MATSA RESOURCES

ASX Announcement

22nd January 2021

Positive Concept Study 600,000tpa Gold-Ore Treatment Plant Lake Carey Project

Highlights

- Matsa appointed CPC Project Design (CPC) to undertake an Engineering Concept Study ("Study") on a 600,000 tonnes per annum gold-ore treatment plant to be constructed at the Lake Carey Gold project
- The Study's results demonstrate a Matsa owned and operated treatment plant located centrally to the existing Fortitude gold mine to significantly and positively impact the financial results of Matsa's mining opportunities
- Key results of the Study (accuracy level +/- 40%) show:
  - Capital cost of a 600,000tpa gold-ore treatment plant to be A$35.4M, plus a contingency of A$7.1M
  - Additional capital cost of associated Infrastructure to be A$13.6M, plus a contingency of A$2.7M
  - Ore processing costs to be A$32.26/t, plus a contingency of A$5.54/t
  - Overall project duration of 18 months from decision to proceed with a construction time of 12 months
  - Potential to increase ore treatment capacity to 1,000,000tpa

Impact on Fortitude Stage 2

- A review of the Fortitude Stage 2 mine study shows a clearer pathway to production with a Matsa owned and operated treatment plant which would eliminate existing delays due to a lack of suitable gold-ore treatment options
- The projected positive cashflow from mining operations substantially increases **to A$55.4M** compared to A$21.8M (at A$2,500/oz Au)
- Potential exists to increase recoverable ounces through re-optimisation of existing pit shells using the Study's lower haulage and processing cost profiles

CORPORATE SUMMARY

Executive Chairman

Paul Poli

Director

Frank Sibbel

Director & Company Secretary

Andrew Chapman

Shares on Issue

271.14 million

Unlisted Options

77.78 million @ $0.17 - $0.35

Top 20 shareholders

Hold 56.86%

Share Price on 21st January 2021

10 cents

Market Capitalisation

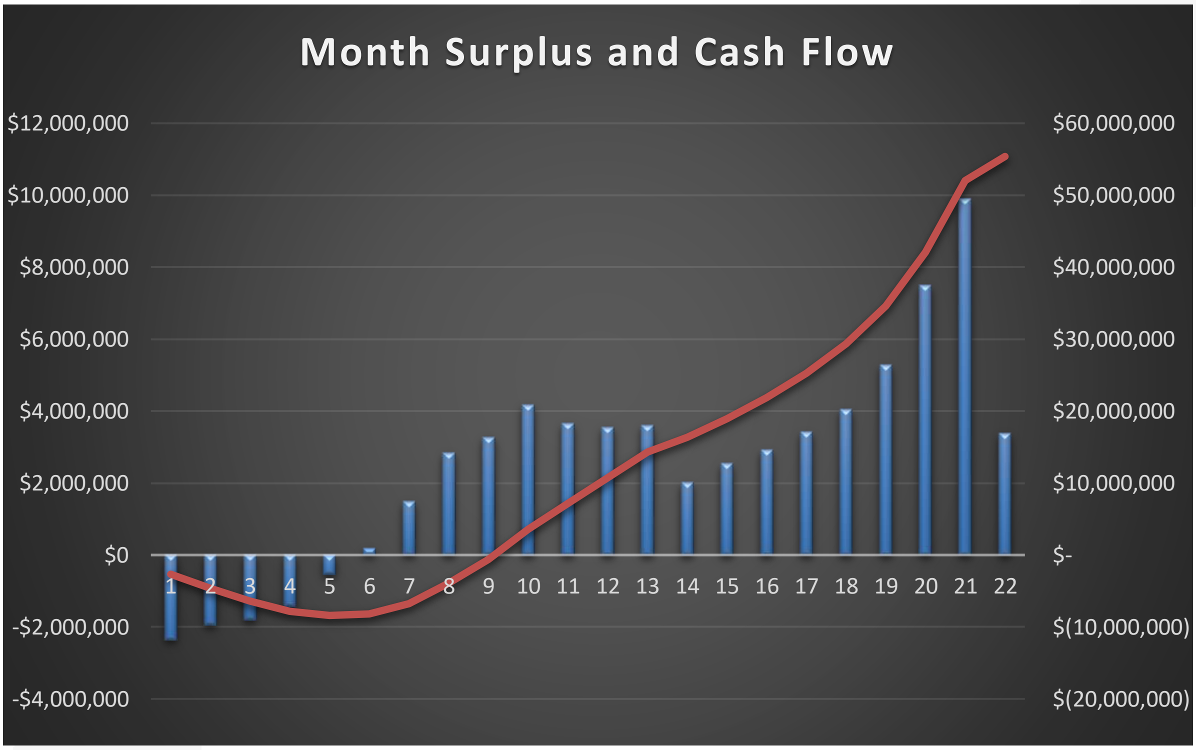
$27.11 million

Impact on Red October

- The Study's results highlight the potential for a dramatic positive impact on the economics of the Red October underground gold mine by reducing production costs and increasing ounces produced
- Applying the Study's new costs to the actual costs incurred at the Red October underground operation for the 30 September 2020 quarter production of 28,278t[1] cost **savings of A$1.6M** would have been realised. This represents a **A$641/oz reduction in AISC** to **A$2,180/oz** for that quarter
- Potential exists to increase recoverable ounces as the reduction in production costs would enable lower grade ore to be treated rather than classified as waste

Impacts on Lake Carey Gold project

- A Matsa treatment plant has the potential to unlock a number of other mining opportunities (Gallant and Devon GMP) and potentially recover gold from several low grade stockpiles which are currently uneconomic within the Lake Carey Gold project area due to dramatic savings in operational expenses and higher potential for economically viable projects



Revised Fortitude Stage 2 Mining Study Projected Cash Flow, cumulative A$55.4M cashflow demonstrated in red

[1] 30 September 2020 Quarterly Report

Matsa Resources Limited ("Matsa" or "the Company" ASX: MAT) is pleased to announce the results of an Engineering Concept Study ("Study") commissioned by Matsa and completed by CPC Project Design ("CPC") for a proposed Matsa owned 600,000tpa gold-ore treatment plant for the Lake Carey Gold project.

While the location of the plant has not been finalised, the likely position of the plant would be suited to the Red October, Devon and Fortitude mines and be well located centrally to any potential future resources such as Fortitude North (Figure 1).

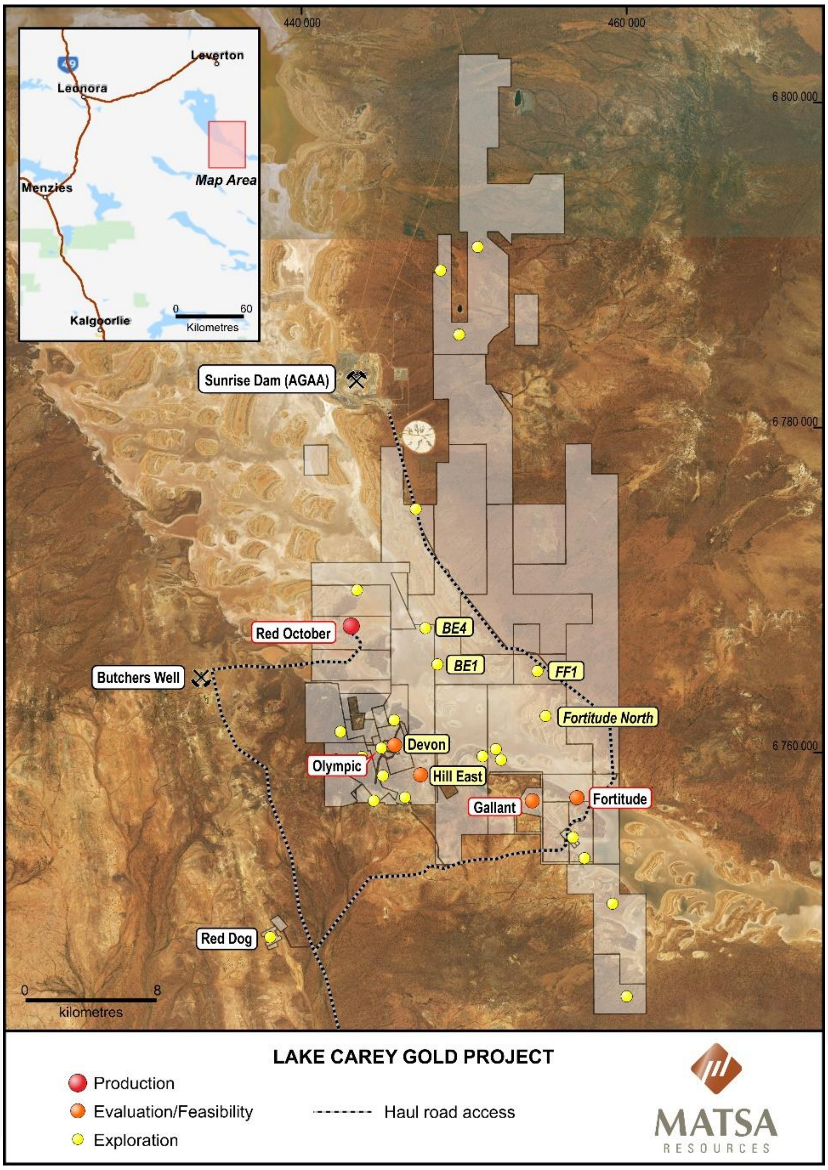


Figure 1: Lake Carey Gold Project

CPC Study Outcomes

Plant Description and Design Criteria

CPC undertook the Study on the basis of requirements and information supplied by Matsa. The full study document is included as Appendix 2.

Battery limits to estimating the Capital Cost were defined as ore feed into the primary crusher feed bin and discharge of tailings to an approved tailings storage facility. Additionally other non-process infrastructure such as offices, workshop, store buildings, power station, construction of the TSF, water supply etc were excluded from the CPC scope.

The 600,000tpa gold-ore treatment plant will consist of a primary and secondary crushing circuit, stockpile reclaim ahead of a single stage ball mill which will grind to a target size of P_{80} 125µm. The ball mill will be arranged in a standard configuration with a cyclone cluster and associated pumps. A single stage closed circuit ball mill has been selected for the grinding process. The ball mill has a diameter of 4.2 metres (m), and effective grinding length of 5.4m fitted with a 1,300 kW motor. The availability is expected to be 91.3%. Process water will be added to the mill to maintain the mill discharge slurry density at 70-75% solids. It is anticipated that the design criteria will cater for a range of feed types from oxide to fresh rock. Targeted metallurgical recovery was anticipated between 90 and 95% based on an average feed grade of 2 to 2.5g/t.

Power would be provided by a power supply contractor under a power purchase agreement. The contractor would build, own and operate a power station of the appropriate size (with diesel generators considered as the base case) and sell the power to Matsa. This is a typical contractual arrangement found throughout the Goldfields and other remote areas in WA.

Limited metallurgical data was available to CPC, so it is acknowledged that the cost estimates have been prepared at a Study accuracy level of ± 40%. Whilst the overall estimate is based on all available test work and information collected so far, a number of additional assumptions have been made based on industry norms and CPC's experience. Risks to CAPEX are limited as the design is fairly robust, considered a standard CIL (carbon in leach) plant design throughout the industry and indicative parameters are in most cases adequate to make informed equipment selections.

Gold recovery will include a gravity circuit where gold will be recovered via a centrifugal concentrator combined with an intensive leach reactor. The cyclone underflow will be split 50/50 and directed to regrind and the gravity circuit via a screen. The gold collected from the gravity circuit will be processed using a drying oven and smelting furnace to allow for separate metallurgical accounting of the gravity circuit. The final doré gold bars will be stored in the gold room safe.

Cyclone overflow slurry will be directed to the CIL circuit where it will be cyanide leached and gold adsorbed on to activated carbon. Loaded carbon will be recovered periodically to recover the gold using acid washing and hot solution elution, before being regenerated. Regenerated carbon will be returned to the last CIL tank.

The leach and CIL circuit consist of a single agitated leach tank followed by six agitated CIL tanks all connected, in series, by launders with bypass capability. The total combined retention time in the leach/CIL circuit is 24 hours. To avoid short circuiting within the leach tank, feed slurry will enter opposite to the submerged outflow position. Slurry leaves the tank by an overflow launder. Each CIL tank will be equipped with an interstage screen and a recessed impellor slurry pump. The interstage screen will allow the carbon to be retained in the respective CIL tank while permitting the pulp to flow through the screen to the next CIL tank in the circuit.

Loaded carbon from the CIL circuit is transferred to the elution circuit to start each elution cycle (once per day). The carbon is acid washed prior to desorbing the gold back into solution, electrowinning and smelting into gold doré.

Tailings slurry discharged from the last CIL tank that flows through the carbon safety screen will be collected in a hopper and pumped directly to the tailing storage facility. Decant solution from the tailings storage facility is returned to the process water pond for reuse in the plant.
Provision has been made for reagent delivery, storage and mixing and for provision of all services including process water, potable water and compressed air.

Power will be provided to all areas of the plant from the outgoing side of the switchroom located immediately adjacent to the power station building. Overall power consumption was estimated to be 15.5M kwhrs per annum. On the basis of 8,000 operating hours per annum the average power draw would be 2MW and the power station will be designed with sufficient residual capacity to reliably start the mill (largest load) on demand.

Item	Cost (A$M)
Direct Cost	28.1
Indirect Cost	5.6
Owners Cost	1.7
Contingency (20%)	7.1
Total Plant only Cost	**42.5M**

***Table 1:* Capital Cost for plant only estimated by CPC**

The estimation of the Other Capital Cost items are based on internal work by Matsa and experience with other site establishment costs.

Item	Cost (A$M)
Plant site bulk earthworks	0.5
Tailing storage initial lift	2.5
Non Process Infrastructure (NPI): (Offices, workshop, store buildings, crib & ablutions, First aid facility, pad for power station, fuel facility, HV switch room, communications and IT)	5
Mill vehicles (forklift, IT machine, LV's, Franna crane, bobcat etc)	0.6
Camp expansion (assumes a significant camp expansion will be required for firstly the construction workforce plus additional mining personnel plus the direct personnel associated with the processing plant)	5
Contingency (20%)	2.7
Total Other Capital Costs	**16.3M**

***Table 2:* Other Capital Cost Expenditure estimated by Matsa**

The total capital cost is A$58.8M.

Figure 2 shows a schematic of the Plant General Arrangement.

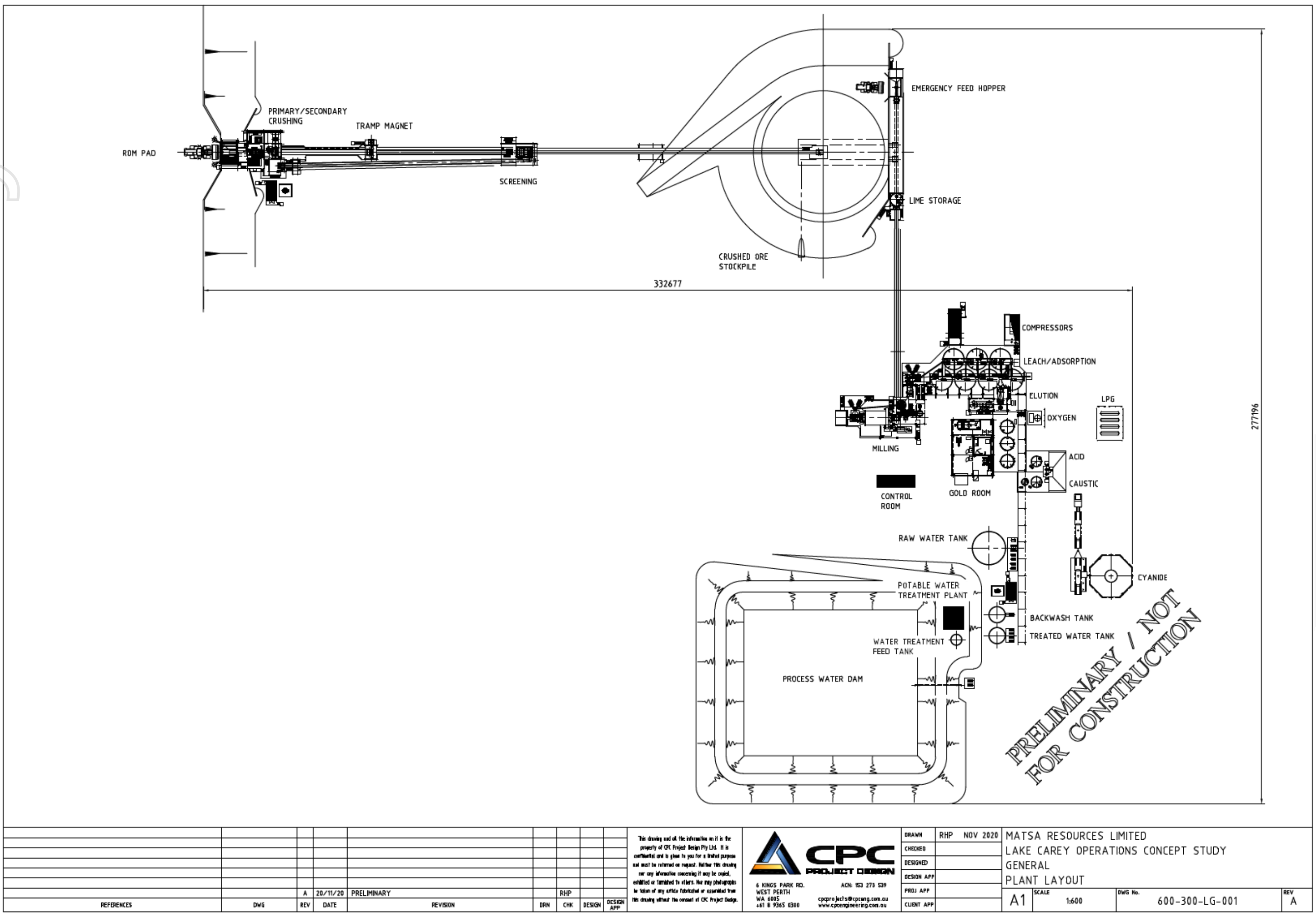


*Figure 3: **Schematic of the Plant General Arrangement***

Operating Costs

CPC estimated the operating cost based on the following:

- Labour for the onsite management and technical service activities associated with the processing plant
- Labour for the operation and maintenance of the processing plant
- Costs associated with the direct operation of the processing plant, including reagents, consumables
- Supply of maintenance materials and analytical services
- Cost of power supplied from an onsite generating plant.

Exclusions were cost of flights and accommodation for the plant workforce and cost of crusher feed and ROM pad services (to be provided by the mining contractor). These were separately estimated by Matsa based on industry experience and reference to other site costs.

Derivation of operating cost estimate by CPC and Matsa as per Tables 3, 4 and 5.

Labour	Typical manning numbers, shift patterns and current industry salaries
Power	Consumption based on motor list, sizing and utilisation
Power unit cost	Assumed to be A$0.20 per Kwhr which includes diesel fuel supply
Reagents	Based on testwork from Fortitude and industry norms & historical data
Reagents	Estimated usage and supplier pricing
Maintenance	Calculated as percentage of direct capital cost, in this case ~4.6%
Analytical services	Typical lab running costs for consumables
Exclusions	Flights and accommodation
Exclusions	Crusher feed, ROM pad services

Table 4: ***Basis of CPC Operating Cost Assumptions***

Item	Cost (A$M)
Labour	6.3
Power	4
Reagents & Consumables	5.2
Maintenance Services and Parts/Store items	0.46
Analytical services	0.67
Sub total	16.63
Contingency (20%)	3.326
Total	19.956
Cost per tonne AUD$ per dry tonne	**$33.40**

Table 5: ***Summary of CPC Operating Cost Expenditure Estimation by Area per annum***

Item	Cost (A$M)
Flights & accommodation for 35 to 40 personnel	1.22
Crusher feed and ROM pad services, dayworks provided by the mining contractor	1.5
Total	2.72
Cost per tonne A$	**$4.50**

Table 6: ***Other Operating Costs Estimated by Matsa***

Total operating cost $37.90 per tonne.

Project duration and timing

An estimated project duration of 18 months is anticipated from award of EPCM contract to operation of the new process facility. The critical path long lead items range from 20 weeks to 46 weeks. The delivery of the ball mill would be the longest lead time item.

Additional timing risks would be managed by the early appointment of a Project Director who would be tasked with co-ordination of statutory approvals, design of the TSF, co-ordination of specification, tendering, evaluation and delivery of Non Process Infrastructure and ensuring the EPCM contractor is fully integrated with the Matsa team.

Potential Throughput Upgrade from 600Ktpa to 1Mtpa

Provision would be required in the plant layout for additional leach tanks and/or a pre-leach thickener. Additionally, a second ball mill would be required to reach 1Mtpa so layout and positioning within the 600Ktpa footprint would need to be planned to allow for this to occur seamlessly at a later date. Neither of these considerations would incur any significant additional cost for construction of the 600Ktpa initial plant.

Additionally, the 2-stage crushing circuit will be initially sized for 70% utilization for day shift operation only resulting in an instantaneous rate of 200 t/h. The circuit design has the capacity for the upgraded plant throughout and can be operated for more hours. The stockpile live capacity will be reduced to 9.6 hours for the expanded 1.0 Mt/y case, down from 16 hours initially. Stockpile reclaim feeders will need to be operated duty/duty. The emergency feeder is still available as back up.

Overall the case for upgrading throughput by ~ 60% will be determined on its own merits however with some careful design and pre-planning, the expansion costs will be minimised and payback will be relatively short based on reduced operating costs.

Plant feed will be sourced from the company's Red October, Fortitude, Devon Gold mines, low grade stockpiles, and other newly discovered projects arising from systematic exploration and resource definition drilling campaigns.

Impact on Fortitude Stage 2 Mining Study

On 21st August 2019, Matsa announced an ore reserve for the Fortitude Stage 2 gold mine. This was based on a toll treatment option for the ore and a gold price of A$2,150 per ounce. All assumptions used in the ore reserve calculation and have been re-examined and are reported again below. Changes where applicable have been made based on the results of the CPC Study and these are highlighted below.

	Reserve assumption 2019	Jan 2021 - CPC Assumptions	Cashflow Impact $A million
Gold price A$ per ounce	$ 2,150	$ 2,500	$ 19
Treatment cost fresh ore	$ 55	$ 38	$ 15
Ore haulage	$ 6	$ 2	$ 4
Overall surplus (million)	$ 22	$ 55	$ 34

The revised study shows that Fortitude Stage 2 mining is attractive, with a potential cash surplus of **A$55.4M over a period of 22 months**. A sensitivity analysis indicates that the project is robust with potential for improvement to the financial model as new optimisations come to hand. Finalisation of discussions with key parties and completion of the tender process may deliver further improvements.

Metallurgical test work indicates that Fortitude ore is amenable for treatment at any of the nearby processing facilities, and will deliver very good-to-excellent gold recoveries with no deleterious elements. Gold-ore treatment costs have been based on the recently completed CPC Study.

Matsa has completed a revision of the mining study into the Fortitude Stage 2 mining operation gold deposit. The Fortitude Stage 2 gold mining operation becomes cash flow positive after month 6 (Refer Figure 3), and has the following positive financial summary:

- Capital outlay for the mining is **A$6.6M** which includes A$5.1M for pre-stripping of overburden

- Maximum cash outlay exposure **A$8.4M**
- Cash surplus **A$55.4M** after 22 months
- Assumed gold price of **A$2,500 oz**
- Production 1.029Mt @ 1.8g/t (**58,080 oz contained**) with **54,400 recovered oz gold**
- Total movement of **5.85M bcm's**
- Waste to ore ratio **14.4**
- Operating cash cost per ounce **A$1,483**

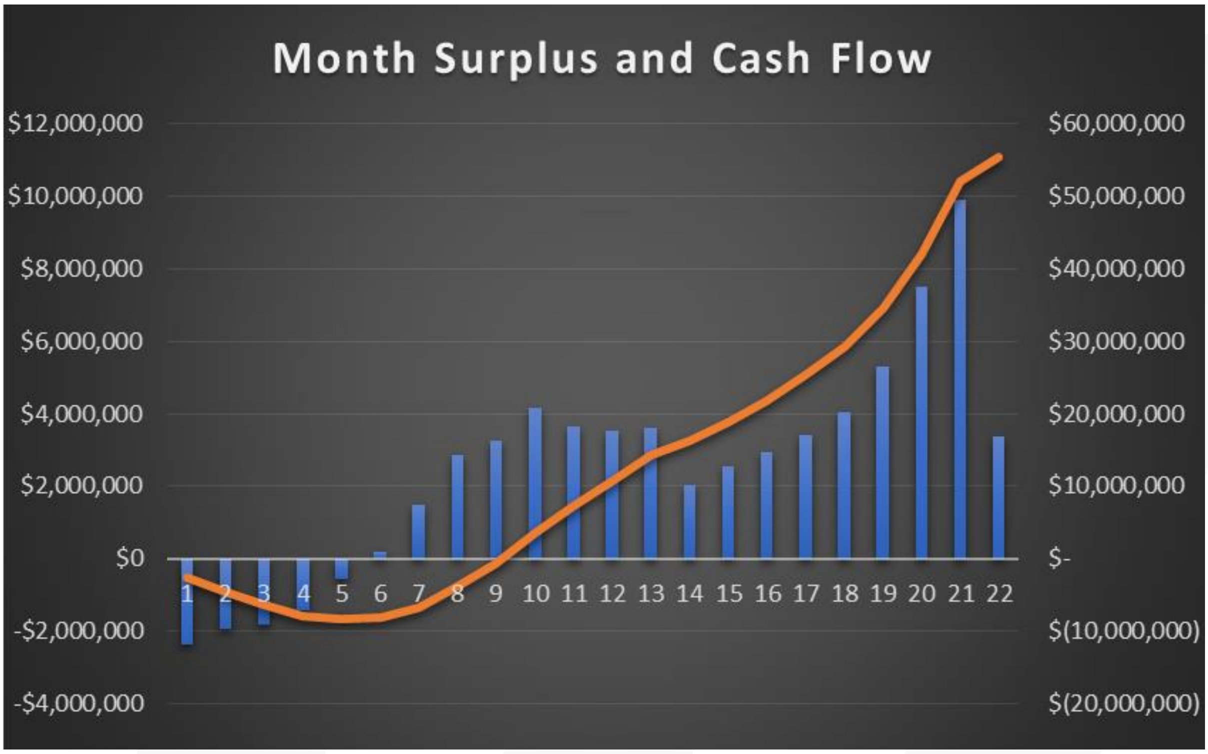


***Figure 3:* Mining Study Projected Cash Flow ($AUD)**

Importantly, all mine regulatory and statutory approvals are already in place which permits immediate commencement of mining activities.

Upon completion of drilling in 2016 and subsequent modelling and depletion of the resource due to the Stage 1 Trial Mining, the Indicated Mineral Resource at Fortitude is 2,945,000t @ 1.8g/t (170,400 oz). The total Indicated and Inferred Mineral Resource estimate for Fortitude now stands at 5,449,000 @ 2.0g/t (342,600 oz)[2].

Cut-Off grade

The Mineral Resource has been reported at a cut-off grade of 1 g/t. This is reasonable considering the style of the deposit, the proximity to process facilities and to the selection of open cut mining equipment and methods used.

[2] 30 June 2020 Annual Report

Fortitude Project Background

Matsa's Fortitude gold deposit is located in the southern portion of the prolific Laverton Tectonic Zone (LTZ). The deposit is located just 25km south of AngloGold Ashanti Australia Ltd's Sunrise Dam Gold Mine (10Moz), 60km south of Gold Fields Ltd's Granny Smith Gold Mine (11.6Moz) and 12 km southeast of Matsa's Red October Gold Mine (0.342Moz).

The Fortitude gold deposit was discovered in 1998 during regional exploration by Aurora Gold Ltd. The project was acquired by Midas Resources Ltd in 2002 who divested it to Fortitude Gold Pty Ltd in 2014.

The bulk of historic work was completed by Aurora and Midas who drilled 523 RC, AC and diamond drill holes into the deposit area. Also completed, was a number of prefeasibility and feasibility studies into a heap-leach/dump leach operation and the viability of constructing a 600tpa CIL treatment plant.

Matsa purchased the Fortitude project from the receivers of Fortitude Gold Pty Ltd in 2016[3]. Matsa subsequently completed an audit of the Resource figures and the Mineral Resource estimate was confirmed as a JORC 2012 compliant Mineral Resource[4]. Matsa subsequently commenced a diamond drilling program to provide drill core for metallurgical, geotechnical and resource definition purposes[5].

Matsa has also previously completed the required heritage, hydrogeological, flora, fauna, community consultation and geotechnical studies which were included in the mining proposal at the trial mining stage, which was approved by DMIRS in 2017. Matsa successfully conducted the Stage 1 trial mine during 2017/2018 which was completed in May 2018[6].

Study Scope Update

A mining study at Fortitude gold deposit commenced in July 2016 to evaluate the technical and financial viability of mining the Fortitude deposit. Trial mining was successfully undertaken in 2017/2018 with gold ore from the trial mine taken to AngloGold Ashanti Australia Ltd's (AGAA) Sunrise Dam Gold Mine processing plant and this current study evaluates the financial and technical viability of mining the remainder and larger part of the gold deposit (Stage 2) using a conventional open pit operation.

The Ore Reserve Estimate data, economic evaluation and Fortitude Stage 2 mining study has been comprehensively reviewed by Matsa senior management.

The study is classified as a Scoping study with a confidence level of +/-20%.

Geology and Mineralisation

Gold mineralisation at Fortitude is associated with the NNW Fortitude Shear Zone, which extends the length of the project. Ductile shearing and mineralisation is focussed within an intermediate volcanic unit adjacent to relatively undeformed mafic rocks.

Gold mineralisation forms continuous steeply dipping quartz lodes along the Fortitude Shear and is accompanied by pervasive wallrock siderite-sericite-silica alteration and vein quartz (locally +/- carbonate) with pyrite +/- arsenopyrite in the deeper sulphide zones.

Vein intensity, siderite/sericite alteration and sulphide minerals are indicative of better Au grade.

Mineral Resource Update

Matsa updated the Mineral Resource estimate to allow for depletion due to the Stage 1 trial mining.

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3 ASX Announcement 21 July 2016 – Acquisition of Lake Carey Gold Project
4 ASX Announcement 1 September 2016 – Fortitude Deposit JORC 2012 Resource
5 ASX Announcement 22 November 2016 – Drilling Commenced on New Targets at Lake Carey Gold Project
6 ASX Announcement 19th June 2018 – Fortitude Trial Mine Final Results

Total Indicated and Inferred Mineral Resources for the Fortitude gold project stands at 5,449,000t @ 2.0g/t for 342,600 oz (Refer Table 6).

Fortitude Deposit 2019 Mineral Resource Estimate (1 g/t Au cut off)							
Type	Indicated		Inferred		Total Resource		
	Tonnes kt	Au g/t	Tonnes kt	Au g/t	Tonnes kt	Au g/t	Au Oz
Oxide	222	1.9	51	2.1	273	1.9	16,900
Transition	377	1.8	125	2.0	502	1.8	29,700
Saprock	227	1.9	1	2.1	228	1.9	14,100
Fresh	2,119	1.8	2,326	2.1	4,445	2.0	282,000
Total	2,945	1.8	2,503	2.1	5,449	2.0	342,600

***Table 6:* Fortitude Gold Project Mineral Resource Estimate**

*	Figures have been rounded in compliance with the JORC code. Rounding errors may cause the column not to add up precisely.
**	Mineral Resources are reported in situ (undiluted).
***	Mineral Resources are reported to a cut-off grade of 1g/t Au.

Sections 1, 2 and 3 JORC tables for the Mineral Resource estimate are presented in Appendix 1.

Competent Persons Statement

The information in this report that relates to Mineral Resources has been compiled by Matthew Cobb, who at the time was a full-time employee of CSA Global Pty Ltd, and Richard Breyley who is a full time employee of Matsa Resources Limited. Dr Cobb is a Member of both the Australian Institute of Geoscientists and the Australian Institute of Mining and Metallurgy. Mr Breyley is a member of the Australian Institute of Mining and Metallurgy. Both Dr Cobb and Mr Breyley have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activities which they are undertaking to qualify as a Competent Persons as defined in the JORC Code (2012). Dr Cobb and Mr Breyley consent to the disclosure of this information in this report in the form and context in which it appears.

Cautionary Statement

This belief is expressed in good faith and believed to have a reasonable basis.

The material in this announcement is intended to be a summary of current and proposed activities, selected geological data, as well as Mineral Resource estimates and Ore Reserves. This data is based on information available at the time.

It does not include all available information and should not be used in isolation as a basis to invest in the Company.

This announcement includes information and graphics relating to a conceptual mining study, completed Mineral Resource estimate and a scoping study and includes "forward looking statements" which include, without limitation, estimates of gold production based on mineral resources that are currently being evaluated.

While the Company has a reasonable basis on which to express these estimates, any forward looking statement is subject to risks, uncertainties, assumptions and other factors, which could cause actual results to differ materially from future results expressed, projected or implied by such forward-looking statements.

Risks include, without limitation, gold metal prices, foreign exchange rate movements, project funding capacity and estimates of future capital and operating costs.

The Company does not undertake to release publicly any revisions to forward looking statements included in this report to reflect events or results after the date of this presentation, except as may be required under applicable securities regulations.

Any potential investor should refer to publicly available reports on the ASX website and seek independent advice before considering investing in the Company.

Matsa completed 21 diamond drill holes in 2016 for 2,257.7m. The results from the drilling were reported on 15th November 2016 and 14th December 2016. The purpose of the diamond drilling was to:

- Infill and convert Inferred Mineral Resources in the potential mining area to Indicated Mineral Resources
- Provide diamond drill core for metallurgical test work
- Provide diamond drill core for bulk density determination
- Geotechnical assessment

CSA Global consultants were contracted to carry out grade estimation for the Fortitude Mineral Resource estimate. The estimate was based on geological constraints and bulk densities provided by Matsa. Three domains were interpolated. Two primary lodes were domained striking 330^0 and 350^0 and steeply dipping towards the northeast. The supergene domain is more variable, striking between 330^0 and 360^0 and shallow dipping towards the northeast.

Top cuts applied for the estimate range from 20g/t Au to 40g/t Au. The chosen method of estimation was ordinary kriging (ok) using a two pass search strategy where the number of samples required being reduced for the second pass. Detailed commentary on all assumptions and methods used in the Mineral Resource estimate can be located in Appendix 1. In 2019, the resource was recalculated to account for the depletion of the Stage 1 trial mining.

Diamond Drill Core Sampling Techniques

The diamond core was marked up for orientation and then for assay sample selection. The core was marked up for sampling, to a maximum of 1m intervals as well as for shorter intervals which targeted visual geological boundaries. The core was then cut length wise to produce half core in a manner to protect the orientation markings. Sampling was completed from one half of the core.

The samples were placed into pre-numbered calico bags and sent off for assay.

Sample Analysis Method

The half core sent for assay sampling was crushed and riffle split and pulverised to produce a 30-50gm sample for fire assay.

Mine Design and Scheduling

The study demonstrates that under the current market conditions, Fortitude can be economically mined.

A geotechnical assessment was completed by Peter O'Bryan and Associates from which the wall design criteria were selected. Walls have been designed at 55^0 in the weathered zone with lower walls having berm widths from 5m to 8m with face angles 60^0 to 75^0 with berms every 10 vertical metres in the weathered zone and 20m in the fresh zones.

An optimisation study was completed by Orelogy using a gold price of A$1,700 per ounce and industry based costs. The optimisation studies were based on Probable Ore Reserves only. The results from this optimisation were used to design the pit. Ramps are designed at a gradient of 1 in 10 and are 24m width for dual lane in the upper portion of the pit at 1 in 10 gradient and 14m width at 1 in 9 for single lane in the lower portion of the pit.

Ore loss of 5% and dilution of 10% at zero grade was allowed for in the study.

Pit Production	Oxide	Transitional	Fresh	Total
Ore tonnes kt	141	277	611	1,029
Ore grade g/t	1.8	1.6	1.8	1.8
Waste bcm				5,447,000
Total bcm				5,853,000
Contained ounces	8,000	13,900	36,200	58,100

***Table 7:* Mine Production Fortitude Stage 2 Pit**

The pit has been scheduled to be mined to completion within 22 months. Mining will be initially worked on double shift, and then from month 14 on single shift as the work area decreases towards the bottom of the pit. Production will be 54,400 ounces of recovered gold at an operating cost of A$1,483 per recovered oz of gold. The ore tonnes mined are hauled to the treatment facility proposed in the Study for processing. Ore haulage commences in month 2 and continues to month 22 when all ore stockpiles on site have been delivered to the processing facility.

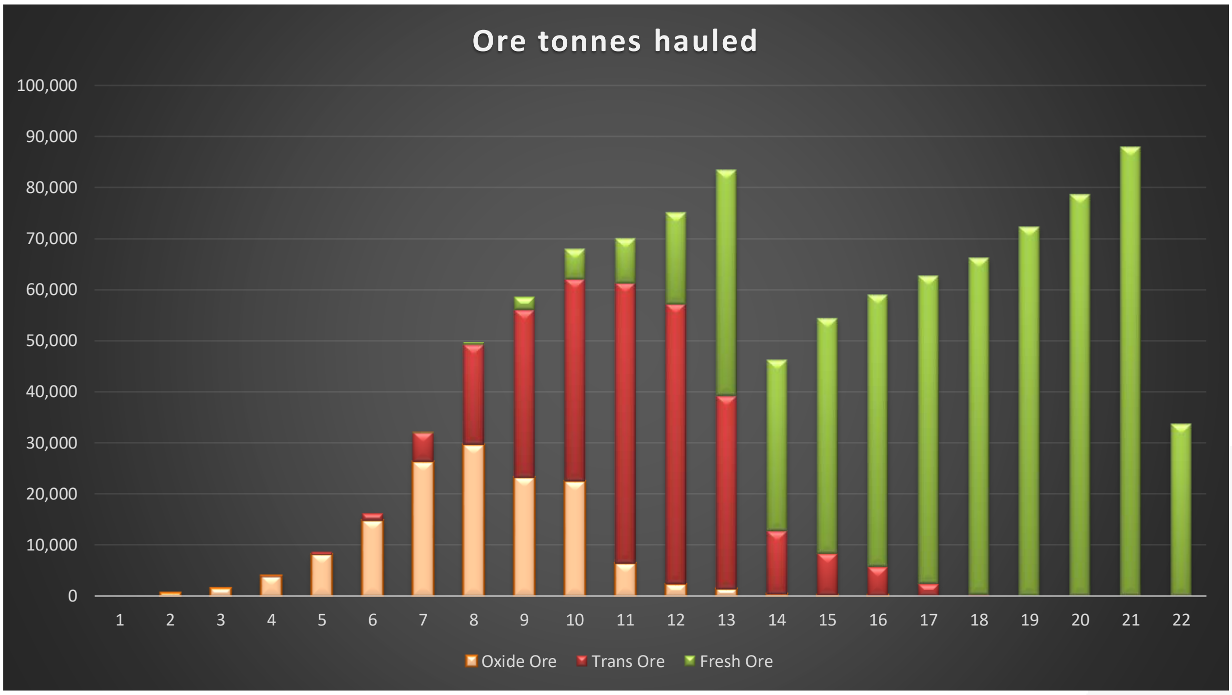

Figure 4: Ore Tonnes Mined per Pit per Month

Ore Reserves

The total Ore Reserve for the Fortitude Stage 2 mining study is 1,029,000t @ 1.8g/t (58,100 oz Au). The entire Ore Reserve is classified as Probable under the JORC 2012 code.

Fortitude Deposit 2019 Ore Reserve Stage 2 Mining Operation (1 g/t Au cut-off)							
Type	**Proven**		**Probable**		**Total**		
	Tonnes t	**Au g/t**	**Tonnes t**	**Au g/t**	**Tonnes t**	**Au g/t**	**Au Oz**
Oxide	0	0	141,000	1.8	141,000	1.8	8,000
Transitional			277,000	1.6	277,000	1.6	13,900
Fresh			611,000	1.8	611,000	1.8	36,200
Total	**0**	**0**	**1,029,000**	**1.8**	**1,029,000**	**1.8**	**58,100**

Table 8: Ore Reserve Statement

* Figures have been rounded in compliance with the JORC code. Rounding errors may cause the column not to add up precisely.

** Ore Reserves are reported inclusive of marginally economic material and diluting material delivered for treatment (diluted).

*** Ore Reserves are reported to a cut-off grade of 1g/t Au.

Dilution parameters applied to the Mineral Resource estimate as modifying factors for Reserve calculation include a mining loss of 5% and dilution of 10% at zero grade. This is considered appropriate for the open pit operation.

The reported Ore Reserve estimations are considered representative on a global scale.

Competent Persons Statement

The information in this report that relates to Ore Reserves has been compiled by Franciscus Sibbel who is a non-executive director of Matsa Resources Limited. Mr Sibbel is a Fellow Member of the Australian Institute of Mining and Metallurgy. Mr Sibbel has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activities which they are undertaking to qualify as a Competent Persons as defined in the JORC Code (2012). Mr Sibbel consents to the disclosure of this information in this report in the form and context in which it appears.

Metallurgy

The test work program scope included detailed head assays, basic comminution characterisation testing and gravity-cyanidation testing under conditions encompassing the potential treatment plants, including cyanidation testing under a range of applicable grind sizes. Test work was undertaken on composite samples derived from part diamond drill core intervals selected to spatially represent the ore types and from the experience of the trial mining. In general, the outcomes of the program results received to date are very encouraging and include:

- Detailed head assays demonstrated no significant levels of potentially deleterious elements and relatively low sulphide contents
- Comminution characterisation test work demonstrated relatively soft to moderate competency, hardness and abrasiveness characteristics including Bond Ball Mill Work Index values ranging from 8.6 kWh/t to 14.6 kWh/t
- Slurry rheology testing was undertaken on the oxide ore type samples and indicated that no viscosity issues are expected from these samples under the tested conditions
- Gravity Au recovery at the tested 180 µm P_{80} grind size demonstrated good (27%) to high (57%) Au extraction to intensive cyanidation solution under the tested conditions
- Cyanidation of the combined gravity tail samples exhibited very good (89.8%) to excellent (98.2%) Au recoveries (including gravity) at 36 hours and excellent Au extraction kinetics which allowed for similar 24 hour Au recoveries (including gravity) which range between very good (89.8%) and excellent (97.5%) values for all tested ore types
- Moderate cyanide consumption and typical lime addition values were reported

The results demonstrate characteristics suitable for treatment via any of the potential nearby toll treatment operator's processing facilities and the proposed treatment plant in the Study, with moderate comminution demands, very good to excellent gravity and cyanidation gold recoveries, moderate reagent requirements and no significant deleterious elements. Some grind size and leach residence time dependencies to gold recovery have been shown and the reported technical relationships can be assessed along with commercial related aspects to obtain maximised economic utilisation of the Fortitude gold deposit.

Processing

CPC has developed the process plant engineering design, operating and capital cost estimates for all of Matsa's Lake Carey's potential mining operations. The cost estimates have been prepared at a Concept Study accuracy level of ± 40%. The design is for a gold=ore treatment plant located at the Lake Carey gold project to treat nearby orebodies at a rate of 600,000 tonnes per year (t/y). The results of this Study have been used in this Concept study update.

Infrastructure

The Fortitude gold deposit is located in close proximity to multiple fully established mining operations complete with processing facilities. There is an established haul road connecting Fortitude deposit to the nearby operations.

The infrastructure required for the mining of the Fortitude Stage 2 are:

- Dewatering of the trial mining operation
- Recommissioning of dewatering bores and the dewatering pipeline network and the sediment and discharge water management ponds
- Minor refurbishment of existing haul roads
- Set up an administration complex which will include the office, crib, ablution and 1st aid room
- Construction of the proposed treatment plant

Approvals and Permitting

All approvals necessary for the Stage 2 mining project, except the proposed treatment plant, are in place:

- 5C License to take water – Department of Water – **Approved**
- 26D License to construct wells – Department of Water – **Approved**
- Native Vegetation Clearing Permit – Department of Environmental Regulation – **Approved**
- License to discharge water – Department of Environmental Regulation – **Approved**
- Mining Proposal – Department of Mines and Petroleum – **Approved**
- Works approval – Department of Environmental Regulation – **Approved**

Land Tenure and Social Heritage

The Mineral Resource and proposed mining area covers 3 granted mining leases which do not expire until 2029. Matsa Gold Pty Ltd (a wholly owned subsidiary of Matsa) is the 100% owner of the tenements which are located on the Mt Weld pastoral lease.

Harmony Australia Ltd holds a 1.5% net smelter royalty for production in excess of 250,000oz of gold. This royalty will not be triggered by this mining proposal. No other 3rd party royalties apply apart from the normal state government royalties.

There are no native title implications over the mine area. The archaeological and anthropological survey located one heritage site outside of the operational envelope and will be easily protected. The traditional owners have been extensively consulted and have given their approval for the mining project.

There are no impediments to operate the license.

Capital Costs Mining operation

The total estimated capital setup costs are A$6.6M which includes A$5.1M for pre-stripping overburden removal. The major capital items are pit dewatering system (A$100k), the site mobilisation and the administration office complex (A$600k). These capital cost items are based on quotes sourced through market suppliers and rates supplied by mining contractors on pre-tender submissions.

Mining Method

The mining method selected for this study is based on a conventional open pit operation involving approximately 120 tonne tracked backhoe excavators and 90 tonne haul trucks. Allowance has been made for drilling and blasting of the material as needed. Ramps are designed at a gradient of 1 in 10 and are 24m wide for dual lane in the upper portion of the pit at 1 in 10 gradient and 14m width at 1 in 9 for single lane in the lower portion of the pit.

Waste Rock Characterisation

A waste rock characterisation study has been completed to determine the suitability of the waste rock to resist erosion, as well as confirming that the waste rock is classified as non-acid forming (NAF).

Operating Costs

Operating costs have been based on unit mining costs supplied by mining contractors from pre-tender submissions and from previous actual costs incurred during the Stage 1 trial mining. Mining costs are yet to be finalised by contract.

An allowance as been made for in pit grade control and is applied on a cost per tonne of ore mined basis.

Gold-ore treatment costs used in the mining study have been adopted from the CPC Study.

Ore haulage costs have been based on expected costs to haul to the treatment plant using existing haulage rates incurred at Red October underground gold mine.

The study was based on a FIFO workforce working on double shift for the initial 13 months and then single shift. The workforce will be accommodated at Matsa's nearby Red October Mine accommodation village and commuting on a shift basis between the village and the mine site.

Market Assessment

The study was completed using an average gold price of A$2,500 per oz of gold, which is considered achievable for the project. There is a transparent quoted liquid market for the sale of gold in Australia.

Economic Factors

The total cost of the Stage 2 mining operation to Matsa is A$95.3M and consists of:

- Start-up cost of A$6.6M including A$5.1M pre-stripping costs
- Mining and haulage costs of A$40.0M
- Processing costs of A$35.2M

The revenue factor is based on the delivery of 1,029,000 ore tonnes at 1.8g/t to a central processing plant as per the CPC Study. This equates to an operating cost to Matsa of A$1,483/oz of gold for the life of the Stage 2 mining project.

Processing Method

The ore is proposed to be processed at a Matsa owned plant based on the CPC Study.

The current metallurgical test work was supplied to the CPC project team and was used in that study. Historical and current recovery data indicates that recovery for Fortitude oxide and transitional material is 95% which was used in the study and for the fresh ore the metallurgy recovery of 93% has been used.

Cut-Off Grade

The Ore Reserve has been reported at a cut-off grade of 1g/t.

Financial Modelling

Financial modelling was undertaken using monthly schedules and cash flows. A flat gold price of A$2,500 per oz was used for the project. The operation has a maximum cash exposure of A$8.4M.

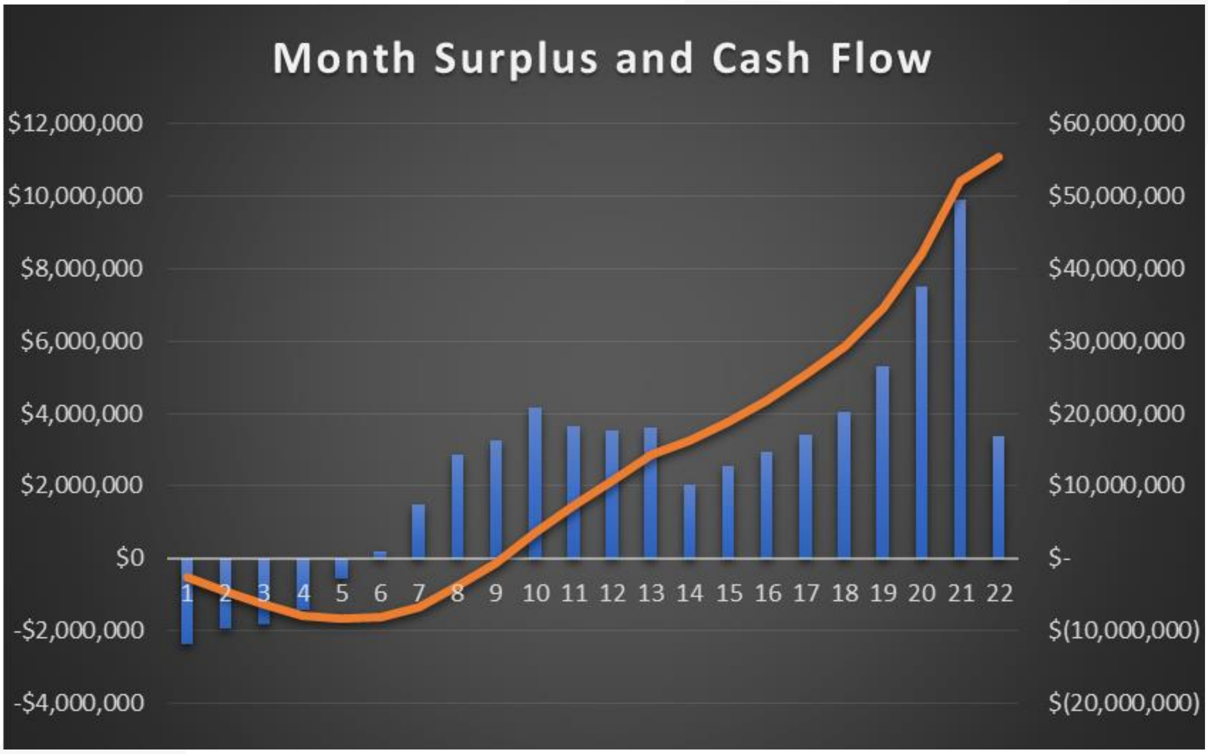

*Figure 5: **Mining Study Projected Cash Flow***

Funding Requirements

Matsa is currently evaluating options to finance the project.

Opportunities

The Fortitude Stage 2 mine represents an opportunity for Matsa, to combine Fortitude with the mining operation at Red October underground gold mine and from other mines to exploit its gold resources for the benefit of all shareholders.

Risks

A key number of risks that are normal for this type of operation have been identified, such as:

- Reduction in the A$ gold price will negatively impact on revenue
- Confidence in the geological model
- Exploration success to justify the construction of a Matsa owned treatment facility
- Achieving the assumed unit cost mining rates as used in the study
- Geotechnical stability of the pit walls

Impact on Red October

Matsa has conducted a review of the impact on the Red October underground gold mine utilising its own processing plant, as described by the CPC study. The result has a dramatic positive impact on the economics of the mine because of reduced processing and haulage costs compared to currently incurred costs.

It is envisaged that production at Red October mine would be processed along with production from other Matsa mining operations such as Fortitude. Furthermore, as the plant would likely be positioned in close proximity to existing Matsa resources, haulage distance would be expected to be reduced to less than 15km and thereby reducing the load and haulage costs to $4/t.

If these new costs were to be applied to the actual costs incurred at the Red October underground operation, the 30 September 2020 quarter production of 28,278t[7] of gold-ore, cost **savings of A$1.6M** would be realised for the quarter. This represents a **A$641/oz reduction in AISC** to **A$2,180/oz** for that quarter as illustrated in Table 9 below.

	September 2020 Quarter Actuals	**September 2020 Quarter Proforms Using CPC Study Costs**	**Difference**
Total Tonnes	28,278	28,278	-
Avg Gold Price (A$/oz)	2,668	2,668	-
Cash (C1) Costs (A$/oz)	1,781	1,781	-
AISC (A$/oz equivalent)	2,821	**2,180**	**641**

***Table 9:** Red October Gold Production Summary 30 September 2020*

This ASX announcement is authorised for release by the Board of Matsa Resources Limited.

For further information please contact:

Paul Poli
Executive Chairman
T 08 9230 3555
E reception@matsa.com.au

[7] 30 September 2020 Quarterly Report

Appendix 1 - Matsa Resources Limited – Fortitude Gold Deposit

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.*	The sampling methodology below is for Matsa drilling only. DD holes – After the core was oriented, marked up and logged the geologist marked up the sample intervals which honored geological contacts or a 1m sample interval if no geological contact was observed. Where the core was unconsolidated it was split (halved) using a paint scraper along the orientation line with the left side of the core being sampled and the right side retained. In competent core the core was quartered using an Almonte core saw with the lower left side of the core (looking down hole) being sampled. The sampling methodology below is known for Midas drilling only. RC sampling procedures adopted by Midas varied pre- and post- 2005. Prior to 2005 (FTRC001 – FTRC153) 1m bulk samples were collected from the cyclone using plastic bags. A 5m composite was then collected in a calico bag using a metal scoop. Upon receiving assays, the plastic bags containing the bulk samples within the mineralised zones were routinely re-split using a Jones riffle splitter to obtain a 2-3kg sample (1/8th split) for submission. Post 2005 drilling (FTRC154 – FTRC266) the bulk sample was collected for 1m sample intervals in plastic bags, while sub- samples were collected in calico bags at the time of drilling by splitting the bulk 1m sample through a Jones riffle splitter to get a 1/8th split. Sampling of AC cores – Drill cuttings were collected every metre in a plastic bag. 4m composite samples were collected by using a trowel or ridged plastic spear, and the approximate 2kg sample was and sent for analysis. Upon receipt of assays the bulk sample within each plastic bag in the mineralised zone was then re-sampled using on 1m intervals by scooping the sample from the bag. DD holes - Once the core was correctly matched, orientation marks were drawn onto the drill core and then propagated along the entire length. The core was then marked for sampling by the geologist, to either 1m length or by geological definitions. The core was cut lengthways in a manner to preserve the orientation line. Sampling of ½ core was then completed.
	Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.	For diamond drilling core was orientated marked up and logged prior to being marked up for sampling by the geologist. Any core loss was logged for entering into the database. Core was either halved or quartered the entire sample portion being collected in a calico bag for submission to the laboratory. For RC drilling completed by Midas 1 metre bulk samples were split using a jones riffle splitter or a rig mounted splitter beneath the cyclone. The resulting 2-3kg sample was collected in a calico bag for submission to the laboratory.
	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	The entire nominated sample was sent to the lab, crushed, riffle split to <3kg (if required) and pulverised to produce a 30-50g charge for fire assay or aqua-regia Au determination.

Criteria	JORC Code explanation	Commentary
	inherent sampling problems. Unusual commodities or mineralisation types (eg. submarine nodules) may warrant disclosure of detailed information.	
Drilling techniques	*Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic etc) and details (e.g. core diameter, triple of standard tube, depth of diamond tails, face-sampling bit or other type, whether core is orientated and if so, by what method, etc).*	A total of 25 diamond holes, 12R C/DD holes, 187 RC holes and 93 AC holes were used in the resource estimation. RC holes were completed using a standard face sampling hammer. The core diameter for diamond drilling completed by Matsa was HQ3 triple tube. Previous companies used a combination of HQ and PQ core diameters. Core was oriented using a Reflex digital core orientation tool, orientation methods by previous companies are unknown.
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.*	Core recoveries were recorded on a per run basis and entered into the geotechnical database. Zones of "nil recovery" were logged by the geologist and assigned a grade of <0.01ppm Au for resource calculation. Recoveries from RC and diamond drilling completed by Midas and Aurora were not provided.
	Measures taken to maximise sample recovery and ensure representative nature of the samples.	Diamond drilling completed by Matsa was carried out by HQ3 triple tube to maximise recovery.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	No relationship between recovery and grade has been observed.
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All core, RC and AC chips were logged by either Matsa, Aurora or Midas geologists. Diamond drilling completed by Matsa was logged for RQD's and 6 holes were logged in detail by a geotechnical consultant. Geological and geotechnical logging was completed to an appropriate level of detail required for Mineral Resource estimation, geotechnical studies, metallurgical studies and mining studies.
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	DD core has been wet and dry photographed after metre marking and orientation was completed. Qualitative geological logging was completed using a standard set of codes. These codes are considered suitable for use in defining and modelling of the deposit geology.
	The total length and percentage of the relevant intersections logged.	All drill holes utilised for the Mineral Resource Estimate have been logged.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.*	The subsampling technique below is for Matsa only. Where the core was unconsolidated it was split (halved) using a paint scraper along the orientation line with the left side of the core being sampled and the right side retained. In competent core the core was quartered using an Almonte core saw with the lower left side of the core (looking down hole) being sampled.

Criteria	JORC Code explanation	Commentary
		For Midas, ½ core was sampled. No information exists for the core sub-sampling for Aurora.
	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	For RC drilling, mineralised sample splits from the 1m samples are obtained by Jones riffle splitter or rig mounted splitter to obtain a 2 – 3kg sample for submission. For AC drilling mineralised sample splits are obtained by metal scoop from the 1m sample bags. The size of the sample is not recorded but is assumed to be similar to the RC sampling. Wetness information has not been captured in the database.
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	Samples taken by Matsa were submitted to ALS laboratories in Kalgoorlie. Samples were dried and crushed to a nominal 6-10mm through a jaw crusher. Samples over 3kg were riffle split to below 3kg and pulverized. Pulverising reduced the particle size to 90% passing 75µm. 300-400g were sub-sampled from the pulveriser bowl as an analytical pulp. The majority of sampling completed by Midas was submitted to either Ultra Trace or Genalysis Laboratories in Perth. Both laboratories abide by a generic sample preparation process where drill samples are initially dried in an oven at temperatures of approximately 105^{0}C, before crushing using a jaw crusher to achieve a product of a maximum 3mm size. Samples exceeding 3kg were split to obtain a volume that would fit in the LM5 pulveriser bowl with single pass. The crushed sample is then pulverised for a specified time in order to achieve a nominal 80% to 95% passing 75 micron size. A 250g sub-sample was then collected and placed in a pulp envelope for analysis. The sample preparation techniques are accepted routine procedure for the style and nature of gold mineralisation at Fortitude.
	Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.	QAQC procedures adopted by Matsa included the insertion of appropriate certified standards and course blanks into the sample sequence preferentially in the ore zones as well as the use of laboratory repeats. 5% of samples were also submitted to an umpire laboratory, 2.5% of these were randomly selected and 2.5% selected by the geologist. Midas QAQC protocols involves submission of standards, blanks, and field duplicate samples. Laboratory repeat analyses have also been supplied to Runge and a large number of pulp samples were also submitted to a secondary laboratory for independent checks. In general all certified standards and blanks returned the expected results within an acceptable error. Laboratory repeats and umpire laboratory results had reasonable repeatability with no obvious bias as would be expected from a gold deposit with a moderate – low nugget affect.
	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.	Matsa did not undertake any second half sampling of drill core as the samples were required for metallurgical test work. 802 duplicate samples were taken by Midas. A scatter plot showed reasonable repeatability with some outliers as expected in lode gold deposits. There was no inherent bias observed.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	The split/cut sample size of 2-3kg to be pulverised with 200-300g sub samples are appropriate for the grain size of the material being sampled.
	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is	Matsa submitted all samples to ALS in Kalgoorlie for analysis by fire assay with a 30g charge.

Criteria	JORC Code explanation	Commentary
Quality of assay data and laboratory tests	*considered partial or total.*	Ultra Trace laboratories was the major provider of assay services to Midas. Assay methods were either Fire Assay or Aqua Regia, with 40g charge used in both methods. ALS laboratories were the principal provider of assay services during the Aurora phase of drilling, while Genalysis laboratories also provided assay services. Analysis was conducted using either Fire Assay or Aqua Regia, with both methods using a 50g charge. Genalysis also conducted both Fire Assay and Aqua Regia analysis, using a 25g charge for the Fire Assay, and a 10g charge for Aqua Regia. Fire assay and aqua-regia analysis methods for gold are appropriate gold analysis methods for ore deposits of this type. Both methods can be considered near 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.	Not Applicable.
	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.	QAQC procedures adopted by Matsa included the insertion of appropriate certified standards and course blanks into the sample sequence preferentially in the ore zones as well as the use of laboratory repeats. 5% of samples were also submitted to an umpire laboratory, 2.5% of these were randomly selected and 2.5% selected by the geologist. Midas QAQC protocols involves submission of standards, blanks, and field duplicate samples. Laboratory repeat analyses have also been supplied to Runge and a large number of pulp samples were also submitted to a secondary laboratory for independent checks. In general all certified standards and blanks returned the expected results within an acceptable error. Laboratory repeats and umpire laboratory results had reasonable repeatability with no obvious bias as would be expected from a gold deposit with a moderate – low nugget affect.
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.*	No verification of significant intersections was carried out by either independent or alternative company personnel.
	The use of twinned holes.	Six out of the 93AC holes in the resource area have been twinned by RC holes. Intercepts and grades from both hole types are similar, with the AC having slightly lower mean grade.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	Data entry, verification and storage procedures are not formally documented. All hard copy sample cut sheets and assay files are retained for database validation.
	Discuss any adjustment to assay data.	Not applicable.
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.*	Matsa All drill holes were surveyed using a Sokkia GSR2650 LB differential GPS which has an accuracy of +/-10cm both vertically and horizontally. Down hole surveys were carried out by Gyro Australia Pty Ltd using an SDI high speed true north seeker keeping gyro. Midas

Criteria	JORC Code explanation	Commentary
		All drill holes used in the resource estimate have been accurately surveyed by contract surveyors using an RTK GPS instrument. Downhole surveys have been conducted by the drilling company at regular intervals using either a single shot or a gyro tool for RC and DD holes. Downhole survey of AC holes was not done.
	Specification of the grid system used.	Midas and Aurora used the AMG84_51 grid system. Matsa used the MGA94_51 grid system.
	Quality and adequacy of topographic control.	A high accuracy (method unknown) topographic DTM supplied by Midas has been used.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.*	Drill spacing of approximately 25m (along strike) by 25m (on section) was considered adequate to establish both geological and grade continuity. Towards the edges of the deposit the drill spacing widens to either 50m (along strike) by 25m (on section) or 50m (along strike) by 50m (on section).
	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.	Data spacing and distribution has been sufficient to permit delineation and to confirm grade continuity of the narrow lodes and supergene domains.
	Whether sample compositing has been applied.	Samples were composited to 1m downhole lengths.
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.*	The orientation of bulk of the drilling is approximately perpendicular to the strike of the steeply dipping mineralisation and is unlikely to have introduced any significant sampling bias.
	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.	Not applicable.
Sample security	*The measures taken to ensure sample security.*	Samples were delivered directly to ALS laboratories in Kalgoorlie by Matsa personnel. The chain of custody was not broken by any 3rd parties.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	No audits or reviews of sampling techniques were undertaken.

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 Mineral Resource covers 2 granted mining leases M39/710 and M39/1065. Both tenements expire in 2029. Matsa Gold Pty Ltd is the 100% owner of the tenements which are located on the Mt Weld pastoral lease. Harmony Australia Ltd hold a 1.5% net smelter royalty for production over 250,000oz. There is no native title claim over the area. One mapped heritage site in the area will not impact on mine planning or production.
	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.	There are no known impediments to obtaining a license to operate in the area.
Exploration done by other parties	*Acknowledgment and appraisal of exploration by other parties.*	Exploration drilling was conducted by Aurora Gold Limited (Aurora) between October 1998 and February 2002. Midas Resources Limited (Midas) acquired the project from Aurora in October 2002. Midas has drilled in excess of 380 drill holes both in and around Fortitude to test for extensions to the Fortitude system. Matsa acquired the project in 2016 and has drilled 21 diamond holes for 2,257.7m.
Geology	*Deposit type, geological setting and style of mineralisation.*	The Lake Carey Project of which the Fortitude deposit forms a part is situated on the Fortitude Shear, which along with the Bindah Shear located just west, forms a narrow corridor of ESE trending greenstones which are bounded to the east and west by granitoid terrane. As the Fortitude-Bindah system extends north the greenstone pile thickens and lies host to numerous large gold mineralisation systems. To the south the Fortitude-Bindah system appears to attenuate and eventually terminate against granitoids of the Eastern Gneiss Terrane. The greenstone sequence located within the Fortitude tenement is comprised of highly foliated felsic to intermediate volcanic rocks with relatively undeformed mafic volcanic units to the east and west in contact with granite. The whole greenstone package varies in width from <2km at the southern end of the tenement to approximately 8km at the northern end. Major north to north-northwest trending shear zones occur within the greenstones and the granite to the east, in particular along geological contacts. The main structural features are the Fortitude Shear along the eastern intermediate-mafic contact and the more north- westerly trending Bindah Shear, along the western intermediate-mafic contact Gold mineralisation is typically associated with the Fortitude Shear Zone, a north-northeast striking dextral shear which extends the length of the Lake Carey tenement. To the north, it horsetails into the Wilga fault system and in the south it continues into the Kirgella Gneissic Dome. Gold mineralization is also associated with the Bindah Shear, particularly at the old Bindah Mine to the southwest. The Fortitude deposit is hosted within sheared felsic to intermediate volcanic rocks and minor ultra mafics, and is covered by up to 10m of lacustrine clays and aeolian sands surrounding Lake Carey. Gold mineralisation occurs within a steeply dipping shear system, and is associated with pervasive carbonate-sericite-silica alteration along with pyrite-arsenopyrite mineralisation. Remobilisation

Criteria	JORC Code explanation	Commentary
		of gold has also resulted in the formation of flat lying zones of supergene mineralisation within the regolith. Weathering extends to a depth of 60-80m.
Drill hole Information	*A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length.*	Not applicable, the company is not reporting exploration results.
	If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	Not applicable, the company is reporting a Mineral Resource based on historic drilling information. A summary of the drilling information has been provided in Section1.
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.*	Not applicable, the company is not reporting exploration results.
	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.	Not applicable, no intercepts have been reported.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	Not applicable, no metal equivalent results have been used.

Criteria	JORC Code explanation	Commentary
Relationship between mineralisation widths and intercept lengths	*These relationships are particularly important in the reporting of Exploration Results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	Mineralisation styles tend to change from narrow vertical lodes in the north, to shallow dipping supergene-hypogene mineralisation in the south. The shear hosted lode mineralisation strikes at roughly between 330° and 350° and is vertical to very steeply dipping to the east north-east. The supergene mineralisation is somewhat more variable with strike roughly between 330° and north - south and the lenses are generally flat lying or shallow dipping to the east north-east. The orientation of the drilling is approximately perpendicular to the strike and dip of the shear hosted mineralisation and is unlikely to have introduced any significant sampling bias.
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.*	Not applicable.
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.*	Not applicable.
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.*	Not applicable.
Further work	*The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).*	The mineralisation at Fortitude is open and plunges towards the north. Further drilling is warranted to test for potential underground resources.
	Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	Not applicable.

Section 3 Estimation and Reporting of Mineral Resources (Criteria listed in the preceding section 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.*	The database used to generate the Mineral Resource estimate was supplied to CSA Global by Matsa as a validated Surpac database. Use in Surpac requires the passing of a set of routine validation steps checking for sample overlaps, sample duplications, missing downhole and missing collar survey data.
	Data validation procedures used.	Historic data was validated during importation into the Matsa database and found to be clean. Sections were plotted and validated against historic hard copy sections. Planned drill holes were ground trothed against historic collars in the field. Matsa is satisfied that the drill hole database has been thoroughly validated.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*	CSA Global staff have not visited site. Matsa staff have made numerous visits to site throughout the conduct of exploration campaigns during 2016 and closely supervised the 2016 resource definition, metallurgical and geotechnical drilling programs.
	If no site visits have been undertaken indicate why this is the case.	Not Applicable.
Geological interpretation	*Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.*	The geological interpretation of the Fortitude deposit was completed by Matsa. The model is well constrained by a long history of discovery and mining of similar deposits within the region. Structural and geological data collected from diamond drill core adequately characterizes the mineralization style to permit a high degree of confidence in the interpretation of the Fortitude deposit. The Competent Persons are satisfied that the geological model is robust and correlates well to field observations and drill hole data.
	Nature of the data used and of any assumptions made.	Detailed geological logging, including alteration and oxidation state data, along with logged intensity of shearing and quartz vein content were used, in conjunction with chemical assays, in order to develop the geological interpretation.
	The effect, if any, of alternative interpretations on Mineral Resource estimation.	Narrow Archaean Lode Gold deposits with a supergene expression and a low grade halo are a common style of mineralization encountered in the Eastern Goldfields of Western Australia. Their morphology and petrogenesis are well characterized, and do not readily offer materially different interpretations. The Competent Persons do not consider that an alternative interpretation of the Fortitude deposit is likely to yield material differences to the Mineral Resource estimate.

Criteria	JORC Code explanation	Commentary
	The use of geology in guiding and controlling Mineral Resource estimation.	The Fortitude deposit is hosted by the Fortitude Shear, which represents the sheared contact between undifferentiated intermediate rocks and greenschist facies mafic / ultramafic rocks. The modelling of geology, along with the presence and intensity of quartz veining is a strong guide to the interpretation of the extents of mineralization.
	The factors affecting continuity both of grade and geology.	Continuity of grade along strike and at depth is controlled by the presence / absence of the host shear fabric, intensity of quartz veining, and the degree of chemical alteration the host rocks have undergone. Each of these characteristics may be traced between drill holes using visual characteristics.
Dimensions	*The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	The Fortitude Mineral Resource is contained within an area defined by a strike length of 1,490 m and 200 m across strike, along an azimuth of 350. The deposit is bounded by the extents 456,807 mE to 457,570 mE and 6,756,451 mN to 6,757,880 mN. The deposit lies within 375 m of the surface, and is open at depth, and potentially to the north along strike.
Estimation and modelling techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	The Mineral Resource has been completed using 3 individual statistical domains built using a nominal 0.2 g/t Au cut-off grade. Samples were composited to 1 m intervals base on assessment of the raw input sample intervals. Hi grade cuts ranging from 20 to 40 g/t Au were applied to the mineralization domains following statistical analysis. Analysis was completed using GeoAccess software. Quantitative Kriging Neighbourhood Analysis was undertaken using Supervisor software, to assess the effect changing key neighbourhood parameters had on the block grade estimates. Kriging Efficiency and Slope of Regression were assessed for a variety of block sizes, minimum and maximum input samples, search dimensions and discretization grids. A two pass search strategy was used where the minimum number of samples required for estimation was reduced in the second pass. For blocks not informed after two passes, the Sichel mean grade for that particular statistical domain was assigned. Ordinary Kriging (OK) was the chosen method of interpolation for the grades of mineralized zones and the low grade halo. All grade estimation was undertaken in Surpac 6.6.2 software.
	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	A previously published Mineral Resource estimate was completed in 2012. Statement of this resource is publicly available and, after consideration for updated drilling data and re-interpretation of mineralized lodes, grade and tonnage values for this previous estimate compare reasonably to the current estimate.
	The assumptions made regarding recovery of by-products.	No by or co-products have been considered.

Criteria	JORC Code explanation	Commentary
	Estimation of deleterious elements or other non-grade variables of economic significance (eg. sulphur for acid mine drainage characterisation).	No deleterious elements were recorded within the available assay data, and none have been considered in this Mineral Resource estimate.
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	Blocks of dimensions 5 x 20 x 5 m were used to subcell to a minimum size of 1.25 x 5 x 1.25 m. This block size was selected on the basis of quantitative analysis using data from the most well informed primary mineralised domain. Dimensions represent approximately half the drill hole spacing in the X and Y dimensions for well informed regions of the model.
	Any assumptions behind modelling of selective mining units.	No assumption of selective mining unit has been made as part of the Mineral Resource estimate.
	Any assumptions about correlation between variables.	The model considers only one variable; Au and so no correlations have been considered.
	Description of how the geological interpretation was used to control the resource estimates.	Mineralisation domain boundaries were treated as hard boundaries for the purposes of selection of input samples data. These boundaries were created on the basis of logged geology, alteration and says values.
	Discussion of basis for using or not using grade cutting or capping.	High grade cuts were sued to limit undue influence of extreme outliers values in the dataset described above.
	The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	The Mineral Resource estimate was validated visually via qualitative comparison on screen between estimated block grades in drill hole assays in section, and also via swath plots generated in the X, Y and Z directions.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages have been determined on a dry in-situ basis. No moisture values were reviewed.
Cut-off parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource has been reported at a cut-off grade of 1 g/t Au. The Competent Persons consider this reasonable when considering the style of deposit, its proximity to processing infrastructure and the assumption of open pit mining methods being employed.
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.*	Mining optimisation studies conducted on historic Mineral Resource estimates for the Fortitude deposit show that it is amenable to open pit mining at grade similar to those reported within this MRE. Open pit mining is considered the most appropriate method of extraction to consider in any future studies. Both the Competent Persons believe that there is a likely prospect of economic extraction. A minimum downhole intercept width of 2m has been applied. No other considerations were made. Detailed assumptions regarding dilution and minimum mining widths should be included in any future optimisation and Mine Planning work conducted by Matsa during any Ore reserve estimation.

Criteria	JORC Code explanation	Commentary
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.*	Historic and current metallurgical test work has been completed indicating good recoveries of greater than 92% through a regular CIL processing plant.
Environmental factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	No considerations regarding waste and process residue disposal have been made as part of this MRE. Given the proximity of the deposit to existing processing infrastructure, it is likely that such infrastructure will be used for processing and will include residue disposal options.
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.*	CSA Global used fixed density values, assigned on the basis of regolith classification of the material within the model. Fresh material was given a value of 2.8, Slightly weathered material; 2.7, transitional oxide material 2.4, fully oxidized material and transported (colluvial) material; 2.0. 128 bulk density measurements were undertaken representing all ore types. Bulk density determination was carried out by ALS laboratories using the wax immersion method on dried core for oxidised rocks to account for voids, vugs and porosity. In transitional and fresh rocks bulk densities were analysed by both the water immersion method and the wax immersion method (ALS). The wax immersion method was given priority when assigning the bulk density to the various rock types.
	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.	The wax immersion method on dried core carried out by ALS laboratories adequately accounts for voids, vugs and porosity.

Criteria	JORC Code explanation	Commentary
	Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	The average bulk density rounded to 1 decimal place was used for all material types except for oxide where a lower value was chosen. This is to account for any possible bias in sample selection.
Classification	*The basis for the classification of the Mineral Resources into varying confidence categories.*	The Mineral Resource was classified as Indicated and Inferred, taking into account the geological understanding of the deposit, the density and quality of input data (including drill hole spacing) and kriging estimation statistics.
	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).	Both the Competent Persons consider that the classification is appropriate when consideration is given to all of the above factors.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	The classification appropriately reflects the view of both Competent Persons.
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*	Internal Audits were conducted by CSA Global which verified methodology and parameters used in the generation of the Mineral Resource estimate.
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 Mineral Resource accuracy is communicated through the classification assigned to the deposit. The Mineral Resource estimate has been classified in accordance with the JORC Code, 2012 Edition using a qualitative approach. All factors that have been considered have been adequately communicated in Section 1 and Section 3 of this Table.
	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.	The Mineral Resource statement relates to a global estimate of in-situ tonnes and grade.
	These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	The deposit has not, and is not currently being mined.

Section 4 Estimation and Reporting of Ore Reserves (Criteria listed in the preceding section also apply to this section)

Criteria	JORC Code explanation	Commentary
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.*	The Fortitude Mineral Resource estimate 2017 (fortitude)ok_final20170130.mdl) was prepared by CSA Global Consulting using databases and geological interpretation supplied by Matsa.
	Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	Mineral Resources are inclusive of Ore Reserves.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*	The competent person is a non-executive director and consultant for Matsa and has visited the site a number of times.
	If no site visits have been undertaken indicate why this is the case.	Not applicable.
Study status	*The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.*	The Stage 2 mining study is a pre-feasibility/scoping level study.
	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.	The Stage 2 mining study is a pre-feasibility study with a level of confidence of +/-20%. The mine is technically and economically viable and Modifying Factors have been considered.
Cut-off parameters	*The basis of the cut-off grade(s) or quality parameters applied.*	A cut-off grade of 1g/t Au is based on an economic assessment and current market parameters.
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 reported "Stage 2 mining study" was a pre-feasibility study. Input factors into optimization have a level of confidence of +/-20%. The economic outcome is based on detailed mine designs.
	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 selected mining method of typical open pit truck and shovel is appropriate for this type and configuration of mineral deposit.
	The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control	Pit slopes are based on a detailed geotechnical assessment by external consultants. Walls have been designed at 55^0 in the weathered zone with lower walls having berm widths from 5 to 8m with face angles 60 to 75 degrees with berms every 10 vertical metres in the

Criteria	JORC Code explanation	Commentary
	and pre-production drilling.	weathered zone and 20m in the fresh zones. No further pre-production drilling is required. Grade controls costs have been applied on a per tonne of ore basis.
	The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).	The Fortitude Mineral Resource estimate 2017 (fortitude_ok_final20170130.mdl) for pit optimization. Pit optimizations were carried out with appropriate slope angles, dilution, recovery, mining costs and metallurgical factors.
	The mining dilution factors used.	10% of waste at zero grade was added to the ore to account for dilution.
	The mining recovery factors used.	Recovery factor of 95% was used.
	Any minimum mining widths used.	A minimum mining width of 15m was used.
	The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.	Inferred Mineral Resources have not been used for the mining study.
	The infrastructure requirements of the selected mining methods.	Site establishment will require the dewatering of the trial mining pits, installation of administration complex and the refurbishment of a haul road.
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.*	Ore will be processes at a 600,00 tonne per annum Plant as outlined in the CPC Concept study.
	Whether the metallurgical process is well-tested technology or novel in nature.	CIL technology is well tested and widely used.
	The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.	3 oxide, 1 transitional and 1 fresh composite was selected for metallurgical test work. The selection was made such that the material was spatially representative of the entire deposit at a grade similar to the overall mined grade. The information gained in the trial mining was also used.
	Any assumptions or allowances made for deleterious elements.	Not applicable. No deleterious elements were identified in the test work.
	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.	Not applicable. No bulk sample or pilot scale test work was undertaken.
	For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	The Ore Reserve estimate has been based on mineralogical and metallurgical factors as discussed in the CPC Concept Study.

Criteria	JORC Code explanation	Commentary
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.*	Matsa has completed detailed flora and fauna, waste rock characterization and hydrogeological studies. Waste rock will be stored in nearby waste dumps constructed to form a stable landform. The project does not require the construction of a tailings storage facility. Waste rock characterization test work indicates the waste material mined is non acid forming (NAF). There are no known environmental impediments to the commencement of mining.
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.*	The project is located nearby significant mining infrastructure and processing plants. An existing haul road is owned by Matsa and connects the project to nearby mines. The mine would be developed as a FIFO operation with the workforce residing at Matsa's Red October camp only 30 minute drive from site.
Costs	*The derivation of, or assumptions made, regarding projected capital costs in the study.*	The costs have been derived from direct quotes received from suppliers and pre-tender submissions by contractors.
	The methodology used to estimate operating costs.	Operating costs have been derived from unit rates received in pre-tender submissions by contractors.
	Allowances made for the content of deleterious elements.	Not applicable.
	The source of exchange rates used in the study.	Not applicable, all costs have been quoted in Australian dollars.
	Derivation of transportation charges.	Ore haulage costs have been derived from quotes provided by local contractors and experience from the trial mining project.
	The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.	The treatment costs are based on the CPC Concept Study.
	The allowances made for royalties payable, both Government and private.	WA government royalties are included. No other royalties apply for this project. The Harmony royalty will not be triggered by this project.
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.*	Revenue factors are based on forecast production rates, head grades and predicted metallurgical recoveries from the mine schedule. A flat gold price of A$2,150/ounce was used based on the current market price.
	The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	

Criteria	JORC Code explanation	Commentary
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.*	Not applicable. No detailed market assessment was undertaken or is required for gold.
	A customer and competitor analysis along with the identification of likely market windows for the product.	Not applicable for gold sales.
	Price and volume forecasts and the basis for these forecasts.	Not applicable.
	For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	Not applicable.
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.*	Not applicable, no NPV has been reported.
	NPV ranges and sensitivity to variations in the significant assumptions and inputs.	Not applicable, no NPV has been reported.
Social	*The status of agreements with key stakeholders and matters leading to social license to operate.*	Matsa has completed significant consultation with the traditional owners, local shires and station owners. There are no social impediments to the commencement of mining.
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.	No material naturally occurring risks have been identified.
	The status of material legal agreements and marketing arrangements.	Matsa is required to go to tender on the mining and ore haulage contracts and will need to complete a full feasibility study to justify building its own mill.
	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.	All government approvals are in place.

Criteria	JORC Code explanation	Commentary
Classification	*The basis for the classification of the Ore Reserves into varying confidence categories.*	Probable Ore Reserves are based on Indicated Mineral Resources subject to economic viability.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	The estimate appropriately reflects the view of the competent person who has signed a JORC consent form to that effect.
	The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	No applicable, no Probable Ore Reserves have been derived from Measured Mineral Resources.
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*	The Ore Reserve estimate, data, economic evaluation and pre-feasibility study have been comprehensively reviewed by Matsa senior management.
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.*	The relative accuracy and confidence in the Ore Reserve estimate is consider high. Geostatistical and statistical procedures used in the Resource were completed by qualified external consultants.
	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.	All currently reported Ore Reserve estimations are considered representative on a global scale.
	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.	Appropriate modifying factors for dilution and ore loss have been applied based on the experience of the competent person.
	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.	Not applicable. Previous production exploited oxide ore where future production will target sulphide ore, as such the modifying factors are treated differently.

Appendix 2

MATSA RESOURCES LIMITED

LAKE CAREY OPERATIONS

CONCEPT STUDY

Document Number: 60001-RP-G-001

Revision B

19 November 2020

DOCUMENT HISTORY AND STATUS

B	19/11/2020	Issued for Study	GN	RD	RD
A	18/11/2020	Issued for information	GN	RD	
Revision	Date	Description	Prepared	Reviewed	Approved

Printed:	19 November 2020
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Name of document 2:	
Name of document 3:	
Document version:	Revision B
Revision Date	19 November 2020
Document number:	60001-RP-G-001

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

APPENDICES

1 EXECUTIVE SUMMARY

CPC Project Design (CPC) has developed the process plant engineering design, operating and capital cost estimates for Matsa Resources Limited (MAT) Lake Carey Operations. The cost estimates have been prepared at a Concept Study accuracy level of ± 40%.

The design is for a treatment plant located at the Lake Carey Operation to treat nearby orebodies at a rate of 600,000 tonnes per year (t/y). The treatment plant will consist of the following major processing areas.

- 2 stage crushing to open stockpile
- Reclamation via belt feeders and an emergency bin/feeder
- Single stage ball milling in closed circuit with hydro-cyclones
- Gravity recovery circuit
- Leach/CIL
- Split AARL elution
- Reagent storage and distribution
- Water and air services.

Limited metallurgical test work has been completed; however, this data has been utilised for process flowsheet selection and equipment sizing as well as for the operational cost inputs.

The operating cost estimate summary for the process plant is presented in Table 1.1.

Table 1.1 OPEX Summary

	\$/y	\$/t feed
OPEX	**20,002,554**	**33.4**

The capital cost estimate for the process plant is presented in Table 1.2.

Table 1.2 CAPEX Summary

Item	Cost (AUD \$m)
Direct cost	28.1
Indirect cost	5.6
Owners cost	1.7
Contingency	7.1
Project Total	**42.5**

The overall schedule is approximately 18 months to bring the project into operation. The site construction period is 12 months.

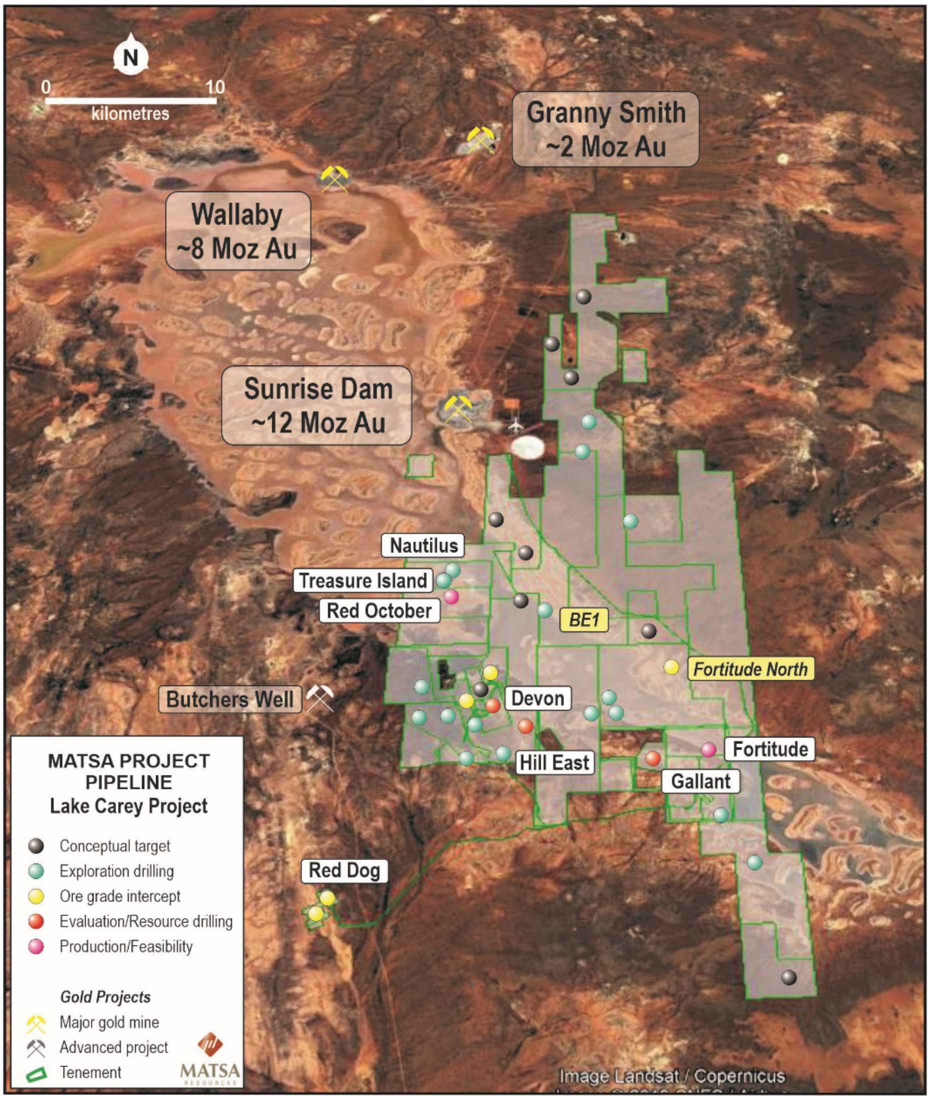
Financial modelling is outside the scope of this report and is to be conducted by MAT.

2 PROJECT BACKGROUND

Matsa holds approximately 563 km^2 of tenement grounds at the Lake Carey Gold Project, located 70 km south of Laverton. Figure 2-1 shows the location of MAT held tenements, in relation to the other known projects.

Mining currently occurs at the Red October mine, with ore delivery to the nearby Sunrise Dam plant for third party toll treatment. A recently completed study outlines a compelling case for the commencement of mining at Fortitude deposit. Current highly prospective targets include both the Devon and Fortitude North deposits.

Figure 2-1 Project Location



The Process Design Criteria (PDC) is based on a combination of available test work results, MAT's basis of design data, standard industry practice and CPC's recommendations.

The Process Flow Diagrams produced has been used as input for other engineering disciplines, and for the generation of the Mechanical Equipment List (MEL).

3 SCOPE

3.1 Scope of Work

The scope of work (SOW) was to provide a conceptual level study including capital and operating costs for a stand-alone 600,000 t/y treatment plant.

3.2 Scope of Services

In order to achieve the scope of work the following activities were conducted:

- Review available ore characteristic and metallurgical test work data
- Develop a process design criterion based on the available data, supplemented with typical values based on experience
- Develop process flow diagrams
- Develop a mechanical equipment list, inclusive of equipment sizing
- Develop capital and operating cost estimates to an accuracy level of ±40%
- Develop general arrangement drawings (plan and elevation) of the process plant
- Estimate of the operating and maximum power draw
- Timelines to construction
- Develop capital and operating cost estimates to an accuracy level of ±40%.

3.3 Battery Limits

The battery limits for this study include:

- Ore feed (by mining contractor) into the primary crusher feed bin
- Construction camp and operations village
- Non process plant infrastructure (offices, ablutions, workshops, laboratory, warehouse, crib rooms)
- Communications
- Raw and fresh water supply to the process plant
- Tailings storage facility
- Power generation and distribution to the process plant.

4 PROCESS DEVELOPMENT

4.1 Overview

This section describes the main features and design of the process plant. Reference can be made to the process flow diagrams attached as Appendix A.

A 2 stage crushing and single stage milling circuit is proposed. Secondary crushed material will feed a primary ball mill.

A gravity circuit is incorporated into the grinding facility to capture and process free gold.

Material exiting the grinding circuit is screened and processed through a conventional leach/CIL circuit. Gold is stripped off the loaded carbon via a split AARL elution circuit.

4.2 Process Design Criteria

The process design criteria and reference sources used as the basis for the study are contained in Appendix B. Key design data is summarised in Table 4.1.

Table 4.1 Design Criteria

Area	Description	Unit	Value
Operating schedule	Annual throughput	t/y	600,000
	Crushing circuit operating hours	h/y	6,162
	Crushing circuit nominal rate	t/h	98
	Crushing instantaneous rate	t/h	200
	Grinding circuit operating hours	h/y	7,998
	Grinding circuit nominal rate	t/h	75
Material characteristics	Moisture content	%	5
	Bond RMWI	kWh/t	11.6
	Bond BMWI	kWh/t	8.6 – 14.6
	Bond abrasion index		0.054 – 0.173
	SG solids	t/m^3	2.7 – 3.05
	SG water	t/m^3	1.03
	Ore bulk density	t/m^3	1.60
	Ore grade, Au	g/t	2.75
Crushing	Configuration		2 stage crush, closed circuit
	ROM feed size F_{100}	mm	800
	ROM feed size F_{80}	mm	400
	Crushed ore P_{80}	mm	20
	Stockpile capacity	h	16 (mill feed)
Grinding	Configuration		Single stage ball mill
	Mill specific power	kWh/t	16.1
	Mill type		Overflow ball mill
	Mill pinion power	kW	1300
	Circulating load, design	%	300
	Cyclone overflow density	%w/w	47
	Product P_{80}	um	125
Gravity	Cyclone u/f split	%	27

Area	Description	Unit	Value
	Feed rate Centrifugal concentrator feed Concentrate mass Treatment type	t/h t/h kg/day	100 60 240 Intensive cyanidation
Leach / CIL	Leach tank qty. Leach tank volume, each	 m^3	1 375
	CIL tank qty. CIL tank volume, each	 m^3	6 375
Elution	Circuit type Carbon batch size Strips per week Heater size Regeneration kiln capacity	 t kW kg/h	Split AARL 2.5 6 1000 150
Electrowinning	# cells - CIL # cells - gravity Cell size - CIL Cell size - gravity	 mm mm	1 1 800 x 800, 9 cathodes 600 x 600, 9 cathodes
Tailings Disposal	Thickener Detox		N/A N/A

5 PROCESS PLANT DESCRIPTION

5.1 Crushing Circuit

The crushing circuit will reduce the ROM ore size from a nominal top size of 800 mm to a product size P_{80} of 20 mm in preparation for the grinding process.

The crushing and screening circuit is designed to process a throughput of 600,000 t/y during a dayshift only basis at 70% utilisation equating to a 200 t/h crushing rate. A level of conservatism is included to account for the lower crushing rates expected when processing higher proportions of oxide ore in the mill feed blend.

Process flow diagrams 600-300-PF-001 and 002 outlines the crushing circuit and should be referenced in conjunction with reading this section.

5.1.1 Process Description

Run of mine fresh ore (-800 mm) will be delivered by truck or wheel loader to the ROM pad and placed in 'fingers' to accommodate blending of the various ore types and gold grades prior to feeding into the process plant.

The ore will be fed by front-end loader from the ROM stockpile to a ROM bin equipped with a stationary grizzly. The grizzly aperture will be 800 mm in size to protect the jaw crusher from over size material. Over size material will be removed from the static grizzly by the ROM loader and then stockpiled before being broken by a mobile rock breaker.

Ore will be withdrawn at a controlled rate from the ROM bin by a variable speed vibrating grizzly feeder with an aperture of 75 mm. Oversize will fall into the primary jaw crusher, where it will be crushed. The expected closed side setting (CSS) of the jaw crusher is 85 mm. The proposed primary jaw crusher will have the capacity to process all ore in the event of a blinded vibrating grizzly.

Undersize from the vibrating grizzly and the primary crushed product will be conveyed using a short sacrificial to the sizing screen feed conveyor. Crushed ore from the secondary crusher will also be conveyed onto the sizing screen feed conveyor combining with the primary crushed product.

The sizing screen feed conveyor will discharge ore onto an inclined double deck vibrating product sizing screen, fitted with 75 mm aperture top deck and a 25 mm aperture bottom deck.

Screen oversize from both top and bottom decks will be conveyed to the secondary crusher feed bin. The material will be withdrawn at a controlled rate from the bin using a pan feeder to feed the secondary crusher. The secondary cone crusher circuit includes a surge bin, a standard configuration cone crusher and pan feeder along with walkways, access ladders, steel supports and other ancillary items.

A self-cleaning type tramp metal magnet will be installed at the transfer between the primary crusher product conveyor and sizing screen feed conveyor to remove any tramp metal.

Screen bottom deck undersize will be conveyed to the crushed ore stockpile.

5.2 Grinding Circuit

The crushed ore will be reclaimed at a control rate from the stockpile by two belt feeders each capable of maintaining design feed rate to the ball mill. Belt feeder discharge will be directed onto the fixed speed ball mill feed conveyor. An emergency feeder system complete with hopper and belt feeder may be used in periods of prolonged crusher maintenance and may also be used as the secondary means for adding grinding media to the ball mill.

The grinding circuit consists of a single stage ball mill operating in closed circuit with hydrocyclones and will reduce the crushed ore size to a final product size P_{80} of 125 µm suitable for gold recovery by gravity concentration and cyanide leaching. The grinding circuit consists of a ball mill, cyclone cluster and associated conveyors and pumps.

Process flow diagram 600-300-PF-003 outlines the grinding circuit and should be referenced in conjunction with reading this section.

5.2.1 Process Description

Ore reclaimed from the stockpile will be transported to the ball mill on the ball mill feed conveyor. The mill feed belt will be equipped with a weightometer to monitor and control the speed of the reclaim belt feeders and maintain mill feed rate.

A quicklime bin discharges lime onto the ball mill feed conveyor by a rotary valve. Lime addition is regulated to ensure a constant slurry pH in the CIL circuit.

In addition to the crushed ore feed and lime addition to the mill, cyclone underflow and gravity tailings are also recirculated back into the mill feed.

A single stage closed circuit ball mill has been selected for the grinding process. The ball mill has a diameter of 4.2 meters (m), and effective grinding length of 5.4 m fitted with a 1,300 kW motor. The availability is expected to be 91.3%. Process water will be added to the mill to maintain the mill discharge slurry density at 70-75% solids.

The mill discharge slurry will pass through the 10 mm aperture trommel into the mill discharge pump hopper. Trommel oversize will be collected in a skip for disposal or re-treatment. The mill discharge slurry will be pumped using duty and stand-by pumps to a cyclone cluster for classification.

The cyclone cluster will operate at a circulating load of 250-300% and consists of five operating 250 mm cyclones and two standbys. A twelve cyclone capacity cluster will be installed to allow for the future addition of more 250 mm cyclones in the 1.0 Mt/y expansion case.

The cyclone overflow, or final product grind size, will have a P_{80} of 125 µm. The cyclone underflow will be split in a distribution box with 50% of the stream being returned to the mill feed chute and the other 50% becoming the feed to the gravity concentration circuit.

Cyclone overflow will report to a vibrating trash screen to remove any misplaced oversize material and oversize trash including mining activity detritus captured with the mined ore. The screen oversize trash will be collected in a skip for periodic collection and disposal. Trash screen underflow will discharge into the leach tank.

5.3 Gravity Circuit

The gravity concentration circuit will produce a concentrate containing coarse gold from the grinding circuit that is intensively leached in a leach reactor to produce a gold rich elute for gold electrowinning.

Process flow diagram 600-300-PF-003 outlines the gravity circuit and should be referenced in conjunction with reading this section.

5.3.1 Process Description

The cyclone underflow stream will flow through a splitter box with 50% of the flow used as feed for the gravity concentration circuit. The cyclone underflow split stream, or gravity feed will initially be passed over a vibrating screen to remove oversize and grit particles greater than 2.4 mm in size. The screen oversize material will be returned to the grinding circuit via the gravity oversize and tailings collection box.

The screen undersize will be the feed to the centrifugal concentrator. The concentrator will operate continuously on a batch basis and will be flushed once an hour to remove the concentrate collected by the unit.

The gravity concentrate will be dewatered in a settling cone and once per day the concentrate will be transferred to an intensive leach reactor. The intensive leach reactor is a batch process, leaching the gravity concentrate to dissolve the contained gold into solution. This solution, pregnant liquor, is pumped to the electrowinning module for electrowinning in a dedicated cell. Leach reactor residue solids will be returned to the mill circuit.

The gold collected from the gravity circuit will be processed using a drying oven and smelting furnace to allow for separate metallurgical accounting of the gravity circuit. The final doré gold bars will be stored in the gold room safe.

5.4 Leach and CIL Circuit

Cyclone overflow slurry will be cyanide leached and gold adsorbed on to activated carbon. Loaded carbon will be recovered periodically to recover the gold using acid washing and hot solution elution, before being regenerated. Regenerated carbon will be returned to the last CIL tank.

Process flow diagram 600-300-PF-004 outlines the leaching and CIL circuit and should be referenced in conjunction with reading this section.

5.4.1 Process Description

Trash screen undersize slurry will be discharged into a dedicated leach tank. The leach and CIL circuit consist of a single agitated leach tank followed by six agitated CIL tanks all connected, in series, by launders with by-pass capability. The total combined retention time in the leach/CIL circuit is 24 hours.

To avoid short circuiting within the leach tank, feed slurry will enter opposite to the submerged outflow position. Slurry leaves the tank by an overflow launder.

Each CIL tank will be equipped with an interstage screen and a recessed impellor slurry pump. The interstage screen will allow the carbon to be retained in the respective CIL tank while permitting the pulp to flow through the screen to the next CIL tank in the circuit.

Barren (and regenerated) activated carbon is added to the last CIL tank. Carbon advances counter-current to the slurry flow. As the slurry flows downstream through the tanks it contacts carbon, solution becomes progressively lower in soluble gold as it moves down the circuit, enabling adsorption of gold to near-completion. Conversely, as the carbon advances upstream, it contacts slurry containing increasingly higher values of gold in solution enabling a higher loading of gold on the carbon.

Loaded carbon will be pumped via a recessed impeller pump from the first CIL tank to a loaded carbon recovery screen. The loaded carbon screen will be a vibrating screen equipped with spray water nozzles to thoroughly wash slurry off the loaded carbon. The loaded carbon will be discharged directly from the screen oversize into an acid wash column. The screen underflow will contain slurry and wash water and will return to the CIL circuit.

Slurry discharged from the last CIL tank will flow to the carbon safety screen. Any carbon which reports to the carbon safety screen will be collected in a bin and returned to the CIL circuit. The carbon safety screen undersize will be collected in a hopper and pumped directly to the tailing storage facility.

5.5 Elution

Loaded carbon from the CIL circuit is transferred to the elution circuit to start each elution cycle (once per day). The carbon is acid washed prior to desorbing the gold back into solution, electrowinning and smelting into gold doré.

Process flow diagram 600-300-PF-005 outlines the elution circuit and should be referenced in conjunction with reading this section.

5.5.1 Process Description

Loaded carbon from the screen oversize is directly transferred into the acid wash column where it is held until the beginning of an elution cycle, a batch process that occurs once per day, six or seven days a week.

The elution circuit has been designed to treat a 2.5 tonne batch of loaded carbon per day. Carbon stripping is achieved using a split Anglo American Research Laboratories (AARL) elution process that uses less water than a standard AARL elution process by recycling solution from the previous carbon elution cooling stage. An alternative ZADRA elution process, which uses less water again was considered, however with a likely future upgrade the AARL process is more easily upgraded and capitally less intensive than upgrading a ZADRA circuit for high carbon processing rates.

Separate acid wash and elution columns are proposed to minimise carbon batch treatment times and enable operation at higher elution temperatures.

The elution circuit is a fully automated semi-batch process controlled by the plant control system ensuring minimal operator involvement. The total elution cycle is expected to take approximately 16 hours, which includes carbon recovery, acid washing and elution.

In the first stage of the elution process, the carbon batch will be washed with 3% hydrochloric acid to remove inorganic contaminants being mainly carbonate precipitates from the carbon. Once this batch of carbon has been rinsed to remove residual acid, it is transferred to the elution column. Elution will include a pre-heat stage, a caustic/cyanide pre-soak stage, a hot solution elution followed by a cooling rinse stage, which elutes precious metals from the carbon to a gold rich electrolyte solution (pregnant eluate). A cool down stage will be included in the process to ensure the carbon temperature is reduced to a manageable level in the column prior to transfer. Elution solutions shall be heated by indirect means using a thermal oil heater and heat exchangers.

5.6 Carbon Regeneration

At the completion of the elution cycle, the batch of barren (gold depleted) carbon is hydraulically transferred back to the CIL circuit or to the feed hopper of the carbon reactivation kiln. The barren carbon will be reactivated in a horizontal rotary kiln. Reactivated carbon will then be transferred back to the last CIL stage.

Process flow diagram 600-300-PF-005 outlines the carbon regeneration circuit and should be referenced in conjunction with reading this section.

5.6.1 Process Description

A 2.5 tonne batch of carbon will be regenerated over a period of approximately 16.7 hours at a nominal rate of 150 kg/h. Thermal reactivation is used to remove organic foulants by subjecting the carbon to temperatures in the order of 650-750°C in a steam environment. The high temperature burns off some of the organic matter whilst steam strips the rest. Steam also serves to keep the reactivation system oxygen free (to prevent the carbon burning) and is involved in the chemical formation of active sites within the carbon.

Reactivated carbon discharging from the kiln is quenched in good quality water then batch transferred using a recessed impeller pump to a vibrating sizing screen located above the last CIL tank. Carbon fines produced within the reactivation circuit are rejected to the tailings stream. Fine carbon generated in the kiln poses a potential gold loss risk if returned to the CIL circuit.

The kiln will be design to treat up to 300 kg/h catering for an expected increase in stripping rate for the 1 Mt/y expansion option.

5.7 Gold Room

The high-grade gold elution solution leaving the elution column is cooled and then pumped to the gold room where the gold is recovered by a process of electroplating. In an electrowinning cell an electric current is passed through the pregnant solution forcing gold to plate out onto stainless steel cathodes. Once the cathodes are laden with gold they are cleaned using high pressure water to remove the gold, which is collected, filtered to remove excess moisture then dried and smelted to produce gold doré.

Process flow diagram 600-300-PF-006 outlines the electrowinning circuit and should be referenced in conjunction with reading this section.

5.7.1 Process Description

Gold rich electrolyte solution produced by the elution process will discharge to a pregnant eluate tank, which is located adjacent to the goldroom. A pump circulates the electrolyte through a single electrowinning cell for approximately 11 hours or until the gold tenor in the electrolyte is acceptably low. There is provision to add caustic soda solution to the pregnant eluate tank to maintain adequate electrolyte conductivity. The electrowinning cell will operate at a voltage between 3.0 to 5.0 VDC and a dedicated rectifier will be capable of supplying up to 1,000 A. A cell with 9 of 800 mm x 800 mm cathodes and 11 anodes is proposed for the electrowinning process.

Gold will electrodeposit as loosely adhering sludge on stainless steel woven wire mesh cathodes. The gold sludge will be manually removed from the mesh using high pressure water washing. This dislodged gold sludge will drain into a settling tank from where it is further dewatered using pressure filter/s. The filtered sludge is then dried in an oven to remove residual moisture prior to smelting.

The oven dried precious metal sludge is mixed with slagging compounds prior to smelting to produce combined gold and silver doré bars.

At the completion of the electrowinning cycle, barren eluate will be pumped back into the head of the CIL circuit.

The gold room shall be a steel framed building incorporating inner high tensile security mesh for additional security. Access to the building shall be via a series of security doors.

5.8 Reagents

Quicklime, caustic, hydrochloric acid and cyanide are the main reagents used within the process plant. Hydrochloric acid will delivered by bulk tanker and transferred to a storage tank. Cyanide will be delivered in liquid form (28% concentration w/v) by a bulk tanker and discharged directly into a cyanide storage tank. Similarly, caustic will be delivered in liquid form (75% w/v concentration). All reagents will be dosed to the circuit via dosing pumps.

Quicklime will be delivered as a solid and pneumatically discharged from the bulk tanker into the quicklime silo. Quicklime addition onto the ball mill feed conveyor belt will be controlled by a rotary valve.

Flux reagents for use in the smelting process will be delivered in bags that are approximately 25 kg and will be stored and handled directly in the gold room.

Carbon for use in the CIL circuit will be delivered regularly in one tonne bulk bags, while grinding balls used within the grinding mills will be delivered in 200 L drums.

Process flow diagram 600-300-PF-007 outline the main reagents used in the process.

5.9 Tailing

Slurry discharged from the last CIL tank that flows through the carbon safety screen will be collected in a hopper and pumped directly to the tailing storage facility.

Decant solution from the tailings storage facility is returned to the process water pond for reuse in the plant.

Process flow diagrams 600-300-PF-007 and 008 outlines the tailings storage facility and shows incoming and outgoing streams.

5.10 Water

A bore field will provide water to a raw water tank, located at the process plant, which in turn will feed the potable water treatment plant, gravity circuit and firewater system.

The lower portion of the raw water tank is dedicated to firefighting use.

A fire water ring main system is provided throughout the process plant. A smaller jockey pump will maintain ring main line pressure and provide water for limited lower demand usage as fire water. When demand is high and the line pressure drops below a set point, the electric firewater pump automatically starts to maintain line pressure. A standby main diesel engine fire water pump is available as back up in the event of a power outage.

The process water pond will receive raw and decant return water from the tailing storage facility. An allowance has been made to top up the process water pond with raw water if required during periods of low decant return. Process water is supplied to the plant from the process water pond by duty/standby centrifugal pumps on a header from which is piped throughout the plant.

The potable water plant will supply water to the elution circuit, offices and for ablutions on site. The potable water storage tank shall have sufficient capacity for the short periods of time that a higher flow is required for the elution circuit, filling up slowly between elution batch operations. The potable water plant shall be sized (17 m^3/h) to accommodate high saline water which may result in up to 40% of its feed discharged as effluent. Given the likelihood of high salinity raw water, potable water will be produced by a high recovery multi-stage reverse osmosis water treatment plant.

5.11 Air Services

A high-pressure air system comprising duty and stand-by compressors, air receivers and a drier will be used to supply high pressure and instrument quality air to sections of the process plant as required.

6 PROJECT IMPLEMENTATION

This section discusses the implementation steps for the Project.

6.1 Project Objectives

The objectives of the project are to:

- Deliver the project with zero safety, environmental and social incidents
- Maintain sound relationships with governments, local communities and other stakeholders
- Conform to statutory requirements regarding licenses and approvals
- Design, procure and construct the project within the approved budget and complete the works within approved milestones and schedule dates
- Maintain a tight cost control system with regular accurate updates
- Design the facility for safety and fitness-for-purpose
- Minimise, to the extent practicable, industrial disputes and deliver no adverse industrial legacies
- Complete plant commissioning to name-plant capacity as scheduled
- Meet or exceed identified project key performance indicators.

6.2 Project Phases

The project implementation phases are:

6.2.1 Studies

- Pre-feasibility study
- Definitive feasibility study
- Drilling and resource definition
- Mine planning and design
- Test work (various).

6.2.2 Approvals

- Funding
- Board approval
- Miscellaneous licences and agreements.

6.2.3 Plant and Infrastructure

- Engineering Design
- Procurement
- Site Works and Construction
- Commissioning and handover to operations
- Ramp up to full production and operation to deliver to specified parameters.

6.3 Project execution strategy

An Engineering, Procurement and Construction Management (EPCM) contract approach for the design and construction of processing plant has been adopted for the Study.

Other construction strategies require further assessment of their capacity to deliver cost, schedule or quality advantages in light of consideration of the distribution of project risk. Options include a greater degree of in-house management of the Project; various degrees of partnering with contractors and packaging the construction into discrete packages for a number of specialists to build.

6.4 Contracting Strategy

The Contracting Strategy Plan for the Project will be based on a number of Schedule of Rates contracts. This approach will ensure Matsa's involvement through all phases of the Project, minimise the construction interface issues and risks and takes into account the following:

- Risk allocation divided among a number of Contractors, not relying on a single contractor for high risk / complex elements of the work
- Contractor competencies by matching contractor skills with work scope and minimising the impact of having too many lower tier contractors
- Fast track philosophy, i.e. schedule to allow for progressive release of engineering, prioritised procurement and delivery of material and equipment
- Harmonise industrial relations and Occupational, Health, Safety, Environment and Community issues by use of contractor skills and resource that are well within proven capacity
- Off-site pre-fabrication of structural steel and plate work
- Identification of and free issue of specific long lead equipment and materials allowing contractor scope to be defined without any requirements for contractors having to manage long lead material supply.

6.5 Project schedule

An estimate project duration of 18 months is estimated from award of EPCM contract to operation of the new process facility.

The equipment that will have the greatest impact on the overall project schedule are listed in Table 6.1 . It is imperative that orders for the critical equipment are placed within the first 2 months to allow the flow of vendor data design information to the EPCM Contractor.

Table 6.1 Long Lead Items, Ex Works

Item	Weeks
Ball mill	46
Lime silo	28
Crushing plants	26
Elution heater package	24
Cyclone cluster	22
Slurry pumps	22
Gravity circuit equipment	20

The estimated project duration is based on the following milestones:

EPCM contract award	Week 0
Place order for nominated long lead equipment deliveries	Month 2
Place order for MCC/Switchroom	Month 4
Civil construction commence	Month 6
Engineering finalised	Month 8
SMP construction commence	Month 9
E&I construction commence	Month 10
Construction practical completion	Month 16
Commence Commissioning	Month 16
Commence Operations	Month 18

7 CAPITAL COST ESTIMATE

7.1 Summary

The capital cost estimate is summarised in Table 7.1. It is presented in Australian Dollars (AUD) and has a reference date of 4th quarter 2020.

The accuracy of the order of magnitude estimate is considered to be ±40%.

Table 7.1 Capital Cost Estimate

Item	Cost (AUD $m)
Direct cost	28.1
Indirect cost	5.6
Owners cost	1.7
Contingency	7.1
Project Total	**42.5**

7.2 Estimate Structure

The structure of the estimate has been divided into the following major categories:

- Equipment cost
- Project direct cost
- Indirect cost
- Owners cost
- Contingency.

7.2.1 Equipment

The mechanical equipment list was developed from the process flowsheet. The mechanical equipment list provides equipment numbers, equipment specification, type, model, size and electrical power draw.

Where time permitted budget quotes were obtained. The remainder of equipment was priced from recent projects and applicable escalation applied as required.

7.2.2 Direct Cost

Direct costs are those expenditures that include supply of the equipment and materials, freight to site and project site labour to construct plant and assembled equipment, temporary construction facilities, supporting facilities and services.

The project direct cost was determined as a factor of the total mechanical equipment supply cost.

7.2.3 Indirect Cost

Indirect costs are those expenditures covering engineering, procurement, construction management, commissioning, commissioning spares and first fills.

The indirect cost value was determined as a factor of the total project direct cost.

7.2.4 Owners Costs

Owner's costs typically cover the following items such as owner's team and associated expenses, insurances, foreign currency rate of exchange variation, Government duties, taxes, permit fees, licence fees, land cost, right of way, royalties, business readiness.

7.2.5 Contingency

Contingency is an allowance additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work. This is an allowance to cover possible costs that cannot be explicitly foreseen or described at the time the estimate is prepared due to lack of complete, accurate and detailed information.

7.3 Exclusions

The capital cost estimate presented excludes the following items.

- Goods and services tax (GST)
- Mine closure and rehabilitation costs
- Government duties, taxes, permit fees and the like
- Licence fees
- Land cost, right of way, royalties
- Finance and interest during construction
- Capital contributions to any statutory authorities
- Value Added and/or Goods and Services Taxes
- Legal costs
- Working and sustaining capital
- Foreign exchange rate exposure
- Sunk costs
- Force majeure issues
- Onsite power station
- Project infrastructure costs
- Bulk earthworks
- Tailings storage facility construction
- Permanent camp
- High security buildings
- Plant vehicles and mobile equipment
- Mining development cost.

7.4 Estimate Clarifications

The following is a list of estimate clarifications which are applicable to the capital cost estimate.

- The estimate is based on the process design criteria, flow sheets and mechanical equipment list developed during the study
- The process and site infrastructure estimate is based on "Stick Built" on site construction (no modular construction)
- All construction waste material can be disposed within 3 km of the project site
- Construction material for concrete can be sourced locally
- The ground geotechnical condition is sound and does not require major structural fill requirements and/or specialist piling.

8 OPERATING COST ESTIMATE

8.1 Summary

The operating cost estimate is summarised in Table 8.1. It is presented in Australian Dollars (AUD) and has a reference date of 4th quarter 2020.

The accuracy of the order of magnitude estimate is ±40%.

Table 8.1 Operating Cost Estimate

Category	Cost $/y	Cost $/t	Cost $/oz
Labour	6,292,850	10.5	125
Power	4,001,370	6.7	80
Reagents & Consumables	5,240,684	8.7	104
Maintenance	456,000	0.8	9
Analytical Services	677,891	1.1	14
Sub-total	**16,668,795**	**27.8**	**332**
Contingency – 20%	**3,333,759**	**5.6**	**66**
Total	**20,002,554**	**33.4**	**398**

8.2 Estimate Structure

The operating cost estimate was developed from a number of sources as summarised in Table 8.2.

Table 8.2 Derivation of Plant Operating Costs

Category	Cost
Labour	Developed manning requirements and salaries/rates
Power	Consumption from the equipment list and power cost of $0.20/kWh
Reagents	Consumption from test work or industry norms and pricing from historical data
Consumables	Estimated usage and pricing from suppliers
Maintenance	Calculated as a percentage of direct capital cost

8.3 Estimate Scope

8.3.1 Inclusions

The operating cost estimate includes:

- Labour for the onsite management and technical activities associated with the processing plant
- Labour for the operation and maintenance of the processing plant
- Costs associated with the direct operation of the processing plant, including reagents, consumables, maintenance materials and analytical services
- Cost of power supplied from an onsite generating plant.

8.3.2 Exclusions

The operating cost estimate excludes:

- Mining, exploration and geology costs
- Run of Mine (ROM) operation and feed into the process plant (by mining contractor)
- Royalties, taxes and duties on items such as fuel and imported equipment and value-added service taxes that may be applied
- Sustaining capital provisions
- Closure and rehabilitation costs
- Flights, accommodation and messing for all labour
- Mobile plant purchase
- Costs associated with maintenance of the tailings storage facility
- Corporate overhead charges
- Corporate functions including safety, environment, accounting, HR, payroll and management.
- Communications costs
- Insurance
- Refining, transport and other sales costs
- Licences/permits and land use fees, water access fees, or other such charges
- Taxes and duties
- Security
- Financing costs
- Mines rescue.

8.4 Labour

Manning levels for operational and maintenance requirements with the associated cost estimates are presented in Table 8.3.

Table 8.3 Labour Cost Summary, excl. 20% contingency

Position Description	Number Required	Total Annual Cost
OPERATIONS		
Management, Metallurgist, Supervisors	7	1,320,800
Plant Operators, Technicians	21	2,755,900
MAINTENANCE		
Mechanical	7	1,257,300
Electrical	6	958,850
TOTAL	**41**	**6,292,850**

8.5 Power

Power to the processing plant will be provided from an onsite diesel powered power station, with a unit cost of 0.20 $/kWh.

Power consumption is based on the load list developed from the mechanical equipment list, accounting for load and motor efficiency factors and equipment utilisations. The design feed rate and comminution parameters were used to calculate typical mill power consumptions.

Table 8.4 is a summary of power costs.

Table 8.4 Power Cost Summary

Plant Area	kWh's	Cost $/y	Cost $/t
Fixed annual fee		900,000	1.50
Crushing	1,898,624	379,725	0.63
Milling	10,159,469	2,031,894	3.39
Leach, CIL & Elution	1,656,804	331,361	0.55
Reagents & Services	1,147,705	229,541	0.38
Tailing & Bores	644,245	128,849	0.22
Process Plant Total	**15,506,847**	**4,001,370**	**6.67**

8.6 Reagents

Reagent consumptions for cyanide and lime are based on the limited test work results provided by MAT. Other consumption rates are based on historical experience from typical operations in the region.

Reagents costs are summarised in Table 8.5.

Table 8.5 Reagents Cost Summary

Consumable	Consumption		Supply Cost		Annual Cost ($)
	Rate	Unit	Rate	Unit	
Sodium cyanide	0.42	kg/t	3.200	$/kg	806,400
Quicklime	3.88	kg/t	0.269	$/kg	625,646
Hydrochloric acid	0.16	kg/t	0.577	$/kg	55,407
Sodium hydroxide	0.01	kg/t	0.606	$/kg	3,638
Activated carbon	0.001	kg/t	2.666	$/kg	1,600
Diesel	1.322	L/t	0.959	$/L	760,900
Silica	0.004		1.300	$/kg	3,120
Borax	0.0015		2.800	$/kg	1,170
				Total	**2,257,881**

8.7 Grinding Media and Consumables

Grinding Media and Consumables cost are summarised in Table 8.6.

Table 8.6 Grinding Media and Consumables Cost Summary

Plant Area	Cost $/y	Cost $/t
Grinding Media	1,125,803	1.88
Consumables	1,857,000	3.10

9 RISK AND OPPORTUNITIES

9.1 Project Risks

Process data is used for both design and financial assessment of the project. Uncertainty around process data and the potential variability of process inputs increases the risk of poor equipment selection and therefore poor plant performance.

This estimate is based on all available test work and information; however, this information is limited, and a number of assumptions have been made based on industry norms, best available information and CPC's experience.

There remains a level of uncertainty in fundamental data that has direct impact on the project financials. This represents a risk to the performance of the project.

Risks to CAPEX are limited as the design is fairly robust, common throughout the industry, and indicative parameters are in most cases adequate to make informed equipment selections.

Risks to OPEX are more significant due to the major impact of reagent consumption on costs, and limited testwork regarding reagent dosages required. In addition, a number of OPEX costs (e.g. diesel) are based on international goods that are historically price volatile and are impacted by foreign exchange rates.

This study has not undertaken any project financial assessment. Caution needs to be exercised in utilising the outputs of this study. The design is based on selected design parameters from limited test work and data.

9.2 Project Opportunities

OPEX reduction opportunities may be in the areas of power generation and reagent usage as discussed in Section 9.1.

Potential capital reductions may be in the following areas:

- A relatively standard comminution circuit comprising 2 stage crushing followed by single stage ball milling has been selected. The limited metallurgical test work to date indicates that the ore is very amenable to semi autogenous grinding (SAG), exhibiting values that suggest little resistance to impact breakage. An alternative single stage crushing followed by SAG milling and recycle pebble crushing would represent savings in both capital and operating costs.
- A split AARL elution circuit has been selected as the basis of the study. This type of stripping circuit is relatively easy to upgrade, should the plant throughout be increased from the baseline 600,000 t/y. The alternative Pressure Zadra stripping circuit is less capital intensive and requires less water than split AARL and offers a potential capital cost savings, albeit minimal. The Pressure Zadra circuit is a continuous stripping and electrowinning process. It is less favorable for throughput upgrading and not as flexible as the alternative split AARL process.
- A recessed impeller pump has been included for each CIL tank to advance carbon. Simple air lifts could be adopted in lieu of pumps for a minimal cost advantage.
- It is proposed that gravity concentrate recovered in the centrifugal concentrator is further processed in a dedicated intensive cyanide leach reactor and electrowinning cell. This circuit is capital intensive. The alternative is to defer the cost of the dedicated intensive cyanide leach reactor and electrowinning cell, instead initially installing a simple gravity shaking table to process the gravity concentrate. Shaking table concentrate is then directly smelted in the gold room furnace.

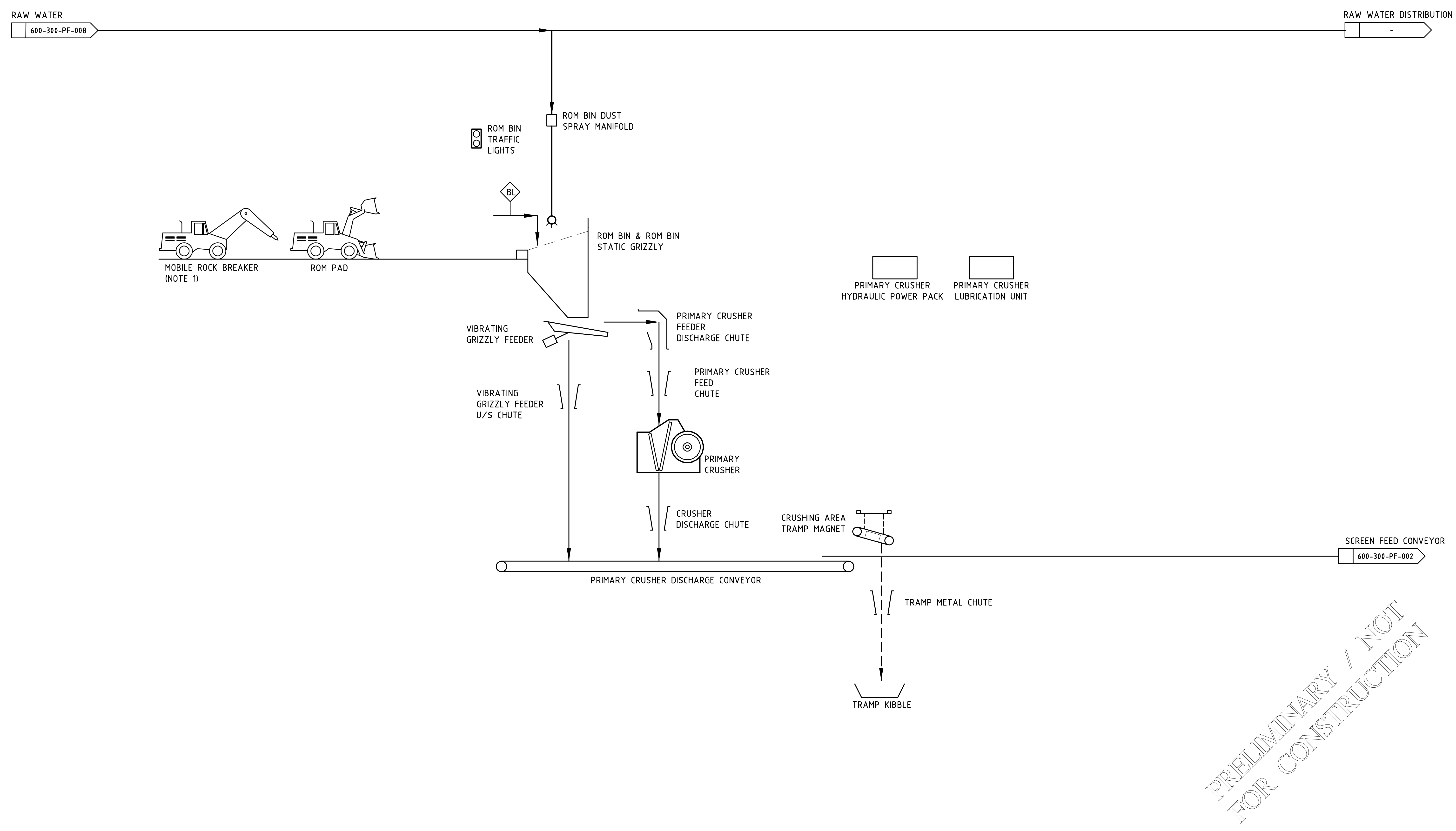
9.3 Upgrade to 1.0 Mt/y

The initial process plant design will enable relatively simple upgrading to a 1.0 Mt/y feed rate. The following is a summary of the likely upgrades required and the provisions to be made into the initial plant design to accommodate.

- 2-stage crushing circuit will be sized for 70% utilization for day shift operation only resulting in an instantaneous rate of 200 t/h. The circuit has the capacity for the upgraded plant throughout.
- The stockpile live capacity will be reduced to 9.6 hours for the expanded 1.0 Mt/y case, down from 16 hours initially.
- Stockpile reclaim feeders will need to be operated duty/duty. The emergency feeder is still available as back up.
- Provision will need to be made in the plant layout for a secondary ball mill or recycle pebble crushing, pending the outcome of further metallurgical test work and comminution modelling.
- The gravity gold circuit can be easily expanded with the addition of a second centrifugal concentrator and leach reactor.
- Provision will need to be made in the plant layout for additional leach, CIL tanks and/or pre-leach thickener.
- The split AARL circuit selected has the capacity to be operated more often and adding additional pregnant eluate tanks prior to electrowinning.
- The gold room layout will need provision for including additional electrowinning cells.

Appendix A
PROCESS FLOW DIAGRAMS

NOTES:
1. MOBILE ROCK BREAKER BY MINING CONTRACTOR



REFERENCES	DWG	REV	DATE	REVISION	DRN	CHK	DESIGN	DESIGN APP
		A	12/11/20	PRELIMINARY	JP			

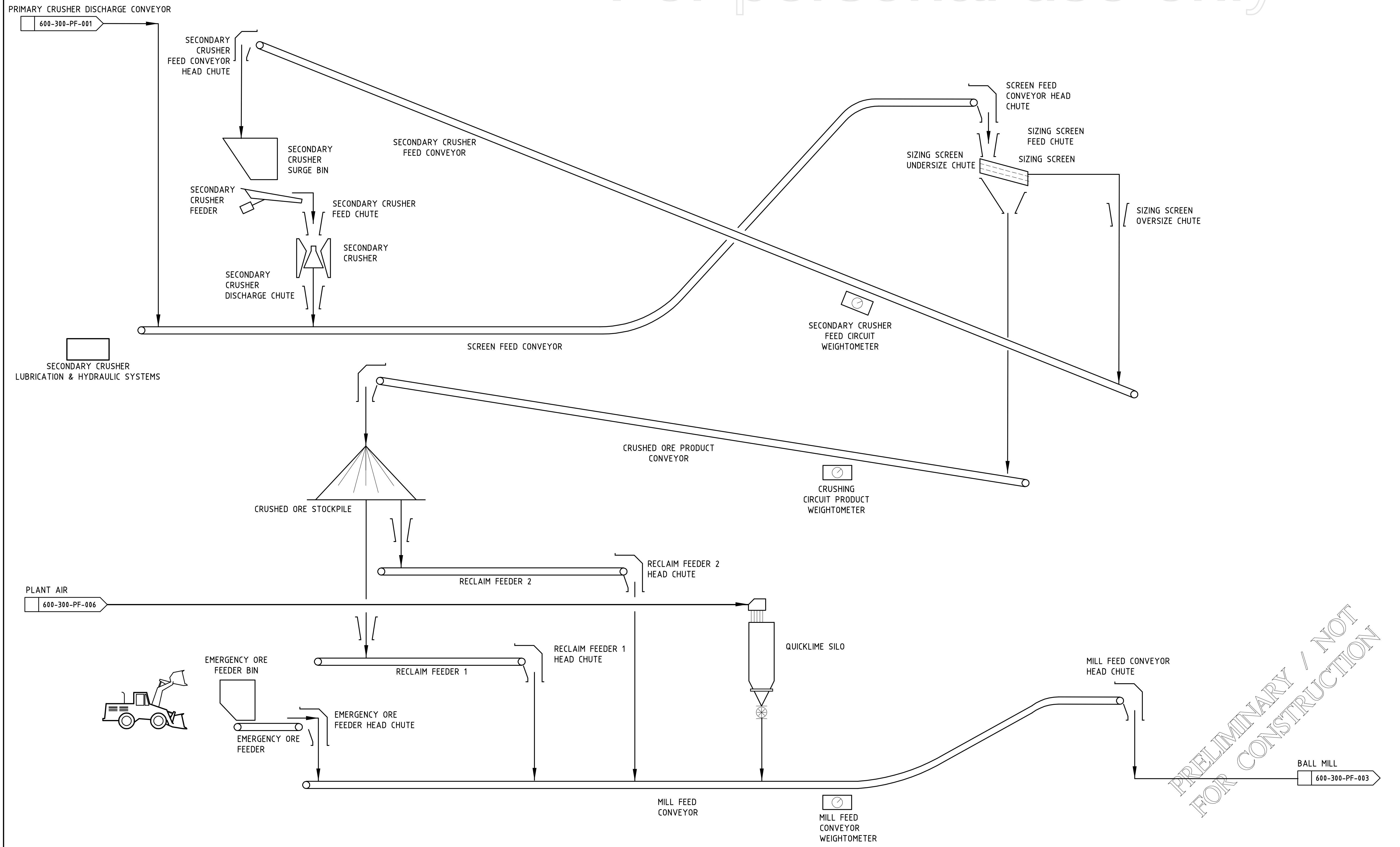
CPC PROJECT DESIGN
6 KINGS PARK RD.
WEST PERTH
WA 6005
+61 8 9365 0300
ACN 153 273 539
cpcprojects@cpceng.com.au
www.cpcengineering.com.au

DRAWN	JP	NOV 20
CHECKED		
DESIGNED		
DESIGN APP		
PROJ APP		
CLIENT APP		

MATSA RESOURCES LIMITED
LAKE CAREY GOLD CONCEPT STUDY
PRIMARY CRUSHING
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-001	A

P:\Projects\600 - Matsa Resources Lake Carey Operations\Drawings\PFD\600-300-PF-001.dwg - Thu Nov 12 2020 - 2:45pm jparry



REFERENCES	DWG	REV	DATE	REVISION	DRN	CHK	DESIGN	DESIGN APP
		A	12/11/20	PRELIMINARY	JP			

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+61 8 9365 0300
ACN 153 273 539
cpcprojects@cpceng.com.au
www.cpcengineering.com.au

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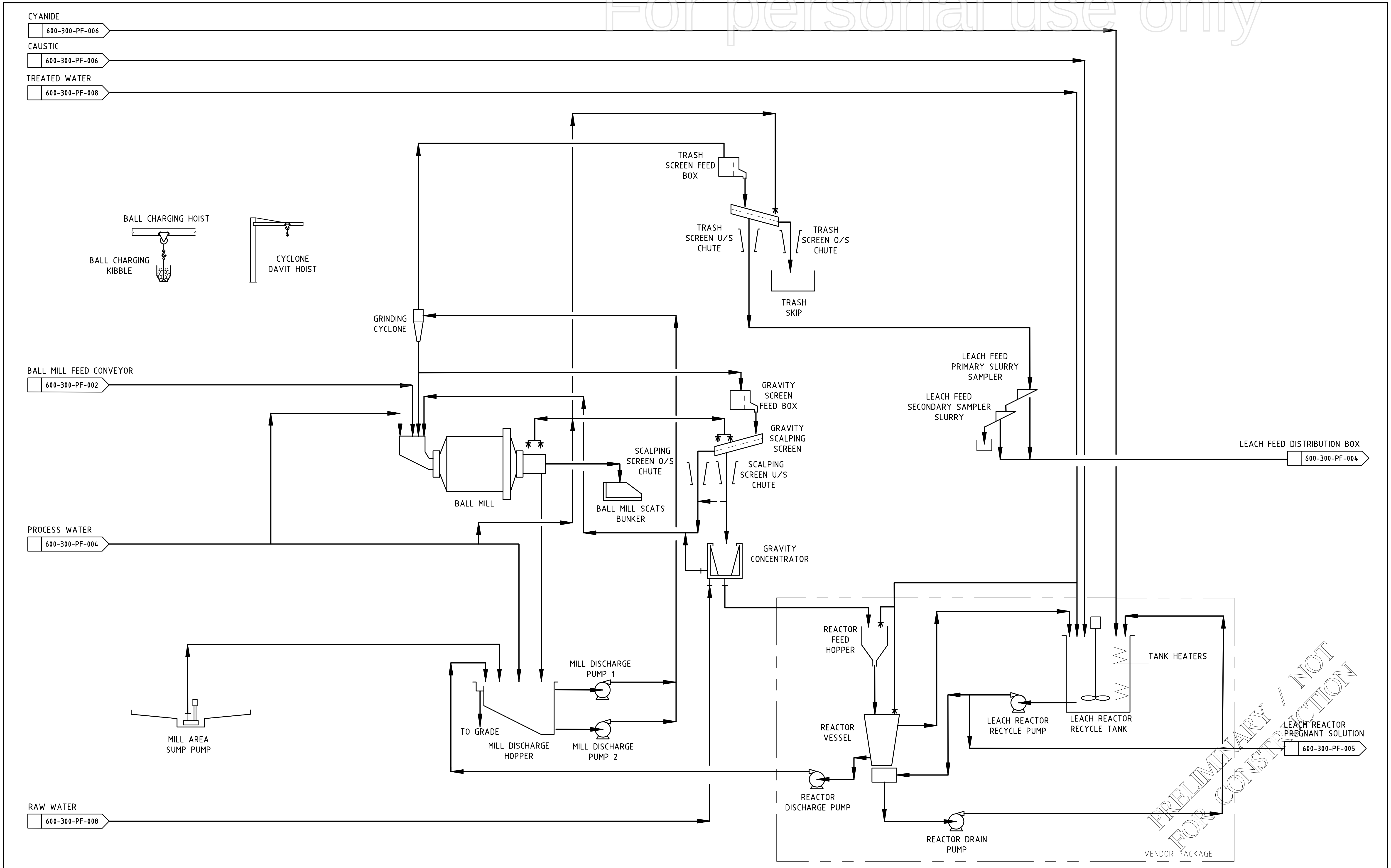
MATSA RESOURCES LIMITED
LAKE CAREY GOLD CONCEPT STUDY
SECONDARY CRUSHING
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-002	A

P:\Projects\600 - Matsa Resources Lake Carey Operations\Drawings\PFD\600-300-PF-002.dwg - Thu Nov 12 2020 - 2:47pm jparry



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MATSA RESOURCES LIMITED
LAKE CAREY GOLD CONCEPT STUDY
MILING & GRAVITY GOLD RECOVERY
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-003	A

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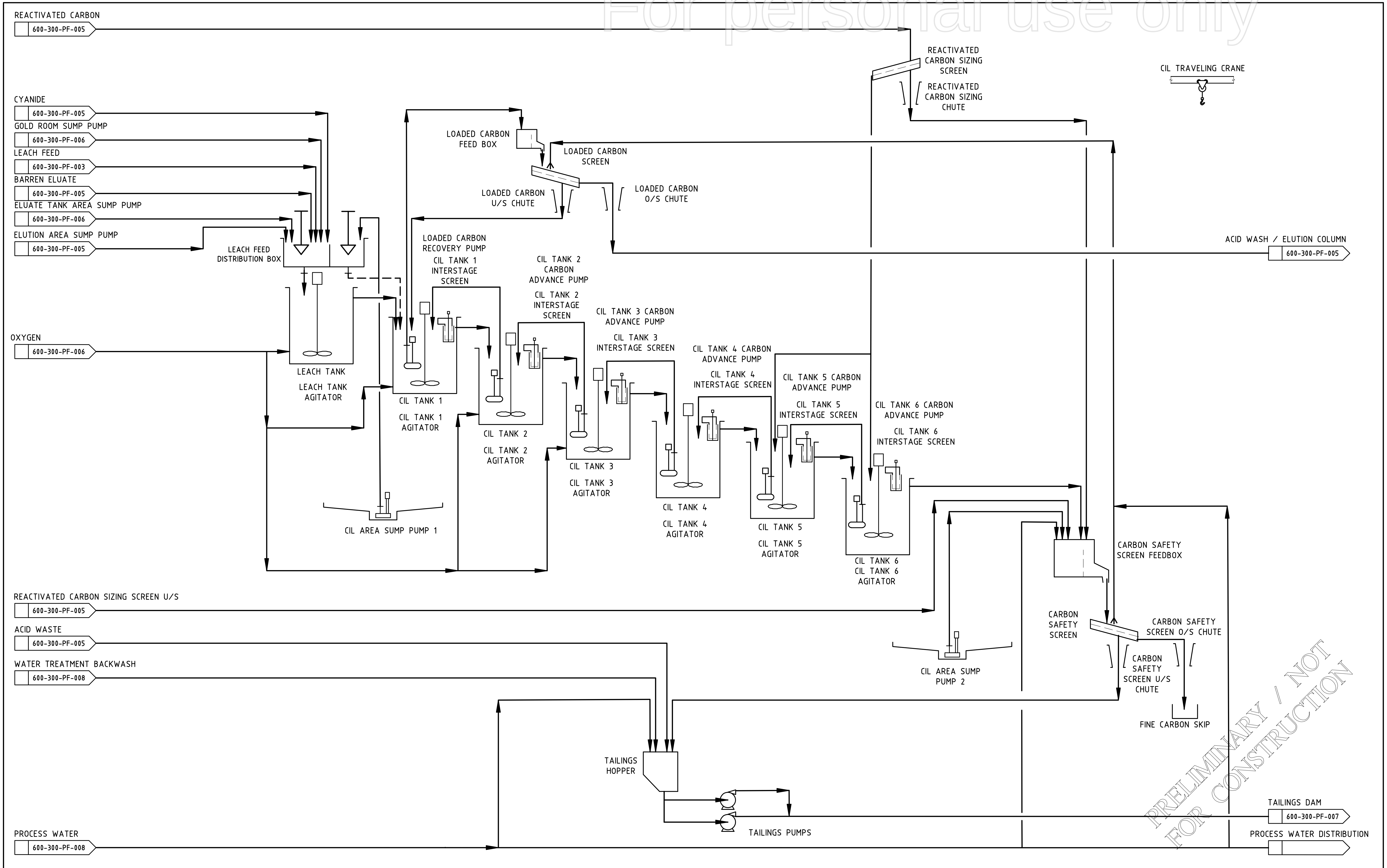

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LAKE CAREY GOLD CONCEPT STUDY
LEACH & CIL
PROCESS FLOW DIAGRAM

A1 | SCALE NTS | DWG No. 600-300-PF-004 | REV A

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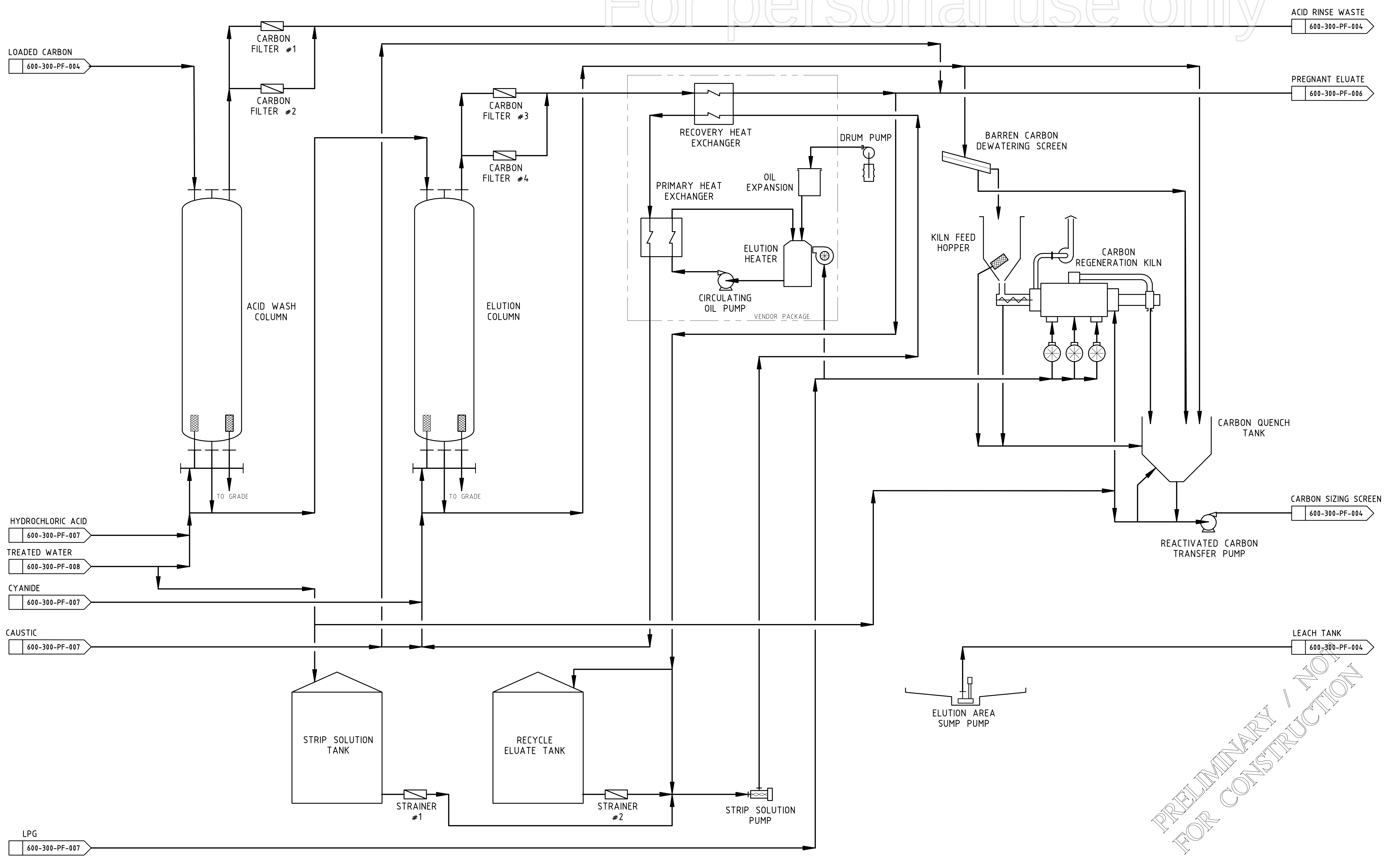

REFERENCES	DWG	REV	DATE	REVISION	DRN	CHK	DESIGN	DESIGN APP
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LAKE CAREY GOLD CONCEPT STUDY
ELUTION
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-005	A

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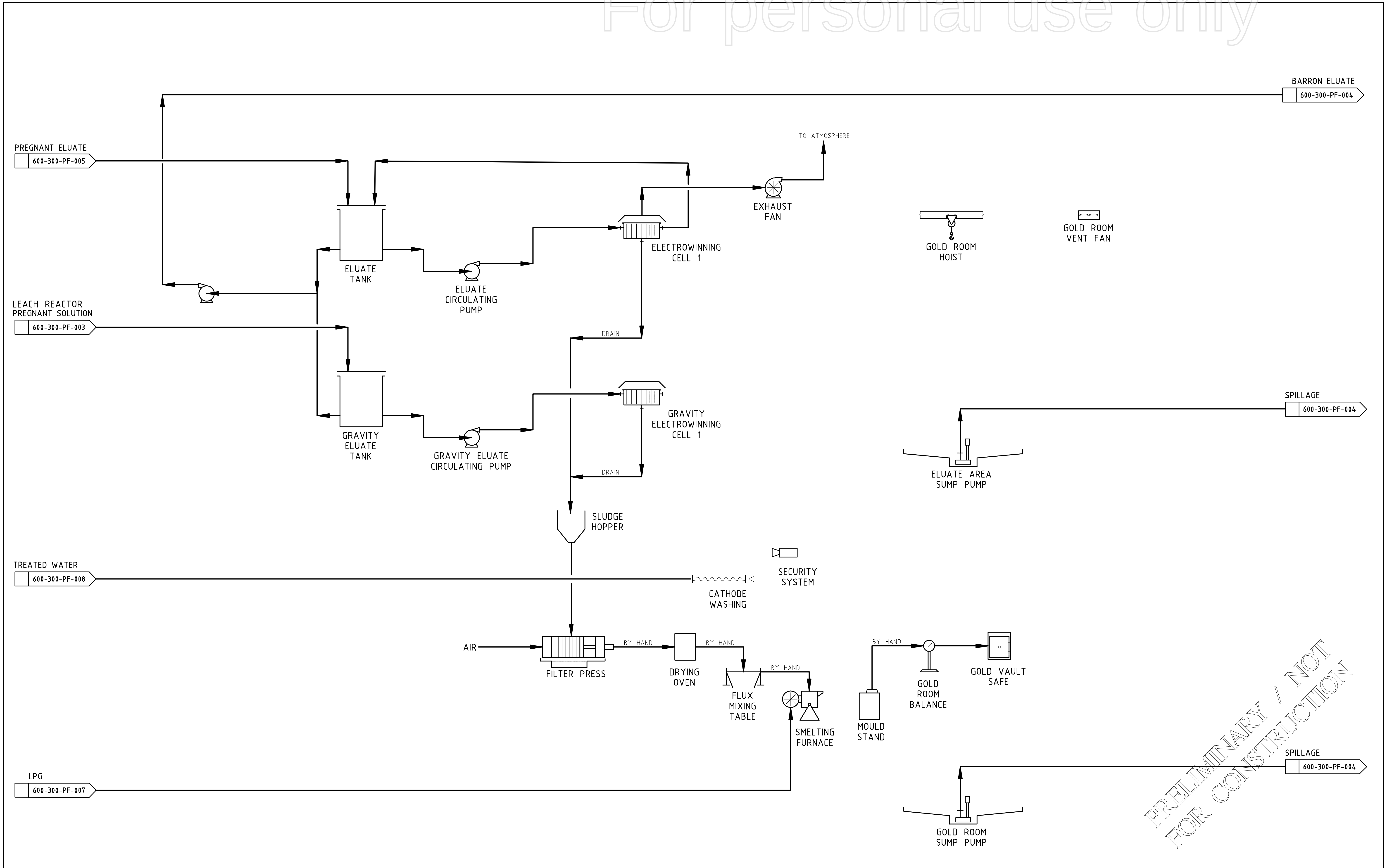

REFERENCES	DWG	REV	DATE	REVISION	DRN	CHK	DESIGN	DESIGN APP
		A	12/11/20	PRELIMINARY	JP			

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LAKE CAREY GOLD CONCEPT STUDY
ELECTROWINNING
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-006	A

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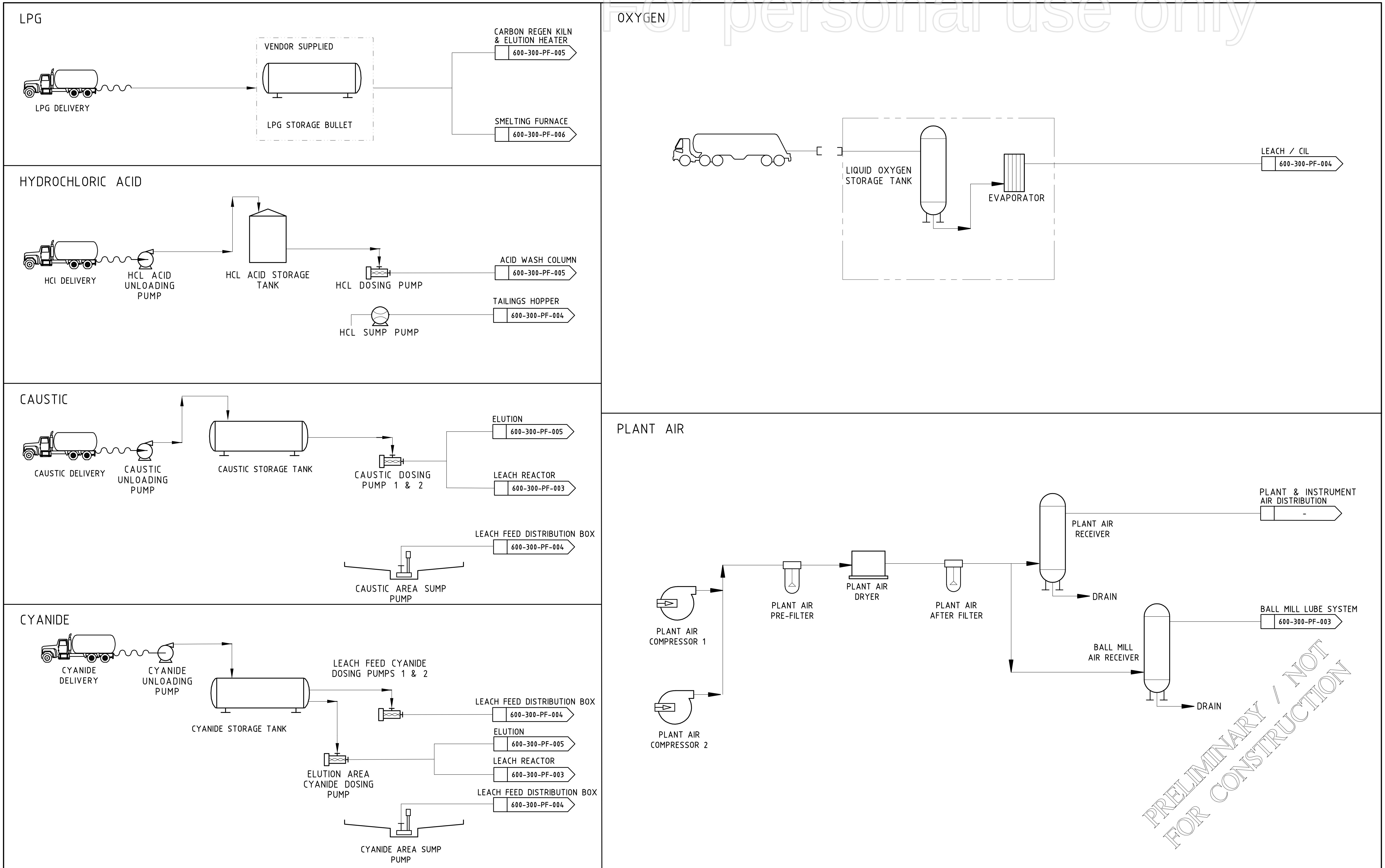

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MATSA RESOURCES LIMITED
LAKE CAREY GOLD CONCEPT STUDY
REAGENTS & SERVICES
PROCESS FLOW DIAGRAM

A1	SCALE NTS	DWG No. 600-300-PF-007	REV A

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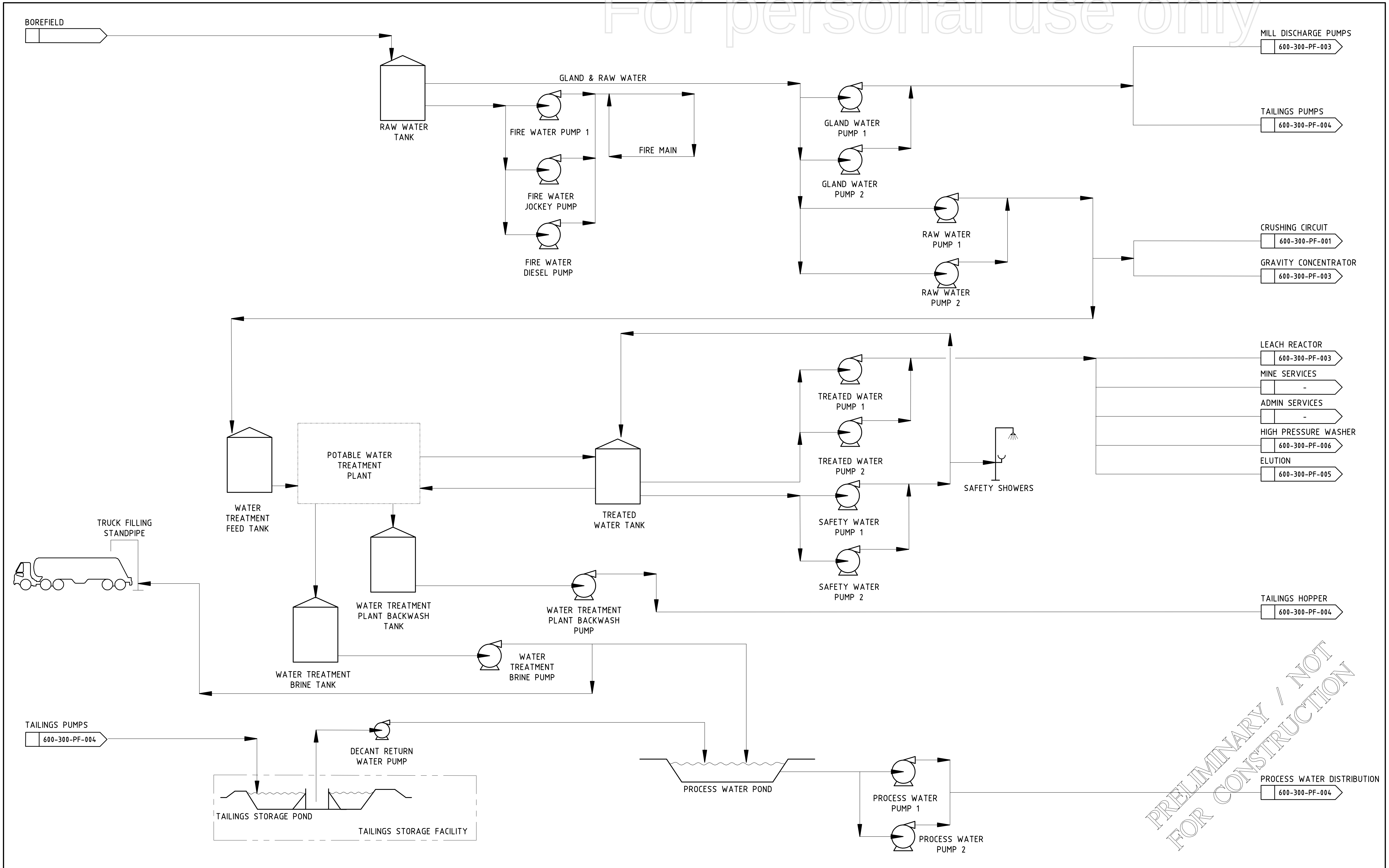

REFERENCES	DWG	REV	DATE	REVISION	DRN	CHK	DESIGN	DESIGN APP
		A	12/11/20	PRELIMINARY	JP			

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MATSA RESOURCES LIMITED
LAKE CAREY GOLD CONCEPT STUDY
WATER SERVICES
PROCESS FLOW DIAGRAM

	SCALE	DWG No.	REV
A1	NTS	600-300-PF-008	A

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Appendix B
PROCESS DESIGN CRITERIA

CPC PROJECT DESIGN

DESIGN CRITERIA

Client: Matsa Resources Limited

Project: Lake Carey Concept Study

Document Code: 60001-DC-R-001

Revision	Date	Revision Description	Signatures		
			Originator	Checked	Approved
A	13-Nov-20	Preliminary Issue	CPC	GN	

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
1	**OPERATING SCHEDULE**				
1.1	**Crushing**				
	Annual throughput	600,000	t/y	A	Client reference documents
	Operating days per year	365	d/y	A	Assumption
	Operating days per week	7	h/wk	A	Assumption
	Operating hours per day	24	h/d	A	Assumption
	Circuit operational availability	70	%	A	Assumption requiring verification
	Circuit operational hours	6,132	h/y	A	Calculated
	Average throughput rate	98	t/h	A	Calculated
	Instantaneous throughput rate	200	t/h	A	Engineering Database/Experience
1.2	**Grinding and Leach**				
	Annual throughput	600,000	t/y	A	Calculated
	Operating days per year	365	wk/y	A	Calculated
	Operating hours per week	7	h/wk	A	Calculated
	Operating hours per day	24	h/d	A	Engineering Database/Experience
	Circuit operational availability	91.3	%	A	Assumption requiring verification
	Circuit operational hours	7,998	h/y	A	Calculated
	Average throughput rate	75	t/h	A	Calculated
1.3	**Ore Grade**				
	Gold	2.75	g/t	A	Client reference documents - advised by client
	Silver	1	g/t	A	Assumption requiring verification - No Data
	Copper	1	g/t	A	Assumption requiring verification - No Data
1.4	**Gravity Recovery**				
	Gold	25	%	A	Client reference documents
	Silver	5	%	A	Assumption
	Copper	5	%	A	Assumption
1.5	**Leach Feed Grade**				
	Gold	2.06	g/t	A	Calculated
	Silver	0.95	g/t	A	Calculated
	Copper	0.95	g/t	A	Calculated
1.6	**Leach and Adsorption Circuit Gold Recovery**				
	Gold - Design (carbon considerations)	95	%	A	Assumption requiring verification
	Silver - Design	54.9	%	A	Assumption requiring verification
	Coper - Design	11.1	%	A	Assumption requiring verification
1.7	**Leach Recovery - Corrected**				
	Gold	94.3	%	A	Calculated
	Silver	45.8	%	A	Calculated
	Copper	0.4	%	A	Calculated
1.8	**Total Recovery**				
	Gold	95.7	%	A	Calculated
	Silver	48.5	%	A	Calculated
	Copper	5.4	%	A	Calculated
2	**ORE PHYSICAL CHARACTERISTICS**				
2.1	**Dry Solids Specific Gravity**				
	Range	2.7 - 3.05	t/m^3	A	Client reference documents - Oxide to primary
	Ore - design	3	t/m^3	A	Client reference documents - Primary for design
2.2	**ROM Ore Properties**				
	Ore Moisture - design	5	% w/w	A	Assumption requiring verification
2.3	**Crushed Ore**				
	Primary crushed ore	1.6	t/m^3	A	Assumption requiring verification
	Angle of repose - minimum	35	degrees	A	Assumption requiring verification
	Draw down angle	70	degrees	A	Assumption requiring verification
2.4	**Unconfined Compressive Strength**				
	Number of tests		No.	A	
	Average		Mpa	A	
	Maximum		Mpa	A	
	Failure Mode			A	
	Design		Mpa	A	
2.5	**Abrasion Index**				
	Design	0.054-0.173	g	A	Client reference documents
2.6	**Crushing Work Index (-76mm to 51mm)**				
	Number of tests		No.	A	
	Range		kWh/t	A	
	Average		kWh/t	A	
	Design		kWh/t	A	

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
2.7	**Drop Weight Test**				
	Parameter A			A	
	Parameter B			A	
	Impact breakage parameters, A x B			A	
	Abrasion breakage parameter, ta	0.08-0.54		A	Client reference documents - oxide low - primary high
	t10 @ Ecs = 1kWh/t			A	
	Design A x B	59.2		A	Client reference documents - primary ore
2.8	**Rod Mill Work Index**				
	Average	11.6	kWh/t	A	Client reference documents - oxide only
	Design - Based on	no primary data	kWh/t	A	Client reference documents
2.9	**Ball Mill Work Index**				
	Closing screen size	150	micron	A	Client reference documents
	Average	8.6-14.6	kWh/t	A	Client reference documents
	Design - Based on	14.6	kWh/t	A	Client reference documents - PRIMARY ORE 14.6
2.10	**Raw Water**				
	Ph	N/A		A	Assumption
	Total dissolved solids (TDS)	high 100,000	ppm	A	Assumption
	Density	1.03	t/m^3	A	Assumption
2.11	**Media Consumption**				
	Mill grinding media	1.5	kg/t	A	Calculated
3	**CRUSHING CIRCUIT**				
3.1	**Description**	2 Stage Crushing		A	Client reference documents
3.2	**Crusher Feed Bin**				
	ROM delivery method	Front End Loader		A	Industry standard
	Front end loader size	Cat 980 or equivalent		A	Engineering Database/Experience
	Crusher feed bin live capacity	50	wet t	A	Engineering Database/Experience
	Residence time when full	14.3	min	A	Calculated
	Oversize handling method	nil		A	Assumption
	Grizzly screen aperture	Fixed Grizzly on ROM Bin		A	Engineering Database/Experience
		700 x 700	mm x mm	A	Engineering Database/Experience
3.3	**Primary Crusher**				
	Feed	ROM Bin discharge via variable speed feeder		A	
	Grizzly	Vibrating Grizzly			
	Configuration	Single stage, open circuit		A	Engineer Database/Experience
	Type	Jaw Crusher - Single Toggle		A	Engineer Database/Experience
	Suggested Size	Metso C120 or equiv		A	Calculated - Bruno
	Crusher closed side settings (CSS)	85	mm	A	
	ROM ore F100	800	mm	A	
	Crushed product P100	150	mm	A	
3.4	**Secondary crusher**				
	Feed	Pre-screened product oversize		A	
	Configuration	Single stage, Closed circuit		A	Engineer Database/Experience
	Type	Cone Crusher		A	Engineer Database/Experience
	Suggested Size	HP300 or equiv		A	Calculated - Bruno
	Crusher closed side settings (CSS)	85	mm	A	
	ROM ore F100	800	mm	A	
	Crushed product P100	150	mm	A	
3.5	**Screening**				
	Feed	Jaw Crushed product		A	Engineering Database/Experience
	Configuration	Closing a single stage cone		A	Engineering Database/Experience
	Type	Inclined Double Decker		A	
	Suggested Size	1.7 x 4.9	m x m		
	Top deck aperture	75		A	
	Lower deck aperture	25		A	
	Suggested Size	HP300 or equiv		A	Calculated
	Crusher closed side settings (CSS)	22	mm	A	
3.6	**Dust Collection**				
	Feed	Provision		A	
3.7	**Stockpile**				
	Residence time - live	16	h	A	CPC Recommendation
	Capacity - live	1,200	t	A	CPC Recommendation

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
4	GRINDING CIRCUIT				
4.1	**Grinding Requirement**				
	New ore feed F80	20	mm	A	Client reference documents
	Circuit product P80	125	micron	A	Client reference documents
4.2	**Stockpile Reclaim**				
	Reclaim Feeder Configuration	Two reclaim belt feeders + one emergency belt feeder		A	Industry standard
	Reclaim Feeder Capacity	100	t/h/feeder	A	Assumption requiring verification
	Emergency Belt Feeder Capacity	100	t/h/feeder	A	Assumption requiring verification
4.3	**Mill Data**				
	Configuration	Single Stage Ball Mill		A	Client reference documents
	Diameter - inside shell	4.2	m	A	Client reference documents
	Diameter - inside shell	13.8	foot	A	Calculated
	Equivalent grinding length, EGL	5.4	m	A	Client reference documents
	Equivalent grinding length, EGL	17.7	ft	A	Calculated
	Length : Diameter ratio	1.3	L : D	A	Calculated
	Discharge arrangement	Overflow		A	Client reference documents
4.4	**Mill Operating Parameters**				
	Mill rotational speed - operating	15	%Cs	A	Client reference documents, Industry standard
	Ball charge - operating	32	% v/v	A	Industry standard
	Make-up ball size	90	mm	A	Industry standard
	Charge volume - operating	32	% v/v	A	Industry standard
	Mill discharge density - design	74	% w/w	A	Assumption requiring verification
	Mill discharge density - range	72-76	% w/w	A	Industry standard
4.5	**Mill Power Requirements - Primary Ore**				
	Pinion power - maximum	1,206	kW	A	Client reference documents
	Installed power	1,300	kW	A	Client reference documents
4.6	**Mill Discharge Screen**				
	Screen type	Trommel		A	Client reference documents, Industry standard
	Screen deck aperture	10	mm x mm	A	Client reference documents
	Suggested Trommel Diameter	1	m	A	Client reference documents
	Suggested Trommel Length	2	m	A	Client reference documents
4.7	**Classification**				
	Method	Hydrocyclones		A	Industry standard
	Recirculating load - nominal	240	%	A	Client reference documents
	Recirculating load- design	300	%	A	Client reference documents
	Cyclone feed tonnage total feed	300	t/h	A	Calculated
	Cyclone feed tonnage total feed	269	m^3/h slurry	A	Calculated
	Cyclone feed	64	% w/w solids	A	Assumption requiring verification
	Cyclone feed pressure	70	kPa	A	Assumption requiring verification
	No. of cyclone clusters	1	#	A	Industry standard
	Size cyclones	250	mm	A	Assumption requiring verification
	Distributor Outlets	10 to 12	No.	A	Assumption requiring verification
	Number of stand-by units - minimum	2	No.	A	Industry standard
	Number Cyclones Operating	5	No.	A	Assumption requiring verification
	Number Cyclones Installed	7		A	Assumption requiring verification
	Cyclone undeflow:				
	Solids rate	225	t/h	A	Calculated
	Solids Concentration	75	% w/w solids	A	Assumption requiring verification
	Slurry flow rate	150	m^3/h slurry	A	Calculated
	Cyclone overflow:				
	Solids rate	75	t/h	A	Calculated
	Solids Concentration	47	% w/w solids	A	Assumption requiring verification
	Slurry flow rate	110	m^3/h slurry	A	Calculated
4.8	**Mill Discharge Pump Hopper**				
	Input streams	Mill discharge, gravity tail, process water, sump pump discharge and ILR residue		A	
	Approximate discharge volume	255	m^3/h	A	Calculated
	Residence time	40	sec	A	Industry standard
	Live volume	3	m^3	A	Calculated

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PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
5	**GRAVITY GOLD RECOVERY CIRCUIT**				
5.1	**General**				
	Circuit configuration	Process portion of screened cyclone underflow stream		A	Industry standard
		Gravity concentrator tailings split between mill feed and mill discharge hopper		A	Engineer Database/Experience
	Proportion of feed stream treated	26.7	% of CUF	A	Calculated
	Equivalent proportion of new feed	80	% of feed	A	Calculated
5.2	**Gravity Screen**				
	Number of gravity screens	1	ea	A	Engineer Database/Experience
	Feed configuration	Split of cyclone underflow		A	Engineer Database/Experience
	Screen solids feed rate - design	100	t/h	A	Calculated
	Screen feed stream percent solids	63	% w/w	A	Calculated
		92.1	m3/h	A	Calculated
	Screen type	Vibrating, step deck		A	Industry standard
	Screen solid feed flux	40	t/h/m2	A	Industry standard
	Screen aperture	2.4 equivalent	mm	A	Industry standard
	Spray water unit flow	0.15	m3/t feed solids	A	Assumption
	Screen Size Estimate - Area Required	2.5	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 (vendor to advise)	m x m	A	
5.3	**Gravity Concentrators**				
	Type	Batch, Centrifugal		A	Assumption
	Number operating	1	ea	A	Engineer Database/Experience
	Model	KC-XD20 or equivalent		A	Assumption
	Maximum unit flow	60	t/h/unit	A	Vendor Specification
	Typical Fluidising water	8, G5 Cone	m3/h/unit	A	Vendor Specification
	Cycle time	1	h	A	Industry standard
	Concentrate production	10	kg/unit	A	Vendor Specification
	Concentrate production	0.24	t/d	A	Calculated
	Design concentrate production	0.75	t/d	A	Assumption requiring specification
5.4	**Concentrate Processing**				
	Method	Batch Intensive Leach		A	Assumption requiring specification
	Batch processing duration	24	h	A	Industry standard
	Reagent Consumptions			A	
	Sodium cyanide	25	kg/batch	A	Assumption requiring specification
	Sodium hydroxide - leach	3	kg/batch	A	Assumption requiring specification
	Sodium hydroxide - electrowinning	1	kg/batch	A	Assumption requiring specification
	Leachaid / leachwell / Oxidant	2	kg/batch	A	Assumption requiring specification
	Hydrogen Peroxide	0	kg/batch	A	Assumption requiring specification
	Flocculant	0	kg/batch	A	Assumption requiring specification
	Pregnant solution quantity per batch	2	m3	A	Vendor Specification
	Leach residue transfer destination	Mill discharge hopper			Industry Standard
5.5	**Intensive Leach Solution Electrowinning**				
	Daily gold recovery	1,130	g/d	A	Calculated
		4,709	g/t	A	Calculated
	Solution grade	565	ppm	A	Calculated
				A	
	Configuration	Sent to dedicated eluate tank for EW		A	Engineer Database/Experience
	Number of cells	1	No.	A	Calculated - EW Calculated
	Number of operating cells	1	No.	A	Calculated - EW Calculated
	Cell configuration	single cell recirc		A	Industry Standard
	Cell Size	600 x 600	mm x mm	A	Calculated - TBC by vendor - size for 50% grg ie 1mtpa
	Electrowinning Time	8	h	A	Industry Standard
	Cathode type	Stainless steel mesh		A	Industry Standard
	Grade of stainless steel wire	125/152		A	Industry Standard
	Number of cathodes per cell	9	No./cell	A	Calculated
	Operating cell volatge	5	V	A	Industry Standard
	Rectifier type	Rectiformer		A	-
	Rectifier size	800	Amp	A	Calculated - Conservative
	Rectifier voltage	10	V	A	Calculated

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PROCESS DESIGN CRITERIA					
No.	Description	Value	Unit	Rev	Source / Comment
6	**LEACH AND ADSORPTION**				
6.1	**Trash Screening**				
	Screen feed stream	Hydrocyclone overflow stream		A	Industry Standard
	Screen type	Linear vibratory		A	Industry Standard
	Screen panel aperture	0.63 x 12mm slots	mm	A	Industry Standard
	Screen specific flow	40	m3/h/m2	A	Industry Standard - conservatism for oxides
	Screen solids feed rate	75	t/h/m²	A	Calculated
	Screen feed stream percent solids	47	% w/w	A	Calculated
		109.6	m3/h/m2	A	Calculated
	Spray water unit flow	0.15	m3/t feed solids	A	Assumption
	Screen Size Estimate - Area Required	2.7	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 vendor to advise	m x m	A	Assumption requiring verification - to be confirmed by vendor
6.2	**Leach Feed Sampler**				
	Configuration	2 stage located on trash screen feed		A	Industry Standard
6.3	**Leach Feed Stream Data**				
	Leach circuit feed stream				
	Design annual throughput	600,000	t/y	A	Calculated
	Leach circuit solids feed rate - design	75	t/y	A	Calculated
	Leach feed stream density - design	47	% w/w	A	Assumption requiring verification
	Leach feed stream solids dry density	3	t/m3	A	Calculated
	Volumetric flow	109.6	m3/h/m2	A	Calculated
	Slurry SG	1.5	t/m3	A	Calculated
	Gold Feed grade - design	2.1	g/t	A	Calculated
6.4	**Gold Recovery Data**				
	Leach circuit gold extraction - design	95	%	A	Calculated
	Gold resuide grade - design	0.1	g/t	A	Calculated
	Tailings solution gold grade	0.013	ppm	A	Industry Standard
	Correct residue grade (includes solution loss)	0.118	g/t	A	Calculated
	Adsorption Efficiency	99.3	%	A	Calculated
	Recovery - design	94.3	%	A	Calculated
	Recovery - design	1.9	g/t	A	Calculated
6.5	**Gold Production Data**				
	Leach Recovered - design	146	g/h	A	Calculated
		3,501	g/d	A	Calculated
	Leach Recovered - design	113	foz/d	A	Calculated
		37,506	foz/y	A	Calculated
	Overall Recovered - design	194	g/h	A	Calculated
		4,667	g/d	A	Calculated
		150	foz/d	A	Calculated
		50,008	foz/y	A	Calculated
6.6	**LEACH CIRCUIT**				
	Configuration	Hybrid CIL		A	Client reference documents
6.7	**Leach Circuit**				
	Tank Style	Leach tank only		A	Assumption requiring verification
	Number of tanks	1	No.	A	Client reference documents
	Effective total leach residence time	3.4	hr	A	Calculated
	Effectiveleach volume	375	m3	A	Calculated
	Leach feed Ph	10.5		A	Industry Standard
	Leach feed NaCN concentration	500	ppm	A	Assumption requiring verification
	Slurry dissolved oxygen concentration	>5	ppm	A	Industry Standard
	Oxygen / air addition method	Downshaft		A	Industry Standard
	Air flow to tank - design	70	Nm3/h	A	Assumption requiring verification
6.8	**CIL Circuit**				
	Configuration	Series tanks with tank by-basses		A	Industry Standard
	Number of tanks	6	No.	A	Client reference documents
	Effective total CIL residence time	20.5	hr	A	Calculated
	Total Leach/Adsorption slurry residence time at design density	23.9	hr	A	Client reference documents
	Effective total CIL volume	2250	m3	A	Calculated
	Effective CIL tank volume	375	m3	A	Calculated
	Residence time at 42% w/w	20.4	hr	A	Calculated
	CIL feed pH	10.5		A	Client reference documents
	CIL feed NaCN concentration	500	ppm	A	Client Reccomendation
	Slurry dissolved oxygen concentration - Tank 2 - 6	>5	ppm	A	Industry Standard
	Oxygen / air addition method	LOX injected via sparges		A	Industry Standard
	Required oxygen gas flow to Tanks 2 to 6 - design	tbc	Nm3/h	A	Assumption requiring verification - No data

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
6.9	**Carbon Circuit**				
	Carbon concentration - Tank 2	10	g/L	A	Calculated
	Carbon concentration - Tank 3 to 6	10	g/L	A	Calculated
	Design carbon concentration all Leach/CIL Tanks	20	g/L	A	Industry Standard
	Carbon mass in Tank 2	3.8	t	A	Calculated
	Carbon mass in each of Tank 2 to 6	3.8	t	A	Calculated
	Total carbon in adsorption circuit	22.5	t	A	Calculated
	GOLD - design carbon loading	1,650	g/t	A	Client reference documents, Assumption requiring verification
	Average carbon movement	91.2	kg/h	A	Calculated
	Average carbon movement	2.2	t/d	A	Calculated
	Carbon adsorption circuit residence time	10.3	d	A	Calculated
	Carbon transfer period	9	h/d	A	Industry Standard
	Intertank carbon and pulp transfer rate	24.3	m3/h	A	Calculated
	Intertank carbon transfer method	Recessed impeller carbon lift pump		A	Industry Standard
6.10	**Loaded Carbon Recovery Screen**				
	Screen feed stream	Slurry from Tank 2 or Tank 3		A	
	Screen type	Horizontal vibratory, with wash troughs or steps in deck		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Loaded carbon recovery time	4	h	A	Industry Standard
	Screen feed flow - design	75	m3/h	A	Calculated
	Screen Size Estimate - Area Required	1.5	m2	A	Calculated
	Screen Size Estimate - LxW	vendor to advise	m x m	A	Assumption requiring verification
	Carbon recovery method	Recessed inpeller carbon lift pump		A	Industry Standard
6.11	**Intertank Screens**				
	Screen type	Cylindrical, mechanically swept		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen cloth construction	Stainless stell wedge wire		A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Effective screen area	2.7	m2	A	Calculated
	Diameter x Height	vendor to advise	m x m	A	Assumption requiring verification
6.12	**Barren Carbon Screen**				
	Screen feed stream	Quench tank discharge stream		A	
	Screen undersize stream destination	Carbon safety screen		A	
	Screen oversize stream destination	Tank 6 or Tank 7		A	
	Screen type	Horizontal vibratory		A	
	Screen panel aperture	0.8	mm	A	Assumption requiring verification
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Screen Size Estimate - LxW	vendor to advise	m x m	A	Vendor specification
6.13	**Carbon Safety Screen**				
	Screen feed stream	Tank 6 or Tank 7		A	
		Carbon transfer water		A	
		Sump pump slurry		A	
	Screen type	Horizontal vibratory		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Screen Size Estimate - Area Required	2.3	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 vendor to advise	m x m	A	Assumption requiring verification
7	**ELUTION, ACID WASH AND ELECTROWINNING**				
7.1	**General Data**				
	Operating schedule	7	d/wk	A	Assumption
	Circuit capacity	2.5	t	A	Client reference
	Average strip frequency	291.6	strips/y	A	Calculated
		5.6	strips/wk	A	Calculated
	Peak strip frequency	6.1	strips/wk	A	Calculated
	Number of elution columns	1	No.	A	Engineer Database/Experience
	Vessel for acid washing	Column		A	Industry Standard
	Dry carbon bulk density	0.47	t/m3	A	Industry Standard
	Carbon bed volume	5.3	m3	A	Calculated
	Loaded carbon gold grade	1650	g/t	A	Calculated
	Barren carbon GOLD grade	50	g/t	A	Engineer Database/Experience

PROCESS DESIGN CRITERIA					
No.	**Description**	**Value**	**Unit**	**Rev**	**Source / Comment**
7.2	**Acid Washing**				
	General				
	Acid wash column carbon capacity	2.5	t	A	Calculated
	Acid wash column fabrication	Rubber Lined Carbon Steel		A	Calculated
	Acid type	Hydrochloric Acid, HCl		A	Industry Standard
	Supplied acid concentration	38.4	% w/w	A	Assumption requiring verification
	Acid solution SG - as supplied	1.16		A	Calculated
	Acid Wash				
	Acid wash cycle time	75	min	A	Engineer Database/Experience
	Acid wash solution acid concentration	3	% w/v	A	Industry Standard
	Neat acid volume required	0.3	m3	A	Calculated
	Neat acid pumping time	15	min	A	Industry Standard
	Neat acid pump flow	1.1	m3/h	A	Calculated
	Water flow	2	BV/h	A	Industry Standard
		10.6	m3/h	A	Calculated
	Temperature	ambient	degrees C	A	Industry Standard
	Acid Water Rinse				
	Acid rinse cycle time	120	min	A	
	Water flow	10.6	m3/h	A	Industry Standard
	Number of bed volumes	4	BV/rinse	A	Industry Standard
	Temperature	ambient	degrees C	A	Industry Standard
	Water Source	Strip Solution tank / Strip Soln Pump		A	
	Location of Output	Tails hopper		A	
	Carbon Transfer				
	Process	Carbon transfer from acid wash column to elution column			
	Carbon transfer time	60	min	A	Calculated
	Carbon transfer concentration	235	g/L		Industry Standard
	Transfer flowrate	10.6	m3/h		Calculated
	Education water required	10.6	m3		Calculated
	Transfer water type	Treated water from strip solution tank (via strip solution pump)			
7.3	**Elution**				
	General				
	Elution method	SPLIT - AARL		A	Engineer Database/Experience
	Elution column construction	Stainless Steel 316		A	Industry Standard
	Elution column operating pressure	300	kPag	A	Engineer Database/Experience
	Dosed NaOH concentration	76	% w/v	A	Engineer Database/Experience - Equivalent to 30% w/w
	Dosed NaOH solution SG	1.52		A	Calculated
	Dosed NaCN concentration	28	% w/v	A	Industry Standard, Calculated
	Pre-treatment				
	Description	The column is primed by supplying water from the strip solution tank		A	
		Following priming, the circuit is heated by circulating solution to column and back to strip solution pump suction		A	
		Sodium cyanide and sodium hydroxide solutions are dosed into the circuit during the heating stage.		A	
	Cycle time	30	min	A	Engineer Database/Experience
	Solution flow	2	BV/h	A	Industry Standard
		10.6	m3/h	A	Calculated
	Temperature operating range	ambient - 125	degrees C	A	Indsurty Standard
	Pre-soak - number of bed volumes	0.8	BV	A	Engineer Database/Experience
	Pre-soak - solution volume - From Starter Tank	4.3	m3	A	Calculated
	Column solution discharge destination	Eluate tank		A	
	Pre-soak solution NaOH concentration	3	% w/v	A	Indsurty Standard
	Pre-soak solution NaNC concentration	3	% w/v	A	Indsurty Standard
	Pre-treatmeant NaOH concentration	127.7	kg/pretreat	A	Calculated
		0.2	m3/pretreat	A	Calculated
	Pre-treatment NaCN consumption	127.7	kg/pretreat	A	Calculated
		0.5	m3/pretreat	A	Calculated
	Elution1 - from starter tank				
	Cycle time	60	min	A	Indsurty Standard
	Elution solution flowrate	2	BV/h	A	Indsurty Standard
		10.6	m3/h	A	Calculated
	Solution volume	10.6	m3	A	Calculated
	Temperature	125	degrees C	A	Indsurty Standard
	Column solution discharge destination	Eluate tank		A	

No.	Description	Value	Unit	Rev	Source / Comment
	Elution2- water (a) - From strip solution tank				
	Cycle time	67	min	A	Engineer Database/Experience
	Elution solution flowrate	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	11.9	m3	A	Calculated
	Temperature	125	degrees C	A	Indusrty Standard
	Column solution discharge dstination	Eluate tank		A	Indusrty Standard
	Elution3- water (b) - from strip solution tank				
	Cycle time	60	min	A	Engineer Database/Experience
	Elution solution flowrate	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	10.6	m3/h	A	Calculated
	Temperature	125	degrees C	A	Indusrty Standard
	Column solution discharge dstination	Starter eluate tank		A	Indusrty Standard
	Cooling				
	Cycle time	30	min	A	Engineer Database/Experience
	Elution solution flowraye	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	5.3	m3	A	Calculated
	Temeprature	Ambient	degrees C	A	Indusrty Standard
	Column solution discharge destination	Starter eluate tank		A	Indusrty Standard
	Total Elution				
	Total bed volumes	8	BV/elution	A	Indusrty Standard
	Total elution water requirement	42.7	m3/elution	A	Calculated
	Total elution time	247	min	A	Calculated
	Electrolyte NaOH addition	59.6	kg	A	Calculated
	Volume of NaOH solution to electrolyte tank	0.1	m3	A	Calculated
	Carbon Transfer				
	Process	Carbon transfer from elution column to reactivation kiln hopper			
	Carbon transfer time	60	min	A	Calculated
	Carbon transfer concentration	235	g/L	A	Indusrty Standard
	Transfer flowrate	10.6	m3/h	A	Calculated
	Eduction water required	10.6	m3	A	Calculated
	Transfer water type	Treated water from strip solution tank		A	Indusrty Standard
	Total Stripping Circuit Water Consumption				
	Acid wash	4.3	m3/Strip	A	Calculated
	Water rinse	21.3	m3/Strip	A	Calculated
	Carbon transfer	10.6	m3/Strip	A	Calculated
	Column prime	4.3	m3/Strip	A	Calculated
	Water elution a	11.9	m3/Strip	A	Calculated
	Water elution b	10.6	m3/Strip	A	Calculated
	Cooling	5.3	m3/Strip	A	Calculated
	Carbon transfer	10.6	m3/Strip	A	Calculated
	Total	78.9	m3/Strip	A	Calculated
	Total RO Water Required per strip / day	62.8	m3/Strip	A	Calculated
7.4	**Elution Equipment**				
	Eluate Tank	5	BV sizing	A	Engineering Database/Experience - conservative
	Number of eluate tanks	1	No.	A	Client recommendation - 2 for 1 Mt/a
	Eluate volume per tank	26.6	m3	A	Calculated - note working vol
	Solution Tanks				
	Starter Elute Tanks	26.6	m3	A	Calculated - note working vol
	Strip Solution Tank	26.6	m3	A	Calculated - note working vol
	Elution Heater				
	Configuration	1 x Recovery Heat Exchanger1 x Primary Heat Exchanger		A	
	Elution heater type	LPG	-	A	Assumption requiring verification
	Elution heater size	1000	kW	A	Calculated - not sized for 1Mt/a

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
7.5	**Electrowinning**				
	Number of cells	1	No.	A	Calculated
	Number of operating cells	1	No.	A	Calculated
	Cell configuration	-		A	Industry standard
	Cell size	800 x 800	mm x mm	A	Calculated
	Effective cell cross-sectional area	0.64	m2	A	Calculated
	Electrwinning cell linear velocity	300	L/min/m2	A	Industry standard
	Electrowinning cell solution feed flow	11.5	m3/h/cell	A	Calculated
		11.5	m3/h	A	Calculated
	Electrowinning time	10	h	A	Industry standard
	Cathode type	Stainless steel mesh		A	Industry standard
	Grade of stainless steel wire	125/152		A	Industry standard
	Number of cathodes per cell	9	No./cell	A	Calculated
	Operating cell voltage	5	V	A	Industry standard
	Current efficiency - Au	8	%	A	Industry standard
	Current efficiency - Ag	13	%	A	Industry standard
	Current per cell	789	Amp/cell	A	Calculated
	Current per cathode	87.7	Amp/cathode	A	Calculated
	Current per cathode area	10.3	Amp/m2 cathode	A	Calculated
	Rectifier type	Rectiformer		A	-
	Rectifier size	1000	Amp	A	Calculated
	Rectifier voltage	10	V	A	Calculated
	Barren solution gold concentration	2	ppm	A	Industry standard
	Barren solution silver concentration	5	ppm	A	Industry standard
8	**CARBON REACTIVATION**				
8.1	**Reactivation Kiln**				
	Type of kiln	Horizontal, diesel fired		A	Indusrty Standard
	Dewatering screen type	Sieve Bend		A	Indusrty Standard
	Carbon reactivation capacity	150	kg/h	A	Calculated
	Carbon reactivation capacity - design	250	kg/h	A	Calculated - Client to Advise - larger for 1mtpa
	Operating temperature - design	720	degrees C	A	Indusrty Standard
	Operating temperature - range	650 - 800	degrees C	A	Indusrty Standard
	Carbon processing time	16.7	h/batch	A	Indusrty Standard
	Number of feed hoppers	1	No.	A	Engineering Database/Experience
	Feed hopper capacity	3	t	A	Engineering Database/Experience
8.2	**Carbon Quench Tank**				
	Required holding capacity	3	t	A	Engineering Database/Experience
	Carbon concentration	400	g/L	A	Indusrty Standard
	Tank type	Closed top with conical bottom		A	Indusrty Standard
	Effective quench tank volume	7.5	m3	A	Calculated
	Carbon transfer method	Recessed impeller pump		A	Indusrty Standard
	Carbon transfer time	1.5	h	A	
	Carbon transfer concentration	150	g/L	A	Indusrty Standard
	Carbon transfer rate	13.3	m3/h	A	Calculated
	Make-up water type	Raw water		A	Indusrty Standard
8.3	**Carbon Sizing Screen**				
	Screen feed stream	Quench tank discharge stream			
	Screen undersize stream destination	Carbon safety screen			
	Screen oversize stream destination	Tank 6 or Tank 5			
	Screen type	Horizontal vibratory			
	Screen panel	0.8			
	Screen specific flow	50			
9	**GOLDROOM**				
9.1	**Cathode Treatment**				
	Method	High pressure washing of sludge from cathodes		A	Industry standard
	Sludge filtration method	Decant and pressure filtration		A	Industry standard
	Drying method	Drying oven		A	Industry standard
9.2	**Smelting**				
	Type of furnace	LPG fired, tilting		A	Industry Standard
	Number of furnaces	1		A	Engineer Database/Experience
	Crucible size	A100		A	Assumption requiring verification - 2 smelts per week at 1 Mt/a; increase to A200 for less smelts but should be ok for 1 Mt/a
	Mould arrangement	Cascade		A	Industry Standard

CPC PROJECT DESIGN

PROCESS DESIGN CRITERIA					
No.	**Description**	**Value**	**Unit**	**Rev**	**Source / Comment**
10	**WASTE DISPOSAL**				
10.1	**Tailings Thickener**	Tickening not inc		A	
10.2	**CIL Discharge Pump Hopper**				
	Input streams	Mill discharge, gravity tail, process water, sump pump discharge and ILR residue		A	
	Discharge density - design	42	% w/w	A	Assumption requiring verification
	Discharge - typical	45	% w/w	A	Assumption requiring verification
	Approximate discharge volume	128.6	m3/h	A	Calculated
	Residence time	90	sec	A	Industry standard
	Live volume	3.2	m3	A	Calculated
10.3	**Tailings Facility**				
	Tailings Settled Density	70	% w/w solids	A	Assumption requiring verification
	Water reclaimed from Tailings Dam	75	%	A	Assumption requiring verification - rest to evaporation
		24.1	m3/h	A	Calculated
	Tailings decant return pump operational time	8	h/d	A	Assumption requiring verification
	Tailings decant return pump - design flow	144.7	m3/h	A	Calculated
11	**REAGENTS**				
11.1	**Quicklime**				
	Consumption	4	kg/t	A	Client reference documents - high saline water - similar project
	Design consumption	10	kg/t	A	Assumption requiring verification
		18005	kg/d	A	Calculated
	Addition point	Mill feed chute		A	Assumption
	Solids SG	2300		A	
	Number of storage tanks	1	#	A	
	Hopper capacity	240	h	A	Industry standard
	Hopper Volume	78.3	m3	A	Calculated
	Nominal lime storage capacity	180	t	A	Calculated - 2 x 100 tonne silos. Consider 1 larger then leave spares for a second fot 1mtpa
11.2	**Sodium Cyanide**				
	CIL usage	240	t/y	A	Calculated
	CIL consumption, design	0.4	kg/t	A	Client Reccomendation
	Elution consumption	0.72	t/week	A	Calculated
	Elution usage	37	t/y	A	Calculated
	Elution consumption	0.06	kg/t	A	Calculated
	ILR Usage	8.3	t/y	A	Calculated
	ILR Consumption	0.01	kg/t	A	Calculated
	Total usage	277	t/y	A	Calculated
	Total consumption, design	0.46	kg/t	A	Calculated
	Hourly consumption	35	kg/h	A	Calculated
	Daily usage	832	kg/d	A	Calculated
	Total Consumption				
	Mixture strength	28	%w/v	A	Industry Standard
	Hourly consumption at concentration supplied % w/w	124	L/h	A	Calculated
	Total daily volumentric flow	3	m3/d	A	Calculated
	Solution Storage				
	Storage live volume required	40	m3	A	Engineer Database/Experience
	Storage residence	13.5	d	A	Calculated
	Reagent Dosing				
	Dosing to CIL, method	Ring main		A	Industry Standard
	Dosing to CIL, design	124	L/h	A	Calculated
	Ring main flow	0.5	m3/h	A	Calculated
	Dosing to elution, method	Dedicated pump		A	Industry Standard
	Time to dose to elution	15	min	A	
	To elution strip at 3% soak, 20% w/v stock solution	0.5	m3	A	Calculated
	Dosing pump capacity	1.82	m3/h	A	Calculated

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
11.3	**Sodium Hydroxide**				
	Elution consumption	1.1	t/week	A	Calculated
	Elution usage	55	t/y	A	Calculated
	Elution consumption	0.09	kg/t	A	Calculated
	ILR Usage	1.33	t/y	A	Calculated
	ILR Consumption	0.002	kg/t	A	Calculated
	Total usage	56	t/y	A	Calculated
	Total consumption, design	0.09	kg/t	A	Calculated
	Hourly consumption	6.99	kg/h	A	Calculated
	Daily usage	168	kg/d	A	Calculated
	Solution Storage				
	Consumption per day	0.221	m3/d	A	Calculated
	Mixture strength	76	% w/v	A	Industry Standard
	Storage live volume required	16	m3/d	A	Calculated
	Storage residence	72	d	A	Assumption requiring verification
	Reagent Dosing				
	Dosing to elution, methos	Dedicated pump		A	Industry Standard - can be directed to ILR when required
	Time to dose	16	min	A	
	Quantity to dose per strip at 3% soak, 39% w/v stock solution	0.17	m3	A	Calculated
	Dosing pump capacity	0.63	m3/h	A	Calculated
11.4	**Hydrochloric Acid**				
	Neat acid wash consumption	1.9	t/wk		Calculated
	Neat acid wash usage	97	t/y		Calculated
	Neat acid consumption	0.16	kg/t		Calculated
	Hourly consumption	12.1	kg/h		Calculated
	Daily usage	291	kg/d		Calculated
	Storage Tank				
	Hourly consumption	10	L/h	A	Calculated
	Total daily volumetric flow	0.25	m3/d	A	Calculated
	Storage live volume required	3.5	m3	A	Engineer Database/Experience
	Storage capacity	14	d	A	Calculated
	Dosing to acid method	Dedicated pump		A	Calculated
	Time to dose	10	min	A	Calculated
	Quantity to dose per strip at 3% soak, 33% w/w stock solution	0.29	m3	A	Calculated
	Dosing pump capacity	1.15	m3/h	A	Calculated
11.5	**Grinding Media**				
	Ball Mill	1.5	kg/t	A	Calculated
	Consumption	2.7	t/d	A	Calculated
	Usage	900	t/y	A	Calculated
11.6	**Carbon**				
	Consumption	0.5	% loss/strip	A	Assumption requiring verification-Typically ~1% for conventional, halved for extruded
		6	g/t	A	Calculated
	Usage	0.01	t/d	A	Calculated
11.7	**Antiscalant**				
	Type				
	Consumption	15	g/t	A	Assumption
	Uage	9	t/y	A	Calculated
11.8	**Sulphamic Acid**				
	Consumption	0.5	kg/Strip	A	Assumption requiring verification
	Usage	146	kg/y	A	Assumption
11.9	**Goldroom Fluxes**				
	Borax flux				
	Consumption	0.05	kg/kg PMs	A	Industry Standard
	Usage	71	kg/y	A	Calculated
	Consumption	0.12	g/t	A	Calculated
	Soda Ash flux				
	Consumption	0.04	kg/kg PMs	A	Industry Standard
	Usage	57	kg/y	A	Calculated
	Consumption	0.1	g/t	A	Calculated
	Potassium Nitrate flux				
	Consumption	0.01	kg/kg PMs	A	Industry Standard
	Usage	14.3	kg/y	A	Calculated
	Consumption	0.024	g/t	A	Calculated
	Silica Flour flux				
	Consumption	0.005	kg/kg PMs	A	Industry Standard
	Usage	7.1	kg/y	A	Calculated
	Consumption	0.01	g/t	A	Calculated

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
12	**WATER SERVICES**				
12.1	**Raw Water**				
	Supply source	To be defined		A	
	Receiving storage type, plant site	Raw water tank		A	
	Minimum storage capacity, plant site w/ fire water allowance	528	m3	A	
	Storage capacity	12	h	A	
	Raw Water Demand				
	Raw water to RO treatment	5	m3/h	A	
	Raw water to crushing	2	m3/h	A	
	Raw water to gravity concentrator	9.1	m3/h	A	
	Raw water to CIL	0.4	m3/h	A	
	Raw water to tailings area	0	m3/h	A	
	Elution & ILR	0	m3/h	A	
	Raw water to reagent mixing	0.8	m3/h	A	
	Miscellaneous	10	m3/h	A	
	Total	27	m3/h	A	Calculated
		653	m3/h	A	Calculated
	Raw water pump flow - design	54	m3/h	A	Engineer Database/Experience
	Raw water outlet pressure	250	kPa	A	Industry Standard
12.2	**Fire Water**				
	Supply source	Allocated portion of Raw Water		A	
	Circuit configuration	Electric jockey pump to maintain main pressure		A	
		Dedicated electric driven main pump with low pressure activation		A	
		Back up diesel driven main pump with low pressure activation		A	
	Firewater reserve allowance	144	m3	A	2 hr storage
	Firemain inlet flow basis	2 attack hydrants operating at 10L/s each		A	
	Firemain inlet flow	72	m3/h	A	
12.3	**RO Water**				
	Supply source	Raw Water, waste to PW dam		A	
	Circuit configuration	Use of raw water to supply essential RW		A	
	Storage capacity	50	m3	A	
	Outflow	12	m3/h	A	
	Capacity from full	4.2	h	A	
12.4	**Elution Water**				
	Supply source	RO Storage		A	Industry Standard
	Elution water requirement - nominal	4.5	m3/h	A	Assumption
	Elution water (strip solution tank) fill rate requirement (ro Water pump sizing)	36	m3/h	A	Assumption - 3hr fill
12.5	**Gland Seal Water**				
	Supply source	Raw water tank		A	Industry Standard
	Gland seal water pump discharge	5	m3/h	A	Assumption
	Gland seal water outlet pressure	1000	kPa	A	Engineer Database/Experience
12.6	**Potable Water**				
	Configuration	RO water treated to UV		A	
	Supply source	Filtered and treated raw water		A	Assumption requiring verification
	Receiving storage type	Plastic lines- squatters type		A	Assumption requiring verification
	Storage capacity	50	m3	A	Assumption requiring verification
	Potable Water Demand				
	Consumption	200	L/man/d	A	Industry Standard
	Personnel on site	300	No.	A	Assumption requiring verification
	Water usage	60	m3/d	A	Calculated
	Potable water pump flow - design	10	m3/h	A	Engineering Database/Experience
12.7	**Safety Shower Water**				
	Supply source	RO storage tank		A	Assumption requiring verification
	Supply method	Circulating ring main		A	Industry Standard
	Ringmain inlet flow	20	m3/h	A	Industry Standard
	Minimum available safety volume	10	m3	A	Engineering Database/Experience

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
12.8	**Process Water**				
	Supply source	decant water and water from settlement ponds to make-up raw water		A	
	Receiving storage type	Pond		A	Assumption requiring verification
	Storage capacity	1,800	m3	A	Calculated
	Storage residence time	24	h	A	Calculated
	Evaporation	6	mm/m2/day	A	Assumption
	Secondary process water sources	Surplus mine dewatering; site stormwater collection ponds		A	Assumption requiring verification
	Mine dewatering excess	0	m3/h	A	Assumption requiring verification - set to neg
	Site water runoff	0	m3/h	A	Assumption
	Site area	0	m2	A	Assumption - set to neg tbc
	Average rainfall	0	mm/y	A	Assumption - set to neg
	Proportion of rainwater collected	0	%	A	Engineering Database/Experience
	Process Water Demand				
	Grinding	71	m3/h	A	Calculated - from mass balance
	Miscellaneous	4	m3/h	A	Calculated - from mass balance
	Total	75	m3/h	A	Calculated
		1,800	m3/d	A	Calculated
	Process water pump flow - design	150	m3/h	A	Engineering Database/Experience
	Process water outlet pressure	150	kPs	A	Engineering Database/Experience - outlet means point of service not d/c
13	**AIR SERVICES**				
13.1	**Configuration**	All HP air will be instrument quality		A	Industry Standard
13.2	**High Pressure Air**				
	Compressor type	Rotary screw		A	Engineering Database/Experience
	Capacity	700	m3/h FAD	A	Assumption
	Discharge pressure	800	kPa	A	Industry Standard
	Number of units	1 duty and stand-by	ea	A	Industry Standard
	Air Receiver capacity	3,000	L	A	

Appendix C
MECHANICAL EQUIPMENT LIST

DESIGN CRITERIA

Client:	Matsa Resources Limited
Project:	Lake Carey Concept Study
Document Code:	60001-DC-R-001

Revision	Date	Revision Description	Signatures		
			Originator	Checked	Approved
A	13-Nov-20	Preliminary Issue	CPC	GN	

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
1	**OPERATING SCHEDULE**				
1.1	**Crushing**				
	Annual throughput	600,000	t/y	A	Client reference documents
	Operating days per year	365	d/y	A	Assumption
	Operating days per week	7	h/wk	A	Assumption
	Operating hours per day	24	h/d	A	Assumption
	Circuit operational availability	70	%	A	Assumption requiring verification
	Circuit operational hours	6,132	h/y	A	Calculated
	Average throughput rate	98	t/h	A	Calculated
	Instantaneous throughput rate	200	t/h	A	Engineering Database/Experience
1.2	**Grinding and Leach**				
	Annual throughput	600,000	t/y	A	Calculated
	Operating days per year	365	wk/y	A	Calculated
	Operating hours per week	7	h/wk	A	Calculated
	Operating hours per day	24	h/d	A	Engineering Database/Experience
	Circuit operational availability	91.3	%	A	Assumption requiring verification
	Circuit operational hours	7,998	h/y	A	Calculated
	Average throughput rate	75	t/h	A	Calculated
1.3	**Ore Grade**				
	Gold	2.75	g/t	A	Client reference documents - advised by client
	Silver	1	g/t	A	Assumption requiring verification - No Data
	Copper	1	g/t	A	Assumption requiring verification - No Data
1.4	**Gravity Recovery**				
	Gold	25	%	A	Client reference documents
	Silver	5	%	A	Assumption
	Copper	5	%	A	Assumption
1.5	**Leach Feed Grade**				
	Gold	2.06	g/t	A	Calculated
	Silver	0.95	g/t	A	Calculated
	Copper	0.95	g/t	A	Calculated
1.6	**Leach and Adsorption Circuit Gold Recovery**				
	Gold - Design (carbon considerations)	95	%	A	Assumption requiring verification
	Silver - Design	54.9	%	A	Assumption requiring verification
	Coper - Design	11.1	%	A	Assumption requiring verification
1.7	**Leach Recovery - Corrected**				
	Gold	94.3	%	A	Calculated
	Silver	45.8	%	A	Calculated
	Copper	0.4	%	A	Calculated
1.8	**Total Recovery**				
	Gold	95.7	%	A	Calculated
	Silver	48.5	%	A	Calculated
	Copper	5.4	%	A	Calculated
2	**ORE PHYSICAL CHARACTERISTICS**				
2.1	**Dry Solids Specific Gravity**				
	Range	2.7 - 3.05	t/m^3	A	Client reference documents - Oxide to primary
	Ore - design	3	t/m^3	A	Client reference documents - Primary for design
2.2	**ROM Ore Properties**				
	Ore Moisture - design	5	% w/w	A	Assumption requiring verification
2.3	**Crushed Ore**				
	Primary crushed ore	1.6	t/m^3	A	Assumption requiring verification
	Angle of repose - minimum	35	degrees	A	Assumption requiring verification
	Draw down angle	70	degrees	A	Assumption requiring verification
2.4	**Unconfined Compressive Strength**				
	Number of tests		No.	A	
	Average		Mpa	A	
	Maximum		Mpa	A	
	Failure Mode			A	
	Design		Mpa	A	
2.5	**Abrasion Index**				
	Design	0.054-0.173	g	A	Client reference documents
2.6	**Crushing Work Index (-76mm to 51mm)**				
	Number of tests		No.	A	
	Range		kWh/t	A	
	Average		kWh/t	A	
	Design		kWh/t	A	

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
2.7	**Drop Weight Test**				
	Parameter A			A	
	Parameter B			A	
	Impact breakage parameters, A x B			A	
	Abrasion breakage parameter, ta	0.08-0.54		A	Client reference documents - oxide low - primary high
	t10 @ Ecs = 1kWh/t			A	
	Design A x B	59.2		A	Client reference documents - primary ore
2.8	**Rod Mill Work Index**				
	Average	11.6	kWh/t	A	Client reference documents - oxide only
	Design - Based on	no primary data	kWh/t	A	Client reference documents
2.9	**Ball Mill Work Index**				
	Closing screen size	150	micron	A	Client reference documents
	Average	8.6-14.6	kWh/t	A	Client reference documents
	Design - Based on	14.6	kWh/t	A	Client reference documents - PRIMARY ORE 14.6
2.10	**Raw Water**				
	Ph	N/A		A	Assumption
	Total dissolved solids (TDS)	high 100,000	ppm	A	Assumption
	Density	1.03	t/m^3	A	Assumption
2.11	**Media Consumption**				
	Mill grinding media	1.5	kg/t	A	Calculated
3	**CRUSHING CIRCUIT**				
3.1	**Description**	2 Stage Crushing		A	Client reference documents
3.2	**Crusher Feed Bin**				
	ROM delivery method	Front End Loader		A	Industry standard
	Front end loader size	Cat 980 or equivalent		A	Engineering Database/Experience
	Crusher feed bin live capacity	50	wet t	A	Engineering Database/Experience
	Residence time when full	14.3	min	A	Calculated
	Oversize handling method	nil		A	Assumption
	Grizzly screen aperture	Fixed Grizzly on ROM Bin		A	Engineering Database/Experience
		700 x 700	mm x mm	A	Engineering Database/Experience
3.3	**Primary Crusher**				
	Feed	ROM Bin discharge via variable speed feeder		A	
	Grizzly	Vibrating Grizzly			
	Configuration	Single stage, open circuit		A	Engineer Database/Experience
	Type	Jaw Crusher - Single Toggle		A	Engineer Database/Experience
	Suggested Size	Metso C120 or equiv		A	Calculated - Bruno
	Crusher closed side settings (CSS)	85	mm	A	
	ROM ore F100	800	mm	A	
	Crushed product P100	150	mm	A	
3.4	**Secondary crusher**				
	Feed	Pre-screened product oversize		A	
	Configuration	Single stage, Closed circuit		A	Engineer Database/Experience
	Type	Cone Crusher		A	Engineer Database/Experience
	Suggested Size	HP300 or equiv		A	Calculated - Bruno
	Crusher closed side settings (CSS)	85	mm	A	
	ROM ore F100	800	mm	A	
	Crushed product P100	150	mm	A	
3.5	**Screening**				
	Feed	Jaw Crushed product		A	Engineering Database/Experience
	Configuration	Closing a single stage cone		A	Engineering Database/Experience
	Type	Inclined Double Decker		A	
	Suggested Size	1.7 x 4.9	m x m		
	Top deck aperture	75		A	
	Lower deck aperture	25		A	
	Suggested Size	HP300 or equiv		A	Calculated
	Crusher closed side settings (CSS)	22	mm	A	
3.6	**Dust Collection**				
	Feed	Provision		A	
3.7	**Stockpile**				
	Residence time - live	16	h	A	CPC Recommendation
	Capacity - live	1,200	t	A	CPC Recommendation

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
4	**GRINDING CIRCUIT**				
4.1	**Grinding Requirement**				
	New ore feed F80	20	mm	A	Client reference documents
	Circuit product P80	125	micron	A	Client reference documents
4.2	**Stockpile Reclaim**				
	Reclaim Feeder Configuration	Two reclaim belt feeders + one emergency belt feeder		A	Industry standard
	Reclaim Feeder Capacity	100	t/h/feeder	A	Assumption requiring verification
	Emergency Belt Feeder Capacity	100	t/h/feeder	A	Assumption requiring verification
4.3	**Mill Data**				
	Configuration	Single Stage Ball Mill		A	Client reference documents
	Diameter - inside shell	4.2	m	A	Client reference documents
	Diameter - inside shell	13.8	foot	A	Calculated
	Equivalent grinding length, EGL	5.4	m	A	Client reference documents
	Equivalent grinding length, EGL	17.7	ft	A	Calculated
	Length : Diameter ratio	1.3	L : D	A	Calculated
	Discharge arrangement	Overflow		A	Client reference documents
4.4	**Mill Operating Parameters**				
	Mill rotational speed - operating	15	%Cs	A	Client reference documents, Industry standard
	Ball charge - operating	32	% v/v	A	Industry standard
	Make-up ball size	90	mm	A	Industry standard
	Charge volume - operating	32	% v/v	A	Industry standard
	Mill discharge density - design	74	% w/w	A	Assumption requiring verification
	Mill discharge density - range	72-76	% w/w	A	Industry standard
4.5	**Mill Power Requirements - Primary Ore**				
	Pinion power - maximum	1,206	kW	A	Client reference documents
	Installed power	1,300	kW	A	Client reference documents
4.6	**Mill Discharge Screen**				
	Screen type	Trommel		A	Client reference documents, Industry standard
	Screen deck aperture	10	mm x mm	A	Client reference documents
	Suggested Trommel Diameter	1	m	A	Client reference documents
	Suggested Trommel Length	2	m	A	Client reference documents
4.7	**Classification**				
	Method	Hydrocyclones		A	Industry standard
	Recirculating load - nominal	240	%	A	Client reference documents
	Recirculating load- design	300	%	A	Client reference documents
	Cyclone feed tonnage total feed	300	t/h	A	Calculated
	Cyclone feed tonnage total feed	269	m^3/h slurry	A	Calculated
	Cyclone feed	64	% w/w solids	A	Assumption requiring verification
	Cyclone feed pressure	70	kPa	A	Assumption requiring verification
	No. of cyclone clusters	1	#	A	Industry standard
	Size cyclones	250	mm	A	Assumption requiring verification
	Distributor Outlets	10 to 12	No.	A	Assumption requiring verification
	Number of stand-by units - minimum	2	No.	A	Industry standard
	Number Cyclones Operating	5	No.	A	Assumption requiring verification
	Number Cyclones Installed	7		A	Assumption requiring verification
	Cyclone undeflow:				
	Solids rate	225	t/h	A	Calculated
	Solids Concentration	75	% w/w solids	A	Assumption requiring verification
	Slurry flow rate	150	m^3/h slurry	A	Calculated
	Cyclone overflow:				
	Solids rate	75	t/h	A	Calculated
	Solids Concentration	47	% w/w solids	A	Assumption requiring verification
	Slurry flow rate	110	m^3/h slurry	A	Calculated
4.8	**Mill Discharge Pump Hopper**				
	Input streams	Mill discharge, gravity tail, process water, sump pump discharge and ILR residue		A	
	Approximate discharge volume	255	m^3/h	A	Calculated
	Residence time	40	sec	A	Industry standard
	Live volume	3	m^3	A	Calculated

CPC PROJECT DESIGN

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
5	**GRAVITY GOLD RECOVERY CIRCUIT**				
5.1	**General**				
	Circuit configuration	Process portion of screened cyclone underflow stream		A	Industry standard
		Gravity concentrator tailings split between mill feed and mill discharge hopper		A	Engineer Database/Experience
	Proportion of feed stream treated	26.7	% of CUF	A	Calculated
	Equivalent proportion of new feed	80	% of feed	A	Calculated
5.2	**Gravity Screen**				
	Number of gravity screens	1	ea	A	Engineer Database/Experience
	Feed configuration	Split of cyclone underflow		A	Engineer Database/Experience
	Screen solids feed rate - design	100	t/h	A	Calculated
	Screen feed stream percent solids	63	% w/w	A	Calculated
		92.1	m3/h	A	Calculated
	Screen type	Vibrating, step deck		A	Industry standard
	Screen solid feed flux	40	t/h/m2	A	Industry standard
	Screen aperture	2.4 equivalent	mm	A	Industry standard
	Spray water unit flow	0.15	m3/t feed solids	A	Assumption
	Screen Size Estimate - Area Required	2.5	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 (vendor to advise)	m x m	A	
5.3	**Gravity Concentrators**				
	Type	Batch, Centrifugal		A	Assumption
	Number operating	1	ea	A	Engineer Database/Experience
	Model	KC-XD20 or equivalent		A	Assumption
	Maximum unit flow	60	t/h/unit	A	Vendor Specification
	Typical Fluidising water	8, G5 Cone	m3/h/unit	A	Vendor Specification
	Cycle time	1	h	A	Industry standard
	Concentrate production	10	kg/unit	A	Vendor Specification
	Concentrate production	0.24	t/d	A	Calculated
	Design concentrate production	0.75	t/d	A	Assumption requiring specification
5.4	**Concentrate Processing**				
	Method	Batch Intensive Leach		A	Assumption requiring specification
	Batch processing duration	24	h	A	Industry standard
	Reagent Consumptions			A	
	Sodium cyanide	25	kg/batch	A	Assumption requiring specification
	Sodium hydroxide - leach	3	kg/batch	A	Assumption requiring specification
	Sodium hydroxide - electrowinning	1	kg/batch	A	Assumption requiring specification
	Leachaid / leachwell / Oxidant	2	kg/batch	A	Assumption requiring specification
	Hydrogen Peroxide	0	kg/batch	A	Assumption requiring specification
	Flocculant	0	kg/batch	A	Assumption requiring specification
	Pregnant solution quantity per batch	2	m3	A	Vendor Specification
	Leach residue transfer destination	Mill discharge hopper			Industry Standard
5.5	**Intensive Leach Solution Electrowinning**				
	Daily gold recovery	1,130	g/d	A	Calculated
		4,709	g/t	A	Calculated
	Solution grade	565	ppm	A	Calculated
				A	
	Configuration	Sent to dedicated eluate tank for EW		A	Engineer Database/Experience
	Number of cells	1	No.	A	Calculated - EW Calculated
	Number of operating cells	1	No.	A	Calculated - EW Calculated
	Cell configuration	single cell recirc		A	Industry Standard
	Cell Size	600 x 600	mm x mm	A	Calculated - TBC by vendor - size for 50% grg ie 1mtpa
	Electrowinning Time	8	h	A	Industry Standard
	Cathode type	Stainless steel mesh		A	Industry Standard
	Grade of stainless steel wire	125/152		A	Industry Standard
	Number of cathodes per cell	9	No./cell	A	Calculated
	Operating cell volatge	5	V	A	Industry Standard
	Rectifier type	Rectiformer		A	-
	Rectifier size	800	Amp	A	Calculated - Conservative
	Rectifier voltage	10	V	A	Calculated

No.	Description	Value	Unit	Rev	Source / Comment
	PROCESS DESIGN CRITERIA				
6	**LEACH AND ADSORPTION**				
6.1	**Trash Screening**				
	Screen feed stream	Hydrocyclone overflow stream		A	Industry Standard
	Screen type	Linear vibratory		A	Industry Standard
	Screen panel aperture	0.63 x 12mm slots	mm	A	Industry Standard
	Screen specific flow	40	m3/h/m2	A	Industry Standard - conservatism for oxides
	Screen solids feed rate	75	t/h/m²	A	Calculated
	Screen feed stream percent solids	47	% w/w	A	Calculated
		109.6	m3/h/m2	A	Calculated
	Spray water unit flow	0.15	m3/t feed solids	A	Assumption
	Screen Size Estimate - Area Required	2.7	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 vendor to advise	m x m	A	Assumption requiring verification - to be confirmed by vendor
6.2	**Leach Feed Sampler**				
	Configuration	2 stage located on trash screen feed		A	Industry Standard
6.3	**Leach Feed Stream Data**				
	Leach circuit feed stream				
	Design annual throughput	600,000	t/y	A	Calculated
	Leach circuit solids feed rate - design	75	t/y	A	Calculated
	Leach feed stream density - design	47	% w/w	A	Assumption requiring verification
	Leach feed stream solids dry density	3	t/m3	A	Calculated
	Volumetric flow	109.6	m3/h/m2	A	Calculated
	Slurry SG	1.5	t/m3	A	Calculated
	Gold Feed grade - design	2.1	g/t	A	Calculated
6.4	**Gold Recovery Data**				
	Leach circuit gold extraction - design	95	%	A	Calculated
	Gold resuide grade - design	0.1	g/t	A	Calculated
	Tailings solution gold grade	0.013	ppm	A	Industry Standard
	Correct residue grade (includes solution loss)	0.118	g/t	A	Calculated
	Adsorption Efficiency	99.3	%	A	Calculated
	Recovery - design	94.3	%	A	Calculated
	Recovery - design	1.9	g/t	A	Calculated
6.5	**Gold Production Data**				
	Leach Recovered - design	146	g/h	A	Calculated
		3,501	g/d	A	Calculated
	Leach Recovered - design	113	foz/d	A	Calculated
		37,506	foz/y	A	Calculated
	Overall Recovered - design	194	g/h	A	Calculated
		4,667	g/d	A	Calculated
		150	foz/d	A	Calculated
		50,008	foz/y	A	Calculated
6.6	**LEACH CIRCUIT**				
	Configuration	Hybrid CIL		A	Client reference documents
6.7	**Leach Circuit**				
	Tank Style	Leach tank only		A	Assumption requiring verification
	Number of tanks	1	No.	A	Client reference documents
	Effective total leach residence time	3.4	hr	A	Calculated
	Effectiveleach volume	375	m3	A	Calculated
	Leach feed Ph	10.5		A	Industry Standard
	Leach feed NaCN concentration	500	ppm	A	Assumption requiring verification
	Slurry dissolved oxygen concentration	>5	ppm	A	Industry Standard
	Oxygen / air addition method	Downshaft		A	Industry Standard
	Air flow to tank - design	70	Nm3/h	A	Assumption requiring verification
6.8	**CIL Circuit**				
	Configuration	Series tanks with tank by-basses		A	Industry Standard
	Number of tanks	6	No.	A	Client reference documents
	Effective total CIL residence time	20.5	hr	A	Calculated
	Total Leach/Adsorption slurry residence time at design density	23.9	hr	A	Client reference documents
	Effective total CIL volume	2250	m3	A	Calculated
	Effective CIL tank volume	375	m3	A	Calculated
	Residence time at 42% w/w	20.4	hr	A	Calculated
	CIL feed pH	10.5		A	Client reference documents
	CIL feed NaCN concentration	500	ppm	A	Client Reccomendation
	Slurry dissolved oxygen concentration - Tank 2 - 6	>5	ppm	A	Industry Standard
	Oxygen / air addition method	LOX injected via sparges		A	Industry Standard
	Required oxygen gas flow to Tanks 2 to 6 - design	tbc	Nm3/h	A	Assumption requiring verification - No data

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
6.9	**Carbon Circuit**				
	Carbon concentration - Tank 2	10	g/L	A	Calculated
	Carbon concentration - Tank 3 to 6	10	g/L	A	Calculated
	Design carbon concentration all Leach/CIL Tanks	20	g/L	A	Industry Standard
	Carbon mass in Tank 2	3.8	t	A	Calculated
	Carbon mass in each of Tank 2 to 6	3.8	t	A	Calculated
	Total carbon in adsorption circuit	22.5	t	A	Calculated
	GOLD - design carbon loading	1,650	g/t	A	Client reference documents, Assumption requiring verification
	Average carbon movement	91.2	kg/h	A	Calculated
	Average carbon movement	2.2	t/d	A	Calculated
	Carbon adsorption circuit residence time	10.3	d	A	Calculated
	Carbon transfer period	9	h/d	A	Industry Standard
	Intertank carbon and pulp transfer rate	24.3	m3/h	A	Calculated
	Intertank carbon transfer method	Recessed impeller carbon lift pump		A	Industry Standard
6.10	**Loaded Carbon Recovery Screen**				
	Screen feed stream	Slurry from Tank 2 or Tank 3		A	
	Screen type	Horizontal vibratory, with wash troughs or steps in deck		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Loaded carbon recovery time	4	h	A	Industry Standard
	Screen feed flow - design	75	m3/h	A	Calculated
	Screen Size Estimate - Area Required	1.5	m2	A	Calculated
	Screen Size Estimate - LxW	vendor to advise	m x m	A	Assumption requiring verification
	Carbon recovery method	Recessed inpeller carbon lift pump		A	Industry Standard
6.11	**Intertank Screens**				
	Screen type	Cylindrical, mechanically swept		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen cloth construction	Stainless stell wedge wire		A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Effective screen area	2.7	m2	A	Calculated
	Diameter x Height	vendor to advise	m x m	A	Assumption requiring verification
6.12	**Barren Carbon Screen**				
	Screen feed stream	Quench tank discharge stream		A	
	Screen undersize stream destination	Carbon safety screen		A	
	Screen oversize stream destination	Tank 6 or Tank 7		A	
	Screen type	Horizontal vibratory		A	
	Screen panel aperture	0.8	mm	A	Assumption requiring verification
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Screen Size Estimate - LxW	vendor to advise	m x m	A	Vendor specification
6.13	**Carbon Safety Screen**				
	Screen feed stream	Tank 6 or Tank 7		A	
		Carbon transfer water		A	
		Sump pump slurry		A	
	Screen type	Horizontal vibratory		A	Industry Standard
	Screen panel aperture	0.8	mm	A	Industry Standard
	Screen specific flow	50	m3/h/m2	A	Industry Standard
	Screen Size Estimate - Area Required	2.3	m2	A	Calculated
	Screen Size Estimate - LxW	1.2 x 2.4 vendor to advise	m x m	A	Assumption requiring verification
7	**ELUTION, ACID WASH AND ELECTROWINNING**				
7.1	**General Data**				
	Operating schedule	7	d/wk	A	Assumption
	Circuit capacity	2.5	t	A	Client reference
	Average strip frequency	291.6	strips/y	A	Calculated
		5.6	strips/wk	A	Calculated
	Peak strip frequency	6.1	strips/wk	A	Calculated
	Number of elution columns	1	No.	A	Engineer Database/Experience
	Vessel for acid washing	Column		A	Industry Standard
	Dry carbon bulk density	0.47	t/m3	A	Industry Standard
	Carbon bed volume	5.3	m3	A	Calculated
	Loaded carbon gold grade	1650	g/t	A	Calculated
	Barren carbon GOLD grade	50	g/t	A	Engineer Database/Experience

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
7.2	**Acid Washing**				
	General				
	Acid wash column carbon capacity	2.5	t	A	Calculated
	Acid wash column fabrication	Rubber Lined Carbon Steel		A	Calculated
	Acid type	Hydrochloric Acid, HCl		A	Industry Standard
	Supplied acid concentration	38.4	% w/w	A	Assumption requiring verification
	Acid solution SG - as supplied	1.16		A	Calculated
	Acid Wash				
	Acid wash cycle time	75	min	A	Engineer Database/Experience
	Acid wash solution acid concentration	3	% w/v	A	Industry Standard
	Neat acid volume required	0.3	m3	A	Calculated
	Neat acid pumping time	15	min	A	Industry Standard
	Neat acid pump flow	1.1	m3/h	A	Calculated
	Water flow	2	BV/h	A	Industry Standard
		10.6	m3/h	A	Calculated
	Temperature	ambient	degrees C	A	Industry Standard
	Acid Water Rinse				
	Acid rinse cycle time	120	min	A	
	Water flow	10.6	m3/h	A	Industry Standard
	Number of bed volumes	4	BV/rinse	A	Industry Standard
	Temperature	ambient	degrees C	A	Industry Standard
	Water Source	Strip Solution tank / Strip Soln Pump		A	
	Location of Output	Tails hopper		A	
	Carbon Transfer				
	Process	Carbon transfer from acid wash column to elution column			
	Carbon transfer time	60	min	A	Calculated
	Carbon transfer concentration	235	g/L		Industry Standard
	Transfer flowrate	10.6	m3/h		Calculated
	Education water required	10.6	m3		Calculated
	Transfer water type	Treated water from strip solution tank (via strip solution pump)			
7.3	**Elution**				
	General				
	Elution method	SPLIT - AARL		A	Engineer Database/Experience
	Elution column construction	Stainless Steel 316		A	Industry Standard
	Elution column operating pressure	300	kPag	A	Engineer Database/Experience
	Dosed NaOH concentration	76	% w/v	A	Engineer Database/Experience - Equivalent to 30% w/w
	Dosed NaOH solution SG	1.52		A	Calculated
	Dosed NaCN concentration	28	% w/v	A	Industry Standard, Calculated
	Pre-treatment				
	Description	The column is primed by supplying water from the strip solution tank		A	
		Following priming, the circuit is heated by circulating solution to column and back to strip solution pump suction		A	
		Sodium cyanide and sodium hydroxide solutions are dosed into the circuit during the heating stage.		A	
	Cycle time	30	min	A	Engineer Database/Experience
	Solution flow	2	BV/h	A	Industry Standard
		10.6	m3/h	A	Calculated
	Temperature operating range	ambient - 125	degrees C	A	Indusrty Standard
	Pre-soak - number of bed volumes	0.8	BV	A	Engineer Database/Experience
	Pre-soak - solution volume - From Starter Tank	4.3	m3	A	Calculated
	Column solution discharge destination	Eluate tank		A	
	Pre-soak solution NaOH concentration	3	% w/v	A	Indusrty Standard
	Pre-soak solution NaNC concentration	3	% w/v	A	Indusrty Standard
	Pre-treatmeant NaOH concentration	127.7	kg/pretreat	A	Calculated
		0.2	m3/pretreat	A	Calculated
	Pre-treatment NaCN consumption	127.7	kg/pretreat	A	Calculated
		0.5	m3/pretreat	A	Calculated
	Elution1 - from starter tank				
	Cycle time	60	min	A	Indusrty Standard
	Elution solution flowrate	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	10.6	m3	A	Calculated
	Temperature	125	degrees C	A	Indusrty Standard
	Column solution discharge destination	Eluate tank		A	

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
	Elution2- water (a) - From strip solution tank				
	Cycle time	67	min	A	Engineer Database/Experience
	Elution solution flowrate	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	11.9	m3	A	Calculated
	Temperature	125	degrees C	A	Indusrty Standard
	Column solution discharge dstination	Eluate tank		A	Indusrty Standard
	Elution3- water (b) - from strip solution tank				
	Cycle time	60	min	A	Engineer Database/Experience
	Elution solution flowrate	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	10.6	m3/h	A	Calculated
	Temperature	125	degrees C	A	Indusrty Standard
	Column solution discharge dstination	Starter eluate tank		A	Indusrty Standard
	Cooling				
	Cycle time	30	min	A	Engineer Database/Experience
	Elution solution flowraye	2	BV/h	A	Indusrty Standard
		10.6	m3/h	A	Calculated
	Solution volume	5.3	m3	A	Calculated
	Temeprature	Ambient	degrees C	A	Indusrty Standard
	Column solution discharge destination	Starter eluate tank		A	Indusrty Standard
	Total Elution				
	Total bed volumes	8	BV/elution	A	Indusrty Standard
	Total elution water requirement	42.7	m3/elution	A	Calculated
	Total elution time	247	min	A	Calculated
	Electrolyte NaOH addition	59.6	kg	A	Calculated
	Volume of NaOH solution to electrolyte tank	0.1	m3	A	Calculated
	Carbon Transfer				
	Process	Carbon transfer from elution column to reactivation kiln hopper			
	Carbon transfer time	60	min	A	Calculated
	Carbon transfer concentration	235	g/L	A	Indusrty Standard
	Transfer flowrate	10.6	m3/h	A	Calculated
	Eduction water required	10.6	m3	A	Calculated
	Transfer water type	Treated water from strip solution tank		A	Indusrty Standard
	Total Stripping Circuit Water Consumption				
	Acid wash	4.3	m3/Strip	A	Calculated
	Water rinse	21.3	m3/Strip	A	Calculated
	Carbon transfer	10.6	m3/Strip	A	Calculated
	Column prime	4.3	m3/Strip	A	Calculated
	Water elution a	11.9	m3/Strip	A	Calculated
	Water elution b	10.6	m3/Strip	A	Calculated
	Cooling	5.3	m3/Strip	A	Calculated
	Carbon transfer	10.6	m3/Strip	A	Calculated
	Total	78.9	m3/Strip	A	Calculated
	Total RO Water Required per strip / day	62.8	m3/Strip	A	Calculated
7.4	**Elution Equipment**				
	Eluate Tank	5	BV sizing	A	Engineering Database/Experience - conservative
	Number of eluate tanks	1	No.	A	Client recommendation - 2 for 1 Mt/a
	Eluate volume per tank	26.6	m3	A	Calculated - note working vol
	Solution Tanks				
	Starter Elute Tanks	26.6	m3	A	Calculated - note working vol
	Strip Solution Tank	26.6	m3	A	Calculated - note working vol
	Elution Heater				
	Configuration	1 x Recovery Heat Exchanger1 x Primary Heat Exchanger		A	
	Elution heater type	LPG	-	A	Assumption requiring verification
	Elution heater size	1000	kW	A	Calculated - not sized for 1Mt/a

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
7.5	**Electrowinning**				
	Number of cells	1	No.	A	Calculated
	Number of operating cells	1	No.	A	Calculated
	Cell configuration	-		A	Industry standard
	Cell size	800 x 800	mm x mm	A	Calculated
	Effective cell cross-sectional area	0.64	m2	A	Calculated
	Electrowinning cell linear velocity	300	L/min/m2	A	Industry standard
	Electrowinning cell solution feed flow	11.5	m3/h/cell	A	Calculated
		11.5	m3/h	A	Calculated
	Electrowinning time	10	h	A	Industry standard
	Cathode type	Stainless steel mesh		A	Industry standard
	Grade of stainless steel wire	125/152		A	Industry standard
	Number of cathodes per cell	9	No./cell	A	Calculated
	Operating cell voltage	5	V	A	Industry standard
	Current efficiency - Au	8	%	A	Industry standard
	Current efficiency - Ag	13	%	A	Industry standard
	Current per cell	789	Amp/cell	A	Calculated
	Current per cathode	87.7	Amp/cathode	A	Calculated
	Current per cathode area	10.3	Amp/m2 cathode	A	Calculated
	Rectifier type	Rectiformer		A	-
	Rectifier size	1000	Amp	A	Calculated
	Rectifier voltage	10	V	A	Calculated
	Barren solution gold concentration	2	ppm	A	Industry standard
	Barren solution silver concentration	5	ppm	A	Industry standard
8	**CARBON REACTIVATION**				
8.1	**Reactivation Kiln**				
	Type of kiln	Horizontal, diesel fired		A	Indusrty Standard
	Dewatering screen type	Sieve Bend		A	Indusrty Standard
	Carbon reactivation capacity	150	kg/h	A	Calculated
	Carbon reactivation capacity - design	250	kg/h	A	Calculated - Client to Advise - larger for 1mtpa
	Operating temperature - design	720	degrees C	A	Indusrty Standard
	Operating temperature - range	650 - 800	degrees C	A	Indusrty Standard
	Carbon processing time	16.7	h/batch	A	Indusrty Standard
	Number of feed hoppers	1	No.	A	Engineering Database/Experience
	Feed hopper capacity	3	t	A	Engineering Database/Experience
8.2	**Carbon Quench Tank**				
	Required holding capacity	3	t	A	Engineering Database/Experience
	Carbon concentration	400	g/L	A	Indusrty Standard
	Tank type	Closed top with conical bottom		A	Indusrty Standard
	Effective quench tank volume	7.5	m3	A	Calculated
	Carbon transfer method	Recessed impeller pump		A	Indusrty Standard
	Carbon transfer time	1.5	h	A	
	Carbon transfer concentration	150	g/L	A	Indusrty Standard
	Carbon transfer rate	13.3	m3/h	A	Calculated
	Make-up water type	Raw water		A	Indusrty Standard
8.3	**Carbon Sizing Screen**				
	Screen feed stream	Quench tank discharge stream			
	Screen undersize stream destination	Carbon safety screen			
	Screen oversize stream destination	Tank 6 or Tank 5			
	Screen type	Horizontal vibratory			
	Screen panel	0.8			
	Screen specific flow	50			
9	**GOLDROOM**				
9.1	**Cathode Treatment**				
	Method	High pressure washing of sludge from cathodes		A	Industry standard
	Sludge filtration method	Decant and pressure filtration		A	Industry standard
	Drying method	Drying oven		A	Industry standard
9.2	**Smelting**				
	Type of furnace	LPG fired, tilting		A	Industry Standard
	Number of furnaces	1		A	Engineer Database/Experience
	Crucible size	A100		A	Assumption requiring verification - 2 smelts per week at 1 Mt/a; increase to A200 for less smelts but should be ok for 1 Mt/a
	Mould arrangement	Cascade		A	Industry Standard

No.	Description	Value	Unit	Rev	Source / Comment
10	**WASTE DISPOSAL**				
10.1	**Tailings Thickener**	Tickening not inc		A	
10.2	**CIL Discharge Pump Hopper**				
	Input streams	Mill discharge, gravity tail, process water, sump pump discharge and ILR residue		A	
	Discharge density - design	42	% w/w	A	Assumption requiring verification
	Discharge - typical	45	% w/w	A	Assumption requiring verification
	Approximate discharge volume	128.6	m3/h	A	Calculated
	Residence time	90	sec	A	Industry standard
	Live volume	3.2	m3	A	Calculated
10.3	**Tailings Facility**				
	Tailings Settled Density	70	% w/w solids	A	Assumption requiring verification
	Water reclaimed from Tailings Dam	75	%	A	Assumption requiring verification - rest to evaporation
		24.1	m3/h	A	Calculated
	Tailings decant return pump operational time	8	h/d	A	Assumption requiring verification
	Tailings decant return pump - design flow	144.7	m3/h	A	Calculated
11	**REAGENTS**				
11.1	**Quicklime**				
	Consumption	4	kg/t	A	Client reference documents - high saline water - similar project
	Design consumption	10	kg/t	A	Assumption requiring verification
		18005	kg/d	A	Calculated
	Addition point	Mill feed chute		A	Assumption
	Solids SG	2300		A	
	Number of storage tanks	1	#	A	
	Hopper capacity	240	h	A	Industry standard
	Hopper Volume	78.3	m3	A	Calculated
	Nominal lime storage capacity	180	t	A	Calculated - 2 x 100 tonne silos. Consider 1 larger then leave spares for a second fot 1mtpa
11.2	**Sodium Cyanide**				
	CIL usage	240	t/y	A	Calculated
	CIL consumption, design	0.4	kg/t	A	Client Reccomendation
	Elution consumption	0.72	t/week	A	Calculated
	Elution usage	37	t/y	A	Calculated
	Elution consumption	0.06	kg/t	A	Calculated
	ILR Usage	8.3	t/y	A	Calculated
	ILR Consumption	0.01	kg/t	A	Calculated
	Total usage	277	t/y	A	Calculated
	Total consumption, design	0.46	kg/t	A	Calculated
	Hourly consumption	35	kg/h	A	Calculated
	Daily usage	832	kg/d	A	Calculated
	Total Consumption				
	Mixture strength	28	%w/v	A	Industry Standard
	Hourly consumption at concentration supplied % w/w	124	L/h	A	Calculated
	Total daily volumentric flow	3	m3/d	A	Calculated
	Solution Storage				
	Storage live volume required	40	m3	A	Engineer Database/Experience
	Storage residence	13.5	d	A	Calculated
	Reagent Dosing				
	Dosing to CIL, method	Ring main		A	Industry Standard
	Dosing to CIL, design	124	L/h	A	Calculated
	Ring main flow	0.5	m3/h	A	Calculated
	Dosing to elution, method	Dedicated pump		A	Industry Standard
	Time to dose to elution	15	min	A	
	To elution strip at 3% soak, 20% w/v stock solution	0.5	m3	A	Calculated
	Dosing pump capacity	1.82	m3/h	A	Calculated

PROCESS DESIGN CRITERIA					
No.	**Description**	**Value**	**Unit**	**Rev**	**Source / Comment**
11.3	**Sodium Hydroxide**				
	Elution consumption	1.1	t/week	A	Calculated
	Elution usage	55	t/y	A	Calculated
	Elution consumption	0.09	kg/t	A	Calculated
	ILR Usage	1.33	t/y	A	Calculated
	ILR Consumption	0.002	kg/t	A	Calculated
	Total usage	56	t/y	A	Calculated
	Total consumption, design	0.09	kg/t	A	Calculated
	Hourly consumption	6.99	kg/h	A	Calculated
	Daily usage	168	kg/d	A	Calculated
	Solution Storage				
	Consumption per day	0.221	m3/d	A	Calculated
	Mixture strength	76	% w/v	A	Industry Standard
	Storage live volume required	16	m3/d	A	Calculated
	Storage residence	72	d	A	Assumption requiring verification
	Reagent Dosing				
	Dosing to elution, methos	Dedicated pump		A	Industry Standard - can be directed to ILR when required
	Time to dose	16	min	A	
	Quantity to dose per strip at 3% soak, 39% w/v stock solution	0.17	m3	A	Calculated
	Dosing pump capacity	0.63	m3/h	A	Calculated
11.4	**Hydrochloric Acid**				
	Neat acid wash consumption	1.9	t/wk		Calculated
	Neat acid wash usage	97	t/y		Calculated
	Neat acid consumption	0.16	kg/t		Calculated
	Hourly consumption	12.1	kg/h		Calculated
	Daily usage	291	kg/d		Calculated
	Storage Tank				
	Hourly consumption	10	L/h	A	Calculated
	Total daily volumetric flow	0.25	m3/d	A	Calculated
	Storage live volume required	3.5	m3	A	Engineer Database/Experience
	Storage capacity	14	d	A	Calculated
	Dosing to acid method	Dedicated pump		A	Calculated
	Time to dose	10	min	A	Calculated
	Quantity to dose per strip at 3% soak, 33% w/w stock solution	0.29	m3	A	Calculated
	Dosing pump capacity	1.15	m3/h	A	Calculated
11.5	**Grinding Media**				
	Ball Mill	1.5	kg/t	A	Calculated
	Consumption	2.7	t/d	A	Calculated
	Usage	900	t/y	A	Calculated
11.6	**Carbon**				
	Consumption	0.5	% loss/strip	A	Assumption requiring verification-Typically ~1% for conventional, halved for extruded
		6	g/t	A	Calculated
	Usage	0.01	t/d	A	Calculated
11.7	**Antiscalant**				
	Type				
	Consumption	15	g/t	A	Assumption
	Uage	9	t/y	A	Calculated
11.8	**Sulphamic Acid**				
	Consumption	0.5	kg/Strip	A	Assumption requiring verification
	Usage	146	kg/y	A	Assumption
11.9	**Goldroom Fluxes**				
	Borax flux				
	Consumption	0.05	kg/kg PMs	A	Industry Standard
	Usage	71	kg/y	A	Calculated
	Consumption	0.12	g/t	A	Calculated
	Soda Ash flux				
	Consumption	0.04	kg/kg PMs	A	Industry Standard
	Usage	57	kg/y	A	Calculated
	Consumption	0.1	g/t	A	Calculated
	Potassium Nitrate flux				
	Consumption	0.01	kg/kg PMs	A	Industry Standard
	Usage	14.3	kg/y	A	Calculated
	Consumption	0.024	g/t	A	Calculated
	Silica Flour flux				
	Consumption	0.005	kg/kg PMs	A	Industry Standard
	Usage	7.1	kg/y	A	Calculated
	Consumption	0.01	g/t	A	Calculated

No.	Description	Value	Unit	Rev	Source / Comment
	PROCESS DESIGN CRITERIA				
12	**WATER SERVICES**				
12.1	**Raw Water**				
	Supply source	To be defined		A	
	Receiving storage type, plant site	Raw water tank		A	
	Minimum storage capacity, plant site w/ fire water allowance	528	m3	A	
	Storage capacity	12	h	A	
	Raw Water Demand				
	Raw water to RO treatment	5	m3/h	A	
	Raw water to crushing	2	m3/h	A	
	Raw water to gravity concentrator	9.1	m3/h	A	
	Raw water to CIL	0.4	m3/h	A	
	Raw water to tailings area	0	m3/h	A	
	Elution & ILR	0	m3/h	A	
	Raw water to reagent mixing	0.8	m3/h	A	
	Miscellaneous	10	m3/h	A	
	Total	27	m3/h	A	Calculated
		653	m3/h	A	Calculated
	Raw water pump flow - design	54	m3/h	A	Engineer Database/Experience
	Raw water outlet pressure	250	kPa	A	Industry Standard
12.2	**Fire Water**				
	Supply source	Allocated portion of Raw Water		A	
	Circuit configuration	Electric jockey pump to maintain main pressure		A	
		Dedicated electric driven main pump with low pressure activation		A	
		Back up diesel driven main pump with low pressure activation		A	
	Firewater reserve allowance	144	m3	A	2 hr storage
	Firemain inlet flow basis	2 attack hydrants operating at 10L/s each		A	
	Firemain inlet flow	72	m3/h	A	
12.3	**RO Water**				
	Supply source	Raw Water, waste to PW dam		A	
	Circuit configuration	Use of raw water to supply essential RW		A	
	Storage capacity	50	m3	A	
	Outflow	12	m3/h	A	
	Capacity from full	4.2	h	A	
12.4	**Elution Water**				
	Supply source	RO Storage		A	Industry Standard
	Elution water requirement - nominal	4.5	m3/h	A	Assumption
	Elution water (strip solution tank) fill rate requirement (ro Water pump sizing)	36	m3/h	A	Assumption - 3hr fill
12.5	**Gland Seal Water**				
	Supply source	Raw water tank		A	Industry Standard
	Gland seal water pump discharge	5	m3/h	A	Assumption
	Gland seal water outlet pressure	1000	kPa	A	Engineer Database/Experience
12.6	**Potable Water**				
	Configuration	RO water treated to UV		A	
	Supply source	Filtered and treated raw water		A	Assumption requiring verification
	Receiving storage type	Plastic lines- squatters type		A	Assumption requiring verification
	Storage capacity	50	m3	A	Assumption requiring verification
	Potable Water Demand				
	Consumption	200	L/man/d	A	Industry Standard
	Personnel on site	300	No.	A	Assumption requiring verification
	Water usage	60	m3/d	A	Calculated
	Potable water pump flow - design	10	m3/h	A	Engineering Database/Experience
12.7	**Safety Shower Water**				
	Supply source	RO storage tank		A	Assumption requiring verification
	Supply method	Circulating ring main		A	Industry Standard
	Ringmain inlet flow	20	m3/h	A	Industry Standard
	Minimum available safety volume	10	m3	A	Engineering Database/Experience

PROCESS DESIGN CRITERIA

No.	Description	Value	Unit	Rev	Source / Comment
12.8	**Process Water**				
	Supply source	decant water and water from settlement ponds to make-up raw water		A	
	Receiving storage type	Pond		A	Assumption requiring verification
	Storage capacity	1,800	m3	A	Calculated
	Storage residence time	24	h	A	Calculated
	Evaporation	6	mm/m2/day	A	Assumption
	Secondary process water sources	Surplus mine dewatering; site stormwater collection ponds		A	Assumption requiring verification
	Mine dewatering excess	0	m3/h	A	Assumption requiring verification - set to neg
	Site water runoff	0	m3/h	A	Assumption
	Site area	0	m2	A	Assumption - set to neg tbc
	Average rainfall	0	mm/y	A	Assumption - set to neg
	Proportion of rainwater collected	0	%	A	Engineering Database/Experience
	Process Water Demand				
	Grinding	71	m3/h	A	Calculated - from mass balance
	Miscellaneous	4	m3/h	A	Calculated - from mass balance
	Total	75	m3/h	A	Calculated
		1,800	m3/d	A	Calculated
	Process water pump flow - design	150	m3/h	A	Engineering Database/Experience
	Process water outlet pressure	150	kPs	A	Engineering Database/Experience - outlet means point of service not d/c
13	**AIR SERVICES**				
13.1	**Configuration**	All HP air will be instrument quality		A	Industry Standard
13.2	**High Pressure Air**				
	Compressor type	Rotary screw		A	Engineering Database/Experience
	Capacity	700	m3/h FAD	A	Assumption
	Discharge pressure	800	kPa	A	Industry Standard
	Number of units	1 duty and stand-by	ea	A	Industry Standard
	Air Receiver capacity	3,000	L	A	

Appendix D
OVERALL PLANT LAYOUT DRAWINGS

CPC Project Design Pty Ltd
ABN 64 153 273 539

6 Kings Park Road
West Perth WA 6005

PO Box 1441
West Perth WA 6872

Contact Person:
Gavin Nowrojee
Study Manager

Phone: +61 8 9365 0300
Mobile: +61 406 011 457
Email: gnowrojee@cpceng.com.au