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SALADIPURA PYRRHOTITE
PROJECT
PHASE II

701

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16371

UNIDO/PPCL

Process Development and Engineering Study

By

Sherritt Gordon Mines Limited

SALACIPURA PYRRHOTITE
PROJECT
PHASE II

CONFIDENTIAL

June 1987

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June 25, 1987

Mr. D. Gardellin
United Nations Industrial
Development Organization (UNIDO)
Purchase and Contract Service
Division of Industrial Operations (PAC/DIO)
P.O. Box 300
A-1400
Vienna, Austria

Dear Mr. Gardellin:

Subject: Saladipura Pyrrhotite Study

Enclosed are 50 copies of the final report on the Phase II work carried out by Sherritt on the recovery of sulphur from Saladipura pyrrhotite. In addition, three (3) copies have been transmitted to Mr. T. N. Jaggi, Managing Director of Pyrites, Phosphates and Chemicals Ltd. (PPCL). The report describes the results of the laboratory testwork and preliminary process engineering studies on the recovery of sulphur, using Sherritt's pressure oxidation process, from Saladipura pyrrhotite concentrate. The commercial plant would treat 400 t/d of pyrrhotite concentrate to produce 102 t/d of elemental sulphur as flotation concentrate. The flotation concentrate could be further upgraded to produce 87 t/d of elemental sulphur.

A process, as defined in the Phase I study on Amjhore pyrite, was confirmed to treat Saladipura pyrrhotite concentrate analyzing 52% iron and 36% sulphur. The flowsheet is similar to the flowsheet in the Phase I report, except there are now two autoclaves operating in parallel instead of a single autoclave due to requirements for a lower feed pulp density. Ground concentrate is pressure oxidized at 150°C using oxygen in the two autoclaves. The autoclave discharge is flash cooled. The flashed slurry undergoes liquid solid separation and flotation to produce a flotation concentrate and tailings slurry. The tailings are neutralized with limestone and lime and are then sent to the tailings pond. The flotation concentrate could be considered as final product, or it could be upgraded by hot filtration to produce a clean sulphur product and filter cake residue.

Sherritt Gordon Mines Limited

Sherritt has confirmed in the laboratory that Saladipura pyrrhotite concentrate can be subjected to pressure oxidation followed by flotation to recover sulphur.

The estimated capital cost for the pressure oxidation and sulphur flotation plant, consisting of grinding, pressure oxidation, liquid solid separation, tailings neutralization, flotation, and oxygen plant is $21.4 million (U.S.). If a hot filtration sulphur recovery circuit is also included, the capital cost will increase to $22.9 million (U.S.). All costs are estimated on the same basis as used in the Phase I report (i.e. third quarter of 1983 for an Indian location). Annual operating costs are estimated to be $3.25 million (U.S.) for the circuit, excluding hot filtration. If the hot filtration circuit is included, annual operating costs would increase to $3.5 million (U.S.). Operating cost estimates were based on the information supplied for the Phase I study.

The next phase of work leading to the commercialization of this project is Phase III, detailed minipilot plant demonstration of the process and preparation of a process engineering design package. Prior to commencing Phase III, it would be necessary to agree to terms under which PPCL would be licensed by Sherritt to use Sherritt Technology in the commercial scale plant, including a non-disclosure agreement between Sherritt and PPCL.

In making changes to the Phase II draft report, we have addressed the comments by PPCL and made appropriate modifications to the report. The use of air rather than oxygen for this process is not appropriate since the large amount of elemental sulphur and sulphidic material in the autoclave slurry would lead to excessive foaming and incomplete reaction in the autoclave. The recovery of elemental sulphur from the autoclave discharge is still by hot sulphur filtration rather than by dissolution in carbon disulphide, since this latter method results, in practice, in excessive reagent consumption.

Yours truly,

D. Robert Weir

D. Robert Weir
Manager, External Technology

DRW/dz
Enclosures

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1.0 SUMMARY

1.1 LABORATORY STUDY

The primary objectives of the laboratory study were: confirmation of the applicability of the process developed in Phase I; and, generation of process design criteria, for a commercial facility for the recovery of elemental sulphur from natural pyrrhotite from the Saladipura deposit. The process chosen for the engineering study is depicted in Figure 1.1

The Saladipura pyrrhotite concentrate, received for the study, contained 52.1% Fe, 35.6% S, and 3.6% SiO_2. Pyrrhotite accounted for about 86% of the sulphur, with the remainder present as pyrite. Five batch pressure oxidation tests were conducted at 150°C, 1 h retention time, 820 kPa total pressure, using oxygen, on concentrate reground to 93% minus 44 m. The extent of sulphur oxidation was 91%, with a 77% yield of elemental sulphur. The yield of elemental sulphur from pyrrhotite, which was essentially completely reacted, was almost 90%. The degree of sulphur oxidation to the sulphate form was 14%. The oxidation tests were followed by sulphur recovery, either by screening or flotation. Recovery of sulphur by screening was in the range of 66 to 72%. The coarse fraction corresponded to 20% of the total residue weight, and assayed 73 to 77% S°. Pyrite, coated and/or occluded within the elemental sulphur was the major diluent. Batch flotation recovered about 87% of the elemental sulphur, in a concentrate containing 61 to 64% S°, with unreacted pyrite and the carry over of iron oxides being the major cause of the lower than desired sulphur grade.

A 24 h continuous oxidation run was conducted at 150°C, 820 kPa total pressure, with oxygen as the oxidant, at 16 to 17% solids, on concentrate reground to 83% minus 44 m. The oxidation was conducted in a multicompartment horizontal autoclave, operated in series with a thickener. Thickener overflow was recycled to the autoclave for temperature control and to provide iron and acid to promote the oxidation. Thickener underflow was collected for batch and continuous flotation testwork. Solids feed rates were 6.4 kg/h for the first 9 h of the run, and 4.1 kg/h for 15 h, corresponding to retention times of 1.0 and 1.5 h, respectively. The degree of sulphur oxidation was, on average, 86.5%, with an elemental sulphur yield of 74%, and oxidation of about 12.5% of the sulphur to sulphate. The pyrrhotite reacted rapidly in the first compartment, and the 1 h retention time appears adequate. The oxidation residues typically assayed 39.4% Fe, 20.1% S°, 1.65% $S(SO_4)$, and 26.4% S, and were considerably finer (98.6% minus 74 m) than the residues produced in the batch oxidation tests.

Batch flotation tests on the continuous oxidation products yielded rougher concentrates containing 62 to 69% S°, with recoveries of 87 to 92% of the elemental sulphur. The highest grade of cleaner concentrate was 8.1% Fe, 81.3% S°, and 88.2% S, with pyrite being the major diluent of the elemental sulphur product. Two continuous flotation tests were conducted in a circuit consisting of rougher, rougher scavenger, cleaner, and cleaner scavenger circuits. Cleaner concentrate grades were initially 61% S°, and improved to 73.5% S° with dilution of the flotation pulp from about 15 to 7% solids. Recoveries of elemental sulphur were in the 94 to 96% range, with scavenger tailings containing 1.16 to 1.72% S°. It is expected that with further circuit and parameter modifications, a higher grade of product with a recovery of 95% or better can be achieved. A practical upper limit on concentrate grade is about 76% S°. The projected recovery of elemental sulphur from the flotation concentrates by melting and hot filtration is about 83 to 86%, for an overall recovery from the pyrrhotite concentrate of about 59%. The filter cake, containing pyrite, iron oxides, and assaying 30 to 35% S°, should be suitable feed for a pyrite roaster.

It is noted that the yield of elemental sulphur in the pressure oxidation, the grade of elemental sulphur in a flotation concentrate, and the ultimate recovery by hot filtration are very much dependent on the relative amounts of pyrrhotite and pyrite in the initial concentrate. Overall sulphur recovery to flotation concentrate is projected to be about 82%. If the pyrrhotite concentrates are expected to contain appreciable amounts of pyrite, consideration should be given to the direct roasting of the flotation concentrates derived from the pressure oxidation (in effect, sulphur enriched, gangue free pyrite) in the pyrite roasters. The suitability of the pyrite roasters for this application would have to be confirmed and any additional financial requirements defined.

1.2 ENGINEERING STUDY

The order of magnitude capital costs for a pressure oxidation and flotation circuit to treat 400 t/d of Saladipura pyrrhotite concentrate to produce 135 t/d of flotation concentrate containing 102 t/d of elemental sulphur, is $21 400 000 (U.S.) for an Indian location.

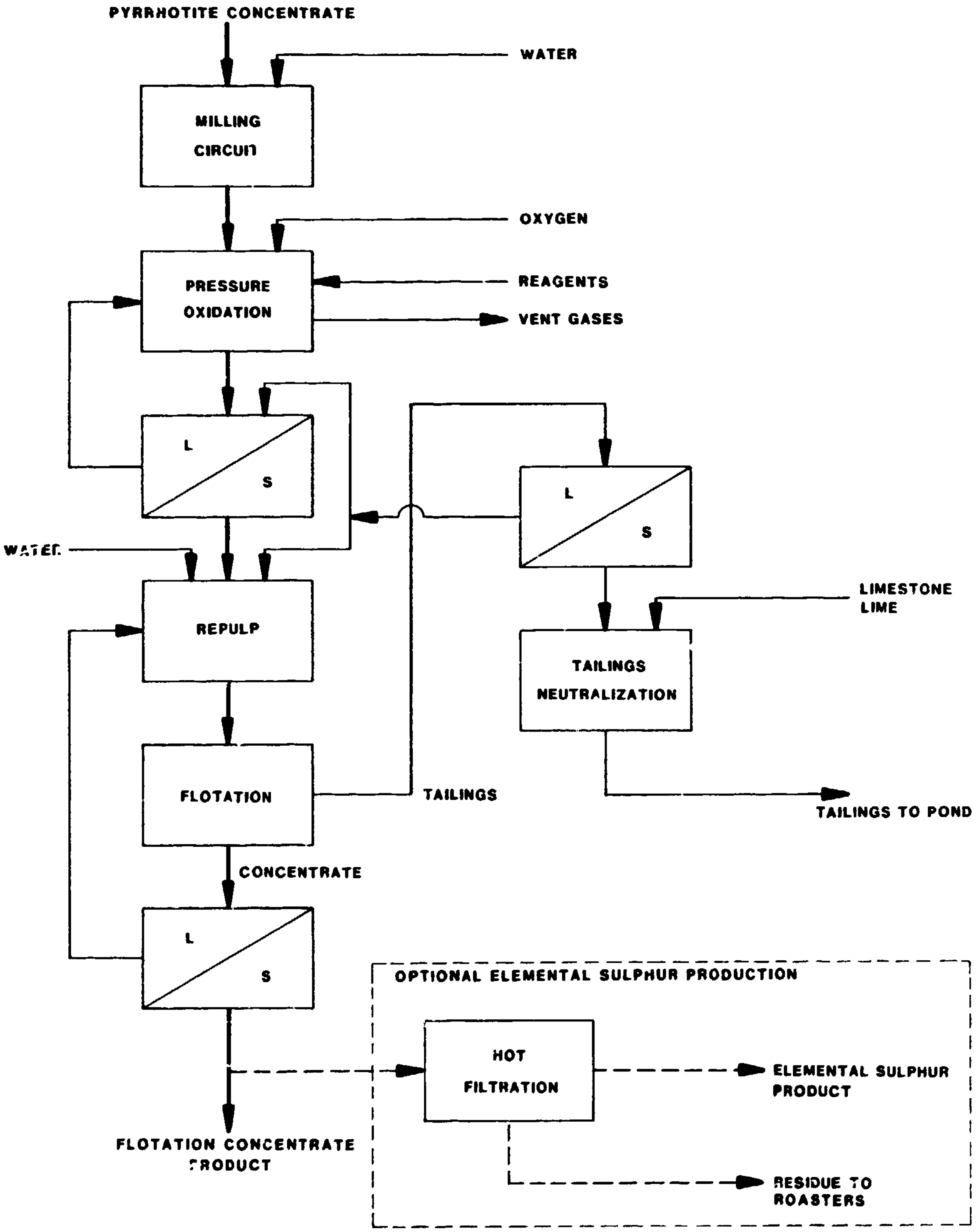
	$'000 (U.S.)
Installed Equipment	7 000
Utilities and Auxiliaries	
- oxygen plant	5 100
- others	600
Indirects, Offsites	1 700
Engineering and Licensing Fee	3 400
Contingency	3 600
Total Capital	21 400
i.e.	$21 400 000

If the flotation concentrate is further treated to produce pure elemental sulphur via hot filtration, the circuit would produce 87 t/d of elemental sulphur and the total capital cost would increase to $22.9 million (U.S.). An estimate of other capital costs, such as working capital, are provided in section 5.6 of this report to enable PPCL to carry out an economic evaluation of this project.

The annual operating costs for the pressure oxidation and flotation circuit are shown below.

	Cost ($'000 U.S.)
Materials and Supplies	800
Utilities	2 140
Labour	40
Maintenance	260
Total	3 240
Say	$3 250 000

If the hot filtration circuit is included, the annual operating costs would increase to $3.5 million.



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SHERRITT GORDON MINES LIMITED
FORT SASKATCHEWAN, ALBERTA, CANADA

SALADIPURA
PYRRHOTITE PROJECT
BLOCK
FLOWSHEET

DWN.	DATE	CKD.	ENG.	APPL. ENG.	D.O. APPL.
LJS	JAN 19/85				

FIG No.	JOB No.	ISSUE
1.1	7987	0
ORIGINATOR	E. McCONAGHY	

2.0 INTRODUCTION

In Phase I of Sherritt's study on the recovery of elemental sulphur from Amjhore pyrite, supplied by Pyrites, Phosphates and Chemicals Ltd. (PPCL), Sherritt had defined a process, based on pressure oxidation, to recover elemental sulphur from a pyrrhotite matte. Batch and continuous pressure oxidation tests had been conducted at 100°C, predominantly on the matte. The continuous pressure oxidation run, of 32 h duration, also included an 11 h period on natural pyrrhotite from the Saladipura deposit. Conversion of the sulphidic sulphur of the matte to elemental sulphur exceeded 85%, and more than 95% of the elemental sulphur was recoverable in a primary or rougher flotation concentrate, grading 44% S°. Attempts to upgrade the product prior to hot filtration were, however, unsuccessful. Subsequent batch pressure oxidation tests defined an oxidation temperature of 150°C, and the use of oxidation additives, as capable of producing high yields of elemental sulphur and producing oxidation residues amenable to upgrading to a high elemental sulphur content by flotation. The Phase I process engineering study was consequently based on a process incorporating pressure oxidation at 150°C. A draft of Sherritt's Phase I report was issued to PPCL on December 19, 1983, and the final report was transmitted to the United Nations Industrial Development Organization (UNIDO) on April 27, 1984.

On December 30, 1983, Sherritt was advised by PPCL that the production of elemental sulphur via the flash smelting and matte production from pyrite concentrate did not appear economically attractive. Sherritt was therefore requested to pursue the Phase II study on the direct production of elemental sulphur by pressure oxidation of natural pyrrhotite from Saladipura. The scope of work for the laboratory study for Phase II was consequently defined as follows:

grinding of Saladipura pyrrhotite concentrate;

batch pressure oxidation and residue flotation tests to confirm conditions for continuous testwork;

one continuous pressure oxidation test for a period of 24 hours, including autoclaving, liquid-solid separation and flotation of sulphur from residue;

batch liquid-solid separation tests on flotation concentrate and tailings slurries;

hot filtration of flotation concentrate to produce elemental sulphur; and

definition of tailings solution composition.

The results of the laboratory tests were to be used to confirm the process design criteria.

The engineering study was to include a description of the process, process flowsheet, an equipment list, an outline specification of the major equipment, and order of magnitude capital and operating cost estimates, along with estimates of working capital, start-up costs, and pre-production costs.

This report on the treatment of Saladipura pyrrhotite concentrate summarizes the laboratory tests performed, and includes results of the process engineering study.

3.0 FEED MATERIAL, PROCESS CHEMISTRY, EQUIPMENT AND EXPERIMENTAL PROCEDURES

An account is provided in this section of the chemical analysis and mineralogy of the concentrate, of the pressure oxidation process chemistry, and of the laboratory equipment and experimental procedures employed in the testwork.

3.1 FEED MATERIAL

Six bags of Saladipura pyrrhotite concentrate were received at the Sherritt Research Centre on June 29, 1984, and were assigned SRC number 2254.

3.1.1 Chemical Analysis and Mineralogy

The total weight of concentrate in each bag, and partial chemical and screen analyses of representative samples from each bag are given below. More complete chemical analyses are provided in Table 3.1.

Bag	Weight kg	Analysis, %			M.R.*	% minus
		Fe	S	SiO_2	S/Fe	44 μm
1	48.7	54.1	37.4	2.26	1,20	27
2	29.9	53.7	37.0	2.30	1.20	26
3	44.9	52.1	36.8	3.49	1.23	27
4	47.6	48.2	33.2	5.38	1.20	35
5	40.0	48.3	32.6	5.28	1.18	31
6	39.6	53.7	36.6	2.81	1.19	26
Average	250.7	51.5	35.5	3.65	1.20	29

*molar ratio

The material in bags 4 and 5 was slightly different in appearance from that in the other bags. This is reflected in the slightly lower sulphidic and higher gangue contents of the concentrates in bags 4 and 5. The sulphur:iron molar ratio was consistent in all the bags, at about 1.2:1.0.

The contents of all of the bags were blended in a cement mixer and divided into five pails. A 10 kg portion of the blended solids was retained for analysis and for batch pressure oxidation testing. The composite analyzed 52.1% Fe, 35.6% S, and 3.62% SiO_2, in good agreement with the weighted analyses of the concentrates in the six bags and with the 35.66% S quoted by PPCL. The sample of concentrate was wet ground in a steel ball mill, to 93.2% minus 44 µm, prior to batch testing. The specific gravity of the concentrate was 4.29. More complete chemical and screen analyses are provided in Tables 3.1 and 3.2.

In preparation for the continuous pressure oxidation test, the remainder of the concentrate was wet ground, in five separate batches in an 88 cm long X 58 cm diameter (inside dimensions) rubber lined ball mill, with a working volume of 50 L. 175 kg of steel balls, ranging from 1.5 cm to 5 cm in diameter were used as the grinding media. The ball mill was operated at 39 rev/min. The reground products were collected as a 70 to 72% solids slurry in a rotary mode in 20 L pails to assure homogeneity. The two hours of grinding reduced the concentrate size to 83% minus 44 µm. Samples of concentrate were collected for analysis for Fe and S and the analyses are given below. The concentrates from pails 2 and 5 were also analyzed for minor components. These analyses are given in Table 3.1.

Pail No.	1	2	3	4	5	Avg.
% Fe	50.6	50.3	50.7	50.5	50.9	50.6
% S	34.1	34.7	34.9	34.8	34.8	34.6
S/Fe M.R.	1.17	1.20	1.20	1.20	1.19	1.19

The concentrate analyses provide by PPCL are given below.

Total Sulphur, %	Pyrrhotite Sulphur, %	Pyrite Sulphur, %	SiO_2, %
35.66	30.18	5.48	4.1

Assuming a zero yield of elemental sulphur from the oxidation of pyrite, the maximum yield of elemental sulphur attainable from the concentrate upon complete oxidation of the pyrrhotite (assuming no oxidation to sulphate) is therefore 84.6%. Also, assuming a chemical composition of Fe_7S_8 for pyrrhotite, and FeS_2 for pyrite, the calculated total amount of iron present as iron sulphides is 50.9%, of which 46.1% is as pyrrhotite and 4.8% is as pyrite.

X-Ray diffraction analysis of the concentrate confirmed pyrrhotite and pyrite as the major sulphides. A point count of the "as received" concentrate indicated approximately 75% pyrrhotite, 10% pyrite, (mostly free grains, with less than 1% interlocking) and 15% gangue and other minor minerals. On this basis the calculated analyses of the concentrate were 53.3% Fe, and 37.5% S, with the sulphur of pyrrhotite accounting for 83.6% of the total sulphur.

As a further determination of the distribution of the sulphidic sulphur between pyrrhotite and pyrite, a sample of ground concentrate (feed for the continuous test) was subjected to a leach in acidified ferric chloride solution at 95°C for 5 h, with excess ferric chloride, followed by organic extraction of elemental sulphur from the leach residue, and analysis of the final residue. Pyrite is virtually inert in a ferric chloride leach under these conditions, such that any residual sulphidic sulphur corresponds to the pyritic sulphur. The analyses of the feed concentrate and of the elemental sulphur free residue are given below.

	Wt.	Al	Ca	Fe	Mg	SiO_2	S	$S(SO_4)$	S°
Concentrate, %	100	0.71	0.24	50.3	0.28	3.53	34.6	0.28	0.7
Residue, %	14.2	2.65	0.76	24.7	0.23	12.0	29.8	0.08	<0.1
% of Feed	14.2	53.0	45.0	7.0	12.0	(104)	12.4	-	-

Microscopic examination of the final residue confirmed the absence of pyrrhotite, and pyrite was the only remaining sulphide. The leach had extracted 47% Al, 55% Ca, 93% Fe, and 88% Mg, and the remaining sulphidic sulphur corresponded to 12.4% of the feed sulphur.

On the basis of the three estimates of the sulphide mineralogy of the concentrate, it was assumed for the purposes of this study, that the pyrite accounted for about 14% of the sulphur, and the remaining 86% was present predominantly as pyrrhotite. As a first approximation, assuming total conversion of the pyrrhotitic sulphur solely to elemental sulphur, with oxidation of neither the elemental sulphur nor pyrite to sulphate, the highest grade of flotation concentrate that could be produced from the oxidation residue, assuming total recovery of the elemental sulphur and the unreacted pyrite in flotation would analyze 10.9% Fe, 89.1% S, and 76.7% S°.

3.2 PROCESS CHEMISTRY

The oxidation of pyrrhotite in a sulphuric acid medium to yield elemental sulphur and ferrous sulphate may be depicted as follows.

$$2Fe_7S_8 + 14H_2SO_4 + 7\ O_2 \text{ ---> } 14FeSO_4 + 16S^\circ + 14H_2O \qquad (1)$$

This reaction is, however, impractically slow in the absence of a species which will facilitate oxygen transfer. One such species is dissolved iron, and in fact the reaction is usually the sum of the reactions below.

$$Fe_7S_8 + 7Fe_2(SO_4)_3 \text{ ---> } 21FeSO_4 + 8S^\circ \qquad (2)$$

$$14FeSO_4 + 7H_2SO_4 + 3.5\ O_2 \text{ ---> } 7Fe_2(SO_4)_3 + 7H_2O \qquad (3)$$

Under the pressure oxidation conditions, the bulk of the ferric iron is hydrolyzed and precipitated as hematite, resulting in regeneration of sulphuric acid.

$$7Fe_2(SO_4)_3 + 21\ H_2O \text{ ---> } 7Fe_2O_3 + 21H_2SO_4 \qquad (4)$$

The overall reaction for the oxidation of pyrrhotite and hydrolysis of iron as hematite is therefore the sum of reactions (1), (3), and (4).

$$2Fe_7S_8 + 10.5\ O_2 \text{ ---> } 7Fe_2O_3 + 16S^\circ \qquad (5)$$

Some of the ferric sulphate may also be hydrolyzed and precipitated as basic ferric sulphate and as a hydronium jarosite, removing sulphate from solution.

$$Fe_2(SO_4)_3 + 2H_2O \text{ ---> } 2Fe(OH)SO_4 + H_2SO_4 \qquad (6)$$

$$3Fe_2(SO_4)_3 + 14H_2O \text{ ---> } 2(H_3O)Fe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \qquad (7)$$

The stoichiometric requirement of oxygen, for a concentrate containing 35.6% S, of which 86% is present as pyrrhotite, according to the reactions represented by equation (5) is 200 kg/t.

Under the pressure oxidation conditions, a portion of the sulphur of pyrrhotite will be oxidized to the sulphate form, either directly, or through further oxidation of the elemental sulphur.

$$2S^\circ + 3\ O_2 + 2H_2O \longrightarrow 2H_2SO_4 \qquad (8)$$

A minor portion of the pyrite may also react. In this case, the product of the oxidation of the sulphur of pyrite is predominantly sulphate, and the overall reaction is given below.

$$2FeS_2 + 7.5\ O_2 + 4H_2O \longrightarrow Fe_2O_3 + 4H_2SO_4 \qquad (9)$$

The extent of the undesirable reactions (8) and (9) can be controlled, to some degree, by maintaining high sulphuric acid and ferric sulphate concentrations during pressure oxidation.

3.3 EQUIPMENT AND EXPERIMENTAL PROCEDURES

Standard Sherritt experimental apparatus and procedures were employed for the batch and continuous pressure oxidation and flotation testwork.

3.3.1 Batch Pressure Oxidation

Batch pressure oxidation tests were conducted in a titanium lined laboratory autoclave. The autoclave is 12.7 cm in diameter and 31.1 cm deep, and has a working volume of 2.5 L. All wetted internal parts are titanium, and include a thermowell. gas sparge tube, baffles, and a central impeller shaft. The experimental procedures were essentially as described in Section 5.4.2 of the Phase I Report, with the exception that all the oxidation tests were conducted at 150°C.

3.3.2 Continuous Pressure Oxidation

The equipment flowsheet for the continuous pressure oxidation is shown in Figure 3.1. The oxidation was conducted in a six compartment horizontal autoclave, having a working volume of 30 L. Appropriate agitators were installed in each compartment, which can be independently heated and cooled. At the start of the run, the autoclave and the oxidation thickener were filled with synthetic thickener overflow solution, containing 9 g/L Fe, 2 g/L Fe^{2+}, and 31 g/L H_2SO_4, and oxygen was sparged into each compartment at the required flow rates. Heating was commenced, and when the temperature reached 140°C, the feed and discharge systems were put into operation. Pyrrhotite slurry and solution were pumped separately into the first compartment. The autoclave temperature was maintained at 150°C, and the working pressure was 820 kPa, using oxygen as the oxidant. The autoclave discharge rate was automatically controlled to match the feed rate and to maintain a constant level in the autoclave. The discharge slurry proceeded via a steam jacketed transfer pipe to a flash tank and then to a thickener feed tank, where it was diluted from about 20% solids to 10% solids with thickener overflow. Flocculant solution (Percol 351) was added to the dilute slurry and the slurry was settled in a 29.2 cm diameter plexiglas thickener, with a cross-sectional area of 0.067 m^2 and a working volume of 97 L. Thickener overflow was collected and recycled continuously to the autoclave. Water was added, as required, to the thickener feed tank as make up for evaporative losses. Thickener underflow was discharged continuously at a rate which maximized the underflow pulp density while maintaining a constant mud level in the thickener, and was stored for flotation testwork.

Samples were taken at predetermined time intervals from the autoclave circuit to provide a measure of the rate of sulphur oxidation, as well as solution concentrations across the autoclave. Several sets of complete profiles were collected at eight sampling points, namely the feed slurry, the six individual compartment slurries, and the discharge slurry. Solutions were analyzed for Fe, Fe^{2+} and H_2SO_4, while washed, dried solids were analyzed for Fe, S, S°, and $S(SO_4)$.

The continuous oxidation run was conducted for 24 h. Further details on the operating parameters and conditions in the oxidation and thickening circuits are provided in Section 4.

3.3.3 Flotation Testwork

Batch flotation tests were conducted in 2,4, and 8 L cells similar to those described in the Phase I report. The continuous flotation tests were conducted in the same circuit that had been used in the Phase I study, but with a different arrangement of the operations. The circuit is shown schematically in Figure 3.2 and consisted of three rougher cells, three rougher scavenger cells, three cleaner cells, and three cleaner scavenger cells. Fresh thickener underflow slurry was diluted to the desired pulp density with water and conditioned with 70 g/t Aerofroth 73. The conditioned slurry entered the rougher cells at 100 mL/min. Rougher tailings were scavenged in the scavenger circuit, and scavenger concentrate was returned to the rougher circuit, while the scavenger tailings were collected as product. Rougher concentrate was diluted with water before being fed to the cleaner circuit. Cleaner concentrate was collected as product. Cleaner tailings were treated in the cleaner scavenging cells, and cleaner scavenger concentrate was recycled to the cleaner cells, while cleaner scavenger tailings were recycled to the rougher circuit.

Table 3.1 **Chemical Composition of Saladipura Pyrrhotite Concentrate**

Species		"As Received"	Ball Milled Concentrate		
			Pail 2	Pail 5	Composite*
Aluminum	Al	n.a.	n.a.	n.a.	0.71
Arsenic	As	0.12	0.14	0.13	0.13
Calcium	Ca	0.40	0.33	0.31	0.24
Cobalt	Co	0.016	n.a.	n.a.	n.a.
Chromium	Cr	0.013	n.a.	n.a.	n.a.
Copper	Cu	0.12	0.11	0.11	0.11
Iron	Fe	52.1	50.3	50.9	50.3
Lead	Pb	0.24	0.13	0.11	n.a.
Magnesium	Mg	0.30	0.28	0.28	0.28
Manganese	Mn	0.04	0.03	0.03	0.03
Molybdenum	Mo	0.013	n.a.	n.a.	n.a.
Nickel	Ni	0.011	n.a.	n.a.	n.a.
Selenium	Se	0.0037	n.a.	n.a.	n.a.
Silica	SiO_2	3.62	3.1	3.0	3.53
Sulphur, total	S	35.6	34.7	34.8	34.6
Sulphur, Sulphate	$S(SO_4)$	0.32	n.a.	n.a.	0.28
Sulphur, Elemental	S°	<0.1	n.a.	n.a.	0.7
Tellurium	Te	0.0022	n.a.	n.a.	n.a.
Titanium	Ti	0.074	n.a.	n.a.	n.a.
Zinc	Zn	0.66	0.65	0.63	0.62

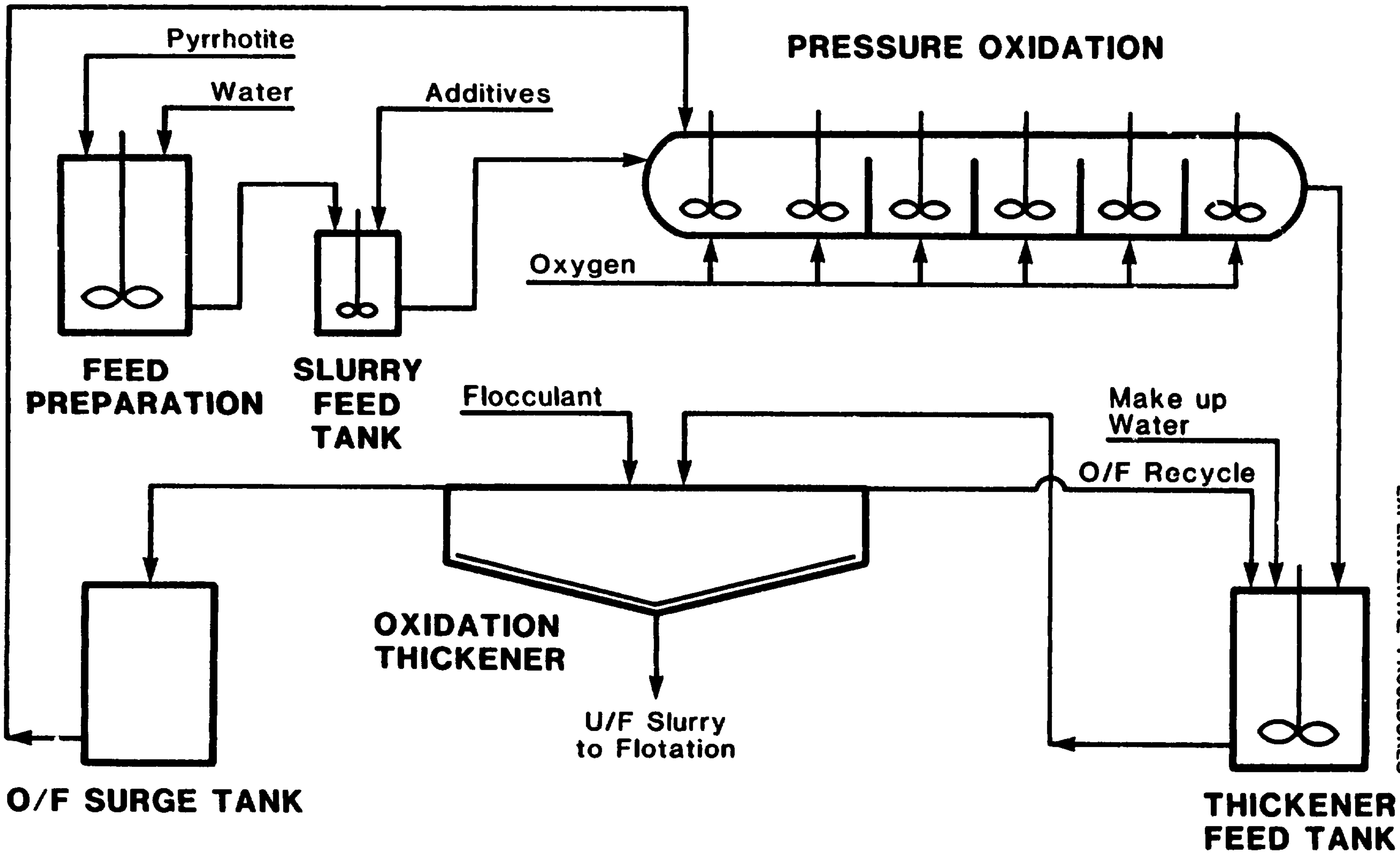
n.a. = not analysed

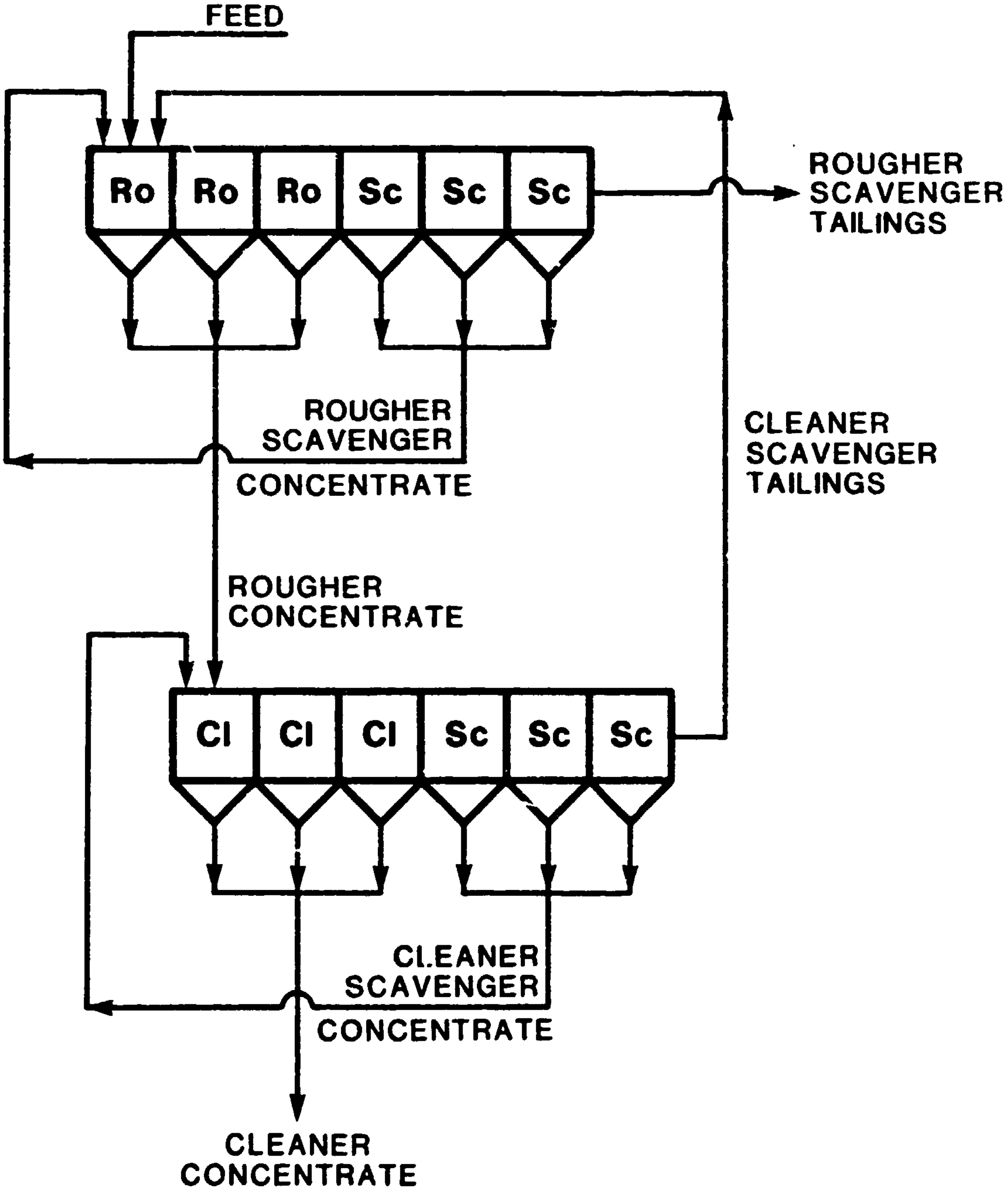
*Composite of Pails 1 to 5 of concentrate ground for continuous oxidation run.

Balance to 100% includes oxides of aluminum, calcium, iron, and magnesium plus bound water.

Table 3.2 Screen Analysis of Saladipura Pyrrhotite Concentrate

Screen Size		% Passing	
Tyler Mesh	µm	As Received	Ball Milled
115	125	82.1	100
170	88	62.1	98.7
250	62	46.0	94.9
325	44	30.2	82.9
	31	21.1	69.7
	22	14.6	57.6
	16	8.0	44.2
	11	4.5	34.6
	7.8	3.7	28.0
	5.5	3.5	24.0
	3.9	1.4	14.1
	2.8	0.6	4.2


Pyrrhotite
Water
Additives
PRESSURE OXIDATION
Oxygen
FEED PREPARATION
SLURRY FEED TANK
Flocculant
Make up Water
O/F Recycle
OXIDATION THICKENER
U/F Slurry to Flotation
O/F SURGE TANK
THICKENER FEED TANK


FEED
Ro
Ro
Ro
Sc
Sc
Sc
ROUGHER SCAVENGER TAILINGS
ROUGHER SCAVENGER CONCENTRATE
CLEANER SCAVENGER TAILINGS
ROUGHER CONCENTRATE
Cl
Cl
Cl
Sc
Sc
Sc
CLEANER SCAVENGER CONCENTRATE
CLEANER CONCENTRATE

4.0 DISCUSSION OF RESULTS

Batch pressure oxidation tests were conducted on the concentrate, at 150°C, to determine the yield of elemental sulphur, the extent of the oxidation of pyrite and elemental sulphur to the sulphate form, and to establish the conditions for the continuous testwork. Batch flotation testwork was conducted to establish the grade and recovery of elemental sulphur containing product. This was followed by a continuous 24 h pressure oxidation run, and batch and continuous flotation testwork on the discharge slurry and thickener underflow slurry. Cylinder settling tests were conducted on the product slurries from the continuous flotation testwork.

4.1 BATCH PRESSURE OXIDATION TESTS

Five batch pressure oxidation tests were conducted at 150°C and at a total pressure of 820 kPa, using oxygen as the oxidant. The test slurries (2.58 L per charge), containing 12.7% initial solids (150 g concentrate/L solution) and 5 kg oxidation additive/t concentrate, were prepared by repulping 150 mL of a 70% solids slurry of reground concentrate in water, with 2350 mL of synthetic electrolyte, containing 5.5 g/L Fe (as ferric sulphate) and 21 g/L H_2SO_4, to promote the oxidation of the sulphidic sulphur to elemental sulphur. Batch retention times were all 1.0 hour. In tests 1, 3, and 4, the autoclave contents were cooled under agitation prior to discharging, while in tests 2 and 5, the bulk of the autoclave contents was flash discharged at the elevated temperature. The results of the first four tests, conducted as material balance runs, are presented in Tables 4.1 to 4.4, and a discussion of the results is provided below. There were brief temperature excursions, to 165° and 155°C in tests 3 and 4, respectively, with no measurable impairment of elemental sulphur yield. The deportment of the sulphur of the concentrate in the four oxidation tests is summarized below.

Oxidation Test	1	2	3	4
Conversion to elemental sulphur, %	75.6	77.5	76.2	78.8
Oxidation to sulphate sulphur, %	14.4	15.7	12.5	13.2
Unreacted sulphur, %	8.1	7.5	11.2	7.9
Totals accounted for, %	98.1	100.7	99.9	99.9
Deportment of SO_4 to solution, %	9.3	11.1	8.0	9.4
Deportment of SO_4 to residue, %	5.1	4.6	4.5	3.8
Conversion of pyrrhotite to S°, %	87.9	90.1	88.6	91.7

The behaviour of sulphur was relatively consistent in the four tests. The average degree of oxidation was 91.3%, with 77.0% conversion to elemental sulphur, and 14.0% oxidation to sulphate. The deportment of sulphate sulphur was 9.5% to the oxidation liquor and 4.5% to the residue. The residues contained only traces of unreacted pyrrhotite. The extent of oxidation of pyrite, accounting for 14% of the sulphur in the concentrate, was about 38%. The yield of elemental sulphur from pyrrhotite was about 89.5%, with the remainder largely oxidized to sulphate. On this basis, the consumption of oxygen was 284 kg/t.

The oxidation liquors typically contained 11.4 g/L Fe, 2.6 g/L Fe^{2+}, and 20.5 g/L H_2SO_4, compared with a starting electrolyte composition of 5.5 g/L Fe (all Fe^{3+}), and 21.0 g/L H_2SO_4. The extraction of iron averaged 7.0%.

The oxidized solids analyzed, on average, 39.9% Fe, 26.6% S, 22.5% S°, and 1.56% $S(SO_4)$. The oxidation resulted in a solids weight increase of about 21% (range of 19 to 23%). The discharge solids from each test were separated into coarse and fine fractions by screening through either a 125 μm (115 mesh, Tyler scale, tests 1 and 2) or a 149 μm (100 mesh, tests 3 and 4) screen. The analyses of the coarse fractions of the residues are summarized below.

Oxidation Test		1	2	3	4
Coarse fraction as % of total residue		21.1	19.5	19.9	21.3
Analysis	% Fe	10.6	12.4	11.5	8.2
	% S	84.2	84.4	86.7	82.9
	% S°	73.9	76.3	73.7	76.2
	% $S(SO_4)$	0.31	0.35	0.06	0.05
	% (Fe + S)	94.7	96.8	98.2	91.1
% Recovery of S° to coarse fraction		71.7	67.5	66.0	72.5
% Recovery of $S^=$ to coarse fraction		90.8	59.9	78.8	62.8

The coarse solids consisted primarily of elemental sulphur and pyrite, the latter either coated or physically associated with elemental sulphur. The deportment of elemental sulphur to the coarse fraction, analyzing 73.7 to 76.3% S°, was about 69%.

By way of comparison, the analysis of a hypothetical, gangue free sulphur-sulphide product, assuming oxidation of 91.1% of the sulphur (including 100% of the pyrrhotite and 38% of the pyrite) of which 77% was oxidized to elemental sulphur and 14% was oxidized to sulphate, and assuming total recovery of elemental sulphur and unreacted pyrite to such a product, is 8.1% Fe, 91.9% S, and 82.6% S°.

The fifth batch oxidation test was conducted at conditions similar to those of the first four tests, and the product slurry was flash discharged, cooled, and subjected to flotation testing. A rougher flotation was conducted in a 2 L flotation cell, at a natural pulp pH of 2.1, employing 65 g/t Aerofroth 73. The rougher concentrate was subjected to a cleaner flotation, in a 1 L cell, with no additional reagents and two cleaner concentrates were collected for separate analysis, the first corresponding to 12 min flotation time, the second for 4 min. The rougher tailings were screened through a 100 mesh (149 μm) screen to recover any unfloated sulphide-elemental sulphur product. The coarse fraction of the tailings weighed only 0.08 g, corresponding to only 0.02% of the total solids. The analysis and distribution of elemental sulphur in the flotation products is summarized below.

Flotation	Fraction	Weight	Analysis	Distribution, %	
		g	% S°	Weight	S°
Rougher	Feed	430.8	18.7	100.0	100.0
	Rougher Concentrate	143.7	51.2	33.3	91.3
	Rougher Tailings	287.1	2.45	66.7	8.7
Cleaner	Rougher Concentrate	143.7	51.2	33.3	91.3
	Cleaner Con 1	91.1	63.6	21.1	71.9
	Cleaner Con 2	20.3	61.5	4.7	15.5
	Cleaner Cons 1+2 (calc)	111.4	63.2	25.8	87.4
	Cleaner Tailings	32.3	9.81	7.5	3.9

The rougher flotation recovered 91.3% of the elemental sulphur, indicating a need for a scavenger flotation to improve the recovery to >95%. The product grade was improved from 51.2% S°, to 63.6% S° in the cleaner flotation, and was appreciably lower than either the 82.6% S° of a hypothetical clean product, or the 73.7 to 76.3% S° in the coarse screen products of tests 1 to 4, because of considerable carryover of iron oxide residue. Subsequent flotation tests were conducted on the oxidation slurries from the continuous oxidation run.

4.2 CONTINUOUS PRESSURE OXIDATION

The continuous pressure oxidation run was conducted for 24 h, at 150°C, 820 kPa total pressure, with concentrate reground to 83% minus 44 µm, analyzing 50.6% Fe and 34.6% S. The reground concentrate, prepared as an aqueous slurry containing 72% solids, and 5 kg oxidation additive per tonne of concentrate, was fed into the first compartment of the autoclave. Electrolyte, initially synthetic solution, and subsequently thickener overflow, was also fed into the first compartment to adjust the solid content of the oxidation slurry to the desired level. The run was commenced at a solids feed rate of about 6.5 kg/h, with a target initial solid content of 17% and a nominal retention time of 1 hour. The heat generated by the oxidation reactions made temperature control to 150°C difficult, and after 4 h (Period 1A), the flow of solution to compartment 1 was increased, to reduce the initial solid content to about 16%. This minor adjustment, resulting in a reduction of the retention time from 56 to about 53 min, facilitated the temperature control, and the circuit was operated under these feed conditions for an additional 5 h (period 1B). The stoichiometric requirement of oxygen for the complete oxidation of the sulphur of pyrrhotite to elemental sulphur only, was 195 kg/t. Based on the results of the batch tests, which showed oxidation of about 10% of the elemental sulphur and 38% of the pyrite sulphur to sulphate, the actual consumption was anticipated to be 273 kg/t. The amount of oxygen provided during the continuous run was about 505 kg/t.

After the first 9 h, the oxidation retention time was increased to 90 min by reducing the flows of concentrate slurry (to ~4 kg/h concentrate), solution, and oxygen, and the circuit was operated under these conditions for 15 h (Period 2). A summary of the operating parameters and the median conditions for the entire run is provided overleaf. Complete operating data are provided in Tables 4.5 and 4.6.

The continuous run was conducted for 24 h, during which the on stream time was 21 h. There were two disruptions, necessitating shut down of the feed system. The first occurred at 10.5 h, and was due to plugging of the autoclave discharge system with elemental sulphur, including the flash pot and several lines, and required 2.25 h to clear. The second occurred at 13.25 h, again due to plugging of the discharge system, which required 40 min for cleaning and further modification. Consequently, no samples were collected during the 9 to 17 h period of the run.

Period		1A	1B	2
Duration,	h	4	5	15
Solids feed rate,	kg/h	6.52	6.60	4.06
Solution feed rate,	L/h	27.6	30.0	17.4
Compartment 1 initial solids,	%	16.9	16.1	16.8
Oxidation retention time,	min	56	53	90
Temperature, °C	C1	152.7	151.2	148.2
	C2	153.5	151.8	148.3
	C3	152.9	152.0	150.3
	C4	151.3	150.8	150.3
	C5	150.8	150.4	150.7
	C6	149.8	149.6	150.0
Total pressure,	kPa	820	820	820
Oxygen flowrate, L/min	C1	15	15	9
	C2	10	10	6
	C3	8	8	5
	C4	5	5	3
	C5	2	2	2
	C6	1	1	1
	total	41.3	41.3	26.2
Oxygen supply,	kg/t	503	497	513

The results from the continuous oxidation run are presented in Tables 4.7 and 4.8 and Figures 4.1 to 4.3. Solution analyses are presented in Table 4.7, and the autoclave solution profiles for Fe, Fe^{2+}, and H_2SO_4 at 6, 17, and 24 h are plotted in Figures 4.1, 4.2 and 4.3, respectively. Sulphuric acid was consumed rapidly in compartment 1, while the ferrous iron concentration increased sharply and total iron concentration increased slightly. In subsequent compartments, the ferrous iron was oxidized to ferric, such that by compartment 6, 80 to 88% of the total iron in solution was in the ferric state. The total iron concentration declined slightly from the level attained in compartment 1, as ferric iron was precipitated. The precipitation of the iron was accompanied by an increase in acid concentration. The net extraction of iron was less than 5%.

Residue analyses for the autoclave compartment and discharge samples are presented in Table 4.8, together with "apparent" sulphur conversion data. The solids weight gain across the oxidation step was in the range of 22 to 30%, averaging about 27%. As in the case of the Phase I continuous oxidation run, the discharge samples contained consistently higher levels of elemental sulphur than the autoclave compartment samples. The reasons for this apparent discrepancy are related to the difficulty of obtaining truly representative compartment slurry samples, through sample dip pipes located near the bottom of each compartment, from sparged slurries where the elemental sulphur tends to float to the surface. For this reason, the final discharge samples were taken as being more representative of the extent of sulphur oxidation. The "apparent" sulphur oxidation and conversion data from the residue profile samples is therefore useful only as a guide, taken in conjunction with the solution analyses. The results indicate a very rapid rate of sulphur oxidation in the first compartment, and the 1.0 h retention time appears adequate.

The analyses of representative samples of oxidized solids, obtained largely as thickener underflow solids used for subsequent batch and continuous flotation testing are presented below.

Sample	% Weight Gain	Composition, %					% Sulphur Oxidation	
		Fe	S	S°	$S(SO_4)$	$S^=$	Total	to S°
1	23.6	40.7	25.9	19.7	2.98	3.22	88.5	70.4
2	35.2	37.2	27.6	20.2	2.80	4.60	82.0	78.9
3	24.8	40.3	25.6	20.0	2.41	3.19	88.5	72.1
4	26.7	39.7	27.1	20.2	2.32	4.58	83.2	74.0
5	25.8	40.0	26.0	20.3	2.34	3.36	87.8	73.8
6	30.3	38.6	26.0	20.4	3.07	2.53	90.5	76.8
Avg.	27.7	39.4	26.4	20.1	2.65	3.65	86.5	74.2

The degree of sulphur oxidation in the continuous run was, on average, 86.5%, with an elemental sulphur yield of 74.2%, and 12.3% oxidation of sulphur to sulphate. For comparison, the corresponding values from the batch tests had been 91.5, 77.0, and 14.5% respectively. The oxidation residues from the continuous run contained only traces of unreacted pyrrhotite, such that the yield of elemental sulphur from pyrrhotite was about 86.3%, compared with 89.5% in the batch tests. The apparent extent of pyrite oxidation in the continuous run was considerably lower than in the batch tests. This may have been due, in part, to the coarser size of the concentrate used in the continuous run i.e., 83% minus 44 µm compared to 93% minus 44 µm in the batch tests.

Towards the end of the continuous oxidation run, a 2 L sample of discharge slurry was collected. The solids were separated into three size fractions by screening on 100 and 200 mesh (149 and 74 µm) screens, and the fractions were subjected to chemical analysis and mineralogical examination. The results are presented in Table 4.9. There was much less formation of sulphur-sulphide agglomerates in the continuous run, as the coarse fraction (+100 mesh) corresponded to only 0.4% of the total residue weight (compared with 20% in the batch tests), and the fine fraction (minus 200 mesh) contained 98.6% of the solids. There was virtually no pyrrhotite remaining in the fines fractions. The precipitated iron was present primarily as hematite, with some also present as hydronium jarosite.

Liquid-Solid Separation

Continuous thickening of the autoclave discharge slurry was carried out in the 97 L volume plexiglas thickener, equipped with a slow speed rake. At the start of the run, the thickener was filled with synthetic overflow solution. Autoclave discharge slurry was diluted to ~10% solids in the thickener feed tank, flocculated, and fed to the thickener. Flocculant, prepared as a 0.5 g/L stock solution, was further diluted to 0.05 g/L before adding to the slurry, at a level of 100 g/t of solids during period 1, and 43 g/t during period 2. The mud level in the thickener was allowed to rise to a depth of 25 to 30 cm before underflow was pumped continuously at a rate to maintain that level. Underflow solids specific gravity was 3.08. A summary of the thickener operating conditions and performance is given below.

Oxidation Period	1A	1B	2
Thickener solids feed rate, kg/h	-	4.5	5.3
Solution specific gravity	1.06	1.06	1.06
Solids specific gravity	3.08	3.08	3.08
Flocculant addition level, g/t	-	100	43
Temperature, °C	63	63	46
Underflow slurry solids, %	-	41.5	48.5
Settler duty, t/m^2·day	-	0.62	0.53

4.3 RECOVERY OF SULPHUR BY FLOTATION

Batch flotation tests were conducted on samples of thickener underflow from the continuous pressure oxidation run to establish potential recoveries and grades of product attainable, and to define parameters for the continuous flotation testwork.

One series of rougher flotations was conducted in the 2 L cell, to evaluate several frothers. The thickener underflow contained 44% solids, analyzing 40.5% Fe and 20.0% S°. The slurry was diluted to 18.6% solids, and after conditioning for 1 min with 50 to 80 g/t frother, at 43°C, pH 1.9, was floated for 3 min. Further details on the experimental procedures and the results are presented in Table 4.10, and a brief summary is given below.

Frother	Analyses, %				Recovery to Concentrate %		
	Concentrate		Tailings				
	Fe	S°	Fe	S°	Weight	Fe	S°
None	15.3	67.0	51.0	0.1	28.9	10.9	99.6
Aerofroth 73	17.0	64.6	50.7	1.5	30.6	12.9	95.0
Dowfroth 200	18.7	60.3	50.9	<0.1	31.8	14.6	>99.6
Pine Oil	16.8	64.6	50.9	0.6	30.7	12.7	97.9

A high recovery (95% and higher) of elemental sulphur was achieved in all the tests. Using the natural frothing characteristics of the system yielded both the highest grade of product and an exceptionally high recovery. Stable froths, which dissipated after the elemental sulphur and sulphides had been floated, were obtained with Aerofroth 73 and with the natural system. Both Dowfroth 200 and Pine Oil produced froths which were less sticky but which persisted throughout.

Two batch flotation tests, incorporating rougher, scavenging, cleaning, and recleaning operations were conducted on larger slurry samples collected during the intermediate and terminal portions of the continuous oxidation run. Rougher and scavenger flotations were conducted in an 8 L cell, at 16 to 18% solids, 41°C, pH ~2, and with 12 to 24 g/t Aerofroth addition to the rougher stage. Cleaner and recleaner flotations were conducted in a 4 L cell, also at 41°C, pH ~2.3, with the addition of Dowfroth 200 to the recleaner stage. There was no recycling of products in these batch tests. Additional experimental details and results are presented in Tables 4.11 and 4.12.

In the first test, (Table 4.11), on material containing 37.2% Fe and 20.2% S°, the rougher concentrate, assaying 17.4% Fe and 62.4% S°, corresponded to 28.1% by weight of the feed and an elemental sulphur recovery of 87.1%. The cleaning operation improved the grade of the product to 12.7% Fe and 71.0% S°, with little loss of recovery (86.2%). The recleaning operation improved the grade only marginally. The scavenger tailings, assaying 2.21% S°, contained 7.7% of the feed elemental sulphur. The second test (Table 4.12), was on material containing 40.3% Fe and 20.0% S°. The rougher concentrate, corresponding to 26.7% of the flotation feed weight, assayed 69.2% S°, and accounted for 92% of the elemental sulphur. Cleaning and recleaning of the concentrate, improved the product grade to 79.2 and 81.3% S°, respectively, with corresponding elemental sulphur recoveries of 91.7 and 73.7%, respectively. Scavenger flotation of the rougher tailings lowered the elemental sulphur content of the tailings from 2.18 to 1.19% S°, and decreased the loss of the elemental sulphur to the tailings to 4.3%.

Microscopic examination of the recleaned concentrate (8.1% Fe, 88.2% S, 81.3% S°) confirmed pyrite as the major species other than the elemental sulphur in the concentrate. All the pyrite particles were either coated or associated with elemental sulphur, preventing further upgrading of the elemental sulphur by flotation.

Two continuous flotation tests, each of 2 h duration, were conducted in the circuit shown in Figure 3.2. The feed solids for the first test analyzed 39.7% Fe, 20.2% S°, 2.32% S(SO_4) and 27.1% S. The diluted and conditioned feed slurry contained 22.6% solids and was fed to the rougher circuit at 100 mL/min. Rougher concentrate was collected in a surge tank and was diluted with an additional 42 mL/min of water prior to cleaning. Additional operating parameters and the results are given in Tables 4.13 and 4.14. Samples of product streams were collected at 1 and 2 h for analysis. Component distributions were calculated by the "two product" formula given in Table 4.14. Recoveries of elemental sulphur were high, at 96.0 and 95.2% at 1 and 2 h, respectively. The corresponding scavenger tailings analyzed 1.22 and 1.40% S°, respectively. The cleaner concentrates, corresponding to 34 and 31% of the feed solids weight, assayed only 56.7 and 61.5% S°, respectively, because of a high carryover of iron oxides.

The second continuous flotation run was conducted at a lower pulp density, resulting in a higher grade of cleaner concentrate, assaying 73.5% S°, and corresponding to 26% by weight of the feed solids, with a sulphur recovery of 94%. It is expected that with further circuit and parameter modification and with reagent optimization, particularly with respect to improved scavenging operation, a higher grade of product with a recovery of 95% or better can be achieved. A practical upper limit on the concentrate grade is about 76% S°, due to dilution of the elemental sulphur by pyrite. With the above flotation concentrate, the projected recovery of elemental sulphur by melting and hot filtration is about 83 to 86%, yielding a filter cake assaying 30 to 35%. The overall recovery of elemental sulphur from the pyrrhotite concentrate through pressure oxidation, flotation, and hot filtration is therefore projected to be about 59%. The filter cake, containing unreacted pyrite, iron oxides, and elemental sulphur should to be suitable feed for a pyrite roaster.

It is noted that the yield of elemental sulphur in oxidation, the grade of elemental sulphur in a flotation concentrate, and the ultimate recovery by hot filtration are very much dependent on the relative amounts of pyrrhotite and pyrite in the starting concentrate. If the pyrrhotite concentrates are expected to contain appreciable amounts of pyrite, consideration should be given to the direct roasting of the flotation concentrates (in effect, sulphur enriched, gangue depleted pyrite) in the pyrite roasters.

4.4 BATCH LIQUID-SOLID SEPARATION TESTS

Standard bench scale settling tests were conducted on the cleaner concentrate and scavenger tailings slurries generated in the final continuous flotation test.

The solid content of both slurries was about 7%. For comparison, settling tests were also conducted on oxidation thickener underflow slurry, diluted to about 12% solids, i.e., flotation feed slurry. The specific gravities of the cleaner concentrate, scavenger tailings, and flotation feed solids were 2.43, 3.65, and 3.07, respectively. All tests were conducted by a standard procedure, using 1.0 L graduated cylinders. The tests were done at 40°C, by placing the cylinders in a thermostatically controlled water bath. Flocculant additions (Percol 351) were 0, 20, 40, and 60 g/t. Settling rates without flocculant were extremely slow for the scavenger tailings and the flotation feed. Thickener unit area requirements were determined using the Kynch method. The results are provided in Tables 4.16 to 4.18, and are summarized overleaf.

	Percol 351 g/t	Initial % Solids	Final (24 h) % Solids	Initial Settling Rate m/h	Thickener Unit Area $m^2/t \cdot d$
Cleaner Concentrate	0	7.3	34.2	*	~0.13
	21	7.0	35.3	10.2	0.21
	42	6.9	35.1	10.4	0.28
	61	7.2	33.8	13.9	0.25
Scavenger Tailings	20	7.2	46.0	1.0	1.86
	40	7.3	44.9	3.8	0.96
	59	7.6	42.5	4.0	1.03
Oxidation Thickener Underflow (diluted)	20	11.4	44.2	0.9	1.81
	36	12.0	47.1	1.0	1.23
	55	11.9	44.0	2.6	0.39

*Interface difficult to establish during initial stages

4.5 TAILINGS SOLUTION COMPOSITION

During the second continuous flotation run, several samples of solutions were collected for chemical analysis. The compositions of the flotation feed (diluted thickener underflow), cleaner concentrate, and the scavenger tailings solutions are tabulated overleaf.

Species		Flotation Feed	Cleaner Concentrate	Scavenger Tailings
Aluminum	Al	87.2	16.9	49.7
Arsenic	As	1.25	0.38	1.09
Cadmium	Cd	0.51	0.10	0.35
Calcium	Ca	72.1	26.6	55.0
Chromium	Cr	0.65	0.12	0.44
Cobalt	Co	44.8	9.1	31.2
Copper	Cu	25.0	4.7	16.8
Iron, total	Fe	3040	480	2030
Iron, ferrous	Fe^{2+}	1200	270	660
Lead	Pb	0.74	0.28	0.55
Magnesium	Mg	73.8	21.1	54.4
Manganese	Mn	19.5	3.8	13.3
Molydbenum	Mo	0.35	0.07	0.24
Nickel	Ni	28.5	5.9	19.8
Phosphorous	P	<0.06	<0.06	<0.06
Silica	SiO_2	67.4	12.7	45.3
Sulphuric Acid	H_2SO_4	900	-	500
Zinc	Zn	159.8	34.2	112.5
pH		1.2	2.3	1.9

All analyses in mg/L (except pH)

The scavenger tailings solution assays were used in the estimation of limestone and lime requirements for the neutralization circuit. Further definition of effluents and their treatment will be required during subsequent project phases.

Table 4.1 Batch Pressure Oxidation Test 1

Charge : 2.58 L slurry containing 12.7% solids (150 g concentrate/L solution)
Conditions: 150°C, 1.0 h, 820 kPa total pressure, 5 kg/t oxidation additive
: oxidized slurry cooled prior to discharge

Component		L, g	Analysis, g/L						
			Fe	Fe^{2+}	H_2SO_4	S	S°	$S(SO_4)$	$S^=$
Feed	Electrolyte	2.35	5.5	0	21.0	11.6	-	11.6	-
	Concentrate	377.5	52.1	-	-	35.6	<0.1	0.3	35.3
Products	Solution	2.35	11.1	1.7	23.0	16.8	-	16.8	-
	+125 μm solids	97.9	10.6	-	-	84.2	73.9	0.31	10.0
	-125 μm solids	367.0	47.9	-	-	10.1	7.77	2.06	0.27
	Total solids (calc)	464.9	40.0	-	-	25.7	21.7	1.69	2.33

Component		L, g	Weight, g				
			Fe	S	S°	$S(SO_4)$	$S^=$
Feed	Electrolyte	2.35	12.9	27.2	-	27.2	-
	Concentrate	377.5	196.7	134.4	-	1.13	133.27
	Totals in		209.6	161.6	-	28.33	133.27
Products	Solution	2.35	26.0	39.6	-	39.6	-
	+125 μm solids	97.9	10.4	82.4	72.3	0.30	9.82
	-125 μm solids	367.0	175.8	37.1	28.5	7.56	0.99
	Total solids (calc)	464.9	186.2	119.5	100.8	7.86	10.81
	Totals out		212.1	159.1	100.8	47.5	10.81
% Accountability			101.2	98.5			

Deportment of sulphide sulphur

% of sulphur reacted : 91.9%
Oxidation to sulphate sulphur : 14.4% (9.3% to solution, 5.1% to solids)
Conversion to elemental sulphur: 75.6%
Conversion of sulphidic sulphur
of pyrrhotite to elemental : 87.9%

Table 4.2 Batch Pressure Oxidation Test 2

Charge : 2.58 L slurry containing 12.7% solids, 5 kg/t oxidation additive
Conditions: 150°C, 1.0 h, 820 kPa total pressure, 345 kPa oxygen partial pressure
: slurry flash discharged

Component		L, g	Analysis, g/L						
			Fe	Fe^{2+}	H_2SO_4	S	S°	$S(SO_4)$	$S^=$
Feed	Solution	2.37	5.5	0	21.0	11.6	-	11.6	-
	Concentrate	377.5	52.1	-	-	35.6	<0.1	0.3	35.3
Products	Solution	2.35	11.9	2.0	24.2	18.0	-	18.0	-
	+125 µm residue	77.0	12.4	-	-	84.4	76.3	0.35	7.73
	-125 µm residue	333.0	47.6	-	-	11.2	8.34	1.94	0.92
	Autoclave "heel"	30.5	40.2	-	-	23.9	19.2	1.63	3.03
	Autoclave "pancake"	10.9	-	-	-	(100)	(100)	-	-
	Total solids (calc)	451.4	40.0	-	-	26.7	22.9	1.60	2.20

Component		L, g	Weight, g				
			Fe	S	S°	$S(SO_4)$	$S^=$
Feed	Solution	2.35	12.9	27.2	-	27.2	-
	Concentrate	377.5	196.7	134.4	-	1.13	133.27
	Totals in		209.6	161.6	-	28.33	133.27
Products	Solution	2.35	28.0	42.1	-	42.1	-
	+125 µm residue	77.0	9.6	65.0	58.8	0.27	5.95
	-125 µm residue	333.0	158.5	37.3	27.8	6.46	3.06
	Heel	30.5	12.3	7.29	5.9	0.50	0.92
	Pancake	10.9	-	10.9	10.9	-	-
	Total solids (calc)	451.4	180.4	120.5	103.3	7.23	9.93
	Totals out		208.4	162.6	103.3	49.3	9.93
	% Accountability		99.4	100.6			

Deportment of sulphide sulphur

% of sulphur reacted	:	92.5%
Oxidation to sulphate sulphur	:	15.7% (11.1% to solution, 4.6% to solids)
Conversion to elemental sulphur	:	77.5%
Conversion of sulphidic sulphur of pyrrhotite to elemental	:	90.1%

Table 4.3 Batch Pressure Oxidation Test 3

Charge : 2.58 L slurry, containing 12.7% solids 5 kg/t oxidation additive
Conditions: 150°C, 1.0 h, 820 kPa total pressure, 345 kPa oxygen partial pressure autoclave slurry cooled prior to discharge
Comments : oxidation temperature rose to 165°C during initial portion of test

Component		L, g	Analysis, g/L						
			Fe	Fe^{2+}	H_2SO_4	S	S°	$S(SO_4)$	$S^=$
Feed	Solution	2.35	5.5	0	21.0	11.6	-	11.6	-
	Concentrate	377.5	52.1	-	-	35.6	<0.1	0.3	35.3
Products	Solution	2.35	11.1	3.7	15.2	16.1	-	16.1	-
	+149 µm residue	91	11.5	-	-	86.7	73.7	0.06	12.9
	-149 µm residue	367	47.7	-	-	12.2	9.40	1.94	0.86
	Total solids (calc)	458	40.5	-	-	27.0	22.2	1.57	3.26

Component		L, g	Weight, g				
			Fe	S	S°	$S(SO_4)$	$S^=$
Feed	Solution	2.35	12.9	27.2	-	27.2	-
	Concentrate	377.5	196.7	134.4	-	1.13	133.27
	Totals in		209.6	161.6	-	28.33	133.27
Products	Solution	2.35	26.0	37.8	-	37.8	-
	+149 µm residue	91	10.5	78.9	67.1	0.06	11.78
	-149 µm residue	367	175.2	44.8	34.5	7.12	3.16
	Total solids (calc)	458	185.7	123.7	101.6	7.18	14.94
	Totals out		211.7	161.5	101.6	44.97	14.94
% Accountability			101.0	99.9			

Deportment of sulphide sulphur

% of sulphur reacted : 88.8%
Oxidation to sulphate sulphur : 12.5% (8.0% to solution, 4.5% to solids)
Conversion to elemental sulphur : 76.2%
Conversion of sulphidic sulphur of pyrrhotite to elemental : 88.6%

Table 4.4 Batch Pressure Oxidation Test 4

Charge	:	2.58 L slurry, containing 12.7% solids, 5 kg/t oxidation additive
Conditions	:	150°C, 1.0 h, 820 kPa total pressure, 345 kPa oxygen partial pressure
	:	autoclave slurry cooled prior to discharge
Comments	:	oxidation temperature rose to 155°C during initial stages of test

Component		L, g	Analysis, g/L						
			Fe	Fe^{2+}	H_2SO_4	S	S°	$S(SO_4)$	$S^{=}$
Feed	Solution	2.35	5.5	0	21.0	11.6	-	11.6	-
	Concentrate	377.5	52.1	-	-	35.6	<0.1	0.3	35.3
Products	Solution	2.35	11.3	3.1	19.7	17.1	-	17.1	-
	+149 µm solids	96	8.50	-	-	86.3	79.4	0.05	6.85
	-149 µm solids	354	47.6	-	-	11.0	8.15	1.74	1.11
	Total solids (calc)	450	39.2	-	-	27.1	23.3	1.38	2.33

Component		L, g	Weight, g				
			Fe	S	S°	$S(SO_4)$	$S^{=}$
Feed	Solution	2.35	12.9	27.2	-	27.2	-
	Concentrate	377.5	196.7	134.4	-	1.13	133.27
	Totals in		209.6	161.6	-	28.33	133.27
Products	Solution	2.35	26.3	39.7	-	39.7	-
	+149 µm residue	96	8.2	82.9	76.2	0.05	6.58
	-149 µm residue	354	168.3	38.9	28.9	6.16	3.89
	Total solids (calc)	450	176.5	121.79	105.1	6.21	10.47
	Totals out		202.8	161.5	105.1	45.9	10.47
% Accountability			96.7	99.9			

Deportment of sulphide sulphur

% of sulphur reacted	:	92.1
Oxidation of sulphur to sulphide	:	13.2% (9.4% to solution, 3.8% to solids)
Conversion to elemental sulphur	:	78.8%
Conversion of sulphidic sulphur of pyrrhotite to elemental	:	91.7%

Table 4.5 Continuous Pressure Oxidation Operating Log

Period		1A				1B					2		
Run Time,	h	1	2	3	4	5	6	7	8	9	10	11*	12*
Slurry feed rate,	mL/min	72.9	71.6	67.3	71.7	70.9	68.9	67.7	66.6	71.1	39.4	42.4	
Slurry pulp density,	g/L	2233	2233	2140	2140	2220	2220	2220	2220	2220	2220	2220	
Slurry solids,	%	72.0	72.0	69.5	69.5	71.3	71.3	71.3	71.3	71.3	71.3	71.3	
Solids feed rate,	kg/h	7.03	6.63	6.01	6.39	6.77	6.58	6.47	6.37	6.80	3.76	4.07	
Solution feed rate,	mL/min	460	460	460	460	500	500	500	500	500	290	290	
Initial C1 solids,	%	18.0	17.2	15.9	16.6	16.4	16.1	15.9	15.6	16.5	15.9	16.9	
Retention time,	min	56	56	57	56	53	53	53	53	52	91	90	
Temperature, °C	C1	159.4	152.5	152.7	151.7	153.2	151.2	152.0	149.8	150.7	148.4		
	C2	160.1	152.9	153.5	152.3	153.7	151.8	152.5	150.4	151.0	148.7		
	C3	153.9	153.9	152.9	151.9	152.4	152.0	152.1	150.4	151.1	150.0		
	C4	153.1	153.1	151.3	150.4	150.4	150.9	150.8	151.0	150.2	150.0		
	C5	15.08	150.8	150.9	150.0	149.8	150.2	150.1	150.2	150.5	150.5		
	C6	149.8	149.8	150.2	149.0	148.9	149.4	149.3	149.6	150.1	150.0		
Total pressure,	kPa	820	820	820	820	820	820	820	820	820	820		
Oxygen flowrate, L/min	C1	15	15	15	15	15	15	15	15	15	9		
	C2	10	10	10	10	10	10	10	10	10	6		
	C3	8	8	8	8	8	8	8	8	8	5		
	C4	5	5	5	5	5	5	5	5	5	3		
	C5	2	2	2	2	2	2	2	2	2	2		
	C6	1	1	1	1	1	1	1	1	1	1		
	total	41.0	41.0	41.7	40.9	41.5	41.9	41.2	41.9	40.9	26.2		
% of stoichiometric O_2		238	252	283	262	250	260	260	270	245	284		
Vent rate, L/min		14	14	14	14	14	14	13	13	13	7		

*Feed to autoclave off from 10.5 to 12.75 h

Table 4.6 Continuous Pressure Oxidation Operating Log

Period		2											
Run Time,	h	13	14	15	16	17	18	19	20	21	22	23	24
Slurry feed rate,	mL/min	34.3	40.6	45.7	46.7	45.0	42.1	42.2	46.0	39.6	41.4	43.1	43.5
Slurry pulp density,	g/L	2220	2220	2220	2220	2230	2230	2230	2240	2240	2240	2240	2240
Slurry solids,	%	71.3	71.3	71.3	71.3	71.8	71.8	71.8	72.4	72.4	72.4	72.4	72.4
Solids feed rate,	kg/h	3.26	3.86	4.34	4.43	4.32	4.03	4.05	4.48	3.84	4.02	4.18	4.23
Solution feed rate,	mL/min	290	290	290	290	290	290	290	290	290	290	290	290
Initial Cl solids,	%	14.2	16.2	17.7	18.0	17.7	16.7	16.8	18.2	16.2	16.8	17.3	17.4
Retention time,	min	93	91	89	89	90	90	90	89	91	91	90	90
Temperature, °C	C1	143.8	146.2	150.4	147.6	147.2	149.7	148.2	146.7	148.2	148.2	149.3	149.7
	C2	144.3	146.4	150.4	147.9	147.4	149.8	148.3	146.9	148.5	148.4	149.4	149.8
	C3	147.2	146.6	150.4	151.2	150.6	151.1	149.9	149.5	150.3	150.2	150.9	150.6
	C4	147.0	146.6	150.3	153.3	150.3	151.5	150.0	149.7	150.1	150.0	150.5	150.2
	C5	146.6	147.9	150.7	153.3	152.3	151.0	150.6	150.3	150.6	150.2	151.0	150.7
	C6	147.1	148.6	150.6	152.5	150.9	150.1	150.0	150.0	150.2	149.1	150.6	150.0
Total pressure,	kPa	820	820	850	820	820	820	820	820	820	820	820	820
Oxygen flowrate, L/min	C1	9	9	9	9	9	9	9	9	9	9	9	9
	C2	6	6	6	6	6	6	6	6	6	6	6	6
	C3	5	5	5	5	5	5	5	5	5	5	5	5
	C4	3	3	3	3	3	3	3	3	3	3	3	3
	C5	2	2	2	2	2	2	2	2	2	2	2	2
	C6	1	1	1	1	1	1	1	1	1	1	1	1
	total	26.1	26.5	27.2	26.7	24.9	25.8	26.6	26.9	25.2	25.9	26.4	26.7
% of stoichiometric O_2		327	280	256	246	235	260	268	245	268	263	258	258
Vent rate, L/min		12	10	10	10	9	9	9	9	8	8	n.d.	n.d.

Table 4.7 Continuous Pressure Oxidation, Solution Analysis

Period	Run time, h	Sample	Analysis, g/L Fe	Fe^{2+}	H_2SO_4	$S(SO_4)$*
1A	1	C1	21.9	13.2	11.6	18.8
		C6	20.6	3.9	21.5	23.5
	2	C1	22.9	13.5	13.2	20.1
		C6	21.2	4.1	19.0	23.2
	3	Thick O/F	14.6	3.0	26.0	20.2
		Feed	12.7	2.8	25.4	18.4
		C1	21.4	11.3	10.0	18.4
		C6	19.4	3.8	14.2	20.2
18	5	Thick O/F	16.0	3.0	16.7	18.4
		C1	21.6	12.0	10.5	18.6
		C6	19.6	3.4	17.0	21.5
	6	Feed	18.8	3.1	16.2	20.5
		C1	21.5	11.9	9.5	18.1
		C2	24.1	12.5	9.0	20.0
		C3	22.3	8.1	11.0	20.4
		C4	22.2	6.8	12.6	21.2
		C5	22.5	5.5	13.8	22.2
		C6	21.5	3.7	15.3	22.4
		Disch.	24.8	4.2	13.5	24.5
	9	C1	19.5	10.6	10.0	16.9
		C3	19.4	8.0	11.4	18.0
		C6	20.9	5.4	14.7	21.2
2	17	Feed	15.0	2.9	12.4	16.1
		C1	18.0	7.1	10.5	16.8
		C2	17.7	7.5	11.1	16.6
		C3	17.6	5.6	13.2	17.8
		C4	17.4	4.2	16.2	19.0
		C5	16.6	2.3	18.9	19.7
		C6	16.4	1.7	20.7	20.3
		Discn.	21.3	2.0	19.1	23.9
	21	Feed	17.4	2.7	14.4	18.8
		C1	21.2	10.2	7.9	17.8
		C3	21.3	6.2	9.0	19.4
		C6	21.7	2.8	15.4	22.8
	24	C1	21.2	10.3	9.8	18.4
		C2	22.0	10.3	9.6	19.0
		C3	20.4	6.3	11.2	19.3
		C4	22.4	5.4	12.8	21.8
		C5	21.7	4.5	15.6	22.4
		C6	21.2	2.7	17.7	23.2
		Disch.	28.2	3.3	15.2	28.2

*Calculated

Table 4.8 Continuous Pressure Oxidation, Residue Analysis

Period	Run time, h	Sample	Composition, %					% S oxidation	
			Fe	S	S°	$S(SO_4)$	$S^=$	Total	to S°
		Feed	50.3	34.6					
1A	1	C6	41.4	25.4	19.8	2.04	3.56	87.5	69.5
	2	C6	40.3	26.1	19.7	2.19	4.21	84.8	71.1
	3	C1	40.6	28.5	19.5	2.07	6.93	75.2	69.8
		C6	41.2	24.6	18.7	2.26	3.64	87.2	66.0
1B	5	C1	40.4	27.6	18.8	2.09	6.71	75.9	67.7
		C6	39.7	25.5	18.9	2.23	4.37	84.0	69.2
	6	C1	40.7	27.4	18.5	2.24	6.66	76.2	66.1
		C2	40.9	26.7	18.2	n.a.	~6.2	~78	64.7
		C3	40.9	25.6	18.9	n.a.	~4.4	~84	67.1
		C4	40.1	26.6	20.2	n.a.	~4.1	~85	73.2
		C5	38.3	31.7	23.5	n.a.	-	-	-
		C6	42.4	21.2	16.4	2.43	2.37	~92	56.1
		Disch.	39.4	29.7	22.4	2.23	5.07	-	-
	9	C1	40.9	27.4	18.6	2.04	6.76	76.0	66.1
		C3	40.7	26.3	19.6	2.16	4.54	83.8	70.0
		C6	40.6	25.5	20.0	2.24	3.26	88.3	71.6
2	17	C1	41.0	25.7	18.6	2.36	4.74	83.2	65.9
		C2	42.3	23.2	17.2	2.44	3.56	87.8	59.0
		C3	42.0	23.0	18.0	2.36	2.64	91.9	62.3
		C4	42.1	21.8	18.5	2.42	0.88	97.0	63.9
		C5	41.8	24.4	19.1	2.46	2.84	90.1	66.4
		C6	43.2	21.1	16.6	2.40	2.10	92.9	55.9
		Disch.	42.2	23.6	18.6	2.40	2.60	91.0	64.2
	21	C1	41.0	26.6	15.6	2.28	8.12	71.2	55.3
		C3	41.1	24.7	16.8	2.80	5.10	82.0	59.4
		C6	41.9	23.7	16.6	2.71	4.39	84.8	57.6
	24	C1	40.7	26.3	17.4	2.47	6.43	77.0	62.1
		C2	41.3	25.7	15.9	~2.6	~7.2	~75	56.1
		C3	41.4	24.0	17.3	~2.6	~4.1	~86	60.7
		C4	41.3	24.3	18.3	~2.6	~3.1	~88	64.4
		C5	41.0	24.1	18.7	~2.6	~2.8	~90	66.3
		C6	42.1	21.6	16.5	2.93	2.17	92.5	57.0
		Disch.	40.1	25.4	20.9	2.81	1.69	93.9	75.7

Table 4.9 Chemical, Mineralogical and Screen Analysis of Continuous Oxidation Residue

a. Chemical and Screen Analysis

Fraction	Weight	Analysis, %				Distribution, %				
Mesh	g	Fe	S°	$S(SO_4)$	S	Wt	Fe	S°	$S(SO_4)$	S
+100	3.2	6.86	83.5	0.19	88.1	0.4	0.07	1.70	0.02	1.40
-100 +200	7.9	13.3	68.4	0.11	75.7	1.0	0.36	3.43	0.04	2.97
-200	763	39.0	19.6	3.11	25.2	98.6	99.57	94.87	99.94	95.63
Head (calc)	774.1	38.6	20.4	3.07	26.0	100	100	100	100	100

b. Mineralogical Examination of Screen Fractions*

Fraction Mesh	Abundance	Species
+100	major minor, trace	pyrite, pyrrhotite hematite, chalcopyrite, sphalerite
-100 +200	major minor, trace	pyrite silica, chalcopyrite, sphalerite
-200	major minor, trace	hematite, hydronium jarosite silica, pyrite

*X-Ray diffraction, electron microprobe, microscope examination of solids after prior removal of elemental sulphur

Table 4.10 Batch Flotation; Frother Evaluation

Feed : Thickener underflow slurry from continuous oxidation, pulp density 1460 g/L, ~42% solids
: diluted to 20% solids

Flotation : 2 L cell, pH 1.9, 43°C
: 1 min conditioning, 3 min flotation
Test 1 : No frother
2 : Aerofroth 73 : 50 g/t
3 : Dowfroth 200 : 84 g/t
4 : Pine Oil : 63 g/t

Test	Fraction	Weight g	Assays, %		Units, g		Distribution, %		
			Fe	S°	Fe	S°	Wt.	Fe	S°
1	Concentrate	123.2	15.3	67.0	18.9	82.5	28.9	10.9	99.6
	Tailings	303.3	51.0	0.1	154.6	0.3	71.1	89.1	0.4
	Head (calc)	426.5	40.7	19.4	173.5	82.8			
2	Concentrate	131.6	17.0	64.6	22.3	85.0	30.6	12.9	95.0
	Tailings	298.2	50.7	1.5	151.3	4.5	69.4	87.1	5.0
	Head (calc)	429.8	40.4	20.8	173.6	89.5			
3	Concentrate	135.6	18.7	60.3	25.4	81.8	31.8	14.6	>99.6
	Tailings	291.3	50.9	<0.1	148.2	<0.3	68.2	85.4	<0.4
	Head (calc)	426.9	40.7	19.2	173.6	~82			
4	Concentrate	131.8	16.8	64.6	22.1	85.1	30.7	12.7	97.9
	Tailings	297.5	50.9	0.6	151.5	1.8	69.3	87.3	2.1
	Head (calc)	429.3	40.5	20.3	173.6	86.9			

Table 4.11 Batch Flotation of Continuous Oxidation Thickener Underflow, Test 1

Feed : 2.38 L thickener underflow from continuous oxidation (Bucket No. 4), pulp density 1500 g/L, ~45% solids, diluted to 8 L with water, ~18% solids
Rougher : 8 L cell, 1500 rev/min, 40°C, pH 2.2
1 min conditioning with 12 g/t Aerofroth 73, 2 min flotation
Scavenger : 8 L cell, flotation of rougher tailings, no additional frother, 2 min float time
Cleaner : 4 L cell, 1000 rev/min, 41°C, pH 2.5, flotation of rougher concentrate, no additional frother, 3 min float time
Recleaner : 4 L cell, 41°C, pH 2.5, 170 g/t Dowfroth 200 (staged), 3 min float time

Flotation	Fraction	Weight	Analysis, %				Distribution, %				
		g	Fe	S°	$S(SO_4)$	S	Wt.	Fe	S°	$S(SO_4)$	S
Rougher	Feed	1697.3	37.2	20.2	2.80	27.6	100.0	100.0	100.0	100.0	100.0
	Rougher Con	477.7	17.4	62.4	0.53	75.4	28.1	13.1	87.1	5.3	77.0
	Rougher Tails	1219.6	45.0	3.62	3.69	8.81	71.9	86.9	12.9	94.7	23.0
Cleaner	Rougher Con	477.7	17.4	62.4	0.53	75.4	28.1	13.1	87.1	5.3	77.0
	Cleaner Con	415.7	12.7	71.0	0.18	85.2	24.5	8.3	86.2	1.5	75.7
	Cleaner Tails	62.0	48.7	4.89	2.89	9.62	3.6	4.8	0.9	3.8	1.3
Recleaner	Cleaner Con	415.7	12.7	71.0	0.18	85.2	24.5	8.3	86.2	1.5	75.7
	Recleaner Con	331.2	11.8	73.0	0.18	86.4	19.5	6.2	70.6	1.2	61.2
	Recleaner Tails	84.5	16.3	63.2	0.16	80.5	5.0	2.1	15.6	0.3	14.5
Scavenger	Rougher Tails	1219.6	45.0	3.62	3.69	8.81	71.9	86.9	12.9	94.7	23.0
	Scavenger Con	29.8	18.1	60.1	0.64	72.2	1.8	0.9	5.2	0.4	4.6
	Scavenger Tails	1189.8	45.7	2.21	3.77	7.22	70.1	86.0	7.7	94.3	18.4

Table 4.12 Batch Flotation of Continuous Oxidation Thickener Underflow, Test 2

Feed	:	2.23 L thickener underflow from continuous oxidation (Bucket No. 10), pulp density 1530 g/L, ~47% solids, diluted to 8 L with water, ~16% solids
Rougher	:	8 L cell, 1500 rev/min, 41°C, pH 1.8 1 min conditioning with 12 g/t Aerofroth 73, 2 min flotation
Scavenger	:	8 L cell, flotation of rougher tailings, no additional frother, 2 min float time
Cleaner	:	4 L cell, 1000 rev/min, 41°C, pH 2.2, flotation of rougher concentrate, no additional frother, 3 min float time
Recleaner	:	4 L cell, 41°C, pH 2.2, 210 g/t Dowfroth 200 (staged), 3 min float time

		Weight	Analysis, %				Distribution, %				
Flotation	Fraction	g	Fe	S°	$S(SO_4)$	S	Wt.	Fe	S°	$S(SO_4)$	S
Rougher	Feed	1478.8	40.3	20.0	2.41	25.6	100.0	100.0	100.0	100.0	100.0
	Rougher Con	394.3	14.3	69.2	0.55	77.0	26.7	9.5	92.0	6.0	80.2
	Rougher Tails	1084.5	49.7	2.18	3.08	6.92	73.3	90.5	8.0	94.0	19.8
Cleaner	Rougher Con	394.3	14.3	69.2	0.55	77.0	26.7	9.5	92.0	6.0	80.2
	Cleaner Con	343.2	8.9	79.2	0.20	87.6	23.2	5.1	91.7	1.9	79.4
	Cleaner Tails	51.1	50.5	1.67	2.89	6.11	3.5	4.4	0.3	4.1	0.8
Recleaner	Cleaner Con	343.2	8.9	79.2	0.20	87.6	23.2	5.1	91.7	1.9	79.4
	Recleaner Con	268.7	8.1	81.3	0.21	88.2	18.2	3.7	73.7	1.6	62.6
	Recleaner Tails	74.5	11.9	71.8	0.15	85.3	5.0	1.5	18.0	0.3	16.8
Scavenger	Rougher Tails	1084.5	49.7	2.18	3.08	6.92	73.3	90.5	8.0	94.0	19.8
	Scavenger Con	18.0	18.4	60.7	0.82	69.9	1.2	0.6	3.7	0.4	3.3
	Scavenger Tails	1066.5	50.3	1.19	3.12	5.86	72.1	89.9	4.3	93.5	16.5

Table 4.13 Continuous Flotation of Oxidation Thickener Underflow

Feed Slurry : Thickener underflow slurry diluted with water
Reagent : Aerofroth 73, 70 g/t, added to feed slurry, with no subsequent reagent addition
Temperature : 43°C (feed slurry)

	Run 1		Run 2		
	1 h	2 h	1 h	1.5 h	2 h
Feed Slurry					
Feed slurry rate, mL/min	100	100	100	100	100
Feed slurry P.D., g/L	1180	1180	1088	1088	1088
Feed slurry solids, %	22.6	22.6	14.6	11.7	11.7
Solids feed rate, g/min	26.6	26.6	15.8	12.7	12.7
Rougher Scavenger Tailings					
Slurry flowrate, mL/min	130	136	115	96	142
Slurry pulp density, g/L	1094	1051			
Slurry solids, %	12.3	12.8	7.1	7.4	7.4
Solids rate*, g/min	17.5	18.3	10.9	8.7	9.4
Rougher Concentrate					
Slurry flowrate, mL/min				40	
Slurry pulp density, g/L					
Slurry solids, %				9.3	
Solids rate*, g/min				3.9	
Water to Cleaner Circuit, mL/min	42	42	42	42	42
Cleaner Concentrate					
Slurry flowrate, mL/min					
Slurry pulp density, g/L					
Slurry solids, %					
Solids rate*, g/min	9.1	8.3	4.9	4.0	3.3

*Calculated from component distributions (Tables 4.14 and 4.15)

Table 4.14 Continuous Flotation Run 1

Product Analyses

Time h	Sample	Composition, %					
		Fe	SiO_2	S	S°	$S(SO_4)$	$S^=$
	Feed	39.7	2.88	27.1	20.2	2.32	4.58
1	Cleaner Concentrate	20.2	0.98	68.7	56.7	0.68	11.32
	Scavenger Tailings	50.4	3.55	4.82	1.22	3.10	0.50
2	Cleaner Concentrate	17.9	1.52	69.9	61.5	0.73	7.67
	Scavenger Tailings	50.5	3.91	4.86	1.40	3.12	0.34

Product Distributions

Time h	Product	Distribution, %					
		Fe	SiO_2	S	S°	$S(SO_4)$	$S^=$
1	Cleaner Concentrate	18.0	8.9	88.4	96.0	9.4	33.2
	Scavenger Tailings	82.0	91.1	11.6	4.0	90.6	6.8
2	Cleaner Concentrate	14.9	22.7	88.2	95.2	10.5	96.9
	Scavenger Tailings	85.1	77.3	11.8	4.8	89.5	3.1

Component distributions calculated by the formula

$$R = \frac{100\,c(f-t)}{f(c-t)}$$

where f, c and t are the assays in the feed, concentrate and tailings, respectively, and R is recovery to the concentrate

Product weight splits were back calculated from the component splits

Table 4.15 Continuous Flotation Run 2

Product Analyses

Time h	Sample	Composition, %					
		Fe	SiO_2	S	S°	$S(SO_4)$	$S^=$
	Feed	40.0	3.06	26.0	20.2	2.34	3.46
1	Cleaner Concentrate	18.5	1.06	68.8	61.0	0.74	7.06
	Scavenger Tailings	50.2	3.89	4.91	1.72	3.10	0.09
1.5	Rougher Concentrate	23.5	1.21	60.2	52.1	0.97	7.13
	Cleaner Concentrate	17.7	1.18	71.6	62.3	0.59	8.71
	Scavenger Tailings	50.4	4.13	4.79	1.16	3.04	0.59
2	Cleaner Concentrate	13.4	0.69	79.5	73.5	0.51	5.49
	Scavenger Tailings	50.2	3.95	4.71	1.64	3.03	0.04

Product Distributions

Time h	Product	Distribution, %					
		Fe	SiO_2	S	S°	$S(SO_4)$	$S^=$
1	Cleaner Concentrate	14.9	10.2	87.3	94.1	10.2	98.7
	Scavenger Tailings	85.1	89.8	12.7	5.9	89.8	1.3
1.5	Cleaner Concentrate	14.1	14.0	87.4	96.0	7.2	89.0
	Scavenger Tailings	85.9	86.0	12.6	4.0	92.8	11.0
2	Cleaner Concentrate	9.3	6.2	87.0	94.0	6.0	99.6
	Scavenger Tailings	90.7	93.8	13.0	6.0	94.0	0.4

Table 4.16 Settling Tests on Flotation Cleaner Concentrate

Test No.	1	2	3	4
Slurry volume, mL	970	970	990	990
Slurry weight, g	1021.5	1021	1037	1039
Slurry P.D., g/L	1053	1052	1047	1049
Solids weight, g	74.3	71.3	71.8	74.3
Initial solids, %	7.27	6.98	6.92	7.15
Solution S.G.	1.009	1.009	1.009	1.009
Solids S.G.	2.43	2.43	2.43	2.43
Percol 351, g/t	0	21	42	60.5
Time, min	Interface level, mL			
0	970	970	990	990
0.5		700	680	650
1.0		400	360	360
1.25	242		325	
1.5	240	300	300	300
2	240	270	270	265
2.5	230	245	250	250
3	220	227	235	235
3.5		217	223	228
4		207	219	220
4.5		200	210	
5	220	196	208	212
5.5		193		
6	225	190	201	210
7	225	188	198	
8		186	195	205
9	230			
10		180	190	200
11	230			
12.5		178	189	
13	230			
14				
15	210	177	187	197
20		175	181	195
30	198	170	180	190
45	197	166	178	
60	196	165	171	187
120	190	161	169	180
24 h	172	160	161	175
Final (24 h) solids, %	34.2	35.3	35.1	33.8
Initial settling rate, m/h	*	10.2	10.4	13.9
Thickener unit area $m^2/t\cdot d$	~0.13	0.21	0.28	0.25

*Initial interface difficult to define

Table 4.17 Settling Tests on Flotation Scavenger Tailings

Test No.	5	6	7
Slurry volume, mL	990	985	945
Slurry weight, g	1046.4	1040	1009
Slurry P.D., g/L	1057	1056	1068
Solids weight, g	75.5	75.5	76.9
Initial solids, %	7.21	7.26	7.62
Solution S.G.	1.003	1.003	1.003
Solids S.G.	3.65	3.65	3.65
Percol 351, g/t	19.9	40	58.5
Time, min	**Interface level, mL**		
0	990	985	945
1			730
1.5		660	630
2		580	565
2.5		550	520
3		520	485
3.5		480	450
4		455	420
4.5		425	395
5		410	375
5.5		390	360
6		375	345
6.5		360	
7		348	320
7.5		337	
8		327	300
9		310	282
10		296	270
11		285	258
12	547	273	249
13	530	264	240
1	510		232
15	495	250	227
17.5	460		213
20	429	225	202
25	378	208	188
30	344	191	178
35	319		170
45	290		160
50	280	167	
60	260	158	150
90	221		
120			145
24 h	109	113	125
Final (24 h) solids, %	46.0	44.9	42.5
Initial settling rate, m/h	0.97	3.76	4.0
Thickener unit area $m^2/t\cdot d$	1.86	0.96	1.03

Table 4.18 Settling Tests on Diluted Thickener Underflow

Test No.	8	9	10
Slurry volume, mL	970	950	940
Slurry weight, g	1161	1162	1140
Slurry P.D., g/L	1197	1223	1213
Solids weight, g	131.8	140.2	136
Initial solids, %	11.4	12.0	11.9
Solution S.G.	1.092	1.092	1.092
Solids S.G.	3.07	3.07	3.07
Percol 351, g/t	19	35.7	55
Time, min	**Interface level, mL**		
0	970	950	940
1			780
1.5			720
2		740	680
2.5		720	630
3		700	580
3.5		680	540
4		655	505
4.5		630	485
5	780	610	465
5.5	750	588	448
6	730	570	438
6.5		550	428
7	700	535	405
7.5	680		395
8	670	508	383
8.5	660	495	
9	650	485	368
10		465	355
11	620	450	343
12.5	606	428	330
15	568	400	313
17.5	540	376	300
20	515	360	290
22.5	494	343	
25	470	330	277
30	435	310	265
35	410	292	258
40	390	283	
45	374	273	247
60	340	253	231
120	275	216	213
24 h	195	190	203
Final (24 h) solids, %	44.2	47.1	44.0
Initial settling rate, m/h	0.87	1[illegible]	2.63
Thickener unit area m^2/t·d	1.81	1[illegible]	0.39

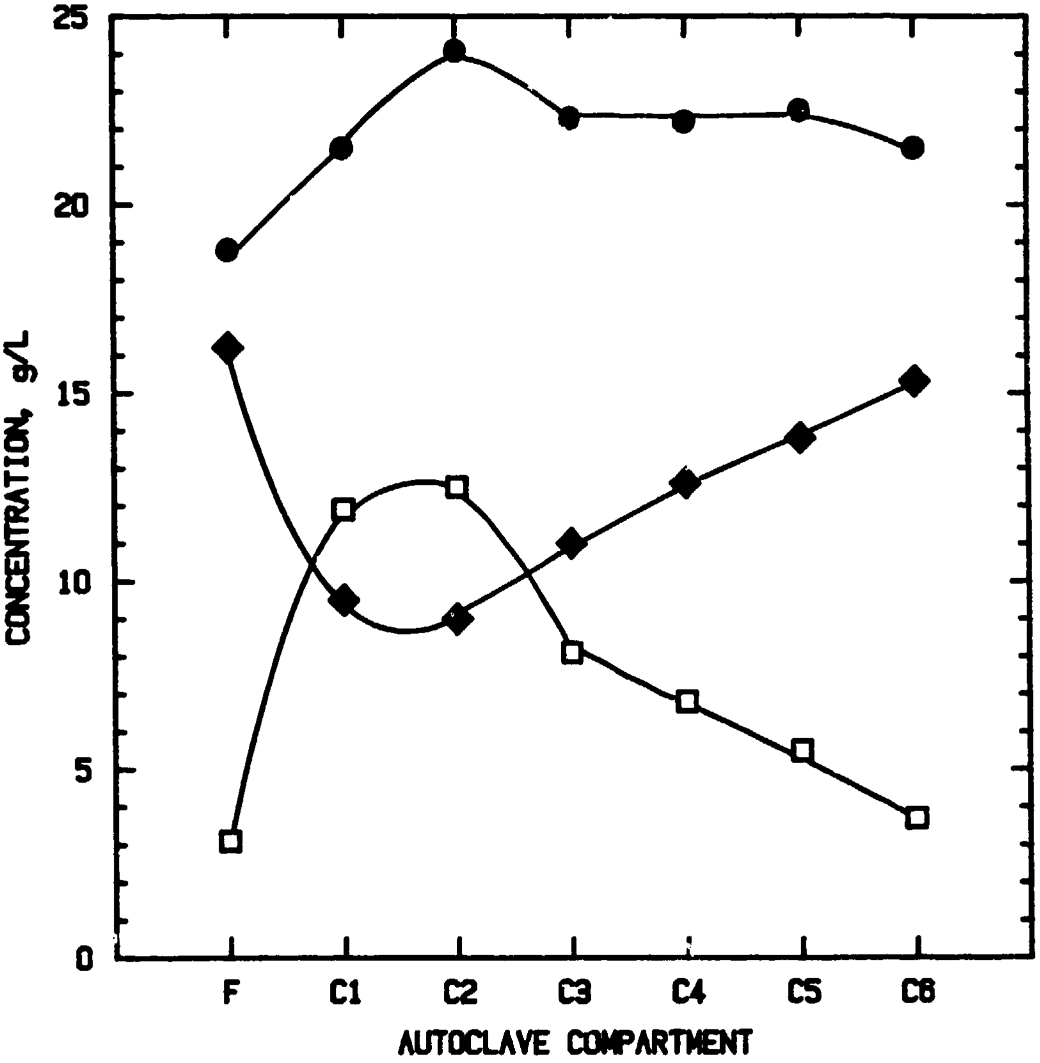


FIGURE 4.1 AUTOCLAVE SOLUTION PROFILES, 6 HOURS

● IRON
□ FERROUS IRON
◆ SULPHURIC ACID

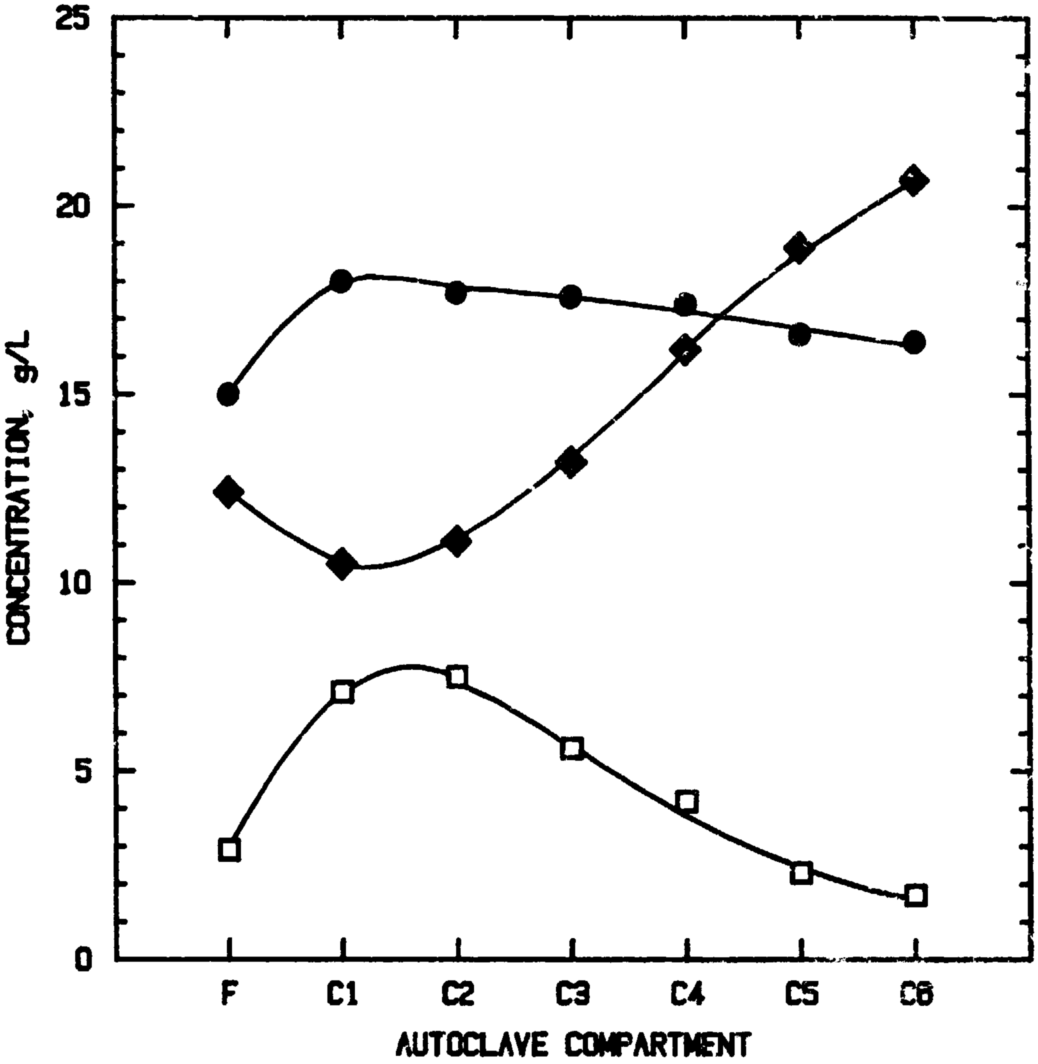


FIGURE 4.2 AUTOCLAVE SOLUTION PROFILES, 17 HOURS

● IRON
□ FERROUS IRON
◆ SULPHURIC ACID

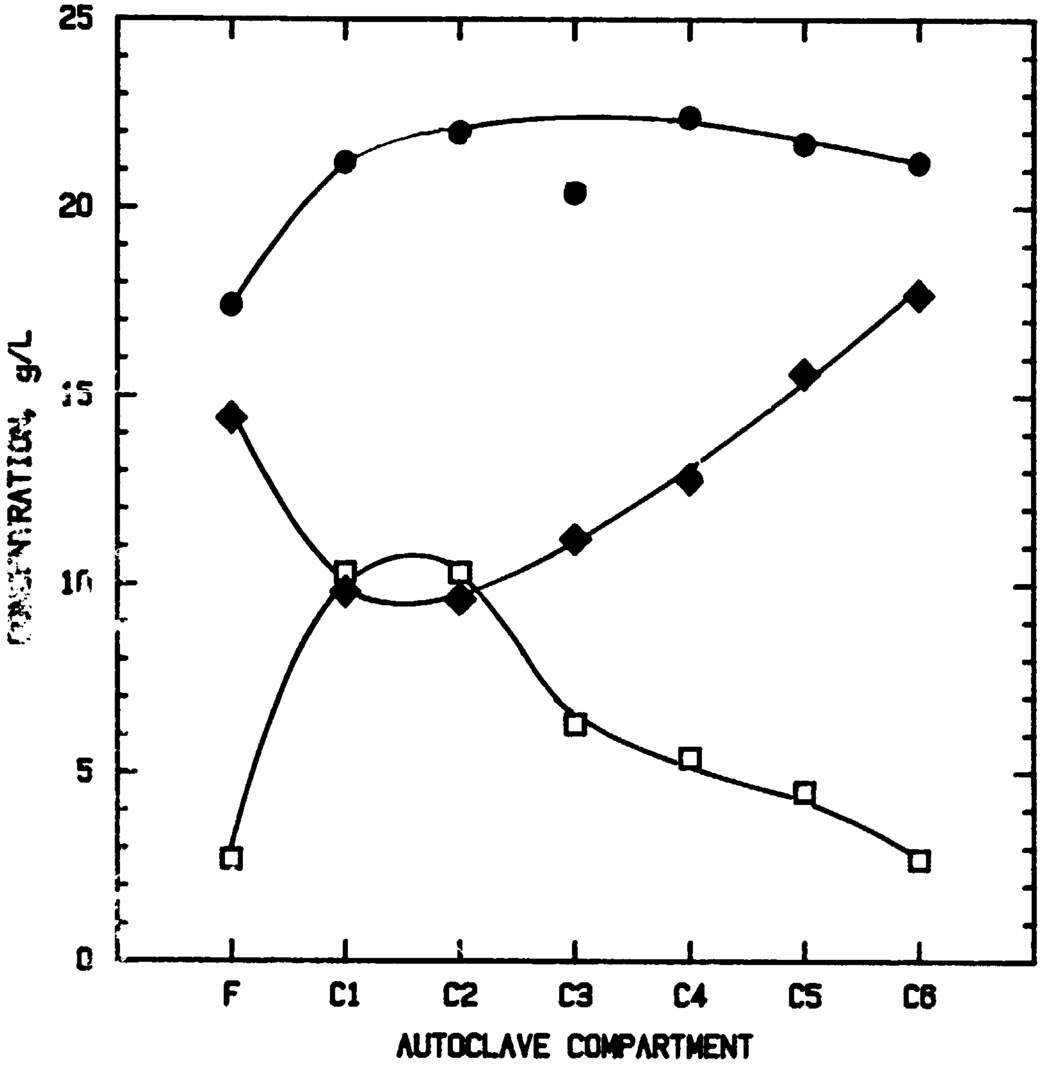


FIGURE 4.3 AUTOCLAVE SOLUTION PROFILES, 24 HOURS

● IRON
□ FERROUS IRON
◆ SULPHURIC ACID

5.0 ENGINEERING STUDY

Based on the results of the laboratory studies and information supplied by the client, preliminary capital and operating cost estimates were prepared. The capital cost of a pressure oxidation and flotation circuit capable of treating 400 t/d of Saladipura pyrrhotite concentrate to produce 135 t/d of flotation concentrate containing 102 t/d of elemental sulphur and 16 t/d of sulphide sulphur (118 t/d total sulphur) is estimated to be $21.4 million (U.S.). The estimated annual operating costs for an Indian location are $3.25 million (U.S.) . The flotation concentrate could be further upgraded via hot filtration to produce 87 t/d of elemental sulphur for a total capital cost of $22.9 million (U.S.). The total annual operating costs would increase to $3.5 million (U.S.)

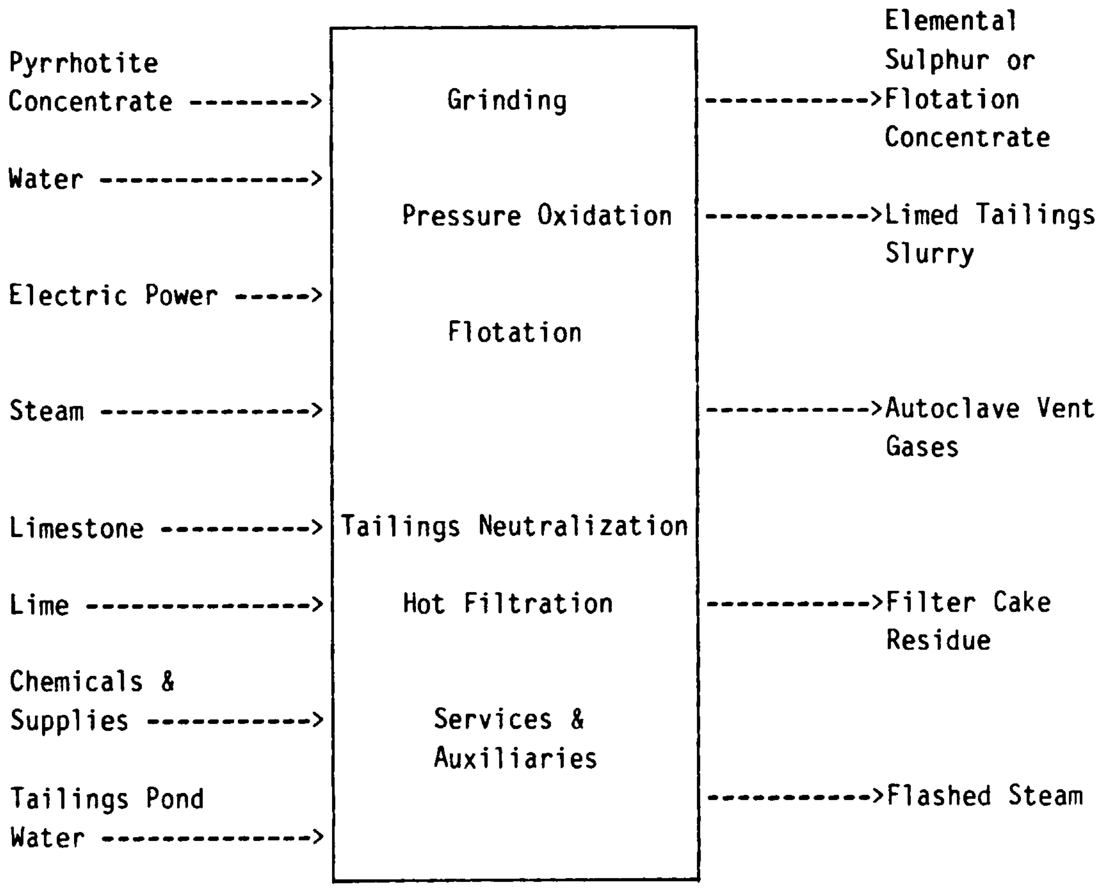
5.1 PROJECT DESCRIPTION

PPCL are considering mining and milling Saladipura pyrrhotite to produce a pyrrhotite concentrate. The concentrate could be further treated to upgrade the sulphur content or recover elemental sulphur by employing Sherritt's aqueous oxidation process. Sherritt's process would convert the pyrrhotite to elemental sulphur and iron oxide.

The pyrrhotite concentrate was assumed to have a composition of 52% iron and 36% sulphur. The pressure oxidation circuit would treat 400 t/d of pyrrhotite concentrate to produce 135 t/d of flotation concentrate containing 102 t/d of elemental sulphur. The flotation concentrate could be upgraded via hot filtration to produce 87 t/d of clean elemental sulphur.

The battery limits of the pressure oxidation, flotation and sulphur recovery circuit and the streams entering and leaving the battery limits, are described following.

Figure 5.1 Saladipura Pyrrhotite Project Battery Limits



The diagram above includes a hot filtration sulphur recovery section to produce an elemental sulphur product along with a filter cake residue. If the hot filtration circuit is not included a flotation concentrate would be produced in place of the elemental sulphur and filter cake residue.

5.2 PROCESS DESCRIPTION

Drawing 09901-68-04-001 depicts the process flowsheet for the PPCL pressure oxidation and flotation circuit.

Pyrrhotite concentrate is delivered from the mill and reclaimed onto a conveyor from the concentrate storage area. The concentrate is metered into the ball mill with a weigh belt. The ball mill discharge is pumped to the hydrocyclones in which the oversize is removed and recycled to the ball mill. The overflow ground concentrate product is sent to the ground concentrate storage tank.

The ground concentrate slurry is pumped from the storage tank to the metering tank with centrifugal pumps, and then into the two pressure oxidation autoclaves with air displacement pumps. Cooled solution is recycled from the autoclave discharge thickener to the autoclaves in order to maintain the autoclave heat balance. The oxidation is achieved using pure oxygen and is carried out in two 3.8 m diameter by 15.3 m long lead and brick lined autoclaves. The vessels are divided into four agitated compartments. Slurry from the autoclave is discharged into agitated flash tanks.

Steam produced in the flash tanks is vented, and the flashed slurry is drained into a seal tank. Slurry from the seal tank is mixed with flotation thickener overflow and is pumped to the autoclave discharge thickener for dewatering. The underflow slurry is sent to the flotation circuit and the overflow solution is recycled to the autoclaves after being cooled in a cooling tower.

Underflow from the autoclave discharge thickener reports to the flotation conditioning tank where the slurry is adjusted to the required pulp density with recycle solutions and where the flotation reagents are added. The slurry is then pumped from the conditioning tank to a constant head tank which feeds the flotation circuit. The flotation circuit consists of rougher, scavenger and cleaner cells. Conditioned slurry is fed to the rougher cells; the resulting concentrate proceeds to the cleaner cells, while the rougher tailings proceed to the scavenger cells. Scavenger concentrate is recycled to the flotation conditioning tank and the scavenger (final) tailings are pumped to the tailings thickener. The cleaner concentrate is pumped to the concentrate thickener while the cleaner tailings are recycled to the conditioning tank. The concentrate thickener underflow is dewatered and washed on a vacuum belt filter and the filtrate is recycled to the flotation conditioning tank. The product filter cake, containing 76% elemental sulphur, can be upgraded to a pure elemental sulphur product by hot filtration. A hot filtration circuit is described later.

Flotation tailings are dewatered in the tailings thickener. Part of the overflow is sent to the flotation conditioning tank, the remainder is recycled to the autoclave discharge thickener. The tailings thickener underflow is partially neutralized with limestone slurry in a series of three agitated tanks. The neutralization is completed with slaked lime in another two agitated tanks in series. The neutralized slurry overflows to the tailings pump tank and is then pumped to the tailings pond. Decant solution from the tailings pond is available for use within the process. The complete analysis of all effluents from the process plant will have to be addressed, when the feed concentrate and neutralization agents are finally defined, in order to meet the local Pollution Control Board regulations.

If required, the flotation concentrate can be upgraded to an elemental sulphur product via hot filtration. The concentrate is first melted with a molten sulphur recycle stream in the sulphur melting cyclone which discharges into a storage pit. Molten sulphur from this pit is circulated through the melting cyclone to supply the heat required to melt the sulphur contained in the incoming concentrate filter cake. The dirty, molten sulphur is filtered through a pressure leaf filter with the clean sulphur filtrate reporting to the clean sulphur pit. The pressure leaf filter cake is discharged periodically and is broken up in a pug mill. This material contains unreacted pyrite and entrained sulphur which would be discharged from the circuit. The molten sulphur product stored in the clean sulphur pit would be delivered to the battery limits of the plant in a pipeline.

5.3 PROCESS FLOWSHEET

The process flowsheet, drawing number 09901-68-04-001 is shown at the end of this section. The equipment for the pressure oxidation and flotation circuit is listed on the flowsheet and a separate list has been sent to PPCL. The flowsheet produces a concentrate for sale; the production of an elemental sulphur product would require the addition to the flowsheet of a molten sulphur pit and molten sulphur filter.

5.4 CAPITAL COST ESTIMATES

5.4.1 Results

The following table contains the summary of the capital cost estimates of the PPCL pressure oxidation and flotation circuit. The capital costs are based on a circuit to treat 400 t/d of concentrate which would produce 135 t/d of flotation concentrate containing 102 t/d of elemental sulphur (118 t/d total sulphur).

		U.S. Dollars
Installed Equipment		6 990 000
Utilities and Auxiliaries:		
Oxygen Plant	- 5 100 000	
Others	- 567 000	5 667 000
Indirects, Offsites		1 712 000
Engineering and Licensing Fee		3 386 000
Contingency		3 592 000
Total Capital		21 347 000
SAY		21 400 000

If a hot filtration sulphur recovery circuit is included, the capital cost would increase to $22.9 million. The hot filtration circuit would produce 87 t/d of pure elemental sulphur.

5.4.2 Basis

The capital cost estimate is based on information received during the Phase I work.

Due to the preliminary nature of the engineering and laboratory work carried out, this capital cost estimate is an order of magnitude estimate with an accuracy of not better than ±30%.

Preliminary material and energy balances were used to estimate the equipment sizes for the complete plant.

The costs tabulated are based on third quarter of 1983 for an Indian location and do not include escalation. Costs for the Indian location were based on information supplied by PPCL.

The exchange rate was Rs. 10 equals $1.00 (U.S.).

All costs were estimated using a modular estimating technique. The installed equipment costs include equipment installation which covers equipment erection, concrete pads, access steel, insulation and paint. The cost estimate was based on equipment complexity, materials and size. Piping cost estimates were based on material cost and the complexity of the piping around a specific piece of equipment. Instrumentation costs were based on an instrument loop count and the complexity of the instrumentation needed for the class of process equipment. Electrical motor hookup cost estimates were based on the motor count with an allowance for control wiring and power supply within the motor control centres.

The oxygen plant would produce 150 t/d of 98% pure oxygen. Oxygen plant costs were based on a quotation for a turnkey plant supplied by PPCL for the Indian location.

Plant utilities include power distribution, pipe racks and pipe on rack. Utilities were estimated using a combination of modular estimating and factoring.

Offsites include the process effluent collection system.

Indirects include allowances for freight, insurance, duty, and sales tax as specified by PPCL. Project insurance was estimated as a percentage of the total project cost. Engineering cost was estimated at 10% of all non turnkey direct cost items. For all turnkey items 5% engineering was allowed. Project contingency was estimated at 25% of total project cost excluding engineering and licensing fee.

5.4.3 Limitations

The following items are not included in the capital cost estimate:

preproduction costs and working capital;

interest during construction;

escalation;

land acquisition costs;

storm water collection system and treatment;

construction camp costs;
tailings pond;
delivery of utilities to the battery limits;
project indirects other than labour related;
contractor's fee;
development of the process area (2000 m^2);
power distribution to the motor control centre;
mobiles;and
buildings.

Allowance for the distribution of the utilities within the battery limits has been included.

Items such as an administration building, water treatment facilities, utilities plant, instrument air, steam, water, tailings pond, and laboratories are not included in the estimate.

The structure around the process plant has not been included as the type of structure would be determined by local standards.

5.5 OPERATING COST ESTIMATES

5.5.1 Results

Operating requirements and operating costs for the proposed pressure oxidation and flotation circuit to produce 137 tonnes of flotation concentrate per operating day in 330 operating days per year, are tabulated overleaf. Major requirements are for lime, electric power, oxidation additive reagent, and maintenance. Oxygen to the process is supplied from a turnkey oxygen plant, so oxygen production costs are included in the overall utilities, labour and maintenance charges.

Annual Operating Requirements and Operating Costs

Item	Unit	Units/Year	Cost/Unit ($ U.S.)	Total Cost ($ U.S.)
Labour	manyear	22	1 900	42 000
Maintenance	% of installed equipment	3.7%		260 000
Materials & Supplies				
Flocculant	t	10	4 000	40 000
Process Additives	t	660	730	482 000
Lime	t	2 500	60	150 000
Grinding Balls	t	66	1 000	66 000
Limestone	t	12 940	3	38 800
Filter Cloth	m^2	255	20.4	5 200
Flotation Reagent	t	7.6	2 300	17 500
Operating Supplies	% of operating labour	6		2 000
Total Materials & Supplies				801 500
Utilities				
Electric Power*	MWh	29 388	65.00	1 910 000
Process Water	m^3	125 400	0.60	75 000
Cooling Water (make-up)	m^3	74 600	0.50	37 000
Demineralized Water	m^3	23 900	0.84	20 000
Mobile Equipment Fuel		Allowance		100 000
Total Utilities				2 142 000
TOTAL				3 245 000
SAY				3 250 000

* Electric Power consumed by oxygen plant = 15,900 Mwh/y

If a hot filtration circuit is included, the annual operating costs would increase to $3.5 million. Additional operating requirements and operating costs for the hot filtration circuit are summarized below.

Hot Filtration Annual Operating Requirements and Operating Costs

Item	Unit	Units/Year	Cost/Unit ($ U.S.)	Total Cost ($ U.S.)
Labour	manyear	5	1 900	10 000
Maintenance	% of installed equipment	2.6		21 000
Materials & Supplies				
Pressure Filter Cloth	m^2	1950	20.4	39 800
Filter Aid	t	50	340	17 000
Operating Supplies				1 000
Total Materials & Supplies				57 800
Utilities				
Electric Power	MWh	100	65.0	65 000
Steam - 1500 kPa	t	15 380	5.2	79 800
- 690 kPa	t	400	5.1	2 000
Total Utilities				146 800
Total Sulphur Recovery				235 600
Total Circuit, Including Pressure Oxidation and Flotation				3 480 000
SAY				3 500 000

5.5.2 Basis

Consumption of operating materials was developed from process requirements based on the material and energy balances and historical data.

Operating costs were estimated using data supplied by PPCL to establish unit prices for labour and materials. When costs for various items were not specified, in house data were used.

An exchange rate of Rs.10 equal $1 (U.S.) was used.

Operating labour requirements were developed from historical data and assumes U.S. Gulf Coast productivities. Operating labour would include four floor operators and one control room operator per shift, one day labourer and one day supervisor. If hot filtration is included one additional operator per shift and one additional day labourer would be needed. The amounts for supervision and overhead can be estimated, based on local practices, when the requirements for the pressure oxidation plant are combined with those for the mine and milling plant, and the auxiliary and utility plants.

Maintenance material and labour requirements were based on an appropriate percentage of installed equipment cost, depending upon the plant section. Maintenance costs do not include supervisory or general personnel.

5.5.3 Limitations

Local taxes, insurances and overheads have been excluded.

No allowance has been included for pumping or maintenance costs associated with the delivery of the tailings pond return water to the process.

5.6 BASIS FOR ECONOMICS

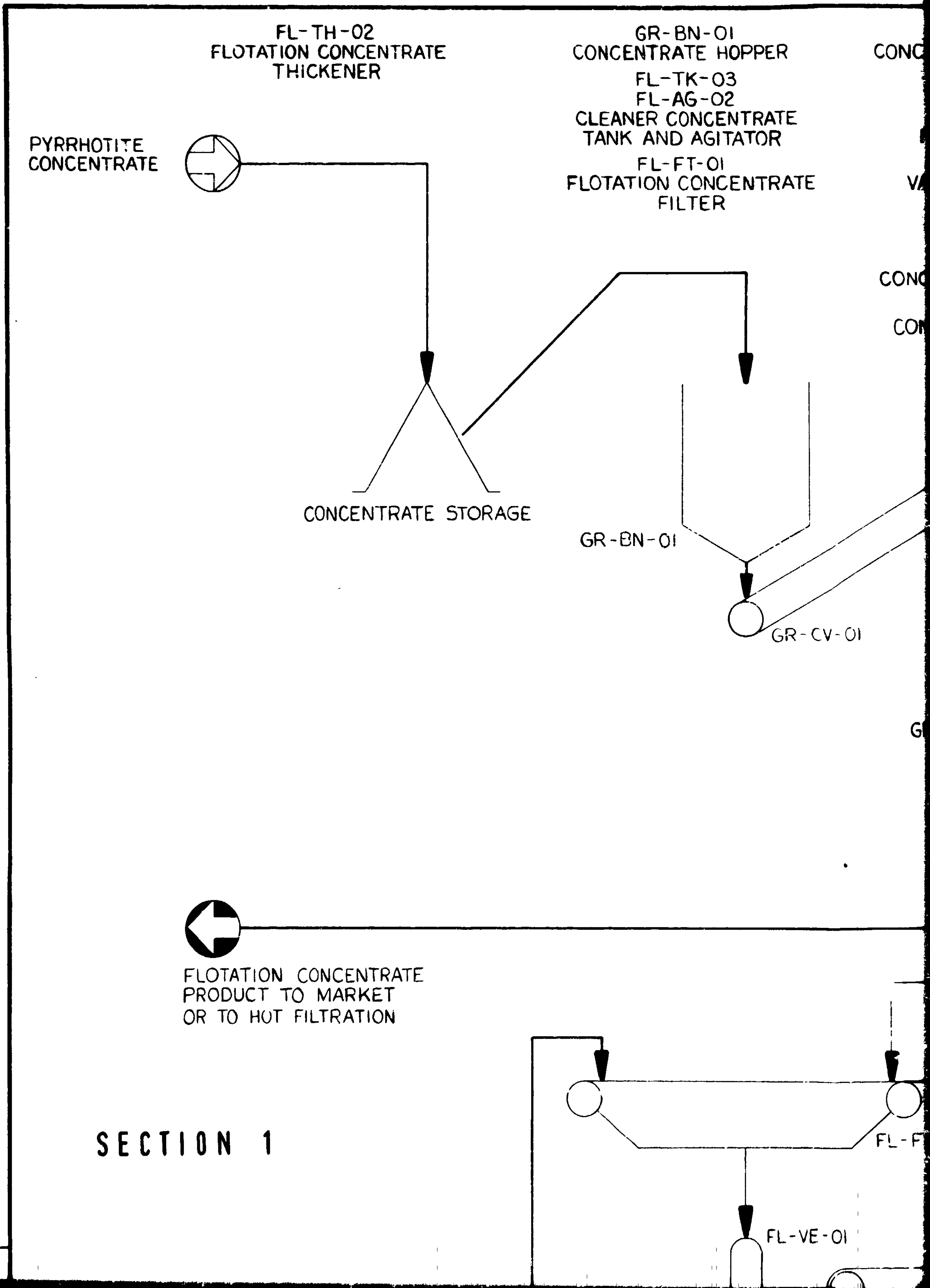
PPCL requested that Sherritt provide the basis required to evaluate the economics of the process. The approximate preproduction expense, start-up costs and the working capital for the process plant are shown overleaf.

Item	$'000 U.S.
Preproduction Expense*	400
Start-up Cost**	200
Working Capital***	600

* Preproduction expense includes detailed mini pilot plant testwork and writing of operating manuals.

** Start-up costs include operator training, cost of a start-up team, one month of maintenance costs, and one week of materials and supplies costs.

*** Working capital includes spares, in process inventory and 30 days of total operating costs less 30 days operating supplies and utilities.


FL-TH-02
FLOTATION CONCENTRATE
THICKENER
GR-BN-01
CONCENTRATE HOPPER
FL-TK-03
FL-AG-02
CLEANER CONCENTRATE
TANK AND AGITATOR
FL-FT-01
FLOTATION CONCENTRATE
FILTER
PYRRHOTITE
CONCENTRATE
CONCENTRATE STORAGE
GR-BN-01
GR-CV-01
FLOTATION CONCENTRATE
PRODUCT TO MARKET
OR TO HOT FILTRATION
SECTION 1
FL-VE-01

GR-CV-01
CONCENTRATE RECLAIM
CONVEYOR

FL-FC-20/25
CLEANER
FLOTATION CELLS

FL-VE-01
VACUUM RECEIVER

FL-VP-01
VACUUM PUMP

GR-BN-02
CONCENTRATE HOPPER

GR-WT-01
CONCENTRATE FEED
WEIGH BELT

GR-ML-01
CONCENTRATE GRINDING
BALL MILL

GR-CY-01/09
GRINDING HYDROCYCLONES

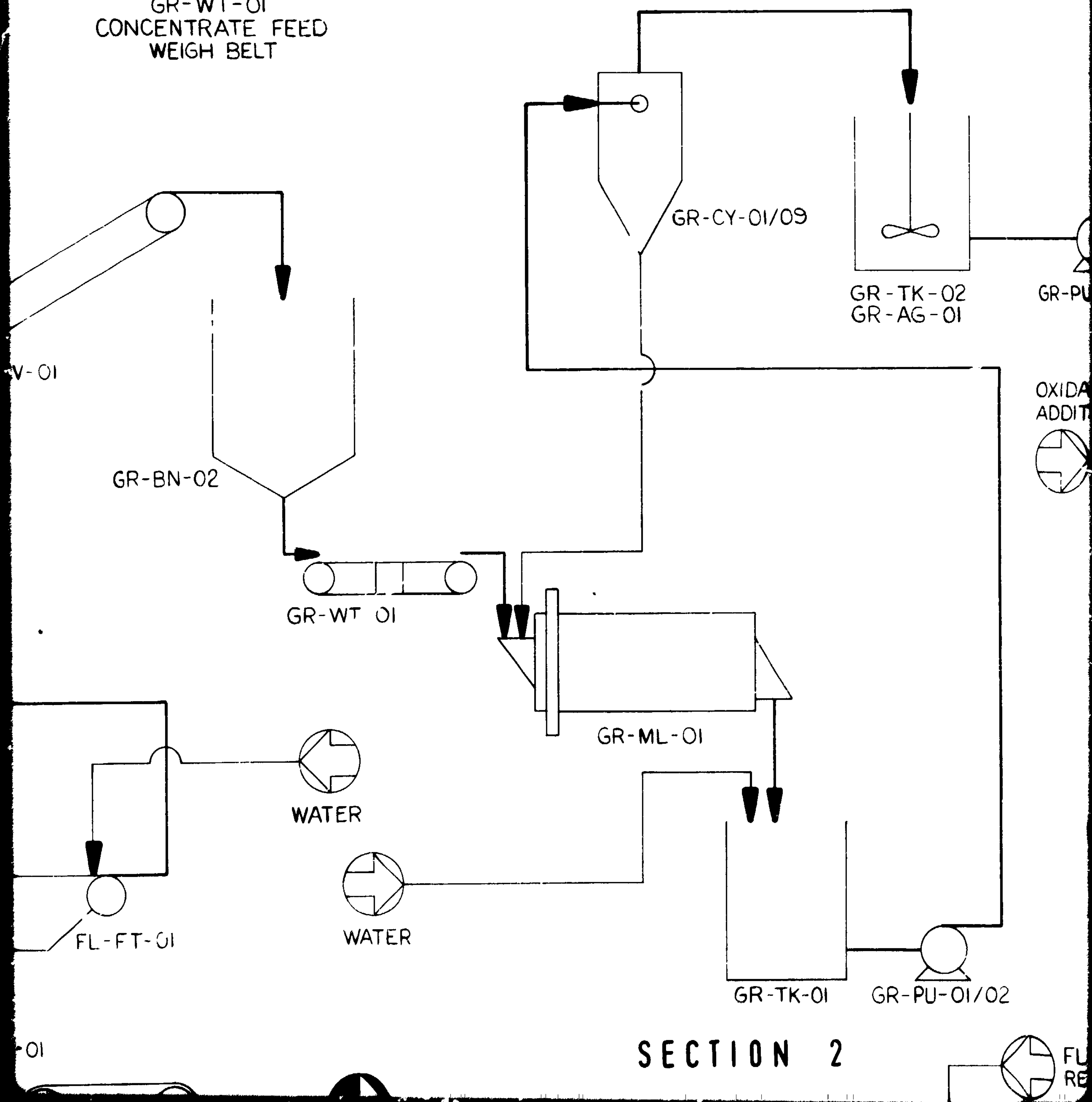
FL-FC-01/20
ROUHER/SCAVENGER
FLOTATION CELLS

GR-TK-01
MILL DISCHARGE
PUMP BOX

FL-TK-02
CONSTANT HEAD TANK

GR-TK-02
GR-AG-01
GROUND SLURRY RETENTIO
TANK AND AGITATOR

FL-TK-01
FL-AG-01
FLOTATION CONDITIONING
TANK AND AGITATOR



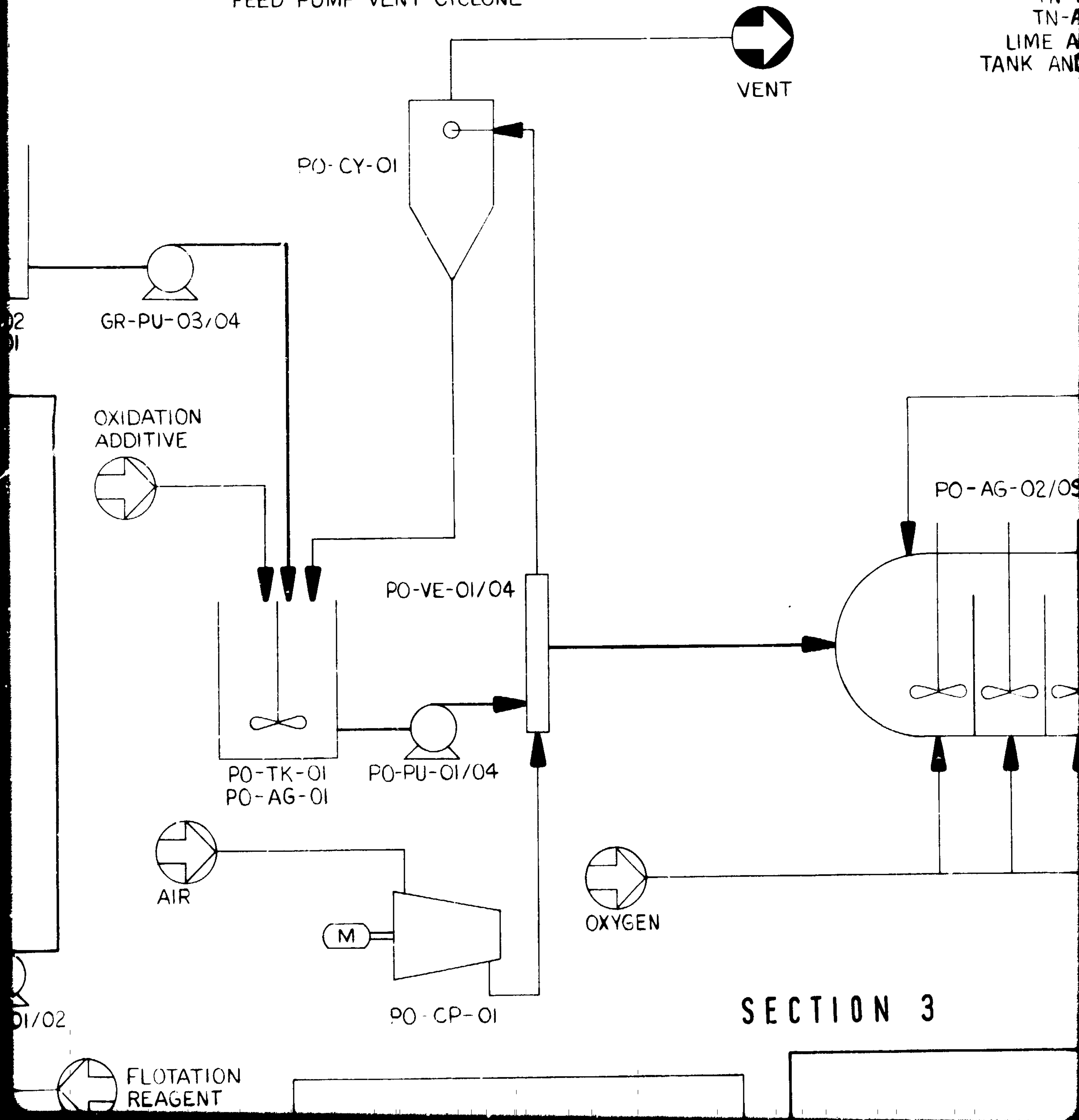
SECTION 2

-TK-02
T HEAD TANK
-TK-02
-AG-01
RRY RETENTION
ND AGITATOR
-TK-01
-AG-01
CONDITIONING
ND AGITATOR

PO-TK-01
PO-AG-01
FEED METERING
TANK AND AGITATOR
FL-TH-01
FLOTATION TAILINGS
THICKENER
PO-CP-01
AUTOCLAVE FEED PUMP
AIR COMPRESSOR
PO-CY-01
FEED PUMP VENT CYCLONE

PO-VE-01/04
AUTOCLAVE FEED PUMPS
PO-VE-05/06
PRESSURE OXIDATION
AUTOCLAVES
PO-AG-02/09
PRESSURE OXIDATION
AUTOCLAVE AGITATORS

PO-T
OXIDATIO
THIC
TN-TK
TN-AG
LIMESTONE
TANKS AND
TN-T
TN-A
LIMESTONE
TANK AND
TN-T
TN-A
LIME A
TANK AND



SECTION 3

PO-TH-01
XIDATION RESIDUE
THICKENER

TN-TK-01/02
TN-AG-01/02
MESTONE ADDITION
NKS AND AGITATORS

TN-TK-03
TN-AG-03
MESTONE REACTION
ANK AND AGITATOR

TN-TK-04
TN-AG-04
LIME ADDITION
NK AND AGITATOR

TN-TK-05
TN-AG-05
LIME REACTION
TANK AND AGITATOR

TN-TK-06
TN-AG-06
TAILINGS PUMP
TANK AND AGITATOR

PO-VE-07/08
FLASH TANKS

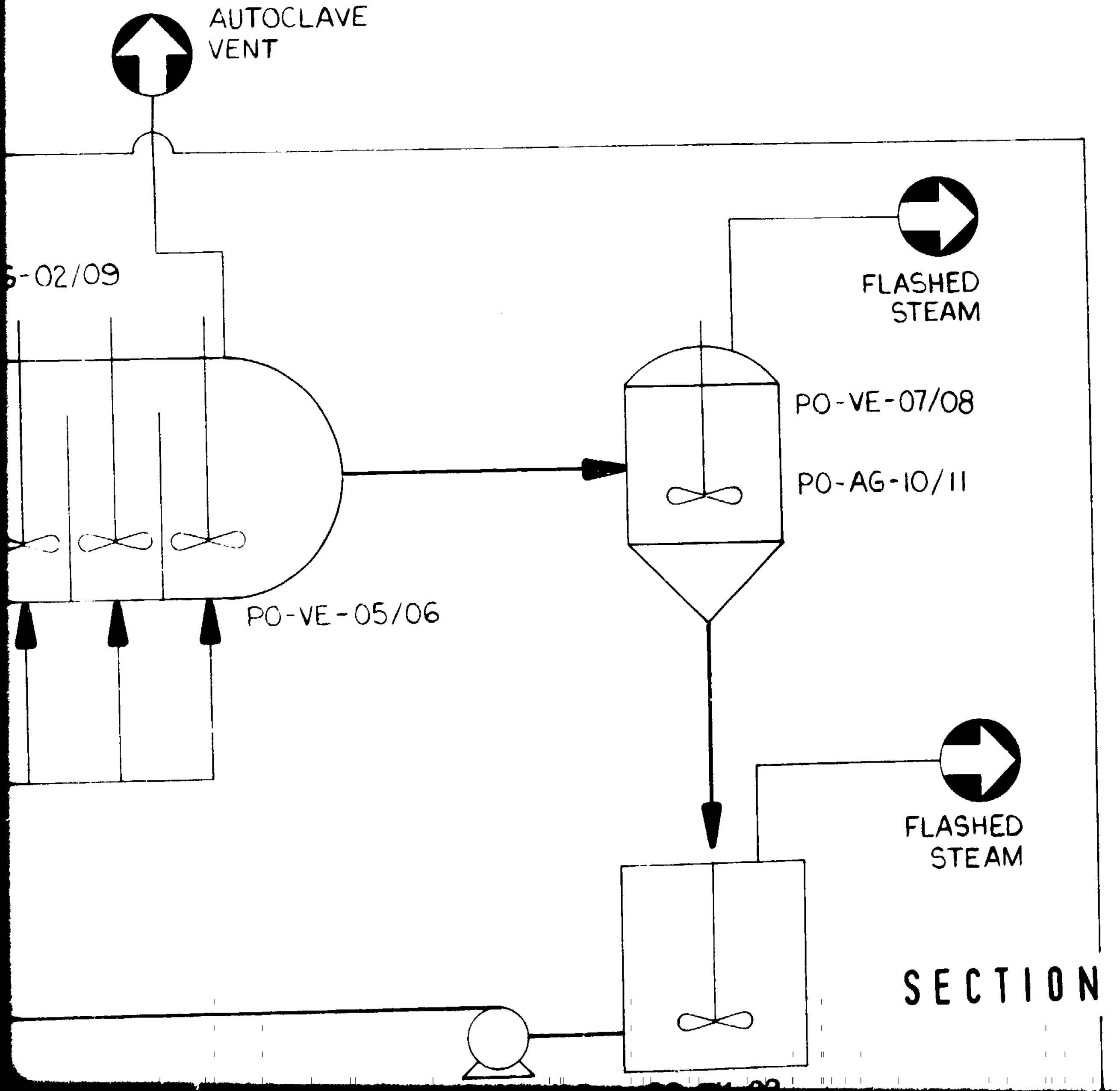
PO-FN-01
COOLING TOWER FAN

PO-AG-10/11
FLASH TANK AGITATORS

PO-CT-01
RECYCLE SOLUTION
COOLING TOWER

PO-TK-02
PO-AG-12
FLASH TANK SEAL
TANK AND AGITATOR

PO-TK-03
RECYCLE SOLUTION
PUMP TANK



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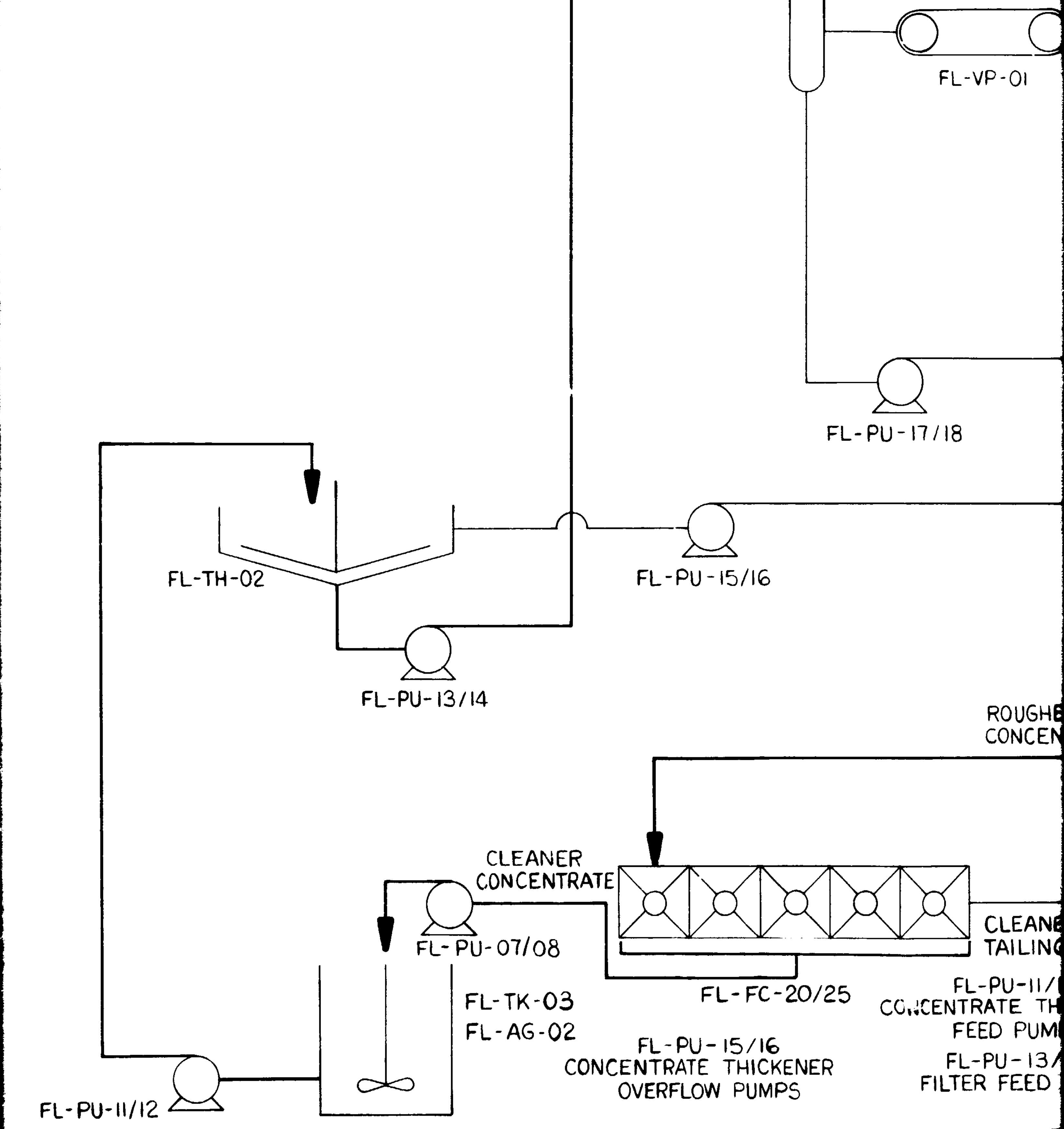
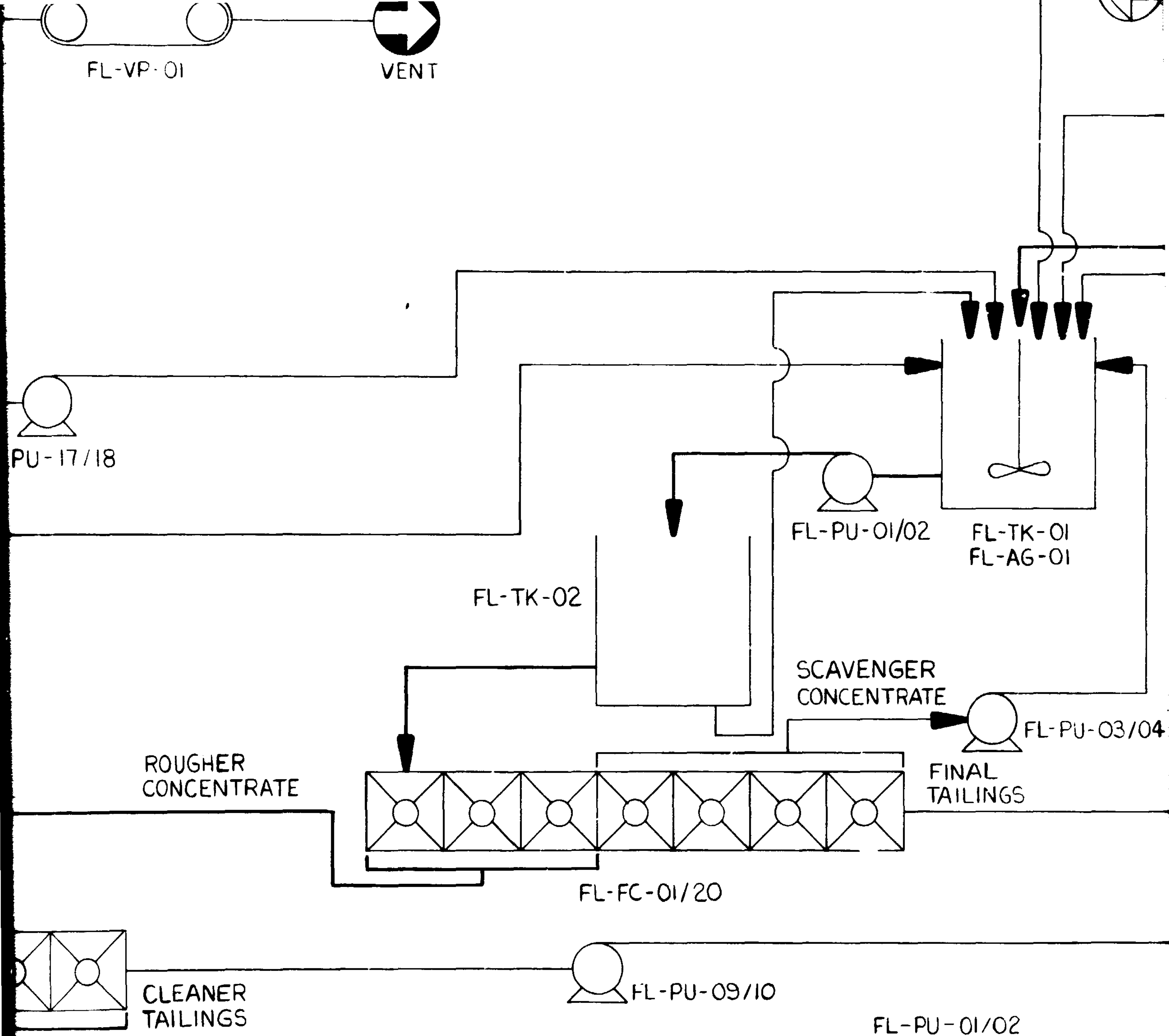


REFERENCE

DWG No	TITLE

SECTION 5



FL-PU-11/12
CONCENTRATE THICKENER
FEED PUMPS
FL-PU-13/14
FILTER FEED PUMPS

FL-PU-07/08
CLEANER
CONCENTRATE PUMPS
FL-PU-17/18
FILTRATE RECYCLE PUMPS

FL-PU-01/02
FLOTATION FEED PUMPS
FL-PU-09/10
CLEANER
TAILINGS RECYCLE PUMPS

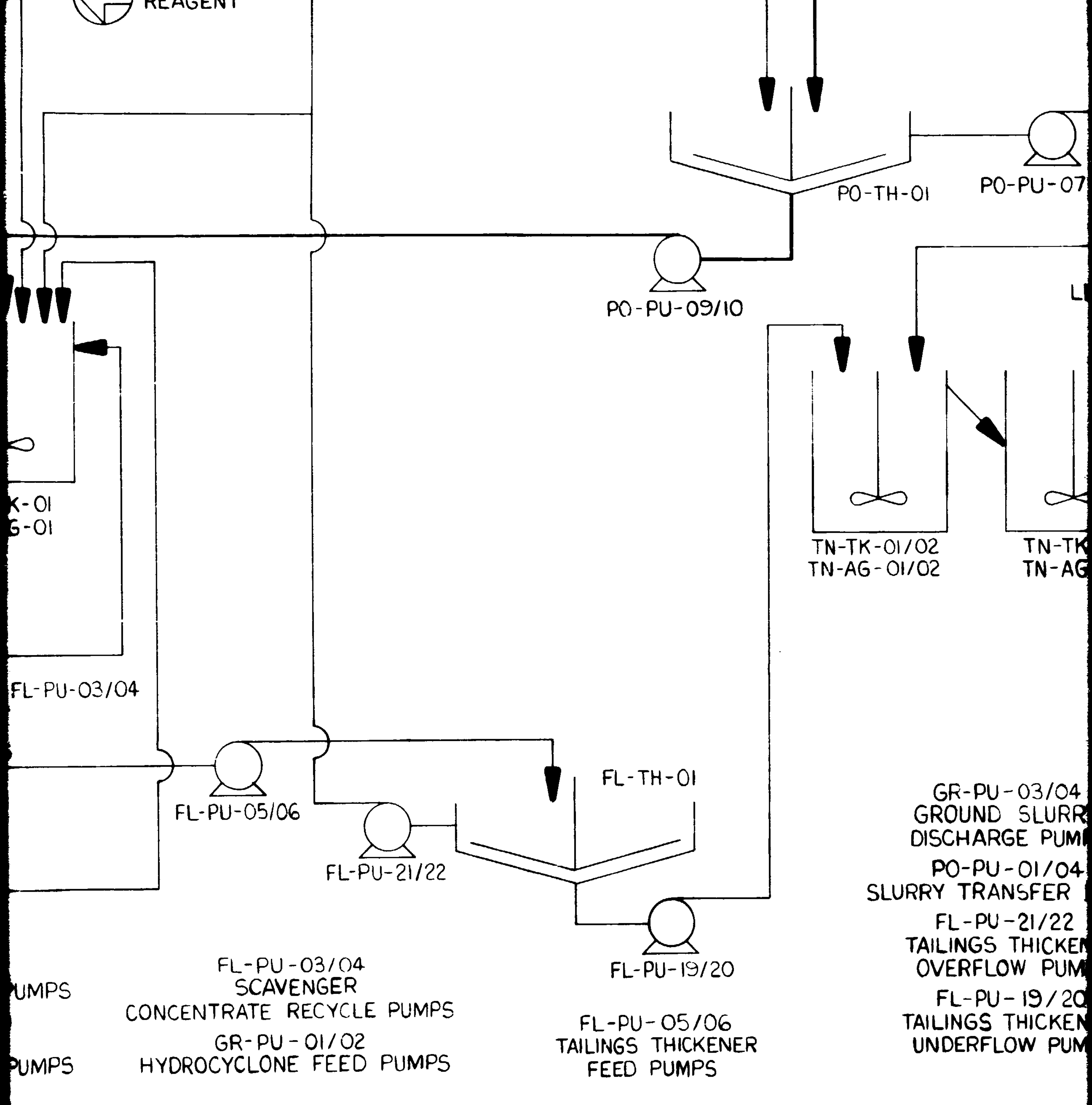
CO

H

SECTION 6

REVISIONS

○	LJS	W
No	BY	CHK'D



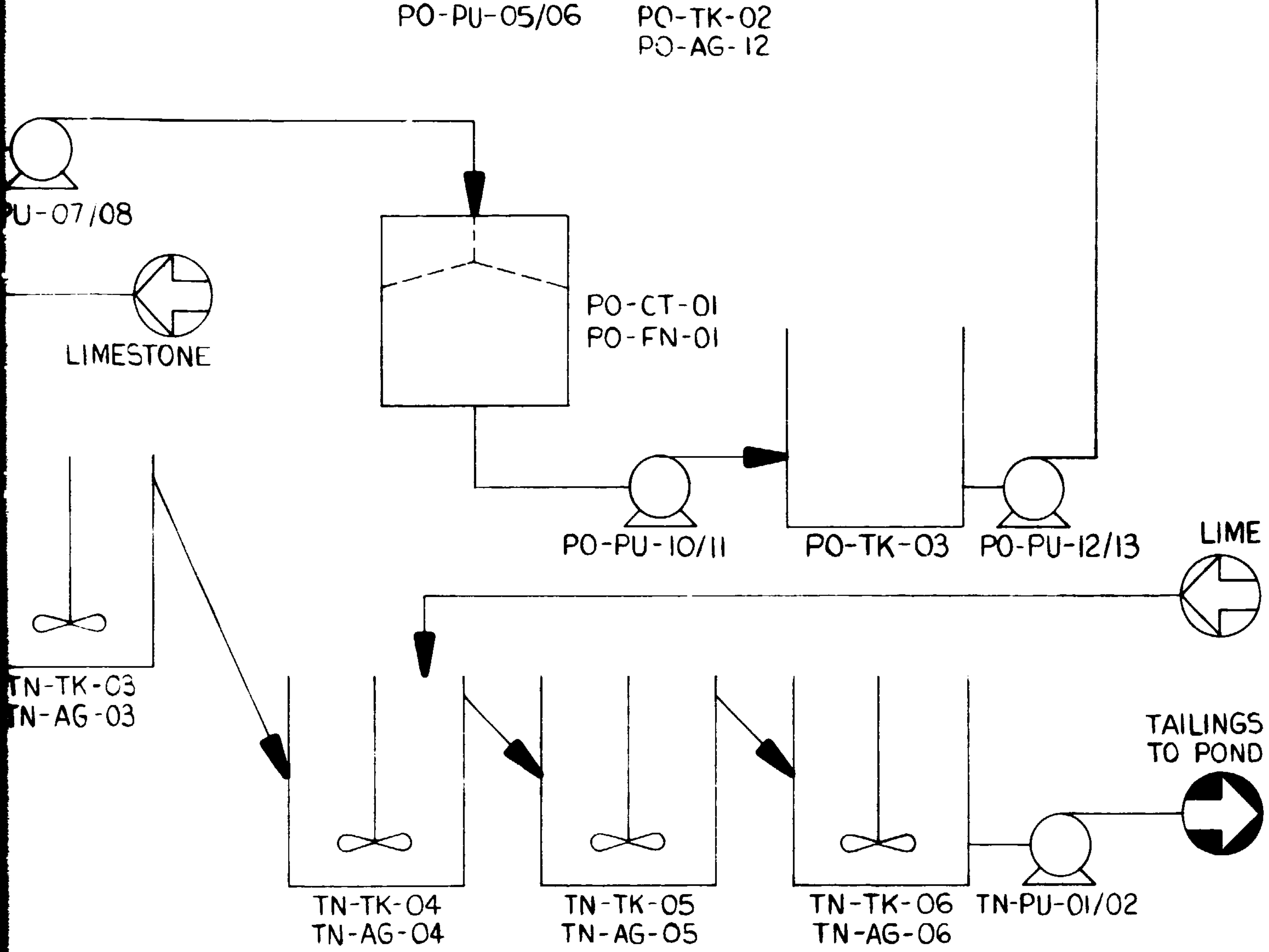
O	LJS	W		Jan 29/85
o	BY	CHK'D	APP'D	DATE

SECTION 7

APPROVALS					
MECH	PROC	ELEC	INST	PROJ ENG	PROJ MGR

JOB No	SCALE
7987	

BILL OF MATERIAL No



3/04
LURRY
PUMPS

1/04
FER PUMPS

1/22
ICKENER
PUMPS

9/20
CKENER
PUMPS

PO-PU-09/10
OXIDATION THICKENER
UNDERFLOW PUMPS

PO-PU-07/08
OXIDATION THICKENER
OVERFLOW PUMPS

PO-PU-05/06
OXIDATION THICKENER
FEED PUMPS

PO-PU-10/11
SOLUTION RECYCLE
PUMPS

TN-PU-01/02
TAILINGS DISCHARGE
PUMPS

PO-PU-12/13
RECYCLE SOLUTION AUTOCLAVE
FEED PUMPS

CONFIDENTIAL
SHERRITT GORDON MINES LIMITED

A

ROJ
GR

TITLE

SALADIPURA PYRRHOTITE
PROJECT
PROCESS FLOWSHEET
PHASE II

sherritt
SHERRITT GORDON MINES LIMITED
FORT SASKATCHEWAN, ALBERTA, CANADA

DRAWING NUMBER	ISSUE
09901-68-04-001	0

SECTION 8