PAPER 19

Goldcorp Inc., Red Lake Mine
Design and Commissioning

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ABSTRACT

The Red Lake Mine is situated in Balmer Township, Red Lake area in northwestern Ontario. It is an underground mine that has been in production since 1948. The property was originally known as the Dickensen Mine and was shut down in 1996.

A feasibility study was carried out in 1998 and it was determined that a new mill should be constructed with a throughput rate of 600 stpd. The life-of-mine average gold grade was projected to be 1.35 oz/t with gold predominantly associated with arsenopyrite and pyrite. Metallurgical testwork indicated that the optimal flowsheet for gold recovery would include an autoclave for oxidation with a carbon circuit for adsorption. To limit capital cost and establish cashflow, the mill was constructed without an autoclave. The recovery by gravity and cyanidation was projected to be 83%. A flotation plant was constructed at the tail end of the circuit to recover the additional gold associated with sulphides from the leach circuit tailings. This material is being stockpiled onsite for future oxidation treatment.

Detailed engineering was started in early 1999 and the first gold was poured in August 2000. One feature of the project was the use of Contact Lake equipment from Cameco's Contact Lake project. The entire mill was purchased and equipment re-used where possible. The capital cost for the process facilities was C$3 1.5 million.

INTRODUCTION

The Red Lake Mine, owned by Goldcorp Inc, is situated in Balmer Township, Red Lake area in northwestern Ontario. A map of the area is shown as Figure 1.

The Goldcorp facility shares a property boundary with Placer Dome's Campbell Mine. The two mines form the foundation of the small town of Balmertown, with both shafts clearly visible form the main street. Although the mineralogy of the two deposits is very similar, the development of the two properties has been very different.

Shaft sinking began on the Goldcorp property in July 1946. Milling started in December 1948. The mill was initially constructed with a flotation circuit and the flotation concentrate was oxidised using a roaster. The roaster was shut down in 1980 for environmental reasons. Following closure of the roaster, flotation concentrate was stockpiled and periodically shipped to a smelter. The mill was shut down in June 1996 for economic reasons.

Geological and metallurgical work continued with the discovery of new high grade orebody with reserves of 2.3 million ounces of gold. A new mill was designed to replace the outdated existing facility. The new mill was started in August 2000 at a throughput rate of 600 T/d.

METALLURGY

Goldcorp has one of the highest grade deposits in the world at the current time. The average grade for the deposit is 1.35 opt (46 g/tonne). The ore stockpile at the commencement of the new operation was over 2 opt with gold clearly visible in the ore. The Goldcorp ore is refractory, with sulphides occurring primarily as pyrite and arsenopyrite. The average sulphur and arsenic grades in the ore are estimated as 2.15% and 1.84% respectively. Approximately half of the gold is gravity recoverable.

PROCESS DESCRIPTION

A feasibility study was carried out by Kilbom SNC-Lavallin in 1998. The feasibility study recommended a flowsheet as shown in Figure 2. This flowsheet included concentrate leaching with subsequent filtration of the product. Subsequent metallurgical testing managed by J.R. Goode and Associates resulted in the flowsheet presented as Figure 3. This flowsheet had some cost advantages and was used as the basis for the detailed design.

ENGINEERING DESIGN

During the feasibility study, it was established that the Contact Lake mill, which was for sale, had equipment which could be reused at Goldcorp. There was also equipment available in the old mill building at Goldcorp. A key element of the engineering was to ensure that capital costs were minimized through the use of available equipment. Hatch completed the detailed engineering for the chosen option. Some of the key design criteria are presented in Table 1:

Table 1: Design Criteria

General		
Crushing	% availability	75%
Process plant	% availability	92%
Daily throughput, nominal	T/d	700
Average grade Au in ore	oz/T	1.35
Maximum grade Au in ore	oz/T	2.50
Average grade As in ore	%	1.84
Average grade S in ore	%	2.15
Annual gold production, average	oz/y	286,978
Silver recovery to bullion (as % of gold)	%	12
Crushing		
Primary Crusher type		Jaw
Crusher product P_{80}	in	4.5
Secondary Crusher type		Omnicone
Crusher product P_{80}	in	0.625
Grinding		
Mill configuration		Single Ball
Product P_{80}	microns	74
Bond Ball mill work index	kWh/T	16.5
Ball mill diameter	m	3.50
Ball mill length	m	4.80
Ball mill installed power	hp	1.200
Gravity Concentration		
Gravity concentrator type		Knelson
Gravity concentrator diameter	in	20
Cyclone underflow to mill	% of mill feed	200
Cyclone underflow to gravity concentrator	% of mill feed	100
Upgrading table		Deister, 5ft x 15ft

Preleach Thickening		
Feed density	% solids	32%
Thickener underflow density	% solids	50%
Thickener unit area requirement	$ft^2/T/d$	0.81
Chosen thickener diameter	m	12.00
Leach		
Required retention time	h	48.00
NaCN consumption	lb/T	4.12
Unit air sparging rate	$m^3/h/m^3$	1.20
CIP		
Number of tanks		6
Retention time per tank	h	1
Calculated carbon loading (fouled)	oz/T	341
Design carbon loading	oz/T	150
Adsorption efficiency	%	99.9%
Inner tank screen type		EPAC
Carbon Treatment		
Stripping method		Zadra
Acid Washing	T	Nitric
Acid washing vessels capacity	T	4
Strip vessel capacity	T	2 x 4
Kiln capacity	T/d	6
Elecrowinning/Refining		
EW cells		2 x $100ft^3$
Cell size	m^3	2.83
Type		Punched plate cathode
Refining furnace type		Induction
Cyanide Destruction		
Process		Inco so2
Number of reaction tanks		2
Retention time per tank	h	3
Feed CN_{WAD}	mg/l	250
Tails CN_{WAD}	mg/l	1
Flotation		
Final concentrate weight recovery	%	10.0%
Final concentrate grade, Au	oz/T	2.21
Total rougher retention	mins	26
Total scavenger retention	mins	26
Total cleaner retention time	min	14
Concentrate Thickening and Filtration		
Thickener unit area requirement	$ft^2/T/d$	1.35
Thickener diameter chosen	m	6
Filter Type		Drum
Filter area	m^2	51.26
Filter operating time	hrs/d	1.84
Backfill		
Method		Paste Deposition
Thickener underflow density	% solids	59
Chosen thickener diameter	m	10.6
Process type		Batch/Continuous

Filter Type		Disk
Slump density control		Mixer torque
Slump of paste to underground	in	**6-9**

Key Aspects of the engineering design included the following:

Site Layout

When the project was initiated, the existing mill, shaft and infrastructure was in place. The property is bordered on the North and East side by Balmer Creek which allowed limited options with regard to locating the mill building. An area was chosen as indicated in Figure 4. The existing headframe and coarse ore bin were included into the new design to limit costs.

The location of the plant was altered from the feasibility study. A review of previous site mapping indicated that the area bordering the Creek to the north and east was largely fill. A number of boreholes were drilled in the process building area and the building was located in a position to minimize earthworks and foundations. The leach area was designed at a higher elevation to take advantage of the natural rock elevations. The fill area near the Creek was designated for use as a coarse ore stockpile to be used for stockpiling ore during mine development.

With any project there is a tradeoff on the costs incurred to get good geotechnical information and the cost of construction. One area where we found conditions to be different to our expectations was the new crusher building. The new crusher building is adjacent to the old building which was on a rock foundation, and it was established through drilling, that there were good rock foundations in the process building area. The rock however dipped sharply in between, and foundation design had to be modified at the time that this was established.

Used Equipment

The mill equipment from Contact Lake was purchased as a package and used where it made sense. The crusher and ball mill were reused as well as the leach and CIP tanks. An aspect of reusing equipment is a comprehensive assessment of the condition of each item. In some cases the cost of refurbishment was very similar to the cost of purchasing new equipment. The re-use of equipment also sometimes has associated with it the loss of process advantages associated with new equipment. The CIP tanks for example had air swept EPAC-type screens. In most new operations, cylindrical swept screens are used. The newer screens allow for easier maintenance and lower operating costs.

For this project a drum filter from the original Goldcorp operation was used for concentrate filtration. This was much larger than required. The optimum design would have been a smaller disc filter with less building area required. The larger unit had a positive impact on the layout in that it allowed for more concentrate storage space. The associated equipment such as the vacuum pump and receiver were also reused. The current arrangement also allows some flexibility for personnel to schedule the operation of the equipment.

The Contact Lake carbon handling and strip circuit was a 1 tpd unit. For the higher grades at Goldcorp it was established that a 6 tpd circuit was required. To minimize capital costs, a circuit with one 4 t acid wash vessel and two 4 t strip columns was purchased. The regeneration kiln was designed for 6 tpd.

Expansion Capability

The projected throughput rate for current operations is 600 stpd. The mill has been designed for 700 stpd, with expansion capabilities to 1000 stpd. A schematic plan of the process building is presented as Figure 5. Space was allowed in the layout for additional CIP tanks, flotation cells and electrowinning cells. One of the main changes required to get to a throughput of 1000 tpd, would be the addition of additional grinding capacity. The area adjacent to the grinding bay was kept open, however, building space was not allocated as this cost was not considered to be warranted. The areas that were allocated for future equipment were useful during construction. Other mechanical equipment such as pumps and thickeners were sized to allow for a throughput of 1000 tpd. The cyanide destruction circuit was also sized to handle the higher throughput.

Flexibility to Restructure the Circuit

Since the potential exists for pressure oxidation of concentrate at a later date, flexibility was allowed to restructure the circuit with flotation prior to leach. Minor piping modifications are required to achieve this. The high grade flotation concentrate will then end up in the filter feed tank and can be pumped to an oxidation facility. The current filter area will be used to load stockpiled concentrate back into the filter feed tank.

Process Considerations

One of the primary concerns with flotation after cyanidation, is the depressing effect the cyanide has on flotation. An Inco system was installed with a target CN_{WAD} of 1 mg/l. This ensures that flotation will operate effectively even after leach. This also meets environmental limits and allows for tailings to be used as backfill.

Addition of the Backfill Plant

It was decided that a backfill plant was required after the construction of the main process facility had started. A separate RFQ was sent out for this work. A combined bid was with Golder Paste Tec and Hatch was accepted. Golder was responsible for the flowsheet, conceptual design and selection of major equipment, with Hatch doing the detailed engineering. This arrangement worked well since existing contractors were familiar with Hatch conventions and specifications. The PLC programming was also consolidated so that all systems were consistent.

The tailings pumpbox and line was in position in an existing building. The southern part of this building was a new extension which also contained a large agitated tank. The design entailed putting a batch type backfill plant into the existing building. The agitated tank was used as a filter feed tank for storage, to allow for increased throughputs when stopes are ready for filling. The plant consists primarily of a disc filter and batch mixer with cement addition from an outside silo. The product to underground is paste, which allows minimum dewatering in the stopes. The plant location allows for flow by gravity to underground.

COMMISSIONING

Construction of the Project commenced during the month of November 1998. Merit Consultants International Limited was retained by Goldcorp Inc. to manage all aspects of the construction phase of the project. This included site preparation, civil workings, mechanical installation, electrical installation, as well as all aspects of underground development. At its peak, Merit, working with Goldcorp personnel, was responsible for the day to day activities and direction of over 15 different contractors, totaling over 300 workers. During the later stages of February 2000 outstanding details of the construction portion of the crushing plant were finalized and it was put online. Towards the end of June 2000, eighteen months after the commencement of the construction phase of the project, the processing plant was brought online. The paste pant followed shortly thereafter.

Through the course of 2000, three different plants were commissioned: the crushing plant, the processing plant, and the paste plant.

Crushing Plant Commissioning

Underground development began in the very early stages of the Project. During this time ore generated was stockpiled underground. Construction of the crushing plant ended during the later stages of February 2000 and commissioning of the first stage began. Development ore continued to be stockpiled underground, and commissioning started utilizing underground waste muck. Site preparations were made for a future ore stockpile on the surface. The crushing plant is a two stage operation, The first stage of crushing involves conveying underground material from the coarse ore bin (COB) to the jaw crusher. Material exiting the jaw crusher can either report to an outside stockpile or the cone crusher for a second stage of crushing. In order for material to report to the cone crusher, however, the processing plant must be available so that this material can be further treated. As a result the second stage of the crushing plant was commissioned in conjunction with the process plant.

Underground muck, screened at 12 inches, is skipped to surface and discharges into the existing 900 T (live load) COB. For a month and a half, underground waste material was subject to one stage crushing. This material's size was reduced from over 12 inches down to roughly 4 inches through the 42 inch by 30 inch Traylor jaw crusher. Material discharges onto the #2 Conveyer where it reports to a transfer chute. Here material can either report to the second stage of the plant for further size reduction or outside for stockpiling. During this time period, the cone crusher, and related equipment, was not ready for commissioning. As a result crushed waste reported outside. A used portable conveyor was purchase and refurbished to achieve this. Virtually 100% of the crushed waste that was produced during this time was used for various site preparation activities such as fill for civil works, road building, regular till, etc.

Some of the modifications that were made as part of commissioning the first stage of the crushing plant included:

- Re-align skirting to prevent build-up of material on the Jaw Crusher discharge conveyer.
- Modifications to the Jaw Crusher discharge chute to prevent an excess dead load
- Adjustments to the COB vibrating feeder.
- Re-aligning the pinion bearing after discovering a localized hot spot.
- Installation of a glycol cooling system while decommissioning the water cooling system (occurred during the commissioning of the Processing Plant).

The skipping and crushing of underground ore commenced in the first week of April 2000. An outside contractor was retained to haul material from the refurbished portable conveyer to one of two stockpiles. A total of roughly 12,400 cubic yards of underground development ore was passed through the first stage of the crushing plant, discharged outside, and subsequently hauled to two separate stockpiles between early April and late June. The intention was to begin using this stockpiled material once the processing plant moved into full twenty-four hour operation. This would supplement underground ore sources as production rates were ramped up to design levels.

Commissioning of the second stage of the crushing plant began in early July 2000, coinciding with the start-up of the processing plant. Commissioning included an Omnicone Cone Crusher, a 16' by 6' WS Tyler vibrating screen, and all related equipment, such as the fine ore handling system, as well as aspects of the reclaiming system. One pre-commissioning detail that was double-checked was skirting installation on all conveying systems. This was particularly important due to the fact that the conveying system in the second stage of crushing is more intensive with longer running belts and a circulating load.

As with the first stage of the crushing plant, the second stage was commissioned on underground waste muck, coinciding with the commissioning of the processing plant.

Some of the modifications that were made during commissioning the second stage of the crushing plant included:

- The original screen was replaced with a screen that had slotted holes rather than square. The reason for the change was to help prevent plugging of the screen surface with wet underground fines. Material is currently being screened at 3/8" before reporting to the processing plant. Availability increased significantly with the installation of this type of screen panel.

- After running with the modified screen panels for 3 weeks, a different panel was used. The long slotted holes allowed large flat stones to pass through the decking. This resulted in reduced grinding rates as significant stone rejects began being rejected from the ball mill trommel whenever the fine ore bin level was low. The third generation of panel was a more flexible rubber with square slots. The flexibility in the panels helped shed the build up of wet fines on the screen surface.

Processing Plant Commissioning

In June and early July 2000, commissioning of the processing plant commenced. Goldcorp Inc. had decided to keep Merit Consultants' on site for three months to assist in the management and commissioning of both the processing and paste plants. Underground waste material was utilized as an initial fill throughout the plant. Thickeners were bedded and dead loads in the fine ore bin were developed. Process lines and general process flows were monitored for proper densities, pump sizing, piping installation, etc. One of the first systems commissioned in the processing plant was the water system. Water was pumped from Balmer Creek, the current water source, to the process water tank and fresh water tank. Water was then pumped throughout the entire process including ball mill, ball mill discharge pump box and pumps, hydrocyclones, pm-leach thickener, leach and CIP tanks, INCO SO_2 reactors, and the flotation circuit. Lines, tanks, and pumps were monitored for sizing, leaks, failure, etc. During this time the ball mill was charged with roughly 60 tons of a mixture of used balls from the old mill and new balls which consisted of a charge of balls ranging in sizes from 1 inch to 2% inches. A ball charge of roughly 35% by volume was, and continues to be, utilized in the mill.

Commissioning of the processing plant was initially carried out as a 12 hour day, 7 day per week schedule. This schedule allowed better utilization of experienced operators and training of less experienced personnel, This schedule also provided excellent practice at starting and stopping the equipment in the plant. This experience proved very valuable during a few unplanned power outages later on during the operation of the plant. The extra downtime also allowed for a much more intensive maintenance schedule during the commissioning period. This was important when it came time to switch from waste to high grade ore. Until this time, underground high grade ore had been brought to surface, subjected to one stage crushing, and stockpiled on site. Once the processing plant was ready to be placed online, the mine switched from high grade ore back to waste so that commissioning of the plant could begin.

On 13 July 2000, the processing plant began treating waste. The grinding circuit, including gravity concentrating and pre-leach thickening, was operated on an 8 hour per day basis.

Throughput for this time period remained at 250 tons per day. Process flows were established, the thickener was bedded, leach and. CIP tanks filled, etc. This continued for roughly one week and on 20 July 2000 the plant switched from waste to high grade ore.

Two considerations were of primary concern once gold bearing material, was treated:

Gold Recovery

While processing waste, the gravity concentrating portion of the grinding circuit had been commissioned. Commissioning of the Knelson Concentrator proved to be quite uneventful. One cyclone is dedicated to the Knelson concentrator and the process stream is screened via a static 4 mm slotted polymet screen. Screened material flows via gravity to the center of the unit where concentration takes place. Once the process stream was established, the only input that was required was the adjustment of flow rates, water addition to the unit, operating pressures, and cycle durations. Concentrated material is upgraded on a Diester table, which was also commissioned during this time. Modifications in this area included:

- Replacement of the table screw feeder to an in-house made weir-type of valve arrangement. The original feeding mechanism involved a knife gate valve feeding the screw feeder which in turn fed the table. This arrangement resulted in an inconsistent feed rate to the Diester table, and was subsequently replaced.

- Re-piping of the table tail line. This originally reported to the ball mill discharge pump box which pumped to the cylopac. It was changed to pump to the cyclone that is dedicated to the Knelson concentrator. The table middlings line continues to report to the concentrate hopper.

Retention time in the plant was extremely high due to the fact that the plant was only processing 250 tons per day. The flotation circuit had to be commissioned under these conditions, and had to be ready to accept feed for recovery of the refractory component of the ore.

Environmental Constraints

Cyanide bearing solution in excess of 1 ppm could not be discharged to the Tailings Management Area. As a result, the INCO SO_2 Plant had to be commissioned before leaching of the high grade ore could commence. This proved to be quite an undertaking, as a result of modifications required to the second hand system. A representative from the INCO Technical Services group remained on site to assist with this. After adapting piping configurations, and obtaining the appropriate approvals pertaining to pressurized vessels, the cyanide destruction portion of the processing plant was ready to accept cyanide bearing material.

On 20 July 2000, the processing plant switched from waste material, which was being reclaimed from surface stockpiles, to high grade ore, which came directly from underground. Grades quickly increased to levels in excess of 1.5 opt. Although the plant continued to process only 250 tons per day, the first official pour was held ten days later, with over 1000 oz poured.

One problem encountered, as previously mentioned in the design portion of this paper, was the CIP interstage screens. Difficulties include blinding and excess wear. Although the current configuration allows the carbon adsorption area to operate well, these screens are maintenance intensive. A different type of screening practice will be investigated in the very near future.

Once flows were established through the circuit, and a relatively continuous stream reported to the flotation circuit, the processing plant began generating a bulk sulphide concentrate. Reagent flows and concentrations were established, and a concentrate was being generated by late August. One problem encountered pertained to handling water generated in the concentrate thickener. The cleaner concentrate from the flotation circuit is de-watered, stored, and eventually reports to a drum filter. Filter cake from this unit operation is subsequently stockpiled for future treatment. Water generated in the concentrate thickener was designed to report back to the process water tank for re-use throughout the plant. Flotation reagents contained in the overflow stream, although low in concentration, caused a variety of problems throughout the circuit. This stream was redirected to tails. The added water requirement for the plant is supplied from Balmer Creek, the current make-up water source.

Paste Plant Commissioning

On 27 August 2000, the paste plant was ready for commissioning. Final outstanding construction details such as calibration of the power transducer and weight scale were completed. Once the thickener was bedded and various unit operations (disc filter, floc system, etc.) were tested, paste was generated at various slumps and discharged to the Tailings Management Area (TMA). During this time period the BHS Mixer was calibrated to determine the correlation between power draw and slump. The discharge of paste to the TMA continued for a five day period.

Prior to discharging paste throughout the underground distribution system, lines reporting to the various underground areas were pressure tested from surface. The first test utilized air at 110 psi and no significant leaks were detected. The second test utilized water at 1000+ psi and again no leaks were detected. Pressure testing was also conducted underground at varying levels. Once the underground distribution system was commissioned the first discharge of uncemented paste to a test stope on the 30^{th} level occurred on 30 September 2000. On 01 October 2000, the first discharge of cemented paste occurred in the same test stope. The very next day the first production stope was poured. The plant operated on day shift only for the next 12 days.

Midway through the month of October this Plant moved to a twenty-four hour production day. Training for these operators was completed by late October.

OPERATING RESULTS

Table 2: Processing Plant's Operating Results

Processing Plant Summary		
Tons Crushed	tons	54,800
Crushing Plant Throughput	tph	60
Tons Milled	tons	50,400
Mill Throughput	tpd	610
Head Grade	opt	1.415
Gravity Recovery	%	52
CIP/Leach Recovery	%	35
Bullion Recovery	%	87
Flotation Recovery	%	8
Overall Plant Recovery	%	95
Bullion Production Summary		
Total Bullion Produced	ozs	55,000
Gravity Ounces Produced	ozs	35,000
Electrowinning Ounces Produced	ozs	20,000
Average Gold Fineness	%	85
Average Silver Content	%	12
Average Impurities Content	%	3
Gravity Concentration		
Knelson Cycle Duration	minutes	15
Tabling Frequency	weekly	3
Diester Grade (includes fineness)	%	73
Average Furnace Charge Weight (excl. fluxes)	Kg	150
Cip/Leach Summary		
Circuit Pulp Density	% solids	48-50-
NaCN Consumption	lb/ton	1.4
Leach Residue Grade	opt	0.180
CIP #1 Carbon Concentration	Gram/Liter	60-70
CIP #2 Carbon Concentration	Gram/Liter	50
CIP #3 Carbon Concentration	Gram/Liter	30
CIP #4 Carbon Concentration	Gram/Liter	30
CIP #5 Carbon Concentration	Gram/Liter	30
CIP #6 Carbon Concentration	Gram/Liter	40
Average Solution Feed Grade	opt	0.480
Average CIP Tail Solution Grade	opt	0.003
Adsorption Efficiency	%	99.4
Loaded Carbon Grade	opt	300
Stripped Carbon Grade	opt	0.5 – I.0
Stripping Efficiency	%	99.6
Average Loaded Eluate Grade	opt	15.0
Average Electrowinning Tails Grade	opt	0.20
Electrowinning Smelting Frequency	weekly	0.75
Electrowinning Sludge Grade (incl. Fineness)	%	78

Flotation Summary		
Final Concentrate Weight Recovery	%	**3 - 4**
Average Concentrate Grade, Au	opt	**2.18**
Concentrate Filtration Frequency	weekly	**3 - 4**
Average Flotation Tails Grade	opt	**0.072**
Backfill Summary		
Underground Fill Ratio	%	**60**
Slump	inches	**9.75** +/- .25
Mixer Power Draw	kW	**7.5**
Cement Content	%	**I0**
Unconfined Compressive Strength	psi	**276**
Cyanide Destruction Summary		
Reactors Required	tanks	1
CN Feed Content	ppm	**250 - 300**
CN Tail Content	ppm	

To date, there is not enough history for a detailed comparison of operating results and costs to predicted values. The initial results are very promising since recoveries are slightly higher than predicted, and cyanide addition is lower. The head grade and gravity recovery are higher than the values used in the original study.

FUTURE DEVELOPMENTS

Short Term

- Additional gravity concentration (currently being investigated)
- Different screening practice in the CIP Circuit (currently being investigated)
- Operating cyanidation circuit utilizing Liquid Cyanide (currently being investigated).

Long Term

The installation of a concentrate autoclave is capital intensive. With a pressure oxidation circuit, the slurry will report to flotation prior to oxidation. This circuit will allow for a high-grade concentrate to go to the pressure oxidation circuit. The overall cyanide addition is more effective with this configuration, since it allows for a higher addition rate to the concentrate leach. Cyanide consumption by sulphides will also be reduced, as the sulphides are oxidized prior to leaching. These costs are offset by the cost of operating an autoclave, and the costs of neutralizing the resulting sulphuric acid.

A comparison of the predicted grades and recoveries for the current flowsheet and autoclave flowsheet are presented in Table 3.

Table 3: Predicted Gold Recoveries

		Whole Ore Leach	Pressure Oxidation	
Feed Grade	oz/T	1.35	1.35	
Gravity Recovery	%	40	40	
Feed to leach	oz/T	0.81	N/A	
Leach recovery	%	72	N/A	
Feed to flotation	oz/T	0.227	0.81	
Flotation recovery	%	71	83	
Feed to leach	oz/T	N/A	Tails leach	Oxide leach
% of throughput		N/A	90	10
Leach feed grade	oz/T	N/A	0.153	6.72
Leach recovery	%	N/A	50	98
Tails	oz/T		0.077	0.134
Combined Tails	oz/T	0.066	0.0827	
Total recovery		83.2*		

* Recovery excludes gold in flotation concentrate since this is not readily recoverable

The pressure oxidation has a predicted recovery of 93.9% which is approximately 10% higher than the current flowsheet. This recovery does not take into account gold which is recovered by the flotation circuit at the tail end of the plant. This material is currently being stockpiled and will add to the viability of the operation in the longer term if pressure oxidation is adopted.

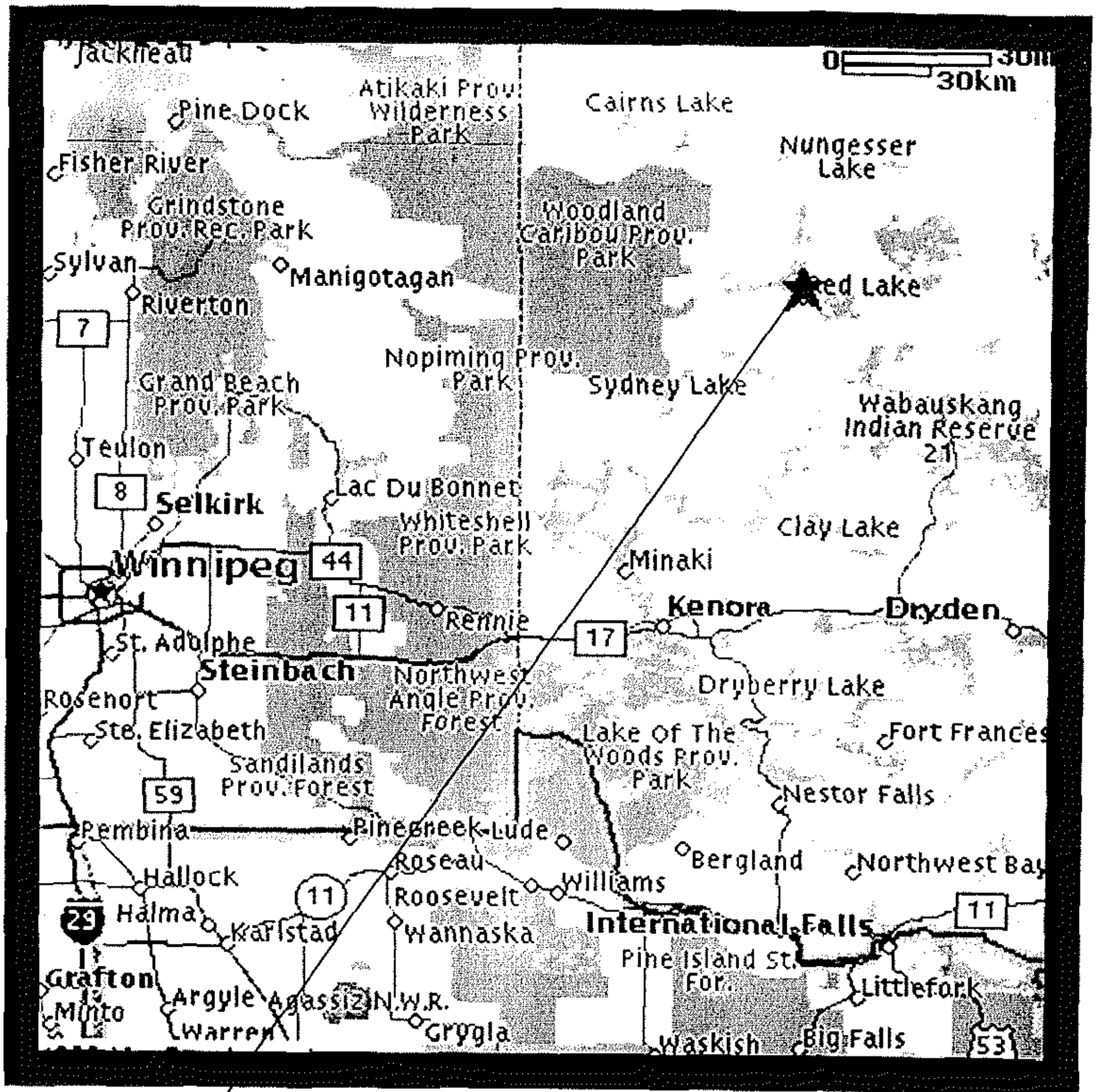

FIGURE 1 - RED LAKE MINE LOCATION MAP

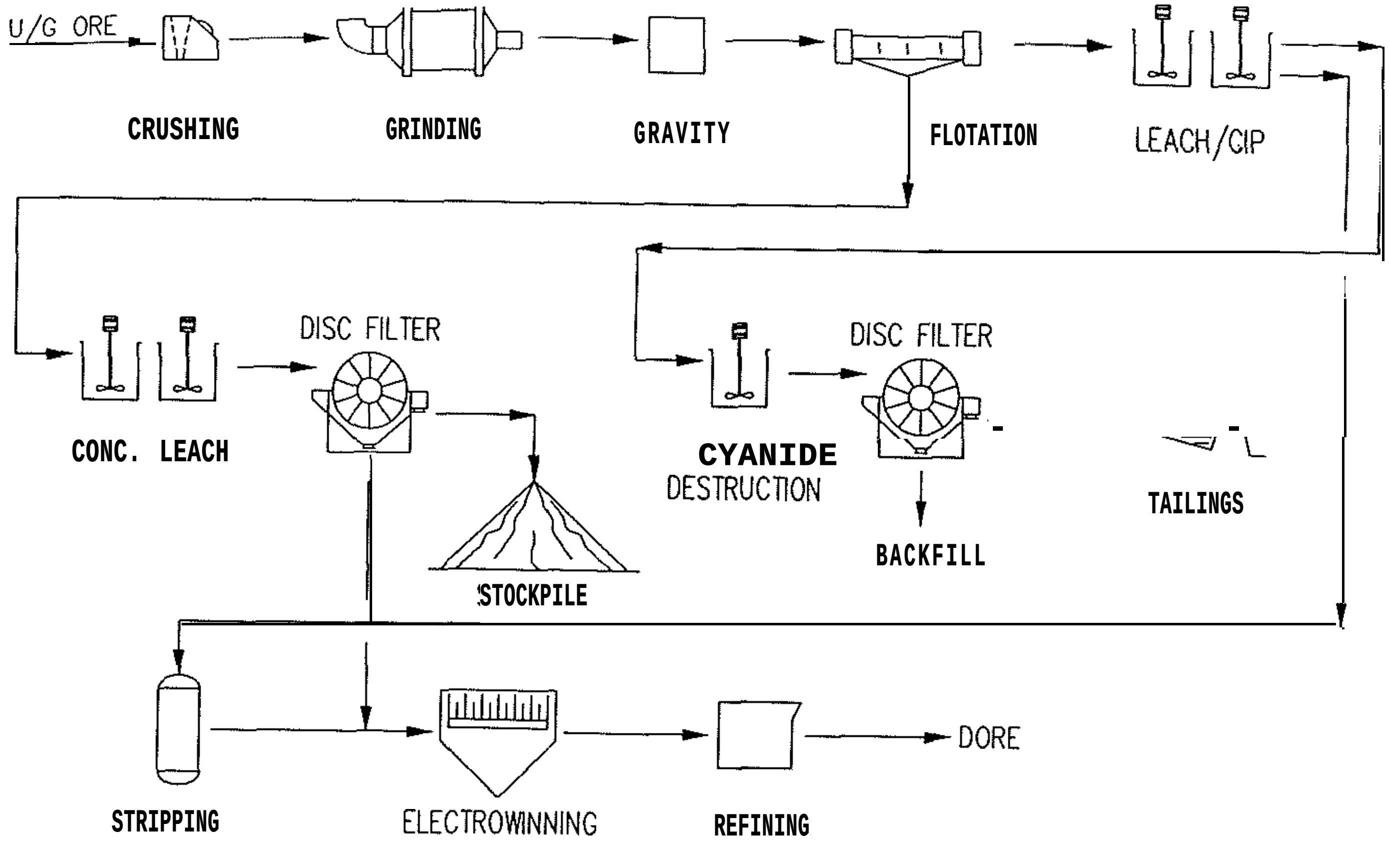

FIGURE 2 – GRIND, FLOTATION, LEACH

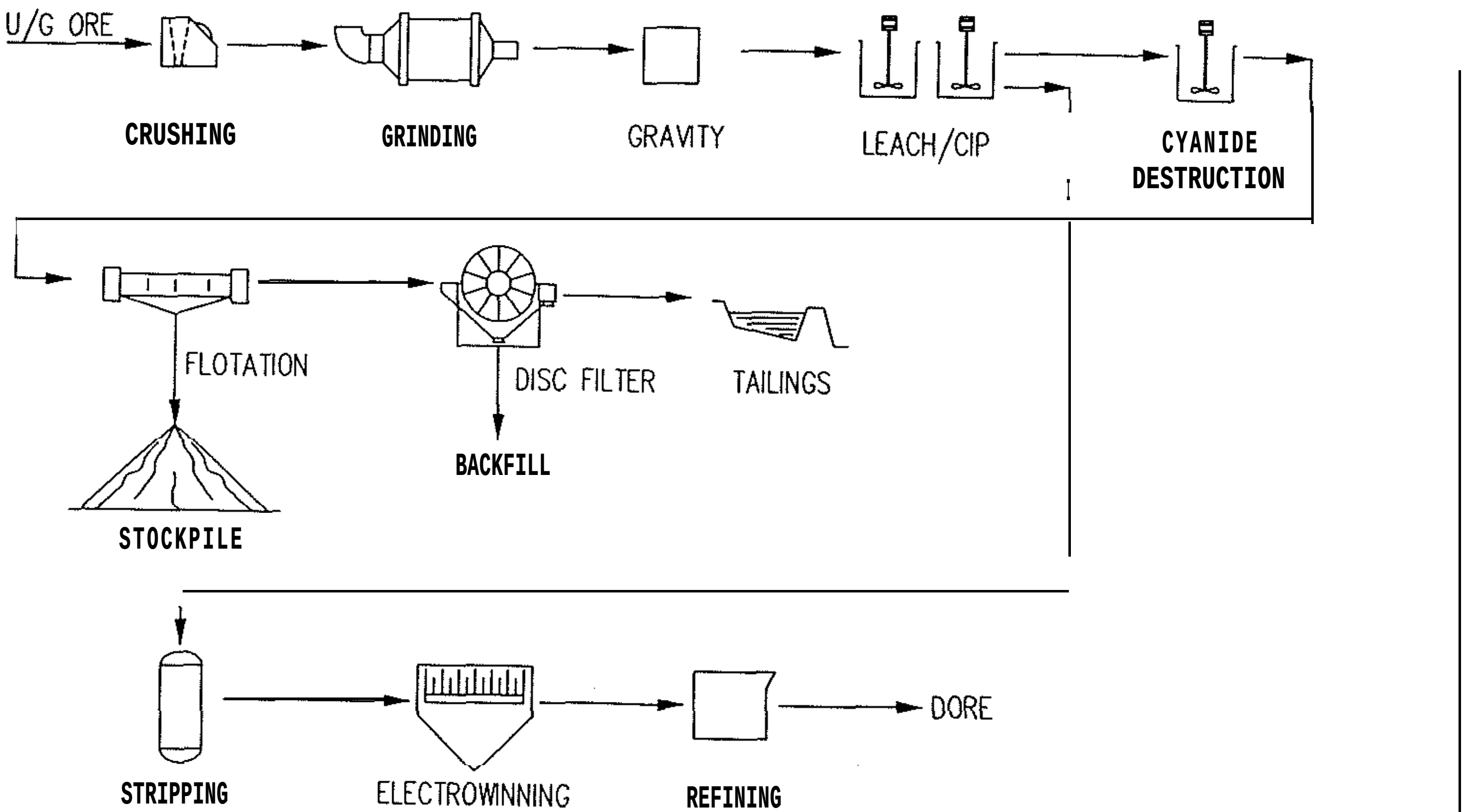

FIGURE 3 – GRIND, LEACH, FLOTATION

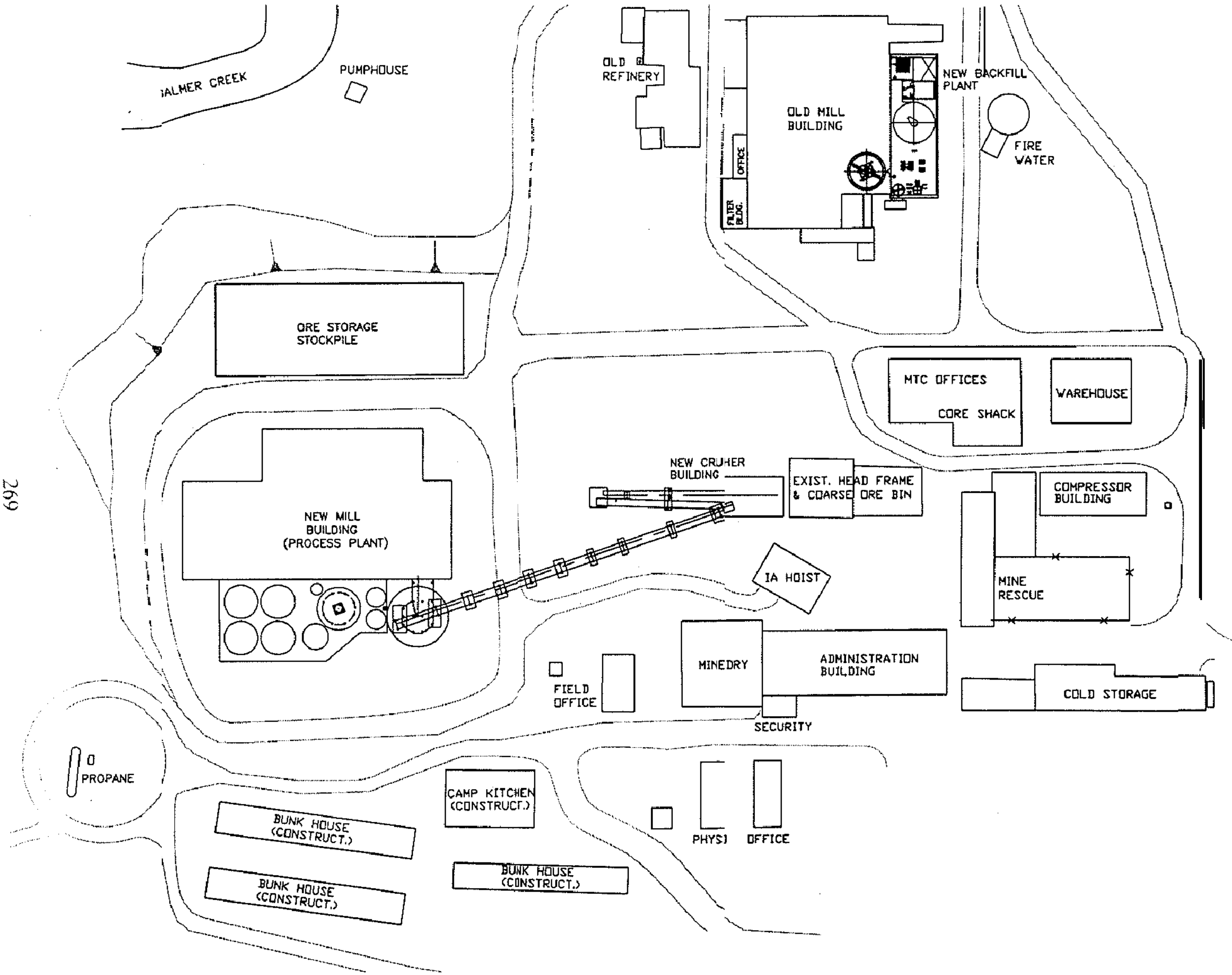

FIGURE 4 – RED LAKE MIE SITE LAYOUT

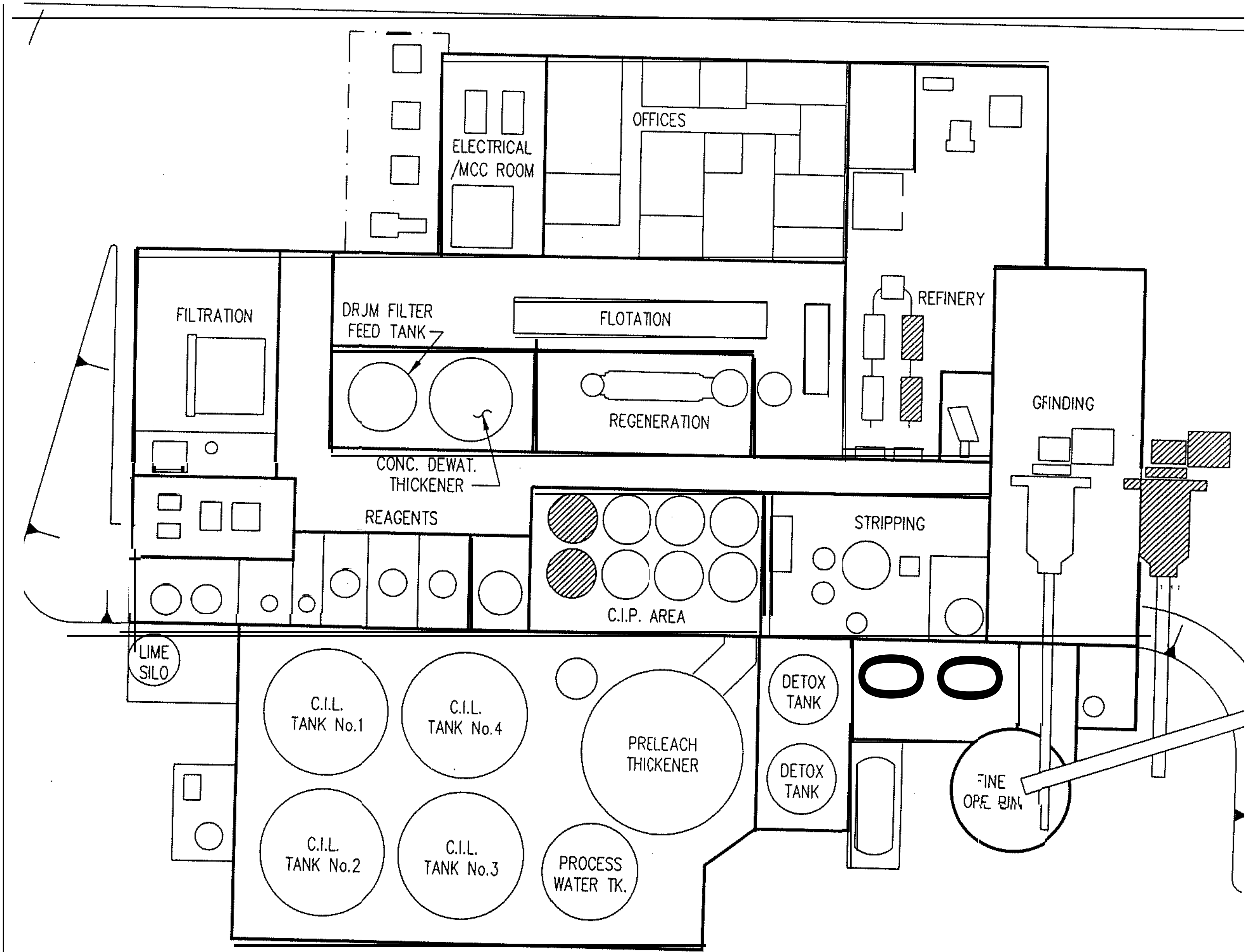

FIGURE 5 - RED LAKE MINE PROCESS PLANT