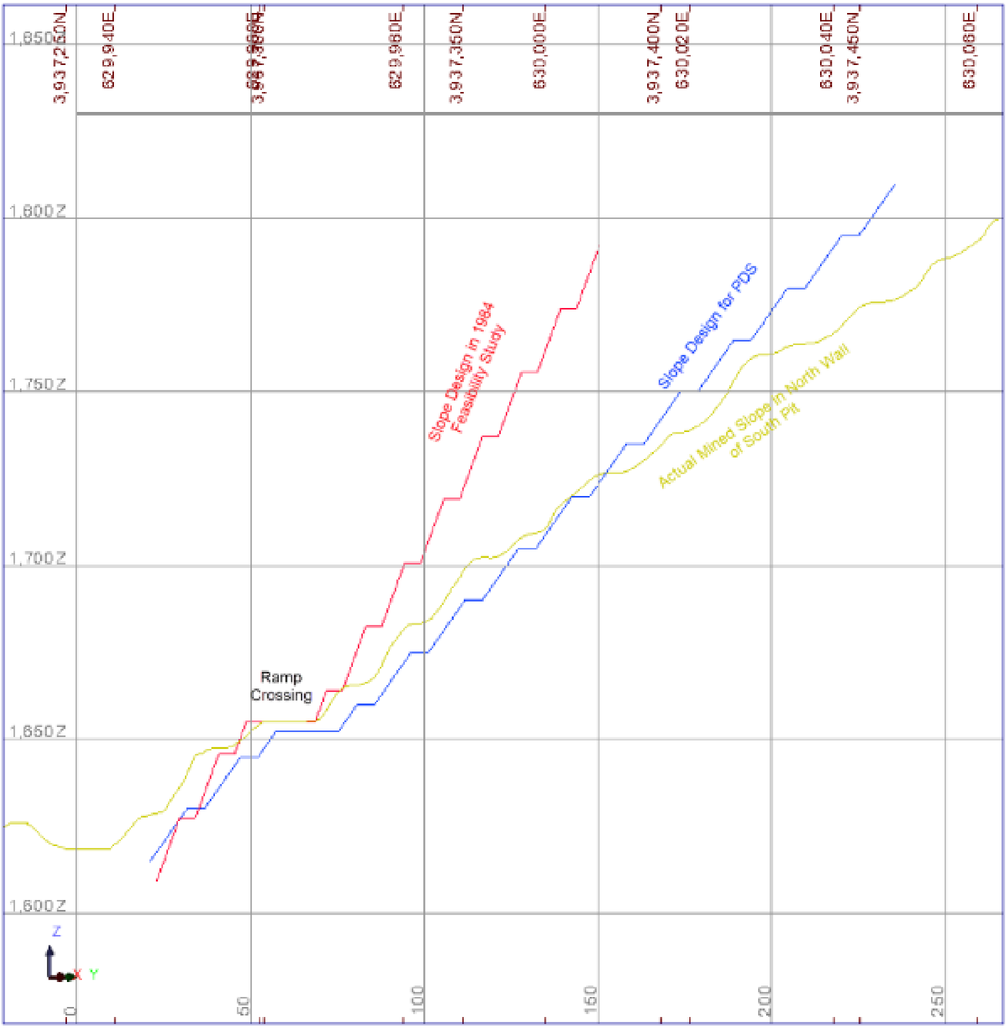

Figure 10: Pit Wall Slope Design

Some upper sections of the pit walls will cut into existing waste rock dumps. Where walls are cut into fill the batter slope is reduced to 37° and the berm width is increased to 10 metres.

Table 5: Pit Wall Slope Design Assumptions

Slope Parameter	Design
Batter angle	
Rock	55°
Fill	37°
Face height	
Rock	15 metres
Fill	15 metres
Berm width	
Rock	5 metres
Fill	10 metres
Inter-ramp slope	
Rock	46.4°
Fill	26.6°

When 18-metre wide ramps are included the average overall wall slope in rock reduces to 43.2°.

Pit Ramps

The pit ramps are designed to accommodate Volvo A60H 55 tonne payload articulated dump trucks. These units are narrower than comparable rigid body trucks and can negotiate steeper grades at similar up ramp speeds. These factors significantly decrease waste volumes and allow mining of smaller benches at the base of the pit recovering ore which may not be mineable with wider rigid body trucks. The following table provides the calculation of ramp widths.

Table 6: Pit Ramp Design Criteria

Haul Road Widths			
Truck		Volvo A60H	
Truck width	m	3.989	Over tray spill guards
Tyres		37.00R57	
Static load radius	m	0.925	
Two way traffic with open side			
x vehicle width		3.5	
Running width	m	13.96	
Safety bund width	m	2.96	37deg sides, 0.5m top width
Drain on wall side	m	1.00	Nominal
Ramp width	m	17.92	
Design width	m	18.00	
One way traffic with open side			Mainly near base of pit
x vehicle width		2.5	
Running width	m	9.97	
Safety bund width	m	2.96	37deg sides, 0.5m top width
Drain on wall side	m	1.00	Nominal
Ramp width	m	13.93	
Design width	m	14.00	
All cut ramps			Two way, no open sides
x vehicle width		3.5	
Running width	m	13.96	
Safety bund width	m	0.00	Not required
Drain on wall side	m	2.00	Nominal, both sides of road
Ramp width	m	15.96	
Design width	m	16.00	
Pit Ramp Grades			
Normal		12.50%	

Pit Design

The pit design mines the North Pipe in two pit stages and a small pit on the western side of the existing South Pit.

Access to the north pit would be down the western side of the existing South Pit. Initially ore and waste from the upper benches would be hauled directly off the benches and across the existing low grade stockpile area to the ROM stockpile area and the East WRE. As the pit becomes deeper all material mined would exit the pit along a ramp cut into the upper eastern wall of the existing South Pit. This would provide access to the ROM area and the East and West WREs.

Ore and waste from the small West Pit would be hauled south around the existing South Pit to the ROM area, the East WRE and the West WRE. The West Pit must be completed before the West WRE, which includes backfilling of the South Pit above the underground mine, reaches the West Pit exit at RL1745.

The third stage of mining would be a push back of the existing South Pit to mine the South Pipe mineralisation below the existing pit and the small zone to the west of it. Stage 3 would use the same pit exit as Stages 1 and 2.

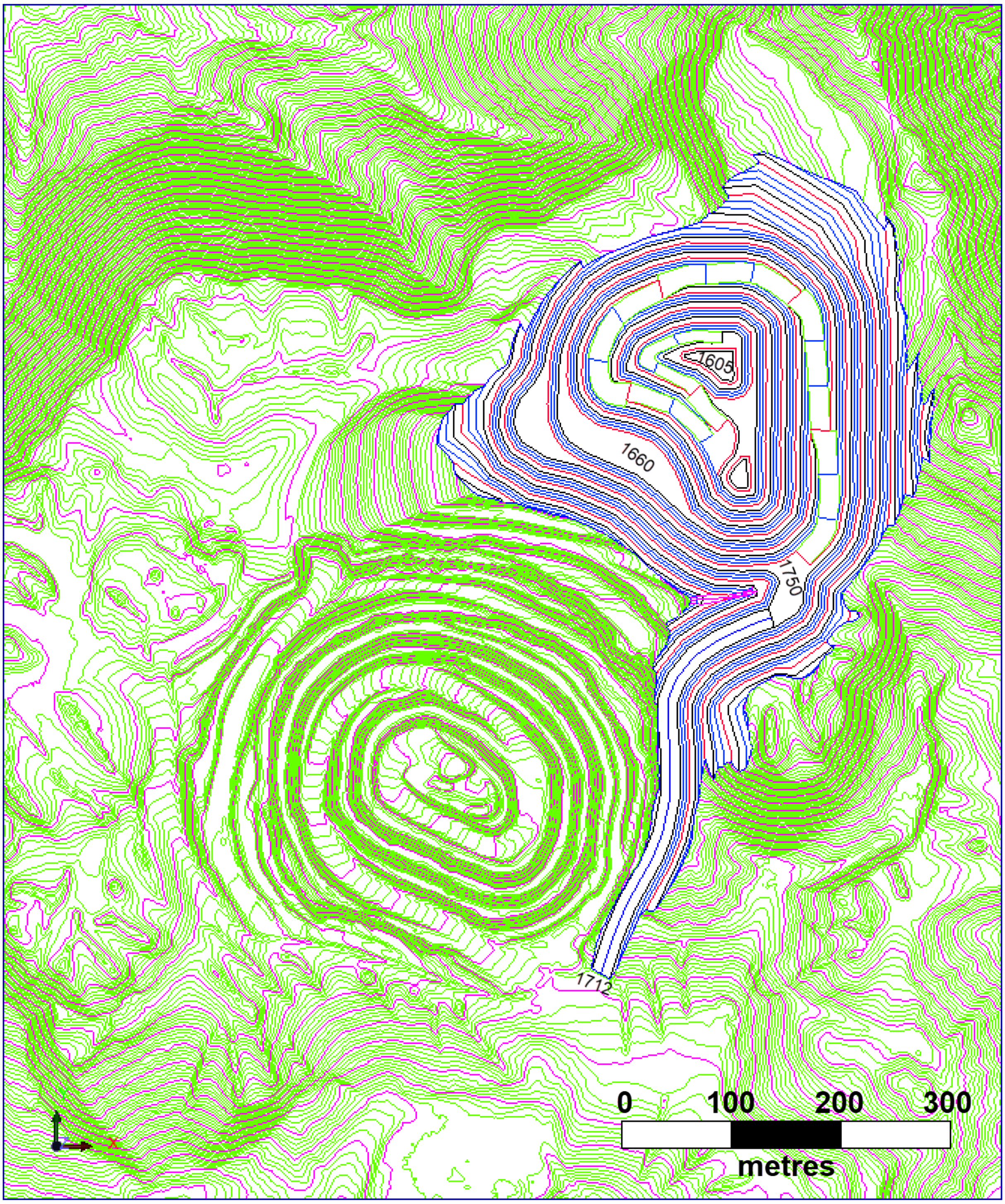

Figure 11: Stage 1 Pit

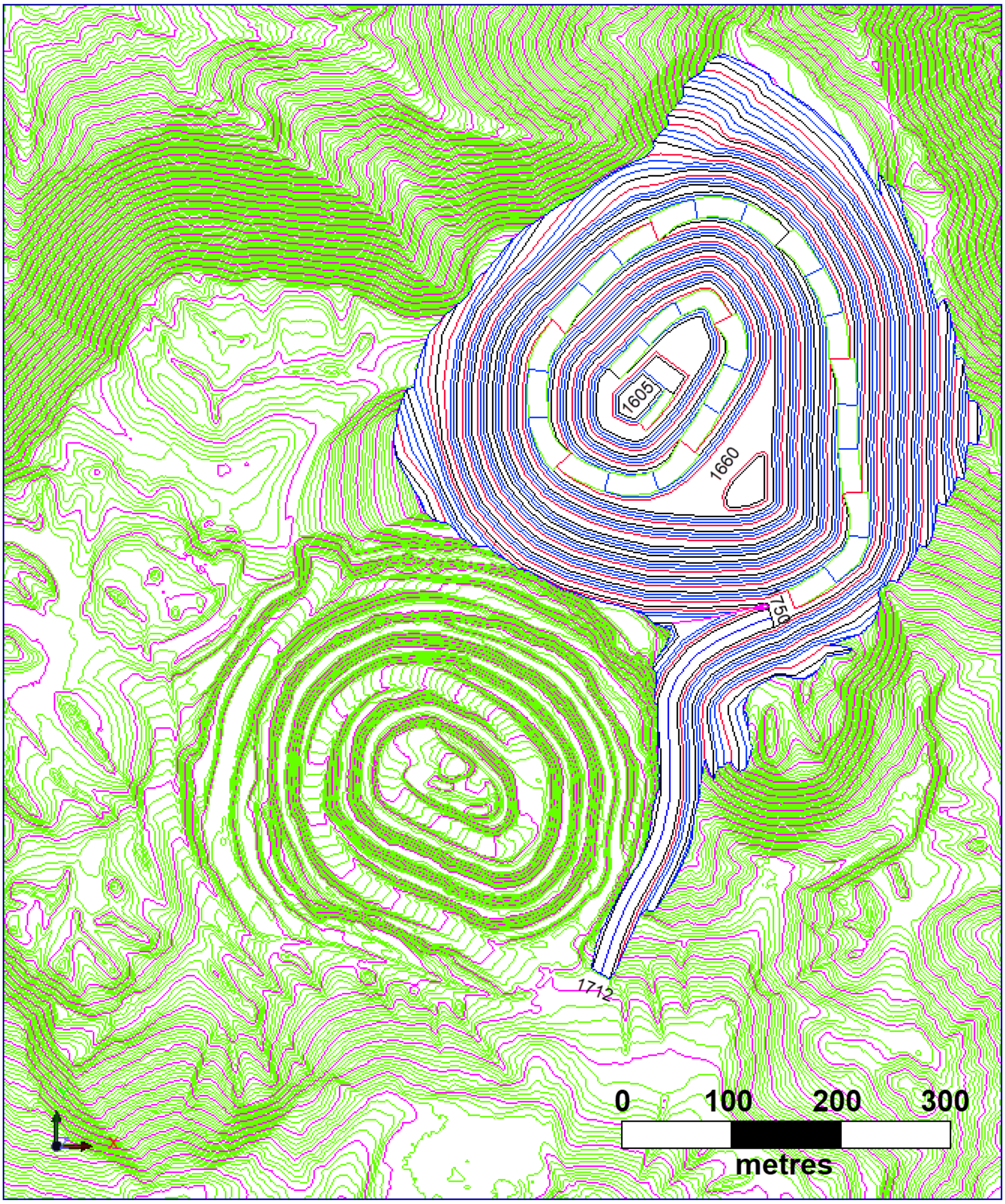

Figure 12 : Stage 2 Pit

Figure 13: Stage 3 Pit

Waste Rock Emplacements (WRE)

Waste rock emplacements for the proposed opencut mine would differ from the 1988 to 1992 operation in that they would be:

- Placed from bottom up rather than from a high tip head so the final landforms can meet design criteria,
- Formed in 10-metre high lifts with 1 in 2 slopes on the final faces separated by 10 metre wide berms to give an overall final slope of 1 in 3,
- Placed to conform more with the surrounding terrain,
- More compacted resulting in lower permeability and reduced total volume,
- Easier to access for rehabilitation work,
- Better able to retain soil, vegetation and moisture on the berms and shallower faces, and
- More amenable to access and habitation by wildlife.

The WREs will be totally within the Unpatented Claims areas and will be designed to facilitate drainage management, erosion controls and containment of any silt run off.

Because the emplacements will be formed from the bottom up, topsoil can be harvested in advance of waste rock placement and placed directly on final faces as they are completed.

Tailings Co-disposal

In order to minimise the overall impact of the project on the site, the Study proposes co-disposal of filtered tailings with the waste rock. The tailings would be filtered to 85% solids by weight and backhauled to the waste rock emplacements by the mine haul trucks. Further work would be required in the DFS to confirm technical, environmental and commercial viability of co-disposal but if proven it offers the following advantages over dam style tailings storage facilities (TSF):

- Greatly reduced surface disturbance because tailings take up void spaces within the body of a WRE instead of lying flat in a valley containment,
- Less prone to wind and rain erosion of the fine tailings material,
- Reduced initial cost because there is no need to build an engineered TSF wall and drainage system, and
- Lower risk of failures impacting downstream ecosystems.

Details of filtered tailings placement with the waste rock would require further planning but in general would entail:

- De-toxification of the tailings to reduce cyanide content to lower than the currently approved limit for storage in the TSF.
- Filtering to 15% moisture.
- Back hauling in mine trucks to the waste rock emplacements.
- Placement, mixing and encapsulation of the tailings in the body of the waste rock emplacement well away from final faces.

All of the WRE would include downstream silt traps and collection dams to capture erosion and water from the WREs. These would be monitored in a similar manner to the existing dams in Curtis Canyon below the former operation TSF.

WRE Design Criteria

Proposed designs for the WREs seek to create stable landforms amenable to revegetation and access to native fauna. Wherever possible they would be placed on currently disturbed ground so the net impact would be an improvement to the current existing WREs.

Design parameters for the WRE's include methodology for estimating the required storage volume and shape of the final landforms.

Table 7: Waste Rock Emplacement Design Criteria

Parameter	Units	Value
Waste rock final swell in dumps		
Fill		10%
Rock		30%
Filtered tailings		
Solids density	w:w	85%
Particle density	t/m3	2.80
Placed volume per dmt	m3	0.534
Final Landform		
Lift height	m	
Face angle	V in H	1 in 2
Berm width	m	10
Overall slope	V in H	1 in 3
Ramp width	m	18.00
Ramp grade		10%

The volume of filtered tailings is based on the dry volume of tailings plus the volume of water to bring it up to 15% moisture by weight.

WRE Design

Four WREs are proposed for the ‘Open Pit Only’ scenario.

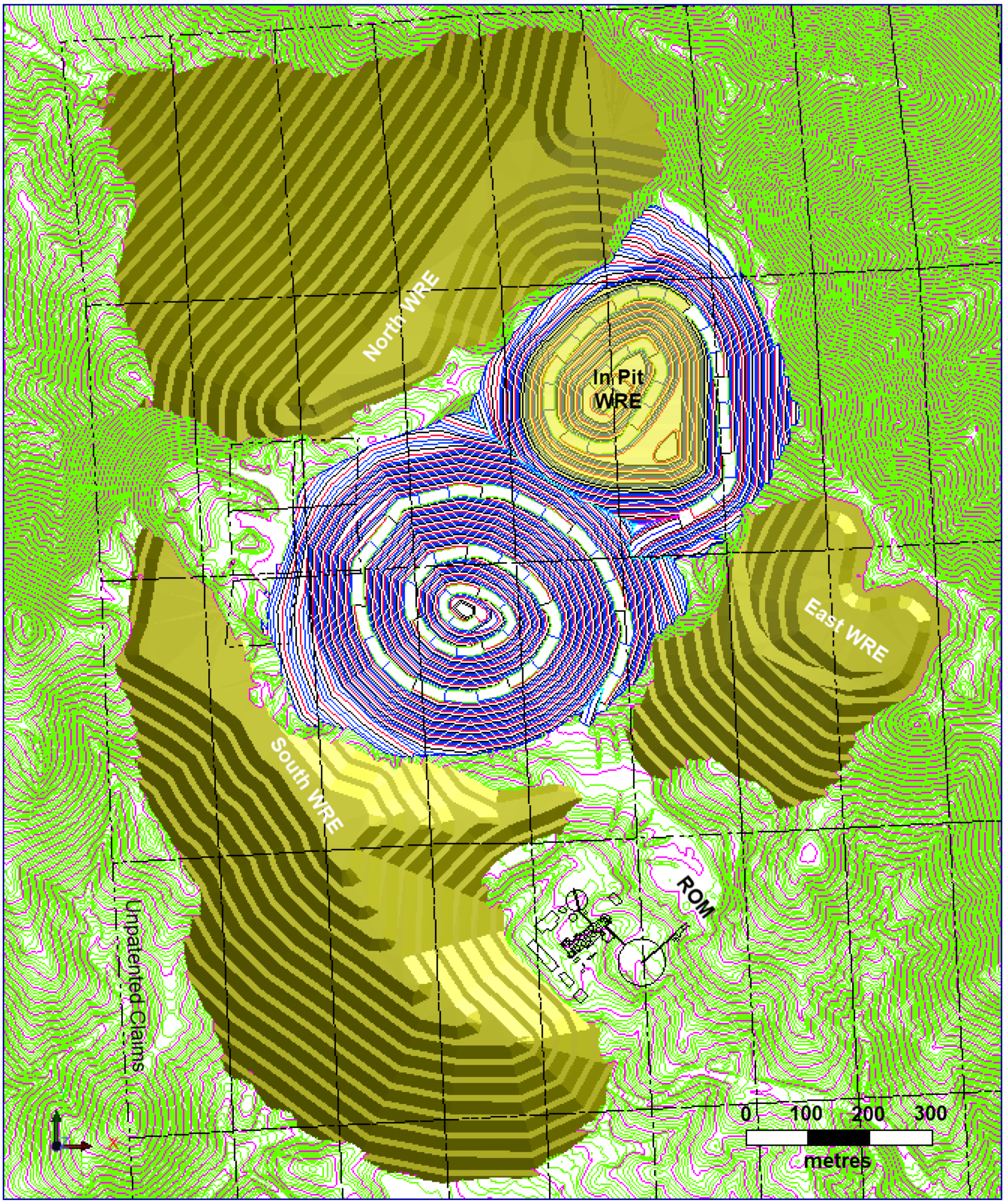

Figure 14: Waste Rock Emplacements

The East WRE would hold all the waste rock from the upper benches of Stages 1 and 2 from the North Pit with access to the lower lifts through the site of the existing low-grade stockpile on the east side of the existing South Pit.

As the North Pit becomes deeper a ramp would be cut into the top of the eastern wall of the existing South Pit to haul mill feed to the ROM area. When the East WRE is full, waste rock would be hauled north west across the ridge between the North and South Pits to the North WRE. This access would be maintained as the South Pit gradually cuts into the North Pit.

As mining progresses in the South Pit pushback of existing void, the South Pit ramp would be designed to join into the ROM access ramp from the North Pit and use the same pit exit point. Waste rock from the South Pit would be placed in the South WRE along with waste rock from the bottom benches of the North Pit once the North WRE is full.

Mining in the North Pit would be completed before the South Pit. The South Pit would still require over 3 Mm^3 of waste rock storage after completion of the North Pit. All of this would be placed in the mined out North Pit.

Opencut Mining Fleet

The proposed opencut mining fleet would be based on hydraulic excavators in backhoe configuration loading haul trucks. For the purpose of the DFS articulated haul trucks are assumed as opposed to rigid body units. This is because they can run at similar speeds on narrower, steeper ramps and work on narrower benches. The load and haul fleet would be supported by bulldozers, graders, water carts and blast hole drills and charging equipment plus a range of smaller equipment such light vehicles, pit pumps and lighting plants.

Mining Rosters

Mining was scheduled on the basis of 2 x 12 hour shifts per day, 7 days per week for a maximum of 14 shifts per week. Potential work hours for each machine type were estimated against this roster after allowance for public holidays, weather stoppages, on-shift delays such as shift and meal breaks or wait on blasting, machine mechanical availability and machine utilisation.

Three possible rosters were assumed:

- 14 shifts per week (2 x 12 hrs, 7 days per week)
- 10 shifts per week (2 x 12 hrs, Monday to Friday)
- 5 shifts per week (1 x 12 hrs, Monday to Friday)

Allowing for some alternation between these rosters over the mine life assists in matching fleet capacity to required mining rates.

Load and Haul Fleet

Trial schedules provided an estimate of required mill feed and waste rock mining rates. Loader/truck combinations were tested to define matches with the required productivity. Selection was also governed by the following assumptions:

- For most of the mine life there should be at least two excavators in operation to help ensure continuity of mill feed.
- For efficient operation at the required dig rates in blasted hard rock, the excavator should be at least in the 90 tonne class.

The loader/truck combinations selected are:

Table 8: Loader Truck Combinations

		Open Pit Only (min/max)	
Excavator operating weight	tonnes	94	120
Truck payload	tonnes	55	55
Dig rate per Excavator	tph	650	1116
Number of Excavators		1	1

Haulage of mill feed, and waste rock is typically the largest component of the mining cost because it requires the largest number of units and the largest number of operators and maintenance crew. Truck fleets were modelled for the Study by defining mill feed and waste rock haul routes from each mining bench to the ROM or WREs and using the haul modelling tools in MineSched software to estimate speeds and truck hours through each period over the mine life. The software tracks growth of the WREs 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.

A further adjustment was made to the truck fleet to allow for haul back of filtered tailings to the WREs.

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 Study mitigates fixed ownership costs by selecting loader / truck combination which match the required production rates as closely as possible.

Labour fixed costs are controlled by alternating between rosters. This is probably only realistic if a mining contractor is used so the workforce can be allocated to other work during periods of 10 or 5 shift rosters.

Blast Hole Drilling and Charging Fleet

Track mounted rotary percussive drills were selected based on the powder factor and blast hole sizes reported from the former operation. The assumed blast pattern uses 152mm diameter holes blasting 5-metre high benches. Drilling rates per metre are based on similar operations and include moving and set up between holes.

Explosives supply was assumed to be a ‘down the hole’ service so the charging unit would be provided by the explosives supplier.

Support Fleet

The load and haul fleet would be supported by:

- Face bulldozers for clean-up and bench levelling around the excavators.
- WRE bulldozers to spread truck dumped waste rock and tailings and form final WRE faces.
- A grader and water truck to maintain haul roads, benches and stockpile areas and to control dust.
- Lighting plants for night operation.

- Pit pumps to remove ground and rainwater from the mining benches.
- A range of other intermittently used equipment such as light vehicles, service trucks, tip truck, small loader and a crane

The Study estimates operating hours, fleet sizes, operator numbers and operating costs for all the major equipment items and applies contingency factors to allow for the smaller intermittently used items.

Tailings / Stockpile Loader

Operating hours were estimated for a large wheel loader to load truck with filtered tailings to back haul to the WREs and to load the crusher from stockpile in periods when the trucks aren't able to direct dump.

The following table shows examples of the types of machines adopted for the Study.

Table 9: Examples of suitable machinery used in the Study for Case 2 (Open Pit Only)

Unit type	Max number	Example
Excavator 110t – 120t	1	Komatsu PC1250, 107t, 7.0m^3
Excavator 90t	1	CAT 395, 94t, 4.1m^3
50t – 60t Truck	12	Volvo A60H, 55 tonne payload, 33.6m^3, 495kW
40t – 50t Truck	0	CAT 745, 41 tonne payload, 26 m^3
Blasthole Drill	2	Epiroc FlexiROC D60 down-the-hole hammer drill 110 mm - 178 mm
Bulldozer	2	CAT D8
Grader	1	CAT 140
Water Truck	1	CAT 745
Loader	1	CAT 988

Load Haul Fleet

Trial schedules for the 'Open Pit Only' scenario showed a good match to required material movement using 1 x 120 tonne excavator and 1 x 94 tonne excavator both loading 55 tonne payload trucks.

The project would operate on a full 14 shift per week roster for most of the mine life. There would be a 15-month period, mostly in Year 2, where the roster drops to 10 shifts per week. By Year 7, mining would be near the base of the pits where the waste to ore ratio is low so the roster would drop to 10 shifts per week and then 5 shifts per week over the final year.

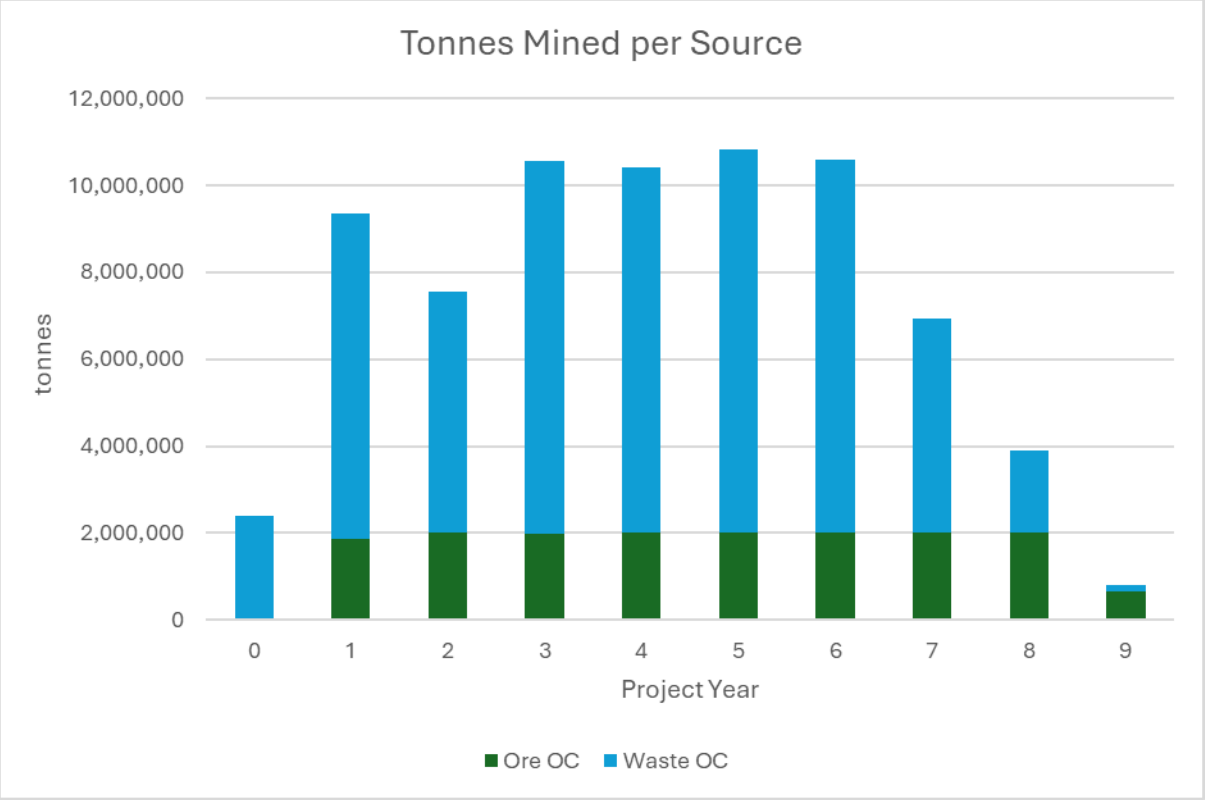

Figure 15: Annual Material Movement

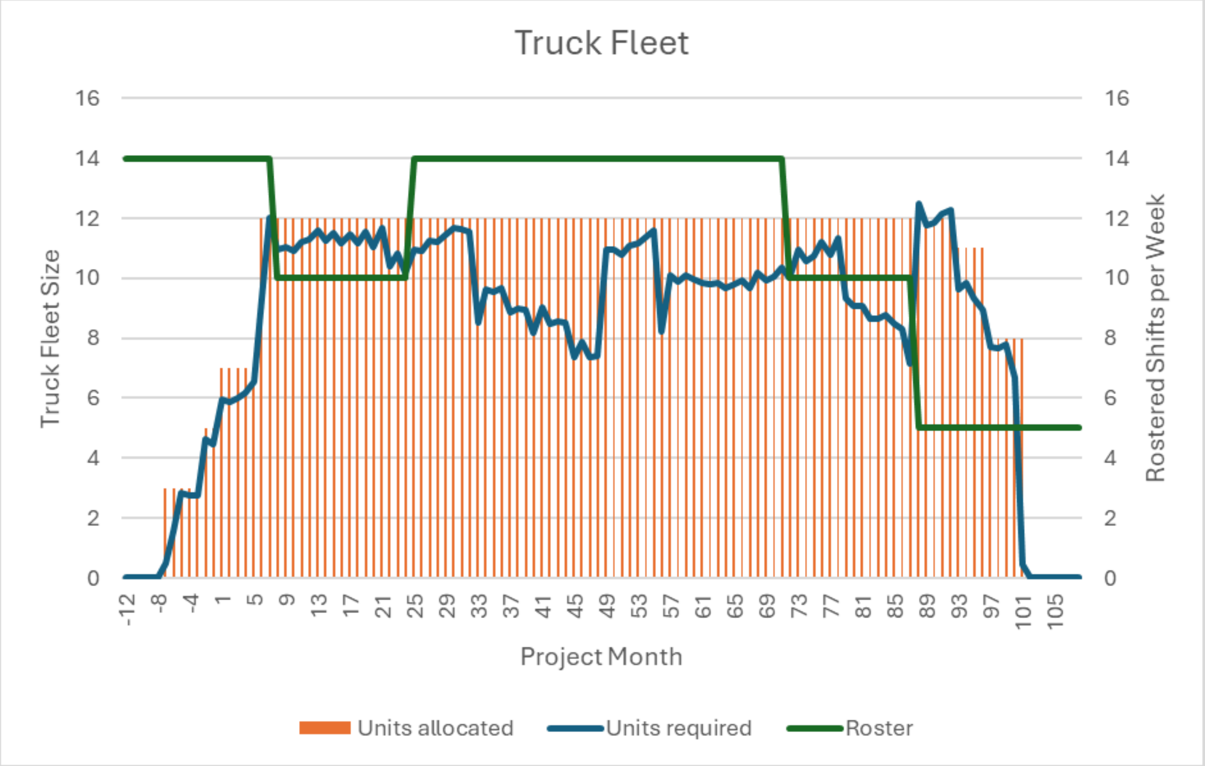

Figure 16: Truck Fleet

The truck fleet would build to a maximum of 12 units and be held at this level for most of the mine life. The excess capacity of the fleet would be used for hauling filtered tailings back to the WREs.

Production Schedule

Underground and opencut mining and mineral processing were scheduled on a monthly basis to assess practical sequencing issues such as underground development in advance of stoping, opencut bench advance rates and run of mine stockpile sizes. Production quantities and grades were summarised annually as shown in Table 10.

Open Pit Only Scenario

Mining would commence in Stage 1 of the North Pit in Month 6 of Year 0. Most of the opencut mining in Year 0 would be waste rock. Opencut mill feed production would increase from Month 1 of Year 1 to meet the full process feed rate of 2.0 million tonnes per year.

Stage 2 of the North Pit would commence early in Year 1 so that it would be deep enough to supply the majority of the mill feed when Stage 1 is completed in mid-way through Year 4.

The South Pit would commence mid-way through Year 3 so the pushback to the existing South Pit would be deep enough to access the mill feed below that pit by the time the North Pit is completed mid-way through Year 7. Mining would then continue in the South Pit until the middle of Year 9.

Mill feed delivery rates from the opencut mine would be matched to the target processing rate of 2.0 million tonnes per year. A ROM stockpile would be maintained adjacent to the crusher feed hopper as a buffer between mining and processing. The ROM stockpile size would be between 20 to 30 thousand tonnes over the mine life. This represents approximately five days of feed to the mill.

Total opencut mining quantities for Case 2 would be 16.6 million tonnes at 1.30 g/t Au and 56.8 million tonnes of waste rock.

Processing was scheduled at 2.0 million tonnes (2.2 million tons) 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. Estimated gold production over the first six years would average 67 koz per year. The last 18 months of production would have much higher-grade mill feed from the base of the South Pit resulting in over 100 koz of gold per year.

Total estimated gold production over the mine life at the estimated head grade and process recovery would be 635 koz.

Table 10: Life of Mine Estimated Production Schedule

Project Year		0	1	2	3	4	5	6	7	8	9	10	11	12	Total
Underground Mine															
Mill Feed	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gold grade	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Waste tonnes	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total tonnes	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Opencut Mine															
Mill Feed	kt	34	1,872	2,003	1,993	2,002	2,001	1,997	2,004	1,997	648	0	0	0	16,551
Gold grade	g/t	1.00	1.18	1.26	1.06	1.05	1.12	1.11	1.31	1.91	2.51	0.00	0.00	0.00	1.30
Waste tonnes	kt	2,346	7,471	5,553	8,564	8,426	8,833	8,604	4,925	1,892	142	0	0	0	56,755
Total tonnes	kt	2,380	9,343	7,557	10,557	10,427	10,834	10,601	6,928	3,889	790	0	0	0	73,306
Total Volume	bcm	928	3,609	2,929	4,111	4,100	4,100	4,001	2,614	1,467	298	0	0	0	28,158
Waste:Mill Feed Ratio	t:t	69.1	4.0	2.8	4.3	4.2	4.4	4.3	2.5	0.9	0.2	0.0	0.0	0.0	3.4
Total Mined															
Mill Feed	kt	34	1,872	2,003	1,993	2,002	2,001	1,997	2,004	1,997	648	0	0	0	16,551
Gold grade	g/t	1.00	1.18	1.26	1.06	1.05	1.12	1.11	1.31	1.91	2.51	0.00	0.00	0.00	0.00
Waste tonnes	kt	2,346	7,471	5,553	8,564	8,426	8,833	8,604	4,925	1,892	142	0	0	0	56,755
Total tonnes	kt	2,380	9,343	7,557	10,557	10,427	10,834	10,601	6,928	3,889	790	0	0	0	73,306
Stockpile Reclaim	kt	0	375	400	400	400	400	400	400	400	135	0	0	0	3,310
End of year ROM Stockpile	kt	34	31	35	27	29	30	27	31	28	0	0	0	0	
Process Plant Feed	kt	0	1,875	2,000	2,000	2,000	2,000	2,000	2,000	2,000	676	0	0	0	16,551
Gold grade	g/t	0.00	1.18	1.26	1.06	1.05	1.12	1.11	1.30	1.89	2.51	0.00	0.00	0.00	1.30
GOLD PRODUCED	**koz**	**0**	**66**	**74**	**63**	**62**	**66**	**65**	**77**	**112**	**50**	**0**	**0**	**0**	**635**

Production Schedule by Resource Category

The production targets for this study include 19% Inferred resources, which have low geological confidence. The following figures show tonnes mined by resource category to assess the resource risk.

Inferred mineral resources make up 19% of the estimated mill feed tonnes.

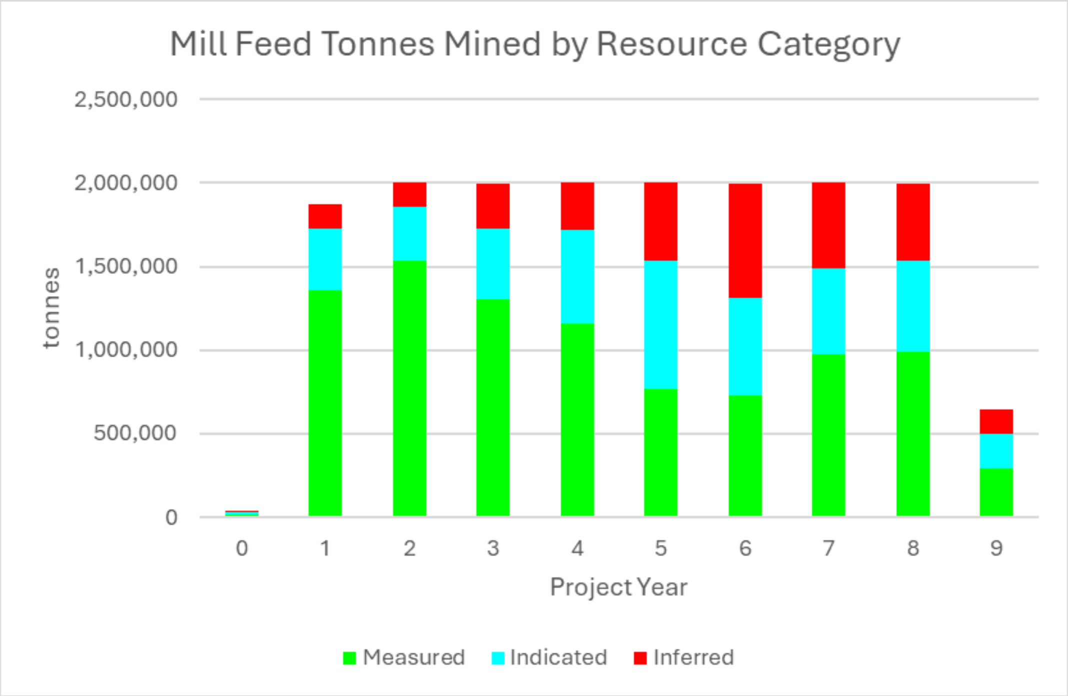

Figure 17: Tonnes Mined by Resource Category

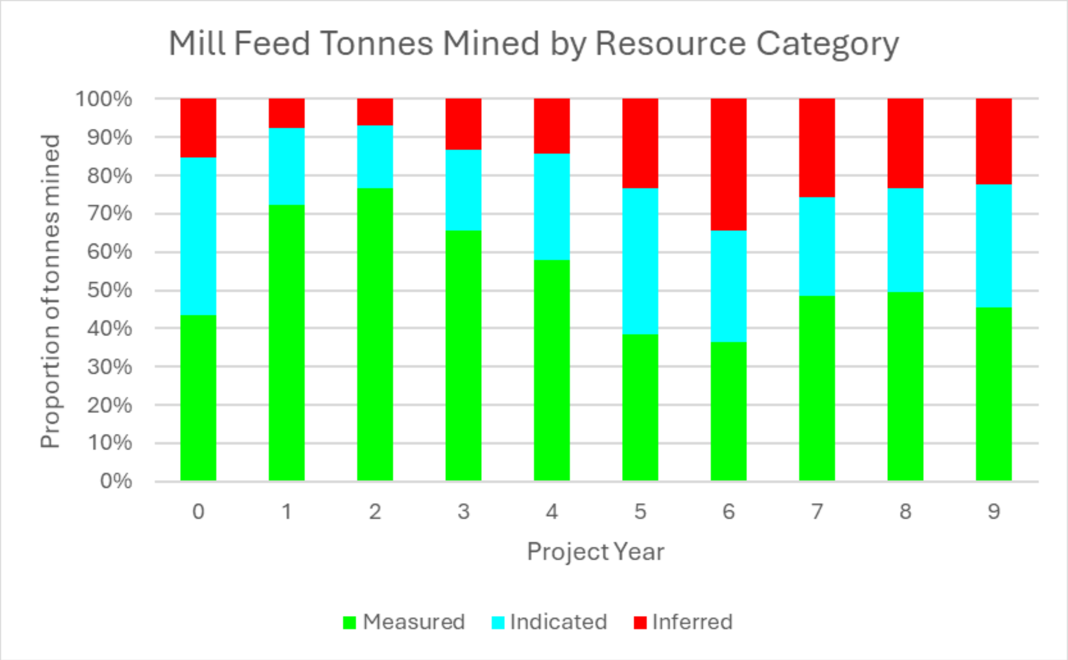

Figure 18: Proportion of tonnes Mined by Resource Category

4. Metallurgy

Gold Recovery

This study assumes a gold process recovery of 92% based on:

- Life of mine production records showing an average of 91% recovery.
- 1984 metallurgical test work results averaging 91.4%
- Assumed improvements in gold processing since 1993.

Table 11: LAC Minerals Inc. Operating Statistics

Statistic	Total
Tonnes Ore Mined	6,594,222
Grade Ore Mined (g/t)	2.07
Tonnes Waste Mined	26,180,306
Total Tonnes Mined	34,757,182
Stripping Ratio	3.97 : 1
Tonnes Ore Milled	5,841,328
Grade Ore Milled (g/t)	2.018
Gold Poured (oz)	344,691

(Source: Lac Minerals Inc., Operating Information Sheets)

Operational records do not include details on ore types processed from 1988 to 1993. Given the location of mining it is likely most of the ore processed was oxide. The drill hole data and resource models available to AMDAD at September 2024 do not provide detail on degree of oxidation in the proposed mine plan. It is possible that it may include partially oxidised or fresh sulphides. The 1984 Amselco Feasibility Study referred to metallurgical test work by Hazen research which included oxide and sulphide mineralisation. The report states “whole ore cyanidation produced high recoveries on both sulphide and oxide ores”.

Some of the reports mention copper in the mineralisation and the 1980s test work was to have included analyses to mitigate excessive cyanide consumption. However, the Amselco feasibility Study notes “results of test work at Hazen indicated copper would not be a problem”.

Other Process factors

A future DFS would need to conduct metallurgical test work to confirm the gold recoveries, grinding characteristics, reagent consumption and other process factors for the ore to be processed in a future mine. Available production records are not comprehensive enough for detailed process design and there is no certainty that the deeper mineralisation will perform the same as ore processed from 1988 to 1993.

5. Mineral Processing

Gold Processing Plant

This Study does not include any analysis or assessment of the gold processing plant. All processing assumptions are based on performance of the carbon in pulp (**CIP**) plant that operated successfully at Colosseum from 1988 to 1993. That plant had a design capacity of 3,400 tons per day and ran at an average throughput of 1.2 Mtons (1.09 Mtonnes) per year.

The DFS would examine process options in detail to incorporate improvements in gold processing technology since 1993.

The main change assumed for the Study is that all tailings would be filtered to 85% solids by weight to facilitate co-disposal with the mine waste rock.

In order to understand how the process plant would work with the mining operation AMDAD:

- Digitised the layout of the 1993 process plant and placed it in the same position on the proposed mine plans.
- Scaled the footprints of existing process plants in the 1 to 2 Mtpa range and compared them to the footprint of the 1993 Colosseum plant to confirm that process plants for Cases 1 and 2 could fit in the available area.
- Scaled the footprint of a comparable size process plant incorporating a tailings filter to assess the additional area required.

These checks confirmed at a scoping level of confidence that a 1 or 2 Mtpa process plant with a tailings filter can be accommodated in the available area at Colosseum.

The following figures show the flowsheet and layout of the 1993 process plant and how it would fit into the mine plan proposed for the PDS.

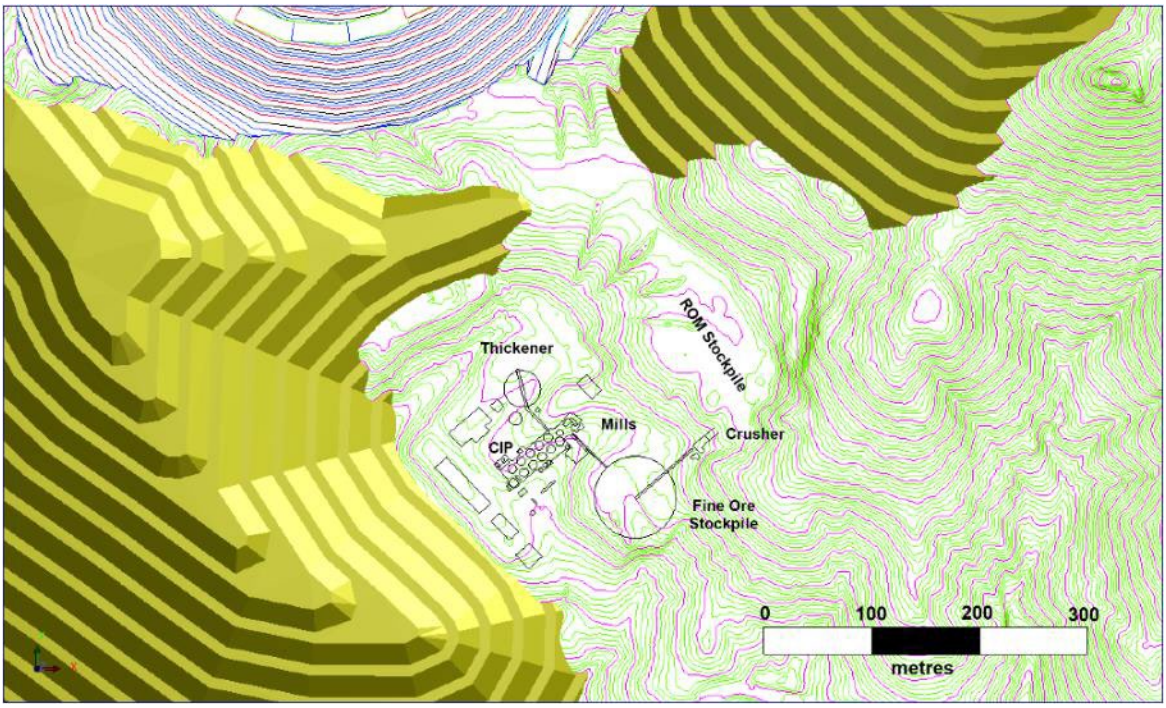

Figure 19: 1993 process Plant Overlaid on Mine Plan

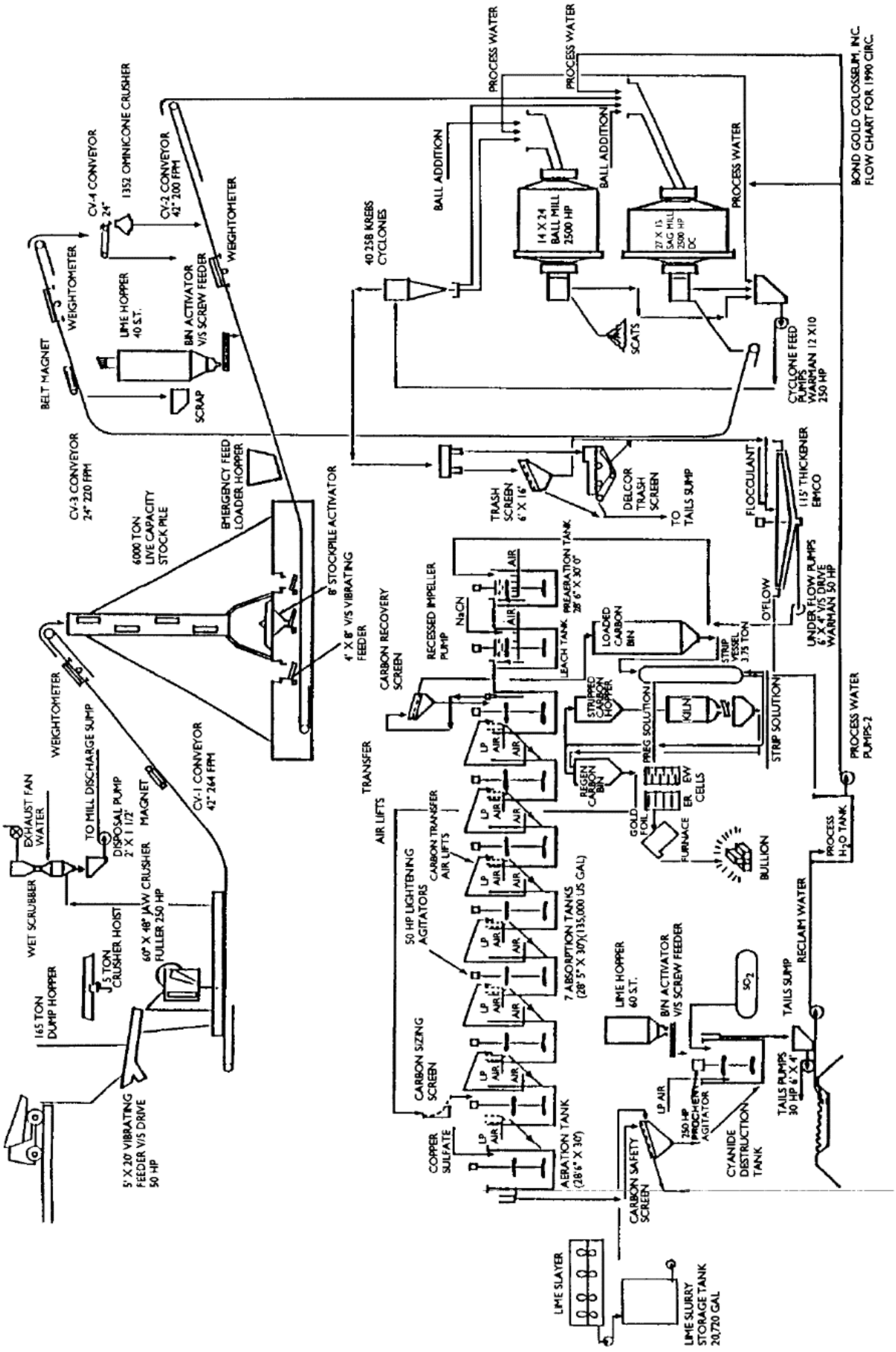

Figure 20: CIP Process Flow Chart 1988 to 1993

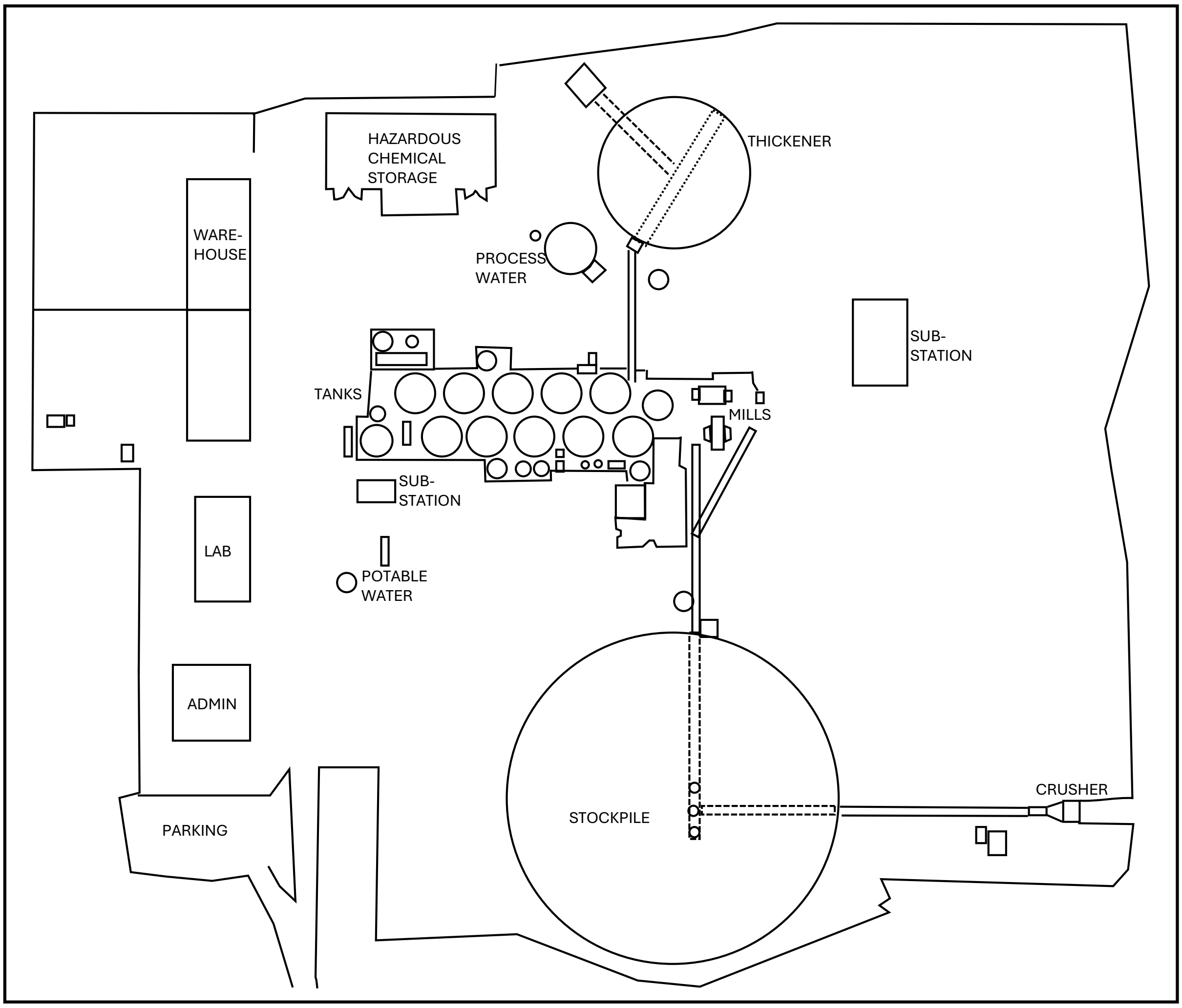

Figure 21: Schematic process Site Layout 1988 to 1993

Tailings Management

The Study assumes that process tailings will be filtered to 85% solids and trucked back to the WREs for co-disposal with the mine waste rock. Tailings and waste rock analyses for the 1982 EIS concluded that with cyanide destruction neither the tailings or waste rock present any danger to surrounding water sources. The tailings and waste rock were assigned as Class 3 waste.

Detailed assessment of co-disposal methods and environmental impact would be required in the DFS. However, the general procedure would be:

- Thicken the tailings in a standard thickener to around 55% to 60% solids by weight.
- Use a belt or plate press filter to increase the solids density to 85%. At this density the tailings filter cake should be handleable by loaders and trucks.
- Stack filtered tailings in a sheltered bay to prevent erosion by wind or rain.
- Use a large wheel loader to periodically load the filtered tailings onto mine trucks which would divert to active WRE benches on route back to the mining faces.

- Dump the filtered tailings on the active WRE bench but well back into the body of the WRE away from future final faces.
- The dumped tailings would be encapsulated within the mine waste rock. This may involve methods such as formation of cells in the WRE benches which are periodically buried in waste rock or paddock dumping of alternate waste rock and tailings piles which are then mixed together by the WRE bulldozer.

Tailings placement would be designed to make the fine tailings take up void spaces in the placed waste rock to reduce permeability of the WRE and fully encapsulate the tailings in waste rock to ensure it is not open to erosion by wind or rain.

6. Infrastructure and Services

Accessibility

The site is accessed by 16.5km of road from Interstate Route 15 (Figure 22). The first 6.2km is sealed and the remaining 10.3km is unsealed.

Proximity to significant locations:

- Las Vegas, Nevada is 76km north along Interstate Route 15 from the Colosseum turn off.
- Ivanpah Solar Facility straddles the Colosseum access road. This 392 MW solar farm includes molten salt heat storage which, combined with supplementary natural gas generation, delivers 24 hour per day electricity supply. A potential power line route from the Ivanpah substation to Colosseum is approximately 10.4km.
- Primm, Nevada is 8km north along Interstate Route 15 from the Colosseum turn off. It is a resort town just inside the Nevada state line.
- Jean, Nevada is 28km north along Interstate Route 15 from the Colosseum turn off. Jean hosts a commercial warehousing facility and light aircraft airport.
- The Mountain Pass rare earths mine is 10km south of Colosseum.

Figure 22: Colosseum Gold Mine access

Power Supply

The former operation accessed grid power from a 33 kV powerline owned by Southern California Edison (**SCE**). There are several high voltage lines (500 kV and 287 kV) 3 km north of the mine, but it may not be feasible to take supply from these. The Ivanpah Solar Facility is located in the Ivanpah Valley 8 km west of the mine. A sub-station at the facility is linked into a 115 kV powerline owned by SCE.

For the purpose of the Study, it is assumed that a new 60 kV powerline will be built from the Ivanpah sub-station to the mine site.

Water Supply

The former operation extracted water from wells in the Ivanpah Valley. AMDAD is not aware if this source is still available. Other aquifer alternatives discussed in the original EIS include Shadow Valley 16 km to the west and Mesquite Valley 30 km to the north.

Employee Accommodation

The Study assumes Colosseum employees will commute daily from their own accommodation. No allowance has been made for permanent employee accommodation and messing.

Other

Other facilities at the mine site would include:

- Mine and process plant workshops and stores,
- Diesel fuel storage,
- Explosives magazine,
- Emergency diesel generator,
- Offices including communications facilities,
- First aid facility,
- Helicopter pad (for emergencies),
- Change house and ablutions,
- Sewage treatment,
- Environmental monitoring and controls, and
- Site security.

Figure 23: Site Access

7. Market Studies and Contracts

Gold Price Forecast

A fixed gold price of US$2,200/oz is used for the Study. This is based 40% on the spot gold price of US$2,500/oz at the start of September 2024 and 60% on the long-term average of publicly available forecasts from reputable sources.

Table 12: Published Gold Price Forecasts

Source	2024	2025	2026	2027	2028
S&P Global	$2,119.97	$2,134.02	$2,075.63	$2,042.84	$2,040.92
CIBC Capital Markets	$2,290.00	$2,600.00	$2,400.00	$2,200.00	$1,975.00
World Bank	$2,100.00	$2,050.00			
JP Morgan	$2,462.50	$2,555.00			
Trading Economics		$2,549.74			
ING	$2,350.00	$2,300.00	$2,240.00		
Average	**$2,264.49**	**$2,364.79**	**$2,238.54**	**$2,121.42**	**$2,007.96**

Gold would be sold in doré produced on site. The drillhole assays and the production records do not show any metals other than gold. It is likely the doré would contain some silver, but the Study does not assign and credits for this.

8. Approvals and Sustainability

Environmental and Social/Community impacts for Colosseum were addressed for the operation by a Draft 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. EPA (1993). Mining and processing from 1988 to 1993 were run 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 operation were suspended in 1993 the EIS and Plan of Operations remained in place and would apply to resumption of activities at Colosseum. These documents would be DTR's minimum standard for the project.

9. Capital and Operating Costs

Basis of Estimates

Capital and operating costs in this Scoping Study are estimated at an order of magnitude level of confidence. The following table summarises the basis of capital and operating cost estimates in each project area.

Table 13: Basis of Cost Estimates

Area	Capital Cost	Operating Cost	Source
Opencut Mine	US cost database for fleet. Pre-production costs from operating costs. Facilities included in Site Infrastructure.	Unit costs from US cost database applied against fleet, workforce and consumables estimates.	Material take-offs from Study production schedule. Unit costs from US cost database.
Process Plant	Cost per tonne of throughput based on benchmarked costs. Additional cost for filtered tailings.	Cost per tonne of mill feed based on benchmarked costs. Additional cost for filtered tailings.	Published studies and company reports.
Site Infrastructure	Cost per tonne of throughput based on benchmarked costs.	General and administration cost as function of process operating cost based on benchmarked projects.	Published studies and company reports.

Opencut Mine

Mining fleet capital and hourly operating costs, labour rates and explosives costs are taken from a US mining cost database.

Diesel fuel costs are published costs for Eastern California for August 2024.

Explosives "down the hole" service charges are based on current Australian contracts.

Fleet productivity estimates and haulage models for the Case 1 and Case 2 production schedules were used to develop monthly equipment hours. These were used to estimate fleet sizes and operator numbers. The maintenance workforce was assumed to be 0.6 x operator numbers.

Blasthole drill hours, fleet and workforce numbers and explosives consumption was estimated for a blast pattern in non-fill material based on the powder factor used from 1988 to 1992.

Fleet ownership costs are based on 60-month lease terms with a 10% residual to take ownership of each unit.

Fleet operation, including operator, maintenance and supervisory labour costs, but excluding explosives costs, was assumed to be by a contractor and a 15% margin was added to these costs.

Mining fleet costs include approximately 10% contingency for equipment and 5% for labour to cover items such as light vehicles, service trucks, minor earthworks machines and a crane which are not fully utilised and do not have full time operators.

Explosives supply was assumed to be on a “down the hole” basis and a monthly service fee was applied for provision of the explosives charging truck and crew.

Mine management, technical and supervisory staff were estimated appropriate for the operation. Staff costs are taken from published West Coast USA salary data.

Additional costs estimated include grade control and environmental management.

Process and Infrastructure

Process and infrastructure capital and operating costs were estimated by reviewing published studies and company reports and the US cost database cost models to define costs as a function of process plant feed rate.

Benchmarking of capital costs included **16 projects** with sufficient published information on the breakdown between process plant, tailings, infrastructure and indirect costs (EPCM, Owner’s costs and contingency) to be adjusted for direct comparison to Colosseum. Four feed rates from the cost database were used.

Benchmarking of operating costs included **38 projects** and four from the cost database. General and administration costs were estimated as a function of the benchmarked process operating cost.

The benchmarked study and report dates range from 2017 to 2024 and were in US, Canadian or Australian dollars. They were adjusted to 2024 using published global inflation rates and then to US$ using current exchange rates at August 2024.

Tailings facility costs were removed from the process costs and capital and operating costs for filtered tailings were added back. The filtered tailings capital and operating costs are based on a Canadian project currently in development.

The following table summarises the benchmarked capital and operating costs adjusted to a filtered tailings operation at Colosseum in 2024 US$.

Table 14: Summary of Process and Infrastructure Cost Estimates

Annual Mill Feed Rate Mtpa	Capital Cost Process Plant US$M	Infrastructure US$M	Indirects and Contingency US$M	Total Process and Infrastructure US$M	Process Opex with Tails Filter US$/ROM t	Project G&A Opex US$/ROM t
0.5	29.52	9.60	9.60	48.72	30.00	11.00
1.0	49.20	16.00	16.00	81.20	23.00	8.33
1.1	52.89	17.20	17.20	87.29	22.10	7.99
1.5	67.65	22.00	22.00	111.65	20.50	7.34
2.0	79.95	26.00	26.00	131.95	18.75	6.63

Additional process costs were added for:

- A large wheel loader to load filtered tailings onto mine trucks for haul back to the WREs. This includes capital costs (ownership) and operating costs (lease, diesel, maintenance and labour).
- Operating costs for the additional mine truck hours for the tailings haul back. The mine fleet has sufficient capacity to include haul back so no additional trucks were required.

Capital Costs

Opencut Mine

Capital costs for opencut mining include:

- Mobilisation and commissioning charges for each mining fleet item,
- Residual payments for each fleet item at the end of the 60-month lease period, and
- Pre-production (Year 0) operating costs.

Opencut mining fleet lease costs are included in operating costs.

Opencut mine facilities such as workshops are included in Site Infrastructure capital costs.

Table 15: Opencut Capital Cost Estimates

Area	Year 0	Deferred	Total
Mining Fleet	$460,900	$3,914,075	$4,374,975
Pre-production	$12,687,319	$0	$12,687,319
Total	**$13,148,219**	**$3,914,075**	**$17,062,294**

Process and Infrastructure

Benchmarking data for the Process and Infrastructure capital costs presented Table 14 are shown in the following chart. The red triangles show total Process and Infrastructure capital costs assumed for Colosseum at a range of annual process feed rates.

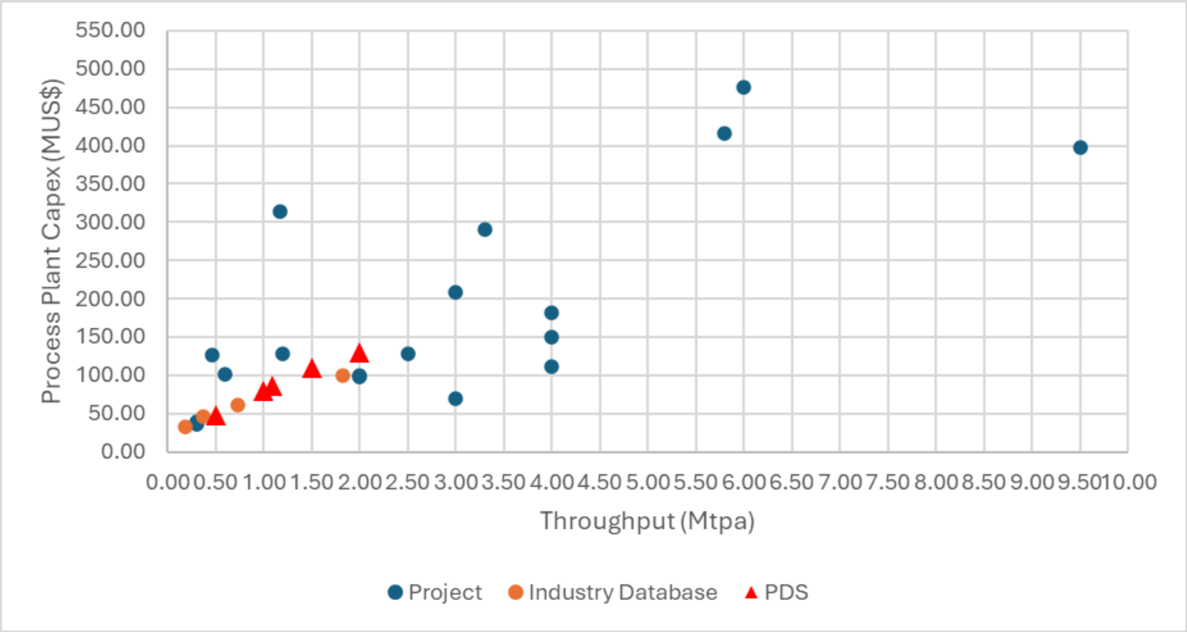

Figure 24: Benchmarking of process and infrastructure capital cost

Additional capital costs specific to Colosseum were estimated:

Powerline and substation	US$4,703,024
Access road upgrade	US$1,030,000

Total estimated construction capital costs for Process and Infrastructure in Year 0 are:

Open Pit Only	US$137,683,024

Sustaining Capital

An amount of 2% of the Process and Infrastructure cost per year was allowed for sustaining capital. Life of project sustaining capital amounts estimated are:

Open Pit Only	US$23,176,642

Mine Closure Costs

Estimated amounts were allowed for mine closure costs:

Open Pit Only	US$30,000,000

These allowances are not based on estimates of the activities involved other than noting that they are well in excess of six months of mining costs at full production. Closure costs will have to cover all activities listed in the Reclamation Plan.

Operating Costs

Opencut Mine

The following tables show the opencut mine operating cost estimates by area and by input.

Table 16: Estimated Opencut Operating Costs by Area

Area	Mill Feed US$/tonne	Waste Rock US$/tonne	Total US$/tonne
Load and Haul	$2.75	$3.04	$2.97
Blast Hole Drilling	$0.43	$0.40	$0.41
Explosives Supply	$0.42	$0.38	$0.39
Grade Control	$0.32	$0.00	$0.07
Management, Technical & Supervision	$0.33	$0.26	$0.28
Environmental Management	$0.02	$0.10	$0.08
Total Mining Cost	**$4.28**	**$4.18**	**$4.20**

Table 17: Estimated Opencut Operating Costs by Input

Input	Life of Mine Cost US$M	US$/tonne	% of Total
Equipment Lease	$39M	$0.54	13%
Diesel & Power	$45M	$0.62	15%
Maintenance	$27M	$0.37	9%
Labour	$102M	$1.39	33%
Contractor Margin	$34M	$0.46	11%
Explosives Supply	$29M	$0.39	9%
Grade Control	$5M	$0.07	2%
Management, Technical & Supervision	$20M	$0.28	7%
Environmental Management	$6	$0.08	2%
Total Mining Cost	**$308M**	**$4.20**	**100%**

Process and Infrastructure

The following chart shows benchmarked data for the basic process cost per tonne of mill feed adjusted to 2024 US$. The red triangles show process operating costs assumed for Colosseum at a range of annual process feed rates.

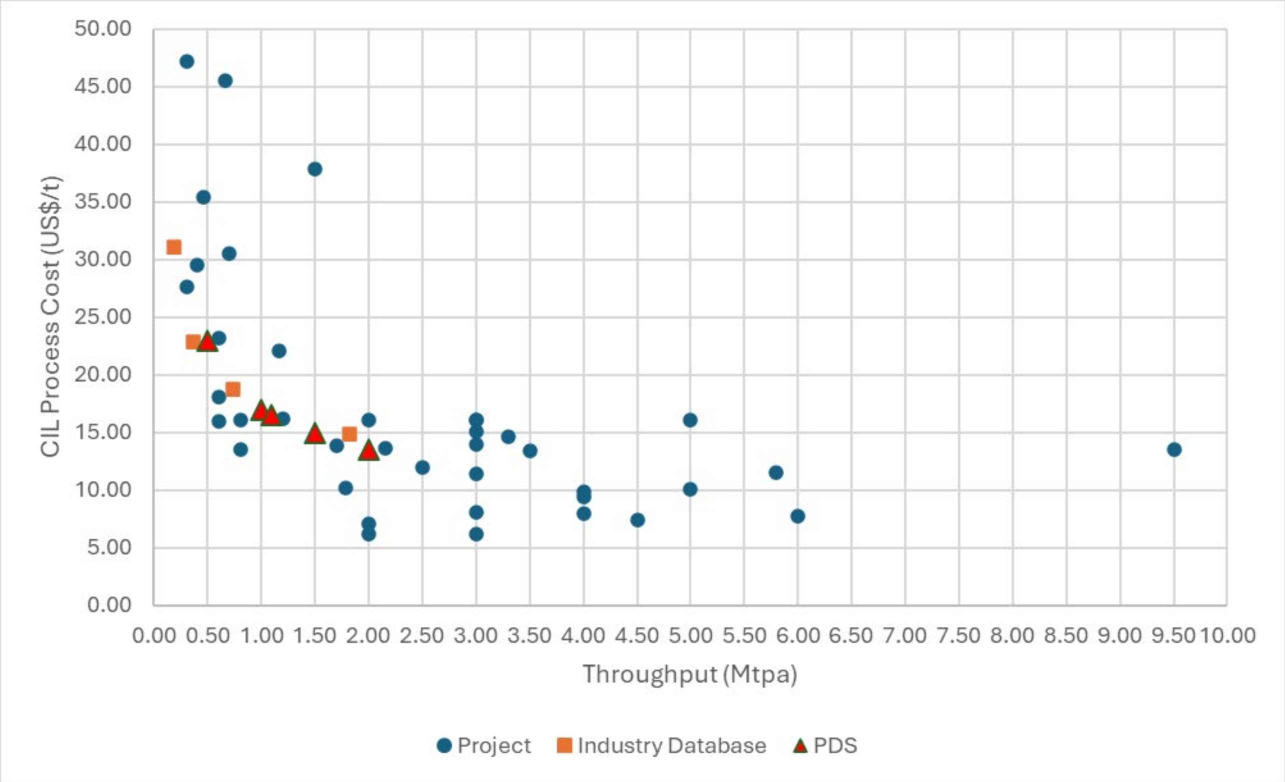

Figure 25: Benchmarking of process operating costs

The estimated cost of operating the tailings filter, which is based on published costs for a similar scale plant, were added to the basic benchmarked process cost and modelled tailings haul back costs per period were added to the total.

Table 18: Estimated Process Operating Costs

Input	Life of Mine Cost US$M	US$/tonne Mill Feed
Processing	$223M	$13.50
Tailings Filter	$87M	$5.25
Tailings Haul Back	$23M	$1.40
Total Process Cost	**$333M**	**$20.15**

General and Administration Costs

General and administration (**G&A**) costs cover items such as operations management, human resources, accounts, safety, community relations, communications and mining tenement costs. These can vary greatly from site to site and consist of a mix of fixed and variable costs. To estimate a representative cost for the Study, G&A costs from the benchmarked studies and reports were plotted against the benchmarked process costs as shown in the following chart.

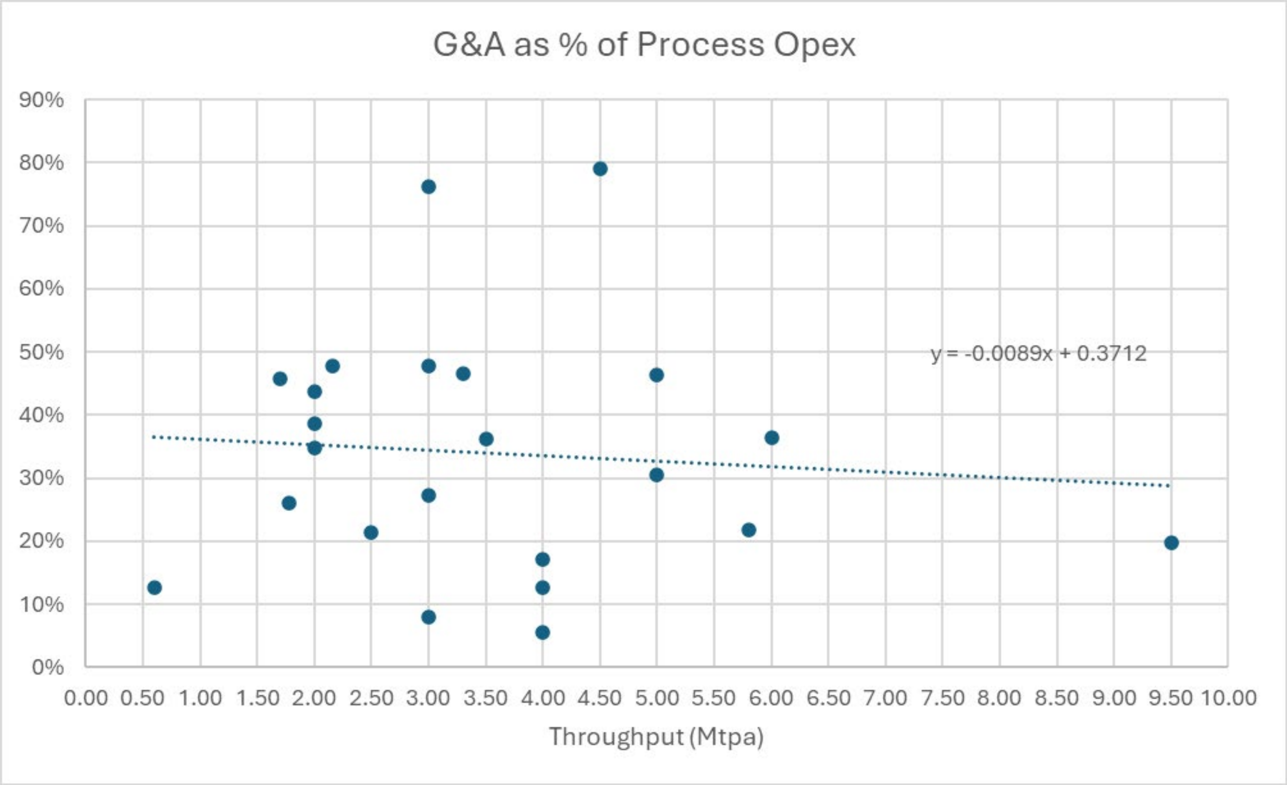

Figure 26: Benchmarking of General and Administration Costs

While the data shows poor correlation there is a general trend of decreasing G&A cost per mill feed tonne as the feed process rate increases. On this basis, G&A costs for the Study were estimated as:

US$0.37 – (Benchmarked basic process cost per tonne x 0.0089)

The resulting estimated G&A costs per mill feed tonne are:

- Open Pit Only US$6.63

Gold Realisation Charges

The following gold realisation costs are based on published reports and the US mining cost database. The royalty payment is part of the purchase arrangement from Barrick Gold.

Table 19: Gold Realisation Charges – estimates and assumptions

Realisation Charge	Cost
Doré transport and insurance	US$1.00/oz
Gold refining charge	US$7.00/oz
Payable gold in doré	99%
Royalties (Barrick)	2.5%

10. Economic Analysis

Cautionary Statement

Economic analyses in this Study are to provide a preliminary assessment of project viability for the two scenarios modelled being ‘Underground & Open Pit Mining’ and ‘Open Pit Only’ as support for decisions on proceeding to a DFS.

The economic outcomes presented are not intended as a Mineral Asset Valuation under the Valmin Code 2015. The authors of this study are not Valmin Practitioners.

Cashflows in this study are based on:

- Production targets which are not Ore Reserves as defined in the JORC Code (2012) and which include up to 19% Inferred mineral resources,
- Capital and operating cost estimates which include a high proportion of benchmarked and factored costs,
- A high proportion of assumed or historical operating inputs in disciplines where the authors of the report are not experts.

There is a high likelihood that the financial outcomes in this study will not be realised in any future project and should not be relied on.

Basis of Cashflow Models

Cashflows are presented as earnings before interest, taxes, depreciation, and amortisation (EBITDA). No analysis of financing costs or corporate taxation are included.

Discounted Cashflow Results

Revenue from gold sales less estimated capital and operating costs applied against the production schedule for the ‘Open Pit Only’ scenario are presented as net cashflows on undiscounted and discounted bases.

Selection of Gold Price

A fixed gold price of US$2,200/oz is used for the Study. See Section 11.

Selection of Discount Rate

Discounted cashflows (DCF) use a discount rate of 6.5%. This rate was selected as a pre-tax estimate of the weighted average cost of capital using the long term US lending rate current in September 2024 and assuming a debt to equity ratio of 80:20. The debt to equity ration in the future may vary from this.

Economic Outcome

The estimate life of mine EBITDA net cashflow would be **US$397M undiscounted** or **US$235M discounted**.

The operating life would be **8.4 years**.

Project payback on an EBITDA basis would be **3.3 years undiscounted** or **4.2 years discounted.**

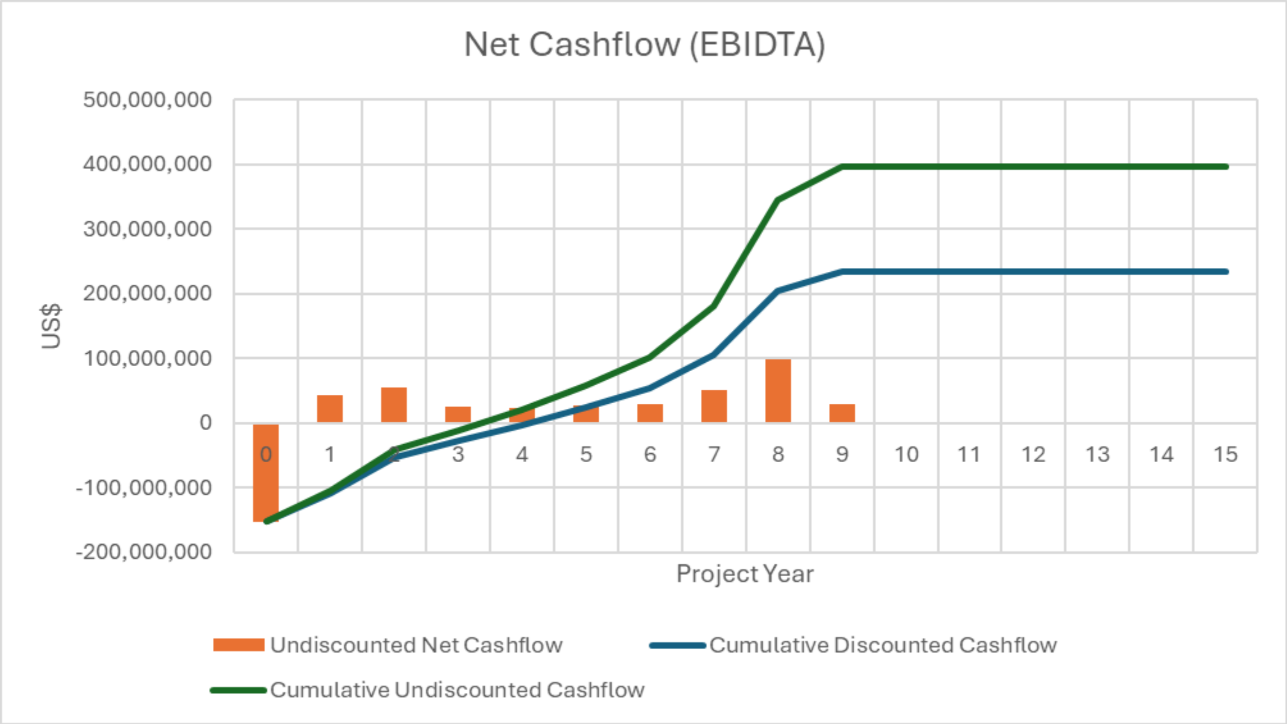

Figure 27: Net Cashflows

Sensitivity Analysis

The net present value of the discounted cashflows was tested against ±15% variations in key factors. The model is most sensitive to gold price and recovery. It shows similar sensitivity to mining and processing operating costs but are less sensitive to capital costs or discount rates.

The 'Open Pit Only' scenario remains positive within the full range of sensitivities tested.

These results do not consider combined effect of changes to multiple inputs.

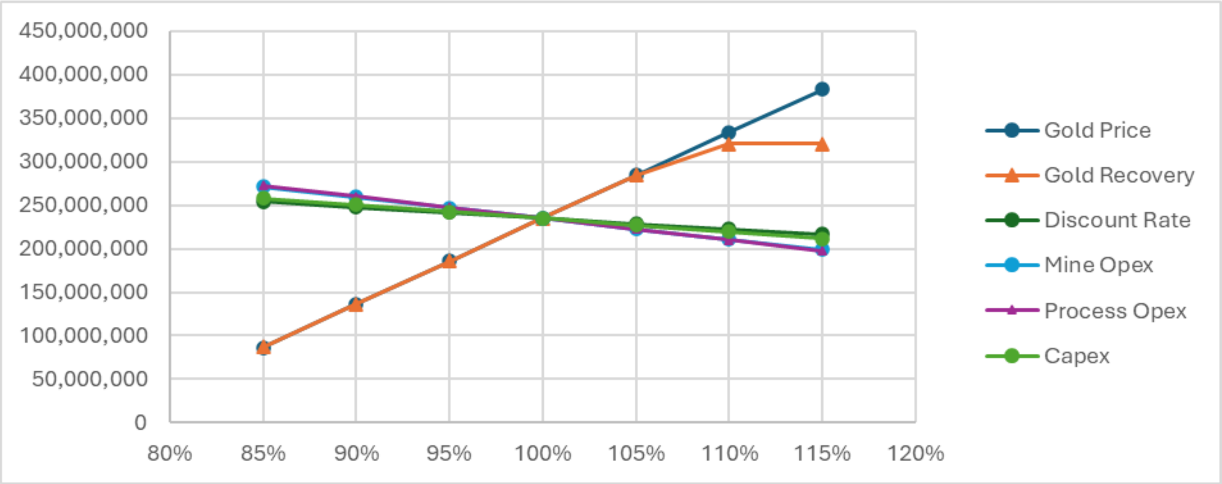

Figure 28: NPV Sensitivity Analysis in US$

Table 20: Sensitivity Analysis – Estimate of variance for each variable

Variable	85%	90%	95%	100%	105%	110%	115%
Gold Price	87	136	186	235	284	334	383
Gold Recovery	87	136	186	235	284	321	321
Discount Rate	255	248	241	235	229	223	217
Mine Opex	271	259	247	235	223	211	199
Process Opex	273	260	248	235	222	210	197
Capex	258	251	243	235	227	220	212

The sensitivity analysis has been estimated by varying one input parameter at a time, leaving all others unchanged. There is the potential that a change in one parameter will result in other parameters changing, either naturally or by changes to decisions. The sensitivity analysis is provided as a guide only and may not reflect actual variations in practice.

As illustrated above, the project is most sensitive to the gold price and gold recovery. The current spot gold price at the date of this Study is approximately 20% higher than the gold price assumed in the Study.

The sensitivity analysis indicates that the project still produces a positive NPV even at a gold price of US$1,870/oz (US$2,200 x 85%). US$1,870/oz is approximately 68% of the level of the current spot price for gold.

Funding

The Company's 100% owned Colosseum Project is in a tier one jurisdiction and regarded as low risk. Historical production records demonstrate in excess of 91% gold recoveries and the project has very strong economics that provide a robust platform for Dateline to source traditional financing through debt and equity markets. There is, however, no certainty that Dateline will be able to source funding as and when required.

To achieve the various outcomes indicated in the Scoping Study, pre-production funding in excess of US$153M may be required. Typical project development financing would involve a combination of debt and equity. Dateline has formed the view that there is a reasonable basis to believe that requisite future funding for development of the Colosseum Project will be available when required.

There are grounds on which this reasonable basis is provided including:

- The Project is in a tier one jurisdiction, with simple non-refractory metallurgy allowing for an industry standard CIL process plant and has a rapid payback of only 3.3 years from commercial process production;
- The very strong pre-tax cashflows of US$398M and rapid payback would support a significant level of conventional debt financing for the Project development;
- The Company has a strong track record of raising equity funds as and when required to further the exploration and evaluation of its assets; and

- The Dateline Board and management has extensive experience in mine development, financing and production in the resources industry.

11. Mining History

Mining history for Colosseum up to 1994 is taken from information compiled on the Mindat website, the USGS and a paper by Gregg Wilkersen.

1860 to 1994

1860 – 1875

Exploration in the Clark Mountain district began in the late 1860s. The district was organized on July 18, 1865, by John Moss, owner of several mines, including the historic Colosseum Mine, which was discovered in 1865.

1900 – 1906

Developed by Devereaux Brothers, Ivanpah Consolidated Mining Co.

1923 – 1938

Purchased by C.H. Gowman, September 1923; operated by Colosseum Mines, Inc. from 1923 to February 1938. The historic Colosseum Mine was developed as underground workings in the West breccia pipe within the lower portion of the gold zone and constitutes the largest historic gold mine within the district. No recorded production occurred until the 1930s, with production of about 615 ounces gold. Recorded production for the mine also indicates that $45,000 (1930s value) in gold and copper was produced prior to 1940. The main historic developments at the mine consist of:

- An adit at 5700 feet accessing 470 feet of drifts.
- An adit at 5877 feet accessing 725 feet of drifts and 250 feet of raises,
- A vertical stope in the central interior of the rubble breccia portion of the West breccia pipe, and
- Semicircular workings on the westernmost edge of the West pipe.

Note that these workings would have been removed in the South Pit between 1988 and 1992.

1938 – 1942

Under lease to Harold Chase and Walter Lineberger, of Santa Barbara from February 1938 to February 1942; 30 unpatented claims, 2 patented claims, Colosseum No. 1 and Colosseum No. 2; 640 acres. The mine was closed in 1942 as a non-essential industry during World War II

1970s - 1985

During this 15 year period the Colosseum property was held by California Gold Properties, which leased some of the older claims and located an additional 176 claims. A series of exploration ventures on the Colosseum property was conducted by Draco Mines, Placer AMEX, and Amselco Exploration using modern methods. Amselco leased the property from Draco Mines in 1982 and conducted extensive drilling and feasibility studies between 1982 and 1984. This work resulted in delineation of ore reserves associated with the southwestern most of two felsite intrusive breccia pipes to a depth of 750 feet. Amselco began the required permit applications in 1983, and a Final EIR/EIS was approved in July 1985.

1986 - 1982

The property was acquired by Dallhold Resources Inc., a subsidiary of Bond Gold International, in September 1986. In November 1986, Royal Resources acquired a 25% interest in the property. Modern operations began in 1986 with mineable reserves estimated to be 10,539,000 tons (9.56 million metric tonnes) with an average grade of 0.062 oz Au/ton (2.13 g/t), for a total of 653,418 oz contained gold. Construction of a 3,400 tons per day (3,084 tonnes per day, or 1.25 Mtpa) carbon-in-pulp, cyanide mill started in November 1986 and was completed in September 1987. Open pit mining commenced in mid-1987 and the mine and mill reached full capacity by early 1988. 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 mining the west pipe and the North Pit mining the east pipe. Most of the mining facility was on unpatented Federal land under the jurisdiction of the Bureau of Land Management (BLM); but two patented claims are located in the South Pit area.

1989 - 1993

Lac Minerals acquired the properties of Bond International Gold, Inc in 1989. Colosseum Inc., a subsidiary of Lac Minerals Ltd, continued to operate the Colosseum Mine until mining was suspended on July 10, 1992. Processing of low grade stockpiles continued until May 1993.

Gold production ran for 5.5 years compared to the original 9 year mine plan. From January 1988 to May 1993:

- 6.59 million tonnes of ore was mined at a waste to ore ratio of 3.97.
- 5.84 million tonnes of ore was milled at an average grade of 2.02 g/t Au.
- 345 koz of gold was poured.

Approximately 750,000 tonnes remains in a low grade stockpile on the east side of the South Pit. A sampling program is required to confirm the grade of this stockpile.

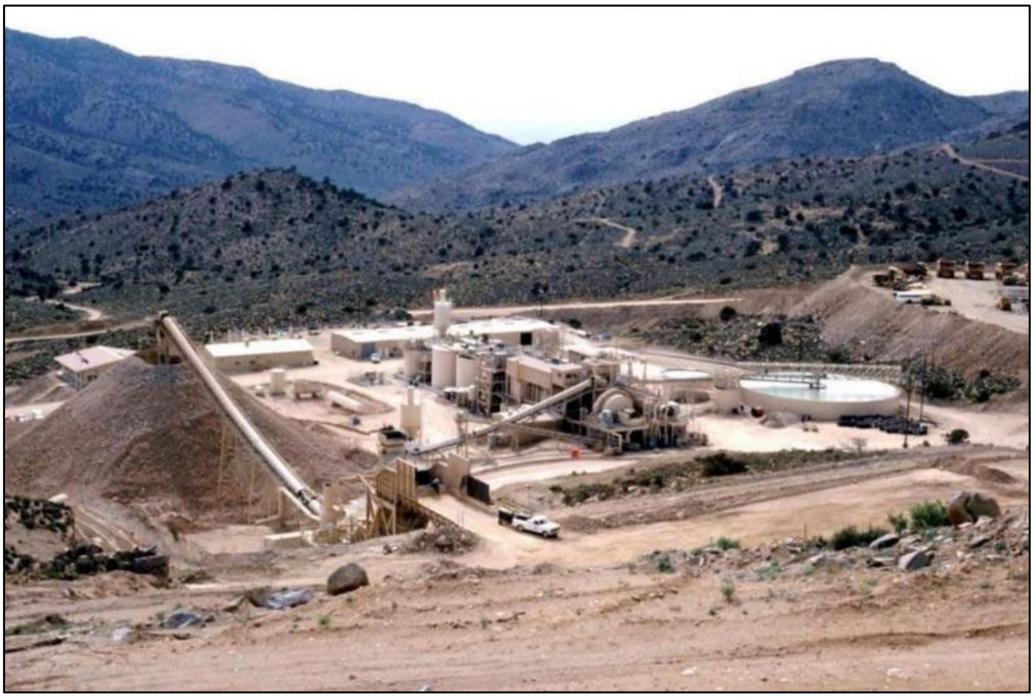

Figure 29: Colosseum gold process plant in 1992

1994 to 2024

Lac Minerals was acquired by Barrick Gold Corporation in 1994. The Colosseum Project lay dormant until Dateline Resources Limited (DTR) acquired it from Barrick Gold in March 2021.

Since the acquisition, DTR has:

- Collated all available exploration and production data,
- Validated the historical drill hole data and converted to the current UTM grid,
- Conducted a confirmatory drilling program on the West Pipe (South Pit),
- Reviewed and updated the geological interpretation,
- Produced two Mineral Resource Estimate updates (July 2022 and June 2024) to bring the Mineral resource up to 1.1 Moz,
- Conducted preliminary mining studies, and
- Progressed discussions and negotiations with relevant stakeholders to establish the status of approval and permits required to re-start mining and processing operations.

12. Geology and Exploration

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 30 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.

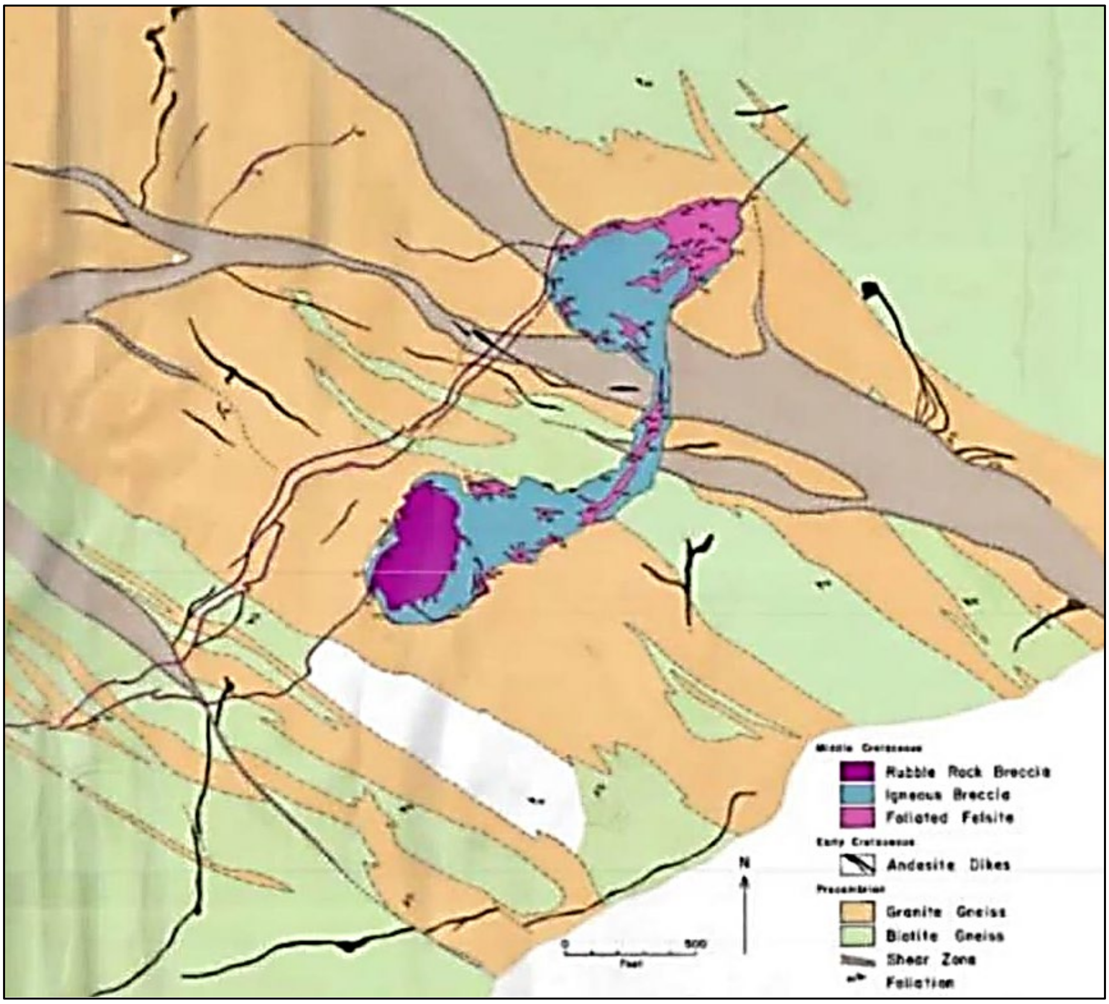

Figure 30: Geology Map for the Colosseum Gold Project (source unknown)

Deposit Types

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.

Exploration

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 meaningful and material data has been included in a previous report by H&SC (2024a).

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.

Future work would be for a feasibility study. If required, additional drilling may be needed for metallurgical and geotechnical purposes.

Processing and interpretation of the geophysical data is ongoing. Current work is on a follow-up program involving IP or MT surveys to test deeper and with greater resolution.

Drilling

A total of 616 holes for a total of 59,137 metres have 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 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 April 2024, DTR drilled 17 diamond core holes (with one abandoned hole) along existing haul roads within the South Pit, for a total of 3,527.65 metres. The majority of this drilling is aimed at confirming mineralisation grades at depth and to better define lateral margins to the deposit.

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

Since April 2024, DTR has drilled two holes in the North Pipe. The first hole intersected a moderate grade zone (0.5 to 2.0 g/t Au) at depth on the eastern side of the North Pipe. The second hole intersected high grade mineralisation within the body of the currently defined resource.

These holes are not included in the June 2024 MRE. When the next MRE update is run they may result in some localised increase of estimated gold grade.

13. Mineral Resource Estimates

DTR's consultants modelled the mineral resource in 2022 and 2024.

The July 2022 MRE used Ordinary Kriging (**OK**) and was mainly based on historic drilling (1991 and earlier) and five holes drilled by DTR 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 DTR in the ASX release "1.1 million ounces of Gold at the Colosseum" dated 6 June 2024.

This section on Mineral Resource Estimates is taken from the June 2024 MRE.

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).

Estimation Methodology

Recoverable MIK was used to complete the gold grade estimation using HSC's inhouse 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 54,313 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) and another for the North Pit (domain 2), 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. 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, in this case, two extreme consecutive samples from one drillhole were top cut to 500g/t.

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 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 AMDAD.

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 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 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. No assessment has been made for any deleterious elements.

Drillhole spacing ranges from 10 to 15m in the core of the two domains but at a variety of directions giving rise to relatively close spaced samples. Downhole sampling was generally at 5 feet (and 2 feet) intervals.

The mineral 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 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 drillholes. HSC also validated the block model using a variety of summary statistics and statistical plots. No issues were noted. Validation confirmed the modelling strategy as acceptable with no significant issues.

Comparison with the 2022 mineral resource estimates indicated a larger tonnage for the 2024 Mineral Resource by 27% and at a very slightly higher gold grade. None of this is unexpected based on the different modelling strategy and the additional drilling data.

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.

Density

No historical density data was supplied.

53 density measurements were supplied by the Company from their recent drilling. Samples consisted of single pieces of core 10-15cm 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

2.66t/m^3 with a range of 1.96 to 3.37t/m^3. Density values tended to show an increase with hole depth.

A default density of 2.65t/m^3 was used for the Mineral Resources and is considered reasonable.

Cut Off Grades

The recoverable MIK resources are reported at a gold cut-off of 0.5g/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 mineral 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 1.3 at a US$2,400/oz gold price with preliminary estimates of mining costs and pit wall slopes.

Classification Criteria

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 Resource, Search Passes 2 & 3 equal Indicated Resources and Search Passes 4 & 5 equals Inferred Resource.

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 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.

Mineral Resource Estimate

The Mineral resource Estimate for June 2024 was reported by DTR in the ASX release “1.1m oz gold for updated Colosseum Resource Estimate” dated 6th June 2024.

Table 21: June 2024 Mineral Resource Estimate

	Category	Cut-off (g/t Au)	Volume (m^3)	Tonnes (Mt)	Grade (g/t Au)	Ounces (koz)
South Pit	Measured	0.5	1.01	2.67	2.23	191.2
	Indicated	0.5	1.13	3	1.28	123.8
	Inferred	0.5	1.89	5.01	1.13	182.6
	TOTAL	0.5	**4.03**	**10.68**	**1.45**	**497.6**
North Pit	Measured	0.5	2.62	6.93	1.18	263.8
	Indicated	0.5	1.59	4.22	1.16	157.7
	Inferred	0.5	1.99	5.26	1.07	182
	TOTAL	0.5	**6.2**	**16.42**	**1.14**	**603.4**
Combined	Measured	0.5	3.62	9.6	1.47	454.98
	Indicated	0.5	2.73	7.23	1.21	281.44
	Inferred	0.5	3.87	10.27	1.1	364.6
	TOTAL	0.5	**10.23**	**27.1**	**1.26**	**1,101.00**

The new recoverable Mineral Resources for the Colosseum gold deposit are reported for a gold cut-off grade of 0.5g/t constrained to the block centroid being above the optimised pit shell and below the current topographic surface.

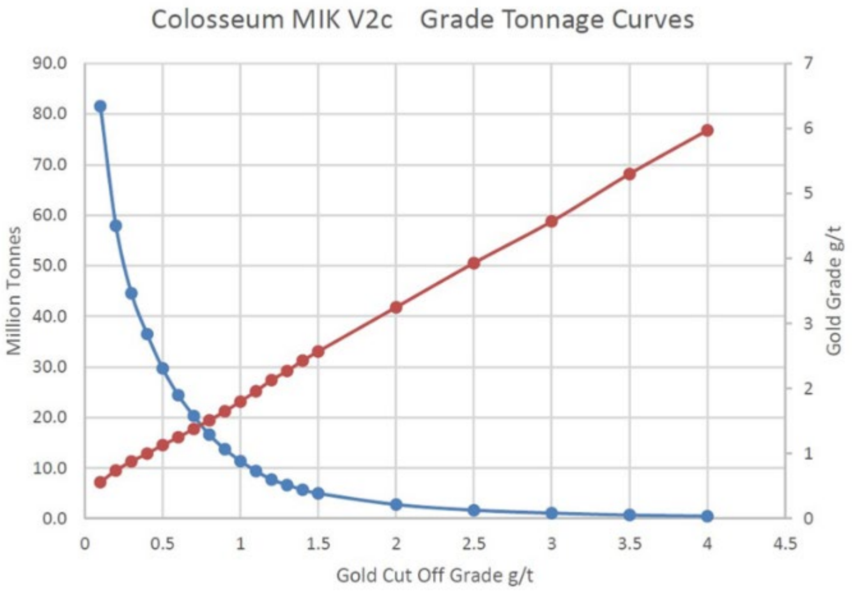

Figure 31: Grade Tonnage Curve for the June 2024 MRE

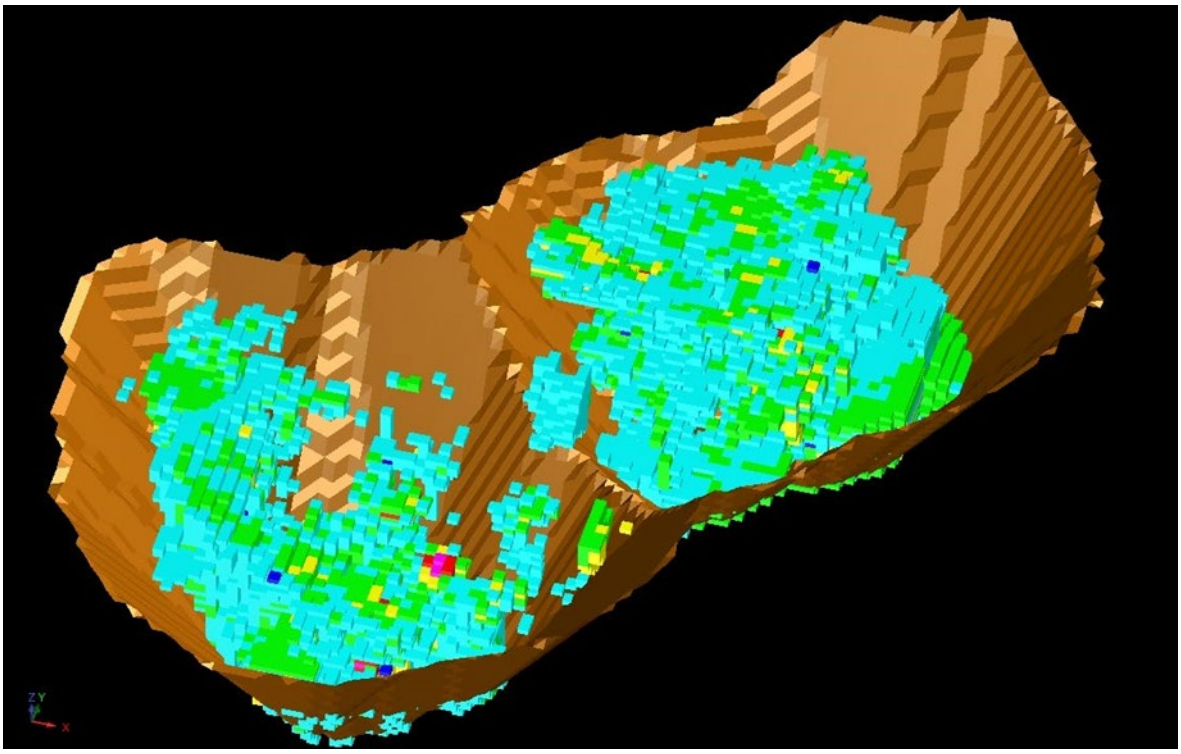

Figure 32: Gold Block Grade Distribution for the Colosseum Mineral Resources (HSC), (view looking down to NNW)

Competent Person Statement – Mineral Resources

The mineral resources underpinning the production target in this Study for the Colosseum gold deposit were prepared by Mr Simon Tear who is a Member of The Australasian Institute of Mining and Metallurgy (MAusIMM) 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 in the form and context in which they appear.

14. Ore Reserve Estimates

An Ore Reserve has not been prepared for this Study.

The Colosseum Gold Mine Study is an order of magnitude technical and economic study of the potential viability of the Colosseum Mineral Resources. It includes appropriate assessments of realistically assumed Modifying Factors together with any other relevant operational factors that are necessary to demonstrate at the time of reporting that progress to a DFS can be reasonably justified. However, the Study is based on low-level technical and economic assessments and is insufficient to support estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Study will be realised.

Additionally, the outcome of the Study is partially (14-19%) supported by Inferred Mineral Resources, which cannot convert to Ore Reserves for this level of study.

15. Risk Assessment and Forward Work

Risk Assessment

The table below provides a preliminary listing of key risks identified for the project as the basis for defining a program of work for a DFS.

Table 22: Summary of Key Risks

Risk Event	Impact	Controls
Mining Cost variability / accuracy	This Study uses preliminary owner operator costs developed from a combination of cost database, first principles and benchmarked against other operations. These costs were developed in mid-2024 and broadly represent contemporary rates. The scale of the open pit and/or underground may be sub-optimal if mining costs are higher than estimated. As well as direct impact on project value, re-optimisation at higher costs than currently estimated may result in a reduction in the size of the optimum pit, the extent of the underground mine and production tonnes.	Review and revise costs in the DFS, with OEM prices, parts costs etc. Contractor pricing Explosives pricing Open pit and underground optimisation and scheduling would be re-run in the next phase of work with final mining costs to confirm extraction shapes.
Confidence level of Mineral Resource forming the basis for the mine plan	The mineral resource has a significant component in the Inferred category. There is a risk that this may not convert to measured and indicated, in which case mill feed shortfalls may result in the schedule. This would reduce revenue and project value.	Further resource drilling as part of the DFS program would target the Inferred Resource, upgrading it to Indicated.
Open cut optimisation – geotechnical assumptions	Optimisations assume that the Study slope designs are appropriate. However, the assumed slopes may be too steep. Should flatter slopes be required additional waste will be mined for stable pit walls, which will impact both project cashflow and schedule material movement profile.	Geotechnical and hydrogeological studies will be undertaken in the DFS to increase the confidence level in these areas and to confirm design parameters for stable excavation in the open cut and underground.
Water impacts on mine plan	Potential water impacts include: - • operational and safety impacts in the open cut and underground mines • environmental impacts from surface contact water Existing documentation suggests that there is a low risk of surface water impacts due to the low precipitation, and limited groundwater.	Hydrological and hydrogeological assessment is required in the DFS phase of work to confirm the low potential for surface water impacts and limited groundwater inflows to the open cuts and underground mines. The hydrological assessment will also confirm requirements and design for surface water management systems such as drains and containment ponds.
Blasting requirements	Underestimation of the powder factor would result in higher drilling and blasting costs than	The powder factors adopted for the Study are broadly consistent with Dyno (2010) and

Risk Event	Impact	Controls
are underestimated	estimated. Underestimation of other blasting considerations such as fly rock, noise/vibration and explosives storage may result in inappropriate site layout and additional costs and impacts for a revised layout.	slightly more conservative than Amselco (1984). A blasting study would be undertaken as part of the DFS program to confirm powder factors and to address fly rock, noise/vibration and explosives storage.
Reactive ground and sulphide dust impacts on mining operations	Low levels of sulphide mineralisation in the rock to be excavated at Colosseum suggest that it is unlikely the ground will be reactive in the underground mine and the deeper benches of the open cuts, and that special precautions would be necessary for blasting.	In the DFS phase of work in conjunction with the blasting study a qualified specialist will make an assessment to confirm that there won't be any reactive ground or sulphide dust explosion risks.
Geotechnical impacts on mine plan – Open Cut	The geotechnical assessment impacts on the open cut wall slopes, which in turn determines the optimal pit shell, material movement, and mining costs. The Study open cut slopes were based on mined slopes in the South Pit which have stood for over 30 years without significant failures. Waste rock dump slope parameters are yet to be geotechnically assessed.	A DFS level geotechnical study will be undertaken for the open cut and waste dump slopes to reduce uncertainty and risk. The open cut geotechnical work will include: • Review of Study open cut designs, • gathering additional data from drilling and testing of samples if required, borehole imaging for structural orientations and to reduce uncertainty, hydrogeological data collection. • assessment of open pit and underground interactions, • assessment of waste rock dump slope stability. • geotechnical assessment for surface infrastructure
Dilution and Recovery, Ore selectivity – impacts and uncertainty	Inadequate selectivity when mining ore may result in excessive dilution and loss. Inappropriate allowances or methods for estimation of dilution and mining recovery may underestimate the dilution and recoveries experienced during operation. Additional dilution will increase operating costs due to loading, haulage, and processing of sub-economic material. Lower recovery will reduce production and revenue.	The Study open cut tonnes and grades have been adjusted for dilution and mining loss application of global adjustment factors. Estimates for dilution skin thickness and percentage adjustments for additional dilution and mining loss would be assessed and refined in the DFS phase of work. Moderate selectivity is required to define economic cut-offs, and to manage dilution and loss. This would be achieved through appropriate grade control practices including RC grade control drilling in the open cut in advance of mining. The ability to achieve reasonable selectivity would also be addressed in the DFS equipment selection.
Pit waste rock storage requirements and potential for acid formation	The Study assumes that waste rock will be non-PAF, and no allowance has been made in the costs for storage of potentially acid forming (PAF) waste rock material. The Study costs for management of waste rock material may be underestimated.	PAF waste characterisation would be completed as part of the DFS to identify whether some waste rock may be potentially acid forming (PAF), posing a risk to the quality of water draining off and through the waste dumps.

Risk Event	Impact	Controls
	Additional costs required to appropriately store PAF waste, above that currently allowed, would impact project economics.	Methods and parameters for management of PAF waste rock would be further investigated and confirmed if PAF is identified.
Contaminants	The mining activities may involve interactions with contaminant elements such as lead, arsenic and mercury. Specific OH&S procedures are likely to be required to address the risk to personnel of exposure to such contaminants. These measures could impact on the mining costs.	Investigations are required to better define this risk and mitigation measures including personal protective equipment, procedures for elimination of dust, personal hygiene, washing of work clothes and monitoring of contaminant levels. Further work to address the risk posed by contaminants includes: • assessment of likely concentrations of contaminants in the mine work areas • Development of OH&S procedures to address the risk to personnel of exposure to such contaminants Assessment of impact on the mining costs of these measures.
Surface topography	Inaccuracies in the as mined/topographic surface may result in additional waste stripping requirements, to the volumes estimated using the current surface model, or inappropriate designs where a high level of precision and accuracy is required to address water courses etc.	The current surface topography model used for the Study mine plan is based on a LiDAR model flown in April 2021. This is considered appropriate for Study level estimates of the open cut and waste rock dump design and volumes.
Initial production schedule is not achieved.	Open cut and underground mining have been modelled using Geovia's MineSched program. This level of detail is considered appropriate for a Study. However, further detail is required to improve confidence in the initial mine development period as well as overall production rates. If productivity estimations for initial mining are optimistic, or if the resource model mineralisation grades and/or extents are overestimated, then insufficient ore may be available to meet mill throughput and metal production.	The detailed DFS Mine Development Plan would update the pit optimisations and pit and dump designs. Modifications to ramp and haul road positions would be incorporated as required. The schedules would allow updating of the mine fleet and workforce. If Case 1 is taken forward to the DFS, stope shapes would be re-optimised against updated costs and revenues, underground designs would be revised and SLC flow modelling would be undertaken. The results of this work would allow a detailed first principles estimation of the underground mining fleet, workforce and consumables. Appropriate grade control will be undertaken in advance of initial mining to confirm initial ore production estimates. A Mining Execution Plan would be prepared as part of the DFS, to help ensure the timely development of the mine leading to ore production.
Equipment maintenance is	Mining targets not met due to lower than estimated fleet availabilities, impacting ore production and cashflows, as a result of: • Maintenance personnel with	The DFS will specifically address mine maintenance including: • consideration of maintenance and repair contracts (MARCs) with

Risk Event	Impact	Controls
inadequate.	inadequate training, skills and/or numbers. - Inadequate maintenance facilities. Inadequate holdings or availability of spare parts and maintenance supplies.	equipment manufacturers - detailed design and estimation of maintenance facilities - assessment of maintenance personnel and support requirements The DFS will also include evaluation of owner-operator versus mining contractor. The latter would give greater flexibility to increase fleet numbers if required. Maintenance capability would be a key consideration in the selection of the contractor. The contractor would be required to commit to a comprehensive maintenance strategy. This may include maintenance and repair contracts with the original equipment manufacturers or their authorised dealers or service agents. Use of a mining contractor should give greater flexibility to increase fleet numbers if mechanical availabilities are sub-optimal.
Sub-optimal pit design and ore reserve for assumed parameters.	If the pit design is not optimised for scale of mining and/or mining fleet, then project NPV will not be maximised.	The final Study pit design is based on Whittle™ open cut optimisation and design investigations for haul roads and pit staging. However, the optimality of the pit design and ore reserve also depends on the project non-mining parameters assumed, including metal prices, non-mining operating costs and processing assumptions. The DFS would include appropriate sensitivity analysis of non-mining parameters to confirm the robustness of the selected optimised pit. It is recommended that the design process be independently reviewed before mining commences, in order to confirm the ultimate pit.
Waste rock dump slopes become unstable and slip.	Potential hazard for mine operators, equipment, and water management measures.	The slopes applied are considered to be reasonable for stability at Study level but require geotechnical analysis to confirm an acceptable factor of safety. Dump stability analysis will be conducted in the next phase of work as part of the geotechnical investigations. Testing of critical sites should be conducted to confirm the waste dump design and requirements for stripping material from the waste dump footprint.
Truck productivity is lower than estimated.	Where modelled truck productivity does not fully account for site-specific factors then lower than expected productivity would require additional trucks to achieve the scheduled material movement rates, adding to project	The Study mining estimates include truck numbers derived from preliminary haulage tkm allowances. While these are adequate for Study level, more detailed modelling would be required for the DFS. The DFS will include detailed haul modelling to

Risk Event	Impact	Controls
	capital and/or operating costs.	increase confidence in required truck hours and numbers to achieve the scheduled material movement. Consideration of contractor mining in the DFS - for an owner-operator scenario the requirement for extra trucks would result in additional capital and operating costs. However, for contract mining with a set contract price per tonne hauled, the cost impact may be minor.
Insufficient stockpile capacity for ore when crushing/ conveying system and/or the mill is broken down.	Potential for annual production to not meet required targets due to instances where the system is unable to sustain ore delivery in the short term.	As broad a filled pad as possible would be built out from the ROM crusher area to provide immediate stockpile buffering. Capacities of the crusher and crushed ore stockpile should be assessed regarding likely outages, considering also the minimum and maximum possible ore digging rate. Consideration should be given to simulation work during the development phase to confirm capacities.
Insufficient material for road sheeting and stemming.	Potential for poor quality roads, reduced haul speeds and production shortfalls, as well as poor blasting performance resulting in higher operating costs. Additional costs may be incurred in sourcing suitable material from an external quarry.	It is likely that reasonable aggregate material could be sourced from selected excavated waste rock within existing waste dumps. Sources of material for aggregate, including a simple quarry design, will be confirmed in the DFS.
Mining activities generate sediment-laden water run-off.	If dirty-water controls are inadequate to control sediment-laden water, then this may jeopardise approvals for operation and put the project at risk of suspension.	Areas of mine disturbance have been identified and water management systems outlined. A detailed mine water management plan would be prepared in the next phase of development.
Incorrect equipment selection.	Impacts include productivities, ore production, and dilution, resulting in higher costs, lower revenue, cashflow and value.	The Study mine plan is based on an industry standard mining fleet for the underground mine and typically sized equipment for the scale and material movement rate of the open cut operations. The DFS will include a detailed evaluation and selection of the mining fleet, and also include evaluation of owner-operator versus mining contractor. Contract mining would provide some opportunity to revise the fleet, and locks in some of the cost.

Risk Event	Impact	Controls
Mine capital costs are higher than expected.	Estimates of capital costs made at this time do not reflect the actual costs when mine development occurs, and this will impact on overall project viability and rate of return.	The main capital costs are the costs for the processing plant, general project infrastructure, mining fleet, establishment of mine facilities and infrastructure, initial road development, and initial mine operations. This includes pre-stripping waste rock in the open cuts, excavation of the boxcut, establishment of portals, and development of the decline and ventilation shafts for the underground mine. The above items would be assessed in detail in the DFS phase of work.

Processing Risks

Risk Event	Impact	Controls
Processing recovery is lower than estimated.	Direct impact on revenue, cashflow and return on investment	Study gold recovery set at 92% as opposed to 91.4% in 1984 test work and 91% from operations. Extra 1% based on assumption of approved technology since 1993. Test core from recent drilling to confirm mineralogy hasn't changed recovery from former operation.
Ore is harder than estimated	Higher than estimated costs for comminution. Adverse impacts on recovery.	Test core from recent drilling to confirm mineralogy hasn't changed recovery from former operation. Detailed crushing and grinding plant design and associated estimates for running costs including power, consumables, personnel
Other metallurgical risks such as high cyanide consumption, copper impact	Higher costs, lower recovery.	Test core from recent drilling to confirm mineralogy hasn't changed recovery from former operation. 1984 FS report notes that cyanidation leaching, and carbon adsorption tests were completed for the FS. 1984 FS report notes that a copper impurities study at Hazen in the 1980s indicated that copper would not be a problem.
Inappropriate process plant design or location	Additional costs for construction, modifications, or higher than estimated operating costs	Detailed plant design including location review, geotechnical investigation, operability study.

Infrastructure Risks

Risk Event	Impact	Controls
Insufficient water supply, poor water management	Interruption to production, higher than estimated operating costs, environmental discharge.	Address water supply as a key part of the DFS. Note that according to the EPA (1993) the previous operation was supplied with water via a pipeline from two wells in the Ivanpah Valley, the wells provided 800 gpm, 16h/day. No water was sourced from Shadow Valley, 10 miles to the west, which had originally been nominated by Amselco (1984) as the primary water supply.
Inappropriate or insufficient power supply	Blackouts, interruption to production, higher than estimated operating costs, safety incidents.	Address power supply as a key part of the DFS. Investigate existing powerlines, obtain pricing for grid supply.
Inappropriate or insufficient tailings management and storage	Safety hazard, environmental discharge, delays to production, higher than estimated costs.	Review Amselco FS (1984) and address tailings testwork, including thickening and filtration, and storage as a key part of the DFS.
Inappropriate or insufficient maintenance facilities	Breakdowns, poor availability, higher than estimated running costs, safety incidents.	Address maintenance facilities as a key part of the DFS.
Inappropriate or insufficient general site facilities and services	Inappropriate or inadequate buildings, offices, amenities, communications, IT, emergency systems, lead to delays, project underperformance, higher than estimated costs.	Address general site facilities and services as a key part of the DFS, including geotechnical assessment of sites for buildings and structures.

ESG Risks

Risk Event	Impact	Controls
Operations ESG impacts/risks not properly addressed.	Greater than estimated impacts: - • on flora and fauna • Potential emissions and pollutants impacting groundwater, air quality including greenhouse gas emissions. • Noise, dust, visual amenity impacts Resulting in: - • jeopardising regulatory approvals and possible suspension of operations • cost impacts reputational impacts	Existing EIS forms minimum standard. DFS would address other measures to for current best practice. Detailed environmental and social studies as required would be undertaken as part of the DFS

Risk Event	Impact	Controls
Mine closure not properly addressed.	Landform instability and erosion, chemical discharges from site, ongoing impacts on flora and fauna resulting in cost and reputational impact	DFS would include detailed closure plan in accordance with approved Reclamation Plan.
Insufficient time allowed for approvals.	Delay to project start, impact on cashflows, returns.	Assess status of all approvals in place at suspension of operations in 1993. Timely program of engagement with community and statutory bodies.
Insufficient stakeholders buy-in	Delays to approvals, permits	Appropriate program of engagement with community and government.
Inappropriate disposal plan for tailings and waste rock	• jeopardising regulatory approvals and possible suspension of operations • cost impacts	Detailed investigations of tailings and waste rock as part of DFS including geochemistry, characterisation, management plans.

Business Risks

Risk Event	Impact	Controls
Inadequate or inappropriate organisation structures, recruitment, training and safety	Project underperformance, delays to production, higher than estimated costs, safety incidents and injuries	Address organisation structures, recruitment, training and safety in DFS
Logistics inadequately addressed.	Safety or environmental incident	Address logistics, warehousing, supply in DFS
Poor construction project management	Delay to project commissioning and startup, cost overrun, adversely impacting cashflow and reputation.	Develop detailed project execution plan as part of the DFS. Establish appropriate project management for construction and initial production.
Inadequate or inappropriate business systems	Project performance is not appropriately monitored and managed, delays to production, higher than estimated costs, safety incidents and injuries	Address business systems and management systems in DFS including IT, communications.

16. Forward Work - Definitive Feasibility Study

This Study has been prepared at a scoping study level of confidence. However, it deals with a project which:

- Operated from 1988 to 1993,
- Had extensive metallurgical test work pre-1988,
- Is extensively drilled,
- Has recently drilled core available for metallurgical, geochemical and geotechnical test work,
- Has high walls exposed for geological and geotechnical mapping,
- Has existing road access which can be upgraded,
- Is in close proximity to services and labour sources, and
- Has an existing framework of approvals.

With this level of information available it is reasonable to build on the findings of this Study to move to a DFS with minimal further work. The proposed path to project development decision based on a DFS is:

- A short Project Selection Study (**PSS**). This study would:
    - Confirm status of all key project approvals,
    - Engage with key experts for the DFS and confirm or adjust key inputs for the DFS such as process and infrastructure capital and operating costs. This would still be at an order of magnitude level of confidence, but it would replace benchmarked estimates in the Study with estimates aligned to current South West USA experience.
    - Select project definition based on either Case 1 (underground and opencut) or Case 2 (opencut only). DTR would make this selection based on commercial outcomes of this Study and the PSS updates and other strategic criteria.
    - Set the detailed scope of work, program for the DFS and engage experts for each area.
- A Definitive Feasibility Study (**DFS**). This study would:
    - Be based on one Scenario (Open Pit Only),
    - Start with a throughput analysis to optimise the mill feed rates assumed for the Study,
    - Include any updates to the Mineral Resource model,
    - Include any further analytical test work identified during the PSS such as process performance test work, tailings and waste rock characterisation, geotechnical assessment, updates to hydrology and geohydrology and updates to environmental and social impact assessments,
    - Include optimisation of the mine plan and fleet and commencement of negotiations with mining contractors and explosives suppliers,
    - Include optimisation of the process flow sheet and equipment selection.
    - Identify and confirm availability of site access, power supply and water supply.

  - Define and estimate all aspects of the project across mining, processing, infrastructure and environmental and social impact to enable the majority of the project construction cost to be estimated at AACE Class 2.
  - Define and estimate all aspects of the project across mining, processing, infrastructure and environmental and social impact to enable the majority of the project operating costs to be estimated at ±15% accuracy.
  - Update gold price and other commercial inputs with industry accepted forecasts.
  - Include a financial model prepared by a qualified practitioner with experience in US corporate law and taxation and with sufficient sensitivity analyses to support an investment decision for the project,
  - Confirm all approvals and permits required for the project to re-commence and, where possible, start application processes for any that need to be updated or renewed,
  - Include a project implementation plan, and
  - Be prepared in conjunction with an Ore Reserves Estimate as defined in the JORC Code 2012.

The proposed two stage approach is to ensure that when the DFS commences the entire project team is focussed on a single, clearly defined scope of work. This is essential to ensure the DFS delivers a reliable basis for the project investment decision and that is completed within the estimated time and budget.

A key decision during the DFS would be if any further drilling is required. Reasons for drilling could include:

- Upgrading Inferred resources in the Study mine plans to at least Indicated status for inclusion in the Ore Reserve,
- Geotechnical and hydrogeological assessment, or
- Obtaining samples for metallurgical test work.

Cores drilled through mineralisation and waste since 2021 are available and the geology is clearly exposed in the existing South Pit walls. If this information can be used instead of new holes it save time and expense for the DFS.

This announcement has been authorised for release on ASX by the Company's Board of Directors.

For more information, please contact:

Stephen Baghdadi	Andrew Rowell
Managing Director	White Noise Communications
+61 2 9375 2353	+61 400 466 226
www.datelineresources.com.au	andrew@whitenoisecomms.com

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About Dateline Resources Limited

Dateline Resources Limited (ASX: DTR) is an Australian publicly listed 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. On 6 June 2024, the Company announced to the ASX that the Colosseum Gold mine has a JORC-2012 compliant Mineral Resource estimate of 27.1Mt @ 1.26g/t Au for 1.1Moz. Of the total Mineral Resource, 455koz @ 1.47/t Au (41%) are classified as Measured, 281koz @1.21g/t Au (26%) as Indicated and 364koz @ 1.10g/t Au (33%) as Inferred.

The Colosseum is located less than 10km north of the Mountain Rare Earth mine. Work has commenced on identifying the source of the mantle derived rocks that are associated with carbonatites and are located at Colosseum.

Forward Looking Statements

Certain statements contained in this Announcement, including information as to the future financial or operating performance of Dateline and its projects may also include statements which are 'forward-looking statements' that may include, amongst other things, statements regarding targets, estimates and assumptions in respect of mineral reserves and mineral resources and anticipated grades and recovery rates, production and prices, recovery costs and results, capital expenditures and are or may be based on assumptions and estimates related to future technical, economic, market, political, social and other conditions. These 'forward-looking statements' are necessarily based upon a number of estimates and assumptions that, while considered reasonable by Dateline, are inherently subject to significant technical, business, economic, competitive, political and social uncertainties and contingencies and involve known and unknown risks and uncertainties that could cause actual events or results to differ materially from estimated or anticipated events or results reflected in such forward-looking statements.

Dateline disclaims any intent or obligation to update publicly or release any revisions to any forward-looking statements, whether as a result of new information, future events, circumstances or results or otherwise after the date of this Announcement or to reflect the occurrence of unanticipated events, other than required by the Corporations Act 2001 (Cth) and the Listing Rules of the Australian Securities Exchange (ASX). The words 'believe', 'expect', ‘anticipate’, ‘indicate’, ‘contemplate’, ‘target’, ‘plan’, ‘intends’, ‘continue’, ‘budget’, ‘estimate’, ‘may’, ‘will’, ‘schedule’ and similar expressions identify forward-looking statements.

All ‘forward-looking statements’ made in this Announcement are qualified by the foregoing cautionary statements. Investors are cautioned that ‘forward-looking statements’ are not a guarantee of future performance and accordingly investors are cautioned not to put undue reliance on ‘forward-looking statements’ due to the inherent uncertainty therein.

Dateline has concluded that it has a reasonable basis for providing these forward-looking statements and the forecast financial information included in this Announcement.

To achieve the range of Colosseum Gold Project outcomes indicated in the Study, funding of in the order of an approximately US$152 million will likely be required by the Company.

Based on current market conditions and the results of studies undertaken, there are reasonable grounds to believe the Project can be financed via a combination of equity and debt, as has been done for numerous comparable projects in the US and other jurisdictions in North America in recent years. Debt may be secured from several sources including Australian banks, US banks, international banks, the high yield bond market, resource credit funds, and in conjunction with product sales via hedging agreements. It is also possible the Company may pursue alternative funding options, including undertaking a corporate transaction, seeking a joint venture partner or partial asset sale.

There is, however, no certainty that Dateline will be able to source funding as and when required. At this point, no formal funding discussions have commenced with potential financiers of the Colosseum Gold Project.

This ASX Announcement has been prepared in compliance with the current JORC Code (2012) and the ASX Listing Rules. All material assumptions, including sufficient progression of all JORC modifying factors, on which the production target and forecast financial information are based have been included in this ASX Announcement.

Appendix 1: Alternative Underground + Open Pit Case (Case 1)

The Study also considered an alternative scenario of Underground and Open Pit mining. This scenario did not produce as attractive returns as the Open Pit Only scenario and included additional risks.

The Company resolved to proceed with investigations into the viability of the Open Pit Only scenario. The following section provides a brief summary of the 'Underground and Open Pit Mining' scenario for completeness.

Underground Mining Method Selection

Underground mining at Colosseum would target the higher grade gold mineralisation remaining below the existing South Pit utilising sub-level caving (**SLC**).

Sublevel Caving (SLC)

SLC is a form of caving where most of the ore mined is blasted and the rest is comprised of dilution that enters the zone of blasted material. SLC is a relatively low-cost mining method which suits the size and grade of resource available for underground mining.

The proposed SLC layout has a 20m vertical interval between levels. Each level has a single access from the main decline. Development on each level is comprised of a main access, footwall drive, truck loading arrangement, return air and parallel SLC ore drives 15m apart and expansion slot development at the end of each SLC ore drive.

Production follows a top-down sequence:-

- On each level production starts with a vertical expansion slot developed at the end of each SLC ore drive using drill and blast methods.
- SLC uphole rings are drilled and blasted starting with the rings closest to the expansion slot then retreating towards the footwall drive.
- Diesel powered load-haul-dump units (LHDs) transfer broken rock from the drawpoints to a truck loading location on each level. Trucks then haul the broken rock to the ROM area on the surface via the main decline.

Underground Optimisation and Mill Feed Definition

AMDAD used the resource block model from the June 2024 MRE in Surpac™ and CAE Mineable Shape Optimiser (**MSO**) software to prepare a conceptual underground extraction shape.

Table 23: Optimisation parameters (USD)

Physical Inputs	Units	Value
Mining Dilution	%	15%
Mining recovery		Varies
Underground mill feed mining rate	ktonnes pa	500
Opencut mill feed mining rate*	ktonnes pa	589
Processing rate	ktonnes pa	1,089
Processing recovery	%	92%
Default (waste) density	t/m3	2.5
Sub-level Spacing	m (vertical)	20
Financial Inputs	**Units**	**Value**
Mine operating cost - SLC	US$/tonne	$37.19
Mine operating cost – aux stopes	US$/tonne	$44.63
Process operating cost	US$/tonne	$20.50
G&A operating cost	US$/tonne	$7.41
Gold price	US$/oz	$2200
Gold realisation charge	US$/g	$5.60
Cut-off grade	g/t	1.25

*The opencut mining rate is set to keep the mill at 1.089 Mtpa. Opencut mining would continue at the full processing rate of 1.089 Mtpa after the underground mine is depleted.

The SLC mining inventory was created using by the following steps:-

- Preparation of conceptual stope shapes and estimation of mineable quantities using the MSO program, with reference to mining, processing, and economic inputs.
- Add extra SLC shapes within the overall set of SLC shapes produced by MSO so that the total mining inventory is a combination of the two sets of shapes.
- Evaluation of these quantities in an Excel spreadsheet to define scheduled development and production quantities, and thereafter project cashflow and net DCF.

The MSO results were re-evaluated within an Excel spreadsheet to confirm economic viability once access costs were considered. This evaluation resulted in a number of marginal SLC shapes and shapes within the opencut void being excluded from scheduling. A small number of these shapes were retained and are included in the total “mining inventory” as open stopes if they cover the additional cost of being mined by long hole open stoping separate to the SLC.

The SLC stope shapes for Case 1 are shown below:-

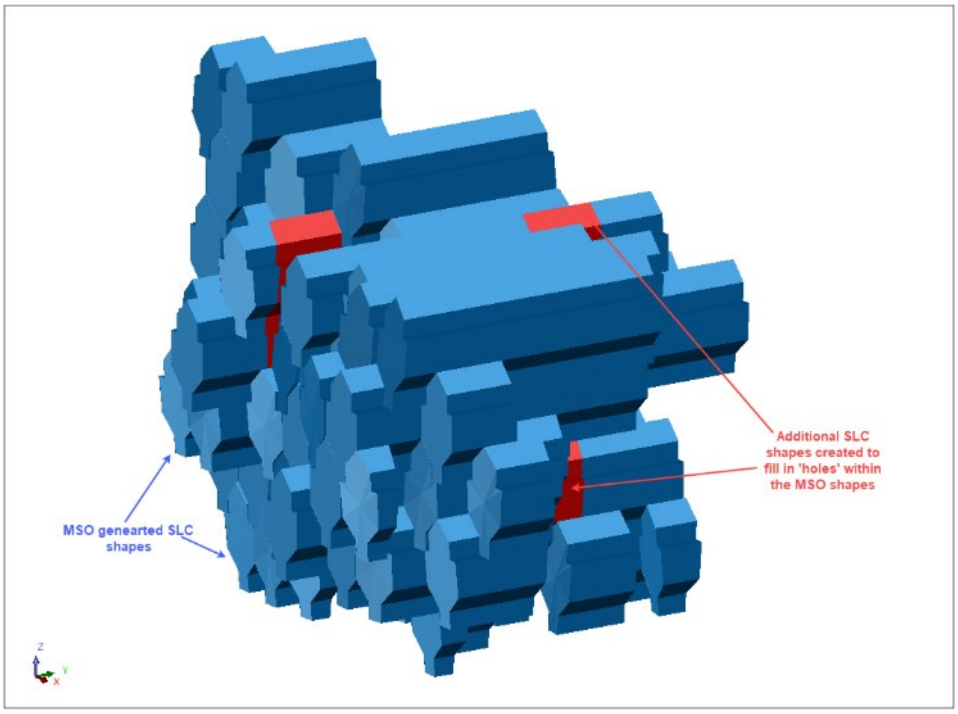


Figure 33: SLC Mining Shapes

The following table shows the progression from the MSO defined shapes through practical and economic rationalisation ('Mining Inventory') to ROM tonnes after adjustments for mining loss and dilution. The table includes mill feed tonnes from the SLC, adjacent open stopes and in-mineralisation development.

Table 24: Underground Production

Evaluation	Mt	Au g/t
MSO	3.13	2.41
Mining Inventory	2.62	2.45
ROM	2.44	2.11

No specific SLC flow modelling software was used to generate an estimate for the tonnes and grades mined from the SLC rings. The tonnes reported inside the SLC shapes for each sublevel are based on the proportion of blasted material mucked from that level, referred to as 'primary draw', and further reducing proportions mucked from the levels below. In addition, each sublevel has between 10% and 25% dilution added at 0.5g/t grade. The resulting draw of tonnes and metal from the entire SLC is 102% of tonnes and 86% of metal. The recovery of blasted SLC rings is only 86% due to the poor overlap of SLC rings from sublevel to sublevel.

Underground Development Design

A development centreline design was prepared based on the defined SLC and auxiliary stope shapes. The development concept is outlined below:-

- the existing open pit ramp is used as the entry point for a main decline developed at 1 in 8 down,
- lateral development at 20m vertical intervals for each sublevel
- stockpile bays, loading bays and ventilation development as required.

The following table lists the development required for each case.

Table 25: Underground Development Quantities

Development Type	Profile	metres
Main decline & stockpiles	5.0m (w) x 5.0m(h)	1,709
Main access on sublevel	5.0m (w) x 5.0m(h)	463
Lateral waste	4.5m (w) x 4.5m(h)	2,405
Ventilation access	4.5m (w) x 4.5m(h)	225
SLC ore drives	4.5m (w) x 4.5m(h)	2,871
Vertical ventilation rise, RAR	3.0m diameter	191
Vertical ventilation rise, FAR	2.4m diameter	174

Figure 34 below shows the development concept:

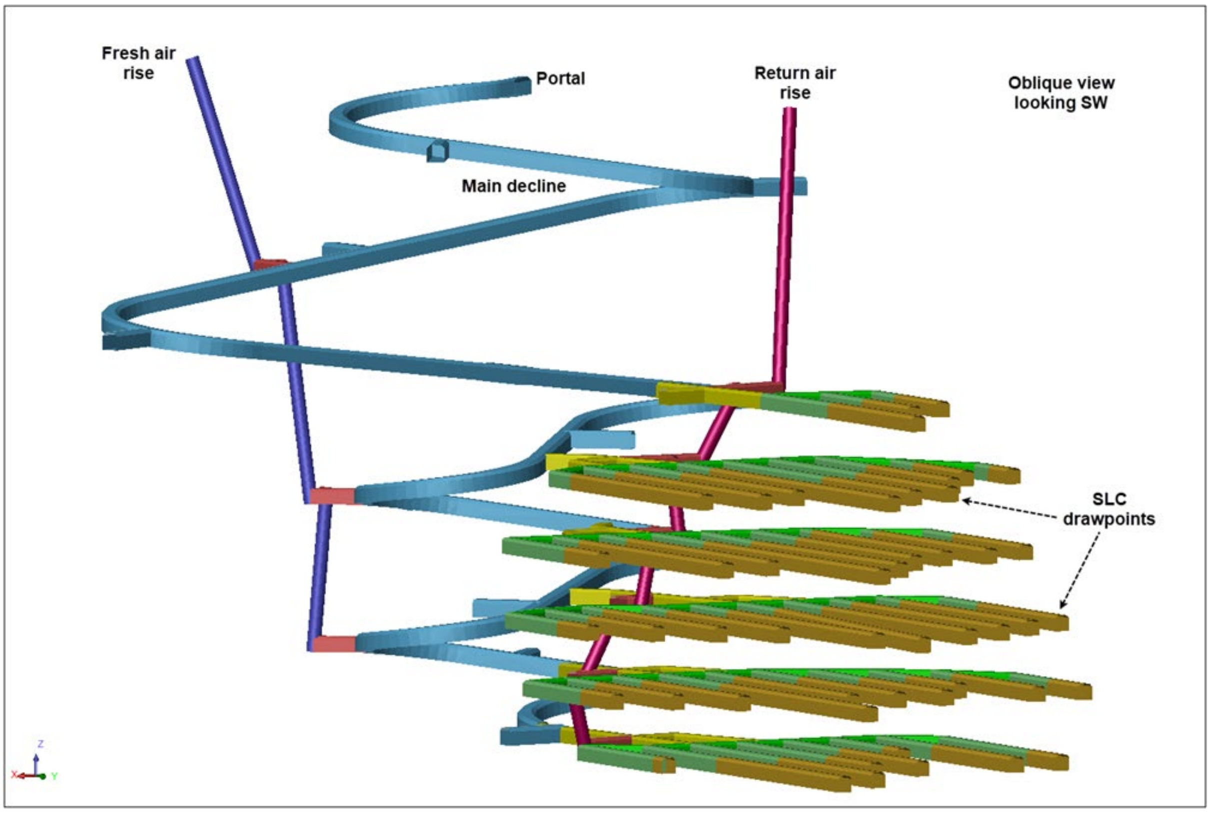


Figure 34: Underground Development Design – 3D view looking SW

Underground Mine Workforce

The proposed workforce associated with the underground operation is listed in the table below. The technical roles listed are total required and operators are number required per shift.

Table 26: Underground Mine Technical and Management Workforce

Role	Max number
Managers	4
Foreman	3
Shift Boss	3
Engineer	4
Geologist/Environment	12
Accountants	4
Purchasing	5
Clerical	10
Total	**45**

Pit Design for 'Underground and Open Pit' scenario

This scenario mines the North Pipe in two pit stages as per the 'Open Pit Only' scenario detailed in the Study.

'Underground and Open Pit' scenario - Load Haul Fleet

The 'Underground and Open Pit' scenario material movement rates vary over the mine life with the opencut initially supplementing underground production and then operating at low waste to ore ratios from the North and West Pits (see Table 27). Efficient matching of the fleet to the required dig rate is further complicated by the relatively low mill feed rate compared to the potential capacity of the excavators.

The following charts show the variability in material movement quantities over the mine life and the strategy for managing the truck fleet against these quantities.

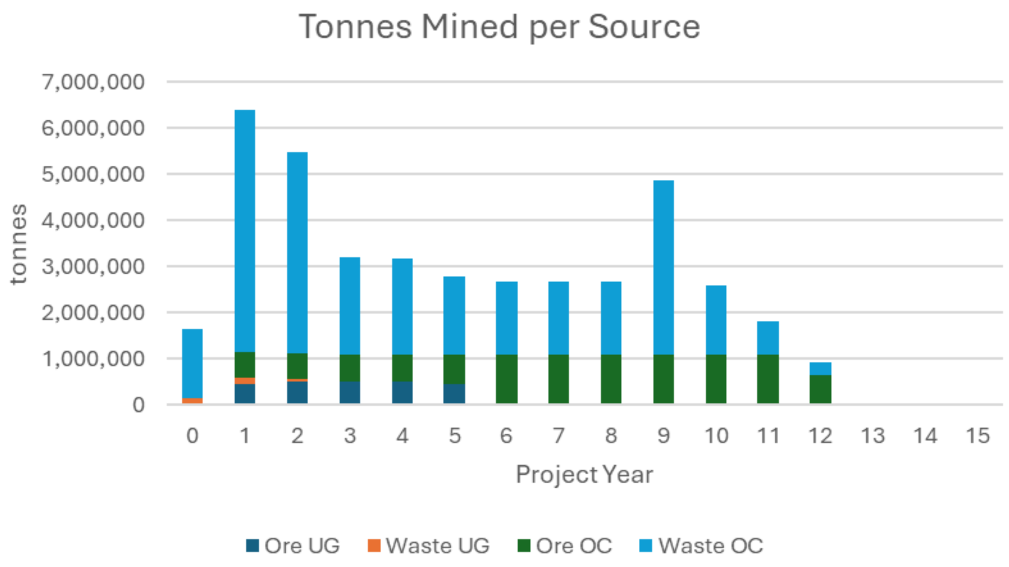


Figure 35: Annual Material Movement – 'Underground and Open Pit' scenario

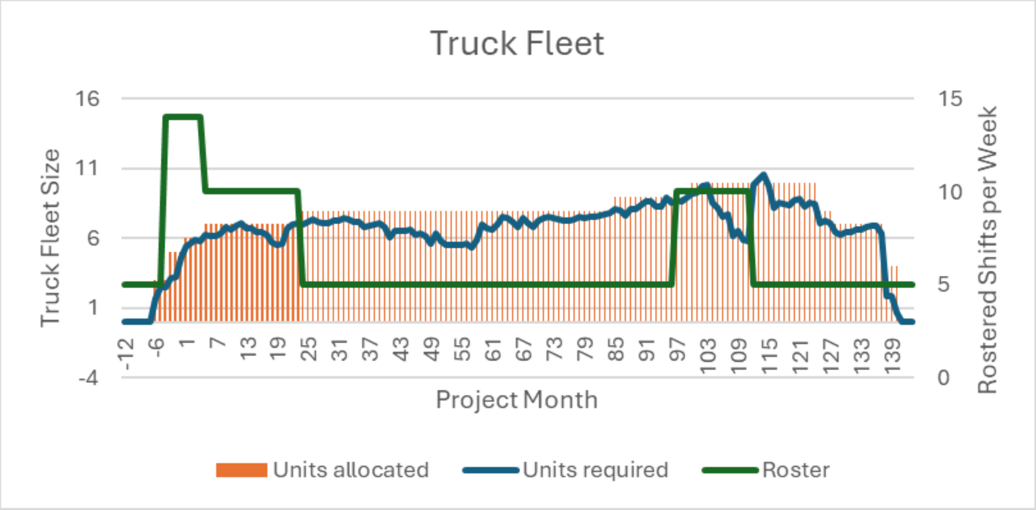
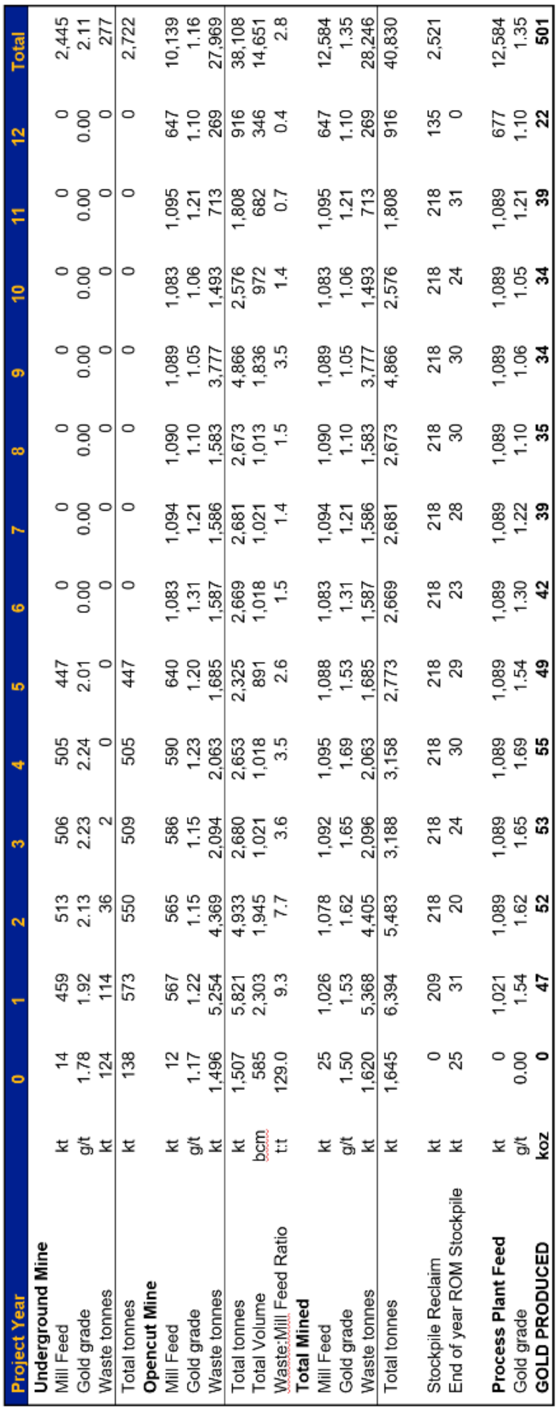


Figure 36: Truck Fleet - 'Underground and Open Pit' scenario

Table 27: Life of Mine Production Schedule – Underground and Open Pit scenario



Project Year		0	1	2	3	4	5	6	7	8	9	10	11	12	Total
Underground Mine															
Mill Feed	kt	14	459	513	506	505	447	0	0	0	0	0	0	0	2,445
Gold grade	g/t	1.78	1.92	2.13	2.23	2.24	2.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	2.11
Waste tonnes	kt	124	114	36	2	0	0	0	0	0	0	0	0	0	277
Total tonnes	kt	138	573	550	509	505	447	0	0	0	0	0	0	0	2,722
Opencut Mine															
Mill Feed	kt	12	567	565	586	590	640	1,083	1,094	1,090	1,089	1,083	1,095	647	10,139
Gold grade	g/t	1.17	1.22	1.15	1.15	1.23	1.20	1.31	1.21	1.10	1.05	1.06	1.21	1.10	1.16
Waste tonnes	kt	1,496	5,254	4,369	2,094	2,063	1,685	1,587	1,586	1,583	3,777	1,493	713	269	27,969
Total tonnes	kt	1,507	5,821	4,933	2,680	2,653	2,325	2,669	2,681	2,673	4,866	2,576	1,808	916	38,108
Total Volume	bcm	585	2,303	1,945	1,021	1,018	891	1,018	1,021	1,013	1,836	972	682	346	14,651
Waste:Mill Feed Ratio	t:t	129.0	9.3	7.7	3.6	3.5	2.6	1.5	1.4	1.5	3.5	1.4	0.7	0.4	2.8
Total Mined															
Mill Feed	kt	25	1,026	1,078	1,092	1,095	1,088	1,083	1,094	1,090	1,089	1,083	1,095	647	12,584
Gold grade	g/t	1.50	1.53	1.62	1.65	1.69	1.53	1.31	1.21	1.10	1.05	1.06	1.21	1.10	1.35
Waste tonnes	kt	1,620	5,368	4,405	2,096	2,063	1,685	1,587	1,586	1,583	3,777	1,493	713	269	28,246
Total tonnes	kt	1,645	6,394	5,483	3,188	3,158	2,773	2,669	2,681	2,673	4,866	2,576	1,808	916	40,830
Stockpile Reclaim	kt	0	209	218	218	218	218	218	218	218	218	218	218	135	2,521
End of year ROM Stockpile	kt	25	31	20	24	30	29	23	28	30	30	24	31	0	
Process Plant Feed	kt	0	1,021	1,089	1,089	1,089	1,089	1,089	1,089	1,089	1,089	1,089	1,089	677	12,584
Gold grade	g/t	0.00	1.54	1.62	1.65	1.69	1.54	1.30	1.22	1.10	1.06	1.05	1.21	1.10	1.35
GOLD PRODUCED	**koz**	**0**	**47**	**52**	**53**	**55**	**49**	**42**	**39**	**35**	**34**	**34**	**39**	**22**	**501**

Underground Mine Capex

The capital cost includes both underground development outside the orebody and specific items or groups of capital costs. The capital cost for this scenario was estimated at US$48.9 million.

Appendix 2: Colosseum Table 1 (JORC Code 2012)

Section 1 Sampling Techniques and Data

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

Criteria	JORC Code explanation	Commentary
Sampling techniques	*Nature and quality of sampling (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.*	As of 5 May 2024, the resource database includes data from 613 holes, for a total of 189,221.07 feet (57,671.77 metres), 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. 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 meters) in 31 core holes, 132,180 feet (40,288 meters) in 273 reverse circulation holes and 11,625 feet (3,543 meters) in 33 rotary/percussion holes. 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

Criteria	JORC Code explanation	Commentary
		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** 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 duplicate samples. Drill hole results and supporting assay certificates are available.

Criteria	JORC Code explanation	Commentary
		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. **2022 Drilling** Dateline Resources Limited 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. Industry standard core handling and sampling procedures were employed to ensure high quality samples. Core 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. 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 overnight shipping to Paragon Geochemical Laboratory in Sparks Nevada. Samples were sent to 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. **2023 Drilling** Dateline Resources Limited 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. Industry standard core handling and sampling procedures were employed to ensure high quality samples. Core samples were collected at maximum of 5-foot intervals or at any

Criteria	JORC Code explanation	Commentary
		lithologic or noteworthy mineralisation changes. All core was logged for rock type, RQD, and recovery and dispatched for assay with usually 5 foot long sample intervals or smaller intervals to break out lithology/mineralisation changes. Logging geologist identified zones of interest, but the entire hole was measured and marked up. Core was halved with half going for assay and half remaining for reference. Core was bagged into pre-numbered bags, and palletised by the Logging Geologist, covered in shrink wrap and handed over to the freight company for shipping to Paragon Geochemical Laboratory in Sparks Nevada or ALS Global in Reno Nevada. 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. **2024 Drilling (To Date)** Dateline Resources Limited completed 558.4 metres (1,832 feet) of drilling in 2 drillholes at the Colosseum Project. All the drilling was done from the surface with HQ diamond drill core. Industry standard core handling and sampling procedures were employed to ensure high quality samples. Core samples were collected at maximum of 5-foot intervals or at any lithologic or noteworthy mineralisation changes. All core was logged for rock type, RQD, and recovery and dispatched for assay with usually 5 foot long sample intervals or smaller intervals to break out lithology/mineralisation changes. Logging geologist identified zones of interest, but the entire hole was measured and marked up. Core was halved with half going for assay and half remaining for reference. Core was bagged into pre-numbered bags, and palletised by the Logging Geologist, covered in shrink wrap and handed over to the freight company for shipping to Paragon Geochemical Laboratory in Sparks Nevada or ALS Global in Reno Nevada. 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

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>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.</td></tr>
<tr><td><i>Drilling techniques</i></td><td><i>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).</i></td><td>Historic Data
<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
2022 Drilling
The drilling program utilized surface core drilling.
The core drilling was conducted with an EVERDIGM ECR 18 drill. All holes utilized triple tube to increase recoveries. The drilling was completed by an experienced diamond drilling core driller.</td></tr>
<tr><td><i>Drill sample recovery</i></td><td><i>Method of recording and assessing core and chip sample recoveries and results assessed.</i>
<i>Measures taken to maximise sample recovery and ensure representative nature of the samples.</i>
<i>Whether a relationship exists between sample</i></td><td>Historic data
Sample recoveries for historic drillholes unknown.
Relationship between recovery and grade unknown
2022 Drilling
All drilling recoveries have been logged and notated each run based on 10 foot tooling.
To maximize sample recoveries, use of triple tube and long chain polymer muds were used to increase recovery.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	Recovery was good overall at better than 90% There has been no analysis between sample recoveries and grade to date.
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 data** Core and chip samples were geologically and geotechnically logged at the mine site to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Geological logging of core samples is qualitative and quantitative in nature. **2022-2024 Drilling** All core was geologically logged. Lithology, veining, alteration, mineralisation and oxides were recorded in the appropriate tables of the drill hole database. Each core box was photographed dry and wet, after logging of unit and structures were notated on the core. Geological logging of core samples is qualitative and quantitative in nature.
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 split core samples were taken. 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 most of the later drilling **2022 Drilling** All drill core was sampled using whole core samples. Samples were placed in heavy-duty, pre-numbered poly sample bags. Samples were placed on pallets and secured with stretch wrap and packing tape and shipped in batches by company personnel directly to Paragon Geochemical via FedEx Freight following standard chain of custody protocols. Routine QAQC samples were inserted at a 20% rate into the sample batches and comprised Certified Reference Materials (CRMs) from CDN Resource Laboratories Ltd. and verified blank granitic material. Rock samples sent to Paragon Geochemical in Sparks, Nevada were dried, weighed, crushed and 1 kg subsample split, which was pulverized to better than 85% passing 75 microns. Rocks samples were analysed by standard 30gm fire assay for gold. Sample size assessment was not conducted but used sampling size

Criteria	JORC Code explanation	Commentary
		which is typical for gold deposits. **2023-2024 Drilling** All drill core was cut in half lengthwise with half being assayed and half remaining for reference and kept in place in original box. Samples were placed in heavy-duty, pre-numbered poly sample bags. Samples were placed on pallets and secured with stretch wrap and packing tape and shipped in batches by company personnel directly to Paragon Geochemical or ALS Global via a local freight company following standard chain of custody protocols. Routine QAQC samples were inserted at a 20% rate into the sample batches and comprised Certified Reference Materials (CRMs) from CDN Resource Laboratories Ltd. and verified blank granitic material. Rock samples sent to ALS Global or Paragon Geochemical in Reno or Sparks, Nevada were dried, weighed, crushed and 1 kg subsample split, which was pulverized to better than 85% passing 75 microns. Rocks samples were analysed by standard 30gm fire assay for gold. Sample size assessment was based on lithologic boundaries and distinct mineralisation changes.
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 labs 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. **2022 Drilling** Samples were assayed by industry standard methods by Paragon Geochemical in Sparks, Nevada. Fire assays for gold were completed using industry standard fire assay methodology. External certified reference materials and blank materials were inserted into the routine sample stream prior to laboratory submission. **2023-2024 Drilling** Samples were assayed by industry standard methods by ALS Global in Reno, Nevada or Paragon Geochemical in Sparks, Nevada. Fire assays for gold were completed using industry standard fire assay methodology. External certified reference materials and blank materials were inserted into the routine sample stream prior to laboratory submission.
Verification of sampling and	*The verification of significant intersections by either independent or alternative*	**Historical Data** Computer printouts and assay certificates are available for the CP, CH and

Criteria	JORC Code explanation	Commentary
assaying	*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.*	CM series holes. The Amselco CM drill hole assays were loaded onto the computer in Denver directly from the Amselco lab. Assay data was then broken down into specific drill hole intervals to form a final data base. All assay data entered in the computer was subsequently checked against original lab 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 data base (Amselco, 1984). 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 assay certificates and submittal forms to scanned printouts from early digital assay databases thru 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 author considers the scans of original assay certificates to be primary sources, whereas the printouts from an earlier database are secondary sources. **2022 Drilling** Sampling, documentation and sample submittal were under the guidance and care of Chris Osterman, PhD Geol (Registered Member SME) and Raymond Harris, Arizona RG. Geologic information was recorded directly on paper drill logs developed specifically for the Colosseum Mine project to collect pertinent information relating to sample depths, RQD, lithology, veining, alteration, mineralisation, and oxides. Sample sheets containing sample depths, QA/QC (duplicates, standards, and blanks inserted in sample runs) was stored in excel spreadsheets. Logs were scanned and sent to database manager along with sample sheets for entry into MX Deposit, the Company's secured data management system available through Seequent. **2023-2024 Drilling** Logging, sampling, documentation and sample submittal were under the guidance and care of Graham Craig, B.Sc. Geol (Registered Member APEGM). Geologic information was recorded directly into MX Deposit logging software to collect pertinent information relating to sample depths, RQD, lithology, veining, alteration, mineralisation, and oxides. Sample sheets containing sample depths, QA/QC (duplicates, standards, and blanks inserted in sample runs) were completed using this same software. MX Deposit is the Company's secured data management system available through Seequent, which feeds directly into the Seequent 3D modelling software, Leapfrog.
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*	**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. Collar survey files were available for most of the 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

Criteria	JORC Code explanation	Commentary
	system used. *Quality and adequacy of topographic control.*	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 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 degrees from true north. Drill hole traces from the historic data base were plotted and compared to plan maps and sections. Azimuth discrepancies were observed in some of the SP91, BD90, ATDH series angle 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. 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 an 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 unsurveyed 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 unsurveyed 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. 2022 Drilling All drill hole collars were surveyed using differential Trimble R12i GPS and Trimble S7 Total Station. The positions are accurate to within 10 cm x-y and height (z) to +/- 20 cm. The holes are surveyed in the California State Plane Zone V coordinate system in feet. Hole locations are reported in UTM WGS84 coordinate system in metres. Downhole survey results were provided by Oretest using a Reflex ACT2 camera to record core orientation. Initial surveys were taken at 50 feet, then 75 feet intervals thereafter inside the drill string and EOH. Outputs were provided on paper and as digital files. 2023-2024 Drilling All drillhole collars were surveyed using handheld GPS. The positions are accurate to within 4 metres.

Criteria	JORC Code explanation	Commentary
		The holes are surveyed in UTM WGS84 coordinate system in metres. Downhole survey results were completed using a Reflex EZ-TRAC magnetic survey tool to record core orientation. Initial surveys were taken at 50 foot, then 100 feet intervals thereafter outside the drill string and at EOH. Outputs were stored on tablets and the REFLEX online storage software.
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 classifications applied.* *Whether sample compositing has been applied.*	**Historic Data** The historic drill hole data was used for prior mining of the Colosseum deposit to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied at the time and is appropriate to be used for the current Mineral Resource Estimate. The original uncut assay intervals were composited to reflect a standard 20 foot bench height based on previous mining at Colosseum. This method computes a length-weighted average of the portions of assay intervals which fall within each 20-foot bench. Composite intervals with less than 10 feet of assayed length were not used for grade estimation. The maximum composite length allowed was 30 feet to allow for inclined holes. 2022 Drilling Current drill holes were drilled to confirm lithological and grade boundaries established from historical drilling. Hole spacing varied depending on target. Data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for resource estimation procedure(s). No sample compositing was done.
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.*	**Historic Data, 2022-2024 Drilling** Drillholes were drilled obliquely to near perpendicular to the known mineralized structures. Definition of structure location was the principal goal. Sample orientation is deemed to be representative for reporting purposes. No 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. **2022-2024 Drilling** Drill hole sampling techniques and QAQC procedures were developed and reviewed by Dale A. Sketchley, M.Sc., P. Geo. of Acuity Geoscience Ltd. and Graham Craig, B.Sc. Geol, GIT of Colosseum Rare Metals. The QAQC program returned only a few CRM and BLK failures, which were deemed to be non-material for resource estimation.
Audits or reviews	*The results of any audits or reviews of sampling*	**Historic Data**

Criteria	JORC Code explanation	Commentary
	techniques and data.	Sampling techniques were developed and reviewed by mine site personnel. **2022-2024 Drilling** Drill hole sampling techniques and QAQC procedures were developed and reviewed by Dale A. Sketchley, M.Sc., P. Geo. of Acuity Geoscience Ltd. and Graham Craig, B.Sc. Geol, GIT of Colosseum Rare Metals. The QAQC program returned only a few CRM and BLK failures, which were deemed to be non-material for resource estimation.

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 exist production-based royalties. Barrick Gold is entitled to a 2.5% Net Smelter Return royalty on all future production of any metals from the Colosseum Gold Mine.
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
Geology	*Deposit type, geological setting and style of mineralisation.*	The Colosseum project is hosted by Proterozoic granites, gneisses. These were intruded by Tertiary age rhyolitic stocks, dykes and breccias. The gold mineralisation occurs in a number of different breccia pipes with both sedimentary and volcanic rock fragments. Gold is associated with pyrite within the breccia pipes.
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*	No Exploration Results are being reported.

Criteria	JORC Code explanation	Commentary
	the drill hole collar ○ *dip and azimuth of the hole* ○ *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.*	
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.*	No Exploration Results 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*	Drillholes are orientated vertically and obliquely to the mineralized structures and disseminated bodies. Interception angles of the mineralized structures are estimated by geometries from known occurrences in the adjacent mine workings and the core drilling intercepts.

Criteria	JORC Code explanation	Commentary
	effect (eg 'down hole length, true width not known').	
Diagrams	*Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	No Exploration Results are being reported.
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.*	No Exploration Results are being reported.
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. 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 extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	The objective of the future work will be a PFS. Additional drilling is likely for metallurgical and geotechnical purposes. Continued processing and interpretation of the geophysical data is ongoing. Currently working on a follow-up program involving IP or MT surveys to test deeper and with greater resolution.

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 current Dateline drill-hole databases were directly created by GeoGRAFX 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 MS Access database. This database is secure, operated by a single database administrator. The drilling data was supplied by DTR to HSC as an MS Access database (2022) plus the most recent 2023-4 drilling as CSV files. This data was re-imported into an MS Access 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. DTR 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 this is the case.*	Database Manager Barbara Carroll (CPG) conducted a field examination of the project area on 4 April 2022 and met with consulting geologist Chris Osterman PhD. 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 time and budgetary 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*	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. Lithological units were delineated for the felsite/felsite breccia, sedimentary

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td>*both of grade and geology.*</td><td>breccia and granite.
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.</td></tr>
<tr><td>***Dimensions***</td><td>*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.*</td><td>The Mineral Resources have an 800m by 800m surface extent. With two separate bodies 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.</td></tr>
<tr><td>***Estimation and modelling techniques***</td><td>*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*
The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.
The assumptions made regarding recovery of by-products.
Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).
In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.
Any assumptions behind modelling of selective mining units.
Any assumptions about correlation between variables.
Description of how the geological interpretation was used to control the resource estimates.</td><td>Recoverable Multiple Indicator Kriging (MIK) with two search domains was used to complete the gold grade estimation using HSC’s in-house 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 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.
A total of 54,313 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) and another for the North Pit (domain 2), reflecting a difference in intensity of drilling and assay grades.
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 in this case two extreme consecutive samples from one drillhole were top cut to 500g/t.
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:
<table>
<tr><th>Dom 1</th><th>X (m)</th><th>Y (m)</th><th>Z (m)</th></tr>
<tr><td>Pass 1</td><td>20</td><td>20</td><td>35</td></tr>
<tr><td>Pass 2</td><td>30</td><td>30</td><td>60</td></tr>
</table></td></tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td><p>Discussion of basis for using or not using grade cutting or capping.</p><p>The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.</p></td><td>
<table>
<tr><td>Pass 3</td><td>40</td><td>40</td><td>70</td></tr>
<tr><td>Pass 4</td><td>60</td><td>60</td><td>120</td></tr>
<tr><td>Pass 5</td><td>60</td><td>60</td><td>120</td></tr>
<tr><td></td><td></td><td></td><td></td></tr>
<tr><td>Dom 2</td><td>Dom 1</td><td>X (m)</td><td>Y (m)</td></tr>
<tr><td>Pass 1</td><td>25</td><td>25</td><td>25</td></tr>
<tr><td>Pass 2</td><td>35</td><td>35</td><td>35</td></tr>
<tr><td>Pass 3</td><td>50</td><td>50</td><td>50</td></tr>
<tr><td>Pass 4</td><td>70</td><td>70</td><td>70</td></tr>
<tr><td>Pass 5</td><td>70</td><td>70</td><td>70</td></tr>
</table>
<table>
<tr><th>Dom 1</th><th>Min Data</th><th>Max Data</th><th>Min Octants</th></tr>
<tr><td>Pass 1</td><td>16</td><td>48</td><td>4</td></tr>
<tr><td>Pass 2</td><td>16</td><td>48</td><td>4</td></tr>
<tr><td>Pass 3</td><td>16</td><td>48</td><td>4</td></tr>
<tr><td>Pass 4</td><td>16</td><td>48</td><td>4</td></tr>
<tr><td>Pass 5</td><td>8</td><td>48</td><td>2</td></tr>
<tr><td></td><td></td><td></td><td></td></tr>
<tr><td>Dom 2</td><td>Min Data</td><td>Max Data</td><td>Min Octants</td></tr>
<tr><td>Pass 1</td><td>16</td><td>48</td><td>4</td></tr>
<tr><td>Pass 2</td><td>16</td><td>48</td><td>4</td></tr>
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<tr><td>Pass 5</td><td>8</td><td>48</td><td>2</td></tr>
</table>
<p>The maximum extrapolation for the Mineral Resources is shown as the Pass 5 search.</p>
<p>No other elements were modelled therefore there are no assumptions about correlation between variables.</p>
<p>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.</p>
<p>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 2022 resource estimates indicated a larger tonnage for the 2024 Mineral Resource by 27% but at a very slightly higher gold grade.</p>
<p>No reconciliation possible because no mining cut off grades are available and low grade stockpiles have no assays.</p>
</td></tr>
<tr><td>Moisture</td><td>Whether the tonnages are estimated on a dry basis or with</td><td>Tonnages are estimated on a dry basis. Moisture not recorded.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	natural moisture, and the method of determination of the moisture content.	
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 0.5g/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 with a revenue factor of 1.3 at a US$2400/oz gold price with preliminary estimates of mining costs and pit wall slopes.
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*	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 opencut mining and ore processing. Future mining operations can be contained within the unpatented mine

Criteria	JORC Code explanation	Commentary
	environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	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. 53 density measurements were supplied by DTR. Samples were from recent drilling by DTR. Samples consisted of weight in air /(weight in air minus weight in water) measurements (Archimedes Principle) on single pieces of core. The average value was 2.66t/m^3 with a range of 1.96 to 3.37t/m^3. Density values tended to show an increase with depth. The default density value used in the resource estimates is considered reasonable.
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 resource estimates is based on the data point distribution which is a function of the drillhole spacing and the search 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.)

Criteria	JORC Code explanation	Commentary – NO ORE IS ESIMATED FOR COLOSSEUM
Mineral Resource estimate for conversion to Ore Reserves	*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.*	The Mineral Resource Estimate was prepared by HS Consultants and announced by DTR on 6 June 2024. No Ore Reserves are reported.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why this is the case.*	The lead author of this report, Mr John Wyche, has not visited the site. A site visit will be required as part of any future Ore reserve Estimate.
Study status	*The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.*	This study is at a Scoping Study level of confidence.

Criteria	JORC Code explanation	Commentary – NO ORE IS ESIMATED FOR COLOSSEUM
	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.	
Cut-off parameters	*The basis of the cut-off grade(s) or quality parameters applied.*	Cut off grades are defined as the gold grade which just covers processing, general and administration costs and gold selling costs after application of process recoveries. In the case of underground mining, the production mining cost to drill, blast, load and haul the mill feed to surface is also included. Cut off grades calculated for this study after allowing for mining loss and dilution are: Case 1 – Opencut 0.55 g/t Au, Underground 1.24 g/t Au Case 2 – Opencut 0.42 g/t Au
Mining factors or assumptions	*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 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 (e.g. 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.*	Opencut Mining: • Global adjustments of 10% dilution at zero grade and 95% mining recovery • Case 1 – 2 x 94 tonne hydraulic excavators loading 45 tonne payload trucks, All material except waste dumps blasted. • Case 2 – 1 x 94 tonne and 1 x 125 tonne hydraulic excavators loading 55 tonne payload trucks. Underground Mining: • Sublevel caving. Simple model using estimated draw and dilution on each 20 metre lift. No flow modelling yet conducted. • Mining recovery and dilution adjusted lift by lift as draw and dilution adjusted manually to account for cave operation and lift geometry. Overall average is approximately 81% mining recovery at 115% dilution at zero grade.
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*	92% process gold recovery based on former operation (91%) and assumed improvements in technology.

Criteria	JORC Code explanation	Commentary – NO ORE IS ESIMATED FOR COLOSSEUM
	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?*	
Environmental	*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.*	DTR advise EIS and Plan of Operations from 1993 remain in place and form minimum basis for proposed resumption of mining and processing.
Infrastructure	*The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.*	Site infrastructure assumed same as 1993. Assumed upgrade of existing road from Intestate Route 15. Assumed power from Ivanpah Solar Facility on new 69kV powerline. Assumed water from Ivanpah Valley wells,
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.*	Opencut estimate from first principles build up using US cost database for unit rates. Underground estimate from factored costs using US cost database for unit rates. Process and Infrastructure costs from benchmarking.
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.*	Fixed gold price of US$2200/oz based on publicly available forecasts from reputable banks and commodity brokers.
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.*	Fixed gold price of US$2200/oz based on publicly available forecasts from reputable banks and commodity brokers.

Criteria	JORC Code explanation	Commentary – NO ORE IS ESIMATED FOR COLOSSEUM
	Price and volume forecasts and the basis for these forecasts. *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply 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.*	Cashflow models on EBITDA basis. Discount rate 6.5%. Both Cases 1 and 2 economically viable at Scoping Study level of confidence. Case 2 likely to have significantly higher value.
Social	*The status of agreements with key stakeholders and matters leading to social licence to operate.*	DTR advise EIS and Plan of Operations from 1993 remain in place and form minimum basis for proposed resumption of mining and processing.
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.*	DTR advise EIS and Plan of Operations from 1993 remain in place and form minimum basis for proposed resumption of mining and processing.
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).*	No ore reserves estimated. Measured, Indicated and Inferred resources used in study. Inferred resources 14% of case 1 mill feed and 195 of Case 2 mill feed.
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*	No audits or reviews of this study 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*	No ore reserves estimated.

Criteria	JORC Code explanation	Commentary – NO ORE IS ESIMATED FOR COLOSSEUM
	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.*	

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