AFRICAN THUNDER KALPLATS

PROCESS AND MINING TECHNICAL STUDY UPDATE 2020

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APPROVAL

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SYSTEM OF UNITS

The international metric system of units (SI) will be used throughout the design in all documentation, specifications, drawings, reports and all other documents associated.

EXECUTIVE SUMMARY

The Kalplats Project is located approximately 350 km west of Johannesburg in the North West Province and 45 km west of Kalgold open pit gold operations on the Kraaipan Greenstone Belt. African Thunder Platinum Limited ("ATPL") engaged DRA to conduct a technical update based on the 2010 Definitive Feasibility Study ("DFS") on the Kalahari Platinum ('Kalplats') Project. The scope of work included restating the economic model to current terms based on the mining optimisation results and associated technical updates.

The project has been baselined on the development of an open cast mine with a processing plant capable of treating 1.5 million tonnes per annum. A technical revalidation has been conducted covering the mining and processing facilities only. The same geology and geotechnical information from the 2010 Kalplats DFS have been used as the basis of design. Knight Piésold undertook a technical review of the tailing's storage solution including associated capital cost estimation. Bulk infrastructure including the supply of water and power services has not been revalidated by DRA as these rely on joint venture agreements still to be formalised. As such, costs associated with bulk infrastructure has been supplied by ATPL.

Mine and Plant Feed Schedule

The mine design and schedule has been revised and optimised for eight known deposits which have measured and indicated resources. The mine and plant feed schedule over life of mine is presented in Figure 1-1 and Figure 1-2 below. Over life of mine, a total of 17.7 million tonnes of mineralized material (oxides and sulphides) is delivered to the processing facility, with 124 million tonnes of waste removed over the same period, equivalent to a strip ratio of 7.01. The average 2E+Au basket grade (Pt, Pd, Au) over life of mine is estimated at 2.92 g/t mineralised material. A concentrate 2E+Au basket grade of 150g/t is estimated equivalent to an approximate 1.3% mass pull.

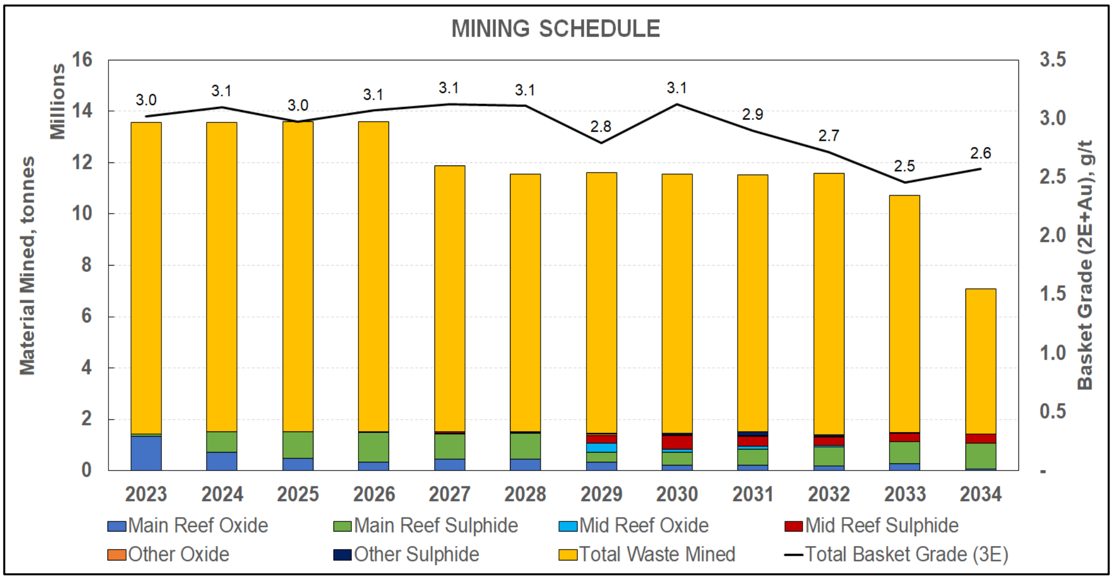

Figure 1-1: Mine Feed Schedule

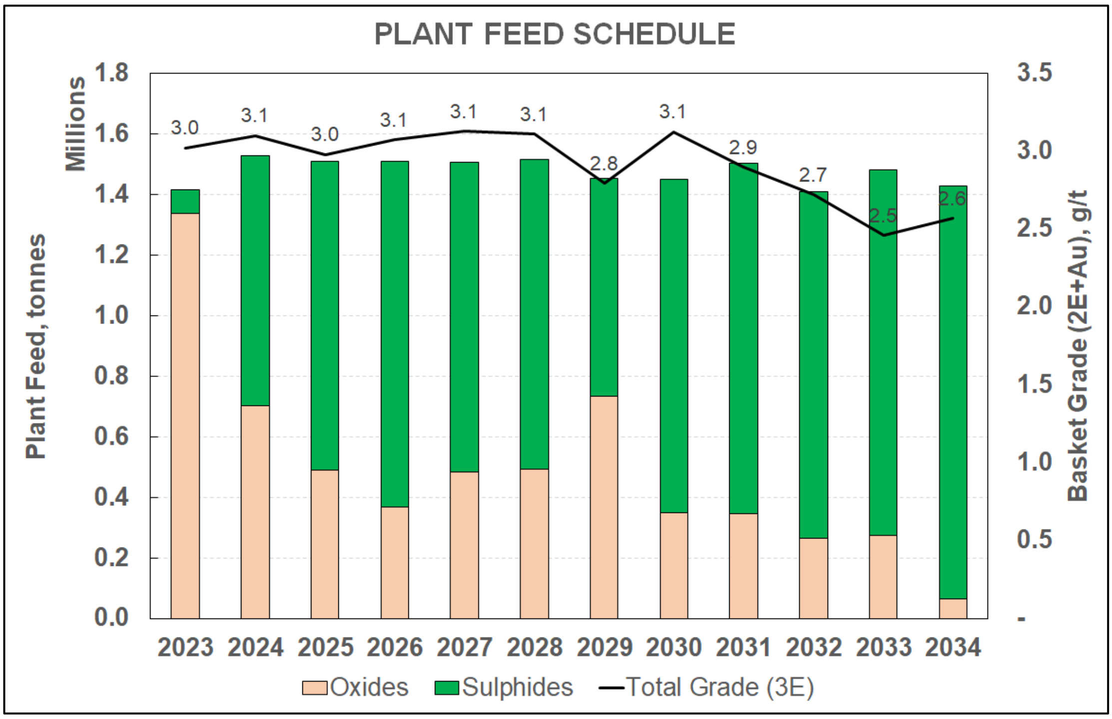

Figure 1-2: Plant Feed Schedule

Capital Expenditure and Phasing

The total initial capital estimate for the project, which includes pre-stripping for mine development, construction, direct cost, indirect costs, TSF and contingency is estimated to be ZAR 2,263m. Mining will be performed using contract mining and as such all initial mining capital costs, apart from pre-stripping, is included in the mining contractor's rate. Capital phasing has been based on the outcomes of the implementation schedule (as per the 2010 feasibility study) defined which allows for an overall construction period of approximately 2 years. Capital phasing relating to the TSF has been provided by Knight Piésold. A USD:ZAR exchange rate of 17 has been applied for cost elements exposed to foreign currency.

Table 1-1: Initial Capital Cost Summary

Description	Unit	Value	% of Total	2021	2022	2023	2024
Direct							
Mining	ZARm	95	4	26	39	30*	-
Process Plant	ZARm	967	43	387	580	-	-
Tailings$^{\beta}$	ZARm	279	12	135	65	30	49
Power Supply$^{\alpha}$	ZARm	181	8	72	108	-	-
Water Supply$^{\alpha}$	ZARm	216	10	86	130	-	-
General Infrastructure	ZARm	138	6	55	83	-	-
Total Direct Capital	**ZARm**	**1,875**	**83**	**761**	**1,005**	**60**	**49**
Indirects							
Land Purchase$^{\alpha}$	ZARm	14	1	14	-	-	-
Other Indirects	ZARm	197	9	79	118	-	-
Total Indirect Capital	**ZARm**	**211**	**9**	**92**	**118**	**-**	**-**
Contingency$^{\#}$	**ZARm**	**178**	**8**	**91**	**86**	**-**	**-**
Total Initial Capital	**ZARm**	**2,263**	**100**	**945**	**1,210**	**60**	**49**

*Mining pre-development costs, $^{\#}$Contingency allowance is exclusive of TSF, $^{\alpha}$Costing supplied by ATPL, $^{\beta}$Costing supplied by Knight Piésold

Operating Costs

The operating costs over life of mine include mine operations, process plant operations, general and administrative costs and tailings disposal and tailings management costs. The cost reported excludes the cost of capitalized mine pre-stripping. A USD:ZAR exchange rate of 17 has been applied for cost elements exposed to foreign currency.

Table 1-2: LoM Operating Cost Summary

Area	Value, ZARm	Unit Cost, ZAR/t ore	% of Total
Mining	5,316	300	59
Plant	3,381	191	38
Tailings Handling$^{\theta}$	44	2	<1
G&A$^{\theta}$	240	14	3
Total Operating Cost	8,981	507	100

$^{\theta}$Costing supplied by ATPL

Metal Pricing and Revenue

No formal market study has been conducted during this revalidation exercise. Project revenue is based on producing a saleale PGM concentrate with pay metals being Pt, Pd and Au using metal prices provided by ATPL.

Table 1-3: Metal Pricing

Area	Unit	2023	2024	LT	Source
Pt	USD/ozt	1,094	1,117	1,101	ATPL
Pd	USD/ozt	1,556	1,414	1,306	ATPL
Au	USD/ozt	1,601	1,499	1,415	ATPL

Net Smelter Return

Revenue has been discounted to account for refining costs applicable to an off-take sale agreement for a PGM concentrate. A payability factor of 82.5% of generated revenue has been included and assumes producing a product below penalty or rejection limits. This factor has been supplied by ATPL and applied in the economic model.

Economic Outcomes

The economic analysis has been carried out using discounted cash flow (DCF) methodologies and has been based on earnings before interest, taxation, depreciation and amortisation (EBITDA). All cash flows are presented in constant terms and as such does not consider the effects of inflation, interest, escalation and other financial adjustments. A 10% constant discount rate has been used in the analysis. No allowance for salvage is included in the model. A lump-sum capital allowance of ZAR 40m during year 7 of operation has been included to cover costs associated with the mining contractor renewal / fleet replenishment. The economic model includes an allowance for annual sustaining capital of 1% of capital expenditure for the process plant. Project royalties have been based on South African legislation relating to the production of an unrefined mineral resource. A USD:ZAR exchange rate of 17 has been applied. The Base Case and Upside key project outputs are presented below. The upside scenario considers a potential recovery improvement (+4%) and the subsequent impact on the project economics. Further metallurgical testwork would be necessary in order to validate the upside potential.

Table 1-4: Key Project Outputs

Item	Unit	Base Case	Upside
USD:ZAR Exchange Rate	-	17	17
Metal Pricing	-	As per Table 1-3	
Initial Capital Cost	ZARm	2,263	2,263
Unit Operating Costs	ZAR/t	507	507
Material Milled	tonnes	17,700,385	17,700,385
Ave LoM Plant Feed Grade	g/t (2E+Au)	2.92	2.92
Metal Ounces Produced			
Pt	ozt	515,526	546,720
Pd	ozt	539,293	571,788
Au	ozt	44,756	47,459
Economic Outcomes (Pre-tax)			
NPV @ 10%	ZARm	2,352	2,881
NPV @ 10%	USDm	138	169
IRR	%	27.8	31.3
Payback Period	years	3.1	2.8
AISC	USD/ozt (2E+Au)	771	747

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ACRONYMS

Acronyms	Descriptions
AAPS	Anglo American Prospecting Services
ARM Platinum	African Rainbow Minerals
ATPL	African Thunder Platinum Limited
BIC	Bushveld Igneous Complex
DFS	Definitive Feasibility Study
DWAF	Department of Water Affairs & Forestry
FX	Foreign Exchange
g/t	Gram per Tonne
Harmony	Harmony Gold Mining Company Limited
Kalplats	Kalahari Platinum
km	Kilometre
kWhr/t)	Kilowatt hours per tonne
LG	LG reef
LM	Lower Main
LoM	Life of Mine
m	meter
m^3	Cubic Meter
mm	Millimetre
MM	Mid-Main
Mtph	Million Tonnes per Hour
MRLG	Mid-Reef Low Grade
PALA	Pala Investments
PFS	Prefeasibility Study
RoM	Run of Mine
SLI	Stella Layered Intrusion
STP	Sewage Treatment Plant
UCS	Uniaxial Compressive Strength
UM	Upper Main

1. INTRODUCTION

African Thunder Platinum Limited (“ATPL”) has commissioned DRA to update the 2010 Definitive Feasibility Study (“DFS”) on the Kalahari Platinum (‘Kalplats’) Project.

The project is based on the development of an open cast mine with a processing plant capable of treating 1.5 million tonnes per annum and associated site infrastructure. A technical revalidation has been conducted covering the mining and processing facilities only. The scope has entailed a review of the mine design, production scheduling, process plant and site infrastructure. The outcomes of the technical revalidation have been incorporated into an overall project valuation model based on current 2020 mineral economics projection and updated 2020 capital and operating costs. Bulk infrastructure including the supply of water and power services has been specifically excluded in the revalidation study as these rely on joint venture agreements still to be formalised. Provisional costing for these items has been based on previous estimates supplied by ATPL. Knight Piésold have reviewed the tailings storage facility design and provided the associated cost estimates for the economic model.

DRA was commissioned by ATPL to assemble this Feasibility Study based on information supplied by ATPL, its consultants and others.

The entities involved in the providing data for the Study and their areas of responsibilities have been established as follows:

➢ ATPL	Exploration Geology and Interpretation
	2010 Geological Report
➢ 2010 Pit Geotechnical report	
	Owner’s Costs
	Test Work Interpretation
	2010 Bulk Infrastructure Capital Costs (Water and Power Supply)
	G&A Operational Cost
	Tailings Handling Operational Cost
➢ Coffey	2010 Resource Estimation
➢ DRA	Open cast mine design
	Open pit production schedule

	Reserve Statement
	Mining Report
	Update Contractor Mining Capital Costs
	Update Contractor Mining Operating Costs
	Update Contractor Mining Stay in Business Costs
	Confirm Rock Dump Design and Capacity
	Review Test Work Interpretation
	Confirm Process Plant Design and Capacity
	Update Capital Cost Estimate (exclusive of bulk infrastructure)
	Update Operating Cost Estimate (exclusive to G&A and Tailings Handling)
	Overall Study Management
	Financial Model and Update Project Evaluation
➢ Knight Piésold	Update Tailings Storage Facility Design and Costing
➢ Digby Wells	2010 Hydrogeological report
➢ AGES	2010 Environmental report
	Associated legal requirements report
	Water

2. PROJECT BACKGROUND

2.1. Project Location

The Kalplats project is located approximately 350 km west of Johannesburg in the North West Province and 45 km west of Kalgold open pit gold operations on the Kraaipan Greenstone Belt. The project area lies approximately 25 km north of Stella within a farming area and a local population of approximately 2,500 inhabitants. The R47 regional highway linking the towns of Mafikeng and Vryburg locates to the south of the project area (Figure 2-1). The topography is slightly undulating to flat-lying with the average surface elevation variable between 1,245 m to 1,375 m above mean sea level.

The proposed mine will be developed on a green-fields site on portions 6 and 8 of the farm

Groot Gewaagd 270, portions 2, 3, 4, 9 and 13 of the farm Gemsbok Pan 309, portions 13, 14, 15, 16 and 18 of the farm Koodoos Rand 321 and on portions 1, 2 and RE of the farm Papiesvlakte A 323 (Figure 2-2)

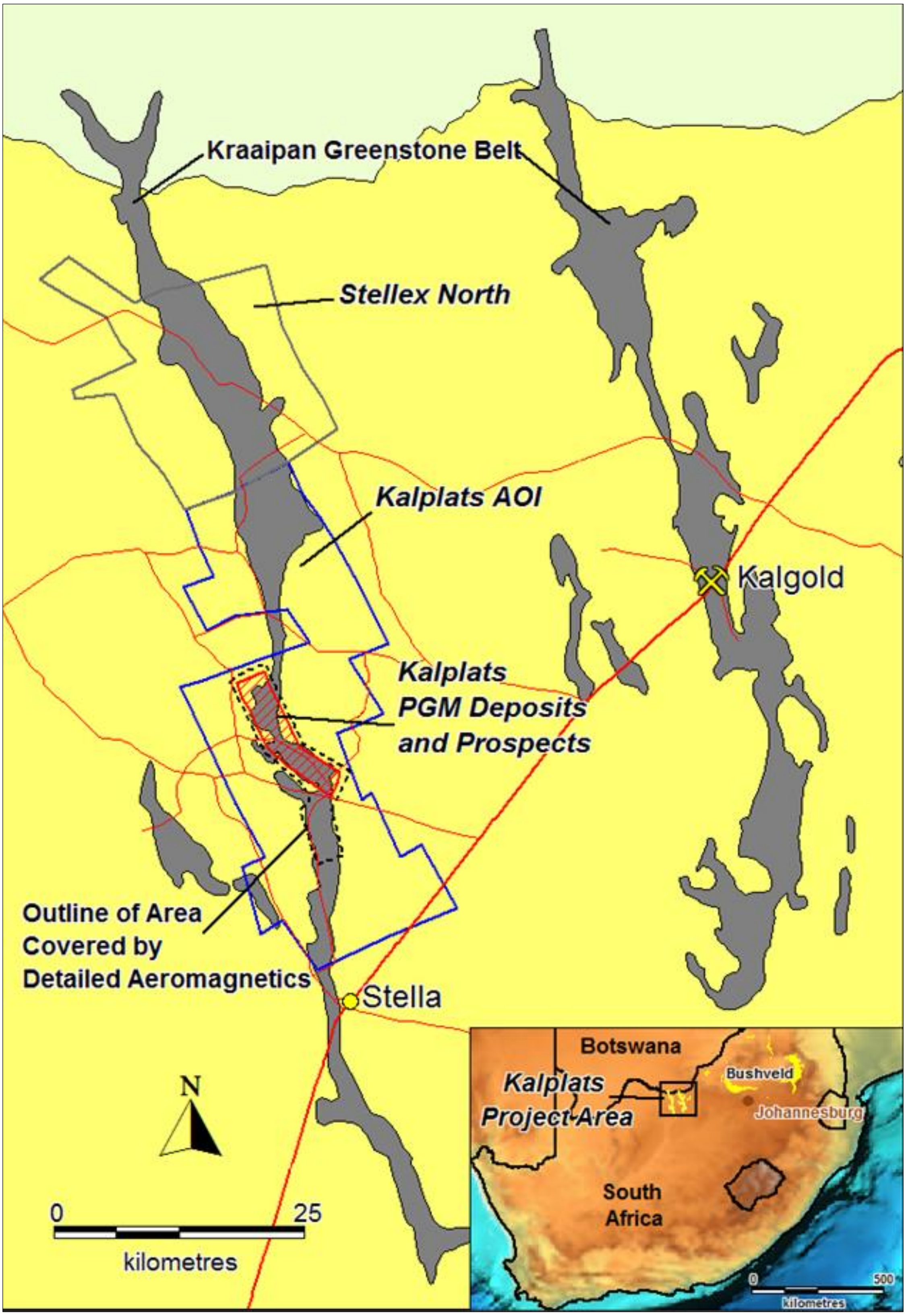


Figure 2-1: Kalplats Project Location

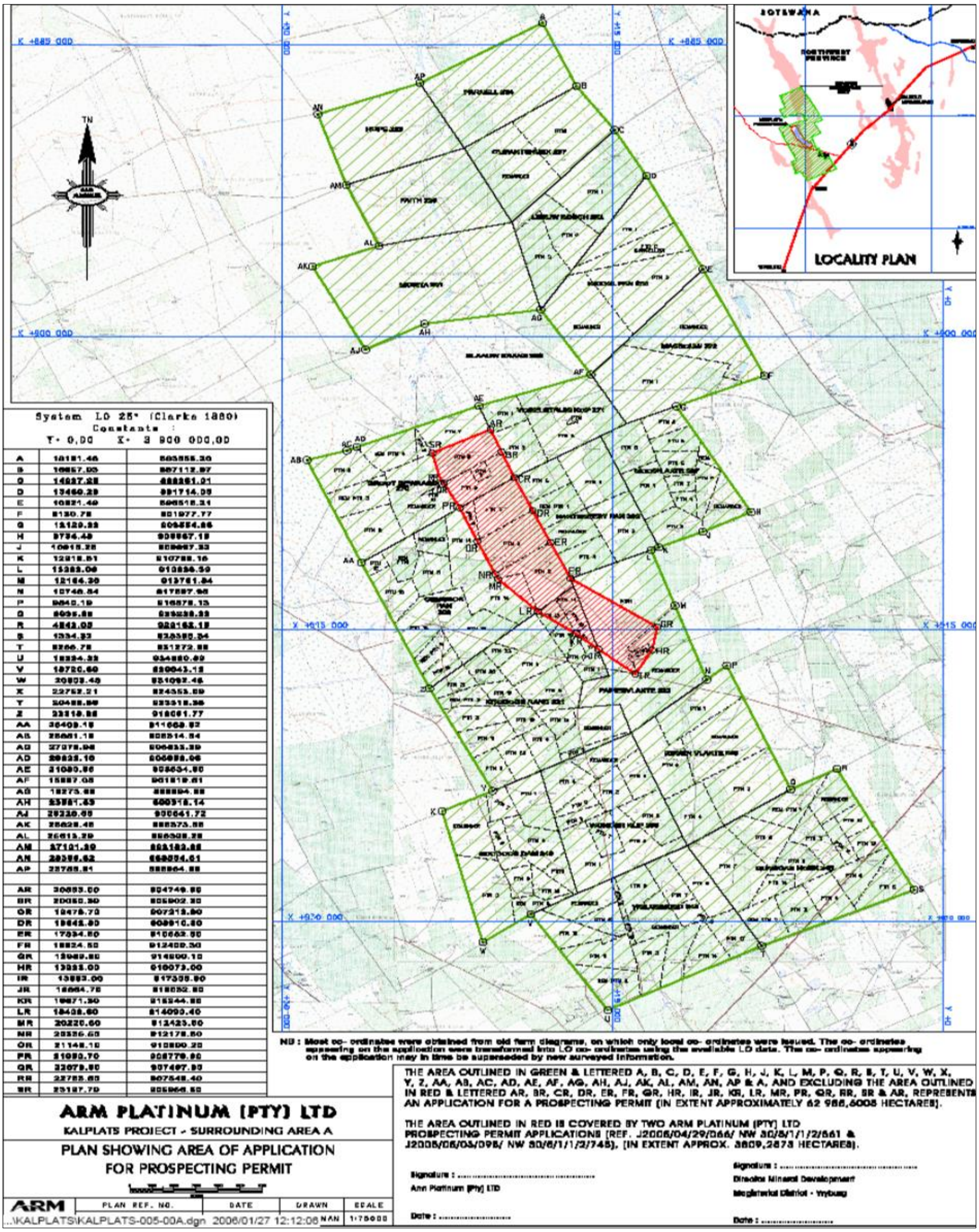


Figure 2-2: Kalplats Project Prospecting Application

3. CLIMATE

The project area falls within the summer rainfall region of South Africa with an average annual rainfall of 500 mm. Summers are warm to hot and winters cold. Temperatures of 380 ºC are common during mid-summer and night temperatures can drop below 00 ºC in winder (May to August).

4. PROJECT HISTORY

The PGM potential of the area was recognised during gold focused exploration programs over the Kraaipan Greenstone Belt in the early and late 1990's carried out by Anglo American Prospecting Services (AAPS). Following the acquisition of the ground by Harmony Gold Mining Company Limited (Harmony), an extensive exploration programme was undertaken, during which some 40,000 m of drilling was completed. Further test work including metallurgical testing, geotechnical, mining and environmental studies and the mining of a box cut were also undertaken and incorporated into a 'Feasibility Study' which was completed in February 2004. The study focused on developing the project as an underground operation, mining a wide, low grade horizon. At the prevailing metal prices this was not considered a viable option.

In 2004 African Rainbow Minerals (ARM Platinum) acquired the Kalplats Project from Harmony. ARM Platinum and Platinum Australia entered into a joint venture in April 2005.

Platinum Australia completed a feasibility study in September 2010 which will form the basis of the current mine deign and economics update.

African Thunder Platinum Limited ("ATP") acquired certain South African PGM assets in March 2015. Part of this acquisition included 49% holding of the Kalplats Project. ATP has been funded by a consortium of experienced mining investors, Aberdeen International ("Aberdeen"), and Pala Investments ("Pala"). Aberdeen and Pala have become shareholders of ATP.

The joint venture Kalplats project partner is African Rainbow Minerals Platinum Pty, and ATP will be the operator.

5. GEOLOGY

Refer to 2010 DFS report.

6. GEOTECHNICAL DRILLING

Refer to 2010 DFS report.

7. MINING

Following the completion of Kalplats 2010 FS Study, DRA was commissioned by ATPL to update the open pit design and optimise the production schedule to 2020 terms. The same geology and geotechnical information from the 2010 Kalplats FS are used as the basis of design.

The project comprises eight known deposits (north to south) namely: Crater, Vela, Mira, Sirius, Orion, Serpens North, Serpens South and Crux that have measured and indicated resources. In addition to these deposits, three exploration targets known as Scorpio, Tucana and Pointer have been identified but did not have sufficient geological confidence to warrant consideration. All the deposits outcrop close to surface, are steeply dipping (+/- 70 degrees) and amenable to open pit mining. (Figure 7-1)

Based on historical work and current economics this update still considers seven of the deposits for which resource modelling has been completed by Coffey Mining Consultants. The upside potential from the Serpens South deposit and three further exploration targets remains to be assessed.

Based on a 1.5 Mtpa production rate, the mining life will be 11 years at an overall life 2E+Au (platinum, palladium and gold) grade of 2.92 g/t, including a pre-production period prior to the plant being commissioned. The overall mine life stripping ratio averages 7:1 if the mined low-grade ore is considered as waste material and not treated. In this study there are no plans to treat the low grade tonnage.

The study included Whittle runs on the deposits using optimised parameters and the pit designs and evaluation of the open pit mining are based on the optimised parameters. The estimates of operating costs, most of which were supplied by the four contractors, have also been considered.

An experienced mining contractor using conventional truck and shovel open pit mining methods will mine ore to satisfy the mill feed requirements of 1.5 Mtpa. An average 2E+Au grade of above 3.0 g/t will be obtained for the first six years of processing.

Open pit optimisation studies were carried out on Kalplats using the geological resource models provided by Coffey Mining Consultants.

The models were then validated to ensure that all the necessary fields were included.

The geological resource block models were not provided in a Whittle compatible format, the surrounding waste rocks were excluded from the model. Therefore, the surrounding waste blocks were created using Datamine™ block modelling processes.

The waste surrounding the ore was partitioned into blocks with dimensions that were similar in size to that of the ore blocks, i.e. 5 x 5 x 5 m (X, Y and Z).

To exclude air blocks in the optimisation process, the waste blocks limits were made bigger than the ore blocks limits.

The waste blocks were then also assigned identifying attributes before adding to the ore blocks.

The Datamine ADDMOD process was used to create the ore and waste models.

The combined block model was subjected to a Datamine process called PROMOD, basically to make sure that there were no blocks overlapping.

To ensure that there was no loss of ore blocks in these processes, the ore in the output block model was evaluated and the results were comparable to the geological resource statement.

Whittle optimisation processes were employed for the optimisation of Kalplats ore bodies.

After a series of runs were completed on all the ore bodies, an optimum pit shell was selected for each of the deposits.

Detailed push back designs were produced using DESWIK CAD to systematically extract the resource inventory contained within the shell envelope as efficiently as possible.

The pit designs were then interrogated and evaluated to assess the available ore and waste tonnage.

The production scheduling was then carried out using the DESWIK Sched scheduling programme.

Production ore summaries for the mining periods were then produced. A treatment schedule was drawn up to satisfy the mill feed requirements.

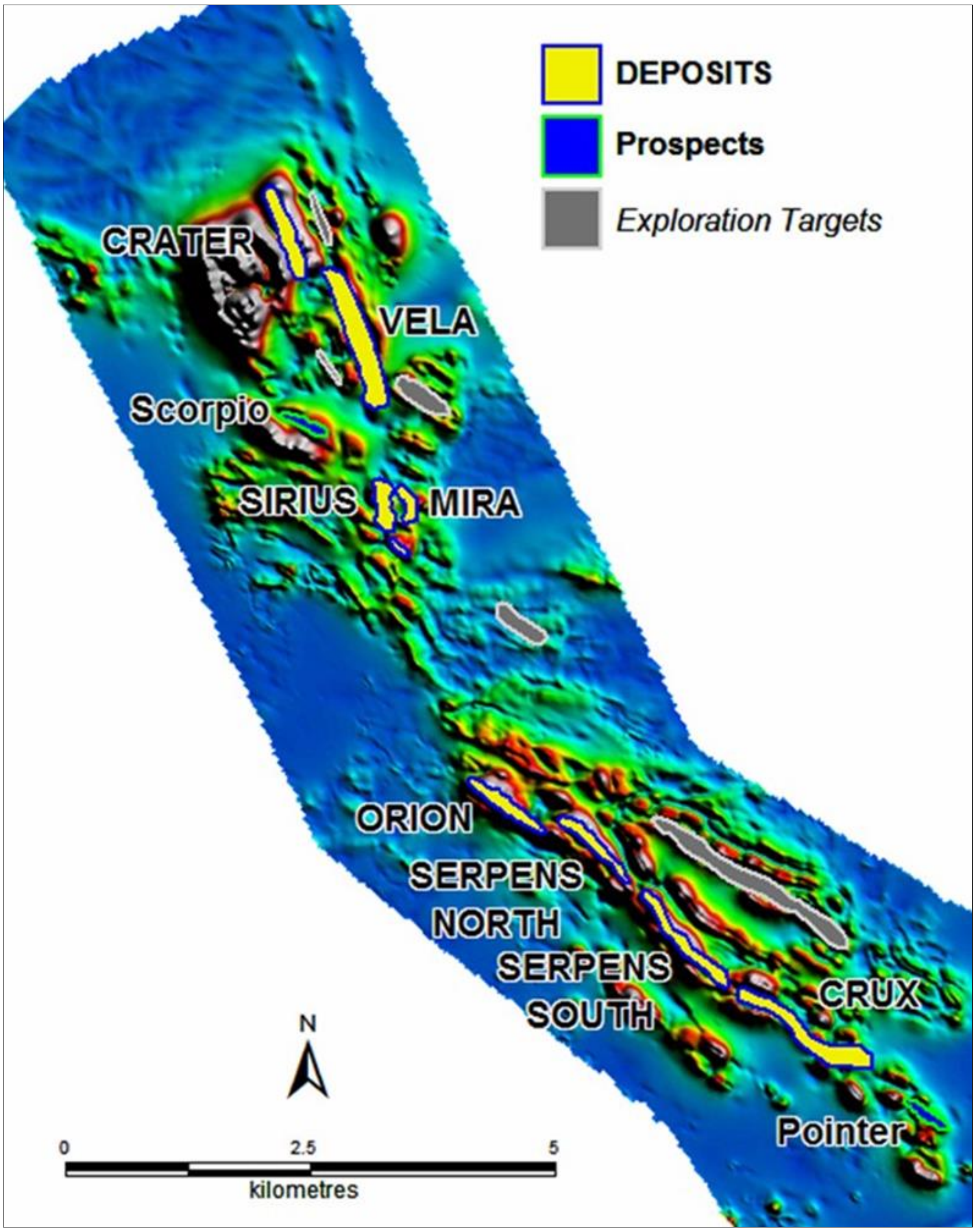


Figure 7-1: Plan of Main Open Pits Deposits Identified

7.1. Resource Summary

In the PFS the measured, indicated and inferred resources were included, however the inferred was excluded for this DFS update.

In order to improve the quality of feed to the plant an ore cut-off grade of 1.9g/t 2E+Au was applied to the mine design.

The exclusion of the inferred resources resulted in the reduction of pit depths. Table 7-1 is a summary of Kalplats resources.

Table 7-1: Kalplats Resource Summary

Deposit	Total Resource[1]			Main Reef Resource[2]			High Grade Resource[3]		
	Tonnes	Grade 2E+Au g/t	Ounces	'000's Tonnes	Grade 2E+Au g/t	'000's Ozs	'000's Tonnes	Grade 2E+Au g/t	'000's Ozs
Crater[5]	26,215,785	2.04	1,719,530	11,543	2.62	967	11,638[4]	3.21	1,202
Orion[5]	11,856,000	1.58	603,210	5,597	2.14	386	2,720	3.33	291
Crux[5]	29,100,000	1.38	1,286,000	13,132	1.67	708	5,229	2.59	436
Vela[6]	36,662,000	1.34	1,579,676	14,804	2.08	992	8,171	3.07	806
Sirius[5]	9,484,000	1.43	431,490	3,103	2.11	210	1,570	3.19	161
Serpens North[6]	7,703,000	1.43	353,950	3,199	1.93	199	1,269	3.20	131
Mira[6]	6,633,000	1.43	304,800	2,246	2.41	174	1,192	3.58	137
Serpens South[7]	10,762,228	1.34	462,071	5,890	1.81	324	848	5.09	139
Total	137,354,013	1.53	6,693,127	59,330	2.21	3,959	32,500	3.16	3,303

[1]Includes the high-grade UM (+UUM in Crater and Vela) and LM Reefs

[2]Includes the high-grade UM (+UUM in Crater and Vela) Main Reef Residual and LM that constitute the Main Reef

[3]Includes UM, UUM, LM, LG, MMW and the Main Reef Residual layers, which is the total mineralised width for all seven layers

[4]For Crater this includes the UUM, UM, LM and the high grade MR2 Reef

[5]Coffey Mineral Resource estimates of Measures, Indicated and Inferred resources

[6] Coffey Mineral Resource estimates of Indicated and Inferred resources

[7]Harmony Mineral Resource estimated of Inferred resources

7.2. Resource and Reserve

Various grade cut-offs where tested to optimise the mining production scheduling process. For previous iterations of study work, various cut-off grades were employed at the different resource areas. For this study it was considered, for the sake of transparency, to employ a single cut-off grade for all mining areas. Various scenarios were considered. A mining cut-off grade of 1.5g/t 2E+Au yields a head grade of around 2.68g/t 2E+Au, whereas a 1.9g/t cut-off yields a 2.92g/t head grade.

Table 7-2 and Figure 7-2 illustrates the calculation of comparative differential revenue gains for the selective scenarios:

Table 7-2: Kalplats Grade Cut-off Comparison

	1.9 g/t 2E+AU	1.8 g/t 2E+ AU	1.7 g/t 2E+ AU	1.6 g/t 2E+AU	1.5 g/t 2E+ AU	Grade	Prill Split	ZAR/g net
Ore Tonnes	17 701 896	18 585 952	19 505 603	20 351 993	21 415 241	1.37	47%	347.04
Grade (2E + AU)	2.92	2.85	2.80	2.70	2.68	1.43	49%	511.43
LoM (y)	11.80	12.39	13.00	13.57	14.28	0.12	4%	493.16
Waste Tonnes	124 133 416	123 249 360	122 329 709	121 483 319	120 420 071	2.92	100%	433.55
S/R	7.01	6.63	6.27	5.97	5.62			
Total t	141 835 312	141 835 312	141 835 312	141 835 312	141 835 312			
Revenue / ore t	883.14	864.93	849.76	819.41	813.34			
Add Revenue	69.80	51.59	36.42	6.07				
Add Mining Cost	40.64	29.49	18.97	10.12				
Diff Revenue Gain	29.16	22.10	17.45	- 4.05				

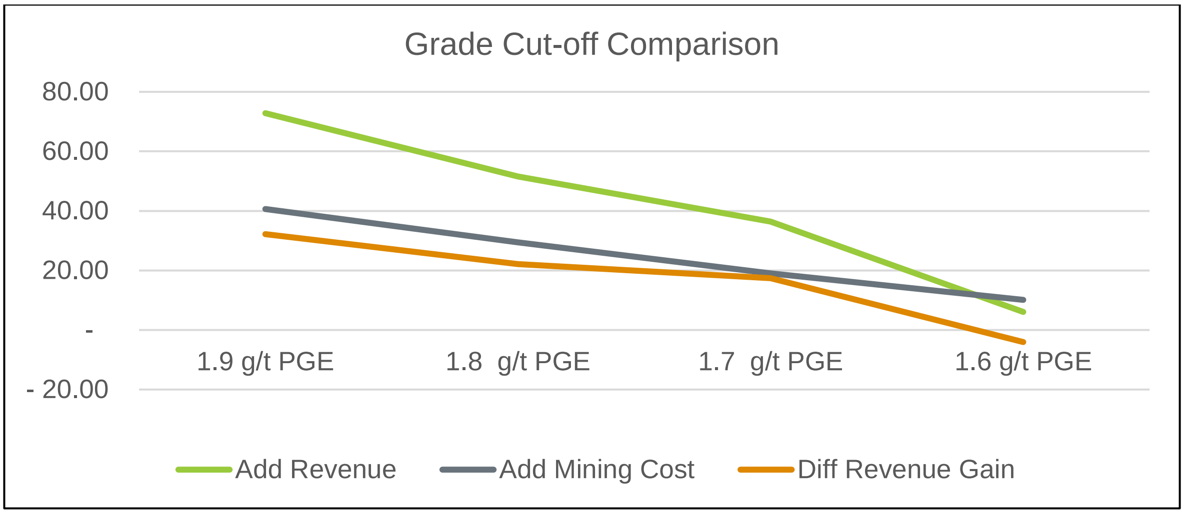


Figure 7-2: Grade Cut-off Comparison

Another consideration to take into account when considering the mining cut-off grade, is the duration of the mine life. As a benchmark, a mine life of less than ten years is not desirable.

When comparing the resultant differential revenue gains, the 1.9 g/t cut-off scenario yields the best differential revenue gain, a head grade of 2.92 g/t 2E+AU which is most favourable, as well as a LOM of around 11 years, also favourable. Based on this comparison a mining cut-off grade of 1.9 g/t 2E+AU was selected.

Refer to Table 7-3 for the comparison of resources to reserves (using a cut-off grade of 1.9 g/t 2E+Au)

Table 7-3: Kalplats Resource to Reserve Summary

Pit		Ore Tonnes (t)	2E+AU Grade (g/t)	Metal Content (ozt)	Pit Depth (m)
Crater	Resource	3,080,460	3.03	300,088	
	Reserve	3,064,546	2.84	279,818	90 - 95
Vela	Resource	6,721,904	3.18	687,243	
	Reserve	5,876,459	3.03	572,466	70 - 105
Orion	Resource	2,076,876	3.29	219,684	
	Reserve	1,481,341	3.08	146,689	110 - 125
Crux	Resource	5,338,385	2.76	473,707	
	Reserve	4,495,546	2.68	387,354	90 - 110
Serpens North	Resource	1,075,985	3.2	110,700	
	Reserve	708,567	2.88	65,609	75 - 100
Sirius	Resource	1,542,602	3.06	151,763	
	Reserve	1,516,050	2.98	145,252	70 - 80
Mira	Resource	563,096	3.51	63,545	
	Reserve	557,875	3.22	57,754	50
Total	Resource	20,399,308			
	Reserve	17,700,384			

7.3. Whittle Optimisation Input Parameters

Separate Whittle optimisations were undertaken for each of the seven deposits. The input parameters used in the Whittle pit optimisations are described in the

Table 7-4 below and cover a range of disciplines. Metal prices, processing costs, administration cost, and processing recoveries were supplied by ATPL, mining costs were provided by four South African Mining Contractors, and the slope angles were recommended by Open House Mining Consulting.

Table 7-4: Kalplats Whittle Input Parameters

Pit Optimisation Input Data for All Mining Areas Completed in April 2020		
Note: All costs in ZAR or USD as stated		
ZAR / USD Exchange	14.41	January 2020
Metal price	$/ozt	ZAR/g
Platinum (Pt)	950	440.13
Palladium (Pd)	1400	648.61
Gold (Au)	1350	625.44
Royalties (%)	Royalties (%) Payable (NSR) %	ZAR/g net
Platinum (Pt)	5% & 83%	347.04
Palladium (Pd)	5% & 83%	511.43
Gold (Au)	5% & 83%	493.16

Pit Optimisation Input Data for All Mining Areas Completed in April 2020		
Mining Costs		
Waste Mining	Depth (m)	R/t
	0-20	21.95
	20+	29.25
	Depth Cost Incremental (USD/t/m)	0.115
Ore Mining	Deposit	R/t
All levels	0-20	25.70
	20+	49.69
Processing Costs	Units	R/t
Processing	R/t	166.43
Other Operating Costs		
Grade Control	R/t	1.2 (in ore cost)
Pit Dewatering	R/t	0.74 (in ore cost)
Owner Fixed	R/t	10.86
Whittle Schedule Parameters		
Ore Production Rate	Mtpa	1.5
Discount Rate	%	10
Exchange Rate	R/USD	14.41
Whittle Physical Parameters		
Pit Slopes Angles	Rock Type	Degrees
	Weathered	35
	Fresh	57
Mining Dilution		30 cm Skin
Mining Recovery	%	97
Process Recoveries	Old	New
Shallow Ore	67%	$Rec = 63.612(HG)^{0.087}-8$
Fresh Ore	79%	$Rec = 63.612(HG)^{0.087}$

NB! The pit optimisation parameters used above are conservative and effectively deplete the ore body to the levels of the measured and indicated resources available in most of the pits. During scheduling various grade cut-offs where applied to optimise the feed grade to the plant.

7.4. Pit Design Parameters

The general design parameters used were as follows:

- Geotechnical parameters as summarized below
- Operating Ramp Width and gradient
- Exit on the pit rims at agreeable locations

The weathered and fresh face angles were battered at 45º and 80º while maintaining their berm widths at 5.7 m and 4m respectively. The overall pit slope angles were therefore 35 degrees for the weathered zone and 57 degrees for the fresh zone based on the recommendation by OHMS. The ramp gradient varies between 8% and 10% and ramp widths are between 20 m and 8 m. Surface haul road widths are 24 m. Table 7-5 below shows slope angle configurations followed to design Kalplats pits.

The following table shows the geotechnical parameters, as recommended by OHMS, used to design the pits for each deposit on the pit shells identified by the Whittle optimisations.

Table 7-5: Pit Design Parameters

Pit Design Parameters						
Material	Batter Angle (deg)	Batter Height (m)	Berm Width (m)		Ramp Gradient (%)	Ramp Width (m)
Weathered	45	10	5.7	Min	8	8
Fresh	80	10	4	Max	10	20

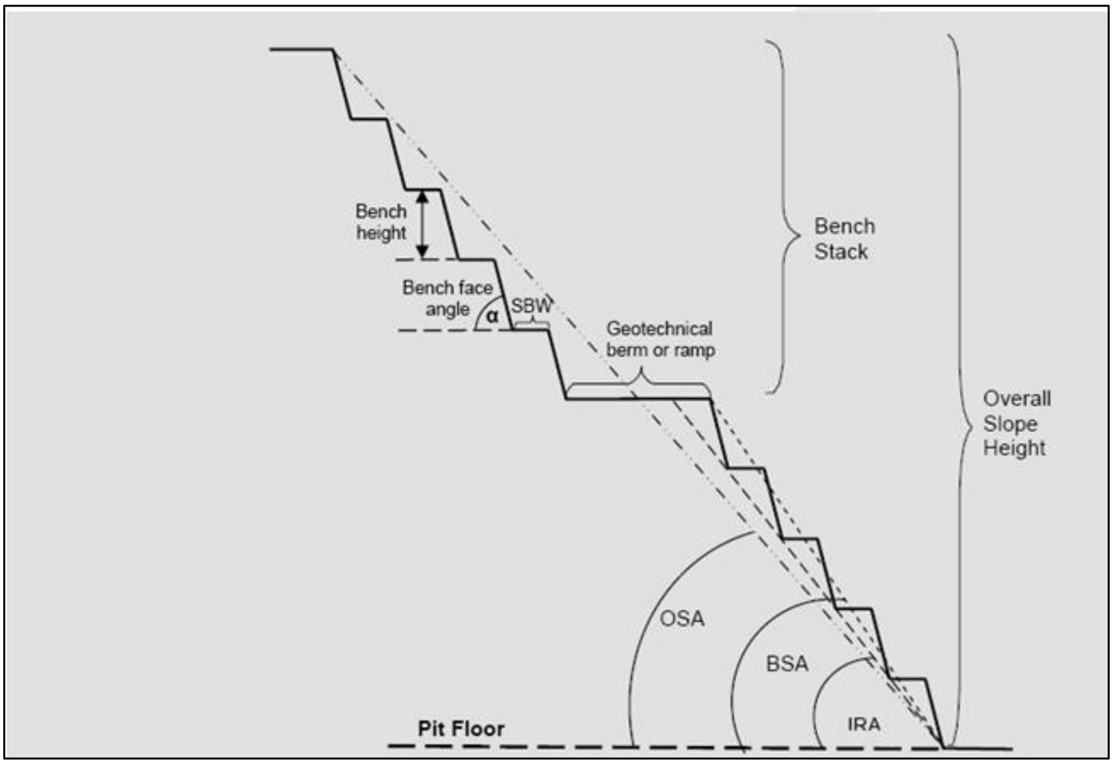


Figure 7-3: Pit Design Terminology

SBW – Spill Berm Width
OSA – Overall Slope Angle measured from toe to crest
BSA – Bench Stack Angle measured from toe to crest
IRA – Inter Ramp Angle measured from toe to toe

An overall pit slope angle was determined at 57 degrees for the fresh ore and 35 degrees for the weathered zone at a factor of safety of 1.5.

7.5. Waste Dump Design

7.5.1. Waste Dump Design Parameters

The dump design was done to ensure adequate capacity exists for waste near each pit. The design has taken local terrain into consideration. The top of the dumps will be graded and flattened so that the area may be useful for grazing purposes once rehabilitated. Table 7-6 show the parameters used in designing the waste dumps. The height, area and volume of each waste dump are shown in Table 7-7.

Table 7-6: Waste Dump Design Parameters

Batter Height	Batter Angle	Berm Width	Ramp Gradient
(m)	(deg)	(m)	(%)
10	22	10	8

Table 7-7: Waste Dump Design Specifications

Waste Dump For	Height (m)	Area (m^2)	Volume (m^3)
Crater, Vela, Sirius and Mira	20	1,500,000	30,000,000
Orion and Serpens North	25	700,000	17,500,000
Crux	20	690,000	13,800,000

7.5.2. Waste Dump Locations

The waste and low-grade dumps are positioned on the west side of the strike of the Kalplats deposits to avoid sterilization of any potential ore resource on the eastern side. Where the dumps have free standing faces, they will be battered back to stable angles of approximately 19º.

The location of the waste dumps in relation to each pit is shown Figure 7-4. Table 7-8 shows the distances from the exits of the ramp to the base of the waste dumps.

Table 7-8: Distances from the Pit Ramp Exit to the Base of the Waste Dump

Pit	Pushback	Distance (m)
Crater	1	500
	2	480
Vela	1	650
	2	780
	3	580
	4	890
	5	640
	6	700
Orion	1	350

Pit	Pushback	Distance (m)
	2	250
Crux	1	280
	2	280
	3	220
	4	220
	5	650
Serpens North	1	900
	2	500
Sirius	1	1,250
	2	1,250
Mira	1	1,280
	2	1,500

The resulting final pit designs for each of the original seven deposits using geotechnical and haul road parameters as defined above are discussed and shown in the figures below. The pit designs were designed from bottom up using the Whittle optimal pit shells for each mining area. Haul road access is provided to all areas within the mining area.

7.6. Pit Designs

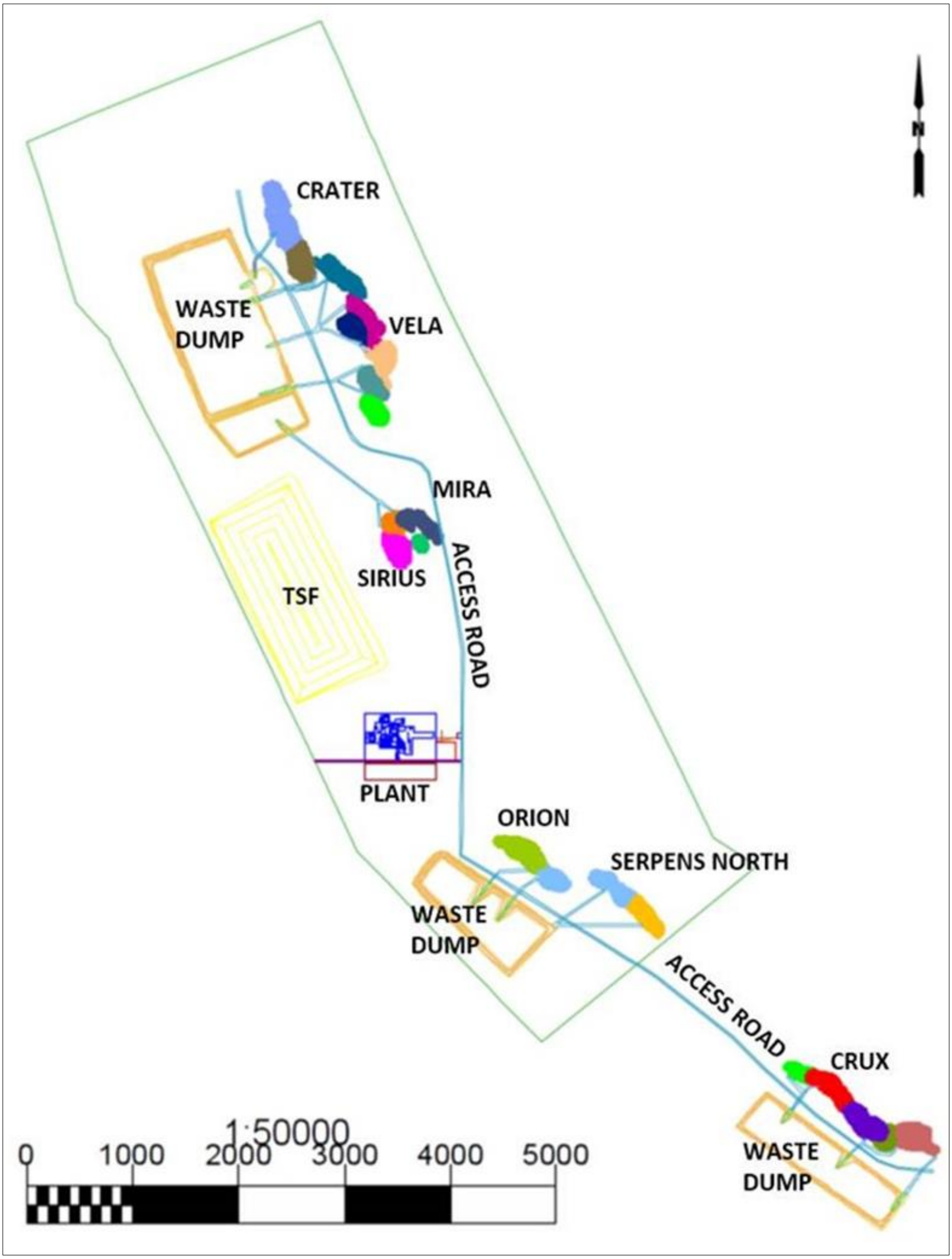


Figure 7-4: Mining Areas, Haul Roads, WRD locations and other Surface Infrastructure

7.7. Crater Pits Design

- Crater Pushback 1 is expected to be 720 m length and approximately 210 m in width at the rim. The pit width at base is 20 m

- Crater Pushback 2 is expected to be 400 m long and approximately 210 m in width at the rim. The pit width at this point is expected to be about 20 m, refer to Figure 7-5

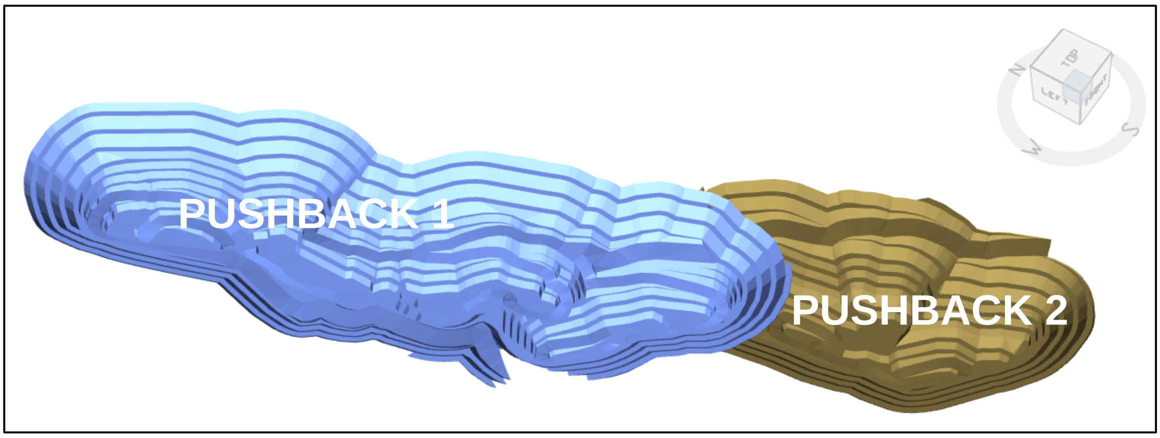


Figure 7-5: Crater Pits Design and Pushback Layout

7.8. Vela Pit Design

- Vela Pushback 1 is expected to be 550 m length and approximately 180 m in width at the rim. The pit width at this deepest point is expected to be about 11 m
- Vela Pushback 2 is expected to be 600 m length and approximately 160 m in width. The pit width at this deepest point is expected to be about 11 m
- Vela Pushback 3 is expected to be 320 m length and approximately 210 m in width. The pit width at the deepest point is expected to be about 15 m
- Vela Pushback 4 is expected to be 370m length and approximately 195 m in width. The pit width at the deepest point is expected to be about 11 m
- Vela Pushback 5 is expected to be 320 m length and approximately 200m wide. The pit width at the deepest point is expected to be about 15 m
- Vela Pushback 6 is expected to be 440 m length and approximately 200 m in width at the rim. The pit width at the deepest point is expected to be about 10 m. Refer to Figure 7-6.

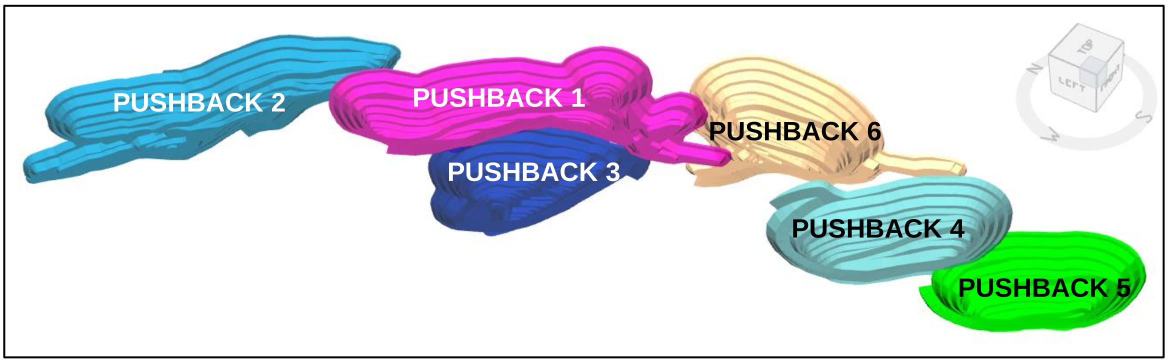


Figure 7-6: Vela Pit Design

7.9. Orion Pit Design

Orion Pushback 1 is expected to be 600 m long and approximately 200 m in width at the rim. The highest point on the pit rim is expected to be approximately 1,275 m RL with the deepest point being 1,155 m RL. The pit width at this point is expected to be about 10 m.

Orion Pushback 2 is the final Pushback and is expected to be 310 m long and approximately 200 m in width at the rim. The highest point on the pit rim is expected to be approximately 1,275 m RL with the deepest point being 1,175 m RL. The pit width at the deepest point is expected to be 10 m. Refer to Figure 7-7.

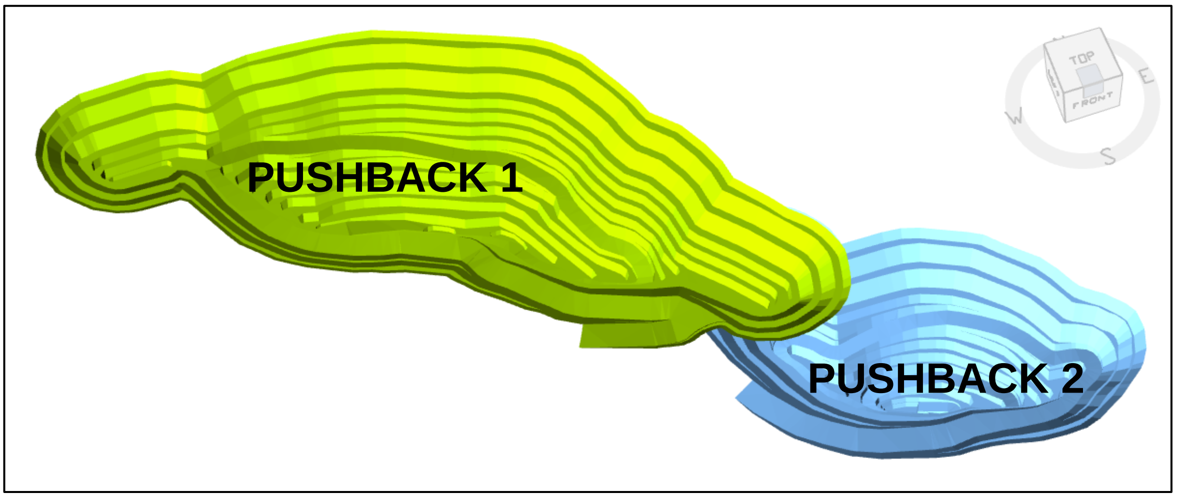


Figure 7-7: Orion Pit Designs and Pushback Layout

7.10. Crux Pit Design

- Crux Pushback 1 is expected to be 540 m length and approximately 190 m in width at the rim. The pit width at this deepest point is expected to be about 10 m
- Crux Pushback 2 is expected to be 450 m length and approximately 195 m in width at the rim. The pit width at the deepest point is expected to be maintained at 10 m
- Crux Pushback 3 is expected to be 510 m length and approximately 260 m in width at the rim. The pit width at this point is expected to be maintained at 35 m
- Crux Pushback 4 is conically shaped and expected to be 250 m length and in width at the rim. The pit width at the deepest point is expected to be about 10 m
- Crux Pushback 5 is expected to be 310 m length and approximately 130 m in width at the rim. The pit width at the deepest point is expected to be about 10 m. Refer to Figure 7-8

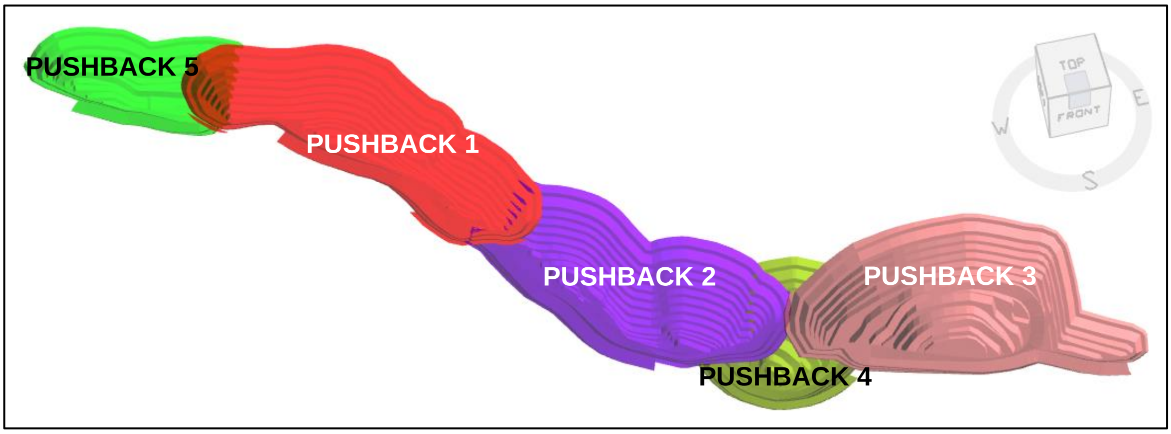


Figure 7-8: Crux Pit Designs

7.11. Serpens North Pit Design

Serpens North Pushback 1 is expected to be 510 m length and approximately 160 m in width at the rim. The pit width at this deepest point is expected to be about 10 m.

Serpens North Pushback 2 is expected to be 450 m length and approximately 180 m in width at the rim. Refer to Figure 7-9.

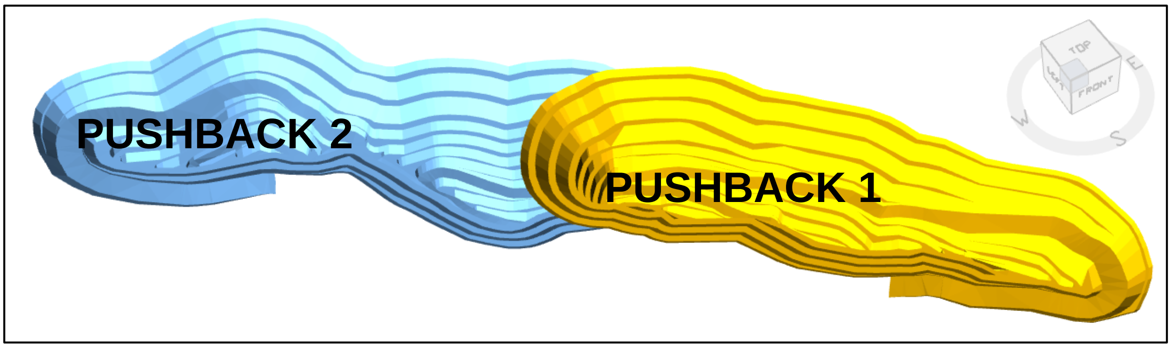


Figure 7-9: Serpens Pit Designs and Pushback Layout

7.12. Sirius Pit Design

- Sirius Pushback 1 is expected to be 390 m length and approximately 245 m in width at the rim. The pit width at this deepest point is expected to be about 20 m
- Sirius Pushback 2 is conically shaped and expected to be 240 m in length and in width at the rim. The pit width at depth is expected to be 20 m. Refer to Figure 7-10

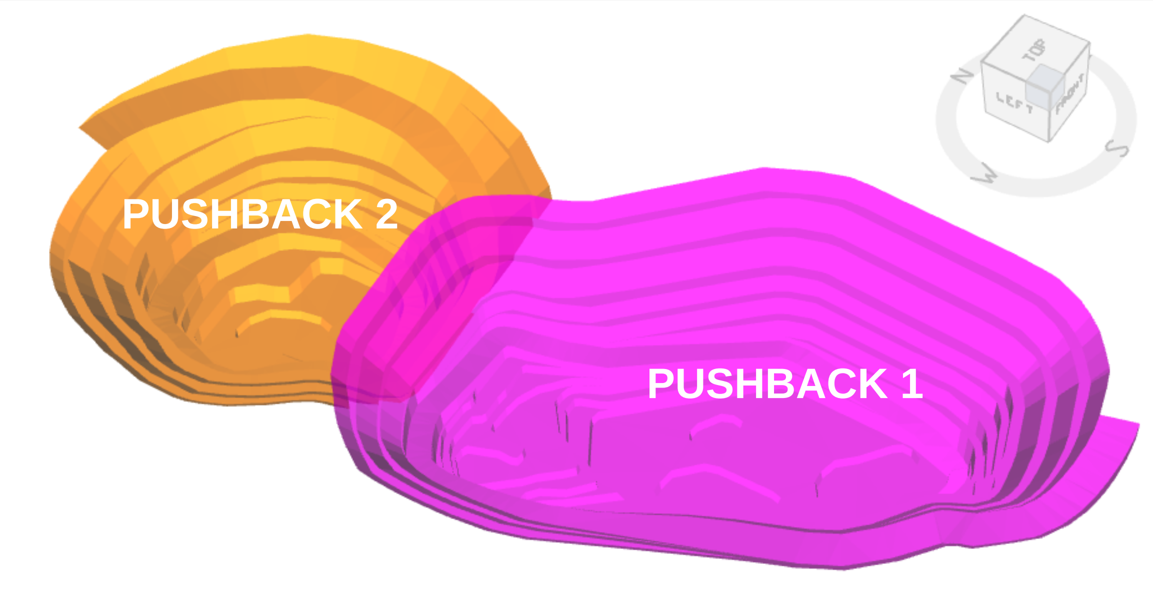


Figure 7-10: Sirus Pit Designs and Pushback Layout

7.13. Mira Pit Design

- Mira pushback 1 is expected to be 480 m in length and approximately 150 m in width at the rim. The highest point on the pit rim is expected to be approximately 1,255 m RL with the deepest point being 1,205 m RL. The pit width at this deepest point is expected to be about 10 m.
- Mira pushback 2 is expected to be 180 m length and approximately 130 m in width at the rim. The pit width at the deepest point is expected to be maintained at 10 m. Refer to Figure 7-11.

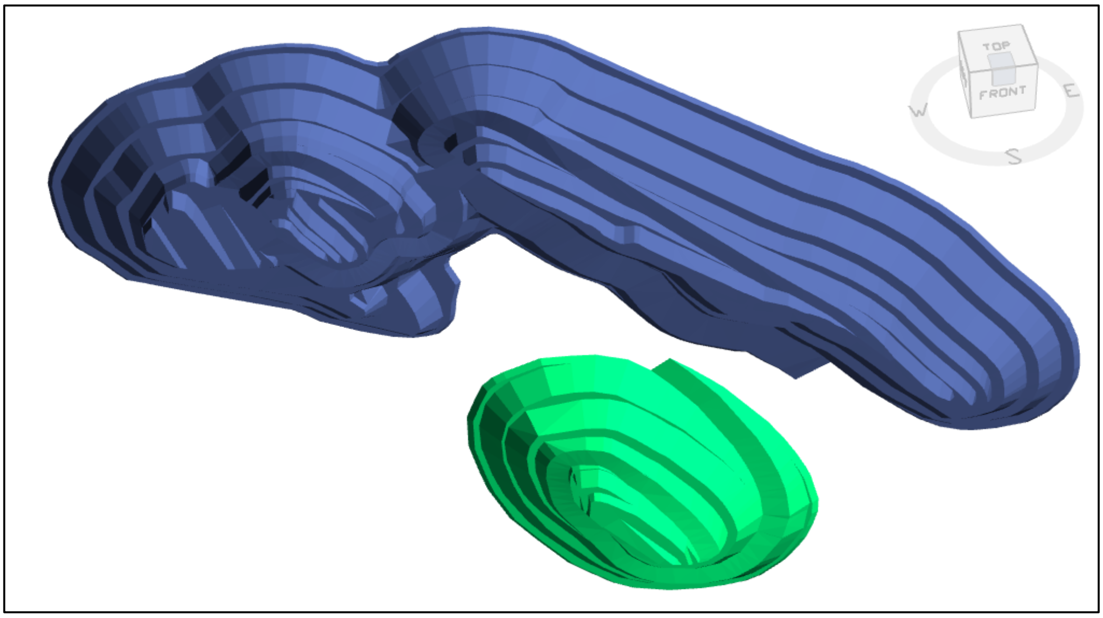


Figure 7-11: Mira Pit Designs and Pushback Layout

The final open pits were designed to extract tonnages and grades as outlined below.

Table 7-9: Per Pit Scheduled Reef and Waste Tonnages

Mining Area		Plant Feed Ore Grade Material >=1.9 g/t 2E+AU				Stockpile Ore Low-Grade Material >=1.0 And <1.9 g/t 2E+AU				Waste Material	
Pit		Ore (T)	Pt (g/t)	Pd (g/t)	Au (g/t)	LG Ore (T)	Pt (g/t)	Pd (g/t)	Au (g/t)	Waste (T)	Strip Ratio
Crater	1	2,215,857	1.10	1.65	0.10	1,942,332	0.53	0.54	0.03	13,283,305	6.87
	2	848,690	1.18	1.53	0.11	602,584	0.53	0.54	0.04	6,385,376	8.23
	Total	3,064,546	1.12	1.62	0.10	2,544,915	0.53	0.54	0.03	19,668,681	7.25
Vela	1	879,141	1.93	1.69	0.20	249,791	0.50	0.55	0.01	5,347,213	6.37
	2	1,421,618	1.50	1.28	0.16	293,224	0.49	0.58	0.02	4,462,618	3.35
	3	1,300,452	1.41	1.20	0.08	320,047	0.61	0.48	0.01	3,138,109	2.66
	4	1,112,967	1.28	1.39	0.02	366,897	0.55	0.53	0.02	4,836,131	4.67
	5	564,791	1.71	1.55	0.16	82,479	0.57	0.52	0.02	4,429,285	7.99
	6	597,490	1.72	1.57	0.09	533,362	0.58	0.56	0.01	4,667,531	8.70
	Total	5,876,459	1.55	1.40	0.11	1,845,800	0.55	0.54	0.01	26,880,887	4.89
Orion	1	1,026,359	1.52	1.38	0.17	1,512,074	0.46	0.59	0.02	7,955,619	9.22
	2	454,981	1.50	1.49	0.13	386,174	0.51	0.61	0.04	3,356,478	8.23
	Total	1,481,341	1.51	1.41	0.16	1,898,247	0.47	0.59	0.03	11,312,097	8.92
Crux	1	888,025	1.96	1.74	0.18	880,134	0.45	0.51	0.10	7 310,979	9.22
	2	1,666,819	0.92	1.34	0.07	5,183,790	0.49	0.70	0.04	7,140,820	7.39
	3	1,218,045	1.09	1.22	0.08	2,152,756	0.53	0.65	0.04	7,106,207	7.60
	4	477,440	1.11	1.14	0.10	769,831	0.57	0.69	0.06	2,585,239	7.03
	5	245,216	1.43	1.09	0.25	221,419	0.50	0.71	0.07	2,630,538	11.63
	Total	4,495,546	1.22	1.35	0.11	9,207,930	0.06	0.07	0.01	26,773,784	8.00
Serpens North	1	396,055	1.51	1.38	0.11	600,209	0.43	0.51	0.05	4,366,770	12.54
	2	312,512	1.37	1.27	0.12	561,894	0.44	0.54	0.03	4,188,873	15.20
	Total	708,567	1.45	1.33	0.11	1,162,103	0.43	0.53	0.04	8,555,643	13.71

Mining Area		Plant Feed Ore Grade Material >=1.9 g/t 2E+AU				Stockpile Ore Low-Grade Material >=1.0 And <1.9 g/t 2E+AU				Waste Material	
Sirius	1	1,241,129	1.44	1.36	0.17	1,366,442	0.44	0.72	0.04	5,411,099	5.46
	2	274,921	1.41	1.57	0.09	724,269	0.43	0.69	0.02	2,693,414	12.43
	Total	1,516,050	1.43	1.40	0.16	2,090,711	0.44	0.71	0.04	8,104,513	6.72
Mira	1	454,597	1.61	1.56	0.14	784,962	0.45	0.54	0.03	2,462,386	7.14
	2	103,278	1.29	1.57	0.03	214,678	0.45	0.56	0.01	627,591	8.16
	Total	557,875	1.55	1.56	0.12	999,640	0.45	0.55	0.03	3,089,977	7.33
TOTAL		**17,700,385**	**1.37**	**1.43**	**0.12**	**19,749,347**	**0.29**	**0.35**	**0.02**	**104,385,581**	**7.01**

7.14. Mining Methodology

The mining operation will be a typical contractor operated open pit truck and shovel, load and haul mining method which requires drilling and blasting of the ore and/or hard overburden below the weathered zone expected to be the top 20 m.

The mining sequence consists of bush leering (where required), clearing and removal of topsoil and stockpiling for use in rehabilitation at the end of the mine life. The removal of overburden is a combination of ripping, dozer push to heaps and loading and hauling to the waste dump. Drilling and blasting will be required to remove the hard overburden.

Ongoing reef zone identification, grade control drilling and logging, ahead of the mining will be an integral part of the mining process. In order to ensure that the ore grade being fed into the plant stays within the planned feed grade limits. Typical mining bench heights of 5 m will be employed since grade control is critical. Due to the number of reefs of variable grade and variable thickness grade control activities will be critical to the project's success.

7.15. Life of Mine Schedule

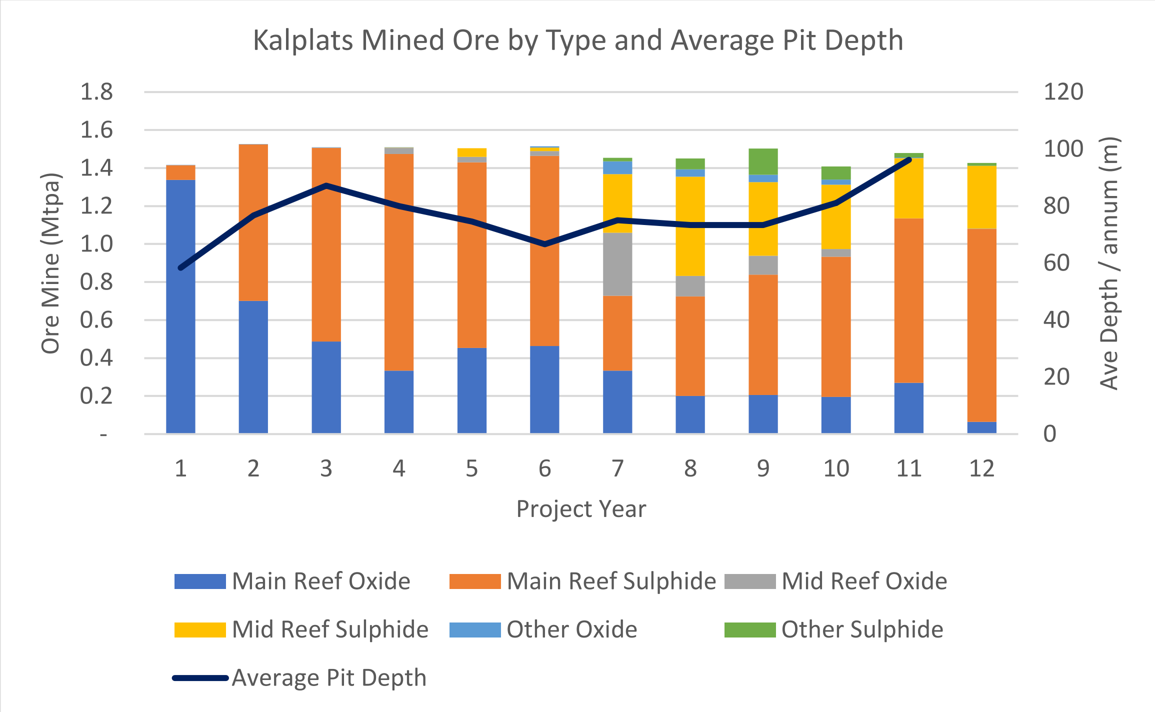
A Life of Mine (LoM) schedule for the Kalplats project was created using Surpac MineSched scheduling programme. Estimated mineable tonnages summaries for each mining period were then produced.

For the Kalplats Project several key parameters were used as constraints to achieve the LoM mining schedule.

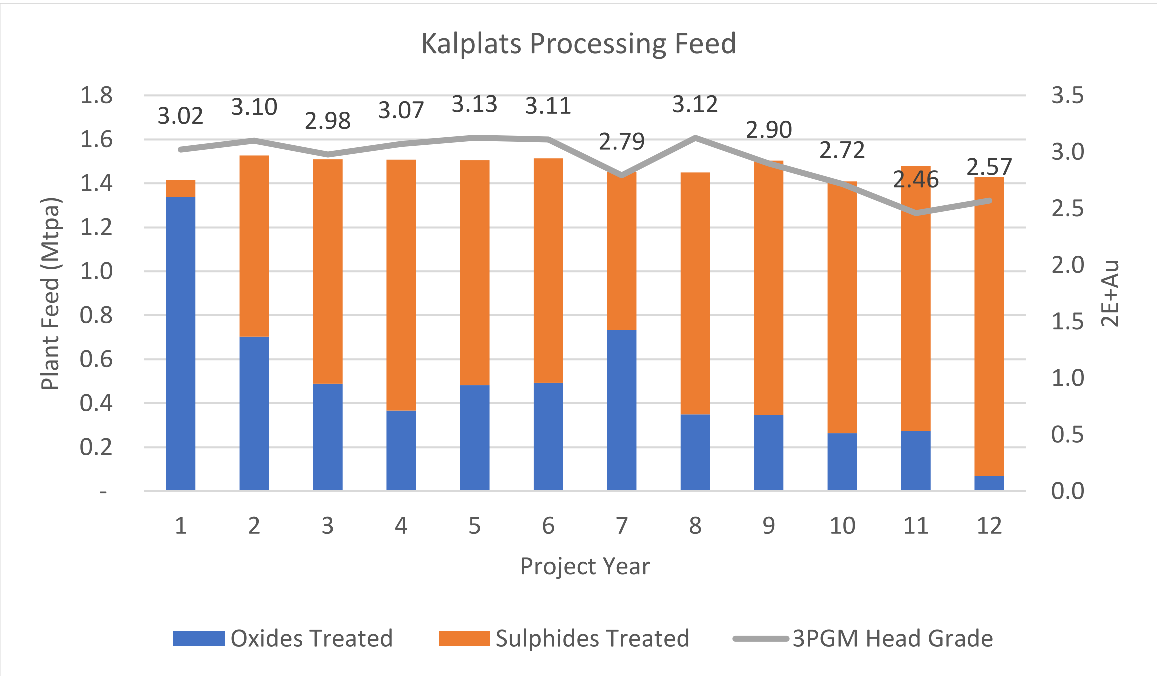
These can be summarised as follows:

- 1.5 Mtpa of high-grade ore to be supplied to the plant
- Minimise stockpiling and unnecessary re-handling costs
- Some of the additional conditions applied to the schedule are:
    - A pre-strip period of three months to allow enough plant feed stocks to be built up to satisfy the ore feed requirements to the processing plant upon start up
- The Plant production ramp up profile is assumed to be post pre-strip period:
    - 50% of plant nameplate capacity processed for the first month
    - 75% of plant nameplate capacity on the second month
    - 125,000 tpm full capacity thereafter

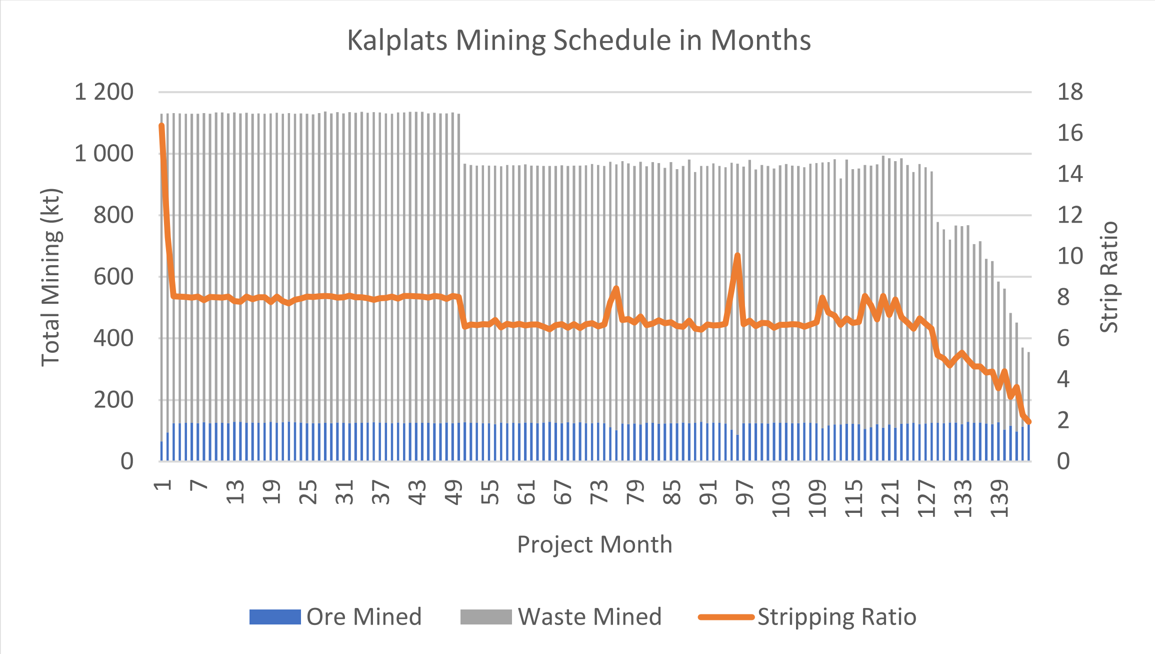
The following tables and figures illustrate the scheduling results for both mining and processing.



Graph 7-1: Graph indicating Mined Ore Tonnes by Type, as well as Average Pit Depth



Graph 7-2: Graph Depicting Oxide vs. Sulphide Ore Tonnage Mined



Graph 7-3: Graph of Monthly Ore vs. Waste Mining Profile and Stripping Ratio

7.16. Grade Control

Rigorous grade control practices with close sampling techniques will be adopted. Selective mining techniques will be employed to ensure the ore is carefully extracted and ore dilution minimised. Selective mining techniques require sequential activities to be executed on a cyclical basis. The cyclical basis includes the following:

- Grade Control Sampling
- Grade Control mark up
- Excavation
- Floor cleans and surveys controls

The ore excavation will be carried out in a highly regulated manner and supervised by the grade control team on site. Grade control sampling practise will also follow historically proven methods.

Drilling should be completed at least one week ahead of mining, to allow adequate time to generate the grade control model and design digging ore blocks.

A conventional grade control modelling software package, such as Datamine, Surpac or Micromine should be used to model ore blocks using the results of the grade control drilling data and the historical resource drilling data.

Staking will be required to assist in minimising dilution by defining the waste/ore contact using coloured tapes.

The pit geologist is to map the excavated area and should commence reconciliation by comparing the ore blocks “as designed’ against ore blocks ‘as mined’.

The intention is that only the identified high-grade mineralised material be directly tipped into the ROM crusher, with the remainder placed on the low-grade stockpiles or the waste dumps as per the grade categories, such as low grade, medium grade, and high grade.

The reconciliation process will include reconciliation of the grade control model with the estimated resource block model.

7.17. Equipment Selection – Primary and Secondary Mining Equipment

The operational opencast mining will be provided by a mining contractor. The service will include de-vegetation, topsoil stripping, removal of overburden to the waste dumps, removal of ore and delivery to the Plant or RoM stockpile.

All the primary and secondary equipment will be provided by the contractor as summarised in Table 7-10.

Table 7-10: Kalplats Equipment Selection

Type	Make	Model	Quantity
Excavator	120 t + 80 t	390 + 374	2 + 2
Trucks / Haulers	Cat / eq	773	12
Trucks / Haulers	Volvo / ep	40 t ADT	12
Dozers	Cat / eq	D9 & D8	2 + 1
Front End Loaders	Cat / eq	988	2
Graders	Cat / eq	16M	2
Water Trucks	Volvo / eq	40t	2
Mobile Compressors	750CFM	750CFM	1
Drill Rigs			2
Diesel Tanker / Service Truck			1

8. METALLURGY

8.1. Introduction

Test work on the Kalplats orebody has been conducted under the management and supervision of various consultants and has been concluded at several testing facilities around the globe. Broadly categorised, the following is a summary of work concluded:

- C.1991: Anglo American conducted test work at Mintek (although referenced in the Harmony PFS Report (2004), no results, data or reports have been shared with DRA)
- 2001 – 2005: Harmony concluded a wide range of bench-scale and pilot scale test work at SGS Lakefield and Mintek. Work was mainly conducted on drill cores from whole zones in Crater areas, followed by the Main Reef zone of Orion. Testing of a large Main Reef bulk sample, taken from the excavation (box cut) of the Crater deposit was used for the pilot campaigns. Additional tests were conducted on near-surface weathered material, whilst drill cores representing high grade samples were also tested in the event that high-grade zones (Upper Main, Lower Main and Mid Reef) could be selectively mined
- 2005 – 2009: Platinum Australia concluded additional test work at Mintek and SGS Lakefield in a bid to simplify the flowsheet developed by Harmony in the PFS
- 2010: Platinum Australia, under the supervision of DRA, conducted flotation and grindmill test work (on drill core samples from the Lower Main Reef and Upper Main Reef Zones) aimed at supporting the revised simplified flowsheet developed by Platinum Australia

Test work results and detailed analysis thereof is not provided in this updated study report, although aspects relating to test work are discussed with reference to recovery correlations, based on all test work concluded to date.

8.2. Kalplats Background

The Kalplats deposit consists of six discreet areas, which have been determined to have different reef packages (low grade, mid reef and main reef). The size and occurrence vary between the areas. Most test work concluded to date has been conducted using samples from the Main Reef of the Crater deposit.

Kalplats ore, is reported to be bi-modal containing silicates and a large magnetic fraction of magnetite, titaniferous and vanadiferous minerals. Magnetics account for 30% of the ore, whilst containing little copper, little sulphides and negligible amounts of nickel. The PGM's are very fine grained (<5 µm) and occur as Pt-Fe alloys and are typically associated with silicates. It is reported that a small proportion of PGMs is associated with magnetite.

Initial test revealed, that unlike UG2 ore, Kalplats PGMs do not benefit from a primary coarse grind. PGM liberation is only achieved at grinds finer than 80% passing 75 µm to both the silicate (non-magnetic) and magnetic fractions.

Table 8-1: PGM Distribution for Platinum Bearing Reefs [Source: Harmony, 2004]

	Liberated PGM's; sulphides and Pt-Fe alloys	PGM's locked in silicates*	PGM's locked in chromitite	PGM's locked in magnetics*	Non floating PGM's
Merensky	80 – 90	0 – 15	<1	<1	5 – 10
UG2	70 – 80	10 – 25	2 – 5	<1	5 – 10
Platreef	77 – 87	0 – 13	<1	<1	10 - 15
Kalplats	55 - 65	15 - 30	<1	8 - 12	5 - 10
* About ⅓ to ½ of PGM's in silicates can be recovered via fine grinding					

Table 8-2: Recovery of PGM's from Platinum Bearing Reefs [Source: Harmony, 2004]

	Liberated PGM's; sulphides and Pt-Fe alloys	PGM's locked in silicates*	PGM's locked in chromitite	PGM's locked in magnetics*	Total PGM Recovery
Merensky	83 - 87	5	0	0	88 - 92
UG2	73 – 77	12	0	0	85 - 89
Platreef	78 – 82	4	0	0	80 - 86
Kalplats	60 - 65	8 - 10	0	4	72 - 79

This shows the difference in expected total recoverable PGM's in the ore. The low Pt-Fe alloy content of Kalplats ore, which is easily liberated, is expected to result in PGM recovery challenges. This as a result of Kalplats ore needing a finer comparative grind to liberate a small percentage of PGM's, whilst having around 8 - 12% magnetite associated PGM's (which have very low recoverable PGM content).

8.3. Pre-Concentration

At a typical mill feed size (down to 6 mm) the PGM's, copper, Fe_2O_3, TiO_2 and V_2O_5 are evenly distributed with size. Test work conducted by Harmony highlighted that the PGM's are also evenly distributed between magnetic and non-magnetic fractions in each class. The liberation size is extremely fine, thus requiring fine milling to ensure that PGMs are liberated from the magnetic fractions.

Although the ore body contains a significant quantity of magnetite, test work conducted by Harmony showed that there was no economic benefit in pre-concentration or removal of the magnetic fraction by magnetic separation. Alternative pre-concentration techniques (inclusive of size scalping and density separation) were, also found, not appropriate to the ore due to the extremely fine nature of the PGM mineralisation within the ore body.

9. PROCESS PLANT

9.1. Process Flowsheet

Saleable PGM containing concentrates can be produced from Kalplats ore by a process of grinding followed by froth flotation. Fine grinding is required to maximise the performance of the flotation process.

Based on test work conducted during the Anglo American ownership period (±1991) and test work managed by Harmony, commencing 2001, the initial flowsheet for the project comprised of a primary mill-float, followed by a secondary mill-float. A target final grind of 80% passing 38 µm was selected. Concentrate thickening and filtration was proposed to produce a saleable filtercake for third-party smelting, with conventional slurry tailings disposal.

The original MF2 flowsheet is traditionally used in the South African mining industry for UG2 type ore processing to avoid increased entrainment of chrome in the final concentrate due to overgrinding. Following the conclusion of additional test work at Mintek and SGS Lakefield, (2005 – 2009) it was demonstrated that in the instance of Kalplats, increased liberation does not result in the recovery of undesired components to the final concentrate (with the possible exception of magnetite).

In addition, the results largely highlight that comparable results are achieved between MF2 and MF1 configurations and that the final grind, not the circuit configuration, affects the achievable recovery and grade. As a result, the Platinum Australia confirmed that an MF2 circuit is not required and proposed a less complex MF1 circuit.

A Feasibility Study was concluded in 2010 by DRA, with additional milling trade-off studies and process optimisation investigations concluded in 2012 to supplement the 2010 Feasibility Study. During this period, additional optimisation bench scale test work on two composite samples (provided by Platinum Australia) was concluded. The FS study was based on an MMF1 configuration with three-stage crushing, followed by two-stage ball milling to achieve a target grind of 80% passing 38 µm. The updated study was based on using conventional technologies and did not consider a wide variety of emerging fine grinding technologies. Two stages of cleaner flotation were proposed with the primary roughers operating with a low slurry density of c.30% solids as directed by test work.

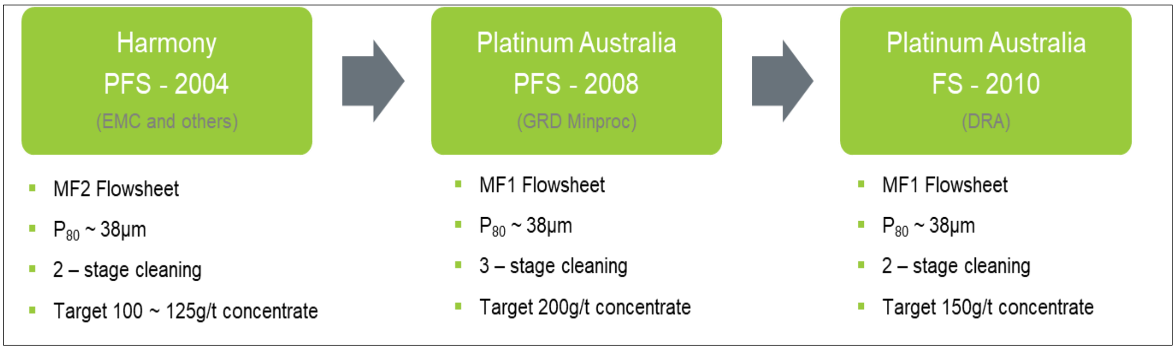


Figure 9-1: Summary of Flowsheet and Design Criteria Progression in Studies

9.5.1. Comminution Circuit

As discussed, the base-case Feasibility Study was based on using conventional ball milling for both the primary and secondary milling duties. During the time of concluding the Feasibility Study for Platinum Australia, emerging technologies such as HPGR fine crushing and numerous fine grind milling technologies, were not considered in detail for the project. This was driven by concerns relating to the proven viability of these technologies, effect of non-inert fine grinding media and desire to keep to a low complexity proven flowsheet.

However, a high-level supplementary investigation was conducted in 2012, to assess the benefit, if any, of replacing one of the secondary ball mills specified in the 2010 study with an IsaMill™. The investigation highlighted that replacing one of the two secondary 4.5 MW ball mills with an IsaMill™ is expected to provide the project with roughly a 10% reduction in milling power. However, based on preliminary estimates for the overall operating cost, inclusive of grinding media and maintenance on the IsaMill™ unit, the operating cost for processing was expected to increase by around 15%. In addition, an increase of around ZAR 34M in process capital (2012 base date) was estimated for the inclusion of the IsaMill™.

For the current economic cost update concluded for ATPL, the use of a Vertimill™ has been modelled for the secondary ball milling duty. Again, at concept level, it is expected that the Vertimill™ will provide significant energy efficiency (±15%) over ball milling, although an increase in overall operating costs, estimated at 10 - 15% higher, is expected due to the high costs associated with inert grinding media, liner replacement and other equipment costs.

For the current update, the 2010 Feasibility Study flowsheet, employing ball milling for the primary and secondary duties has been used. This due to the increase in cost (both operating and capital) expected with alternative technologies.

However, it is recommended that the trade-off studies be revisited based on data from vendor specific test work and detailed cost estimation, using 2020 power supply and consumable rates.

9.5.2. Recovery Models

Over the years, several test work campaigns have been concluded under the management of various owners, consultants and engineers. Test work has predominantly been conducted at Mintek and SGS Lakefield Laboratories.

The Harmony testwork was conducted using borehole samples from the Crater and Orion deposits, plus a bulk sample from trial mining (box-cut to 40 m depth) at Crater. The preliminary test work conducted by Platinum Australia was done on samples from the Crater box-cut whilst the later stages used a mini bulk sample composited from Serpens North boreholes, plus variability borehole samples from the Serpens North, Vela, Crux and Sirius Deposits.

The Harmony test work, on various circuit configurations, highlighted that recoveries (2E+Au) from 64% to 73% could be expected at final concentrate grades of between 60 g/t and 217 g/t. Mintek pilot plant test work achieved recoveries (2E+Au) of around 73% with final concentrate grades ranging between 71 g/t and 107 g/t (2E+Au).

During the Feasibility Study conducted by DRA in 2010, additional samples were provided to simulate the simplified MMF circuit. During this Feasibility Study, limited to no test work reports from previous study phases were shared with the team since it was argued that results obtained did not reflect the simplified MMF flowsheet.

For the 2010 test work, drill core samples were used comprising of a total of individual 243 samples, with a total mass of 876 kg. A detailed description of the exact location of the samples is not provided, although it was reported that 98 samples were from the LM type zone while 145 were from the UM type ore, with different pit areas represented, namely: Crux (75 samples), Mira (14 samples), Serpent North (25 samples), Sirius (93 samples), and Vela (36 samples). These samples were further grouped to represent different depths, namely: 10 – 20 m (44 samples), 20 – 30 m (73 samples), 30 – 40 m (50 samples), and 40 – 50 m (76 samples).

7.5.4.1 *Fresh Material (>50 m)*

Open circuit bench-scale flotation test work was conducted on two composite samples (LM and UM 40 - 50 m depth), followed by locked- cycle tests on the two composite samples. The tests were aimed at providing estimates of expected recovery for the simplified MF1 circuit. The samples had high reported head grades:

- LM ~ 3.5 g/t (2E+Au)

- UM ~ 4.8 g/t (2E+Au)

From the locked-cycle test work, final recoveries (2E+Au) of 74% (LM) and 69% (UM) with final concentrate grades of 270 g/t and 475 g/t produced respectively. Using the limited data available for the simplified MF1 circuit, these data results were used to extrapolate expected final recoveries to produce a 150 g/t concentrate. Based on these two high grade composite samples, a recovery of 78.9% (2E+Au) was estimated, whilst highlighting that additional test work is required to validate the extrapolation assumptions. This work has not been concluded.

Assuming negligible difference between the MF2 or MF1 circuit response, the graphs shown in Figure 9-2 to Figure 9-4, provides scatter plots between mass pull, recovery and upgrade ratios using applicable test work data conducted over the various campaigns. This includes the recent 2010 DRA open circuit and closed-circuit tests, 2004 test work managed by Harmony and 2009 test work managed by Platinum Australia.

The graphs highlight, as expected, a clear correlation between upgrade ratio (concentrate grade divided by feed grade) and mass pull and also recovery and mass pull. For each test the final grade achieved is indicated as a label in Figure zz, which highlights the higher mass pulls resulting in lower grade concentrates.

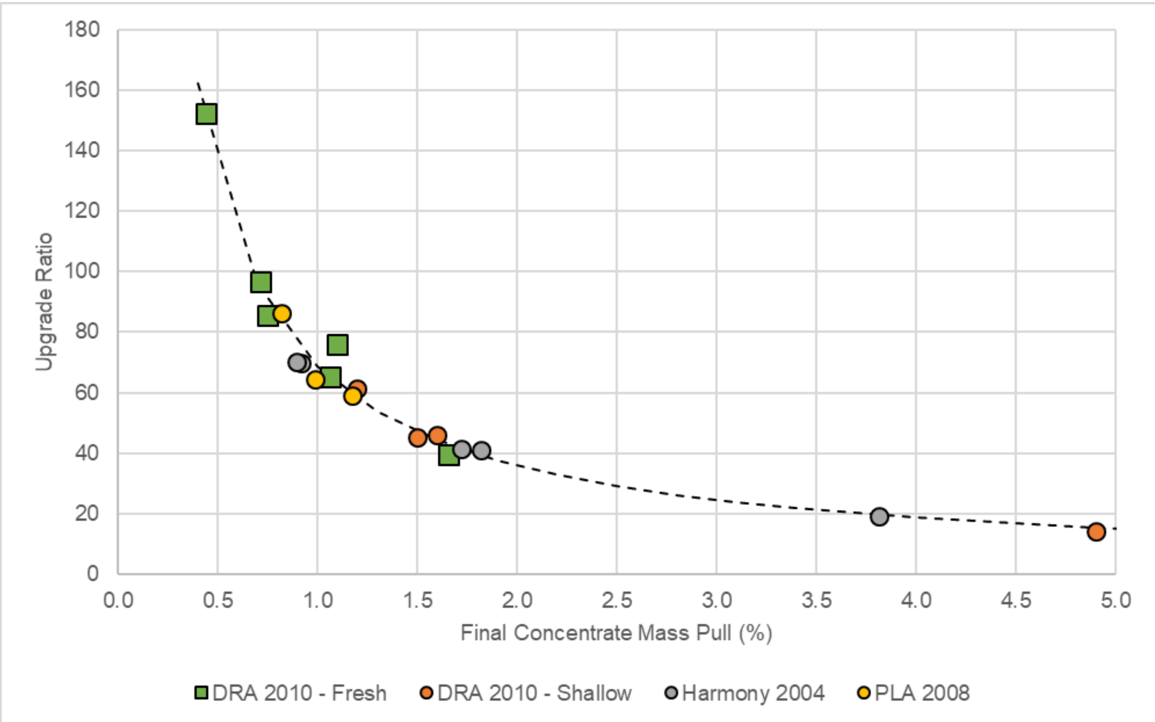


Figure 9-2: Upgrade Ratio vs Mass Pull

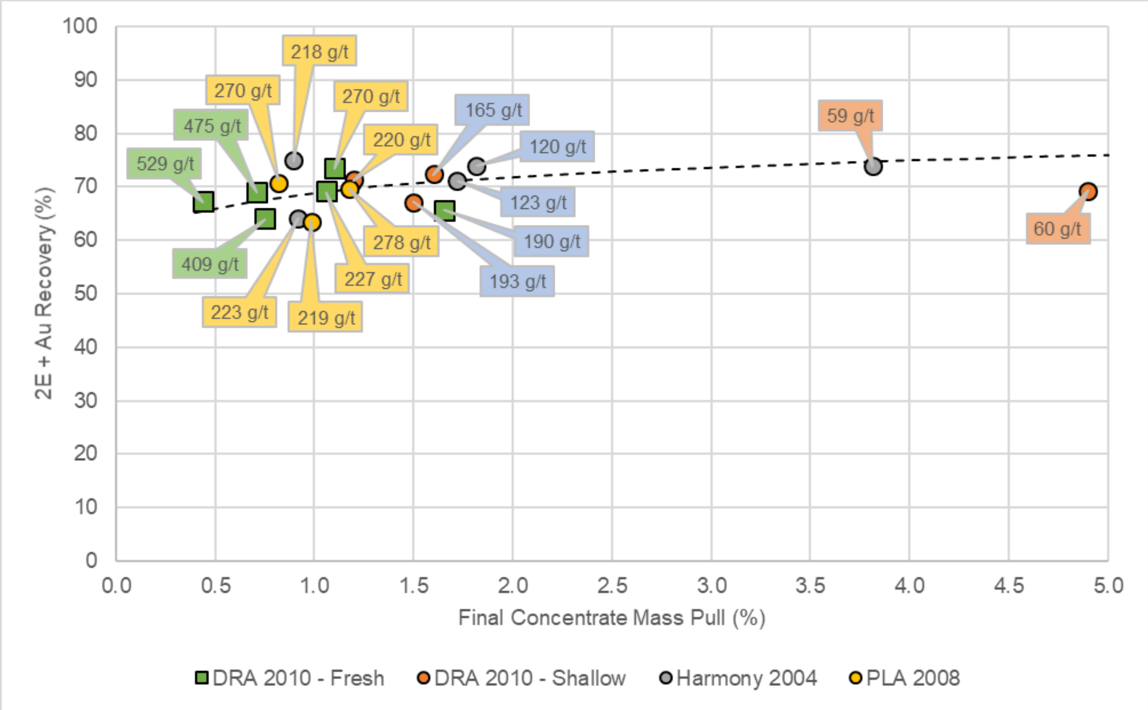


Figure 9-3: 2E+Au Recovery vs Mass Pull

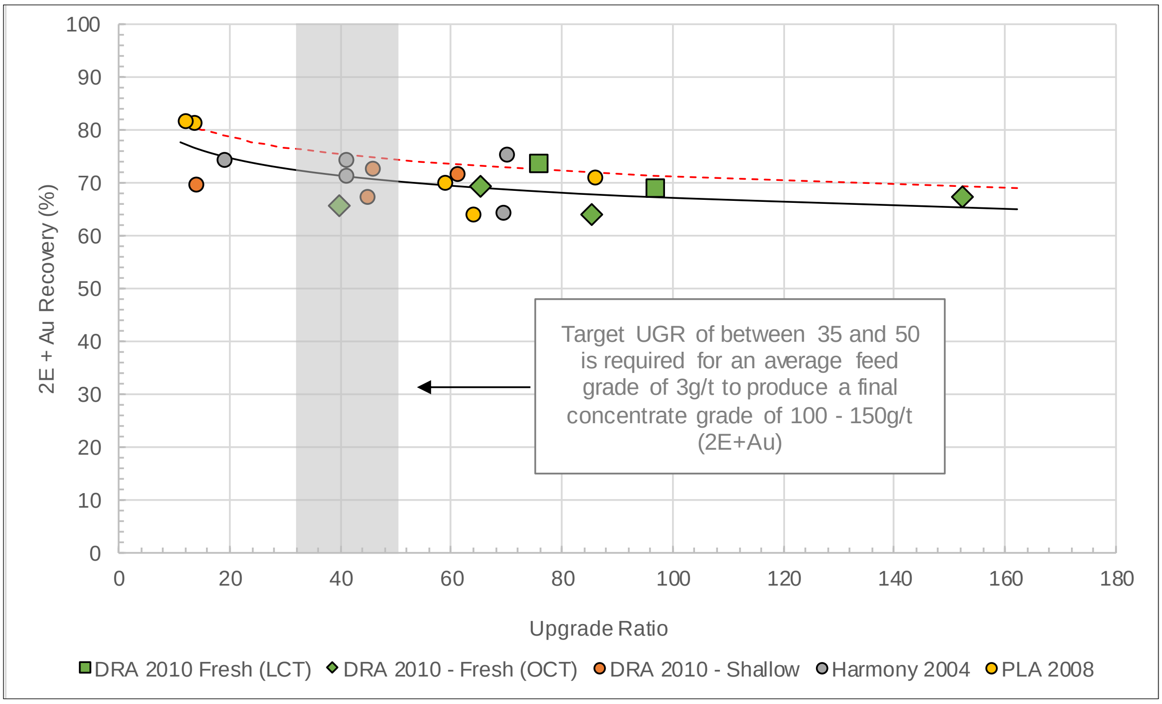


Figure 9-4: 2E+Au Recovery vs Upgrade Ratio

The various graphs highlight that extrapolation of the 2010 results is likely to result in an over-estimation of recovery, if all preceding test work is ignored.

Due to different circuits tested, variances in target grind and different reagent suites used, the previous data is not to be considered definitive either, again highlighting the need to conclude all necessary test on the simplified flowsheet. Insufficient test work, and elemental analysis, has been concluded to provide definitive data aimed at highlighting the recovery differentials between platinum and palladium.

In addition, limited work as detailed in previous study reports, has been concluded to provide any definitive estimates relating to the recovery of weather, oxidised or partially oxidised material. The 2010 study has highlighted that a recovery drop of up 15% can be expected for shallow, highly weathered material, with recoveries increasing as depth increased. Figure 9-4 highlights that, based on open circuit bench scale tests, the shallow (20 – 40 m) samples tested in 2010 appeared to follow a similar grade and recovery relationship to fresh material. However, only single composite samples for Mira and Sirius, were tested and no locked cycle test work was concluded. It should be noted, that earlier work by Harmony, reported significantly lower recoveries for weathered material.

As highlighted in the 2010 Feasibility Study, additional test work is recommended to support the simplified flowsheet selected and provide additional confidence in the recovery correlations. The test work should include optimisation test work (including the use of pre-conditioning ahead of flotation, fine grinding test work and flotation test work) on composite and variability samples that reflect the expected plant feed grade and mining schedule. The 2010 test work has indicated that high concentrate grades are achievable and it is believed that recovery optimisation may be possible (as shown in Figure 9-4, where optimised recovery estimates as a function of upgrade ratio are provided with the dashed line).

9.5.3. Concentrate Processing

There are a number of options that could be considered for downstream beneficiation of a flotation concentrate. In general, one of two typical downstream beneficiation options exist for flotation concentrate:

- internal beneficiation of concentrate using pyro-metallurgical or hydro-metallurgical processes, or
- beneficiation of concentrates by an external third party

Processing by an external third party is the most common strategy employed by newly established mining and processing firms. It is the most likely, low capital option, for the Kalplats mine toll with toll smelting or third party treatment at one of the existing smelters in the Rustenburg / Northam area. Off-take (the outright sale of concentrate to a third party) and toll treatment (beneficiation by a third party with metal return to the supplier) agreements vary significantly due to the multitude of variables that need to be considered. These are typically negotiated between the seller and purchaser, and are defined within numerous contractual arrangements. Agreements are structured such that a reasonable profit is attained after all relevant treatment costs (both operating and capital costs) associated with smelting and refining the flotation concentrates are covered.

These contracts generally consider inter alia: smelting and refining costs, metal grade, base metal grade, moisture content, sulphur content, deleterious element content, payment pipelines, penalties (compositional and other) as well as metal accounting provisions.

Based on initial engagement between Kalplats and Northam, it has been indicated that Northam would be interested in toll treating. Conceptually, Northam has indicated that the current minimum concentrate grade they would consider would be 100 g/t (2E+Au), but depending on volumes could consider low grades. Above 100 g/t their metal return is expected to be the range of 80 - 85%, with low 70% return for concentrate below 100 g/t. Detail of the amount of magnetite in final concentrate is required, and additional test work on the simplified MMF1 flowsheet is required.

Based on the feedback from Northam, targeting a concentrate of closer to 150g/t is believed will provide the best value to the project.

9.2. Process Description

A concise process description and been provided detailing the general treatment method adopted. An overall process schematic is presented in Figure 9-5 and incorporates the main unit operations which should be readed in conjunction with the process description.

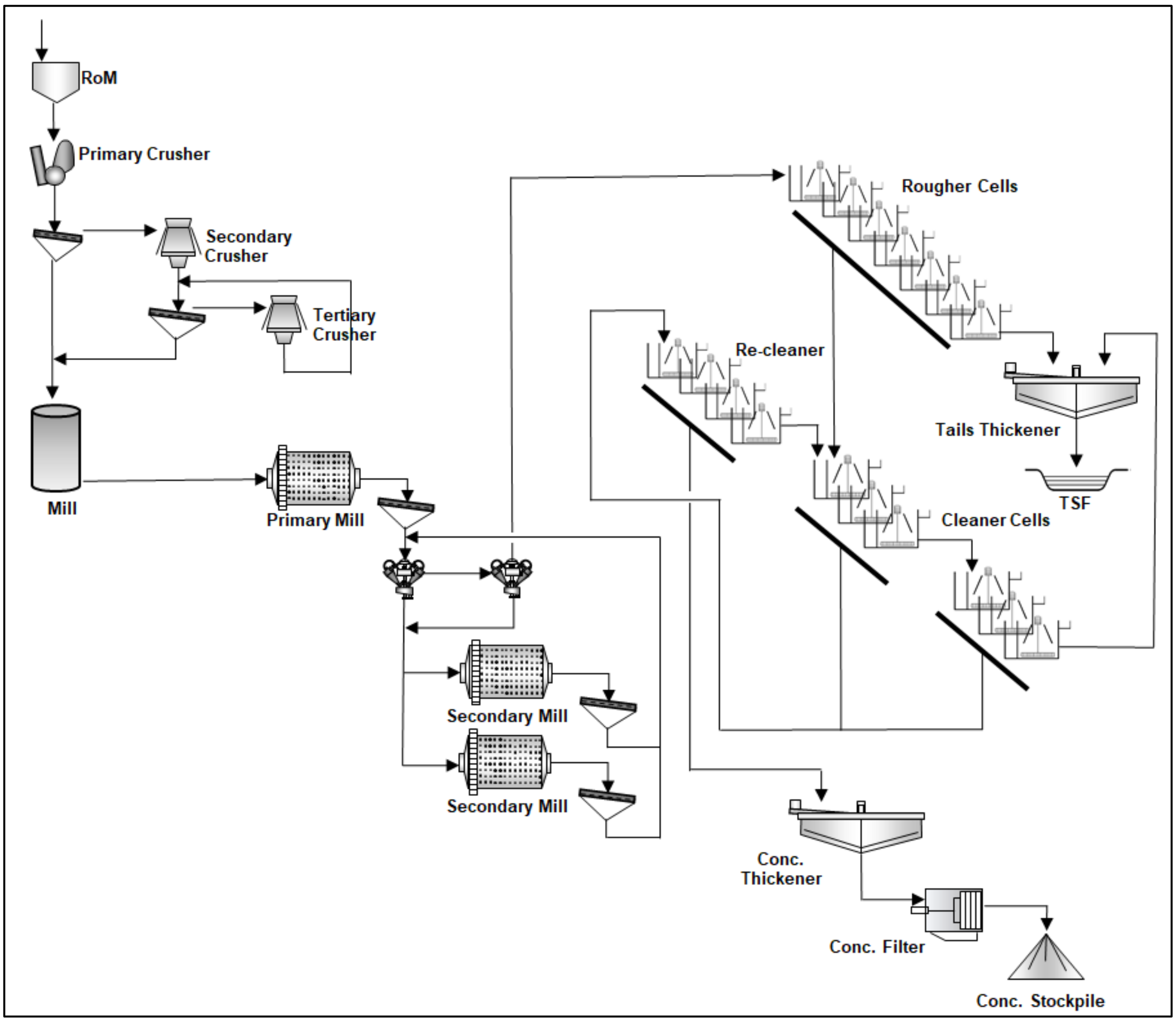


Figure 9-5: Schematic Flow Diagram

9.5.4. Ore Receiving

Run of mine (RoM) ore from the open pit mining operation is transported to the plant using dump trucks. These trucks will, under normal operation, tip directly into the RoM bin to serve as plant feed. In cases where direct tipping is not permitted, these trucks will dump ore onto the RoM pad. Material from the RoM pad would then be transferred using a front end loader for discharge into the RoM bin. A static grizzly with 600 mm aperture will control the top size fed into the bin.

9.5.5. Crushing

An apron feeder extracts the ore from the tip bin feeding onto a grizzly feeder. The grizzly oversize passes into the primary jaw crusher, with the jaw crusher discharge joining the grizzly undersize. The combined crushed ore is then conveyed to the secondary crusher circuit screen.

The secondary crusher circuit screen undersize is conveyed to the mill silos. A belt magnet will clean the oversize portion of the material from the secondary screen to protect the cone crusher. The oversize passes into the secondary crusher feed bin which feeds the secondary cone crusher where material is crushed to <30 mm. The crusher discharge is then conveyed to the tertiary circuit screen.

The tertiary crusher circuit screen oversize passes into the tertiary crusher feed bin. From there it is fed with a vibrating feeder to a cone crusher where it is crushed to <10 mm. The crusher discharge is then conveyed back to the tertiary circuit screen. Screen undersize is conveyed to the mill silos. A common conveyor is used to collect both screen undersize from the secondary and tertiary crusher circuit screen.

9.5.6. Primary and Secondary Milling

The milling circuit will encompass primary and secondary milling. Ore from the feed silo is conveyed to the primary mill. The slurry exits the mill and passes over a vibrating screen with a 4mm deck. The +4 mm is collected on a belt and returned back to the mill. Spray water is added to the primary mill discharge screen with make-up water and the screen underflow discharging into the primary mill discharge sump where it is combined with the secondary mill product.

The combined mill discharges are pumped with dedicated pumps and lines to cyclones. The cyclone overflow gravitates to the deslime tank from where it is pumped to the desliming cyclones. The deslime cyclone overflow gravitates to the flotation circuit. All the cyclone underflows are collected and split evenly to the two secondary mills with the secondary mill discharges combining into the mill discharge sump.

All scats will collect onto a common belt and can be removed from the circuit with a magnet if recycle becomes excessive.

9.5.7. Bulk Rougher and Cleaner Flotation

The primary rougher flotation section consists of a series of tank-type flotation cells operating in series, with gravity flow. The smaller high energy cell is utilised before this bank to break the flocculant down. It can also be used as a conditioner or produce a concentrate.

The concentrate from the first two cells can be sent to final concentrate or to the recleaner. All other cells combine into a rougher concentrate tank from where it is pumped to the cleaners. The rougher tails from the last cell will enter the tank just above the cone section of the sump or with a back-pressure pipe to limit the effect on the last cell as well as to save height on the building. The slurry exiting the final rougher cell passes directly into the rougher tails tank from where it is pumped to the tailings thickener.

The cleaner flotation section consists of twin boxes of three mechanisms each operating in series, with gravity flow. There is a step-in cell height between the two boxes to ensure accurate level control in each box. The recleaner bank consists of one box with four mechanisms. Recleaner tail feeds to the cleaner circuit. The slurry exiting the final cleaner cell passes into the tailings sump, from where the slurry is pumped to the rougher cells.

9.5.8. Tailings Thickening and Disposal

Flotation tails is fed to the thickener with flocculant being added to aid with solids settling. The thickener overflow is used as process water within the plant while the underflow is pumped to the final tails tank.

The tails thickener underflow is pumped to the final tails tank. The tailings are then pumped via three stages to the tailings dam through dedicated pipelines. The penstock and drain water is collected from the dam into a return water pond from where it is pumped back to the process water pond.

9.5.9. Concentrate Dewatering

The final concentrate is fed to the concentrate thickener. From this tank, the slurry is fed to the concentrate filter. Filtrate is circulated back to the concentrate thickener and filter cake is conveyed to the day bunkers ready for dispatch to the purchaser.

Concentrate thickener overflow is pumped through a pressurised cyclone canister for use as spray water within the plant with the excess going to the gland service water tank. The filter is periodically back washed to the concentrate thickener.

A rake lift has been installed on the thickener due to the fact that small stock tanks will be used and the thickener might occasionally be used for storage as well.

9.5.10. Process Water

Tails thickener overflow, tailings dam return water, collected storm water and sewage return are all fed to the process water pond, via an intermediate tank. From this tank, the process water is distributed around the plant via dedicated pumps. A hosing pump and distribution system is also provided from the process water tank.

A pressurised canister system will be used from this process water tank. The cleaned product from the cyclone overflows will feed to the gland service water tank, which previously has been fed from the fresh water supply. It will be an excess supply with all excess flowing back into the process water tank. If the level in the process water tank is above set point (as a result of sufficient return water), no make-up water will therefore be required, thus reducing the fresh water consumption.

9.3. Raw, Clear, Grand Service, Fire, Potable Water

9.5.11. Raw Water

The only source of fresh water supplied to the plant will be the bulk header water tank. The bulk water header tank will receive water from the main fresh water supply / supplies. This tank will be kept full by the municipal/borehole supply. This tank will provide the plant with a low-pressure medium volume steady supply. In the event of a supply failure, this tank can then be emptied before a plant shut down needs to commence.

The installation of this tank reduces the peak supply required as it serves as a buffer to the plant and demands can be prioritised and controlled during an emergency.

Distribution inside the plant will be done from various tanks that will be made up from this bulk water header tank.

7.5.5 Gland Service / Clear Water

The gland service water tank (clear water) will overflow into the process water tank and will have excess supply while the process water tank still maintains level via the process water cyclones. The process water cyclones will clean process water to make it suitable for gland service usage. Mill seal water, dust suppression, and all other applications capable of using clear water with some reagent in it, will utilise water from this tank.

7.5.6 Clean Water

The clean water tank will be used in applications where no solids or reagents are allowed i.e. regent make-up, tailings HP GSW pumps and concentrate filter clean water.

7.5.7 Potable and Fire Water

Water supplied from the bulk water header tank will be cleaned in an ozone water treatment plant. From there it will distribute to three tanks. One tank will be dedicated to supplying all human consumption requirements, another two for fire water. Electric jockey pumps are used to maintain normal water pressure in the fire main system.

The potable water and fire water supplies will be vendor suppy packages. The plant will be self-sufficient without these systems.

9.4. Air Reticulation

A number of compressed air systems are provided, including the common high-pressure systems;

- Pressing and drying air for the concentrate filters
- General plant air
- Instrument air

These systems include high-pressure air compressors, dryers (for instrument air), receivers and filters.

Concentrate filter air will be provided using a dedicated compressor while the plant and instrument air will be generated by seperate system.

A low-pressure air system is provided for the plant flotation air requirements. The low-pressure air is provided by positive displacement type blowers.

9.5. Collector Make Up and Distribution

A dry powder system will be utilised for make-up with dedicated dosing pumps. Bags will be added to a hopper above a make-up tank. Made-up collector (batch) will discharge into a day tank and be used dose to the required points in the plant.

9.5.12. Senkol 2 Storage and Distribution

Senkol 2 dosage will be distributed directly to the required dosing points by positive displacement reagent pumps from mother unit tanks or drums.

9.5.13. Frother Storage and Distribution

Frother will be distributed directly to the required dosing points by positive displacement reagent pumps from mother unit tanks.

9.5.14. Flocculant Storage and Distribution

Bags of 250 kg or 1 t will be dropped into a hopper and then fed to a continuous make-up system using a feeder, blower, and jet wet head. The continuous make-up system tank will allow time for mixing, hydration, and dosage in sections of the tank. From this tank, the flocculant will be distributed to the plant by positive displacement pumps to the various dosing points.

10. TAILINGS DAM

Knight Piésold was appointed by ATPL to update the Kalplats tailings dam design and budget pricing to include the latest legislation and include this in the Feasibility study update.

See appendix A - Kalplats TSF PFS Final Design Report 10 Jun 2020 for details.

11. PROJECT IMPLEMENTATION

The overall anticipated project implementation schedule is summarised in the table below.

Table 11-1: Kalplats Project Implementation Timelines

Event	Time	Start
Project approval	1 months	October 2020
Mining Licenses	2 months	November 2020
Detailed Design	6 months	January 2021
Construction	18 months	July 2021
Cold Commissioning	1 months	December 2022
Hot Commissioning	3 months	January-March 2023
Full Production		April 2023

12. CAPITAL COST ESTIMATE

12.1. Mining Cost

Capital expenditure incurred is for site establishment amounts to ZAR 65m. A capital contigency of 10% has been applied. Pre-development costs of ZAR 29.6m have been capitalised during the first year of production.

12.2. Process Plant Cost

The capital cost estimate (CCE) is based on a process plant capable of treating 1.5 million tonnes per annum and as defined by the process flowsheets, design criteria and the infrastructure and services described in this document.The capital cost estimate is summarised in Table 12-1:.

Table 12-1: Kalplats Project Process Plant Capital Cost Summary

Capital Costs	Value, ZAR	Source
Concentrator Plant (Direct Costs)	966,528,906	DRA
Plant and General Infrastructure (Direct Costs)	138,332,733	DRA
Process Plant (Indirect Costs) incl. EPCM	197,013,514	DRA
Power Supply*	180,690,783	ATPL
Water Supply*	216,124,823	ATPL
Land Purchase*	13,537,500	ATPL
TSF#	278,760,047	Knight Piésold
SUB TOTAL	**1,990,988,306**	**Calculated**

Capital Costs	Value, ZAR	Source
Contingency – (excluding TSF)	177,722,826	Calculated
TOTAL	**2,168,711,132**	**Calculated**

*No cost revalidation conducted, as provided by ATPL. #Costing provided by Knight Piésold.

DRA was commissioned to revalidate the 2010 Definitive Feasibility Study ("DFS") on the Kalahari Platinum ('Kalplats') Project. This section describes the basis and methodology used to revalidate the CCE for the Process Plant, which includes the project capital costs, project indirect costs, infrastructure costs and contingencies.

The following documents provided the basis for the 2010 Capital Cost Estimate, and still apply:

- Block Plan and Layouts
- Process Flow Diagrams
- Process Design Criteria
- General Engineering Design Criteria
- General Arrangement drawings
- Supplemental Sketches as Required
- Equipment Quotations from Vendors and Suppliers
- Fabrication and erection rates from Vendors and Suppliers
- Mechanical Equipment List
- Electrical Motor lists
- Electrical, Control and Instrumentation Schedules
- Project Execution Programme
- Battery Limits as set out in the Process Design Criteria

The estimate is still categorised as a "Feasibility Type" with a combination of detailed, semi-detailed and factored costs.

The estimate has been produced using vendor firm and budget quotations and where required SEIFSA escalation percentages and is based strictly on the equipment as described within the CCE. Any changes to the equipment would require a modification to the costs. Contingencies for work have not been built into quantities and rates but the estimating inaccuracy, as a result of limited detail engineering, is covered in the overall contingency provision. Costing for the TSF has been provided by Knight Piésold.

12.3. Estimate Criteria

14.3.1. Base Date

The base date for the revalidated CCE is January 2020.

14.3.2. Base Currency and Foreign Exchange Rate

The revalidated estimate has been presented in the South African Rand (ZAR) currency. Capital costs exposed to foreign exchange have been predominately linked to the United States Dollar (USD). These costs have been recorded in USD and rebaselined to a ZAR equivalent. A USD:ZAR exchange rate of 17 has been applied for the estimate (as advised by ATPL).

14.3.3. Accuracy

The revalidated estimate meets the required accuracy criteria of ±15%.

14.3.4. Escalation

Approximately 10% of all costs (plant only) in CCE could not be revalidated. Escalation was applied to these cost items from the base date of the original CCE May 2010 to the current terms. SEIFSA escalation indices were identified based on the nature of work to be undertaken, i.e. labour, steel, earthworks, civils, EC&I, mechanical equipment. Each discipline-specific future index is calculated using the previous two-year average, increase / decrease, from which an overall escalation rate for each discipline was extrapolated.

The escalation indices as well as the calculation tables are contained in the CCE on separate tabs.

14.3.5. Estimating System & Format

The revalidated estimate has been prepared in the DRA estimating system in MS Excel format.

14.3.6. Estimate Report Requirements

The revalidated CCE is presented as a fully detailed estimate, together with a summary sheet.

12.4. Cost Revalidation Basis

This section describes the method and basis of the revalidation. The same categories for the allocation of costs to the engineering disciplines have been maintained for ease of comparison, and are as follows:

14.4.1. Earthworks, Civil Works, Infrastructure and Building Works.

The existing BOQ's from the 2010 CCE were used and populated with current rates and P&G's from Contstruction Contractors as per recent DRA projects. Items that could not be revalidated were escalated as described above.

14.4.2. Structural Steelwork, Platework Supply & Erection and Mechanical Erection, incl. Conveyors.

The existing BOQ's from the 2010 CCE were used and populated with current rates and P&G's from Contstruction Contractors as per recent DRA projects. All items were revalidated.

7.5.7.1 *Mechanical Equipment*

All plant mechanical equipment including conveyor mechanical equipment (as per the 2010 CCE) has been revalidated from quotes received from the original equipment suppliers.

The mechanical erection P&G's were as per the current SMPP quotes received on recent DRA projects, and are applied to the mechanical erection component only.

7.5.7.2 *Piping*

The in-piping for the plant has been factorised from the mechanical equipment supply cost by the same factor applied in the 2010 CCE. As the mechanical supply price has increased, the piping has also increased.

Overland pipework, valves and fittings were quantified by DRA in the 2010 CCE, and these MTO's were revalidated using supply and erect rates from current 2020 DRA projects.

7.5.7.3 *Electrical, Control and Instrumentation*

The existing EC&I BOQ's from the 2010 CCE were used and populated with current rates and P&G's from selected EC&I Contractors as per recent DRA projects. Items that could not be revalidated were escalated as described above.

14.4.3. Transport

Transport costs for major mechanical equipment were obtained from vendor quotations . Transport costs for steelwork and platework and EC&I were as per the quoted rates from contractors as per current DRA projects. All other transport costs were based on a tonnage or per load basis using DRA's data base.

14.4.4. Preliminary and General Costs

Preliminary and General costs include all contractors' overheads such as contractual requirements (safety, sureties, insurance, etc.), the site establishment and the removal thereof, and company and head office overheads. They also cover supervision, travel to and from the site, contractor supplied temporary facilities, offices and Laydown areas, tools and contractors' equipment.

P&G costs been allowed for at a percentage of the value of the works to be executed and are shown separately against each estimate sub-heading as summarised below:

The P&G's are from recent 2020 quotes and are in line with current market trends experienced on DRA projects. Refer to Table 12-2 below.

Table 12-2: Discipline P&G's

P&Gs	%
Earthwork, Roads and Infrastructure P&Gs (on total suncontract value)	40.00%
Civils and Buildings P&Gs (on total suncontract value)	45.00%
SMP P&Gs (on total supply and erect value)	46.00%
Piping P&Gs (on total supply and erect value)	46.00%
EC&I P&Gs (Plant) (on total supply and erect value)	20.00%
EC&I P&Gs (Infrastructure) (on total supply and erect value)	24.24%

14.4.5. Spares

Spare parts costs have been included in the estimate to cover commissioning spares for both mechanical equipment and electrical equipment. These spares are mainly required to facilitate the DRA commissioning process and reduce possible delays due to equipment failure.

The spares holding costs have been derived as a factor of 2% of the mechanical and EC&I equipment supply costs.

14.4.6. First Fill and Consumables

First Fill of Grinding Media for 1 x Primary and 2 x Secondary Ball Mills has been costed and included in the CCE using current quotes from a current project.

First Fill of Reagents, existing 2010 CCE costs have been escalated as per SEIFSA escalation indices described above.

14.4.7. External Services

The External Services section contains EPCM type labour costs for consultants, these 2010

CCE costs have been escalated as per SEIFSA escalation indices described above.

The following escalated costs have been included in the estimate:

- QS services,
- Geotechnical investigations
- Fire protection assessment consultants
- Mass Flow consultants,
- PLC / SCADA consultants
- Surveying services
- Hazop services

14.4.8. Pre-Production Costs

The Pre-Production section contains vehicles, security during construction, construction power, hire of cranage on site during construction, temporary ablution facilities and EPCM site offices and establishment. All of which are existing 2010 CCE costs and have been escalated as per SEIFSA escalation indices described above.

14.4.9. EPCM Costs

The EPCM and external costs of the Kalplats 2010 CCE were calculated from first principles using DRA manhour rates and the project execution schedule. The 2010 CCE was 12% of total plant cost. The EPCM cost for the revalidated 2020 CCE is 12,5%.

14.4.10. Power Supply

No revalidation conducted, as provided by ATPL.

14.4.11. Water Supply

No revalidation conducted, as provided by ATPL.

14.4.12. Land Purchase

No revalidation conducted, as provided by ATPL.

14.4.13. TSF

Revalidated, as provided by Knight Piésold.

14.4.14. Contingency

The estimating inaccuracy is assessed as a measure of the confidence in the design and estimating processes / outputs. The confidence in each design and engineering discipline has been assessed against the following scale with the following scale with the associated contingency allowance. Contingency has been assessed as a percentage for the individual cost components, from which an overall estimating inaccuracy is derived.

This contingency (Estimating Inaccuracy) percentage has been calculated to be approximately 10% (exclusive of TSF).

14.4.15. Exclusions

- The following (but may not be all) are not included in the capital cost estimate
- No allowances have been made in the estimate for financing costs or interest during construction
- Taxes and duties, interest charges, etc. have been excluded from the estimate
- VAT, fees, statutory and import duties are all excluded (except for certain reagent costs)
- Cost of schedule delays such as those caused by
- Scope changes
- Labour disputes
- Environmental permitting activities
- Cost of financing
- Acquisition costs
- Sunk costs
- Additional studies prior to EPCM
- All VAT, import duties, surcharges and any other statutory taxation, levies or government duties (except for certain reagent costs)
- All royalties, commissions, lease payments, rentals and other payments to landowners, title holders, mineral rights holders, surface right holders, and / or any other third parties not mentioned in this documentation
- Forward cover for any foreign content (it is assumed that this would be accommodated by the client if necessary)
- All operating costs

- Any work outside the defined battery limits
- Any provision for project risks outside of those related to design and estimating confidence levels
- Fuel Storage
- Interest on capital loans
- Any costs to be expended following completion of the feasibility study and prior to Board approval for project implementation
- Mineral rights and the purchase or use of land
- The costs of any trade off studies
- Social impact studies
- Client project staff costs
- Client travelling and accommodation
- Client Site transport and fuel
- Insurances

13. OPERATING COST ESTIMATE

13.1. Mining

15.1.1. Mining Operating Cost Estimate

Four mining contractors were approached for open pit budget contractor mining pricing in March 2020. These were WBHO, Stefanutti Stocks, Trollope Mining Services and Andru Mining. Three of the submissions where reasonably close to each other and the best two of these formed the basis of the open pit mining costs used for the capital and operating costs for the Kalplats Project. The contractor costs have been used and include allowance for the varying haul distances from the pits to the Plant that range between 2,500 m and 5,500 m over the scheduled LoM. The waste and low-grade ore dumps are generally within 1, 500 m from the pits.

15.1.2. Contractor Fixed Cost

The contractor monthly fixed cost has been calculated to be R 4,700,000 per month. Items which are included in the monthly fixed cost are as follows:

- Salaries and Wages

- Housing
- Safety, PPE and Legal Compliance
- Travelling
- Equipment excluded in the production unit rates
- General overheads

15.1.3. Contract Miner Rates for the Life of Mine

The base cost provided by the four Mining Contractors that submitted pricing are summarised in the Contractor Mining Rates in Table 13-1 below.

Table 13-1: Kalplats Mining Contractor Rates Summary

Mining Contractor Rates Comparison (ZAR)	WBHO	Stefstocks	Trollope
1. Site Establishment	68,597,675	62,073,867	16,725,867
1. Site Disestablishment	10,320,000	2,096,642	8,461,650
2. Time Related G&A (total 144 months)	676,483,200	690,383,949	868,703,973
3a. Bush Clearing Hectares	19,435.69	64,629.53	72,750.00
3b. Topsoil Stripping L&H and Dump and Level 2 km free haul	45.57	48.30	53.09
4. L and H Free Dig 2.5 km free haul	60.80	54.21	59.43
5a. Hards Drilling Cost per meter	166.55	260.44	143.50
5b. L and H Hards 2.5 km free haul	67.53	68.19	76.83
6a. Ore Drilling cost per meter	195.46	260.44	150.00
6b. L and H Ore 3.5 km free haul	26.62	25.92	31.21
7. Ore Re-handle at ROM Tip (max 400 m)	20.52	14.75	11.30
8. EO Haul Rate per 200 m (applied to increased haul distances)	0.53	0.52	1.33

The costs are based on March 2020 cost calculations and price indices.

13.2. Process Plant

The 2010 Feasibility Study process operating cost estimate was based on contracted operations using MINOPEX. As a result, a full breakdown of costs was not presented in the Feasibility Study report. For the current cost update study, a first principle zero-based build-up of operating costs, including a breakdown of the fixed and variable costs, was concluded using the 2010 PDC, design and mechanical equipment list as the basis. DRA's database of recent 2020 studies and projects was used for supply costs of power, consumables, reagents and labour.

The operating cost estimate presented includes all labour, materials, reagents and consumables necessary to support the operations.

The operating cost estimate is presented in Unites Stated Dollar (USD) and has been based on an exchange rate of ZAR:USD of 17. The estimate has is expected to have an accuracy of ±15 to 20%.

Fixed Costs are defined as the costs that will be incurred irrespective of production rates. These costs would typically include, labour costs, environmental costs and the fixed portion of the plant maintenance costs. Variable costs are defined as costs that vary depending on the level of production. These costs are based on unit consumption rates and are the costs that are incrementally incurred as production rates vary. The costs would typically include reagent consumption, power, water and variable consumable costs (e.g. mill and crusher liners).

All steady state costs presented are based on processing 1.5 Mtpa (125 ktpm) with an overall run-time of 7,800 hours per annum.

The following are items specifically excluded from this estimate:

- All value-add taxes (except for certain reagent costs), import duties and/or any other statutory taxation, levies and/ or national and local institutions
- Contributions to social programmes
- Contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements
- Concentrate marketing, off take agreement and concentrate transport costs
- All costs beyond the battery limits of the process plant fence (except for TSF)

15.1.4. Labour Costs

Labour costs are based using rates for similar South African operations with a complement estimate of 148 people. The estimate makes provision for production plant management, technical services, plant administration, production teams, engineering, laboratory and maintenance personnel. For the production team continuous operation is assumed with 4 shift teams on rotation.

A summary of the labour breakdown and cost is provided in **Table 13-2**.

Table 13-2: Process Labour and Cost

Description	Number	Labour Cost (ZAR per annum)
Management and Technical Services	5	4,713,675
Administration	10	3,976,424
Process Operators	95	27,166,527

Description	Number	Labour Cost (ZAR per annum)
Engineering	38	15,676,446
Sub-Total	148	51,533,073

15.1.5. Power Costs

Power consumption for all processing equipment (excluding primary and secondary mills) was determined using estimates for power draw (based on installed power), equipment utilisation and mechanical efficiency factors where applicable. The mechanical equipment list developed in the 2010 Feasibility Study was used as the basis for the estimation.

For milling energy consumption, actual modelling results were used. Power estimates provided are based on the average ore hardness data derived from test work. It should however be noted that noteworthy ore competency variability is observed when analysing all comminution data from the various test work campaigns conducted (work indices range from 15.0 and 22.0 kWh/t over the 90% confidence range). With more definitive geomet understanding and modelling, a more conclusive model aligned with the mining schedule could be concluded.

Cost estimates for the office buildings, change houses and equipment outside of the process plant fence have not been included in the estimate reported below. The operating cost estimate has been based on an average power cost of ZAR0.95 /kWh. Eskom, the national power provider, has not been consulted to provide updated costs.

Table 13-3: Process Power Consumption and Cost

Description	Power Consumption (kWh/t)	Steady State Power Cost (ZAR per annum)
Crushing	5.02	7,160,078
Milling	62.1	88,554,048
Other Process	17.8	25,425,125
Sub-Total	85.0	121,139,250

15.1.6. Reagents and Consumables

The cost of consumables includes the cost of crusher circuit liners, mill circuit liners, grinding media and reagents as outlined below.

All crusher liner, mill liner and grinding media consumption has been derived using modelling based on an abrasion index of 0.18 derived in test work. Limited understanding of the variability is known. Recent 2020 supply costs for high chrome grinding media, delivered to site, has been used along with cost estimates for liners as provided in recent quotations received from equipment suppliers for the Kalplats project.

Table 13-4: Liner and Grinding Media Cost Estimates

Description	Steady State Cost (ZAR per annum)
Crusher Liners	1,390,905
Primary Mill Liners and Grinding Media	20,632,723
Secondary Mill Liners and Grinding Media	33,640,748
Sub-Total	55,664,375

Reagent costs were calculated using the average consumption rates for optimised conditions defined in the 2010 Feasibility Study. Recent 2020 supply costs, from reputable reagent suppliers have been used and costs include transport costs for delivery to site.

Table 13-5: Reagent Cost Estimates

Description	Steady State Cost (ZAR per annum)
Collector	17,822,337
Co-Collector	322,221
Frother	6,311,060
Flocculant	1,842,439
Sub-Total	26,298,056

15.1.7. Maintenance Costs

Maintenance costs (excluding crushers, primary mill and secondary mills) have been estimated based on the updated capital cost estimates for the project and typical annual replacement rates ranging between 10 and 20 years. The maintenance cost for the additional equipment was based on the recommended annual operating spares estimates from chosen equipment vendors. These additional maintenance costs are summarised in Table 13-6.

Table 13-6: Maintenance Cost Estimates

Description	Steady State Cost (ZAR per annum)
Platework Maintenance	2,323,398
Piping Maintenance	3,498,929
Mechanical Maintenance	11,183,992
EC&I Maintenance	6,459,365
Sub-Total	23,465,683

15.1.8. Water Costs

Raw water consumption estimates are based on benchmark figures for similar projects employing conventional tailings disposal. An overall site water balance has not been developed in conjunction with the selected tailings consultant for the study. The cost model is based on an average process plant raw water consumption, of 0.60 m^3 per tonne milled. A water supply cost of ZAR 7.0 /m^3 has been used based recent studies concluded in the area. No allowance has been made for any water treatment costs.

Table 13-7: Water Cost Estimates

Description	Steady State Cost (ZAR per annum)
Water Cost	6,300,000
Sub-Total	6,300,000

15.1.9. Laboratory Costs

Allowance for on-site laboratory services has been included. This includes all metal accounting and process control services required to support the operations. An annual allowance of ZAR 1.2M for consumables is included.

15.1.10. TSF

No revalidation conducted, as provided by ATPL.

15.1.11. G&A

No revalidation conducted, as provided by ATPL.

15.1.12. Process Operating Cost Summary

Table 13-8 provides a summary of the process plant operating costs, based on a steady state annual treatment rate of 1.5 Mtpa.

Table 13-8: Summary of Steady State Process Operating Costs

Description	Process Cost	
	ZAR per annum	ZAR/t
Labour	51,533,073	34
Power	121,139,250	81
Stores & Maintenance	23,465,683	16
Liners & Grinding Media	55,664,375	37
Reagents & Consumables	26,298,056	18
Water	6,300,000	4
Laboratory	1,200,000	1
Tailings Handling (ATPL)	3,750,000	3
G&A (ATPL)	20,000,000	13
Total	**309,350,437**	**206**

14. ECONOMIC ANALYSIS

See :-

- Appendix B Kalplats Financial Model Rev G - Base Case
- Appendix C Kalplats Financial Model Rev G – Upside

For full financial modelling input and results

14.1. Introduction

The economic viability and performance of the Kalplats project has been determined through developing an economic model founded on the results derived from the revalidated feasibility study. The economic analysis has been carried out using discounted cash flow (DCF) methodologies and has been based on earnings before interest, taxation, depreciation and amortisation (EBITDA). All cash flows are presented in constant terms and as such does not consider the effects of inflation, interest, escalation and other financial adjustments. The economic model has been populated on a 100% equity basis and does not consider alternative financing scenarios. Financing related costs such as interest expense, withholding taxes on dividends and interest income, are excluded from the economic model. Exclusions pertaining to capital and operating costs are described in the relevant report chapter within this report.

Cash flows considered in the cash flow model include revenue, operating costs, initial capital expenditure, stay in business (SIB) capital allowance, capital contingency, environmental rehabilitation capital allowance and royalties presented on a year-by-year basis. A two year construction timeline has been scheduled with a production time span of 12 years. All currency figures are reported in South African Rand (ZAR). A project start date of 2021 has been assumed.

14.2. Mine Production Profile

Mine production is reported as mineralised material and waste from the open cast mining operation. The mine design and schedule has been revised and optimised for eight known deposits which have measured and indicated resources. The mine schedule has been modelled based on achieving an approximate 1.5 Mtpa plant feed thereby withdrawing the requirement for ore re-handling through intermediate stockpiling.

The mine schedule over life of mine is presented in Figure 14-1 below. Over life of mine, a total of 17.7 million tonnes of mineralized material (oxides and sulphides) is delivered to the processing facility, with 124 million tonnes of waste removed over the same period, equivalent to a strip ratio of 7.01. The average 2E+Au basket grade (Pt, Pd, Au) over life of mine is estimated at 2.92 g/t mineralised material.

Further detail covering the mining schedule can be sourced from subsequent sections of this report.

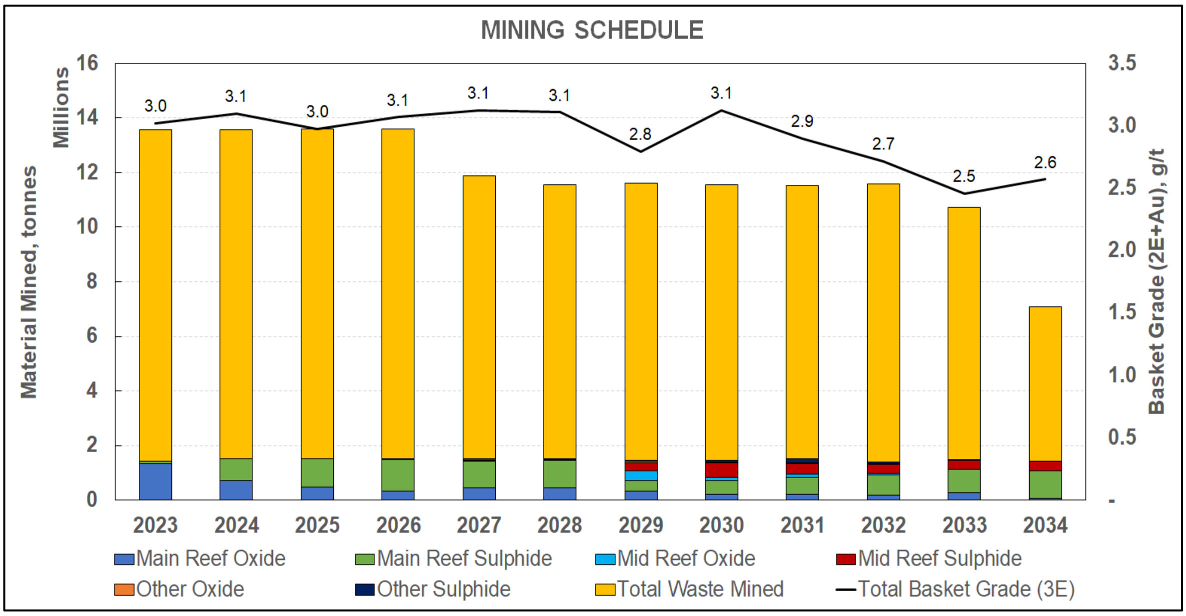


Figure 14-1 Total Material Moved Mining Schedule

14.3. Plant Feed and Production Profile

The tonnage design for the process plant is based on achieving a peak milled tonnage of 1.5 Mtpa. The plant tonnage ramp-up profile has been sequenced over a three month time span thereby achieving nameplate capacity by month 3 of production. A six month recovery ramp-up has been applied. An overview of the ramp-up strategy is presented in **Table 14-1**.

Table 14-1 Production and Recovery Ramp-up Profile

Production Month	Tonnage Ramp-up	Recovery Ramp-up
1	50%	70%
2	75%	75%
3	100%	80%
4	-	85%
5	-	90%
6	-	100%

The mine and plant feed schedule are identical considering there are no intermediate stockpile requirements. A total of 17.7 million tonnes of mineralised ore is fed to the plant over a 12 year life of mine at a 2E+Au basket grade of 2.92 g/t mineralised material. The annual plant feed schedule is presented in Figure 14-2 below.

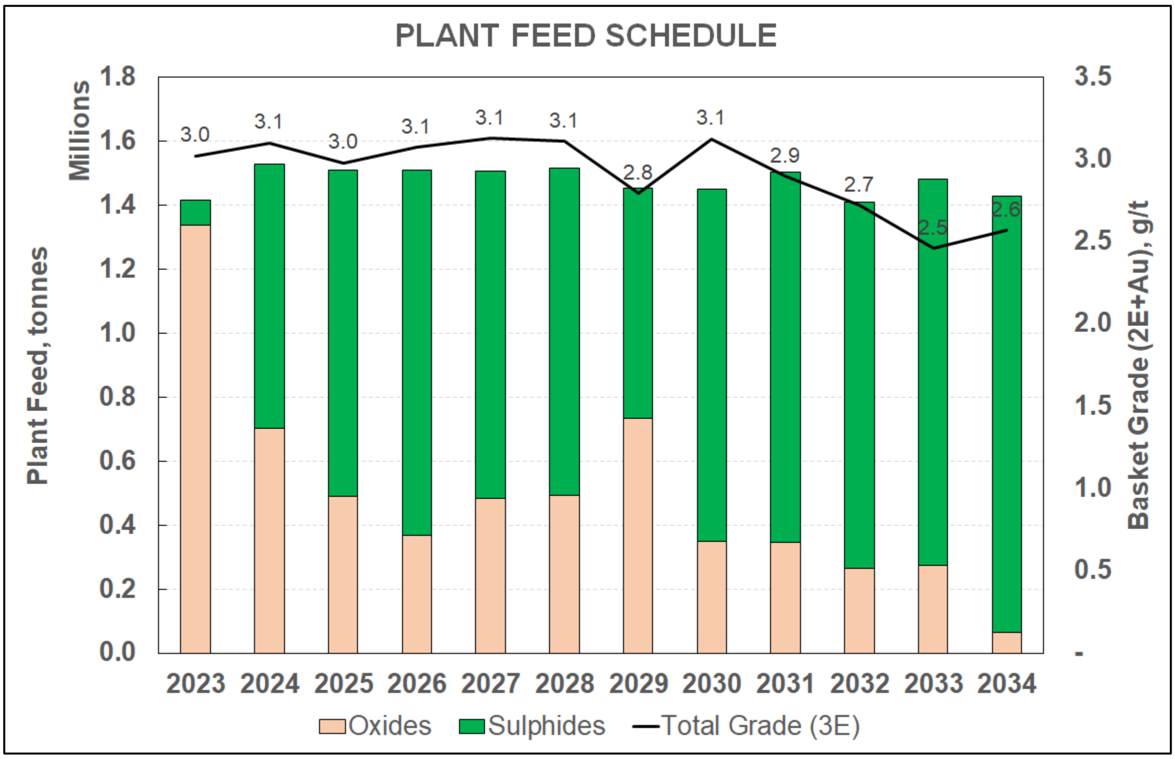


Figure 14-2 Plant Feed Schedule

A concentrate 2E+Au basket grade of 150g/t is estimated equivalent to an approximate 1.3% mass pull. An overall base case metal recovery of 66.2% is expected based on outcomes derived from historical testwork. The annual concentrate production schedule is presented in Figure 14-3 below.

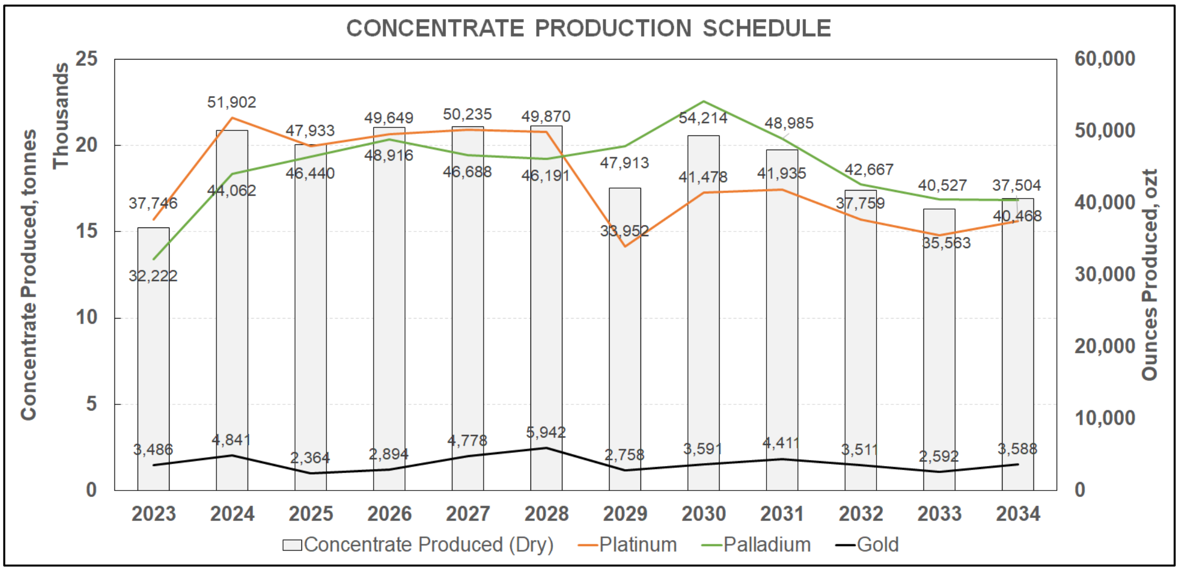


Figure 14-3 Concentrate Production Schedule

14.4. Capital Expenditure and Phasing

The total initial capital estimate for the project, which includes pre-stripping for mine development, construction, direct cost, indirect costs, TSF and contingency is estimated to be ZAR 2,263m. Detailed breakdown of the capital cost is available in Section 12 and is summarised in Table 14-2 below . Mining will be performed using contract mining and as such all initial mining capital costs, apart from pre-stripping, is included in the mining contractor's rate. Capital phasing has been based on the outcomes of the implementation schedule (as per the 2010 feasibility study) defined which allows for an overall construction period of approximately 2 years. Capital phasing (excluding TSF) has been benchmarked to similar completed projects by DRA equivalent to a capital split of 40:60 over the 2 year design, construction and comissioning timeline. Capital phasing relating to the TSF has been provided by Knight Piésold.

Table 14-2 Initial Capital Cost Summary

Description	Unit	Value	% of Total	2021	2022	2023	2024
Direct							
Mining	ZARm	95	4	26	39	30*	-
Process Plant	ZARm	967	43	387	580	-	-
Tailings$^{\beta}$	ZARm	279	12	135	65	30	49
Power Supply$^{\alpha}$	ZARm	181	8	72	108	-	-
Water Supply$^{\alpha}$	ZARm	216	10	86	130	-	-
General Infrastructure	ZARm	138	6	55	83	-	-
Total Direct Capital	**ZARm**	**1,875**	**83**	**761**	**1,005**	**60**	**49**
Indirects							
Land Purchase$^{\alpha}$	ZARm	14	1	14	-	-	-
Other Indirects	ZARm	197	9	79	118	-	-
Total Indirect Capital	**ZARm**	**211**	**9**	**92**	**118**	**-**	**-**
Contingency$^{\#}$	**ZARm**	**178**	**8**	**91**	**86**	**-**	**-**
Total Initial Capital	**ZARm**	**2,263**	**100**	**945**	**1,210**	**60**	**49**

*Mining pre-development costs,

$^{\#}$Contingency allowance is exclusive of TSF,

$^{\alpha}$Costing supplied by ATPL,

$^{\beta}$Costing supplied by Knight Piésold

14.5. Operating Costs

The operating costs over life of mine include mine operations, process plant operations, general and administrative costs and tailings disposal and tailings management costs. The cost reported excludes the cost of capitalized mine pre-stripping. Table 14-3 shows the estimated total and unit operating cost by category over life of mine. See detailed breakdown of the operating cost in Section 13.

Table 14-3 LoM Operating Cost Summary

Area	Value, ZARm	Unit Cost, ZAR/t ore	% of Total
Mining	5,316	300	59
Plant	3,381	191	38
Tailings Handling$^{\theta}$	44	2	<1
G&A$^{\theta}$	240	14	3
Total Operating Cost	8,981	507	100

$^{\theta}$Costing supplied by ATPL

14.6. Metal Pricing and Revenue

No formal market study has been conducted during this revalidation exercise and it is understood that the marketing aspects relating to selling of a PGM concentrate are clearly understood by ATPL. Project revenue is based on producing a saleale PGM concentrate with pay metals being Pt, Pd and Au using a pricing profile provided by ATPL. Metal prices used in the economic model is presented in Table 14-4.

Table 14-4 Metal Pricing

Area	Unit	2023	2024	LT	Source
Pt	USD/ozt	1,094	1,117	1,101	ATPL
Pd	USD/ozt	1,556	1,414	1,306	ATPL
Au	USD/ozt	1,601	1,499	1,415	ATPL

Based on the projected revenue and metal pricing the major revenue driver is palladium contributing 53% of total pay metals. An overview of the revenue breakdown over life of mine is presented in Figure 14-4 below.

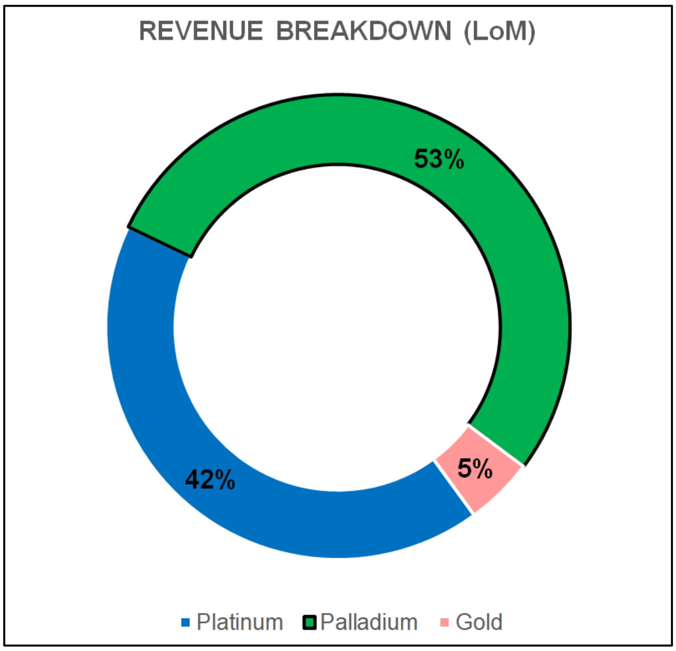


Figure 14-4 Revenue Breakdown (LoM)

14.7. Net Smelter Return

Revenue has been discounted to account for refining costs applicable to an off-take sale agreement for a PGM concentrate. A payability factor of 82.5% of generated revenue has been included and assumes producing a product below penalty or rejection limits. This factor has been supplied by ATPL and applied in the economic model.

14.8. Sustaining Capital

A lump-sum capital allowance of ZAR 40M during year 7 of operation has been included to cover costs associated with the mining contractor renewal / fleet replenishment. The economic model includes an allowance for annual sustaining capital of 1% of capital expenditure for the process plant.

14.9. Salvage Value

No allowance for salvage is included in the model.

14.10. Reclamation and Closure

Environmental reclamation and closure costs have been based on costs provided by ATPL.

14.11. Royalty Tax

Project royalties have been based on South African legislation relating to the production of an unrefined mineral resource. Royalties payable are a function of gross sales and a royalty percentage. The royalty percentage is limited to a minimum of 0.5% and a maximum of 7%

applied to gross sales. Capital expenditure has been recouped through depreciation of capital assets over life of mine using a straight-line basis used in deriving earnings before interest and taxation. The formula used to calculate royalties payable was as follows:

$$Royalty\ (ZAR) = Gross\ Sales\ (ZAR) \times Royalty\ \%$$

Where royalty % was calculated as follows:

$$Royalty\ \% = 0.5 + \frac{EBIT}{Gross\ Sales \times 9} \times 100$$

14.12. Economic Outcomes

The economic analysis is prepared on a 100% equity project basis and does not consider financing scenarios. A 10% constant discount rate has been used in the analysis. An average throughput rate of 1,475,032 tpa producing 228,004 tonnes of concentrate is projected.

The initial capital outlay is estimated to be ZAR 2,263M, inclusive of pre-stripping for mine development, construction, direct cost, indirect costs, TSF and contingency. Operating cost drivers include mining and processing costs and are estimated to be ZAR 300 /t and ZAR 207 /t respectively over LoM. General and administration costs are estimated at ZAR 20M per annum. All-in sustaining costs are estimated at USD 771 /ozt 2E+Au over life of mine. The analysis has revealed a pre-tax Net Present Value (NPV) of ZAR 2,352M with an internal rate of return (IRR) of 27.8% and corresponding to a payback period of 3.1 years. The outcomes of the analysis are summarised and presented in Table 14-5 below.

Table 14-5 Base Case Economic Outcomes

Area	Unit	Value
Pre-tax		
NPV @ 10%	ZARm	2,352
NPV @ 10%	USDm	138
IRR	%	27.8
AISC	$/ozt 2E+Au	771
Payback Period (Undiscounted)	yrs	3.1

The resultant economic metrics were calculated from annual projected cash flows covering revenue, capital costs, operating costs, sustaining capital and royalties. A summary of the life of mine cash flows is presented in Table 14-6 below.

Table 14-6 Base Case Discounted Cashflow

Item	Units	Total / Average	Profile	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Revenue																	
Platinum																	
Total Platinum Gross Revenue	ZAR	**9,658,728,759**		-	-	701,997,766	985,573,868	897,166,517	929,279,504	940,251,689	933,413,403	635,471,462	776,335,008	784,891,700	706,744,212	665,637,363	701,966,266
Palladium																	
Total Palladium Gross Revenue	ZAR	**12,191,225,716**		-	-	852,325,247	1,059,166,823	1,031,062,822	1,086,042,566	1,036,564,785	1,025,528,466	1,063,770,483	1,203,652,466	1,087,568,024	947,284,066	899,781,195	898,478,774
Gold																	
Total Gold Gross Revenue	ZAR	**1,094,551,265**		-	-	94,874,697	123,364,988	56,866,926	69,624,728	114,931,035	142,927,017	66,355,592	86,390,638	106,113,066	84,445,748	62,342,842	86,313,987
Total Gross Revenue	ZAR	**22,944,505,740**		-	-	1,649,197,711	2,168,105,679	1,985,096,265	2,084,946,798	2,091,747,509	2,101,868,886	1,765,597,537	2,066,378,112	1,978,572,790	1,738,474,026	1,627,761,400	1,686,759,027
Concentrate Transportation Charges																	
Total Concentrate Transport Charge	ZAR	**(214,592,332)**		-	-	(14,335,092)	(19,673,144)	(18,879,189)	(19,800,812)	(19,847,871)	(19,906,691)	(16,515,023)	(19,375,916)	(18,604,743)	(16,381,005)	(15,355,510)	(15,917,336)
Smelting and Refining Charges																	
Total Smelting and Refining Charge	ZAR	**(4,015,288,504)**		-	-	(288,609,599)	(379,418,494)	(347,391,846)	(364,865,690)	(366,055,814)	(367,827,055)	(308,979,569)	(361,616,170)	(346,250,238)	(304,232,955)	(284,858,245)	(295,182,830)
Total Net Revenue	ZAR	**18,714,624,903**		-	-	1,346,253,020	1,769,014,041	1,618,825,230	1,700,280,296	1,705,843,824	1,714,135,140	1,440,102,944	1,685,386,026	1,613,717,808	1,417,860,067	1,327,547,645	1,375,658,861
Operating Costs																	
Mining Cost	ZAR	**(5,315,886,396)**		-	-	(366,687,880)	(476,305,865)	(515,260,116)	(542,997,613)	(468,263,075)	(448,850,275)	(426,458,509)	(448,753,335)	(442,754,761)	(450,016,143)	(421,173,692)	(308,365,132)
Process Plant Cost	ZAR	**(3,381,192,055)**		-	-	(272,813,666)	(289,637,992)	(286,863,217)	(286,828,165)	(286,201,646)	(287,646,198)	(278,441,517)	(278,017,004)	(285,991,829)	(271,764,349)	(282,398,084)	(274,588,388)
Tailings Handling Cost	ZAR	**(43,680,950)**		-	-	(3,501,575)	(3,765,399)	(3,721,658)	(3,718,631)	(3,708,153)	(3,731,867)	(3,588,779)	(3,574,165)	(3,707,988)	(3,478,802)	(3,657,236)	(3,526,697)
G&A Cost	ZAR	**(240,000,000)**		-	-	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)	(20,000,000)
Total Operating Costs	ZAR	**(8,980,759,402)**		-	-	(663,003,121)	(789,709,256)	(825,844,991)	(853,544,409)	(778,172,874)	(760,228,340)	(728,488,805)	(750,344,504)	(752,454,578)	(745,259,293)	(727,229,013)	(606,480,217)
Initial Capital Expenditure																	
Direct																	
Mining	ZAR	**(94,601,848)**		(26,000,000)	(39,000,000)	(29,601,848)	-	-	-	-	-	-	-	-	-	-	-
Process Plant	ZAR	**(966,528,906)**		(386,611,562)	(579,917,343)	-	-	-	-	-	-	-	-	-	-	-	-
Tailings	ZAR	**(278,760,047)**		(134,779,724)	(64,788,831)	(30,120,411)	(49,071,082)	-	-	-	-	-	-	-	-	-	-
Power Supply	ZAR	**(180,690,783)**		(72,276,313)	(108,414,470)	-	-	-	-	-	-	-	-	-	-	-	-
Water Supply	ZAR	**(216,124,823)**		(86,449,929)	(129,674,894)	-	-	-	-	-	-	-	-	-	-	-	-
General Infrastructure	ZAR	**(138,332,733)**		(55,333,093)	(82,999,640)	-	-	-	-	-	-	-	-	-	-	-	-
Total Direct Capital	ZAR	**(1,875,039,139)**		(761,450,621)	(1,004,795,177)	(59,722,259)	(49,071,082)	-	-	-	-	-	-	-	-	-	-
Indirects																	
Land Purchase	ZAR	**(13,537,500)**		(13,537,500)	-	-	-	-	-	-	-	-	-	-	-	-	-
Other Indirects	ZAR	**(197,013,514)**		(78,805,406)	(118,208,108)	-	-	-	-	-	-	-	-	-	-	-	-
Total Indirect Capital	ZAR	**(210,551,014)**		(92,342,906)	(118,208,108)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency (exclusive of TSF)	ZAR	**(177,722,826)**		(91,226,011)	(86,496,814)	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital	ZAR	**(2,263,312,979)**		(945,019,538)	(1,209,500,100)	(59,722,259)	(49,071,082)	-	-	-	-	-	-	-	-	-	-
SIB Capital																	
Total SIB Capital	ZAR	**(198,104,508)**		-	-	(3,424,545)	(13,698,178)	(13,698,178)	(13,698,178)	(13,698,178)	(13,698,178)	(57,698,178)	(13,698,178)	(13,698,178)	(13,698,178)	(13,698,178)	(13,698,178)
Environmental Rehabilitation																	
Closure Cost and Reclamation	ZAR	**(119,531,043)**		-	-	(1,712,500)	(301,388)	(452,082)	(452,082)	(452,082)	(452,082)	(452,082)	(452,082)	(17,694,342)	(39,940,829)	(46,620,716)	(10,548,778)
Royalties																	
Royalty	ZAR	**(888,341,677)**		-	-	(61,853,791)	(95,573,612)	(73,557,962)	(79,938,088)	(88,958,690)	(91,915,241)	(58,734,784)	(89,675,354)	(79,203,618)	(54,790,002)	(45,564,881)	(68,575,654)
Pre-Tax Cash Flow																	
Pre-Tax Cash Flow (EBITDA)	ZAR	**6,264,575,295**		(945,019,538)	(1,209,500,100)	556,536,805	820,660,525	705,272,016	752,647,539	824,562,000	847,841,299	594,729,094	831,215,908	750,667,092	564,171,764	494,434,856	676,356,033
Cumulative Undiscounted Pre-Tax Cash Flow	ZAR	**6,264,575,295**		(945,019,538)	(2,154,519,638)	(1,597,982,833)	(777,322,309)	(72,050,293)	680,597,246	1,505,159,246	2,353,000,546	2,947,729,640	3,778,945,549	4,529,612,641	5,093,784,405	5,588,219,262	6,264,575,295
Discounted Pre-Tax Cash Flow (EBITDA, 10%)	ZAR	**2,352,096,722**		(945,019,538)	(1,099,545,546)	459,947,773	616,574,399	481,710,276	467,334,906	465,443,753	435,076,646	277,445,512	352,516,687	289,414,660	197,738,762	157,542,183	195,916,251
Cumulative Discounted Pre-Tax Cash Flow (10%)	ZAR	**2,352,096,722**		(945,019,538)	(2,044,565,084)	(1,584,617,311)	(968,042,913)	(486,332,636)	(18,997,730)	446,446,023	881,522,669	1,158,968,180	1,511,484,867	1,800,899,527	1,998,638,289	2,156,180,472	2,352,096,722

14.13. Sensitivity Analysis

A sensitivity analysis has been conducted assessing the impact of variations in capital cost, operating cost and metal selling price. The results of the sensitivity analysis are shown in Table 14-6 below. Each variable is assessed in isolation to determine the impact on NPV and IRR.

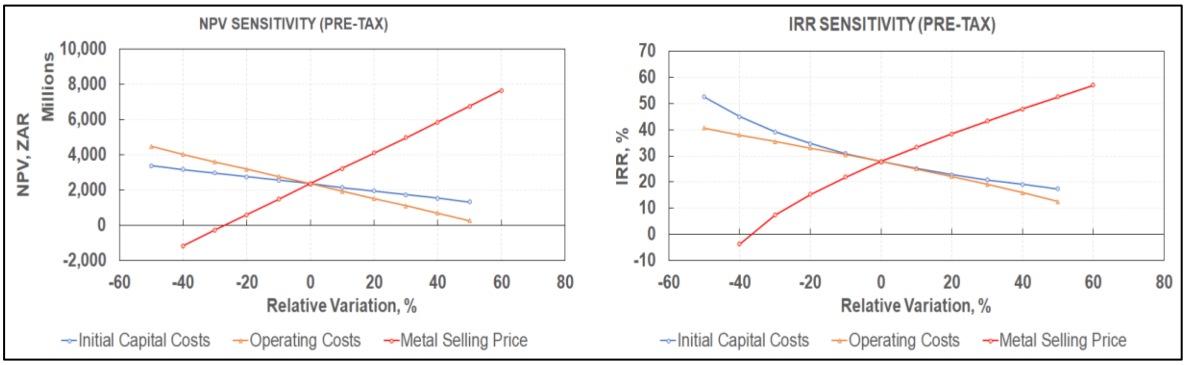


Figure 14-5 Sensitivity Analysis Summary

14.14. Upside Metal Recovery Scenario

Based on the historical metallurgical test work campaigns conducted, there is a potential to realise an improvement in metal recoveries modelled in the economic analysis. In order to quantify this potential benefit, an upside metal recovery scenario was modelled on an expected additional 4% improvement in overall recovery. The recovery improvement has been guided by extrapolated historical test work results. Further test work would be necessary in order to validate a potential recovery upside. The impact on project economics is presented in **Table 14-7** below.

Table 14-7 Economic Outcomes – Upside Metal Recovery

Area	Unit	Value
Pre-tax		
NPV @ 10%	ZARm	2,881
NPV @ 10%	USDm	169
IRR	%	31.3
AISC	$/ozt 2E+Au	747
Payback Period (Undiscounted)	yrs	2.8

15. PROJECT RISK SUMMARY

15.1. Mining

Due to Kalplats geology and metallurgical composition mining will require:-

- The geology of the seven deposits is highly complex, however continuity of the mineralised layers within the fault blocks has been established with a reasonable degree of confidence
- High levels of reef identification and grade control will be required to ensure the limits between waste and ore and high grade and medium grade ores can be determined before blasting to ensure the mining planning targets are met
- Management of dilution and ore losses will require integration into operations management and controls,
- The geotechnical parameters may vary from pit to pit depending on depths of weathering, localised faulting and other variables,
- Detailed short term planning will be essential to ensuring waste stripping and ore mining target and grades are achieved to ensure operating costs are controlled to budgeted levels

15.2. Metallurgical Testwork / Process Recovery

More testwork would be recommended to due to the unique nature of the Kalplats geology and metallurgical composition:

- complete the grade recovery profile of the ores in the target grade area
- optimize density feeds to the roughers
- to isolate whether it was the lower feed density or the NaSi that resulted in the improvement, to review the rougher mass pull conditions
- and to enhance the recovery testwork on the 10 to 40 m depth samples