PAULSENS GOLD PROJECT

"What Panel Beating & Metallurgy Have In Common"

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ABSTRACT

The Paulsens gold deposit in the Pilbara region of Western Australia was discovered by CRA in 1998. In 2001 St Barbara Mines Limited (SBML) made a successful takeover offer for Taipan Resources and then began an update of the previous Bankable Feasibility Study (BFS) undertaken by Minproc Limited.
This paper describes the history of the metallurgical testwork, plant design and initial development to optimise a project with complex metallurgical process characteristics.

The Paulsens ore is preg robbing, has cyanide soluble copper and nickel and is highly sulphidic. The ore is relatively hard and requires a very fine grind size of P80 53 microns. The metallurgical challenges have been met for this complex ore and the process flowsheet include aspects not commonly associated with a typical gold plant.
The latest study changed the mining plan, utilised a second hand plant and improved the robustness of the project economics.

1.0 INTRODUCTION & PROJECT HISTORY

The Paulsens Gold Deposit is located in the Ashburton geological province of Western Australia, 180 km west of Paraburdoo. St Barbara recently became a major shareholder in Taipan Resources and thus have the controlling interest over the Project.
The Paulsens Deposit was discovered in the early 1930s and a small underground mine, with associated battery and treatment plant, was established. Recorded gold production was approximately 12,000 oz.
Modern exploration at Paulsens commenced in the 1980s. Surface mapping at both prospect and regional scale, together with RC percussion and core drilling was undertaken by CRA Exploration, Hallmark Resources and Taipan Resources NL.

Location of Paulsens Gold Project in WA

2.0 GEOLOGY

● Regional Geology

The Paulsens Project area is located in the north-western closure of the Wyloo Dome. The dome comprises a doubly plunging anticlinorium, exposing granite and greenstone of the Archaean Pilbara Craton in the dome core, overlain by Archaean to early Proterozoic clastic sediments, dolomites, banded iron formation, basalt, and dolerites of the Fortescue and Wyloo Groups on the dome flanks.

The Paulsens Deposit is hosted within Archaean rocks (circa 2.7 Ga) of the Fortescue Group, which consists predominantly of tholeiitic basalt flows, but contains intercalations of fine grained, largely pelitic, sedimentary rocks.
The Fortescue Group rocks form the basement to the overlying Wyloo Group sedimentary sequence of Lower Proterozoic age (circa 1.8 Ga). Wyloo Group rocks form an angular unconformity with the older rocks.
Following deposition of the Wyloo Group, the area was compressed and deformed, producing the doubly-plunging anticline called the Wyloo Dome. The Wyloo Dome is the dominant regional structure of the area, and its associated structures (minor folds, axial-plane cleavage and reverse faults).
At least two suites of dolerite dykes cut basement rocks of the Wyloo Dome. The oldest and most numerous are the Billeroo Suite dykes. These are relatively narrow dykes (up to 50m thick) with irregular trends that cut across the Paulsens Deposit. A second suite of dykes – known as the Black Hill dolerites are less numerous than the Billeroo suite, but can be up to 200m thick, however, there are no known Black Hill dykes in the immediate vicinity of the Paulsens Deposit.

● Deposit Geology

The Paulsens Deposit is hosted within and along the margins of a north-north westerly trending, south-westerly dipping gabbro dyke intruded into metasediments of the Archaean Fortescue Group. The sedimentary sequence around the deposit has been informally defined to comprise several members. In stratigraphic order, they are:
Tin Hut Basalt (top);
Madang 'Breccia'; and,
Melrose Argillite (base local sequence).
The Paulsens Gabbro has intruded as a dyke at a high angle across bedding. The dyke averages 80m thick, strikes northwest and dips around 70° to the southwest. The gabbro can be traced along strike, at surface, for around 1100m. The southern termination of the dyke is caused by a bedding plane fault, similar to the Melrose Fault, whereas the northern boundary of the dyke has not been located, as it is overlain by the Tin Hut Basalt.

● Gold Mineralisation

Gold mineralisation occurs within a quartz-carbonate-sulphide vein system coinciding with an east-northeasterly dipping, north-north-westerly plunging kink in the gabbro dyke. A 50m high hill, capped with quartz, marks the surface expression of the vein system.
The Paulsens Deposit is hosted within a quartz vein that occupies the flat-dipping portion of a listric normal fault zone named the Melrose Fault. The Melrose Fault is a bedding-plane fault that cuts pelitic metasediments of the Fortescue Group. Massive quartz veining, up to 30-40m in thickness, occurs within the fault where it cuts and displaces a steep-dipping gabbro dyke. The fault probably formed during the period of crustal extension that initiated sedimentation in the early Proterozoic Ashburton Trough.

Gold mineralisation is associated with sulphide-rich portions (pyrite, pyrrhotite, minor arsenopyrite) of the vein. Such zones occur on the upper and lower contacts of the otherwise barren massive quartz veins, and along a number of hanging wall splays. The gold typically occurs as fine grains (av. 6-10μm) within and on the boundaries of the pyrite crystals.
Gold mineralisation is interpreted to have formed during late-stage movements on the fault zone.

● **Paulsens Resources**

The Paulsens resource is illustrated in the following table.

Category	Tonnes	Grade	GOLD
		Au g/t	OUNCES
Measured	1,440,581	4.26	197,305
Indicated	3,207	3.52	362,955
Inferred	732,710	3.1	73,027
Total	**5,380,440**	**3.66**	**633,287**

3.0 MINING

The mine plan has changed from a small open pit and underground operation (BFS) to a large open pit (LOP) followed by a possible underground operation.
The economics of the LOP have been improved by the use of large trucks and excavators. Owner operator mining studies have been undertaken and the design of the open pit is optimised by the geotechnical considerations.

The mine plan is governed by the large amount of waste to be stripped before feed can be delivered to the mill.

Detailed outcrop and interpretive geological mapping was carried out over a 6km2 area that included the proposed waste dump, tailings dam, plant, office and camp sites.

A review of the owner operator alternative indicated that there were significant savings available to Taipan with this option. Mining will be undertaken with two fleets:

1. The Demag (27m3 bucket) shovel owned by SBML with 3 to 6 of 145 tonne trucks for a period of 31 months, and
2. A 7m3 bucket backhoe type excavator with 2 to 5 of 145 tonne trucks dor 36 months.

High grade ore will be given milling priority and at the current gold price realised in the financial model, the cut of grade is 0.8 g/t/

The standing water table is 10 metres below surface and dewatering of the open pit will be required at a rate of 4,000 m3 per day to maintain wall stability and trafficable pit floor conditions. Grade control will be achieved using RC drilling in 30 metre passes. Drilling programmes have been designed to take into account the nature of the mineralisation in each of the three lodes as well as copper, nickel and carbonaceous material that could affect mill performance.

4.0 PREVIOUS FEASIBILITY STUDY

4.1 BFS

Ore samples from the “Paulsens Project” were first tested by AMMTEC for CRA in October 1989. The testwork was conducted over a four stage programme.

Stage 1 Preliminary Testwork Programme

Stage 2 Flowsheet Testwork Programme

Stage 3 Additional Testwork Programme

Stage 4 Variability Testwork Programme

Minproc undertook a Bankable Feasibility Study on behalf of Taipan Resources NL this was completed in July 1999.

- **Sampling**

The metallurgical samples were selected based on ore body representivity and drilled to test the main ore types for comminution characteristics and benchscale testwork with approximately 300 intervals for variability testing.

4.2 Laboratory Testing

- **Mineralogy**

The ore was found to be characterised by quartz carbonate, quartz chlorite schist with pyrite and minor chalcopyrite and arsenopyrite. The gold is predominately 10-45 microns and occurs attached to and within pyrite grains and low in silver.
Covellite and gersdorfite have been observed
The massive sulphide ore contains up to 85% pyrite.

● Comminution Aspects

The massive sulphide ore has a relatively high UCS. The wall rock is relatively softer.

UNCONFINED COMPRESSIVE STRENGTH (MPa)				
Sample Type	Number Tests	Minimum	Maximum	Average
Massive Sulfide	3	157	212	183
Quartz/Carbonate	5	54	122	69
Wallrock	5	27	75	51

The Crushing Work Index is relatively low indicating high crushing plant capacity and a low power requirement for crushing the ore.

BOND IMPACT CRUSHING WORK INDEX (kWh/t)				
Sample Type	Number Tests	Minimum	Maximum	Average
Massive Sulfide	5	3.9	11.6	7.0
Quartz/Carbonate	20	4.0	11.3	6.6
Wallrock	10	3.8	16.5	8.9

The Rod and Ball mill Work Indices are high indicating a high power requirement for grinding the ore. The ore is not abrasive which will ensure long crusher liner life.

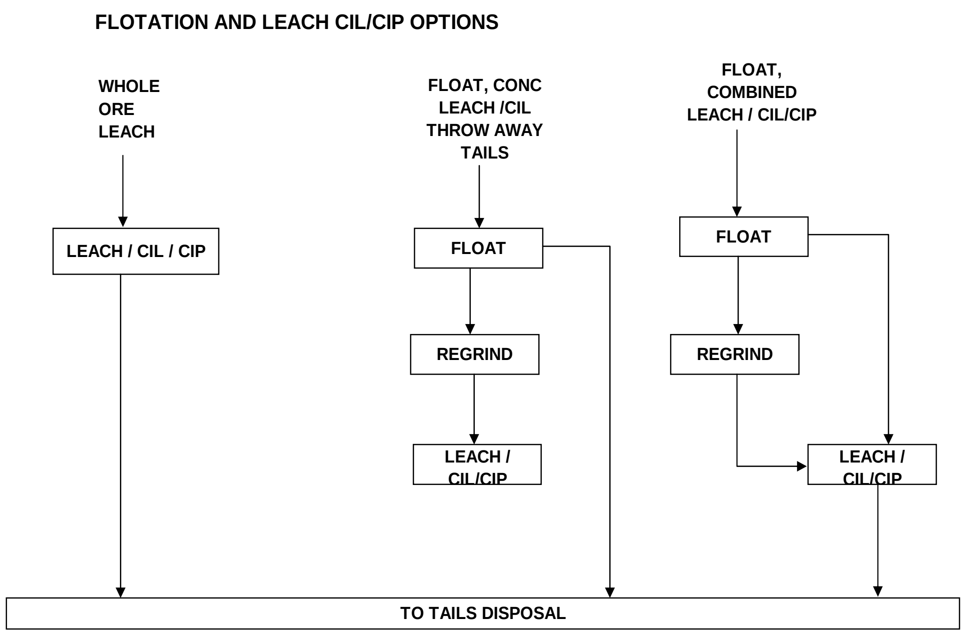
BOND ROD AND BALL MILL WORK INDEX AND ABRASION INDEX RESULTS							
Sple. Type	Bond Rod Mill Work Index			Bond Ball Mill Work Index			Abrasion Index
	F_{80} (µm)	P_{80} (µm)	Work Index (kWh/t)	F_{80} (µm)	P_{80} (µm)	Work Index (kWh/t)	
Massive Sulfide	9000	900	11.0	2700	86	14.2	0.326
Quartz	8500	950	14.7	2400	86	19.1	0.226
Wallrock	9000	900	20.1	2650	86	17.2	0.115
Stage 2 Composite	—	—	—	2450	90	18.7	—
Stage 3 Composite	7562	933	14.1	2274	59	17.6	0.229

The Autogenous Media Competency indicated that the ore readily breaks down with a low energy requirement. The Paulsens ore is amenable to SAG milling.

AMDEL AMC RESULTS SUMMARY						
Size Fractn. (mm)	Weight		Crushing Work Index (kWh/t)			
	(kg)	(%)	Minimum	Maximum	Average	Std Dev
+76	17.59	9.7	7.9	23.4	12.8	4.3
-76 + 51	11.03	6.0	8.3	24.6	11.8	4.3
-51 + 38	12.81	7.1	5.8	17.5	9.6	2.6
-38 + 25	8.86	4.9	5.4	13.8	8.1	2.1
-25 + 19	7.08	3.9	3.3	19.9	6.2	3.5
-19 + 12.7	9.83	5.4				
-12.7 + 6.3	13.74	7.6				
-6.3	100.56	55.4				

● **Flotation Aspects**

Flotation was evaluated on the basis of producing a throw away tail and a reduced CIP plant. The bulk of the ore is massive pyrite and minor pyrrhotite, therefore flotation while appealing as a concept it does not offer any significant process advantages.



The sulphide ore was floated at a P80 of 75um and a natural pH of 7 with copper sulphate for activation. The flotation tail was "throw away" and therefore no cyanidation was carried out on the flotation tailings. At a grind size of P80 150um the tail was 0.20g/tonne Au.

The flotation concentrate was ground to 130, 50 and 20 um and while recoveries of 93% to 95% were achieved the cyanide consumption was 3.3 to 4.4 kg/t. The fine grinding readily liberated reactive sulphides and, because of this, flotation was eliminated very early as a processing option.

FLOTATION RESULTS SUMMARY

					GRADE		RECOVERY	
Test	Product	Head	Tail	WT %	Au	S	Au	S
		Au g/t	Au g/t		g/t	%	%	%
GL3328	Rougher	5.36	0.08	19.74	26.8	22.1	98.8	98.4
GL3329	Cleaner	4.88	0.07	14.8	26.7	26.7	96.6	92.3

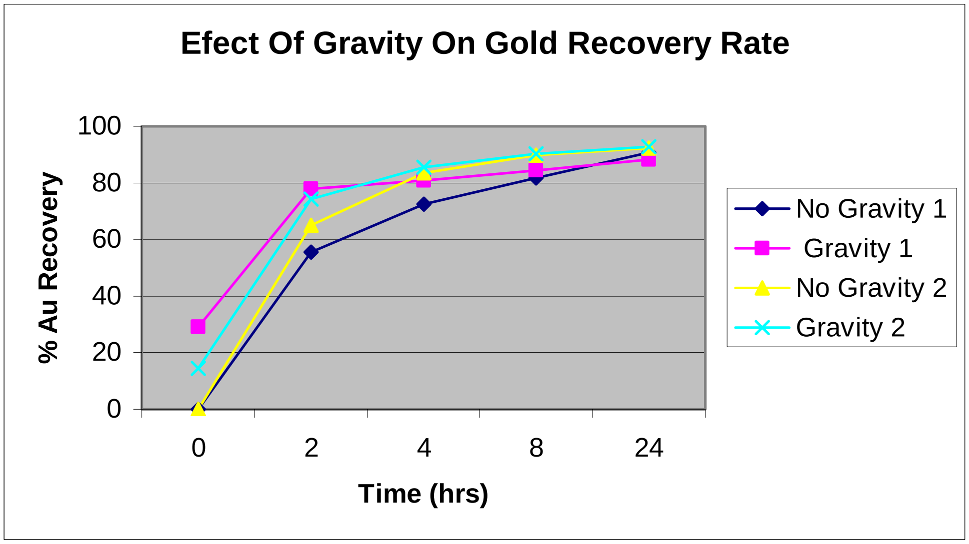
A summary of the flotation reagent conditions and further results follows.

STAGE 2 FLOTATION TESTWORK RESULTS SUMMARY

Flotation Test Details					Gold Grades		Flotation Concentrate				
		Reagent Additions						Grade		Recovery	
Lab Test #	P_{80} Grind (µm)	$CuSO_4$ (g/t)	A238 (g/t)	PAX (g/t)	Head (g/t)	Tail (g/t)	Wt (%)	Au (g/t)	S (%)	Au (%)	S (%)
H6576	75	25	25	75	7.76	0.154	19.1	40.1	71.9	98.4	99.7
H6577	75	25	0	95	7.36	0.102	18.4	39.6	42.0	98.9	99.7
H6578	75	0	25	95	7.17	0.094	19.2	36.9	40.4	98.9	99.7
H6580	106	0	25	95	7.60	0.108	18.8	40.0	43.5	98.8	99.7
H6582	75	25	25	95	7.33	0.088	18.5	39.1	42.2	99.0	99.7
H6579	106	25	25	95	7.17	0.126	18.1	39.1	43.8	98.6	99.7
H6581	150	25	25	95	7.21	0.110	17.7	40.1	45.2	98.7	99.7
Bulk	150	25	25	95	7.34	0.138	18.7	38.6	41.5	98.5	99.7
Arithmetic Average					7.37	0.115	18.6	39.2	42.6	99.7	98.7

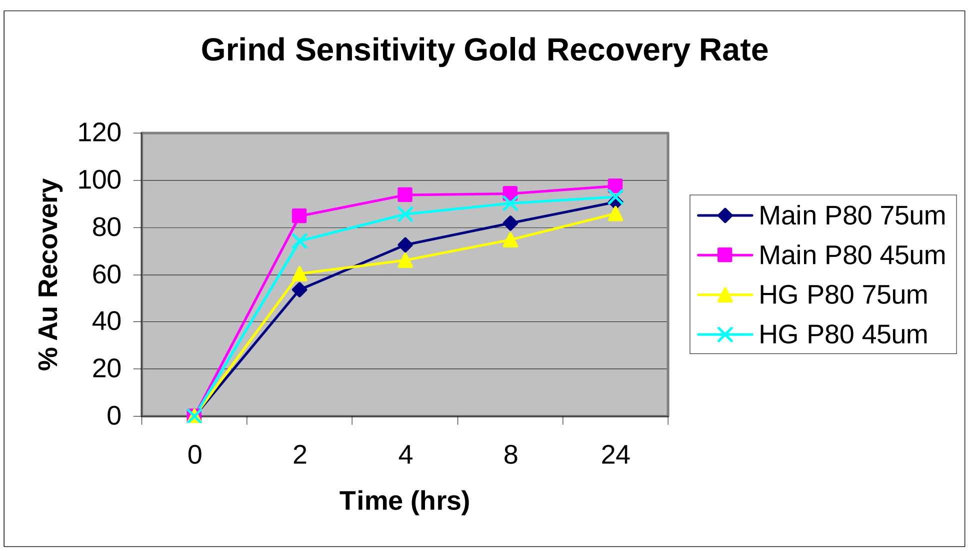
● **Gravity Recovery**

The overall difference with or without gravity was small, more than likely because the gold is not coarse. For the BFS gravity was not included in the flowsheet.
However because the Bounty plant had a gravity circuit and the advantages of including it outweighed the disadvantages the gravity circuit was retained. In addition the industry does not have a good record in predicting gravity gold recovery pre production.



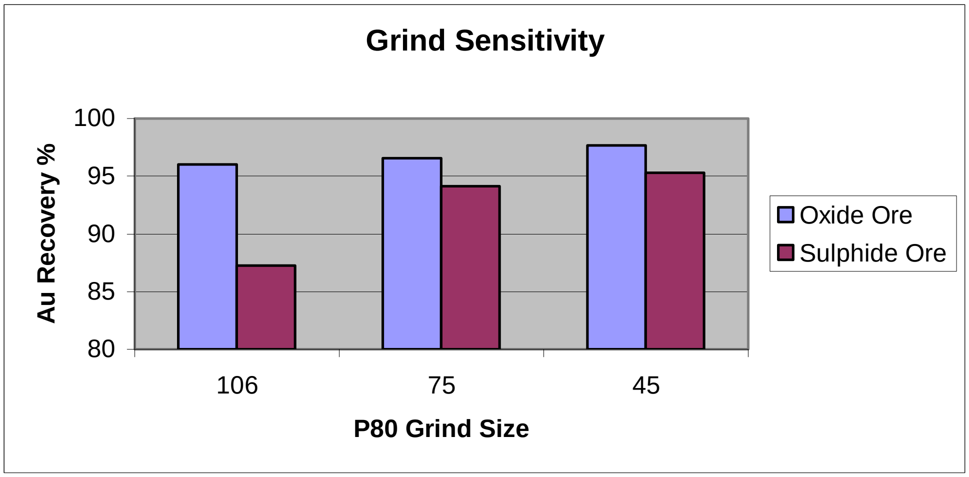
● **Ore Grind Sensitivity**

The gold occurs predominantly as inclusions in pyrite. A grind size P80 of 53um resulted in an additional 2-3% in gold recovery compared to P80 of 75um. The fine grind size necessitated a pre leach thickener to allow classification at overflow densities of 25-35% solids w/w. A grind size P80 of 53um was chosen as the basis of design.
A gold recovery of 94% was selected based on a mill feed grade of 8.0 -9.7 Au g/tonne.



The cyanide consumption for the oxide ore was typically 0.38 kg/tonne whereas for the sulphide ore the cyanide consumption decreased with grind size 1.16 kg/t, 0.83 kg/t, 0.78 kg/t.

The oxide ore is not grind sensitive. The sulphide ore is grind sensitive and requires a very fine grind size P80 of 53um.



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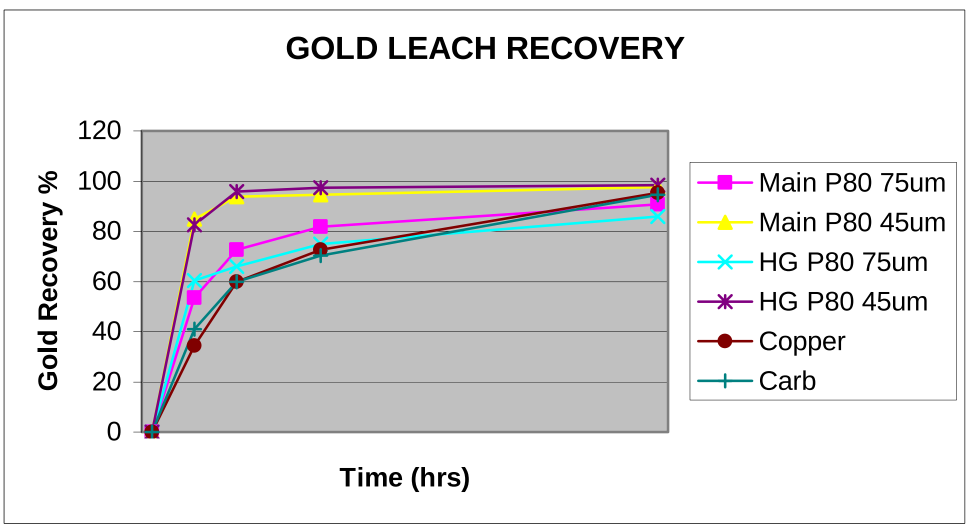
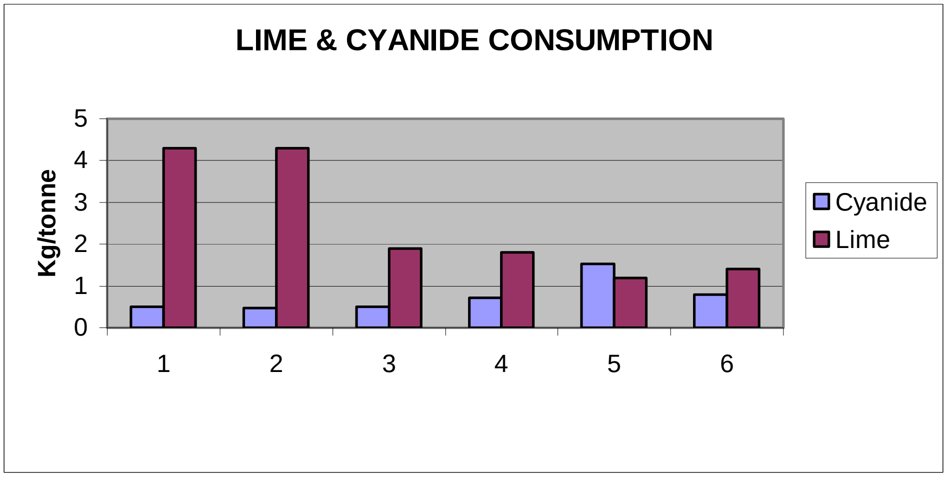




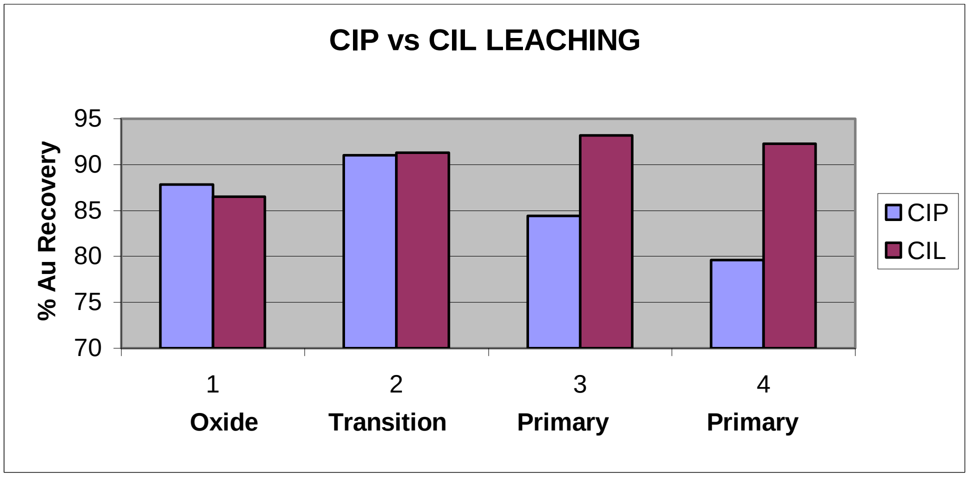
1. Main P80 75um 2. Main P80 45um 3. HG P80 75um 4. HG P80 45um
5. Copper ore 6. Carbonaceous ore

- **Preg Robbing Aspects**

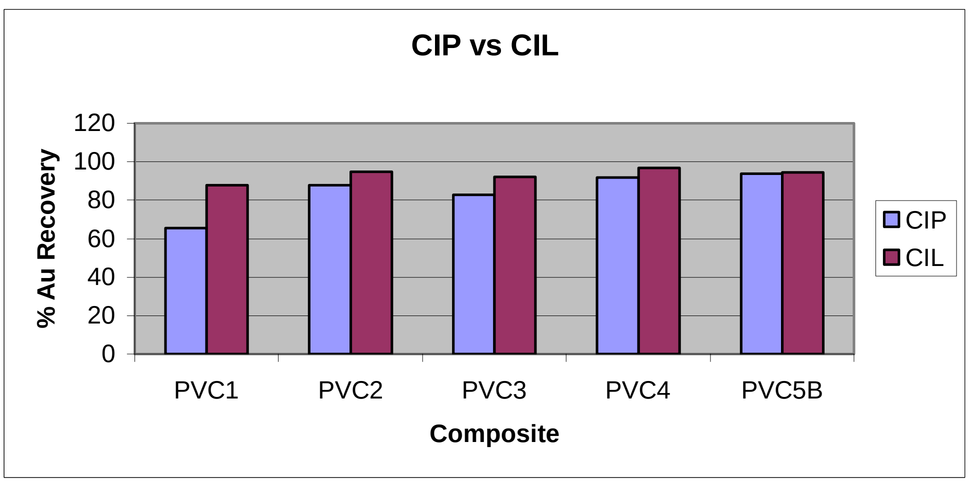
The preg robbing capacity of the ore was tested using a 10 ppm gold standard solution. The preg robbing phenomena was identified very early because of occasional very poor gold recoveries achieved with the early testing programme.

Preg Robbing Test

Time	Soln Grade	Gold Loaded	Calc Loading
mins	ppm Au	%	Au g/t
0	9.62	0	0
15	1.72	82.1	40
30	1.48	84.4	41
45	1.32	85.7	41
60	1.22	86.4	42
120	1.14	86.9	42



The results highlighted the improvement by the CIL process and the fact that the carbonaceous preg robber was associated with the primary ore contamination and its potentially aggressive nature.
The preg robbing carbonaceous ore is associated with the waste and not the ore however it will enter the mill feed as a result of dilution during mining operations.



CIL is superior in all cases and increased cyanide consumption was evident for all tests. A corresponding consistent decrease in lime consumption was noted at the same pH. All tests were at a P80 of 53um.

- **Reagent Consumption**

The below reagent consumptions developed from testwork were typical for the Paulsens ore i.e. low cyanide and lime consumption.

LIME & CYANIDE CONSUMPTION

Reagent	Minimum	Maximum	Average
	(kg/t)	(kg/t)	(kg/t)
Lime	1.09	1.54	1.33
Cyanide	0.56	1.07	0.77

The above reagent consumptions were typical for the Paulsens ore i.e. low cyanide and lime consumption.

● Assay By Size Leach Residue

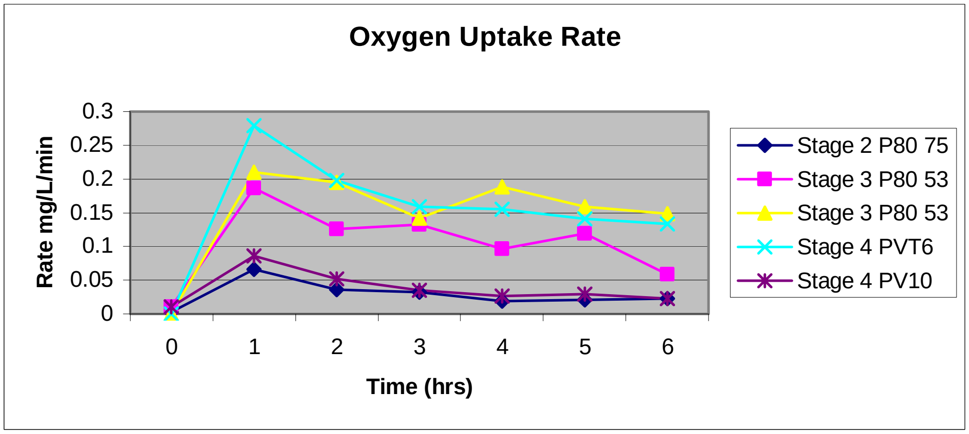
The results from the assay by size fraction procedures indicated a relatively uniform grade across the size ranges tested. with the minus 28 micron fraction the only consistently reported low grade residue.

STAGE 2 LEACH RESIDUE SIZE ASSAY RESULTS

Parameter	Test Conditions				
AMMTEC Test Number	H6569	H6570	H6571	H6573	H6572
Grind Size	75 μm	53 μm	45 μm	45 μm	45 μm
Cyanide Concentration (ppm)	500	500	500	1000	300
Size Fraction	**Size Fraction Gold Grade (g/t)**				
+ 75 μm	0.584	0.455	0.422	0.458	0.480
- 75 μm + 53 μm	0.740	0.524	0.486	0.494	0.480
- 53 μm + 38 μm	0.622	0.528	0.525	0.515	0.518
- 38 μm + 28 μm	0.535	0.505	0.318	0.318	0.545
- 28 μm	0.346	0.322	0.328	0.340	0.340
Calculated Residue Grade	0.528	0.409	0.382	0.389	0.400

● Oxygen Uptake Rate

The Paulsens ore is a massive sulphide ore consisting mainly of pyrite and pyrrhotite. The oxygen uptake demand is high enough to necessitate oxygen sparging.



- **Settling Tests**

The settling characteristics of the ore are excellent exhibited by rapid settling.

STAGE 3 AMMTEC BENCH SCALE SETTLING TEST RESULTS SUMMARY

Test Parameter	Unit	AMMTEC Test # H6769	H6770	H6771	H6772
pH		7.8	10.5	10.5	10.5
Flocculant Dosage	g/t	0	0	5	10
Flocculant Type		none	none	M333	M333
Final Density (% solids)	%	66.0	62.9	59.3	60.3
Time to Achieve 50% soilds	mins	30	19	8	8
Conventional Thickener Area @ 50% solids	m^2/t/day	0.245	0.151	0.063	0.063
Conventional Thickener Area @ 55% solids	m^2/t/day	0.286	0.183	0.119	0.118
Free Settling Rate (Conv. First 10 secs)	m/h	0.45	0.94	4.44	1.45
Supernatant Clarity		Cloudy	Cloudy	Fair	Clean

SUPAFLO DYNAMIC SETTLING TEST RESULTS SUMMARY

Test #	Feed Slurry Solids (t/m^2h)	Feed Slurry Density (% solids)	Flocculant Dosage (g/t)	Underflow Density (% solids) Measured	Underflow Density (% solids) Calculated	Overflow Clarity (ppm)
1	0.98	9.46	24.8	57.0	57.0	90
2	0.95	9.46	15.6	51.8	51.9	120
3	1.61	9.46	23.1	48.8	48.7	300
4	1.50	9.46	36.8	53.5	53.2	85

The settling characteristics of the ore are excellent.

● Viscosity Results

The viscosity results are characteristically very low, even at high densities, indicating that there will be no issues with inter tank screens or de rating of pump capacity.

SATGE 3 VISCOSITY RESULTS SUMMARY				
Slurry Density (% solids)	Measurement System	Shear Rate (S^{-1})	Viscosity (cp)	Shear Stress (Pa)
50	Brookfield 'VL' (#1 Spindle)	1.25 2.51 6.27 12.5	13 34 18 14	
	Bholin Visco 88 (System 6)	67.0 118.5 208.3	30 40 59	1.9 4.7 11.8
55	Bholin Visco 88 (System 6)	67.0 118.5 208.2	40 53 72	2.8 6.5 15.1

● Adsorption Constants

Sequential batch CIP tests were undertaken to determine k and n, the Fleming adsorption constants. The results indicated very fast adsorption.
The CIL circuit was originally based on achieving 6,000 Au g/t on loaded carbon but later reduced in light of the copper and nickel loading onto the carbon.

FLEMING CONSTANTS

Stage	Test	% w/w	k	n	Loading
					Au g/t
2	H6586	40	141.5	0.639	3327
3	H6768	55	149.6	0.665	5603

● Variability Testing

The variability testwork programme included tests under the standard leach conditions and grind size established from the previous programmes. Most of the samples exhibited relatively fast kinetics, with the majority of the gold leached within the initial 8 hours. Cyanide consumption was moderate between0.33 kg/t and 1.18 kg/t, depending on ore type. Lime consumption varied from 2 kg/t to 4 kg/t.

Copper dissolution varied between 4.8% and 38.5% over 24 hrs.

STAGE 4 VARIABILITY SAMPLES CYANIDATION TEST RESULTS

Parameter	Unit	Variability Sample Number					
		PVT 1	PVT 2	PVT 3	PVT 4	PVT 5	PVT 6
Sample Ore Type		Oxide	Trans	Prim	Prim	Prim	Prim
Calculated Head Grade	g/t	3.59	3.86	6.50	5.51	6.33	17.99
Calculated Residue Grade	g/t	0.439	0.374	0.537	0.255	0.986	0.844
Assay Residue Grade	g/t	0.440	0.348	0.505	0.164	0.985	0.860
Gold Extraction at 8 hr	%	79.5	62.2	71.3	72.1	82.6	92.5
Gold Extraction at 24 hr	%	87.8	91.0	92.2	97.0	84.4	95.2
Copper Head Grade	g/t	633	764	1380	1680	125	666
Copper Dissolution at 24 hr	%	12.6	38.5	15.2	4.8	16.7	10.5
Cyanide Consumed	kg/t	0.39	0.95	1.18	0.52	0.63	1.00
Lime Addition (@ 60% CaO)	kg/t	1.80	2.89	2.20	2.17	2.21	4.28
Parameter	Unit	Variability Sample Number					
		PVT 7	PVT 8	PVT 9	PVT 10	PVT 11	PVT 12
Sample Ore Type		Prim	Prim	Prim	Prim	Prim	Prim
Calculated Head Grade	g/t	6.34	2.56	25.34	5.41	2.97	6.36
Calculated Residue Grade	g/t	0.445	0.195	0.459	0.310	0.206	1.149
Assay Residue Grade	g/t	0.430	0.186	0.470	0.355	0.240	1.295
Gold Extraction at 8 hr	%	86.1	84.8	88.8	89.0	88.0	74.9
Gold Extraction at 24 hr	%	93.2	92.8	98.2	93.4	91.9	79.6
Copper Head Grade	g/t	735	503	171	149	248	245
Copper Dissolution at 24 hr	%	17.8	22.9	10.9	10.2	13.9	13.3
Cyanide Consumed	kg/t	0.98	0.62	0.75	0.33	0.35	0.43
Lime Addition (@ 60% CaO)	kg/t	2.55	3.02	3.17	2.90	2.16	2.28

5.0 FLOWSHEET DEVELOPMENT

A number of process options were considered based on:

• Flotation at a coarse primary grind size, followed by regrind and cyanidation
• Direct cyanidation
• AG, SAG, SABC, ABC, Rod Ball, etc

Direct cyanidation was chosen based on a P80 of 53um.

• ability to treat oxide & sulphide ore at P80 53 um
• low risk
• simplified water management
• low operating cost and medium capital cost

● Comminution

A SAG Ball grinding circuit was chosen during the BFS. This was based on:

• low risk
• ease of expansion
• low capital & operating cost

The main rock types identified were:
• Massive sulphide ore
• Quartz carbonate ore
• Wallrock

The wallrock is more competent than the sulphide or quartz carbonate ore, but forms less than 15% of the plant feed.

Paulsens ore compared with all of the ores included appears to be a good SAG mill candidate. Resistance to breakage is indicated by the values A and b (JK Tech). The product A*b allows comparison with other ore types. Resistance to abrasion is indicated by ta , a smaller value of ta indicates greater resistance to abrasion.

JK SAG/Autogenous Mill Model Parameters

Ore				A	b		A*b		ta
White Foil upper				78.4	0.44		34.5		0.56
White Foil lower				82.7	0.36		29.8		0.26
Kanowna Belle				60	0.45		27.0		0.29
Kidston				43	0.98		42.1		0.17
Big Bell				50	0.7		35.0		0.3
Granny Smith				50	0.53		26.5		0.19
Red Dome				50	0.58		29.0		0.22
TMH				50	0.54		27.0		0.17
Paulsens				81	1.28		103.7		0.9

Resistance to breakage	A	b	independent
Product	A*b		allows comparison of ores
ta	ta		the smaller the value the greater the resistance

From an initial study three milling circuits were investigated. The operating costs of the circuits were obtained at P_{80} grind sizes of 106, 75, 53 and 45 micron. In addition, flotation versus whole ore leach was also investigated by obtaining the operating costs. From this study the most cost effective circuit was found to be a single stage crush followed by a SAG/ball mill grinding circuit. Subsequent to the comminution circuit the whole ore leach treatment of the ore was the most cost effective at a P_{80} grind size of 53 micron.

Design Criteria

Aspect	Unit	Quantity
Throughput	t/h	61.4
Crusher Work Index		
- Crusher design	kWh/t	11.8

- Mill design	kWh/t	11.8
Crushing work index rock size	mm	61
Rod mill work index	kWh/t	14.1
Rod mill WI P_{80} achieved	um	950
Ball mill work index	kWh/t	17.6
Ball mill WI P_{80} achieved	um	59
Abrasion index		0.325

- **Jaw Crusher**

A single toggle jaw crusher was chosen due to low abrasion index of the ore. The jaw crusher was sized assuming the largest rock will fit within 80% of the crusher gape.

Design Criteria

Aspect	Unit	Quantity
Maximum lump size	mm	700
Chosen jaw dimensions	mm	1066 x 1219
	Inches	42 x 48
Open side setting	mm	150
Capacity	t/h	420
Power required from Bond	kW	20
Motor power	kW	125
Maximum product size	mm	150

- **SAG Mill**

The power split between the SAG and ball mills was determined by sizing the equipment at a P_{80} of 150 um and splitting the power 45/55 between the mills. At this power split the mill transfer size is T_{80} of 1000 microns. By retaining this transfer size the SAG mill was sized. The product size was then reduced to a P_{80} of 53 micron and the BWI at a P_{80} of 59 micron was used for sizing the ball mill. Although the grind size of 59 micron was not as low as the required grind size, it was assumed the BWI would not significantly change.

Design Criteria

Aspect	Unit	Quantity
Inside shell diameter	m	4.27
EGL	m	2.7
L/D ratio		0.63
Percent critical speed	% Nc	72
Percent total charge	% vol	27
Percent ball charge	% vol	8
Maximum percent ball charge	% vol	12
Ball size	mm	120

F_{80}	mm	150
T_{80}	µm	1000
Rock specific gravity	t/m^3	3
Milling density	% solids	70
Efficiency of drive	%	98
Liner thickness (average)	mm	100
Liner material		Steel
Installed motor power	kW	650
SAG mill inefficiency		1.25
Uncertainty factor	%	10
Grate aperture	mm	15
Discharge screen aperture	mm	25

- **Ball Mill**

Design Criteria

Aspect	Unit	Quantity
Inside shell diameter	m	3.81
EGL	m	6.4
L/D		1.68
Percent critical speed	% Nc	72
Percent ball charge	% vol	37
Ball size	mm	50-65
P_{80}	µm	53
Rock specific gravity	t/m^3	3
Milling density	% solids	70
Circulating load	%	300
Efficiency of drive	%	98
Liner thickness (average)	mm	137
Liner material		Rubber
Shell loner thickness	mm	80
Lifter projection height	mm	120
Lifter width	mm	200
Installed motor power	kW	1400
Uncertainty factor	%	10
Circulating load	% C L	300

- **Cyclones**

A high circulating load of 300% was chosen due to the high ball mill work index and fine grind size. Warman have determined the size and number of cyclones using 250Cavex10 cyclones with 73 mm inlets, 54 mm spigots and 90 mm vortex finders.

Design Criteria

Aspect	Unit	Quantity
Cyclone diameter	mm	250
Number of cyclones required		5
Number of standby cyclones		2
Operating pressure	kPa	62
Cyclone feed density	% solids	55
Cyclone overflow density	% solids	31
Cyclone underflow density	% solids	74
Overflow product size	P_{80} um	53

● **Thickening**

Design Criteria

Aspect	Unit	Quantity
Diameter	m	8

Settling tests were undertaken using Supaflo's high rate thickener test rig. To achieve the nominated underflow density of 50% w/w solids using a 1.5 t/m^2/h settling rate, a 7.2 metre diameter thickener is required. An 8 metre thickener is selected as this is the nearest standard size thickener.

● **Leach Adsorption**

The CIL was designed at the head grade of 8 g/t and reviewed at head grades of 10 and 12 g/t. 94.7% of the gold leaches within 24 hours at a P_{80} of 53 micron, therefore a residence time of 24 hours was chosen for design. The gold leach kinetics are very fast, 90% within 4 hours; a representative model was unable to be produced using the 24 hour test data. To overcome this problem the 24 hour data point was ignored; leaching constants were obtained using all the other data points. As the ore has a preg-robbing component CIL is configured with seven adsorption tanks. Seven adsorption tanks are necessary to obtain an average stage efficiency of less than 60%.

Leaching testwork at densities of up to 55% w/w indicated a slower leaching rate; caused from lower oxygen levels in the slurries. It was more difficult to aerate therefore a CIL feed density of 50% solids has been chosen for the design.

Design Criteria

Aspect	Unit	Quantity
Design head grade	g/t	8
Feed rate to CIL	t/h	61.4
Operating feed density	% solids	50
Design leach data	Time (hrs)	Ore conc (ppm)

	0	7.65
	2	1.92
	4	0.74
	8	0.40
Leach constants		
- Ko		0.3337
- q		1.597
Residence time per tank	hrs	3.5
Live volume per tank	m^3	287

A triple contact test indicated slow adsorption kinetics as the carbon did not load significantly even though the carbon concentration was low. For the design the kinetic constants were lowered from those obtained in testwork.

Design Criteria

Aspect	Unit	Quantity
Carbon kinetic constants		
- k	h^{-1}	141, 149
- n		0.63, 0.66
Carbon loading	g/t of carbon	3327, 5503
- k	h^{-1}	100
- n		0.55
Carbon concentration		
- 8 g/t	g/l	16
- 10g/t	g/l	14
- 12 g/t	g/l	12.5
Carbon residence time (8 g/t)	Hrs/stage	79

A carbon loading of 8000 g/t, normally chosen for high head grades, was not appropriate for the design due to the poor adsorption kinetics. 6000 g/t was chosen as this gave a reasonable carbon concentration of 16 g/l. If the carbon grade only loads to the testwork minimum of 3327 g/t then a carbon concentration of 29 g/t would be required. This is considered to be within the limits of the CIL design. A barren grade of 100 g/t was used as the stripping water will be potable quality.

Carbon movement is based on recovering 2.5 tonnes of carbon per day within 8 hours. By recovering carbon over an eight hour period there is sufficient time available for recovering a second batch in the same day, in the event the CIL head grade increases and two strips per day are required.

- **Intertank Screens**

Design Criteria

Aspect	Unit	Quantity

Pulp flow rate	m^3/h	82
Pulp back Flowrate	m^3/h	21
Viscosity of pulp (2.5 s^{-1})	CP	34 (50% solids)
Screen superficial velocity	m/h	50

- **Stripping**

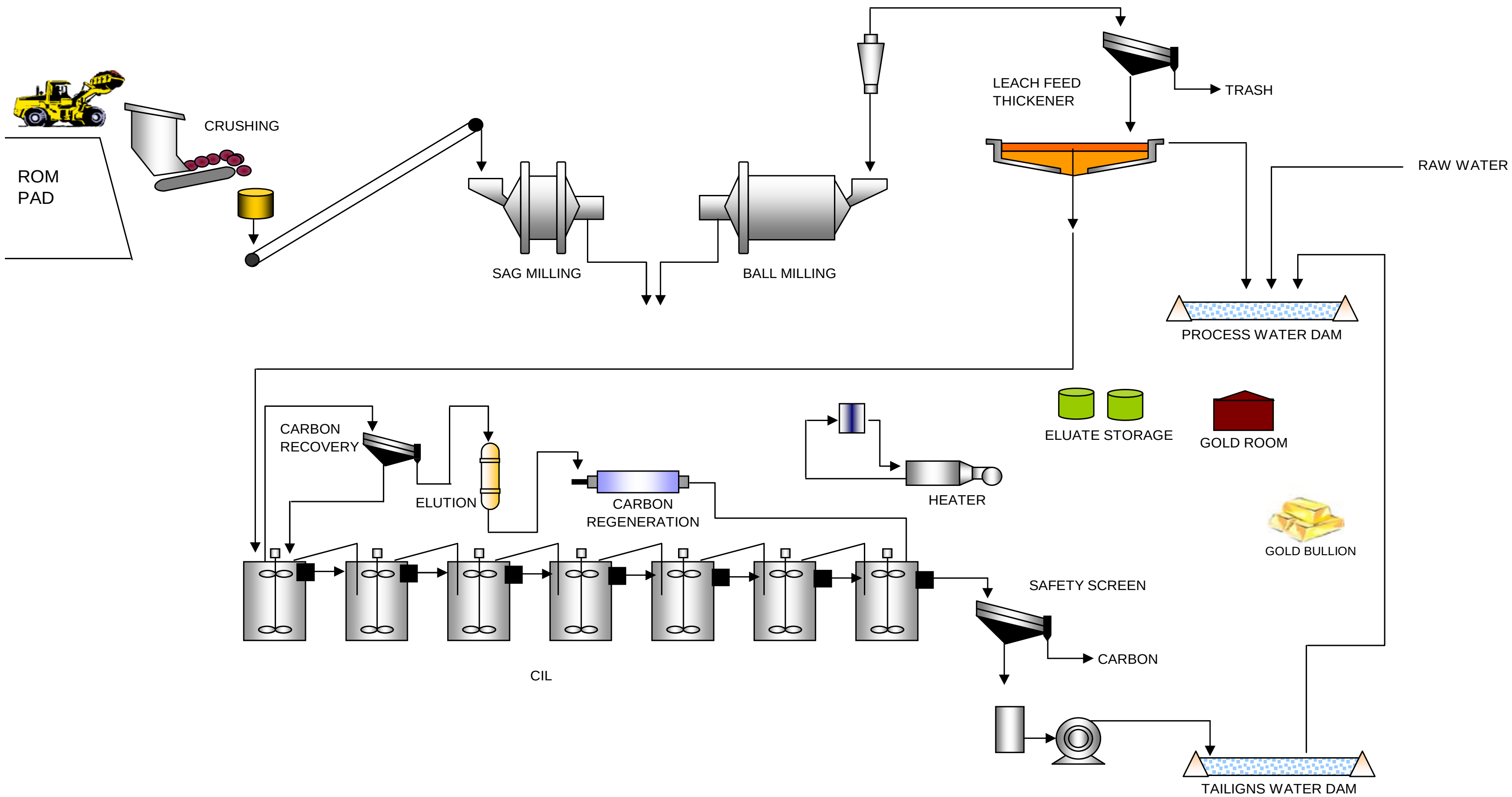
Design Criteria

Aspect	Unit	Quantity
Gold smelted (8 g/t feed grade)	Kg/week	96
Barring furnace size		A100
Smelts per week		2

At the ore head grade of 8 g/t and a carbon head grade of 6000 g/t a 2.3 tonne elution column is required. A 2.5 tonne column has been chosen. An eluate volume of 10 BV has been used in the design as this is expected to be the maximum volume required.

The electrowinning circuit was designed so that only one cell is required. If the head grade increases above 12 g/t a second Electrowinning cell and eluate tank are required. Due to the tight stripping schedule it was recommended that the second eluate tank be installed if the ore feed grade reaches 12 g/t. Knitted steel wool cathodes have been selected due to the ease of harvesting the gold and the lower cost as the knitted steel wool is reused and the mass of smelting reagents are reduced. A small drying oven is supplied to dry the gold after removal from the cathodes.

To be able to regenerate the carbon from every strip a 250 kg/h kiln is required. This kiln has sufficient capacity to regenerate carbon from each strip within 12 hours.


CRUSHING
ROM
PAD
SAG MILLING
BALL MILLING
LEACH FEED
THICKENER
TRASH
RAW WATER
PROCESS WATER DAM
CARBON
RECOVERY
ELUTION
CARBON
REGENERATION
HEATER
ELUATE STORAGE
GOLD ROOM
GOLD BULLION
SAFETY SCREEN
CARBON
CIL
TAILIGNS WATER DAM

6.0 OPTIMISATION OF THE BFS

Following on from the takeover of Taipan Resources NL St Barbara Mines assembled a study team to optimise the BFS study and resolve some of the technical issues identified as requiring further work.
This covered the geology, mining, metallurgy, environment and financial modelling aspects of the project.

6.1 Metallurgical Testwork

The bulk of the testwork had been completed during the Minproc BFS however further refinements were required to optimise the project economics.

- **Sequential Cyclic Carbon Loading Testwork**

The previous testwork did not assay loaded carbon for copper adsorption. Given that fouling of the carbon was a concern with recycled tailings return water it was decided to investigate this further. While bench scale testing does not indicate a potential problem previous experience with cyanide soluble copper and nickel in closed loop circuits indicated that the behaviour of the copper and nickel needed to be better understood. Cyclic carbon loading tests were undertaken to determine the amount of copper and nickel on the loaded carbon.

Sequential Carbon Contact

Leach Residue			Solution		
Wt (g)	Au (ppm)	Cu (ppm)	Vol (ml)	Au (mg/l)	Cu (mg/l)
7000	0.388	123	7000	3.66	39.1

Calculated Head		Extraction	
Au (ppm)	Cu (ppm)	Au	Cu
4.05	162	90.42	24.12

Fleming k = 108.19 hr^{-1}

Fleming n = 0.808

Loaded Carbon

Copper	845	862	ppm
Nickel	807		ppm
Gold	4018	3816	ppm

Three split samples of a composite Paulsens ore were leached under standard conditions to generate pregnant liquor for cyclic carbon loading under equilibrium conditions. The result was that the copper loadings were 845 ppm and the nickel loadings were 807 ppm from a feed containing 162 ppm of copper and 429 ppm of nickel.

The testwork confirmed a Fleming k =108.9 (equilibrium rate constant) and the Fleming n =0.808 which were in line with previous test results. The nickel solubility is lower than copper and both metals must be removed as the carbon is continually recycled.

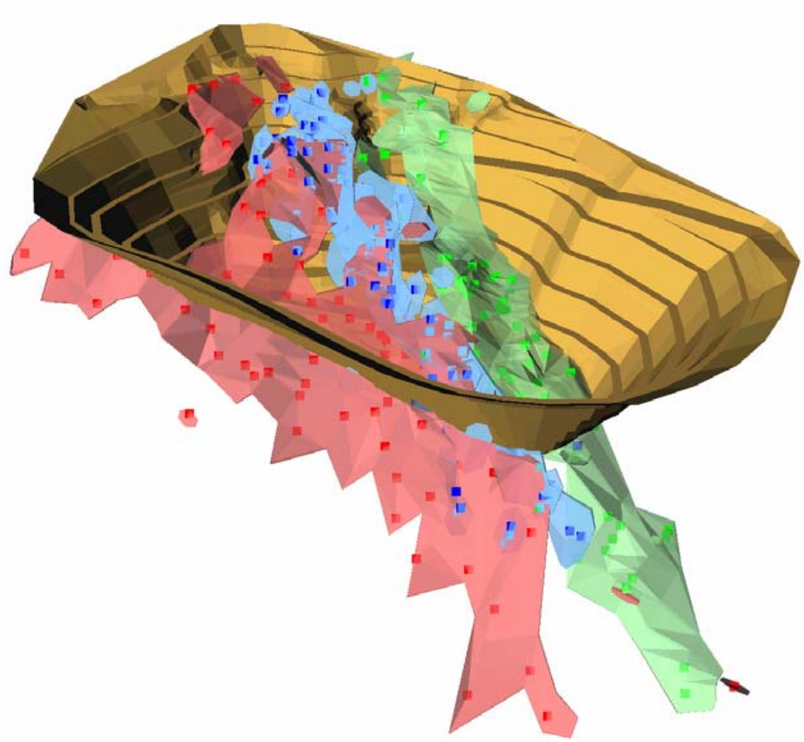
A strategy is now in place to measure the contaminants in the ore feed and blend these to achieve acceptable levels and in addition a mitigation process to remove these from the loaded carbon during the elution cycle.

- **Copper & Nickel Issues.**

Approximately 212 RC drill samples were assayed for total copper and nickel and at the same time the percentage of cyanide soluble copper and nickel were determined as a percentage of the total metals contained by bottle rolling the samples in a 1% cyanide solution.

Number Samples	% Cyanide Soluble Cu/ Total Cu	% Cyanide Soluble Ni/ Total Ni
39	31.8	4.6
69	36.6	2.6
58	37.3	4.1
45	40.7	4.0

The results were input into a block model of the orebody and the proposed open pit.



The red dots are cyanide soluble copper greater than 250 ppm. The blue dots are for cyanide soluble nickel greater 100 ppm.
The important finding from this work was that the copper and nickel levels did not increase with depth and it was likely that blending would minimise the impact of copper and nickel in the feed.

●Carbonaceous Ore Tests

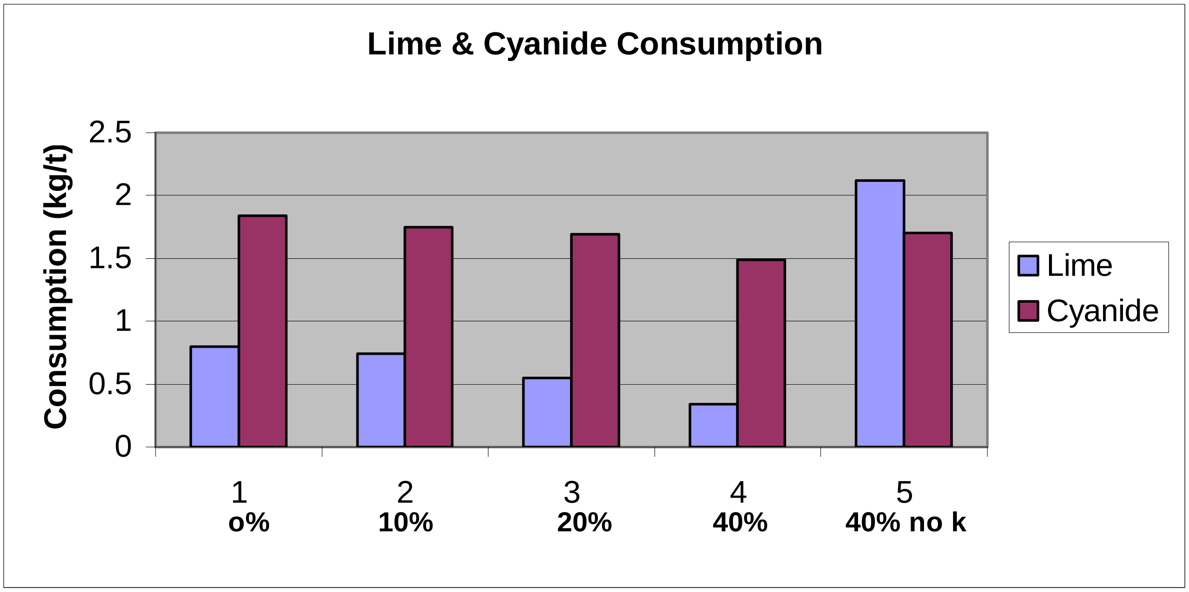
This testwork was carried out to determine the impact of increased carbonaceous ore as a result of mining dilution ending up as mill feed. In addition kerosene was evaluated as a passivation agent for the carbonaceous ore.

Composite	Blend (%)		Au (g/t)	Au (g/t)	Cu (ppm)	Ni (ppm)	Total C (%)	Organic C (%)
	Waste	ROM						
1	0	100	11.2	6.74	655	151	1.04	0.15
2	10	90	6.74	8.34	595	150	1.10	0.16
3	20	80	6.08	5.40	551	147	1.20	0.20
4	40	60	4.24	4.70	459	143	1.44	0.27

Composite	Blend (%)		Calc'd Head Au (g/t)	Leach Residue Au (g/t)	Gold extraction (%) @ Time (hours)				Reagent Consumption	
	ROM	Carbonaceous			2	4	8	24	Lime	NaCN
1	100	Nil	8.69	0.301	52.82	73.05	89.55	96.54	0.80	1.84
2	90	10	7.55	0.281	57.12	75.51	90.66	96.28	0.74	1.75
3	80	20	7.84	0.294	54.88	72.57	83.98	96.25	0.55	1.69
4	60	40	5.61	0.205	68.07	83.29	92.70	96.35	0.34	1.49
4 (no kero)	60	40	5.35	0.214	65.78	83.88	92.17	96.00	2.12	1.70

Composite	Blend (%)		Carbon Assay : Cu (ppm)				Carbon Assay : Ni (ppm)			
	ROM	Carbonaceous	2 hrs	4 hrs	8 hrs	24 hrs	2 hrs	4 hrs	8 hrs	24 hrs
1	100	Nil	1160	709	997	1510	39	34	29	80
2	90	10	1140	704	922	1542	37	25	29	87
3	80	20	1029	700	800	1226	35	21	29	75
4	60	40	761	553	623	864	24	19	23	64
4 (no kero)	60	40	743	557	613	740	57	27	24	44

This testwork confirmed that increasing waste dilution up to 40% could be tolerated without impacting on the gold recovery.



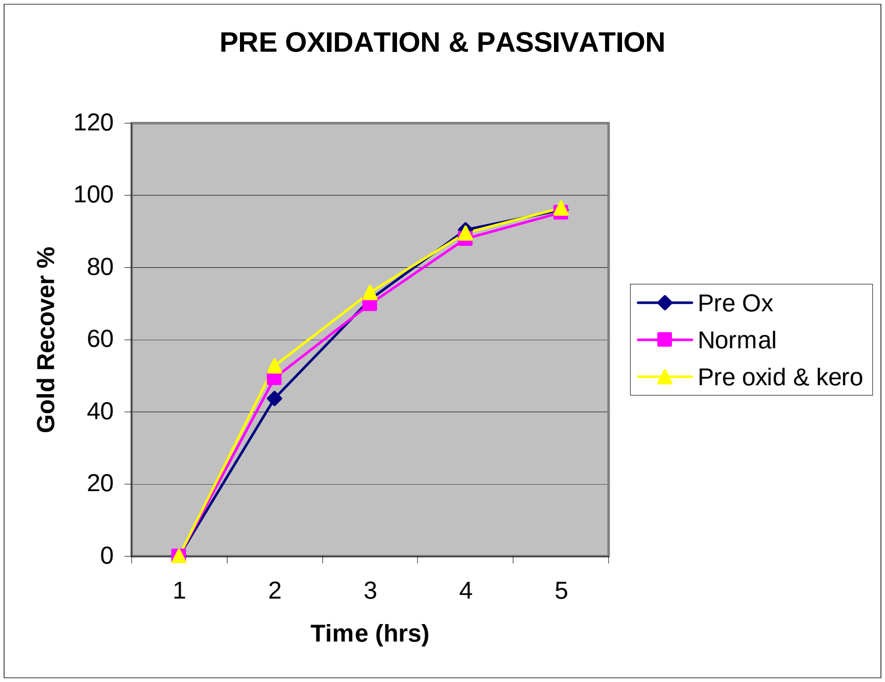
The kerosene affects the lime consumption which is not clearly understood.

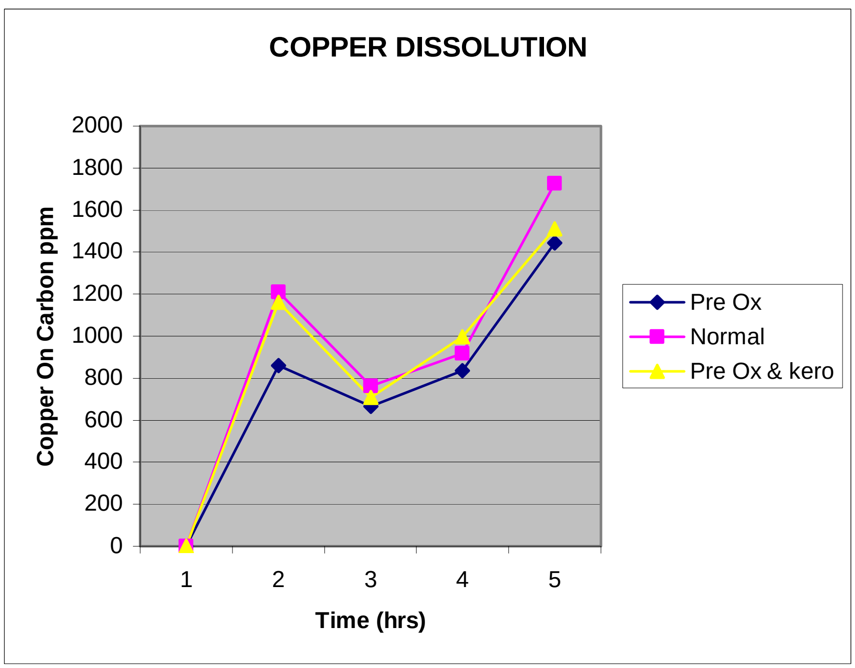
● Pre Oxidation

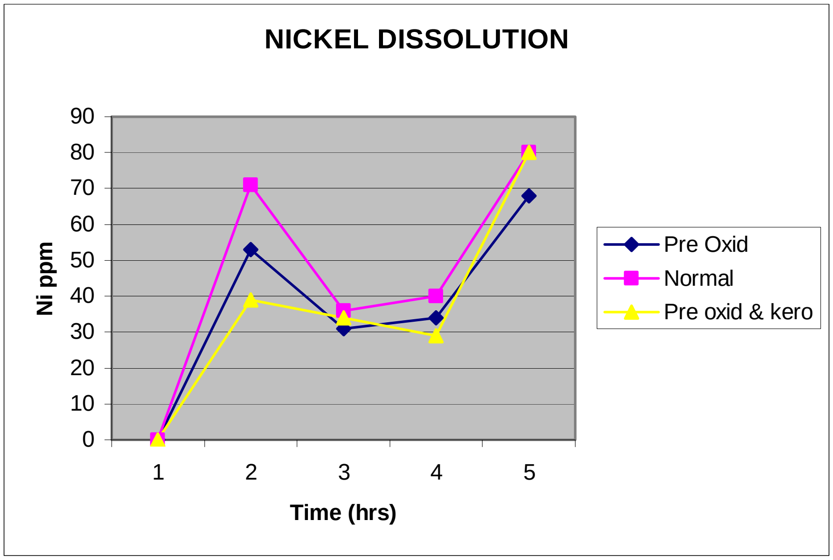
Pre oxidation was suggested because the ore contains pyyrhotite and the potential to reduce cyanide consumption and increase leach kinetics and overall gold recovery was considered high. The result was increased lime consumption (low cost) but reduced cyanide consumption and increased overall gold recovery.

Leach Type	Calculated Head Au (g/t)	Leach Residue Au (g/t)	Gold Extraction (%)				Reagent Consumption	
			2	4	8	24	Lime	NaCN
Pre-conditioned	8.96	0.367	43.76	71.07	90.43	95.90	3.01	1.89
Normal	8.92	0.425	49.38	69.92	87.95	95.24	2.64	2.06
A8253 Kero CIL	8.69	0.301	52.82	73.05	89.55	96.54	0.80	1.84

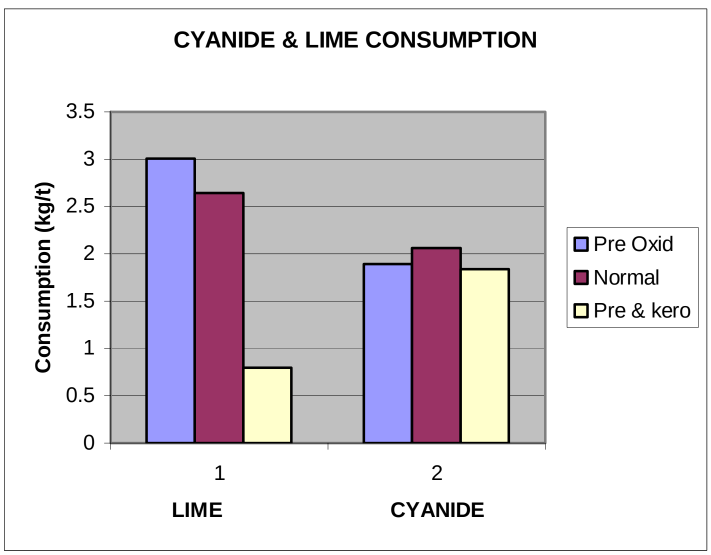
Leach Type	Carbon Assay Data							
	Cu (ppm)				Ni (ppm)			
	2 hrs	4 hrs	8 hrs	24 hrs	2 hrs	4 hrs	8 hrs	24 hrs
Pre-conditioned	860	667	836	1443	53	31	34	68
Normal	1210	764	917	1727	71	36	40	80
A8253 Kero CIL	1160	709	997	1510	39	34	29	80


PRE OXIDATION & PASSIVATION
Gold Recover %
120
100
80
60
40
20
0
1
2
3
4
5
Time (hrs)
Pre Ox
Normal
Pre oxid & kero


COPPER DISSOLUTION
Copper On Carbon ppm
2000
1800
1600
1400
1200
1000
800
600
400
200
0
1
2
3
4
5
Time (hrs)
Pre Ox
Normal
Pre Ox & kero



The tests were carried out CIL by removing carbon and adding fresh carbon. The copper and nickel assays are on carbon. The pre oxidation reduces the copper and nickel dissolution.



The pre oxidation increases the lime consumption (low cost), and reduces the cyanide consumption. Once again the kerosene reduces the lime consumption.

• Carbonaceous Ore Issues

With the Paulsens project some carbonaceous waste rock will enter the circuit via ore dilution. This material has the potential to compete for dissolved gold adsorption.

The passivation potential of kerosene was evaluated on samples of carbonaceous waste using a 10ppm standard gold solution.

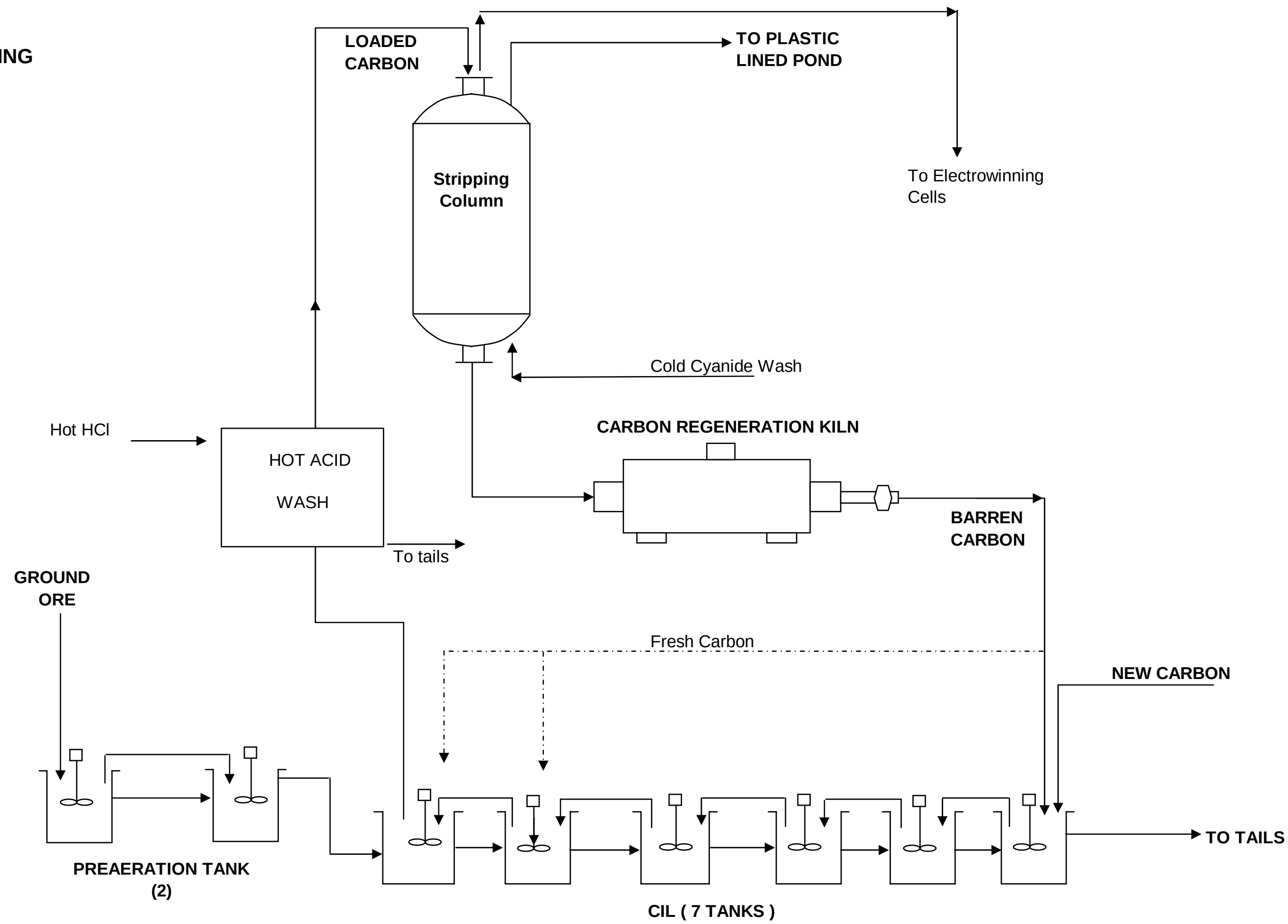
Samples of Paulsens ore were composited and then mixed with carbonaceous waste in proportions of 5, 10, 20 and 40% to establish the preg robbing capacity of increased amounts of waste.

The outcome from this work was that kerosene will passivate the preg robbing characteristic of the waste and CIL leaching eliminates the problem even at levels up to 40% waste dilution.

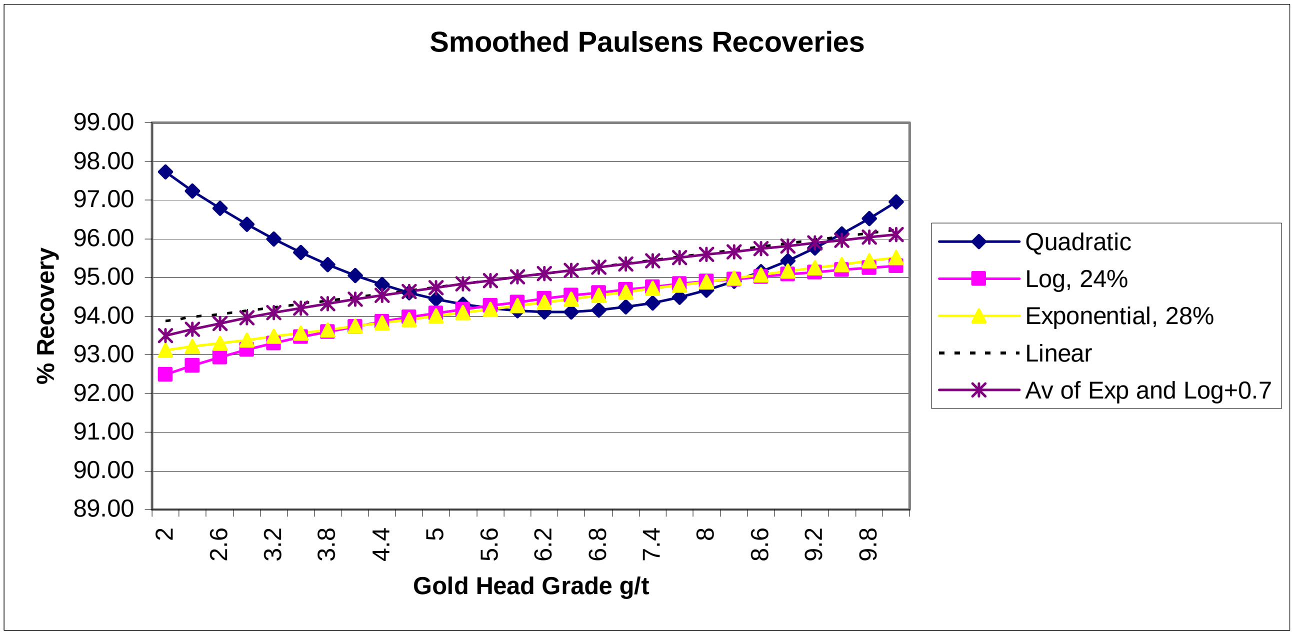
The addition of trace levels of kerosene may be justified and this can be evaluated after commissioning.

The importance of maintaining fresh carbon in contact with the pregnant leach solution also resulted in a change to the carbon management circuit. (see flowsheet)

CARBON MANAGEMENT TO MINIMISE PREG ROBBING



• Correlation Of Recovery With Head Grade



A correlation was developed based on CIL tests to allow input into the financial model and adjustment of the recovery levels to fit with changes in the head grade. The calculated gold recovery over the life of the project based on the mine schedule and head grade was calculated at 94.7%.

A fixed recovery was used in the first financial model. The model was very sensitive to gold recovery. A second iteration incorporated the fact that recovery would be related to head grade. As the grade varied significantly in the mine plan a regression equation was developed based on the CIL tests conducted. This second approach was deemed to be a more accurate representation.

6.2 Process Plant Selection

A number of used plants were assessed from the perspective of possible use. The costs of purchasing, relocating and refurbishing these plants were estimated in varying degrees of detail.
Plants reviewed in detail were.

• Fortnum	500, 000 tpa
• Mt McLure	2,000, 000 tpa
• Bounty	700, 000 tpa

A variety of other plants were considered:

Nimary, Cosmo Howley, Harbour Lights

The Bounty plant was selected after an exhaustive evaluation of the Bounty plant by a number of SBM and independent engineering consultants. In light of the study SBM recommended purchasing the Bounty plant.

7.0 THE BOUNTY PLANT

The Bounty plant was designed to treat hard ore and the 700,000 tpa plant capacity offered economies of scale over the 500,000 tpa rate previously considered.

- The Bounty plant is a good fit for the project.
- The gravity gold circuit will be retained.
- The rod mill can be close circuited with a DSM screen to control the rod mill transfer size.
- Pre aeration ahead of seven stages of CIL will ensure maximum leach gold recovery.
- Cyanide soluble nickel and copper will be removed from the loaded carbon using hot acid washing and a cold cyanide wash sequentially.
- A site based laboratory and assay strategy has been put in place based around relocating the Bounty assay laboratory.
- The operating costs, water balance and engineering warranties covering performance tests have been well developed.

• **Bounty Plant Process Description**

The plant has been repainted from a crème to a battleship grey and has deteriorated significantly from my last visit. The plant is now 14 years old and the hyper saline water has caused severe corrosion of the steelwork and resulted in concrete fretting. The plant is looking old and tired and has also suffered from lack of maintenance and repair because of the poor financial position of the owners.
The Bounty ore is very hard, not amenable to SAG milling and has a high abrasive index which requires careful consideration in optimizing a practical and flexible comminution flowsheet. The Bounty plant is currently three stages of crushing followed by rod mill ball mill a very common circuit twenty years ago.

- Rom pad with lay down capacity for separate ores
- Three stage crushing
- Fine ore bin storage and an emergency stockpile and feeder
- Two stage rod ball milling to a P80 of 63um
- Manual Knelson concentrator and concentrate clean up using a Gemini table and direct smelting
- A pre leach thickener
- A 2 tank pre aeration system followed by 4 leach tanks and a six stage CIL circuit using recessed impellor pumps for carbon transfer (CIP rather than CIL old thinking)
- A 2.5 tonne AARL split elution circuit
- Reagent circuit using lime, cyanide, oxygen and occasionally lead nitrate
- A PSA oxygen generation system with bulk liquid back up facility
- A conventional tailings system with central deposition and water recovery system

Process Control

The original process control system was a Fischer & Porter system and this was upgraded to a Cytec system when the rod mill was installed. The crushing plant and stripping plant had PLC programmed sequence stop/start.
The grinding circuit had feed rate control, sump level control, density control with a minimum number of control loops.
The process control system needs updating.

Run Of Mine (ROM) Laydown

The provision of separate lay down areas for underground and open pit ore in fingers allows blending for grade reconciliation and ore hardness. Each ore could be treated in campaigns to allow a full metallurgical balance and reconciliation to be undertaken.

Crushing & Screening

The ROM ore is fed by front end loader to a bin covered by a square 900mm grizzly. The oversize is removed and stockpiled for contract rock breaking.

The ROM bin ore is fed via a vibrating feeder over a vibrating finger grizzly with the fines by passing the jaw crusher. A double toggle jaw crusher was chosen because of the high unconfined compressive strength of the ore.
The fines and jaw crusher product are combined on conveyor 1 with the cone crusher product and transferred onto the inclined conveyor 2 which feeds the vibrating product screen. The screen undersize falls directly onto CV5 conveyor which delivers ore to the fine ore bin while the screen oversize top deck falls onto CV3 and feeds the secondary crusher while the bottom deck oversize falls onto CV 4 and feeds the tertiary crusher. Both the secondary and tertiary crusher products feed onto CV1 and then CV2 which feeds the product screen.
Conveyor 1 and conveyor 5 have weightometers to allow control and measurement of the federate. Conveyor 2 has a tramp metal magnet and a metal detector. Dust collectors were taken out because they failed after several years. The crushing plant is dusty and the steel work has corroded due to the hyper saline water.

Emergency Stockpile & Reclaim

The crushing plant will not have as high availability as the grinding circuit and provision needs to be allowed for continuous milling during crusher bowl and mantle change outs. The Bounty circuit has several options for emergency feeding and allows for ploughing off CV5. Reclaim from the emergency stockpile can also be by front end loader to a small bin with a vibrating feeder which feeds onto the rod mill feed conveyor.

Grinding & Classification

The two stage milling circuit has was selected for Bounty because the Bounty ore is not suitable for SAG milling and the high abrasive index limits three stage crushing because of bowl and mantle change outs as frequently as every three to four days.
The addition of the rod mill allowed higher mill throughput and reduced the need for fine crushing.
Paulsens may be able to use two stages of crushing followed by the rod ball combination based on the power required for grinding.

Primary Rod Mill

The primary mill is steel lined 3.2m by 5.6m Five Cail Babcock rod mill (750 kW) with large rods (97mm by 5.115m long) and the rod charge represents 39% by volume...
The mill is not close circuited and produces a sand consistency product.
The trommel screen oversize discharges into bunker. Provision could be made to return the scats to the mill feed or stockpile the scats.
The Bounty system has been designed to run the ball mill with the rod mill off line.

Secondary Ball Mill

The secondary mill is steel lined Boliden Allis 3.9m by 6.9m overflow ball mill close circuited with a nest of 8 cyclones including 2 standby cyclones. The cyclone overflow grind size is P80 63um and allow some flexibility to change this.
The mill operates with 40% ball charge using a mix of 65 and 80mm balls.

Gravity Gold Recovery

The original Bounty plant had no gravity circuit.
An automatic Knelson concentrator would be better with the concentrate being periodically pumped to storage tanks in the gold room.
The gold concentrate is then intensively leached and the leached solids returned to the grinding circuit. The pregnant gold solution will be electro won onto steel wool and smelted after calcining.
The Bounty system uses clean up of the concentrates on a Gemeni table followed by direct smelting. The Bounty plant has enough electrowinning cells to use intensive leaching.

Pre Leach Thickener

A vibrating Malco screen is used as a trash screen ahead of the thickener and this would be suitable for relocation. A 30 ft Jord conventional pre leach thickener is used to thicken the cyclone overflow before going to the CIL circuit. Only thirty percent of smaller plants have this facility and there is an additional capital cost however for the very hard Bounty ore where control of the grind size is important the thickener can be justified. If the thickener is removed the overflow density will be a constraint on the grinding overflow density. The worst outcome would be a situation where the overflow density required for the optimum grind size is 40% w/w and at 38% solids the carbon settles in the CIL tanks due to the low viscosity of the ore.
Hard ores similar to Bounty have high circulating loads and the only means of grinding finer is to reduce the cyclone feed density and hence the cyclone overflow density.
The conventional thickener is old technology and a HIRATE thickener of 7.2 m would achieve the same outcome. The Paulsens ore flowsheet includes a pre leach thickener because the grind size is P80 53um and the same arguments apply.
Provision can be made for bypass of the thickener for maintenance if required.

Leaching CIL

The thickener underflow is pumped to two pre aeration pre liming tanks which are very important in reducing the cyanide consumption for the sulphide ore by sparging with oxygen from the PSA system.
This is followed by three leach tanks where cyanide is added. The design is CIP whereas Paulsens needs CIL.

Six carbon in leach (CIL) tanks allow the ground ore to be leached with a residence time of greater than 24 hours at 50% solids w/w. The flow of slurry is controlled by the head difference between the first and the last tank and adjustable weir arrangement on adsorber tank 6. The tanks are baffled and have dual propeller Lightnin agitators to ensure good mixing and minimize short circuiting. Oxygen is sparged into each tank to ensure oxygen levels are maintained to ensure fast leach kinetics.
Cyanide addition is by needle valve adjustment to either tank 3, 5 or 6.
The carbon advance is by Warman recessed impellor pump and the intertank screens are cylindrical wedge wire with rotating wiper blades.
Provision has been made to bypass a tank for maintenance if required.
A vibrating tailings security screen is used ensure that carbon does not report to the tailings.

Elution, Reactivation & Electrowinning

Elution

Loaded carbon from the CIL circuit is recovered and washed on a horizontal vibrating screen deck. It is then washed with 3% hydrochloric acid in a column, to remove calcium magnesium and other contaminants that could interfere with gold elution.
The loaded carbon is then eluted using a slit Anglo strip.

Carbon Regeneration

The barren carbon is thermally reactivated in a horizontal Allis Chalmers regeneration kiln which is gas fired. The carbon is recycled to the CIL circuit. The kiln needs a new stainless steel shell.
A fine carbon removal screen is used after regeneration to remove the fines before returning the carbon to the CIL circuit.

Electrowinning

CIL Gold

The gold from the Anglo strip is electro won onto steel wool cathodes in a high efficiency Mintek type cell using stainless steel mesh anodes and rectifiers to provide the necessary direct current power supply. Bounty has three such cells.
This gold is accounted for separately.

Reagents

Quicklime is transported in bulk from Kalgoorlie or Perth and stored in a 250 tonne silo and fed onto the mill feed conveyor belt using a screw feeder.
Cyanide is transported from Perth as bulk liquid in isotainers and pumped into storage tanks on site for distribution and use. Facilities also include mixing for solid cyanide.
Caustic soda as a liquid is transported from Perth and stored in a storage tank for distribution and use principally with the Anglo strip.

Oxygen is generated on site using a Pressure Swing Adsorption (PSA) system and sparged into the leach tanks. A bulk liquid facility exists on site as back up.

Liquid petroleum gas (lpg) is supplied by tanker and used for the packaged boiler in the gold room and the carbon regeneration kiln.

Sampling

Two automatic samplers are included one for the leach feed and another for the CIL tailings.
A very good sample pressure filter sample station has been installed on the leach tank floor.

Power Supply

The Bounty power supply is from a transmission line from the plant site running adjacent to the road down to the road through to Hyden and ties into the 11kV line.
Western power charges a tariff of $0.092/kWhr. A step down transformer and switchgear is located at the plant site. Stores Area.
The 6MW diesel power station has no value because the engines have 25,000 hours up and even with a major rebuild they are old technology and two strokes which are not fuel efficient.
The power station cannot supply the total plant requirement and has not been used since the connection to the Western Power grid supply in June 1992. It has been used as emergency power.

Paulsens will generate power on site from a contract power station at a considerably higher operating cost.

Workshops

A steel framed sheet clad building which could be relocated is included on site.

Stores

A steel framed sheet clad building which could be relocated is included on site.

Plant Admin Building & Camp

A typical Atco type mill administration and office building is included which could be relocated. The camp at Bounty is old by current standards of design but would be suitable for relocation.

● **Proposed Layout**

The layout of the plant has been addressed to take into account the different site conditions at the Paulsens site.

The operability and maintainability have been addressed keeping the number of changes from the basic plant design to a minimum. A number of condition reports have been undertaken particularly with respect to major items of equipment and visual inspections of tank internals etc. Where necessary capital has been judiciously applied and refurbishment will be applied based on inspections of the equipment.

The final process flowsheet and changes made will result in a robust plant design with some operational flexibility

● **Operating Philosophy**

The decision was made very early in the piece to install a high level process control system (DCS) and utilise a small number of highly trained staff to operate the plant.
It was also agreed that minimal changes would be made to the Bounty plant unless a significant benefit could be achieved or there was a process reason necessitating the change.
A buffer stockpile of crushed ore between the crushing plant and the milling section will be retained in order to ensure the mill was not shut down because of a lack of ore.
It was also agreed that only equipment that was required would be utilised and excess tankage for example would be left behind.

The basis of refurbishment would be on an as required basis and the fact the plant was operating at the time of purchase meant there was a high degree of certainty that the plant could be made reliably operational at the Paulsens site.

● **Bounty Plant Unit Operations**

A comparison of the Paulsens and Bounty ores reveals the Bond Work Indices are very similar. The Paulsens ore is less abrasive, the CWI and RWI are also very much lower than for the Bounty ore.

Characteristic	Units	Bounty	Paulsens
UCS	Kpa	164-311	183
Crushing Work Index	kWhr/t	10.9-40	12.8-6.2
Abrasive Index	kWhr/t	0.58811	0.115-0.326
Rod Mill Work Index	kWhr/t	24.4	14.1
Ball Mill Work Index	kWhr/t	18.8	18.23
Specific gravity		2.9	3.0

• Crushing

The crushing plant capacity and final product size was evaluated with respect to specifying the optimum design rod mill feed size.

The crushing plant throughput was based on 2016 tonnes for a single 12 hour shift.

The benefits of this intermediate stockpile system are:

- Mill downtime due to lack of ore can be eliminated
- The live capacity of the fine ore bin will be improved by ploughing off
- Operational flexibility will be maximized with a large emergency stockpile (10,000 tonnes)
- Front end loader costs are minimized

The disadvantages are:

- Increased capital cost

● Grinding

The basic design premise was 700,000 tpa (86 tphr) at a grind size of 53 microns.

Concern was expressed that the rod mill transfer size could be different to that calculated and this could see the ball mill overloaded. Therefore it was proposed to install a DSM screen and close circuit the rod mill to ensure a suitable transfer size.

● CAVEX Cyclones

The existing Warman 15CE cyclones are old technology and Warman were asked to:

• provide the cost and details for replacing the cyclones with CAVEX design and

• confirm the optimum cyclone diameter for a product size of 53 microns

The outcome of the study was that the cyclone overflow density was far more critical in determining cut size and classification efficiency and therefore the existing cyclones were retained.

• Throughput & Optimisation

A number of process calculations and reviews were run with the aim being to provide assurance over throughput and grind size.
A number of scenarios were considered including crushing finer, reducing the rod mill transfer size and optimising the milling circuit cyclones.

The benefits of two stage milling are:

- Milling has a higher availability than crushing
- Milling is environmentally more acceptable than crushing (dust)
- Two stage milling allows better optimization and operational flexibility than a single mill
- Two stage milling is a technically lower risk than a single mill
- The design can incorporate features not possible with a single mill (DSM, scats crusher)
- Second hand mills could be used due to the common availability of smaller mills.

The disadvantages are:

- A single mill would have a higher capital cost however it would need to be a 2 compartment mill
- Ore variability could change the grinding balance with respect to capacity limitations

• Hi Rate Thickener Selected

A 12 metre Hi Rate thickener was selected ahead of the leach circuit rather than using the Bounty conventional thickener. This is up to date and superior technology.

This will allow the cyclone overflow density to be reduced to achieve a fine product size without compromising the leach feed density and residence time.

The benefits of the pre leach thickener are:

- Greater operational flexibility with the hard ore grind size and overflow density
- Increased residence time in the CIL circuit because of the higher density achievable
- Reduced cyanide, lead nitrate and lime usage because of the higher density
- Minimise carbon settling not uncommon on primary ore and densities < 40% solids w/w.
- Reduced raw water requirement
- Allow central tailings dam deposition (subject to testwork)
- Reduced environmental impact in the tailings dam

The disadvantages are:

- Increased capital cost

• Gravity Concentration

The Knelson concentrator has been retained in its current location and fed from a bleed of the cyclone underflow rather than the mill discharge as currently configured at Bounty.

Any additional recovery above the CIL recovery has not been included as the extent of gravity gold is indeterminate.

The current Bounty plant has a gravity circuit. It was considered beneficial to use this facility.

The benefits of gravity gold recovery are:
- Faster cash flow
- Higher overall gold recovery (coarse gold slow leach kinetics
- Reduced stripping requirement and lower overall cost/ounce
- Reduced risk of gold loss to preg robbers

Disadvantages are:
- Increased risk of gold theft

An automatic Knelson concentrator would be better with the concentrate being periodically pumped to storage tanks in the gold room.
The gold concentrate is then intensively leached and the leached solids returned to the grinding circuit. The pregnant gold solution will be electro won onto steel wool and smelted after calcining.
The Bounty system uses clean up of the concentrates on a Gemeni table followed by direct smelting.
The Bounty plant has enough electrowinning cells to use intensive leaching.

The benefits of the intensive leach are:
- Greatly improved gold security (theft coarse gold)
- Reduced labour requirement (tabling concentrates)
- Proven and accepted processing route (Sunrise Dam, Carosue Dam etc)

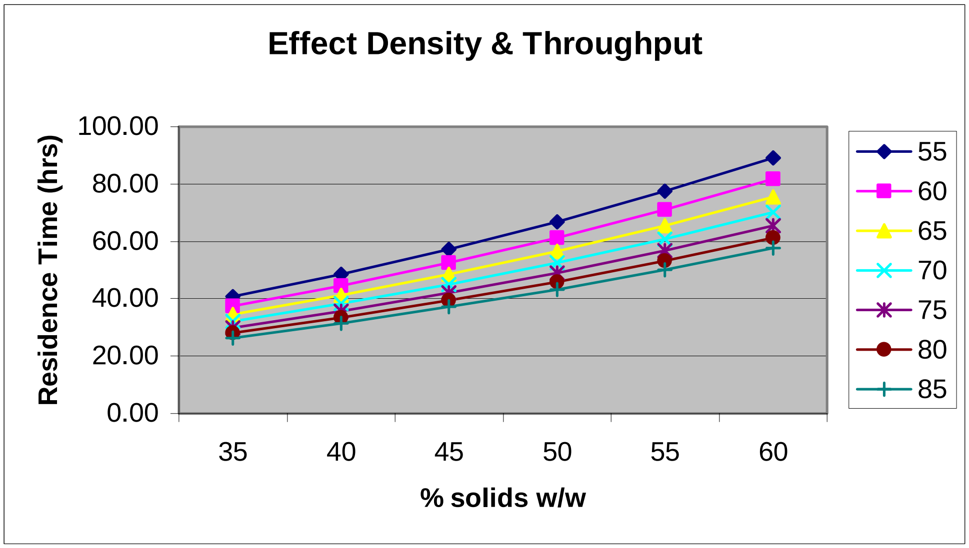
• Leaching CIL

Pre aeration (using oxygen) and pre liming tests conducted at AMMTEC indicated that cyanide consumption was reduced, the leach kinetics were faster and a higher overall gold recovery was possible (95.9%). The final circuit therefore includes two pre aeration tanks each with approximately four hours residence time followed by seven CIL tanks all relocated from Bounty (26.95 hrs residence time).

The testwork results also determined the level of copper on the carbon was 1443 ppm from a feed of 411 ppm while the nickel was 68 ppm from a feed assay of 149 ppm.

The nickel solubility is lower than copper and both metals must be removed as the carbon is continually recycled.

A strategy is in place to measure the contaminants in the ore feed and blend these to achieve acceptable levels and in addition there is a mitigation process to remove these from the loaded carbon as required.



The original Bounty plant was a CIP plant however for the Paulsens ore this will be converted to a CIL circuit.

The advantages of CIL compared to CIP are:

- Minimal capital cost
- Simplicity with fewer pieces of equipment
- Faster leaching of gold
- Absolutely essential for preg robbing ores

The disadvantages are:

- Higher gold inventory
- No opportunity for pre aeration

● **Stripping & Goldroom**

Stripping Column

The existing Bounty column will need a new neoprene rubber liner installed prior to reinstating it at the Paulsens site.

Hot Acid Wash

Hot acid washing will be utilised to remove nickel from the loaded carbon for every strip. Hot water will be provided via the heat exchanger plates coupled to the boiler. The acid wash liquor will be disposed of to the tailings.

Cold Cyanide Wash

A cold cyanide wash has been included in the flowsheet to remove copper from the loaded carbon. The cold cyanide wash liquor will be pumped to a small HDPE lined evaporation pond. If the copper levels are lower than anticipated then the cold cyanide wash will be bypassed until the copper loadings increase.

Regeneration Kiln

The regeneration kiln stainless steel shell tube needs to be replaced prior to relocation to the Paulsens site.

The advantages of the Split Anglo Strip are:
- Very efficient strip with low barren carbons
- Amenable to modularisation and automation
- Well proven system in common use in the gold industry

The disadvantages are;
- Potable water required
- Cyanide required for pre treatment

The advantages of horizontal kilns are:
- Carbon activity usually higher

The disadvantages of horizontal regeneration kiln are:
- Higher capital cost
- More manual rather than automatic operation
- Higher maintenance costs

● **Water Balance**

Typical Paulsens Water Analyses

Water Analyses								
Sample	**TDS**	**pH**	**SO4**	**HCO3**	**Cl**	**Ca**	**Mg**	**Na**
	ppm		ppm	ppm	ppm	ppm	ppm	ppm
Stage 3	1680	7.55	160	290	190	73.8	72.9	213

Water Balance

A water balance was undertaken to establish quantities and flows to be managed and the size of process water dams, tanks and pipelines. The amount of water in each circuit (tails, return, process, potable) was assessed.

Stripping Water

An analysis of the site bore water revealed that it could be used for stripping, reducing the size of the required RO plant.

Raw Water

166 m3/hr of water will be recovered from dewatering and 84.3 m3/hr will be pumped to the process plant with the balance used for dust suppression and 42.5m3/hr will be disposed of to a creek.

Tails Return Water

The tails return water is approximately 17% based on high evaporation rates and water being lost to retention and seepage.

Potable Water

The potable water requirement for the camp is 1.9m3/hr. The potable water will be produced from a small reverse osmosis unit. The raw water quality is excellent with a sample tested assaying 400 TDS.
It is planned to use raw water for stripping as a result significantly reducing the size of the reverse osmosis plant.

Excess Water Disposal

This is allowed for under the license conditions.

Process Water Dam

The process water dam will be a HDPE lined pond with sufficient capacity to feed the plant during a 24 hour shutdown of the dewatering pumps.

● **Process Control**

A preliminary control philosophy was developed to take into account the deficiencies and age of the Bounty process control equipment.

A new Citec Distributed Control System (DCS) has been selected for the relocated Bounty plant. The control loops for the crushing, milling, thickening and CIL areas were developed from plant inspections, discussions with operators, and basic P&ID's.

8.0 CAPITAL COSTS

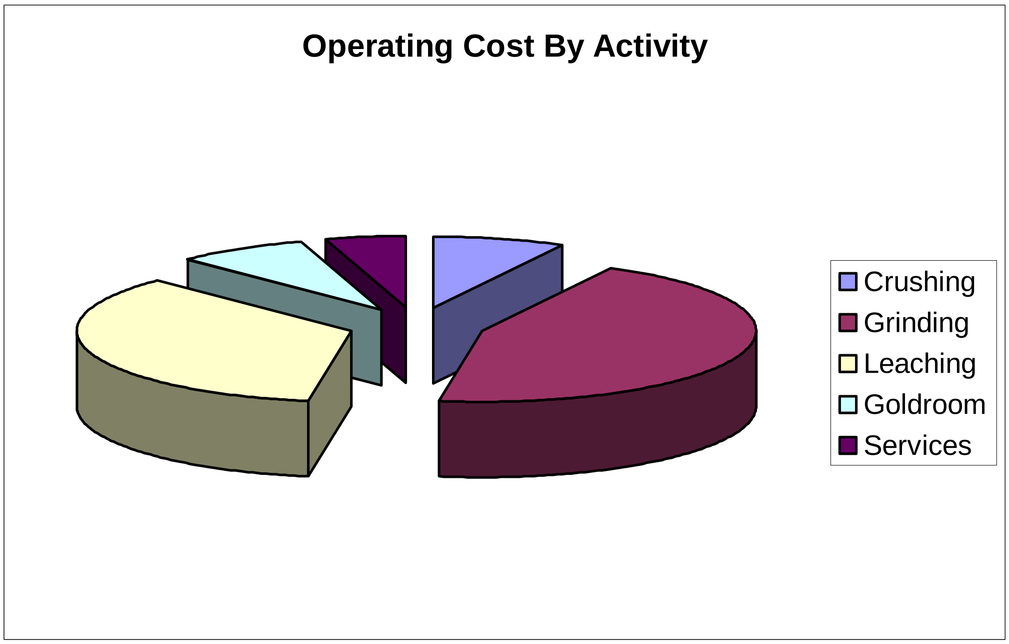
The estimate as summarised and tabulated in Table 2.1 is for a 700 000 t/a nameplate capacity Gold Processing Plant complete with supporting infrastructure such as access roads, airstrip, accommodation village, plant buildings, water supply, tailings disposal lines and electrical distribution. The total estimated value of the initial capital expenditure is $23.0 M.

Table 2.1
CAPEX ESTIMATE

Area No	Area Title	Estimated Cost ($)
00	Site Access and Preparation	1 553 170
01	Crushing	1 917 960
02	Grinding	1 051 730
04	Leaching and Adsorption	822 940
05	Desorption and Gold Room	224 270
06	Reagents	69 200
07	Air Water Services and Tailings	1 247 680
08	Plant Site Services	1 820 930
09	Plant Buildings	1 036 930
10	Mobile Equipment	795 190
11	Purchase & Relocate Bounty Mine Plant	5 134 700
13	Power Supply & Substation	84 870
14	First Fill & Spares	592 000
15	Preliminaries	1 152 470
16	Permanent Village (ex Bounty)	1 626 640
17	Construction Camp	525 900
	Total Direct Cost	**19 656 580**
	EPCM	1 497 380
	Total Bare Cost incl EPCM	**21 153 960**
	Accuracy Provision	1 314 680
	Fee	558 900
	TOTAL CAPITAL COST	**23 027 540**

9.0 OPERATING COSTS

The operating costs were re evaluated from the BFS and applied to the Bounty plant plus more recent reagent quotations**.** Operating costs are a very significant driver with the Paulsens project economics.



Summary	**Variable**	**Fixed**	**Total**	**%**
	$/tonne	**$/tonne**	**$/tonne**	
Crushing	$0.58	$0.95	$1.53	10.19
Grinding	$3.29	$2.43	$5.71	38.04
Leaching	$2.59	$1.34	$3.93	26.16
Goldroom	$0.56	$0.65	$1.21	8.06
Services	$0.36	$2.28	$2.64	17.55
Total	**$7.37**	**$7.65**	**$15.02**	

Reagents Re Quoted

The principal reagents were requoted by suppliers resulting in a substantial saving, particularly for cyanide based on both bulk shipment and the ability of AGR to now ship a paste product more cost effectively than a liquid.

This resulted in a small saving in operating costs.

Manpower

The manning numbers were reviewed based on a survey of existing gold plants and comparing the proposed manning with actual numbers at several operating mines. This review resulted in a reduction of manpower and a small saving.

Power

The power cost which is a major cost contributor was reduced because of the switch from diesel over gas power generation.

10.0 RISK ASSESSMENT

A process risk review was undertaken based on the following and will be used as a checklist to minimise process risk.

			Consequence		
Likelihood	Catastrophic	Major	Moderate	Minor	Insignificant
Certain	High	High	High	Significant	Significant
Likely	High	High	High	Significant	Moderate
Moderate	High	High	Significant	Moderate	Low
Unlikely	High	Significant	Moderate	Low	Low
Rare	Significant	Significant	Moderate	Low	Low

- Risks discovered as a result of reviewing the Taipan DFS information and visiting the Bounty plant

- Risks of relocating the Bounty gold plant equipment for the Paulsens gold project in the Pilbara.

- A risk comparison of the different ore comminution characteristics and differences between the Bounty flowsheet and the Paulsens Minproc Feasibility study flowsheet.

- A risk assessment of the infrastructure facilities.

- Risk factors with regards to Operating & Capital cost considerations

- Project schedule risk issues relocating the Bounty plant.

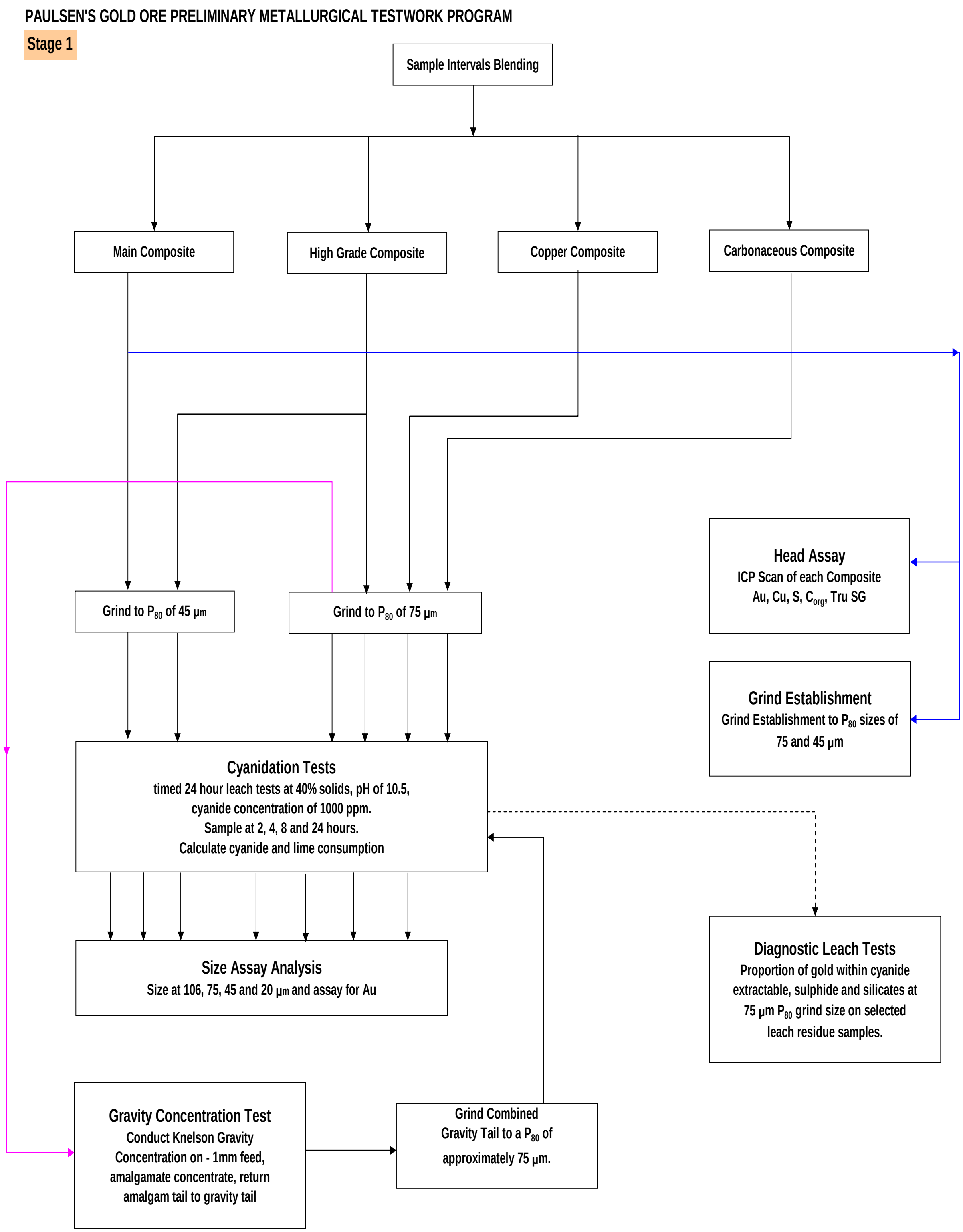
The Risk Assessment report addressed the project risks and the proposed mitigation steps.

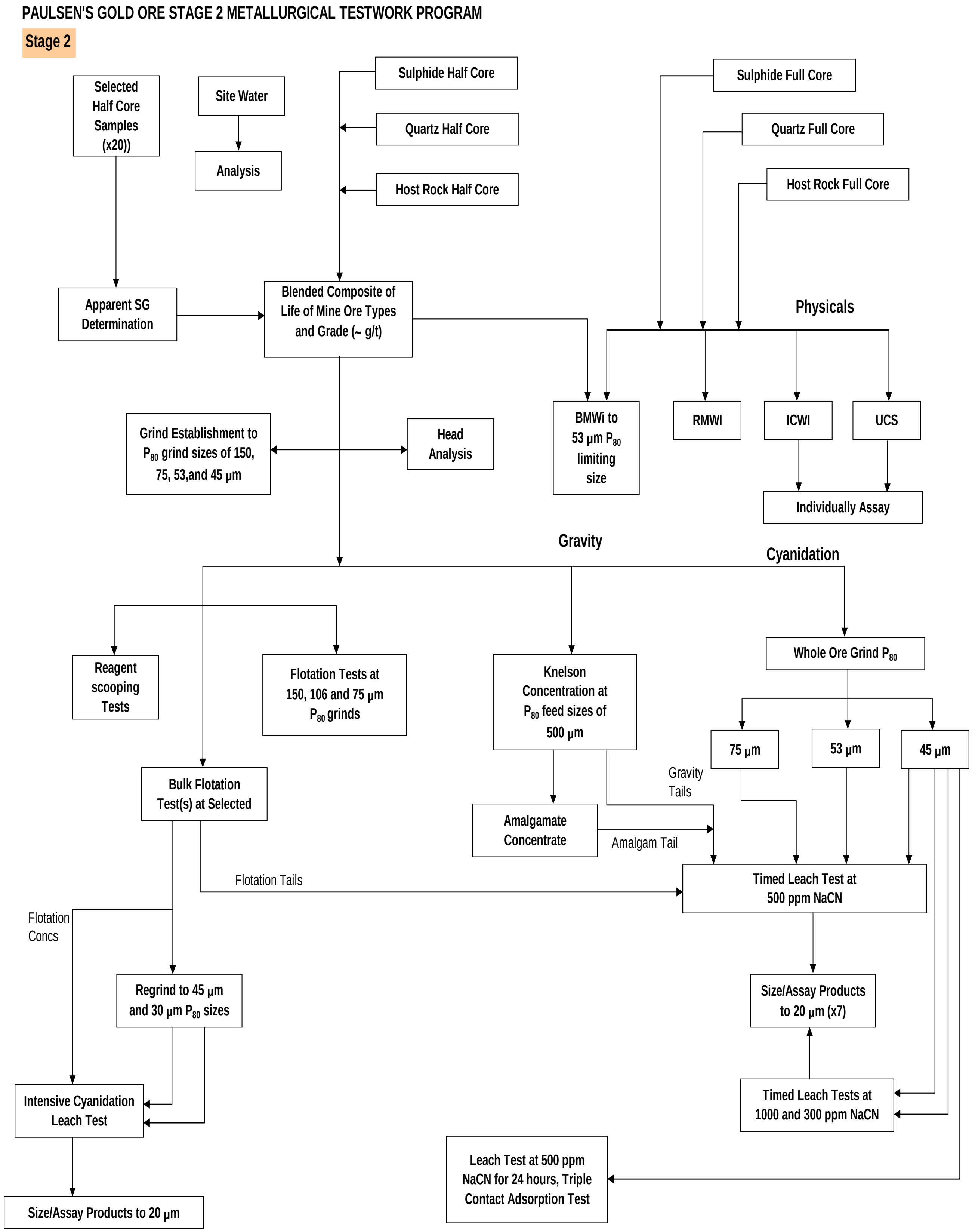
ACKNOWLEDGEMENTS

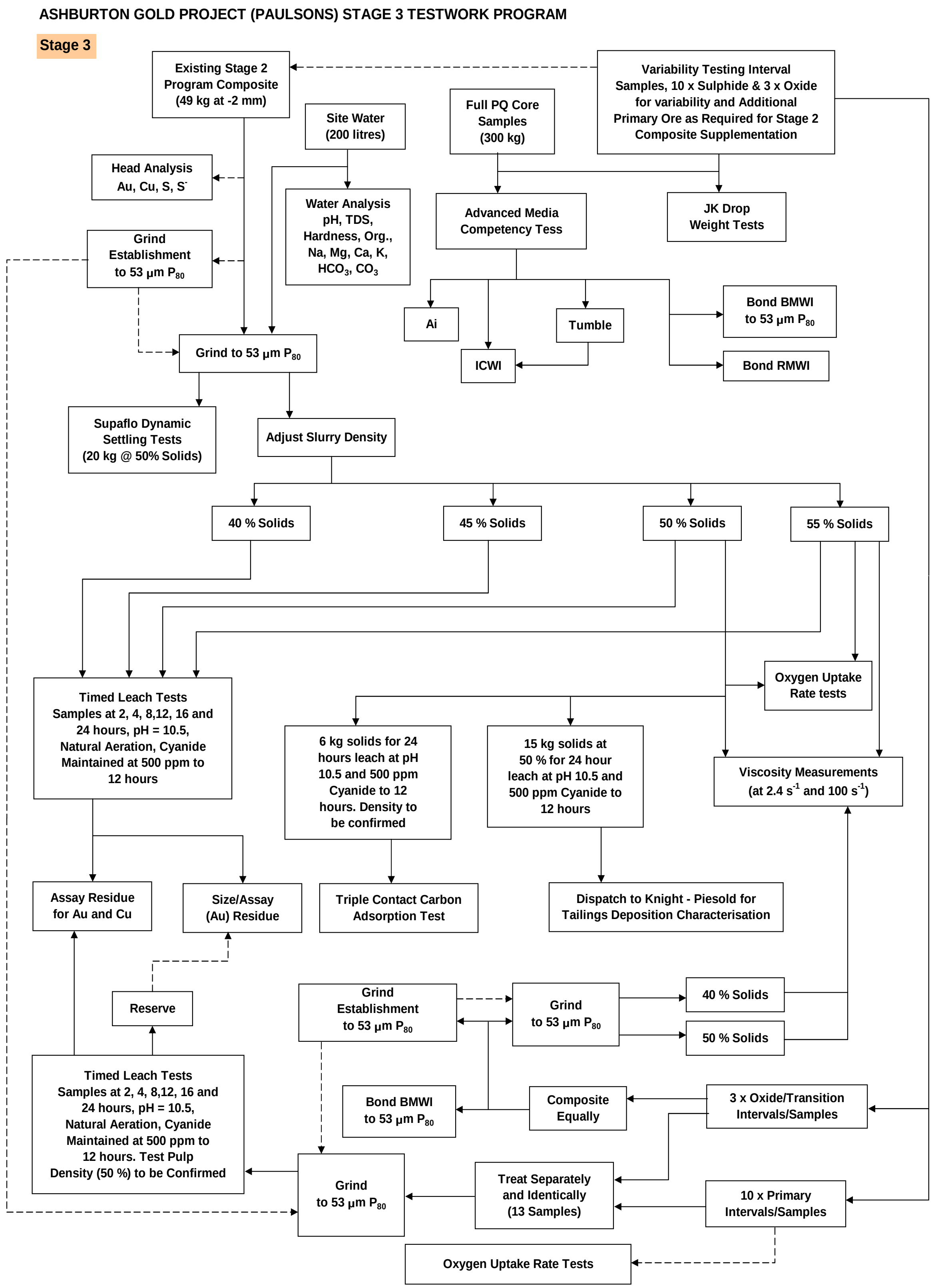
The success of the updated Feasibility Study is a credit to the dedication and hard work of a number of people working on the Paulsens Optimised Project Study. These endeavours culminated in a more robust project. The authors would like to thank all staff and consultants for their contribution and the management of St Barbara for their permission to publish this paper.

APPENDICES

1. Stage 1 Metallurgical Test Programme
2. Stage 2 Metallurgical Test Programme
3. Stage 3 Metallurgical Test Programme
4. Stage 4 Metallurgical Test Programme
5. Paulsens Flowsheet Updated
6. Original Bounty Flowsheet
7. Water Balance
8. Head Assays Met Testwork Composites

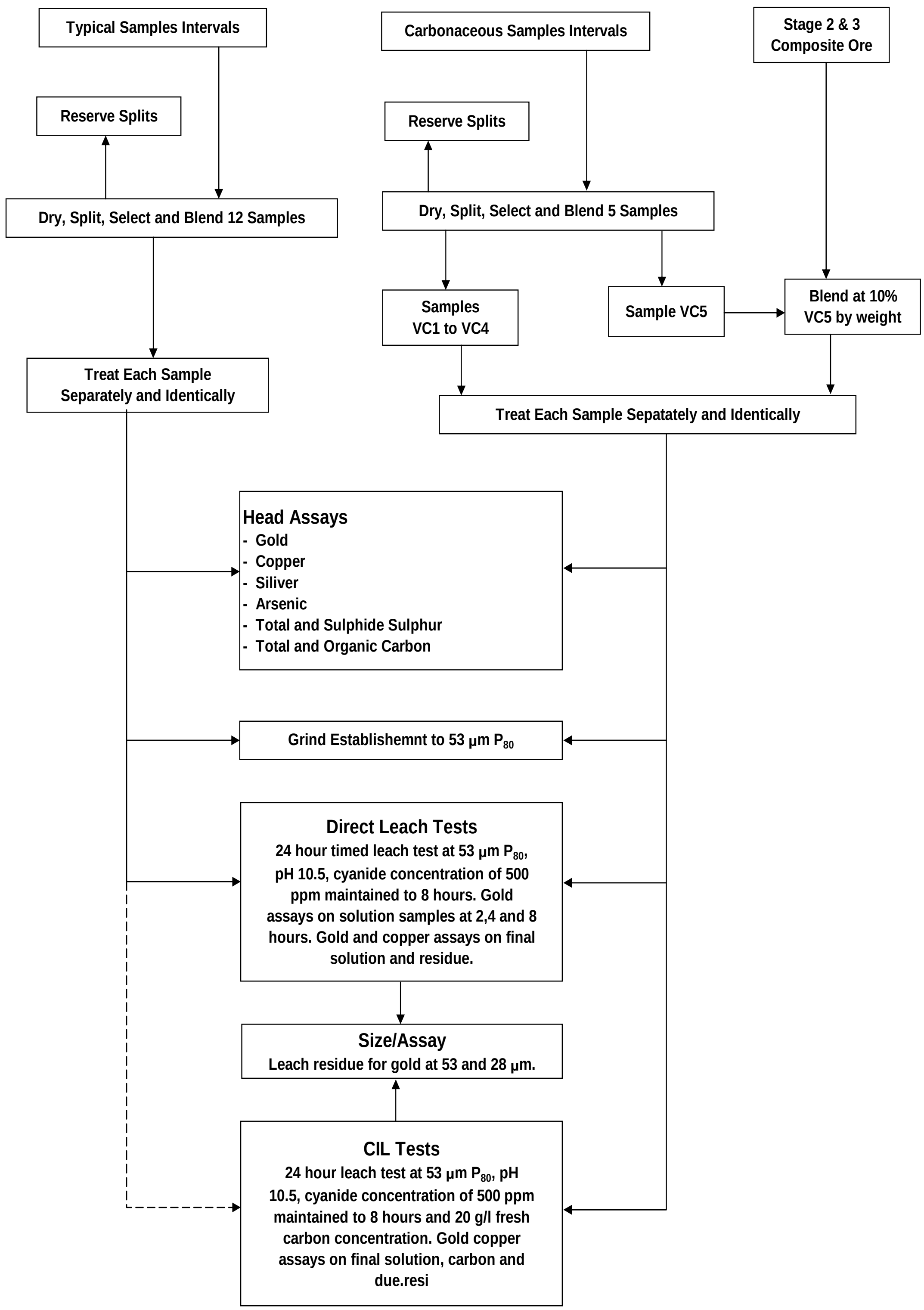

PAULSEN'S GOLD ORE PRELIMINARY METALLURGICAL TESTWORK PROGRAM
Stage 1
Sample Intervals Blending
Main Composite
High Grade Composite
Copper Composite
Carbonaceous Composite
Head Assay
ICP Scan of each Composite
Au, Cu, S, C$_{org}$, Tru SG
Grind to P$_{80}$ of 45 µm
Grind to P$_{80}$ of 75 µm
Grind Establishment
Grind Establishment to P$_{80}$ sizes of 75 and 45 µm
Cyanidation Tests
timed 24 hour leach tests at 40% solids, pH of 10.5, cyanide concentration of 1000 ppm.
Sample at 2, 4, 8 and 24 hours.
Calculate cyanide and lime consumption
Size Assay Analysis
Size at 106, 75, 45 and 20 µm and assay for Au
Diagnostic Leach Tests
Proportion of gold within cyanide extractable, sulphide and silicates at 75 µm P$_{80}$ grind size on selected leach residue samples.
Gravity Concentration Test
Conduct Knelson Gravity Concentration on - 1mm feed, amalgamate concentrate, return amalgam tail to gravity tail
Grind Combined Gravity Tail to a P$_{80}$ of approximately 75 µm.

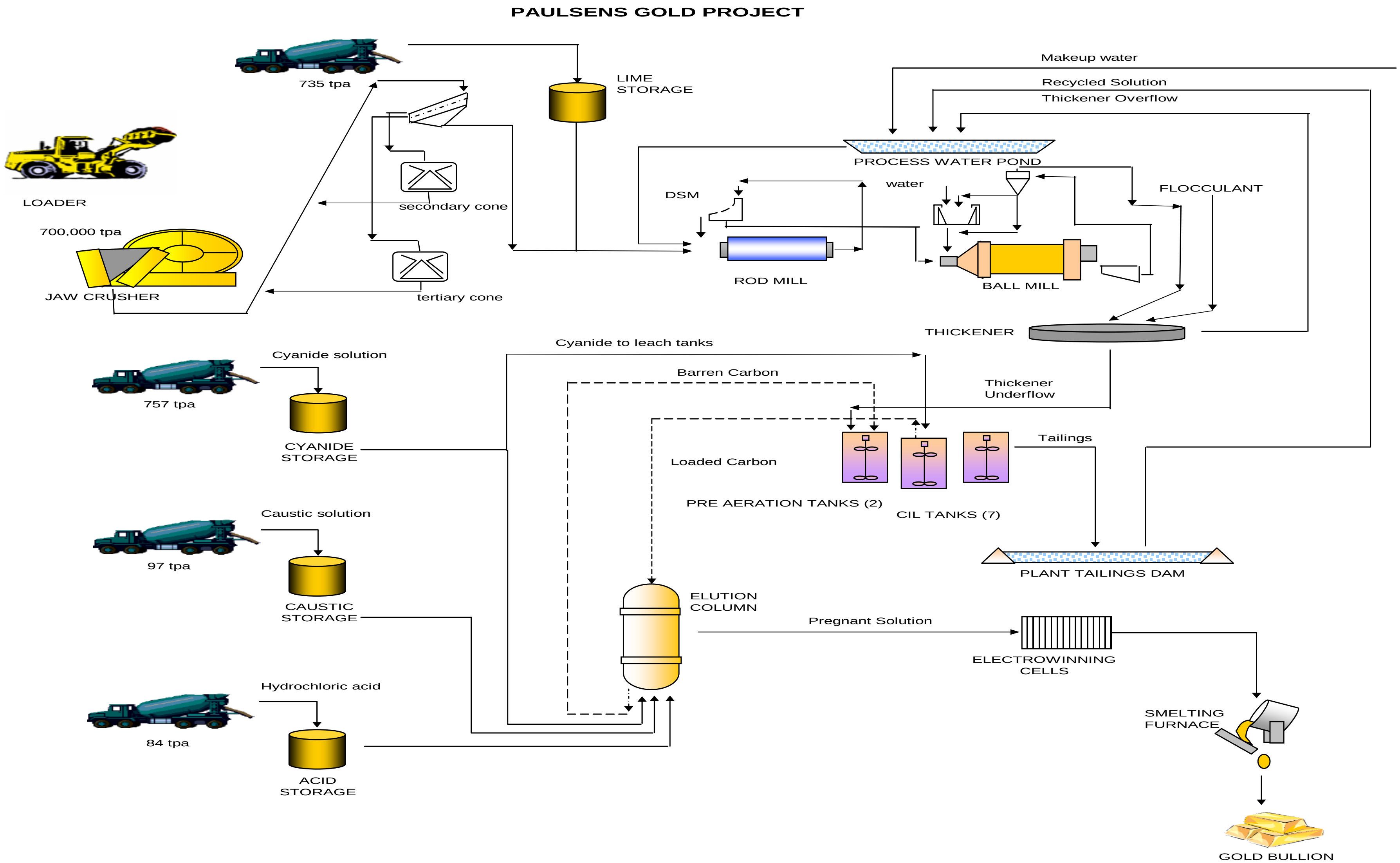

PAULSEN'S GOLD ORE STAGE 2 METALLURGICAL TESTWORK PROGRAM
Stage 2
Selected Half Core Samples (x20))
Site Water
Analysis
Sulphide Half Core
Quartz Half Core
Host Rock Half Core
Sulphide Full Core
Quartz Full Core
Host Rock Full Core
Apparent SG Determination
Blended Composite of Life of Mine Ore Types and Grade (~ g/t)
Physicals
Grind Establishment to P_{80} grind sizes of 150, 75, 53,and 45 μm
Head Analysis
BMWi to 53 μm P_{80} limiting size
RMWI
ICWI
UCS
Individually Assay
Gravity
Cyanidation
Reagent scooping Tests
Flotation Tests at 150, 106 and 75 μm P_{80} grinds
Knelson Concentration at P_{80} feed sizes of 500 μm
Whole Ore Grind P_{80}
75 μm
53 μm
45 μm
Gravity Tails
Bulk Flotation Test(s) at Selected
Amalgamate Concentrate
Amalgam Tail
Timed Leach Test at 500 ppm NaCN
Flotation Tails
Flotation Concs
Regrind to 45 μm and 30 μm P_{80} sizes
Size/Assay Products to 20 μm (x7)
Intensive Cyanidation Leach Test
Timed Leach Tests at 1000 and 300 ppm NaCN
Leach Test at 500 ppm NaCN for 24 hours, Triple Contact Adsorption Test
Size/Assay Products to 20 μm

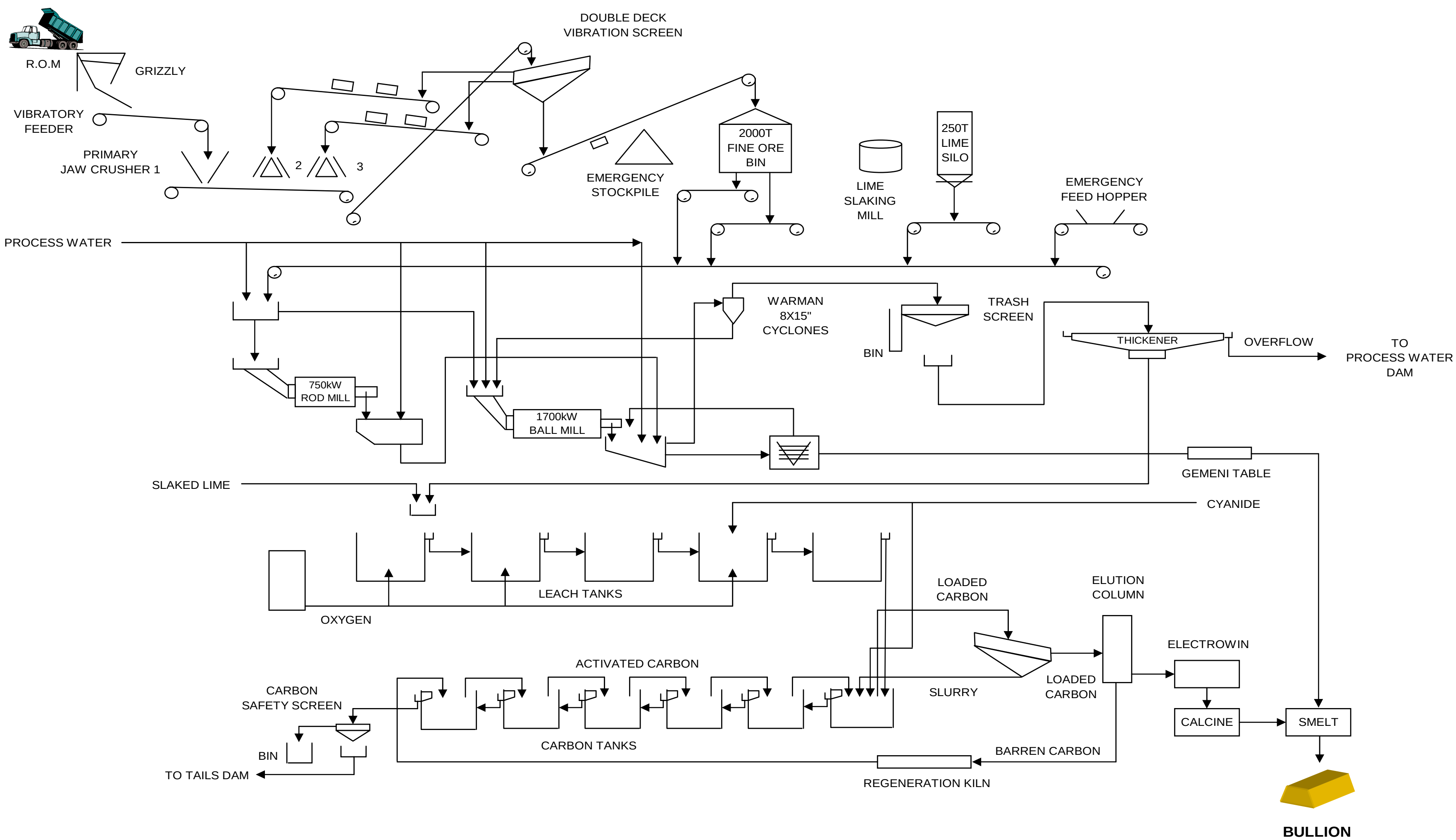

ASHBURTON GOLD PROJECT (PAULSONS) STAGE 3 TESTWORK PROGRAM
Stage 3
Existing Stage 2 Program Composite (49 kg at -2 mm)
Variability Testing Interval Samples, 10 x Sulphide & 3 x Oxide for variability and Additional Primary Ore as Required for Stage 2 Composite Supplementation
Site Water (200 litres)
Full PQ Core Samples (300 kg)
Head Analysis Au, Cu, S, S⁻
Water Analysis pH, TDS, Hardness, Org., Na, Mg, Ca, K, HCO_3, CO_3
Advanced Media Competency Tess
JK Drop Weight Tests
Grind Establishment to 53 µm P_{80}
Ai
Tumble
ICWI
Bond BMWI to 53 µm P_{80}
Bond RMWI
Grind to 53 µm P_{80}
Supaflo Dynamic Settling Tests (20 kg @ 50% Solids)
Adjust Slurry Density
40 % Solids
45 % Solids
50 % Solids
55 % Solids
Oxygen Uptake Rate tests
Timed Leach Tests Samples at 2, 4, 8,12, 16 and 24 hours, pH = 10.5, Natural Aeration, Cyanide Maintained at 500 ppm to 12 hours
6 kg solids for 24 hours leach at pH 10.5 and 500 ppm Cyanide to 12 hours. Density to be confirmed
15 kg solids at 50 % for 24 hour leach at pH 10.5 and 500 ppm Cyanide to 12 hours
Viscosity Measurements (at 2.4 s^{-1} and 100 s^{-1})
Assay Residue for Au and Cu
Size/Assay (Au) Residue
Triple Contact Carbon Adsorption Test
Dispatch to Knight - Piesold for Tailings Deposition Characterisation
Reserve
Grind Establishment to 53 µm P_{80}
Grind to 53 µm P_{80}
40 % Solids
50 % Solids
Timed Leach Tests Samples at 2, 4, 8,12, 16 and 24 hours, pH = 10.5, Natural Aeration, Cyanide Maintained at 500 ppm to 12 hours. Test Pulp Density (50 %) to be Confirmed
Bond BMWI to 53 µm P_{80}
Composite Equally
3 x Oxide/Transition Intervals/Samples
Grind to 53 µm P_{80}
Treat Separately and Identically (13 Samples)
10 x Primary Intervals/Samples
Oxygen Uptake Rate Tests

ASHBURTON GOLD PROJECT (PAULSONS) STAGE 4 TESTWORK PROGRAM

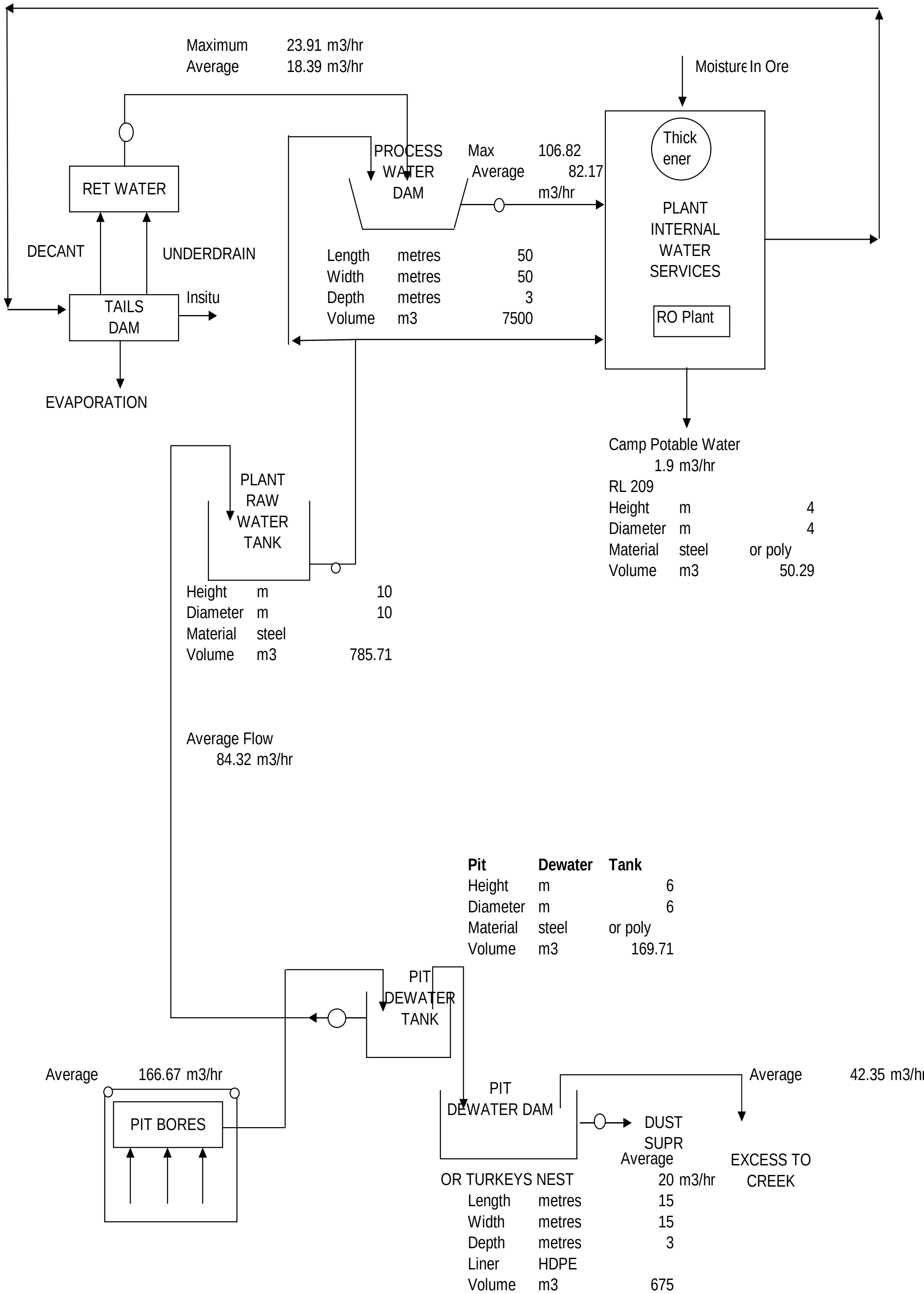
Stage 4




PAULSENS GOLD PROJECT
735 tpa
LIME
STORAGE
Makeup water
Recycled Solution
Thickener Overflow
LOADER
PROCESS WATER POND
DSM
water
FLOCCULANT
secondary cone
700,000 tpa
JAW CRUSHER
tertiary cone
ROD MILL
BALL MILL
THICKENER
Cyanide to leach tanks
Cyanide solution
757 tpa
Barren Carbon
Thickener
Underflow
CYANIDE
STORAGE
Loaded Carbon
Tailings
PRE AERATION TANKS (2)
CIL TANKS (7)
Caustic solution
97 tpa
PLANT TAILINGS DAM
CAUSTIC
STORAGE
ELUTION
COLUMN
Pregnant Solution
ELECTROWINNING
CELLS
Hydrochloric acid
84 tpa
SMELTING
FURNACE
ACID
STORAGE
GOLD BULLION


DOUBLE DECK
VIBRATION SCREEN
R.O.M
GRIZZLY
VIBRATORY
FEEDER
PRIMARY
JAW CRUSHER 1
2
3
2000T
FINE ORE
BIN
EMERGENCY
STOCKPILE
LIME
SLAKING
MILL
250T
LIME
SILO
EMERGENCY
FEED HOPPER
PROCESS WATER
WARMAN
8X15"
CYCLONES
TRASH
SCREEN
BIN
THICKENER
OVERFLOW
TO
PROCESS WATER
DAM
750kW
ROD MILL
1700kW
BALL MILL
GEMENI TABLE
SLAKED LIME
CYANIDE
LEACH TANKS
OXYGEN
LOADED
CARBON
ELUTION
COLUMN
ELECTROWIN
ACTIVATED CARBON
CARBON
SAFETY SCREEN
SLURRY
LOADED
CARBON
CALCINE
SMELT
BIN
CARBON TANKS
BARREN CARBON
TO TAILS DAM
REGENERATION KILN
BULLION

PAULSENS WATER CIRCUIT



SAMPLE HEAD ASSAYS

Sample Description	Gold (g/t)	Copper (g/t)	Siliver (g/t)	Arsenic (g/t)	Sulfur (%)		Carbon (%)	
					Total	S^{-}	Total	Org.
CRA Oxide Comp.	10.2	N/A	N/A	N/A	N/A	N/A	N/A	N/A
CRA Sulfide Comp.	5.31	N/A	N/A	390	5.96	N/A	N/A	N/A
Stage 1 - Main	9.40	116	<2	N/A	8.62	8.12	0.57	0.05
Stage 1 - High Grade	21.60	246	<2	N/A	5.01	4.42	0.27	0.06
Stage 1 - Copper	5.50	7900	<2	N/A	6.23	6.17	2.72	0.12
Stage 1 - Carb.	9.55	344	<2	N/A	7.09	6.70	1.57	0.09
Stage 2 & 3 Composite	7.26	466	<2	619	7.93	N/A	N/A	N/A
Stage 4 - Carb. PVC1	2.17	384	2.8	396	4.17	3.72	2.48	0.63
Stage 4 - Carb. PVC2	3.60	1360	16.0	563	7.06	6.40	2.13	0.28
Stage 4 - Carb. PVC3	4.58	230	0.6	414	4.49	4.21	1.28	0.68
Stage 4 - Carb. PVC4	3.50	121	0.8	261	2.58	2.02	1.48	0.52
Stage 4 - Carb. PVC5	0.15	76	<0.1	173	0.63	0.48	2.08	0.78
Stage 4 Crab. PVC5B	8.52	476	0.8	617	7.68	7.27	1.17	<0.03
Stage 4 - Var. PVT1	2.70	647	1.3	620	0.50	0.26	0.36	<0.03
Stage 4 - Var. PVT2	3.12	720	1.7	517	4.88	4.73	0.45	0.04
Stage 4 - Var. PVT3	5.55	1410	24.0	317	6.02	5.40	2.10	0.25
Stage 4 - Var. PVT4	4.10	1680	2.0	286	3.48	3.18	3.23	0.06
Stage 4 - Var. PVT5	4.95	116	0.3	558	4.91	4.62	1.21	0.44
Stage 4 - Var. PVT6	16.00	624	1.2	1057	17.40	16.70	0.063	0.06
Stage 4 - Var. PVT7	5.95	710	1.2	596	9.38	8.99	0.88	0.06
Stage 4 - Var. PVT8	3.28	518	0.9	393	6.44	5.92	1.04	<0.03
Stage 4 - Var. PVT9	14.75	140	0.8	663	8.66	8.14	0.46	0.06
Stage 4 - Var. PVT10	4.25	124	0.5	450	6.29	5.99	0.82	0.22
Stage 4 - Var. PVT11	3.14	217	0.4	324	4.40	4.37	0.99	0.14
Stage 4 - Var. PVT12	6.33	229	.04	524	7.89	7.29	0.76	0.10