

How Swedish metallurgists developed

For many years attempts have been made to concentrate scheelite by flotation in the presence of fluorspar, calcite, and apatite which have very similar flotation properties. Until recently, however, it was considered impossible to separate it

At Yxsjöberg in central Sweden (Figure No. 1), AB Statsgruvor mines a low-grade scheelite ore for production of scheelite, chalcopryrite, and fluorspar concentrates. Formerly the scheelite was recovered by gravity concentration. But because of unfavorable prerequisites for this beneficiation method, the operations have given unsatisfactory results until recently. Comprehensive experimental work, however, has resulted in a new selective scheelite flotation process, which is drastically improving the mine economies.

According to laboratory and pilot plant tests the new process yields concentrate which exceeds 65 percent WO_3 in grade and 75 percent in recovery from a feed assaying less than 0.4 percent WO_3 . The process came on stream in late 1977.

This article describes research and development work which resulted in improvements to the original gravity concentration and finally led to the novel process for selective flotation of scheelite in the presence of fluorspar and calcite. Results from full-scale operation will be presented in a later article.

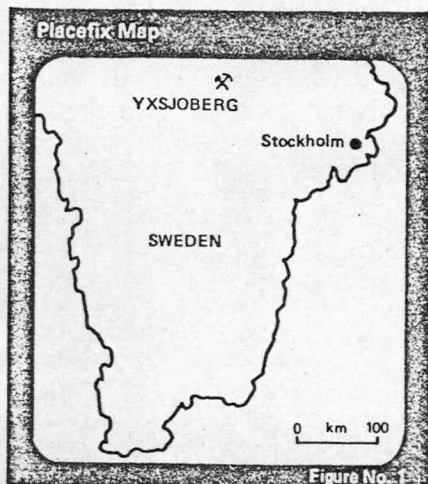
Statsgruvor is a wholly-owned subsidiary of LKAB, the large, state-owned iron ore mining group. Stopping at the Yxsjöberg underground scheelite mine is mainly cut-and-fill. The run-of-mine ore averages 0.3 to 0.4 percent WO_3 . The scheelite in the ore is partly fine-grained and is associated with several times larger quantities of calcite and fluorspar.

Skarn Ore

The Yxsjöberg field belongs to the upper, sodium-rich part of the Svecofennian leptite region, which in the area consists of moderately metamorphic sediments and volcanic rocks of intermediate to acidic composition. After folding and uprising, the complex has been tectonized and amply cut by metabasitic dykes. These strike

roughly east-west and are in parallel with the folding. Younger granites, traditionally believed to represent the parent magma of the ore-forming solutions, crop out some 4 miles to the east of the mine.

Mineralization occurred along a limestone horizon in the series, pre-



served in places but mainly transformed into three somewhat distorted bodies of amphibole and pyroxene skarn. The scheelite is associated with pyrrhotite and chalcopryrite as well as garnets, other skarn materials and abundant amounts of fluorspar.

The pyroxene skarn is characterized by xenomorphic hedenbergite normally coarser than 200 microns and seems to be unevenly impregnated with fairly coarse-grained scheelite. The amphibole skarn on the other hand is on average more fine-grained than the pyroxene skarn. It consists mainly of common hornblende with small scheelite grains fairly evenly distributed in the matrix. Some grain counts exemplifying the scheelite grain size distribution in these ore types are given in Figure No. 2. To a minor extent mineralization also occurred in garnet skarn and in pegmatite veinlets. The fluorspar and calcite

proportions vary considerably in all ore types.

On average the run-of-mine ore assays 0.38 percent WO_3 , 0.25 percent Cu, 2.2 percent S, 0.02 percent P, and 7.5 percent CaF_2 , which corresponds to the following mineral composition:

0.5 percent scheelite
0.7 percent chalcopryrite
5.5 percent pyrrhotite
0.1 percent apatite
7.5 percent fluorspar
5.1 percent calcite
5.2 percent quartz
10 percent biotite
15 percent feldspar
50 percent skarn minerals

Reopen Mine

In 1963 low tungsten prices forced the former owner, AB Yxsjö Gruvor, to cease operations. In 1969 AB Statsgruvor acquired the mine and after construction of new facilities production recommenced in 1972. Basically the same flowsheet was adopted for beneficiation as in the old plant.¹

The ore was crushed in three stages to minus-20-millimeters, each time bypassing the fines, and then ground by rod mills in two stages to minus-0.5-millimeter (Figure No. 3). A rougher scheelite concentrate, mainly consisting of pyrrhotite, skarn minerals, and garnet, was obtained on three-deck shaking tables and further upgraded in two stages on single-deck diagonal tables. Most of the pyrrhotite was discarded by wet magnetic separation and then the scheelite product was dried, roasted, and cleaned from residual impurities by dry low and high intensity magnetic separation.

Rejects from the roughing tables were reground in a ball mill operated in closed circuit with a hydrocyclone. After reducing the pyrrhotite content by magnetic separation, chalcopryrite was floated by conventional methods. The copper rougher concentrate was reground and cleaned in four stages.

selective scheelite flotation process

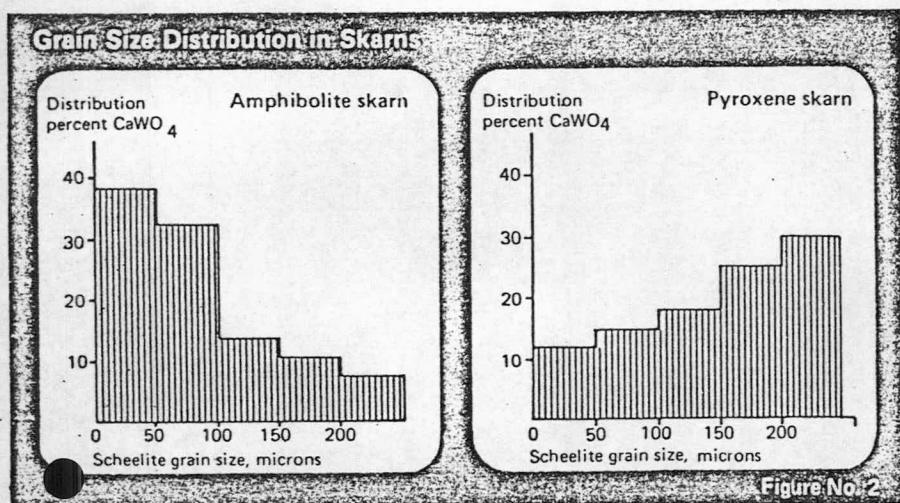
from fluorspar, and calcite-bearing ore gave only low grade concentrate or unacceptable recoveries. The new flowsheet described here has successfully solved these problems.

From the copper circuit rejects fluorspar and fine-grained scheelite were floated with tall oil as a collector. Scheelite in the final concentrate and some of the cleaner stage middling was recovered by three-stage tabling and then dried, roasted, and treated as described above.

In spite of great efforts to improve efficiency by adjusting the process, scheelite concentration results re-

tation methods were developed in bench and pilot plant tests and introduced at the concentrator. These modifications allowed steady progress and together with rising prices helped operations to become profitable. Of particular importance in the development was replacing the three-deck shaking tables by Reichert cone concentrators. This measure raised recovery in the gravity circuit from about 60 percent to about 80

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mained unsatisfactory for a long time. In 1974 overall scheelite recovery in concentrate assaying 68 to 70 percent WO₃ was still less than 50 percent. Due to this fact, and low tungsten prices, the mine was operating at loss. Therefore a project team with members from AB Statsgruvor and LKAB was appointed in order to develop a beneficiation process for more efficient recovery of scheelite.

Two different approaches were chosen aiming at short and long term improvements respectively. The former comprised modified gravity processes later to be combined with or replaced by flotation, depending on how selective such a process could be made.

Various modifications of the original flowsheet by gravity, magnetic, and flo-

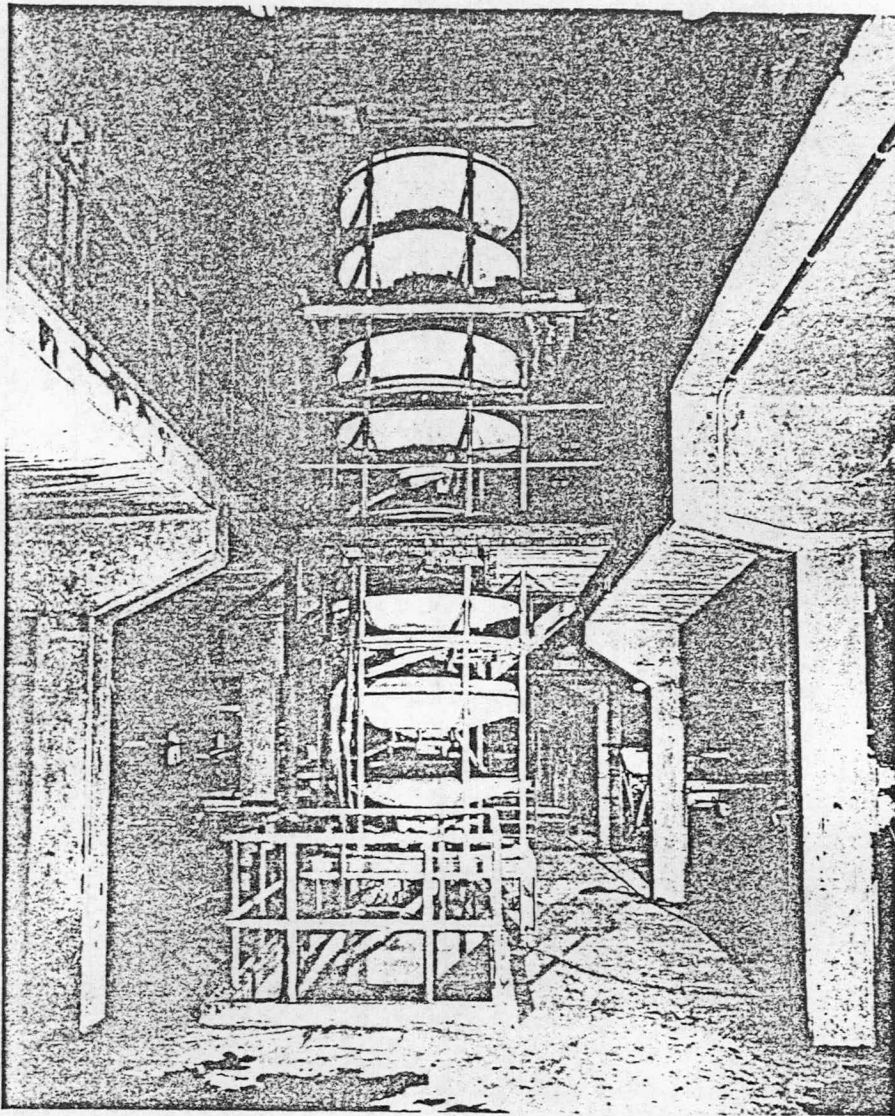
percent. However, the overall scheelite recovery was increased by only some 2 percent. The flowsheet in operation at Yxsjöberg the last year before the switch-over to all flotation is shown in Figure No. 3.

The three-decked tables were originally installed in order to reduce space requirements, but have proven troublesome to operate. Due to their excessive need for mechanical service and the difficulties to control the process, both grade and recovery were inconsistent, thus causing disturbances along the production chain. Replacing the tables with Reichert cone concentrators was a success because the latter stabilized the process, despite the fact that the cone theoretically falls short of tables as a concentrating machine.

Scheelite Flotation Process

For many years attempts have been made to concentrate scheelite by flotation in the presence of fluorspar, calcite, and apatite, which have very similar flotation properties. Until recently, however, it was considered impossible to separate it from fluorspar and only low-grade concentrate or by far unacceptable recoveries were achieved from calcite-bearing ores. Apatite seems to cause comparatively minor troubles.

The primary objective of the flotation tests in this study was to find a method to produce a saleable scheelite concentrate assaying at least 65 percent WO₃ at the highest possible recovery. Otherwise, it was to be investigated



REICHERT cone concentrator stabilized the modified gravity circuit.

whether low grade concentrates were marketable or could be chemically treated as to give overall economy.

At first, bulk flotation of calcium minerals with subsequent separation was studied. Selective desorption of the collector film was attempted utilizing a method devised by Petrov^{2,3} which involves conditioning the bulk concentrate in hot solution with sodium silicate.

With sodium oleate as a collector, desorption was fairly unselective, but it improved if the fatty acid were slightly modified. Very encouraging results were achieved with an amphoteric high molecular unsaturated carboxylic acid (KemaNord Lilaflo OS). In batch tests scheelite concentrate assaying 67 percent WO_3 was produced at an 85 percent overall recovery.

It proved, however, difficult to reproduce these results in the pilot plant as the process is sensitive and difficult to control. Particularly varying calcite contents in the bulk concentrate called for rapid changes of process parameters. Hence it seemed doubtful whether sufficiently steady conditions could be created without using very sophisticated equipment. The flowsheet of this Petrov model process is shown in Figure No. 4.

In parallel to the studies mentioned above tests were also carried out on selective flotation. Here it was tried to find suitable combinations of calcite and fluorspar depressing agents which would permit scheelite to be floated selectively. Chemicals such as soda ash, sodium silicate, and polyphosphates

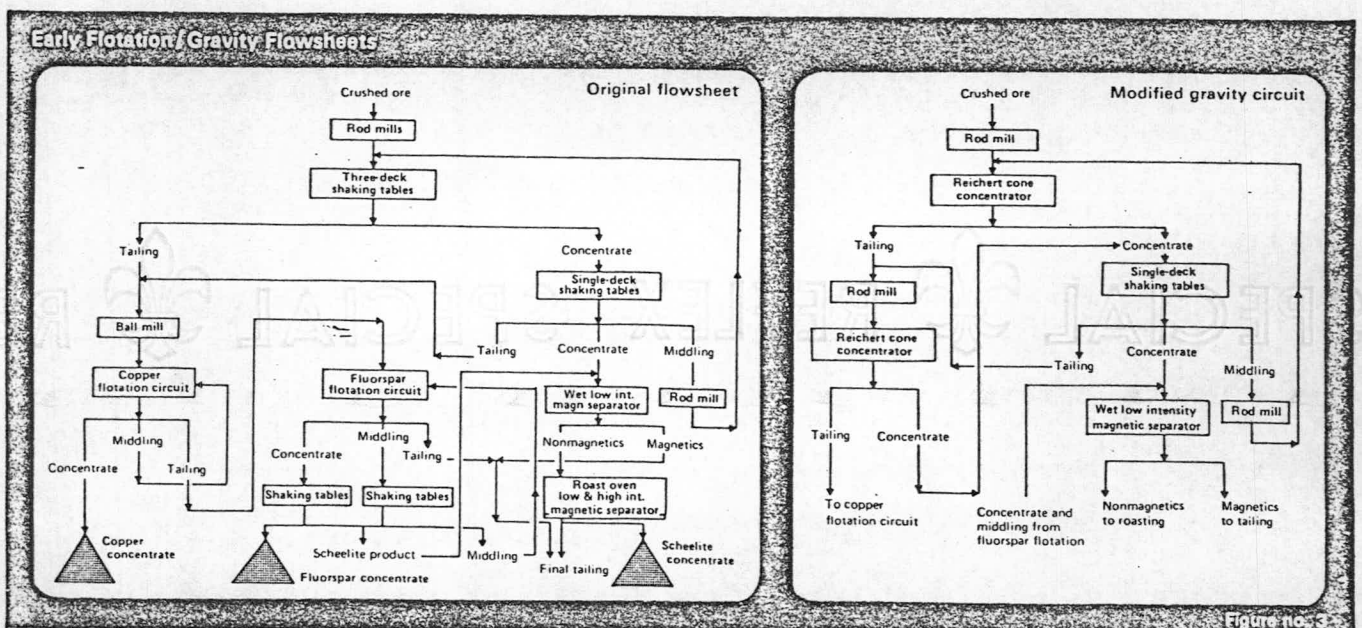


Figure no. 3

Pilot Plant Flowsheets

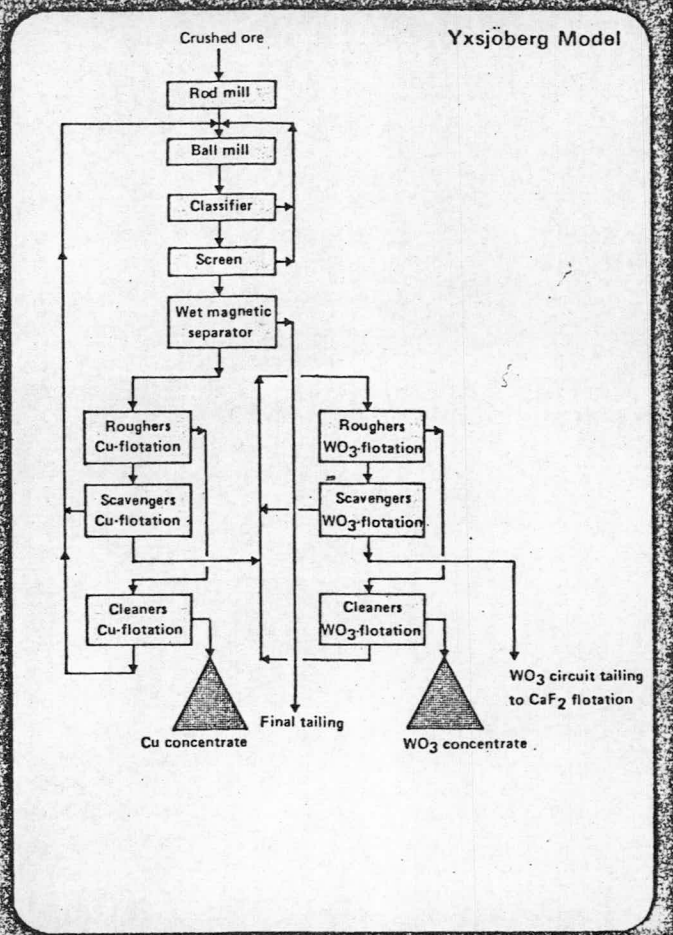
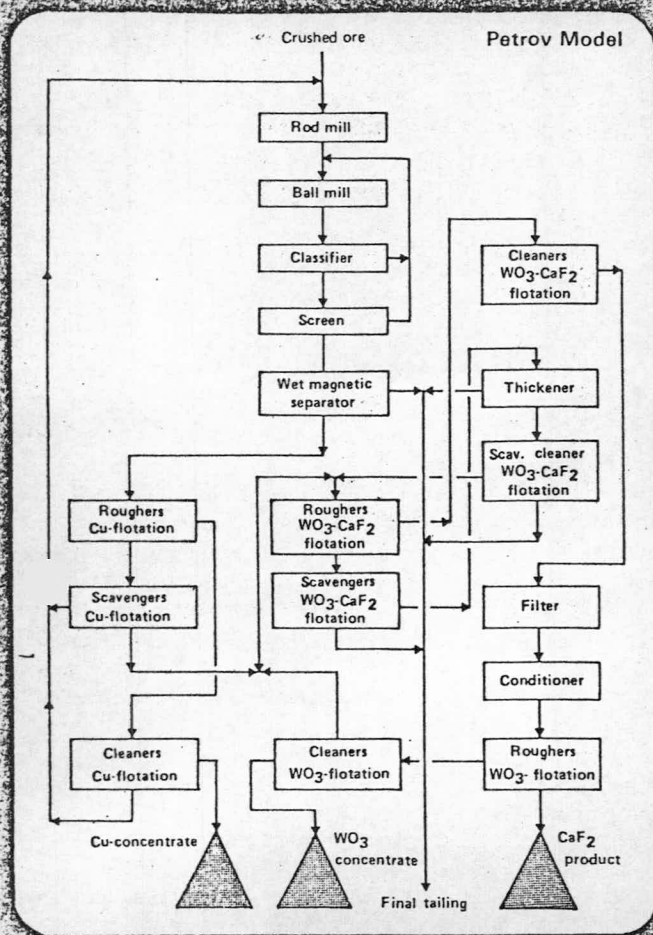


Figure No. 4

Table No. 1
Pilot Plant Flotation Tests

Products	Weight percent	Assay, percent		Distribution, percent	
		Cu	WO ₃	Cu	WO ₃
Impurities	3.70	0.10	<0.01	2.7	<0.1
Copper concentrate ¹	0.66	16.3	<0.01	75.8	<0.1
Scheelite concentrate	0.53	0.04	68.5	0.3	78.9
Final flotation tailing	95.11	0.03	0.095	21.3	21.1
Feed	100.00	0.14	0.45	100.0	100.0

¹ First cleaner concentrate

Table No. II
Chemical Composition of Scheelite Concentrate, Percent

Component	Percent	Component	Percent
CaWO ₄	83.2	Mo	0.008
CaF ₂	9.2	Mn	0.013
CaCO ₃	2.5	Fe	2.10
Fe ₂ O ₃	2.64	Cu	0.022
FeO	0.32	Be	0.004
MnO	0.02	Y	0.015
MgO	0.06	Sn	0.0015
Al ₂ O ₃	0.27	Pb	<0.0005
SiO ₂	0.60	As	<0.0005
TiO ₂	0.02	Sb	<0.002
V ₂ O ₅	0.09	Bi	0.023
P ₂ O ₅	0.09	P	0.04
Na ₂ O	0.04	S	0.085
K ₂ O	0.03		
	99.08		

with and without addition of metal salts, lignin-sulphonates, and quebracho were tested alone or together in various combinations. As collector sodium oleate was utilized throughout. To obtain optimum results it was also necessary to choose suitable pH and pulp temperature. The best test conditions resulted in high grade rougher concentrate at 80 to 85 percent WO₃ recoveries.

In order to confirm laboratory results, pilot plant tests were run with a capacity of one ton per hour according to the Yxsjöberg model flowsheet given in Figure No. 4. The ore was ground in two stages to 80 percent minus some 140 microns and then pyrrhotite was dis-

carded by magnetic separation. In the first flotation step chalcopyrite and the remainder of pyrrhotite were floated off. In the second stage the rougher scheelite concentrate was cleaned twice. The results of a typical pilot plant test are summarized in Table No. 1.

The scheelite concentrate contains extremely small amounts of impurities except some fluor spar, as seen in Table No. II.

In January 1977 it was decided to go ahead with the necessary modifications for a switch-over from gravity concentration to flotation. The concentrator has been reconstructed and production went on stream with the new flowsheet in late 1977. The plant has a rated capacity of 200,000 tons per year.

References

1. Rothelius, E., *Swedish Mineral Dressing Mills*, Stockholm, 1957.
2. Klassen, V. I. and Moukrosov, V. A., *An Introduction to the Theory of Flotation*, London, Butterworth, 1963, 333-334.
3. Mitrofanov, S. I., *Solution of Some Problems Concerning the Theory and Practice of Selective Flotation in the USSR*, Proceedings, 4th International Mineral Congress, Stockholm, Sweden, 1957.

END.

Flowsheet to separate scheelite concentrate from fluor spar, apatite, and calcite gangue proves more successful in full-scale plant than pilot process

New Swedish scheelite flotation process results

As mentioned in a previous article, (WORLD MINING, June 1978, pages 40 to 43), a new flotation process has been developed in Sweden for concentration of scheelite in the presence of fluor spar, calcite, and apatite, which have very similar flotation properties. The first successful pilot plant test with Yxsjöberg ore was performed by Gränges Mineral Processes at Stråssa, Sweden, in November 1976.

In January 1977 the board of AB Statsgruvor granted the necessary means for modification and reconstruction of the Yxsjöberg plant. The old process was continued until the summer holidays 1977 and the new one started production at the end of September that same year. Favorable circumstances, such as common business depression, decentralized reconstruction with a few responsables, and ample working space in the plant are the main reasons for the short reconstruction time.

After a few weeks operation and flowsheet adjustments the process yielded acceptable results. Gradual improvement has continued and results are now better than forecast. For practical reasons, results given here mainly describe the situation as of the end of April 1978.

Mining and Plant Capacity

All mining at Yxsjöberg is done underground with horizontal cut-and-fill as the main method. The annual output has been up to 150,000 metric tons, but it will decrease to 125,000 tons. However, the concentrator throughput in the near future will be gradually increased to nearly 250,000 tons per year by treating other scheelite ores in the area and by feeding gravity tailings from old dumps.



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The new process operates with a conventional flowsheet, comprising crushing in three stages, grinding in rod and ball mills, and copper and scheelite flotation. The fluorite content of the ore is not recovered at present. Figures 1 to 3 show the grinding, copper, and scheelite circuits respectively, with material, pulp, and metal balances as recorded during a detailed circuit sampling on January 17, 1978. Weekly average results of copper and scheelite flotation during a four-month period are given in Figure 4.

Grinding Circuit

The grinding circuit, Figure 1, is a compromise between equipment at hand and desired design. The ball mill is rubber lined and operates in closed circuit with a 10-inch cyclone. Mill power data are given in Table 1.

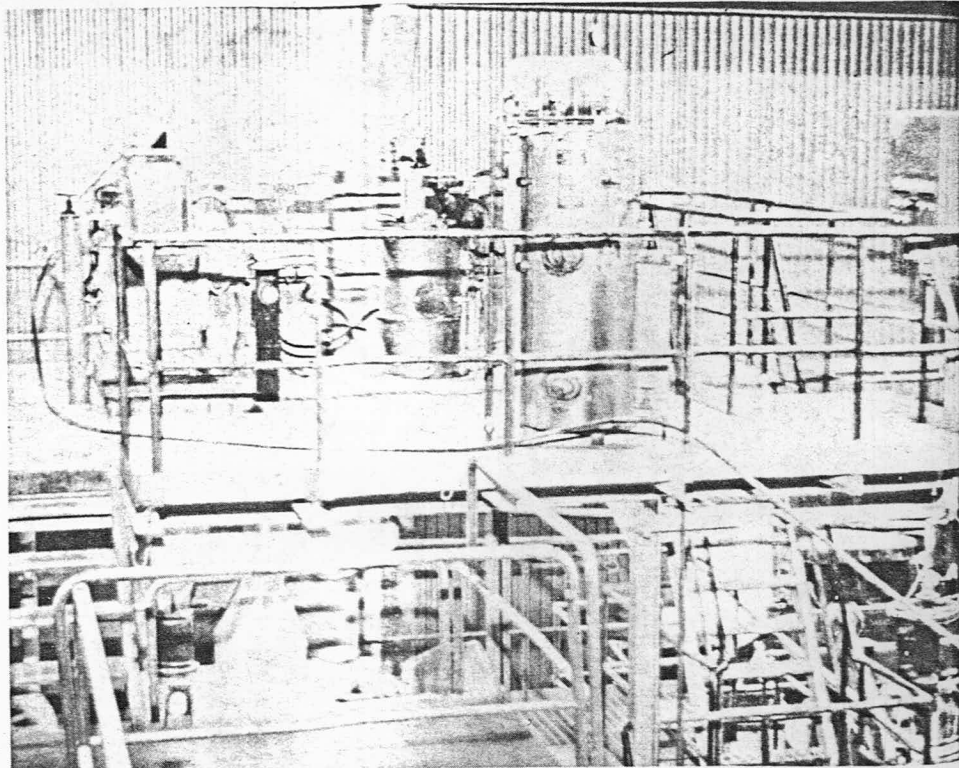
Scheelite is very brittle and has a

high density; hence it is easily ground and accumulates in the finer sizes; part of it is lost in the ultrafines, as will be shown below. Though the feed to the cyclone contains 63.5 percent solids, classification is acceptable. The cyclone is provided with two automatic control circuits; one adjusts the pump speed by comparing the actual level in the pump sump with its preset value; the other regulates the water addition to the cyclone feed, guided by the solid-ratio in the cyclone overflow. These control circuits are of utmost importance for the classification efficiency.

The final grinding product leaving

Table 1.
Yxsjöberg Mill Data

	Mill power, kilowatts		Consumption Gross	Kilowatt hours per ton Net
	Gross	Net		
Rod mill	130	95	5.9	4.3
Ball mill	135	99	6.1	4.5
Total	265	194	12.0	8.8



SCHEELITE FILTER CAKE is jumped into a steam-heated drying oven, which discharges directly into the storage bin.

the circuit in the cyclone overflow is sized as follows:

Screen Opening, Microns	Cumulative Percent Weight Passing	Screen Opening, Microns	Cumulative Percent Weight Passing
583	100	74	50
297	95	44	32
147	77	9	6.5

The 80 percent passing size is about 160 microns. Hence it is a comparatively coarse material with a high percentage of solids that goes to flotation. In fact, the solid-ratio of the flotation feed has gradually been increased and is now about 50 percent, which seems to be suitable for the process. The equipment chosen could be adapted to these somewhat unusual parameters. After proper adjustment, operation now runs almost undisturbed.

Copper Flotation

The chalcopyrite content of the ore is recovered as a valuable byproduct. A comparatively long scavenger series and four cleaner stages are used (see Figure 2). Results for a four month period are given in Figure 4 and summarized in Table 2.

Chalcopyrite selectivity from both scheelite and pyrrhotite is very good and is hardly affected by varying reagent addition or operating characteristics.

Scheelite Flotation

The scheelite at Yxsjöberg is very pure and contains less than 0.05 percent Mo; thus its composition can be calculated stoichiometrically to comprise 80.5 percent WO_3 . However, the ore also contains accessory amounts of wolframite, estimated to about 0.01 percent in WO_3 ; as these are not recovered by the flotation process, but reported in the analysis, the theoretically possible WO_3 recovery in scheelite from the ore is lower than 100 percent.

Scheelite flotation at Yxsjöberg has strictly speaking not yet reached steady state conditions and, as the assaying routines at the plant are too slow to serve as an operational guide, metallurgical results hitherto have varied considerably (see Figure 4). Average results for a four-month period are given in Table 3.

Figure 3 shows the flowsheet and gives the circuit sampling results for a day when the plant treated a fairly low-grade ore with only 0.34 percent WO_3 . The circuit was designed for large circulating loads from the scavengers to the rougher cell, but under these circumstances a high-grade concentrate with 73.4 percent WO_3 was also ob-

Table 2.
Results of Cu-flotation at Yxsjöberg January to April 1978

	Percent Weight	Assays, Percent		Distribution, Percent	
		Cu	WO_3	Cu	WO_3
Feed	100.00	0.235	0.398	100.0	100.0
Concentrate	0.84	22.5	0.01	80.6	0.0
Rejects	99.16	0.046	0.401	19.4	100.0

Table 3.
Results of Scheelite Flotation at Yxsjöberg January to April 1978

Period/Product	Weight-Percent	Assay, Percent			Distribution, Percent		
		WO_3	$CaCO_3$	CaF_2	WO_3	$CaCO_3$	CaF_2
Feed	99.16	0.401	5.6	6.5	100.0	100.0	100.0
Concentrate	0.46	69.0	11.1	2.2	79.8	0.9	0.2
Rejects ¹	98.70	0.081	5.57	6.5	20.2	99.1	99.8

1. Flotation tailings, as the fluorite is not recovered

tained at an 80 percent recovery. The data also show that rougher and scavenger flotations are very selective with concentration ratios of 40 to 80 times and that the cleaners serve mainly as a concentrate buffer. This mode of operation seems to be quite essential for achieving high grade concentrates at acceptable recoveries with the process. The sampling results also demonstrate that the selectivity from fluorite is better than from calcite. In fact, fluorite causes very little trouble.

In connection with the circuit sampling mentioned above, sizing sorting assay tests were made on feed, concentrate, and rejects of the scheelite circuit. Results are given in Table 4. As seen, selectivity is very good in the range 26 to 125 microns with a 97.6 percent recovery into a concentrate with 73.4 percent

WO_3 or 91.2 percent $CaWO_4$. The total losses were 20.4 percent WO_3 , altogether 11.9 percent report in the slimes minus 9 microns, where the recovery is only 23.6 percent. Hence it can be stated that this range offers a great potential for further development.

A complete analysis of the concentrate obtained during the period January to April 1978 (see below) shows that the main impurities are calcite and fluorite. The percentage of other harmful elements, such as Si, Mo, As, P, etc., is low:

$CaWO_4$	85.7	P	0.06
$CaCO_3$	11.1	S	0.034
CaF_2	2.2	Sn	<0.05
Total	99.0	Bi	<0.05
Si	0.22	Mn	<0.05
Mo	0.024	Fe	0.16
As	<0.05	Cu	0.01

Grinding circuit

Figure 1

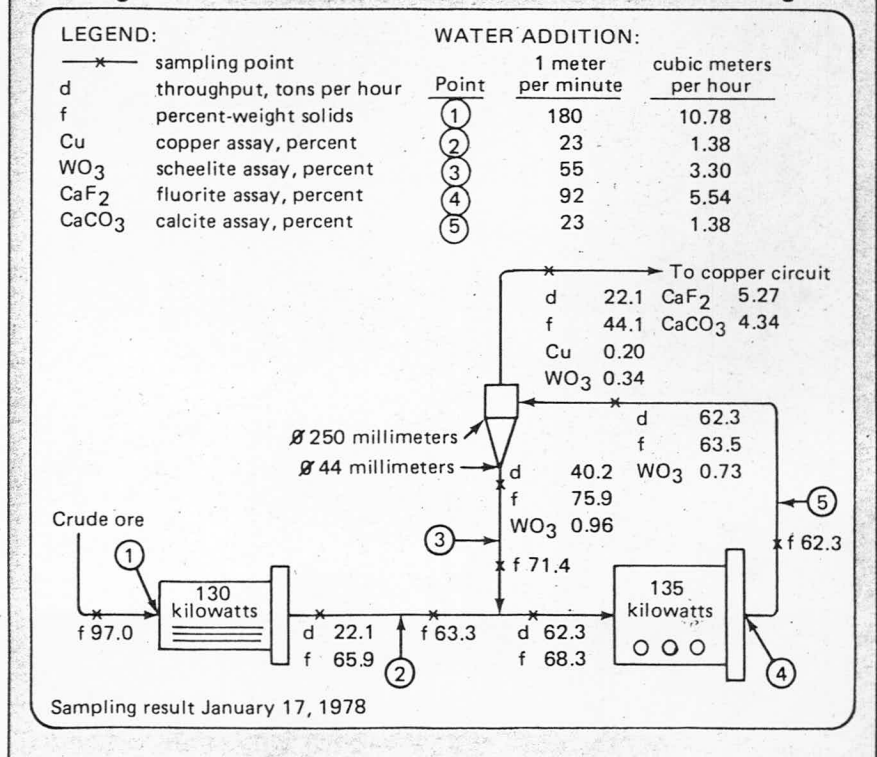


Table 4.
Scheelite Flotation at Yxsjöberg January 17, 1978, Sizing Sorting Assay Tests

Fraction Microns	CIRCUIT FEED			CONCENTRATE				TAILINGS		
	21.96 tons per hour = 100 percent			0.083 tons per hour = 0.373 percent				21.88 tons per hour = 99.627 percent		
	Weight Percent	Assay WO ₃ Percent	Distr. WO ₃ Percent	Weight Percent	Assay WO ₃ Percent	Distr. Percent	Recovery Percent	Weight Percent	Assay WO ₃ Percent	Distr. Percent
+ 250	8.0	0.13	3.0	1.1	17.5	0.3	8.8	8.0	0.12	13.7
250-125	21.7	0.13	8.3	6.9	59.2	5.6	77.0	21.7	0.03	9.3
125- 75	20.4	0.25	15.0	15.3	72.3	15.3	96.0	20.4	0.010	2.9
75- 45	15.9	0.39	18.2	22.9	74.3	23.4	97.4	15.9	0.010	2.3
45- 26	13.0	0.48	18.3	23.6	73.5	24.0	99.0	13.0	0.005	0.9
26- 9	14.4	0.51	21.5	26.0	75.8	27.2	88.3	14.4	0.06	12.4
9	6.6	0.81	15.7	4.2	72.6	4.2	23.6	6.6	0.62	58.5
Total:										
Calculated	100.0	0.34	100.0	100.0	72.4	100.0		100.0	0.070	100.0
Analyzed					73.5		79.6		0.069	

Concentrate Treatment

The scheelite concentrate is continuously filtered and dried. Low production rate, high product value, and a demand for almost complete dryness, which permits air sizing, have required a special flowsheet design.

From the cleaner, circuit concentrate is pumped to a vacuum filter and only filter overflow is thickened and pumped back to the filter. In order to avoid slime losses, the thickener overflow is recirculated as wash water in the launders of the scheelite circuit. For the same reason the water seal of the vacuum pumps is provided with a large sedimentation basin, which is emptied

once or twice a year. It then contains some 10 kilograms of concentrate.

The drying oven is indirectly steam-heated and discharges directly into a concentrate bin. Oven temperature is regulated by presetting the value of the steam pressure. To avoid dust losses, the exhaust gases are cooled and the condense water with the slime is discharged into the thickener.

Scheelite concentrate is shipped in sealed 45 gallon barrels.

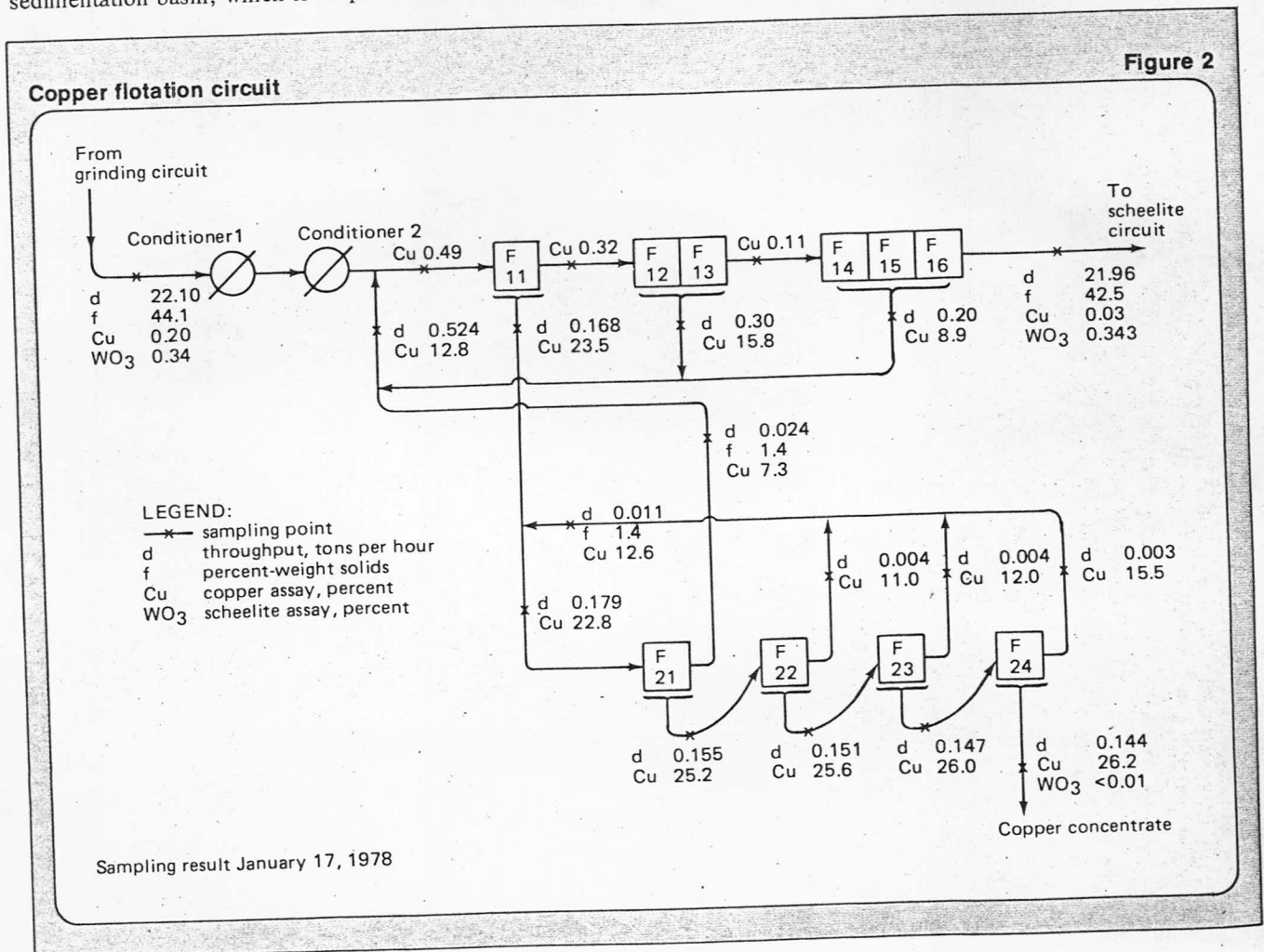
Process Description

In the grinding circuit, the pH-value of the pulp is roughly adjusted and then the flotation chemicals are added to two

conditioners ahead of copper flotation (Figure 2). Chacopyrite is recovered in the primary flotation stage.

The rejects from copper flotation are conditioned with the necessary amount of fatty acid. In the conditioner, the scheelite flocculates selectively into aggregates of up to a hundred particles. The aggregates are so stable that special efforts are required to destroy them sufficiently in the cleaner circuit to liberate and remove entrapped particles, particularly of calcite and fluorite.

Tailings are flocculated with caustic lime, which is added in the pumps ahead of the 1,300-meter-long pipeline leading to the pond. The effluent from the sub-



sequent clarification pond has a turbidity of <10 FTU and contains <6 milligrams per liter of suspended matter. Its content of base metals is very low, that of copper being highest with about 50 parts per million. The total fluorine content, however, is 4 to 5 milligrams per liter and a special investigation is in progress to find means to diminish it.

Personnel and Costs

The plant is operated by six shift crews, each consisting of one foreman, one mill operator, and one flotation attendant. The shift cycle is 6 weeks long and each crew works one week per cycle on day shift and serves as replacement for absentees in other crews.

Maintenance and repair require three mechanics and two electricians on day-shift basis, foremen partly being shared between mine and plant.

For sample preparation and assaying there are five part-time workers guided by one part-time laboratory engineer.

Plant personnel thus totals 28.5 positions including the mill superintendent, or 19.5 workers and 9 employees.

Direct operating costs total about 38 Swedish Crowns (\$8.75) per metric ton of ore and can be divided as follows:

Table 5.
Full Scale Test Runs with Wigström Scheelite Ore

Test Run Number	Crude Ore			Concentrate			Tailings	
	Weight Percent	Assay Percent WO ₃	Percent CaF ₂	Weight Tons	Assay Percent WO ₃	Percent CaF ₂	Recovery Percent WO ₃	Assay Percent WO ₃
1	7,390	0.367	9.1	35.9	61.1	0.8	80.7	0.071
2	10,231	0.261	8.5	30.7	66.8	0.8	76.7	0.061
Total	17,621	0.306	8.8	66.6	63.7	0.8	78.7	0.065

Item	Crowns per Ton	Dollars per Ton
Wages	10	2.30
Reagents	9	2.07
Electrical energy	3.50	0.81
Maintenance	6	1.38
Miscellaneous	9.50	2.19
Total	38.00	8.75

Other Ores and Old Tailings

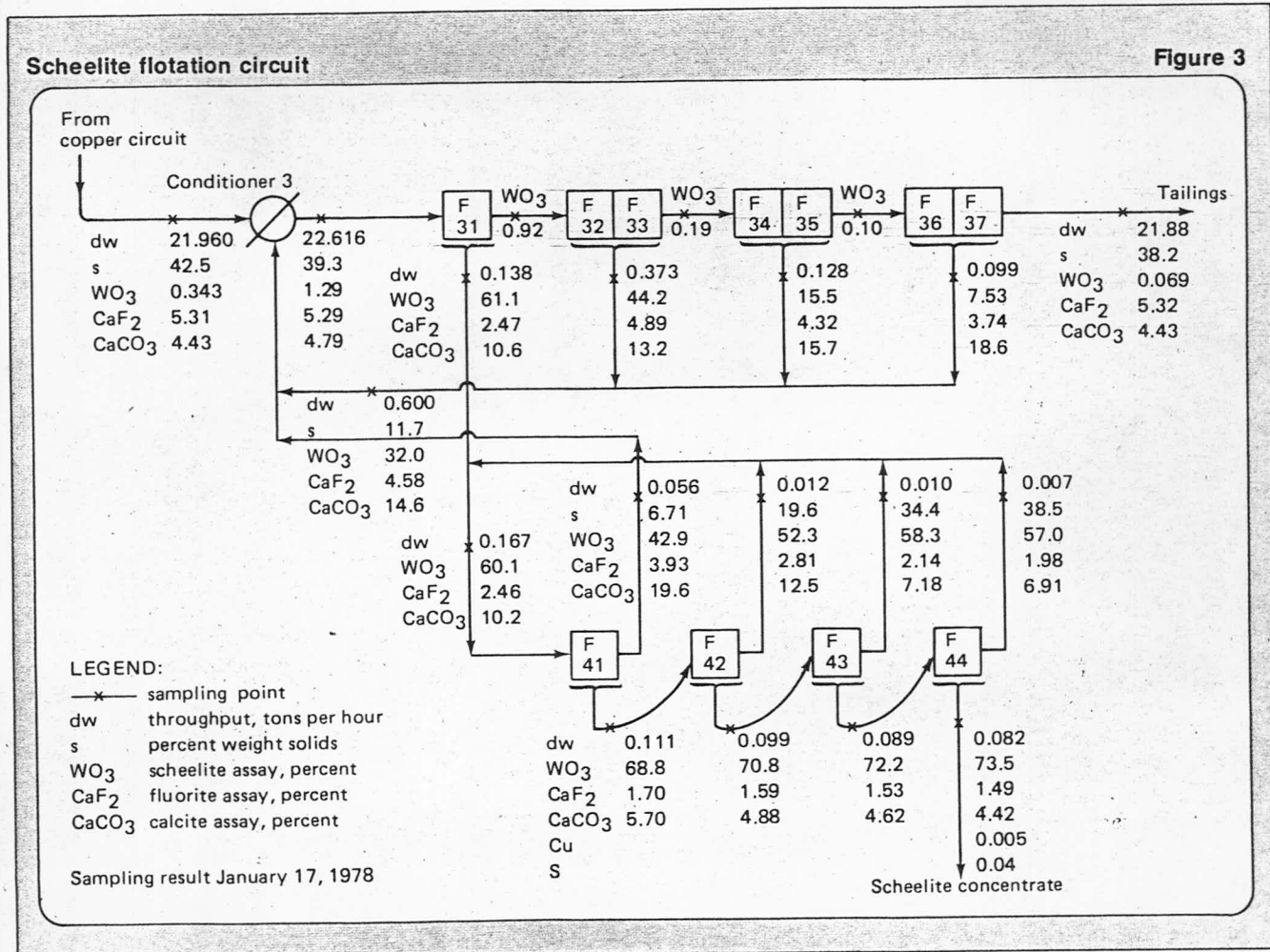
During the period May to August 1978 full scale test runs were made with scheelite ore from another deposit in the area and with old tailings. The other deposit, known as Wigström, lies 15 miles east southeast of Yxsjöberg and is currently mined as an open pit. The ore is almost completely free of sulphides and contains about 0.3 percent WO₃, 8.5 percent CaF₂ and varying proportions of calcite (0.5 to 3 percent). The scheelite is of the orange type and carries about 2 percent Mo in the lattice. After a few amenability tests in the laboratory two full scale trial runs lasting 14 and 15 days respectively were performed with the ore. Results are given in Table 5.

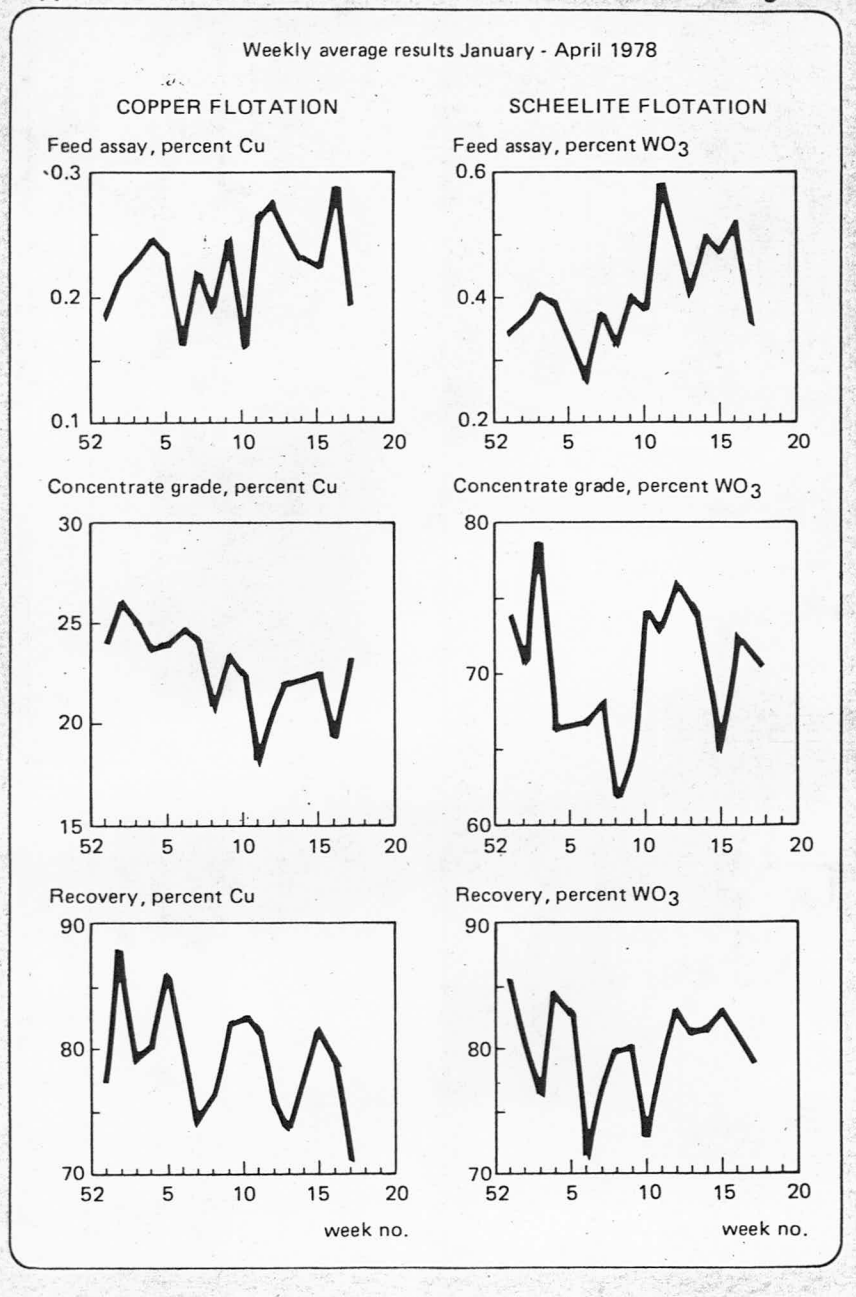
Part of the old tailings at Yxsjöberg contains on average 0.15 to 0.18 percent WO₃. The material is heavily coated

As production is small and costs for supplies large, the result must be accepted. Specific costs can hardly be diminished, except by increasing throughput. Profitability is also strongly affected by changes in feed assay and metallurgical result. Thus, for example, the net return diminishes by about half if the feed assay drops from 0.40 percent to 0.30 percent and it increases by about 50 percent if the feed assay rises to 0.50 percent WO₃.

Scheelite flotation circuit

Figure 3





with depressing reagents, as fluorite flotation was the last beneficiation step during the time of deposition. Laboratory tests with the new method indicated that about 60 percent of the scheelite could be recovered as a concentrate with 45 percent WO₃ and 35 percent calcite. However, a trial run in the plant during nine days with 5,735 tons of tailings assaying 0.124 percent WO₃ yielded on average a 66.5 percent concentrate at 56.4 percent recovery. The day by day development of the run indicates that it will be possible to reach a 65 percent recovery at 68 percent concentrate grade.

Judging from these results, it seems

likely that the process suits most scheelite ore types. This fact considerably facilitates ore evaluation during the current prospecting in the surroundings of Yxsjöberg. For more precise estimates, however, continuous testing is necessary. Full scale test runs in the existing plant with some 1,000 tons of ore offer a great advantage over customary pilot-plant testing as the net return of the test work immediately will pay for developing the new deposit.

Future Development

Daily variations of metallurgical results in the plant are largely caused by lack of adequate process information.

For one thing, retention time in the crude ore bin is too short to permit valid information for planned ore treatment; corresponding information from the mine is too vague and can be used as long term guidance only. At the same time it has proved almost impossible in plant practice to judge visually whether a concentrate assays 65 or 70 percent WO₃, or if a tailing contains 0.06 or 0.08 percent WO₃. However, these differences are worth about 13 Swedish Crowns (\$3.00) per ton of ore and justify a stake in closer process information. Hence, discussions are now in progress with suppliers of systems for on-line analysis. Such a system could possibly be installed in 1980.

In May 1978 a quick method for concentrate assay was introduced at the plant, based on practical experience. As the filtered concentrate contains 4 percent moisture and a certain relationship exists between its scheelite and calcite contents, the latter is determined by titration with weak acid and then the WO₃ content can be read from a graph. A determination takes only 10 minutes and the accuracy is ± 1 percent in the range above 70 percent WO₃ and ± 2 percent in the range 60 to 70 percent WO₃. During the test period, the concentrate grade has been improved without deteriorating the recovery.

Grinding is another topic for further development. As explained earlier, scheelite losses now mainly occur in the ultrafines, minus-9-microns. Independent of the grinding method chosen, recovery in this fraction will be poor, but probably ball mill grinding is completely unsuitable in this case and should be replaced by pebble grinding. However, mill dimensions and pebble acquisition are obstacles. To start with, a full scale test with discpebs 75 millimeters diameter by 15 millimeters long will be conducted in order to diminish the proportion of slimes at the same mesh of grind as hitherto. If this test fails, nevertheless the potential of secondary pebble grinding must be considered anew.

As to fluorite production, this problem is at present put aside because of the current market depression. According to laboratory tests it will be possible to achieve 70 to 80 percent recoveries at concentrate grades of 98 and 95 percent respectively. But considerable investment will be necessary, particularly if fluorite pellets for the steel industry are requested. However, if market conditions change, the company is prepared to install the necessary equipment and begin fluorite production. ■