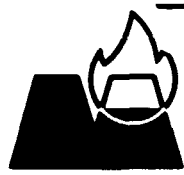


MINTEK

REPORT

No. M103D

THE RECOVERY OF A CONCENTRATE OF TIN AND TUNGSTEN FROM ORE MINED AT VAN ROOIS VLEY

by

R.N. Guest

10th June, 1983
Reissued 29th March, 1985

COUNCIL FOR MINERAL TECHNOLOGY
200 Hans Strijdom Road
RANDBURG
South Africa

MINTEK

(ORE-DRESSING DIVISION)

REPORT
No. M103D

THE RECOVERY OF A CONCENTRATE OF TIN AND TUNGSTEN FROM ORE MINED AT VAN ROOIS VLEY

by

Robert Nelson Guest

10th June, 1983
Reissued 29th March, 1985

Printed and published by the Council for Mineral Technology

ISBN 0 86999 715 7

Investigators R.L. Abela, R.N. Guest, L. Jonker, M.J. Nyai, and J. van Velde
Assistant Director: Applications P.W. Overbeek
Director of Division I.J. Corrans

Programme No. 009
Project No. 126611
Project Report No. 1

Sponsor Shell S.A. (Pty) Ltd Metals Division

All correspondence to Mintek, Private Bag X3015, Randburg, 2125 South Africa

SYNOPSIS

Concentration tests using gravity separation, flotation, and magnetic separation showed that it is possible for the high-grade ore mined at Van Roois Vley to be upgraded to a concentrate containing 61 per cent $Sn-WO_3$ at a recovery of about 80 per cent, and for the low-grade ore to be upgraded to 53 per cent $Sn-WO_3$ at the same recovery.

However, it is suggested that, under normal mining conditions and with large-scale mining equipment, less fines would be produced and this should lead to an increase in grade and recovery.

Mineralogical examinations showed that intermediate fractions produced by tabling contained very few locked particles, and that the grade of the final concentrate on a full-scale plant would probably be higher than that obtained with laboratory equipment.

Molybdenite could be a valuable byproduct, but flotation tests of gravity concentrates for selective recovery of the mineral gave variable results. Flotation tests for the removal of fluorspar from the final concentrate were unsuccessful.

SAMEVATTING

Konsentrasietoetse met gebruik van swaartekragskeiding, flottasie en magnetiese skeiding het getoon dat dit moontlik is om die hoëgraadse erts wat by Van Roois Vley ontgin word, op te gradeer tot 'n konsentraat wat 61 persent $Sn-WO_3$ bevat met 'n herwinning van ongeveer 80 persent, en om die laegraadse erts op te gradeer tot 53 persent $Sn-WO_3$ met dieselfde herwinning.

Daar word egter aan die hand gedoen dat daar normale mynboutoestande en met grootskaalse mynboutoerusting minder fynmateriaal voortgebring sal word en dit behoort tot 'n verhoging van die graad en herwinning te lei.

Mineralogiese ondersoeke het getoon dat tussenfraksies wat deur vlaktafelkonsentrasie verkry is, baie min geslote partikels bevat het en dat die graad van die eindkonsentraat in 'n volskaalse aanleg waarskynlik hoër sal wees as dié wat met laboratoriumtoerusting verkry word.

Molibdeniet kan 'n waardevolle neweproduk wees, maar flottasietoetse met swaartekragkonsentrate om die mineraal selektief te herwin, het veranderlike resultate opgelewer. Flottasietoetse om vloeispaat uit die finale konsentraat te verwyder was nie geslaagd nie.

CONTENTS

1. INTRODUCTION 1

2. EXPERIMENTAL METHOD 1
   2.1. PREPARATION OF SAMPLES 1
   2.2. PRIMARY CONCENTRATION OF COARSE MATERIAL 3
   2.3. CONCENTRATION OF FINES 3
   2.4. RETREATMENT OF JIG AND TABLE PRODUCTS 3
   2.5. MAGNETIC SEPARATION 3
   2.6. FLOTATION 3
   2.7. OVERALL FLOW DIAGRAM 3
   2.8. REMOVAL OF MOLYBDENITE AND FLUORSPAR 3

3. RESULTS 3
   3.1. PRIMARY CONCENTRATION 6
      3.1.1. Heavy-medium Separation 6
      3.1.2. Jigging 6
   3.2. HIGH-GRADE SAMPLE (J175) 8
      3.2.1. Treatment of Coarse Material 8
      3.2.2. Treatment of Natural Fines 8
      3.2.3. Treatment of Fines from HMS 8
      3.2.4. Treatment of Fines from Secondary Jig Tailing 8
      3.2.5. Retabling of Middling Fractions 8
      3.2.6. Upgrading of Final Concentrates 8
      3.2.7. Summary of Results 8
   3.3. LOW-GRADE ORE (J176) 8
      3.3.1. Treatment of Coarse Material 8
      3.3.2. Treatment of Natural Fines 16
      3.3.3. Treatment of Secondary Fines 16
      3.3.4. Retabling of Middiing Fractions 16
      3.3.5. Upgrading of Final Concentrates 16
      3.3.6. Summary of Results 16
   3.4. MINERALOGICAL EXAMINATION 16
   3.5. ANALYTICAL RESULTS 16
   3.6. REMOVAL OF MOLYBDENITE AND FLUORSPAR 16

4. DISCUSSION AND CONCLUSIONS 21

LIST OF TABLES

Table 1. The main minerals of highest density found in the ore 22
Table 2. Operating conditions for HMS tests 22
Table 3. The results of the size split during comminution 22
Table 4. Size assay on the 6 to 1 mm fraction of Sample J175 23
Table 5. Size analysis of material finer than 6 mm from Sample J176 23
Table 6. Results of exploratory tests with the DWP separator on the concentrates or sink fractions from Sample J175 23
Table 7. Results of exploratory HMS tests on Sample J175 using the DSM cyclone 24
Table 8. Results of the final HMS cyclone runs on the bulk sample J175 24
Table 9. Results of HMS cyclone test on coarse material from Sample J176 24
Table 10. Primary jigging results for Sample J175 25
Table 11. Heavy-liquid separation test on primary jig tailing from Sample J175 25
Table 12. The size distribution, determined by Microtrac, in the cyclone overflow of the hydrosizer slimes from natural fines 25
Table 13. The results of cyclosizing the hydrosizer overflow from Sample J175 26

Table 14. Size assay on the overflow from the slimes cyclone from Sample J175 26
Table 15. Final concentrates from Sample J175 26
Table 16. Pyrite-flotation products from Sample J175 27
Table 17. Total magnetic fraction from Sample J175 27
Table 18. Final tailings from Sample J175 28
Table 19. Final results for Sample J175 28
Table 20. Results of semi-quantitative analysis on a sample of final concentrate from Sample J175 29
Table 21. Final concentrates from Sample J176 29
Table 22. Pyrite-flotation products from Sample J176 30
Table 23. Total magnetic fraction of Sample J176 30
Table 24. Final tailings from Sample J176 31
Table 25. Final results for Sample J176 32
Table 26. Mineralogical composition of table concentrates 32
Table 27. Results of a flotation test on the recovery of molybdenite from the jig concentrate from the HMS fines circuit 33
Table 28. Results of a flotation test on the recovery of molybdenite from the table concentration of the combined primary fine table concentrates 33
Table 29. Results of a flotation test on the recovery of molybdenite from the table concentrate obtained by the retabling of the combined second concentrates from the primary tables 34
Table 30. Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (1) 35
Table 31. Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (2) 36
Table 32. Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (3) 37
Table 33. Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (4) 38

LIST OF ILLUSTRATIONS

Figure 1. Comminution procedure 1
Figure 2. The method of laying down the ore for mixing 2
Figure 3. The method of removing the ore in a direction perpendicular to the direction of placement 2
Figure 4. Primary concentration of coarse material 4
Figure 5. Treatment of fines 5
Figure 6. Procedure for the flotation of pyrite 6
Figure 7. Flow diagram of the tests for Van Roois Vlcy feasibility study 7
Figure 8. Treatment of coarse particles from Sample J175 9
Figure 9. Treatment of natural fines from Sample J175 10
Figure 10. Treatment of fines from HMS concentrate from Sample J175 11
Figure 11. Treatment of fines from secondary jig tailing from Sample J175 12
Figure 12. Retreatment of intermediate products from Sample J175 13
Figure 13. Final upgrading of concentrates from Sample J175 by flotation and magnetic separation 14
Figure 14. Treatment of coarse particles from Sample J176 15
Figure 15. Treatment of natural fines from Sample J176 17
Figure 16. Treatment of secondary fines from Sample J176 18
Figure 17. Retreatment of intermediate products from Sample J176 19
Figure 18. Final upgrading of concentrates from Sample J176 by flotation and magnetic separation 20

1. INTRODUCTION

Mr J. Trouw of Shell S.A. (Pty) Ltd, Metals Division, asked the Council for Mineral Technology (Mintek) to produce data that would complete the feasibility study on Shell S.A.'s deposit of tin and tungsten on the farm Van Roois Vley, about 45 km west of Upington in the northern Cape Province. The deposit was mined between 1945 and 1958, and mainly oxidized ore was treated. Since the mining operations ceased, various mining companies have investigated the deposit.

Two samples were received at Mintek: J175, a 10 t sample of high-grade ore (0,56 per cent tungsten trioxide and 0,26 per cent tin), and J176, a 5 t sample of low-grade ore (0,20 per cent tungsten trioxide and 0,15 per cent tin). All the analyses were carried out by a commercial laboratory, and Mintek's Analytical Division merely carried out check assays on a range of samples.

Table 1 gives a list of the main minerals of highest density found in the ore. Other minor minerals that were found in the ore are ilmenite, chalcopyrite, garnet, and molybdenite. Of the heavy minerals present, fluorite and scheelite have the lowest Mohs' scale of hardness number and are likely to slime, whereas cassiterite and wolframite are harder and brittle and would probably produce some fines on being comminuted.

An examination of the relative densities of the minerals reveals that, if pyrite is used as a marker in the gravity-separation stage, it should be possible for all the silicates, such as tourmaline, to be eliminated from the concentrates. A concentrate having no silicates could be upgraded by magnetic separation for the removal of the magnetite and by flotation for the removal of the sulphides.

2. EXPERIMENTAL METHOD

2.1. Preparation of Samples

The ore was tested for primary concentration at two sizes, viz between 9 and 1 mm, and between 6 and 1 mm. It was necessary that as little material finer than 1 mm as possible should be created, and the comminution procedure adopted is shown in Figure 1.

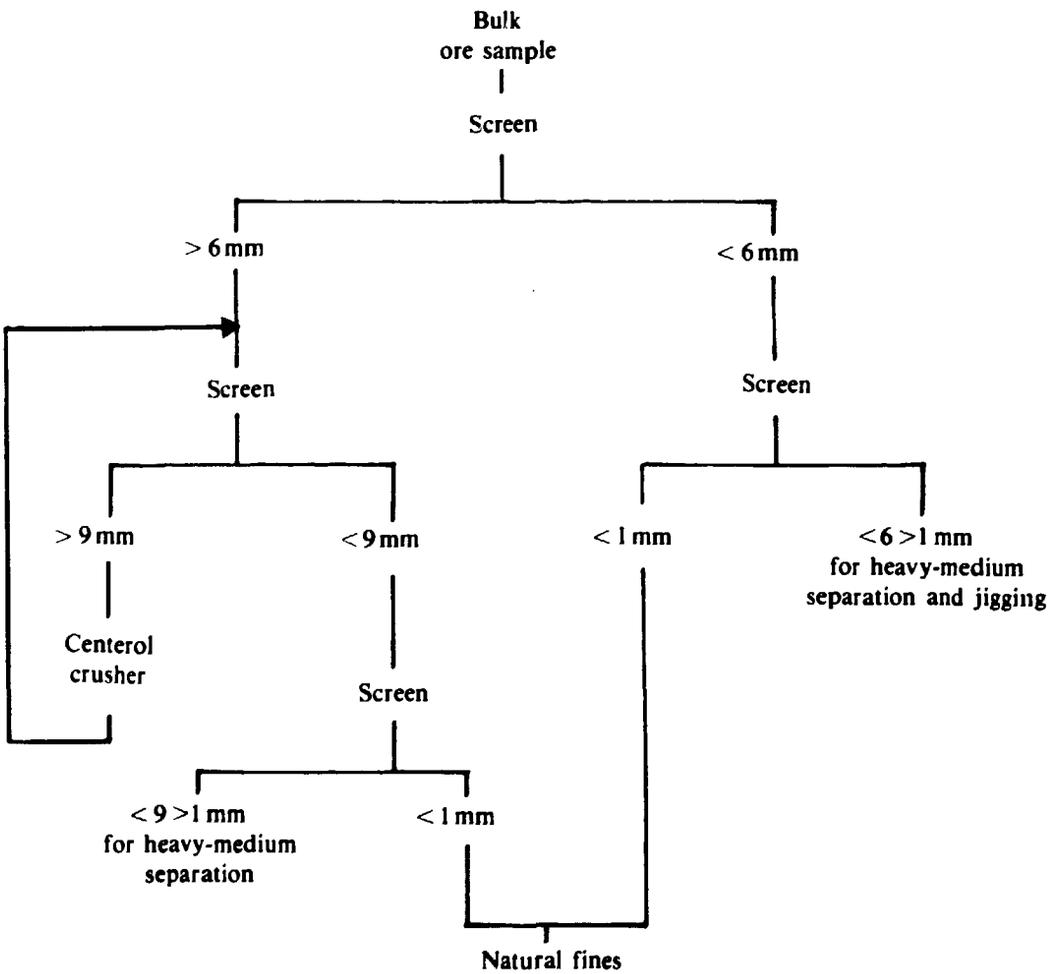


FIGURE 1. Comminution procedure

So that the bulk samples would be thoroughly mixed and representative head samples would be obtained, the ore was mixed as follows. The ore was laid in thin layers in one direction and then removed in a direction perpendicular to the direction of placement (Figures 2 and 3).

FIGURE 2. The method of laying down the ore for mixing

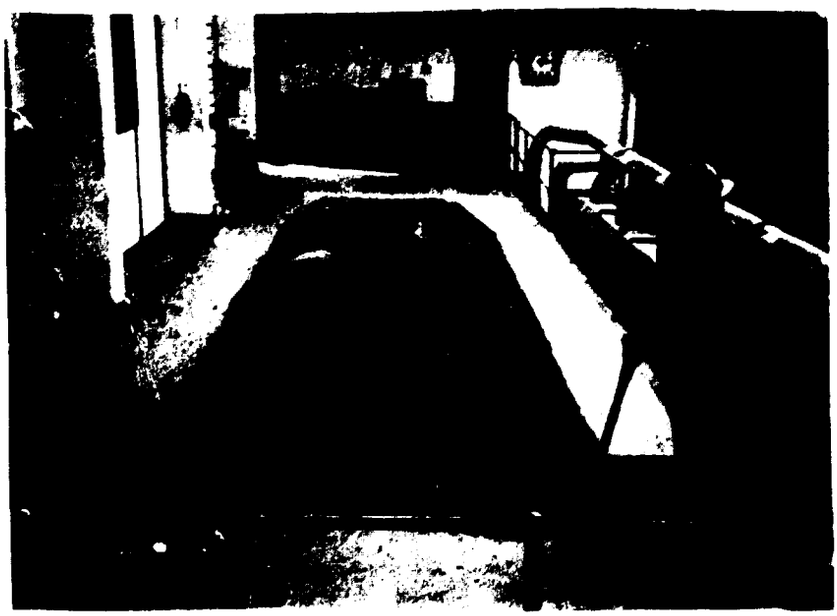


FIGURE 3. The method of removing the ore in a direction perpendicular to the direction of placement

2.2. Primary Concentration of Coarse Material

Jigging and heavy-medium separation (HMS) were the two methods of primary coarse concentration carried out.

The jigging was done, with a small Rema jig having four compartments, on the fraction of the material between 6 and 1 mm. HMS, by an HMS cyclone and a Dyna Whirlpool separator (DWP), was carried out on material from the fractions between 9 and 1 mm and between 6 and 1 mm. Nortons-Tividale SA (Pty) Ltd conducted exploratory work on the DWP, and the results are reported here.

The cyclone used for the HMS had a diameter of 150 mm and was of the Dutch State Mines (DSM) type, the heavy medium used being 150D ferrosilicon. The relative densities of the various media and pressures for the two coarse materials treated are given in Table 2.

The bulk of the coarse high-grade material (J175) was treated by HMS in the cyclone plant. The concentrate from this material and the concentrate from the jig were combined for secondary treatment (Figure 4), whereas the coarse low-grade material (J176) was concentrated by HMS only. The primary concentrate was reduced to material finer than 3 mm, and the fraction between 3 and 1 mm was jigged to give a final concentrate, a pyrite-rich concentrate, and a tailing. (No pyrite concentrate was taken from the low-grade ore, J176.) Magnetic separation and flotation were used to upgrade the pyrite concentrate, and the secondary jig tailing was ground to material smaller than 1 mm.

2.3. Concentration of Fines

The three products finer than 1 mm (natural fines, fines from the primary HMS, and the fine secondary jig tailing) were all treated as shown in Figure 5. The product from spigot 1 of the hydrosizer was concentrated on a spiral, and the spiral concentrate was upgraded by a fines jig. Both the products from spigots 2 and 3 were treated by spiral, and the spiral concentrate was upgraded by tabling. The overflow from the hydrosizer was tabled after desliming. When the fine material generated in the coarse circuit was classified in the hydrosizer, the product from spigot 3 was added to the overflow from the hydrosizer.

When the low-grade ore was treated, the fines from the primary HMS and those from the secondary jig tailing were mixed, and were then treated together.

2.4. Retreatment of Jig and Table Products

The table middlings originating from the jigging operations on the fines were combined and retabled, and all the other table middlings were also combined for tabling. The concentrates from these retabling steps and all the other coarse table concentrates were combined and retabled to give a final gravity concentrate. All the concentrates from the slimes tabling were combined and cleaned by tabling to produce a final gravity concentrate. The middlings from the tabling of the fines jig tailing were combined and retabled, and the fines jig concentrates were upgraded by rejigging. All the final gravity concentrates (excluding the high-grade coarse jig concentrates) were upgraded by magnetic separation and flotation.

2.5. Magnetic Separation

Two separators were used for this concentration: a low-intensity belt magnetic separator and a Permroll separator supplied by Edward L. Bateman Ltd.

2.6. Flotation

The gravity concentrate was ground finer than 150 μm before it was floated in a Denver laboratory flotation cell as shown in Figure 6.

2.7. Overall Flow Diagram

A condensed flow diagram of the testwork is shown in Figure 7.

2.8. Removal of Molybdenite and Fluorspar

The ore contains fluorspar, which can form hydrofluoric acid during the hydrometallurgical step that is used for the separation of the tin and tungsten concentrates, and there is a minor amount of molybdenite present, which could become a valuable byproduct. Flotation tests were therefore carried out on the removal of these two constituents from various concentrates.

3. RESULTS

The results of the size split during the comminution of both the high-grade and the low-grade ore are given in Table 3.

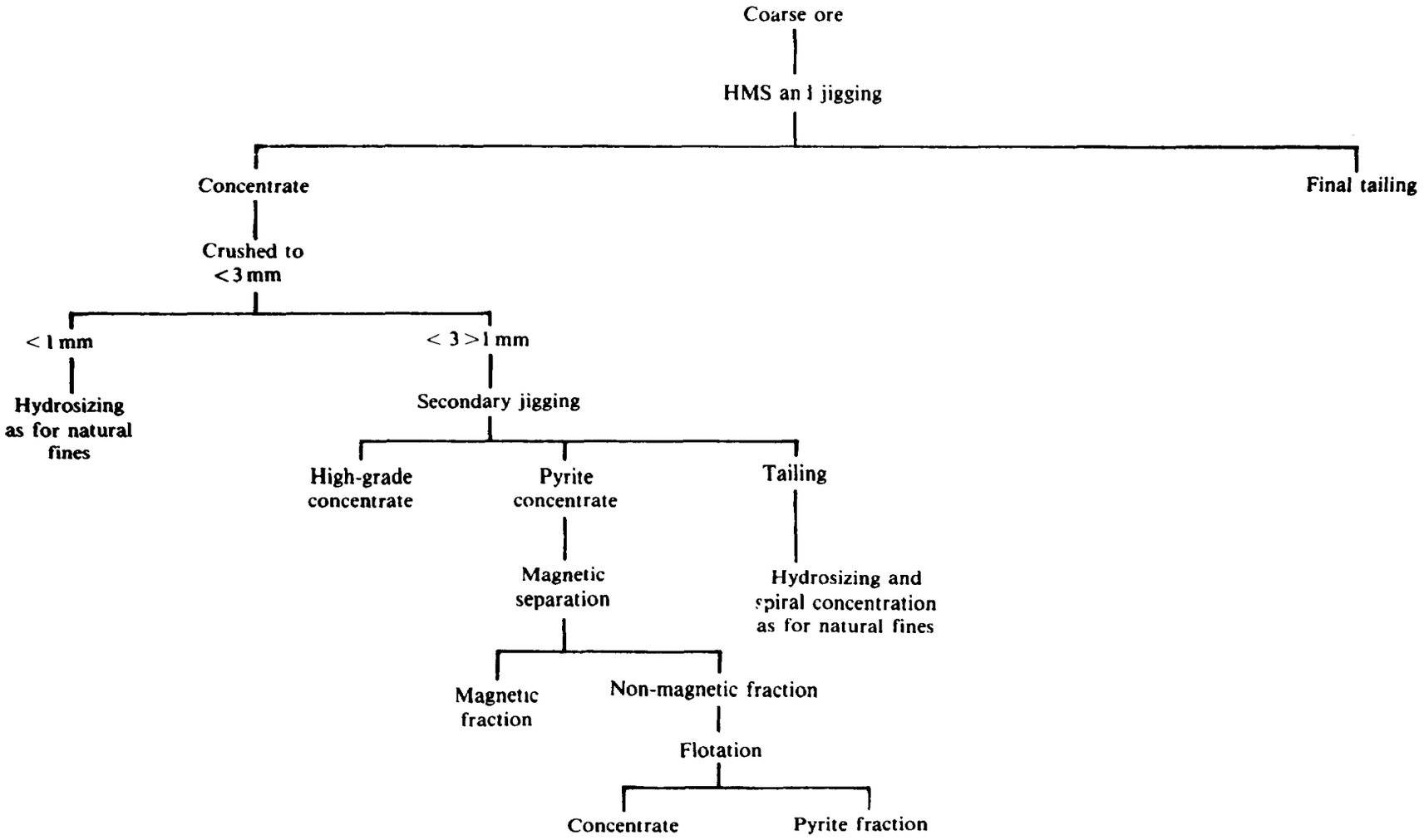
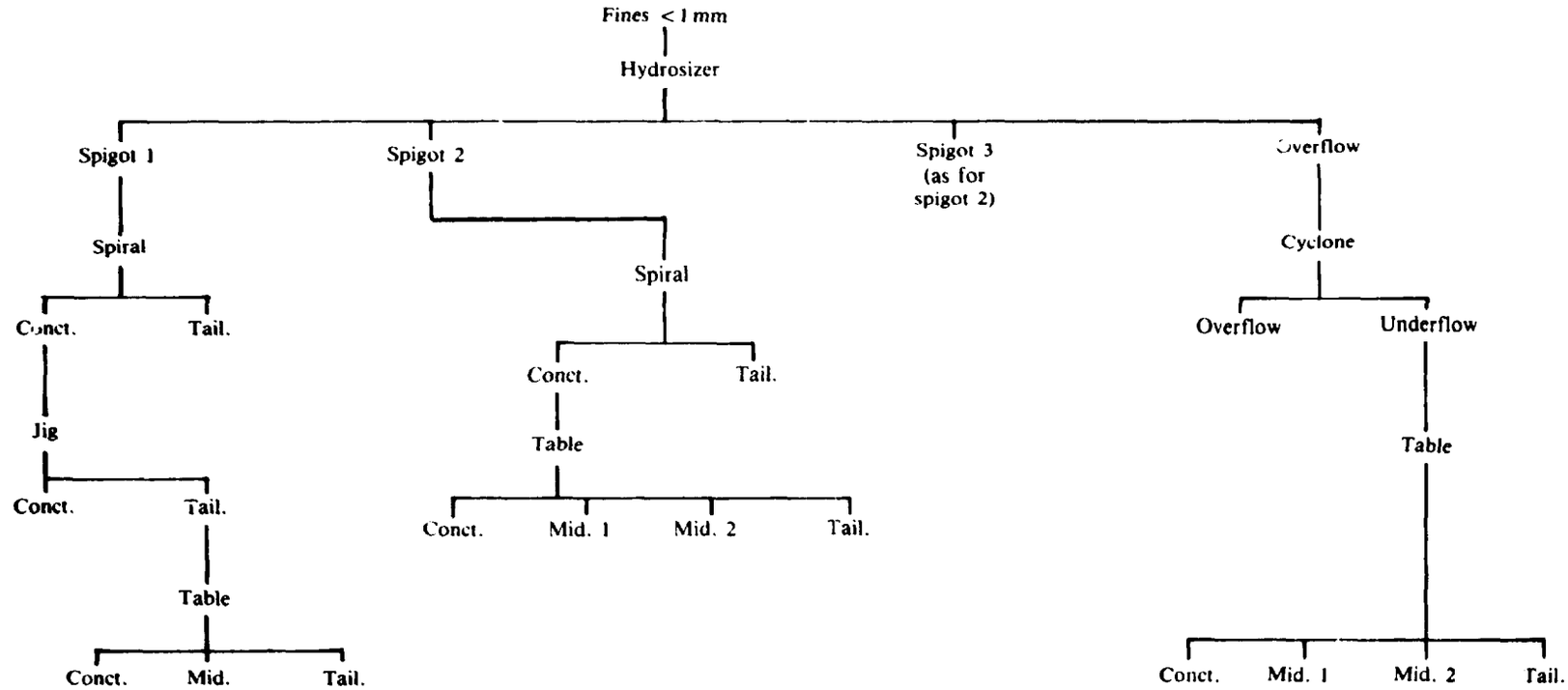


FIGURE 4. Primary concentration of coarse material



Conct. = concentrate
Mid. = middling
Tail. = tailing

FIGURE 5. Treatment of fines

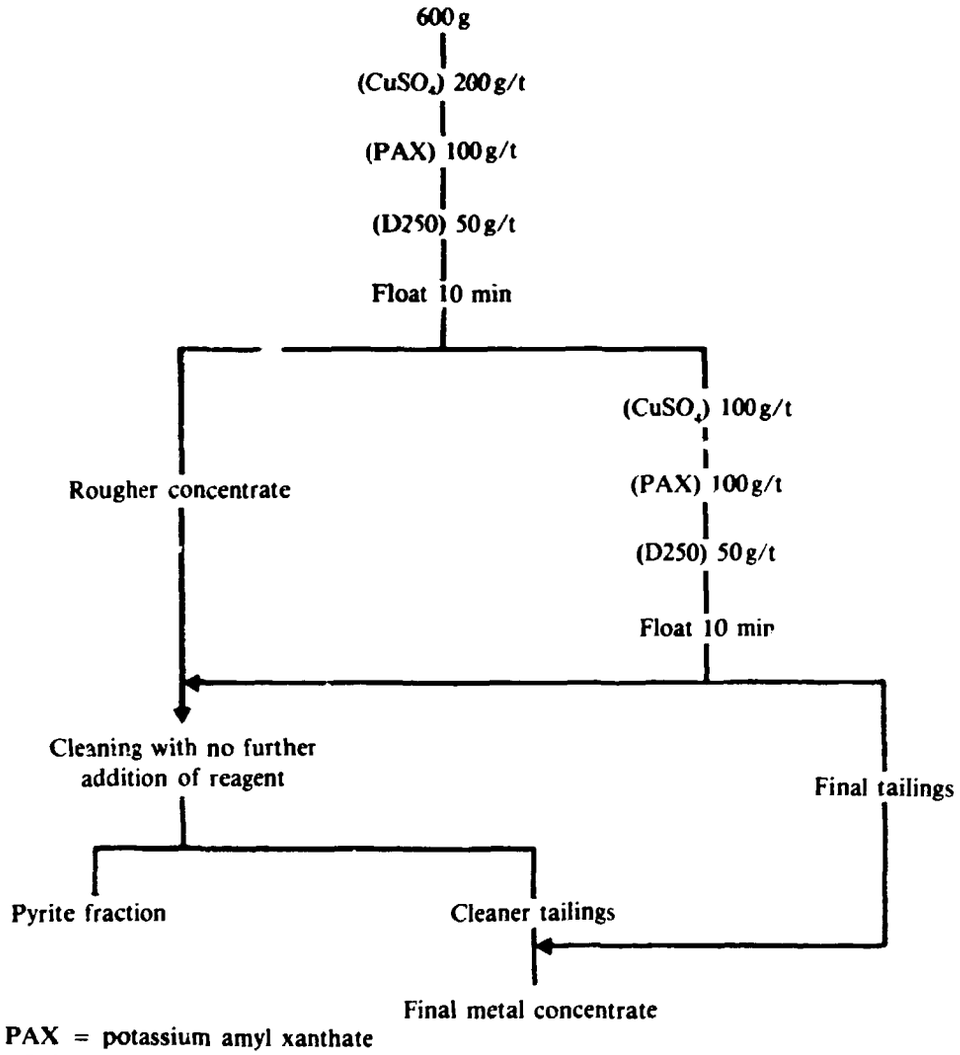


FIGURE 6. Procedure for the flotation of pyrite

3.1. Primary Concentration

The material finer than 6 mm was sized and the individual size fractions were assayed (Tables 4 and 5). There was a slight upgrading in the material coarser than 1 mm from the high-grade ore, but no major upgrading in the low-grade ore.

3.1.1. Heavy-medium Separation

The exploratory HMS tests carried out on a DWP separato. by Nortons-Tividale SA (Pty) Ltd are reported in Table 6. The results of the HMS tests carried out in a DSM cyclone (Table 7) show that, in general, the cyclone recovered more values at the same mass split than did the DWP. The cyclone was therefore used to treat the bulk sample, and the results are reported in Tables 8 and 9. Between 94 and 97 per cent of the values were recovered in this primary HMS step from the high-grade material, and about 93 per cent from the low-grade material.

3.1.2. Jigging

As a comparison of jigging with HMS, numerous preliminary jigging tests were carried out, followed by a final run on approximately 1 t of material between 6 and 1 mm. The results (Table 10) show that jigging gave recoveries and grades of the same order as those obtained by HMS. The material locked in the bed should be taken into account and included in the final concentrate. If this is done, recoveries of

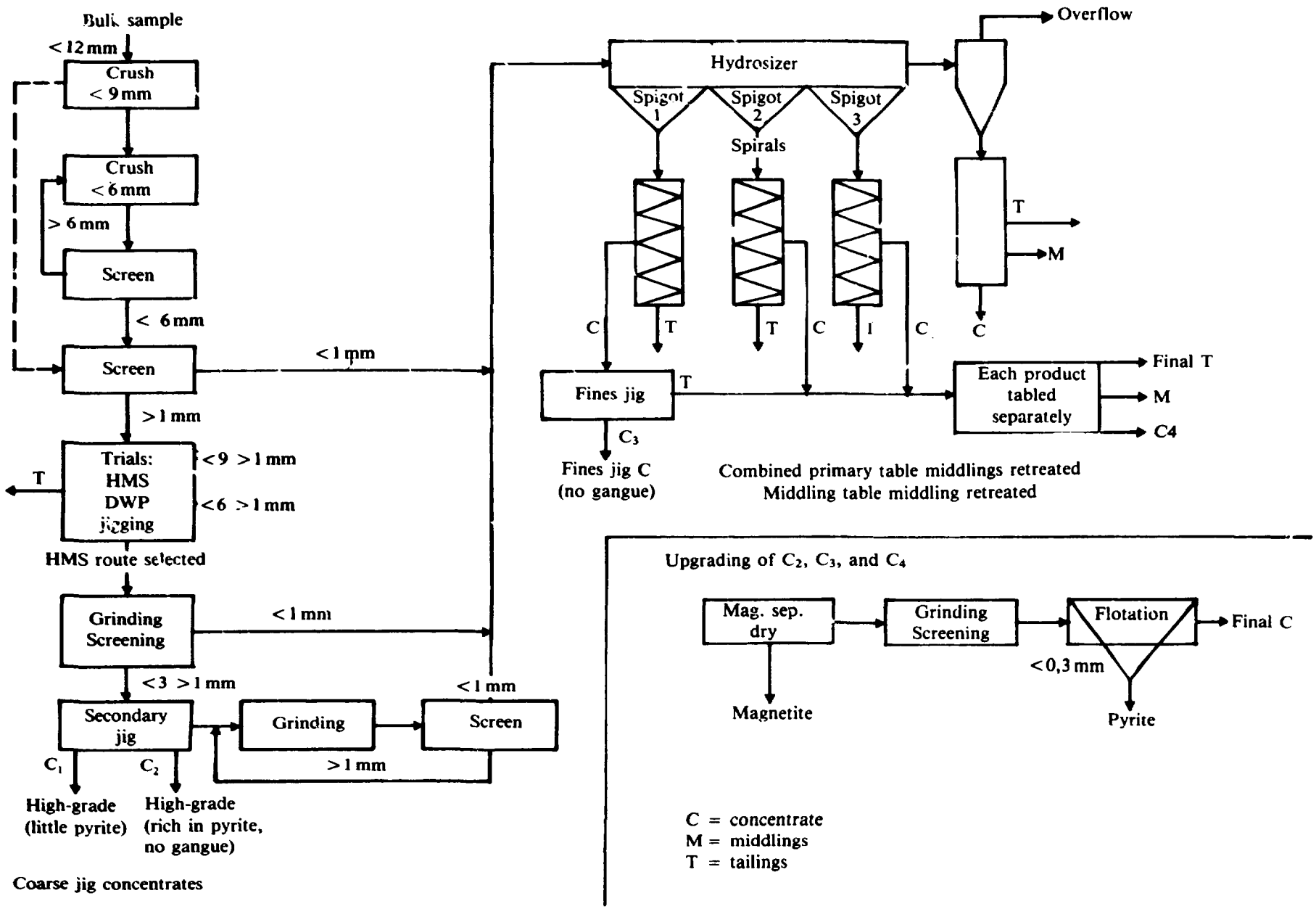


FIGURE 7. Flow diagram of the tests for Van Roois Vley feasibility study

96,2 per cent of the tungsten and 92 per cent of the tin would be obtained in a total mass of 12,5 per cent. As a test of the efficiency of the primary jigging stage, a sample of the tailing was separated by heavy liquids (Table 11), and this showed that over 90 per cent of the valuable mineral was present in ore having a relative density of less than 3,3. This indicates that most of the mineral lost in the primary jigging was probably in the form of small, unliberated chats in gangue material.

3.2. High-grade Sample (J175)

3.2.1. Treatment of Coarse Material

The treatment of the coarse material is depicted in Figure 8. A final low-grade tailing of about 60 per cent of the mass was rejected by HMS.

3.2.2. Treatment of Natural Fines

The material from the crushing stage finer than 1 mm was termed natural fines, and was sized in a hydrosizer and then concentrated by means of spirals, jigs, and tables (Figure 9). A determination of the size distribution was carried out on the cyclone overflow material by use of the Microtrac (Table 12), and over 90 per cent was found to be finer than 20 μm (quartz diameter).

The product from the hydrosizer overflow in the test on natural fines was sized in a cyclosizer, and the individual fractions were analysed. The results (Table 13) show a natural upgrading of tungsten trioxide in the slimes fraction due to the breakdown of scheelite. Conversely, the opposite occurs with the tin assay.

The results of the size assay on the cyclone overflow (Table 14) show a similar trend of upgrading of scheelite and downgrading of cassiterite in the slimes.

3.2.3. Treatment of Fines from HMS

The concentrate from the HMS stage was crushed, and the material finer than 1 mm was screened off (Figure 8) and separated by hydrosizer, spirals, jigs, and tables (Figure 10).

3.2.4. Treatment of Fines from Secondary Jig Tailing

The tailing from the secondary jigging step (Figure 8) was crushed to finer than 1 mm and separated by hydrosizer, spirals, jigs, and tables (Figure 11).

3.2.5. Retabling of Middling Fractions

The various fractions of low-grade concentrate and middlings that were produced (Figures 8 to 11) were combined according to particle size or grade and were rejigged or retabled (Figure 12).

3.2.6. Upgrading of Final Concentrates

The gravity concentrates that were produced contained magnetite and sulphides (the exception being the concentrate from the high-grade secondary jig), and these contaminants were removed by magnetic separation and flotation. The results are given in Figure 13.

3.2.7. Summary of Results

The results for all the products are grouped into individual sections, viz all the final concentrates, all the pyrite-flotation products, all the magnetic fractions, and the final tailings (Tables 15 to 18).

The grade of the pyrite or sulphide concentrates varies considerably, but, when a single flotation circuit comes into operation, it is expected that the losses in this product will be reduced slightly.

Generally, the coarser concentrates were of a higher total-metal grade (i.e. from jigging), and the products with the lowest grade originated from the concentrates obtained on the slimes table.

The combination of all the concentrates into one final fraction and all the other products into one final tailing (Table 19) resulted in the recovery of over 80 per cent of the valuable minerals in a concentrate of 61,3 per cent combined tin and tungsten (by calculation). A sample of this concentrate when analysed (Table 20) was found to have a combined tin and tungsten grade of 65,6 per cent. Semi-quantitative results for 16 other elements are given in Table 20.

3.3. Low-grade Ore (J176)

3.3.1. Treatment of Coarse Material

The treatment of the coarse material is shown in Figure 14. A final low-grade tailing of 56 per cent by mass was rejected by HMS.

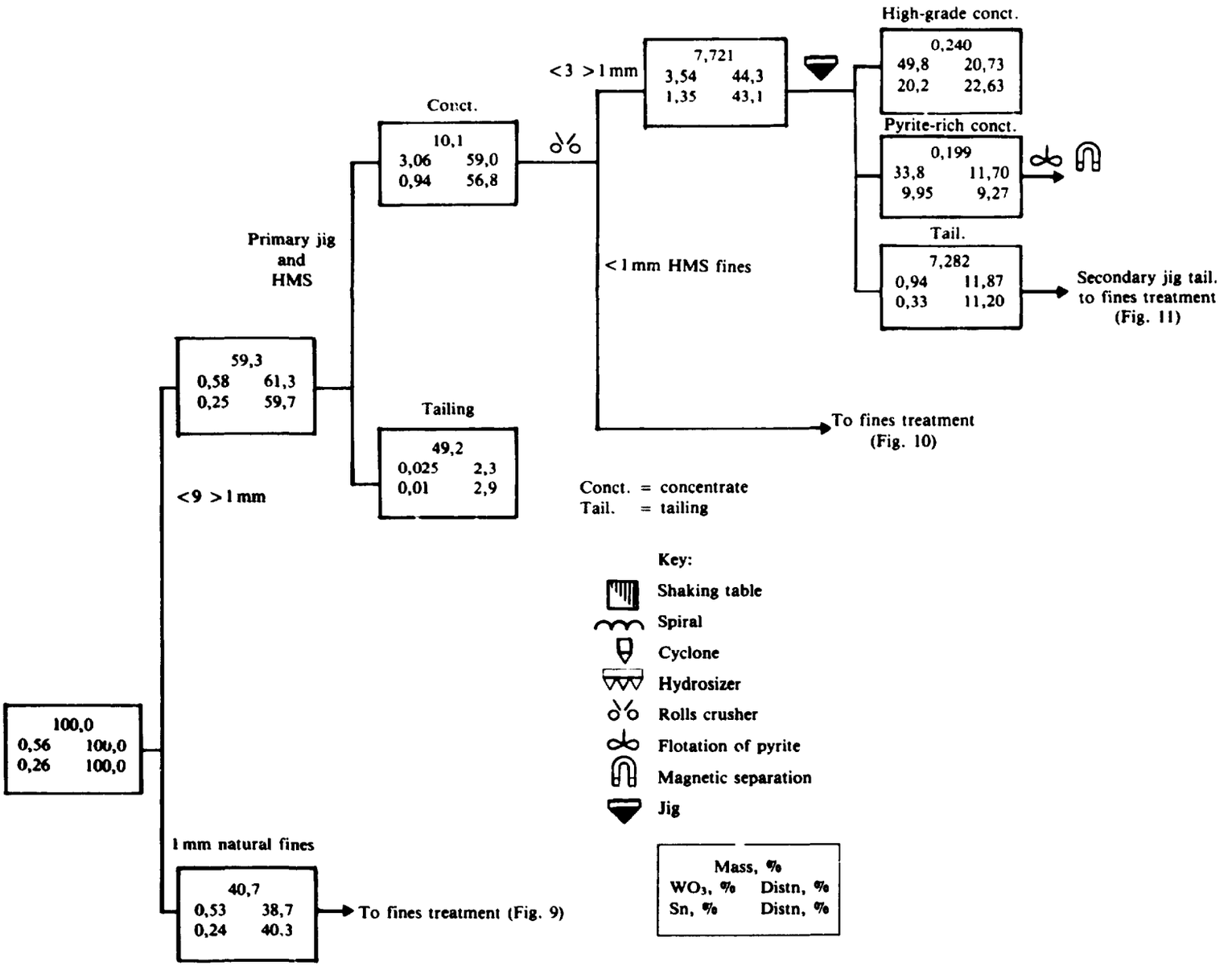


FIGURE 8. Treatment of coarse particles from Sample J175

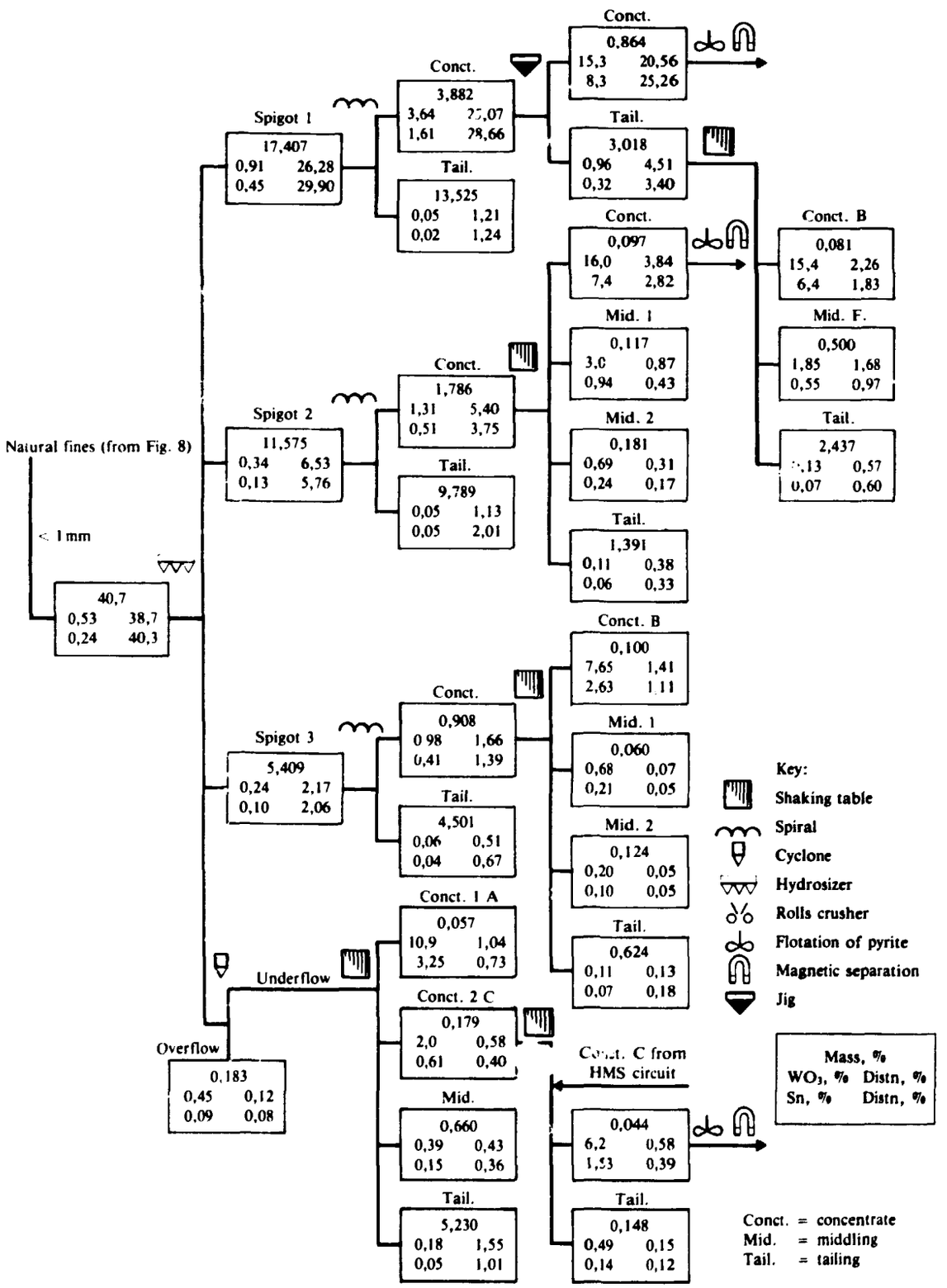


FIGURE 9. Treatment of natural fines from Sample J175

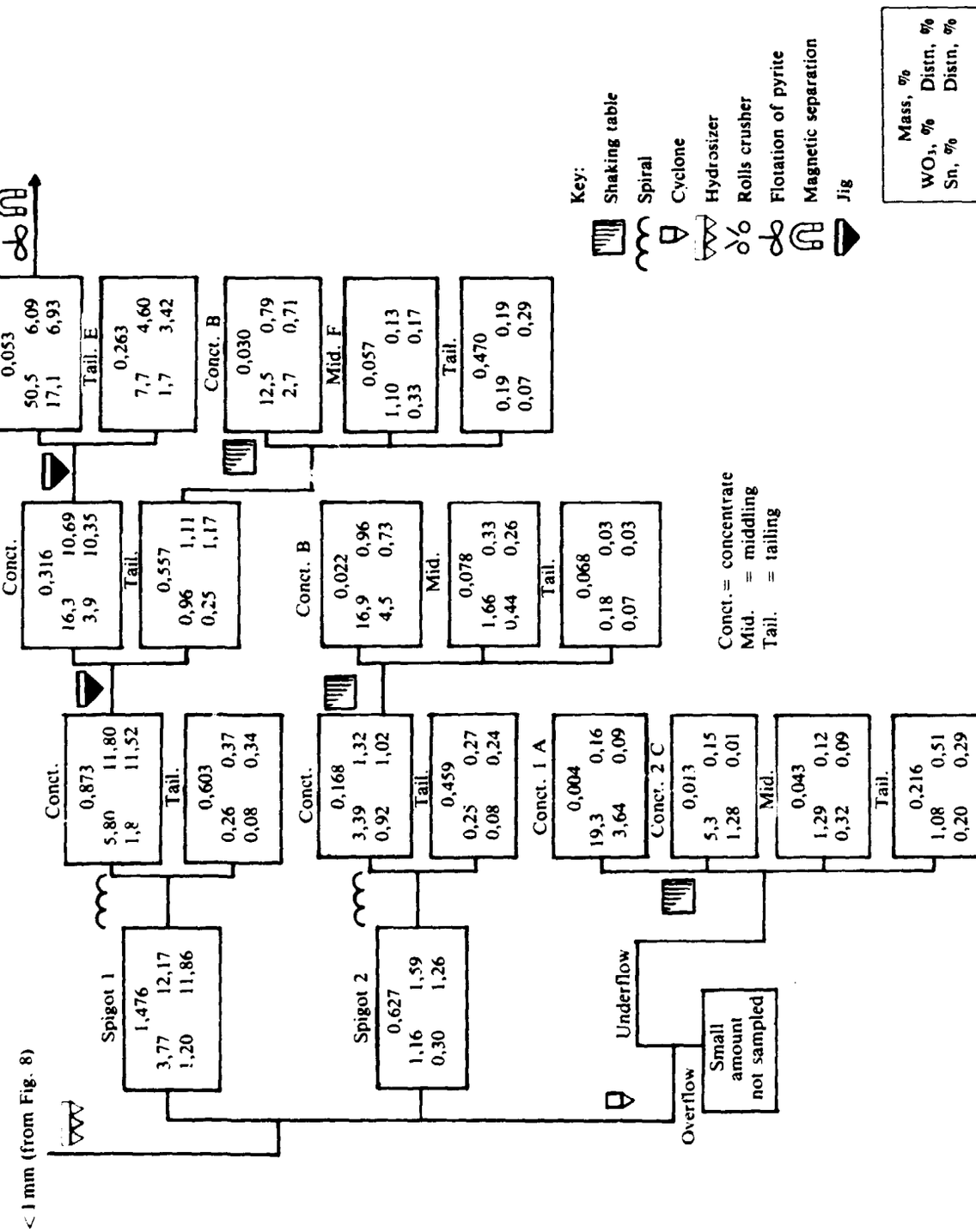


FIGURE 10. Treatment of fines from the HMS concentrate from Sample J175

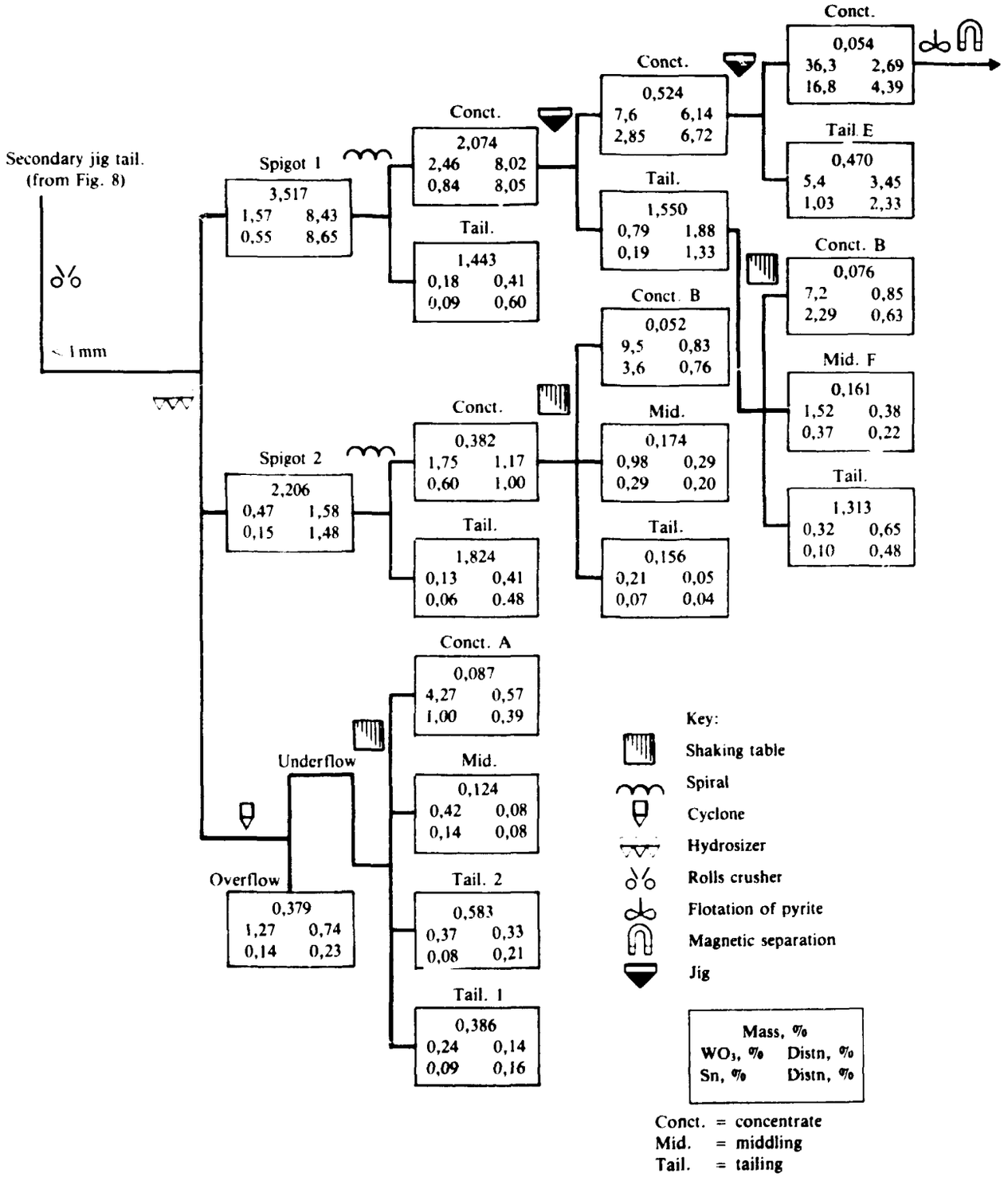


FIGURE 11. Treatment of fines from secondary jig tailing from Sample J175

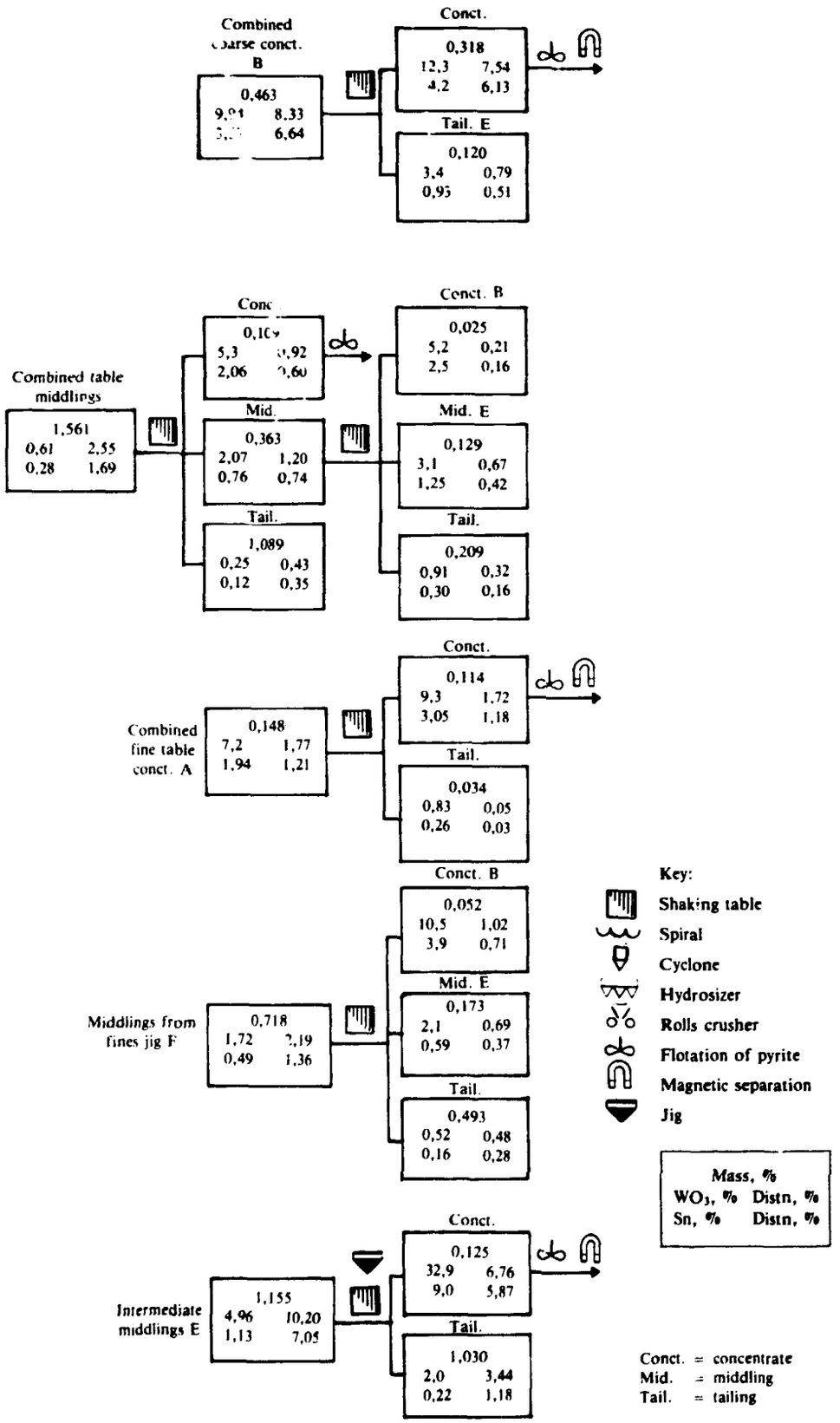


FIGURE 12. Retreatment of intermediate products from Sample J175

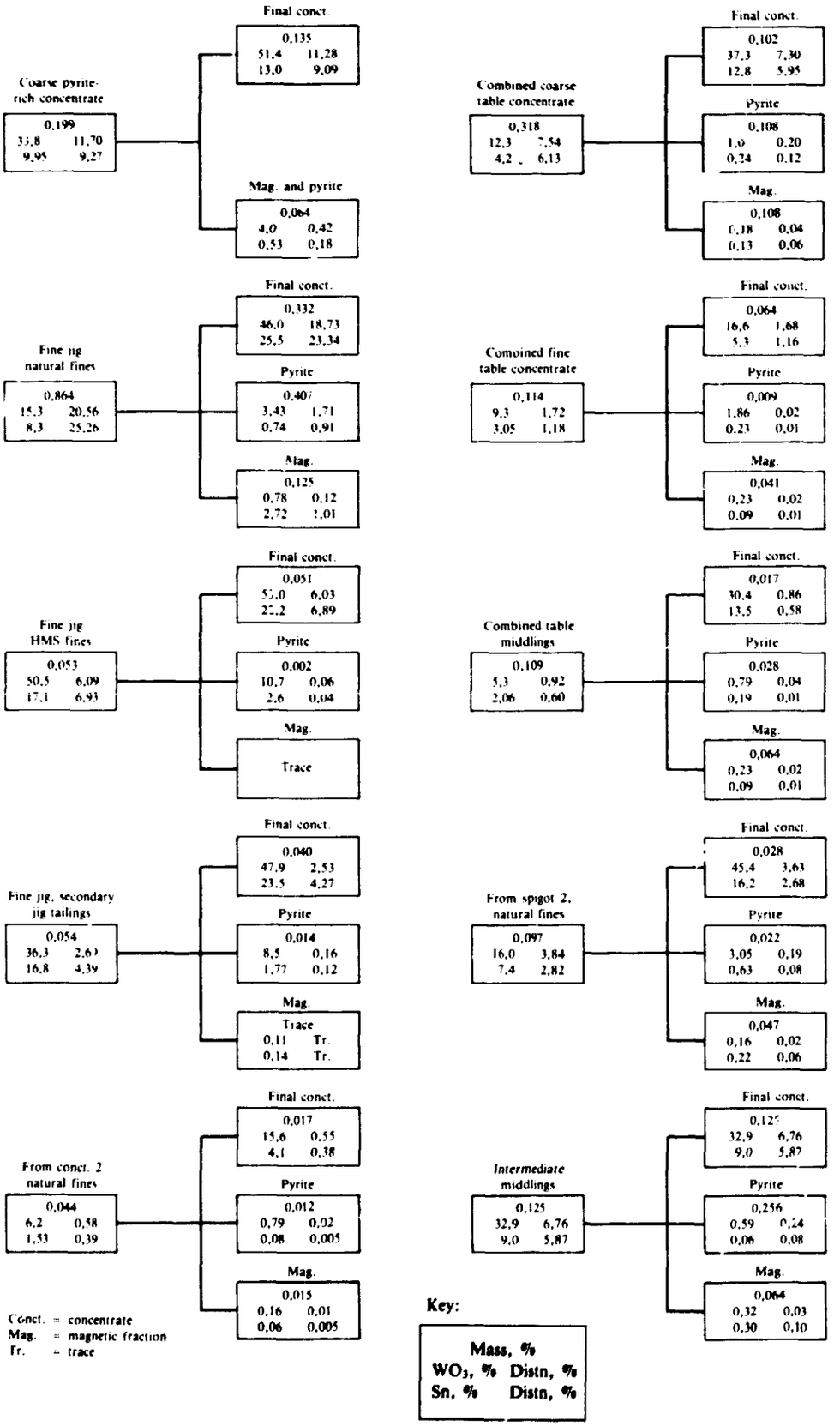


FIGURE 13. Final upgrading of concentrates from Sample J175 by flotation and magnetic separation

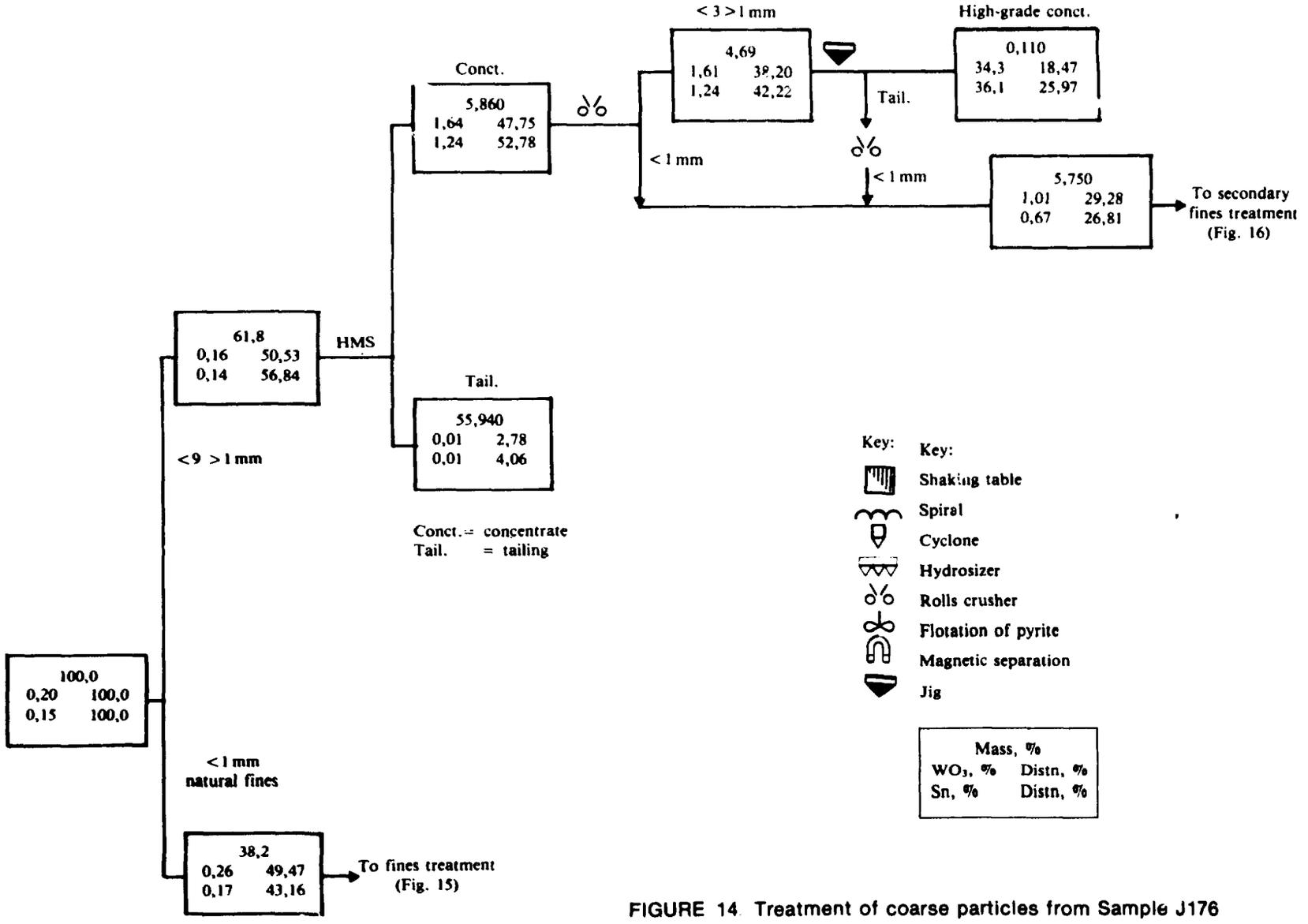


FIGURE 14 Treatment of coarse particles from Sample J176

3.3.2. Treatment of Natural Fines

The material from the crushing stage fine than 1 mm (natural fines) was treated by hydrosizer, spirals, jigs, and tables, and the results are given in Figure 15.

3.3.3. Treatment of Secondary Fines

The material finer than 1 mm that was produced from the primary HMS concentrate was treated by hydrosizer, spirals, jigs, and tables, and the results are given in Figure 16.

3.3.4. Retabling of Middling Fractions

The various middling and low-grade fractions were combined and treated in the same way as for the high-grade ore, and the results are given in Figure 17.

3.3.5. Upgrading of Final Concentrates

The gravity concentrates produced were upgraded by flotation and magnetic separation, and the results are shown in Figure 18.

3.3.6. Summary of Results

Again, the results for all the products have been grouped together in sections and are given in Tables 21 to 24. The results for the two final products combined from all the products are given in Table 25, which shows that the recoveries were similar to those obtained for the high-grade ore, viz 80 per cent, but the combined grade of tin and tungsten was lower, at 53,2 per cent.

3.4. Mineralogical Examination

Certain of the finer concentrates were of relatively low grade, and their addition to the total concentrate caused the final grade to be reduced, especially in the case of the low-grade ore.

A mineralogical investigation was carried out on nine table concentrates in an assessment of whether they could be upgraded further by gravity separation or whether locked particles would interfere with further upgrading. Particles consisting of intergrowths of scheelite and wolframite were not included in the count. The major impurities were found to be free grains of magnetite, pyrite, tourmaline, and garnet, with minor amounts of chlorite, quartz, and sericite.

Table 26 gives the results of the investigation. The amounts of the various minerals present and of silicate gangue locked on ore particles were calculated from the analysis of the concentrates, the amount of pyrite and magnetite removed in the upgrading process, and the mineralogical estimation. The results show that the proportion of valuable ore minerals that are locked on silicate gangue is small, and that it should be possible for these concentrates to be upgraded further by removal of the free gangue on a shaking table.

The reason for the low grade of the concentrates is probably that it is difficult to separate a clean concentrate from the thin band of concentrate that is formed on a laboratory shaking table. It is expected that the band would be spread over a larger area on a full-size table, making it possible for a concentrate that is fairly free of silicates to be separated. Any magnetite or sulphides would be removed by the subsequent upgrading techniques.

3.5. Analytical Results

During the investigation, various standard samples were analysed by four laboratories, and it was found that the results of three of the laboratories were in fairly close agreement with one another. However, the Mintek laboratory reported the low tailing values to be higher than those reported by the other laboratories.

The analysis of the tailings is obviously not straightforward, and it is recommended that accurate international standards should be tested by the laboratories concerned as confirmation of their results.

3.6. Removal of Molybdenite and Fluorspar

The high-grade head sample was found to contain about 100 p.p.m. of molybdenum and 1,23 per cent sulphur. The low-grade contained about 30 per cent molybdenum and 0,12 per cent sulphur. Various table concentrates that were analysed for molybdenum were found to contain up to 0,2 per cent.

Initially, an attempt was made to remove the molybdenite by selective flotation before the other sulphides were floated. Tables 27 to 29 show the results of these tests on three fine concentrates originating from fines jigs and tables. The grade and recovery of molybdenite varied from 3,75 to 25,6 per cent and from 10,5 to 69,3 per cent respectively.

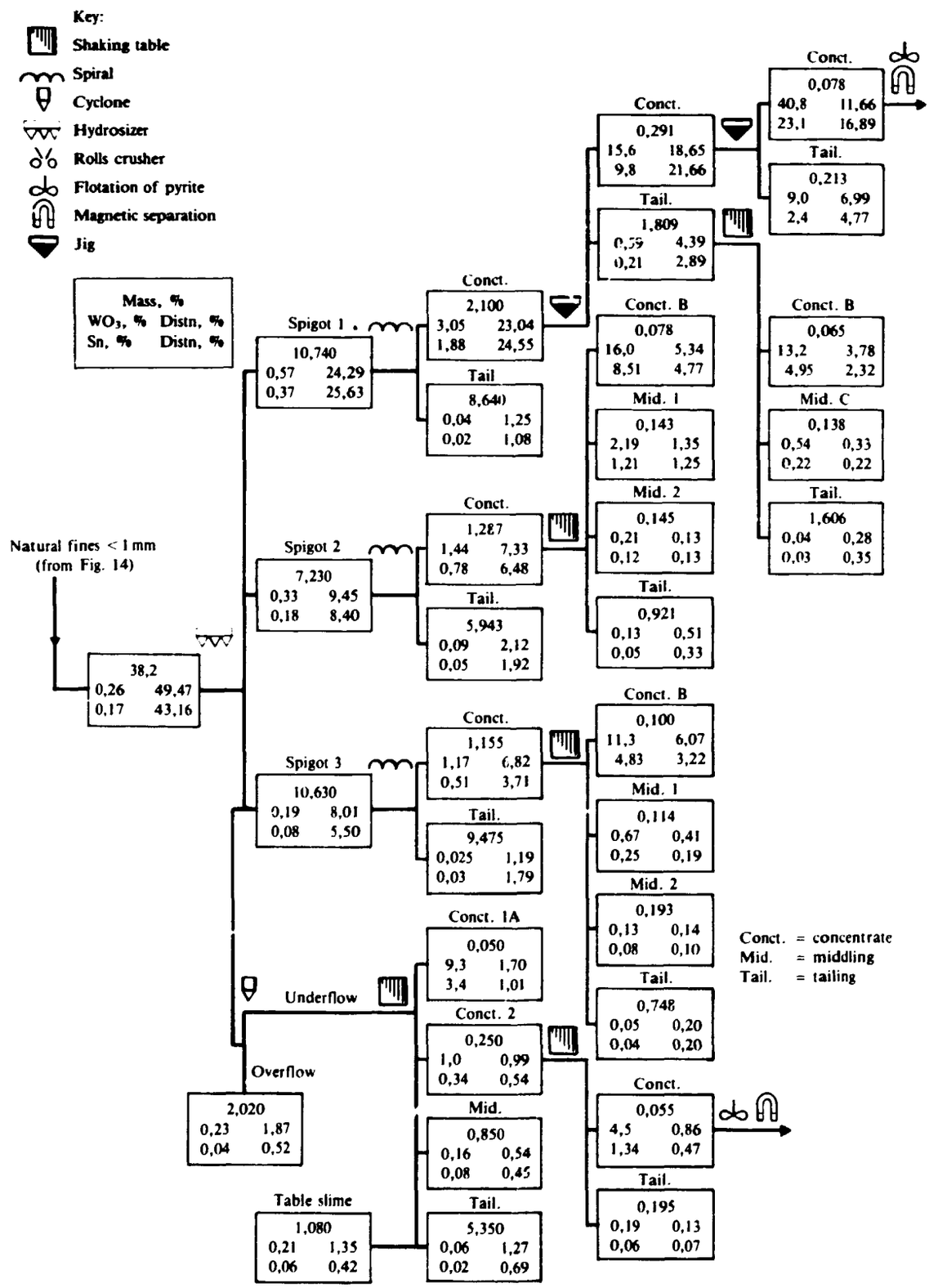


FIGURE 15. Treatment of natural fines from Sample J176

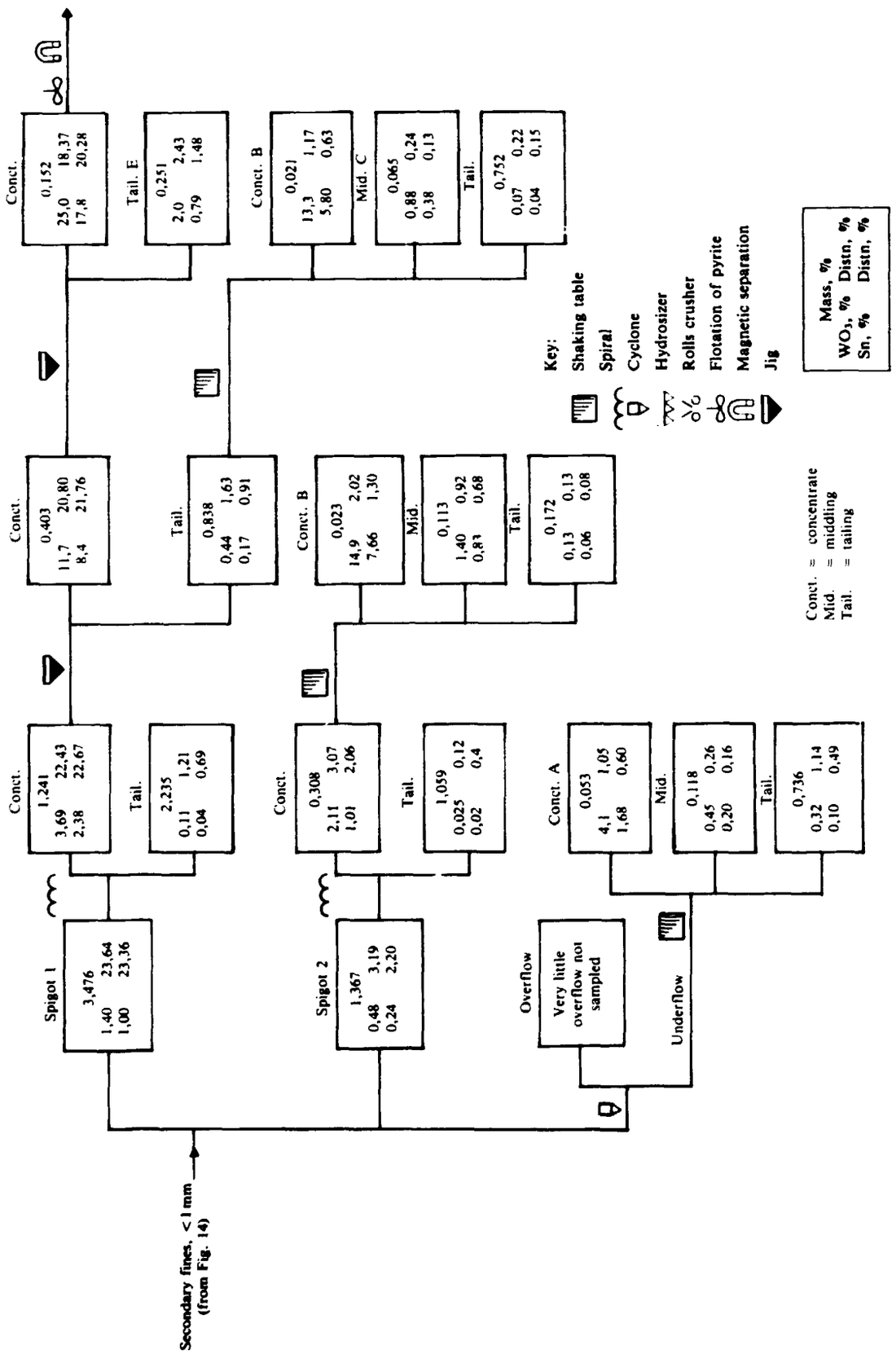


FIGURE 16. Treatment of secondary fines from Sample J176

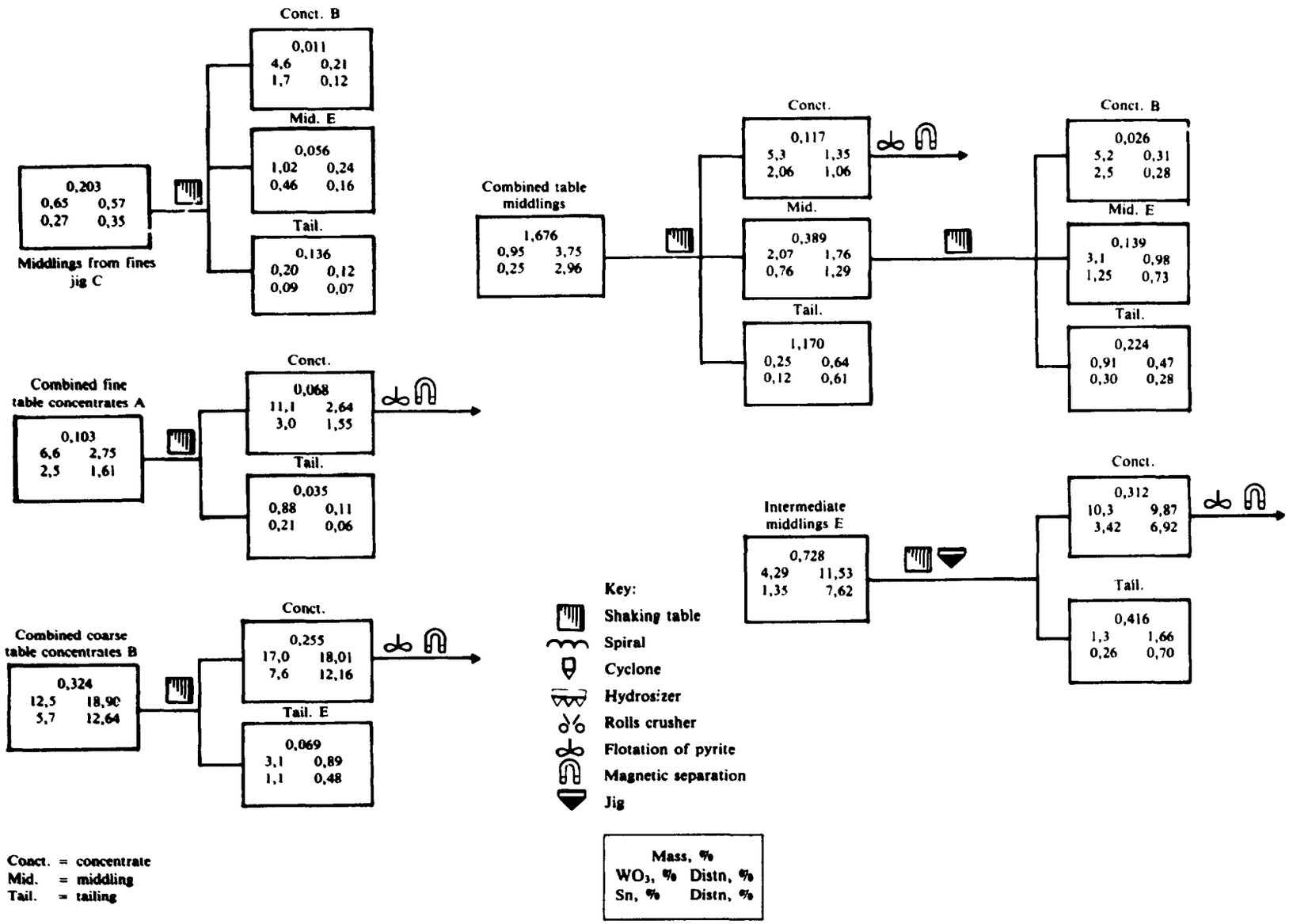


FIGURE 17. Retreatment of intermediate products from Sample J176

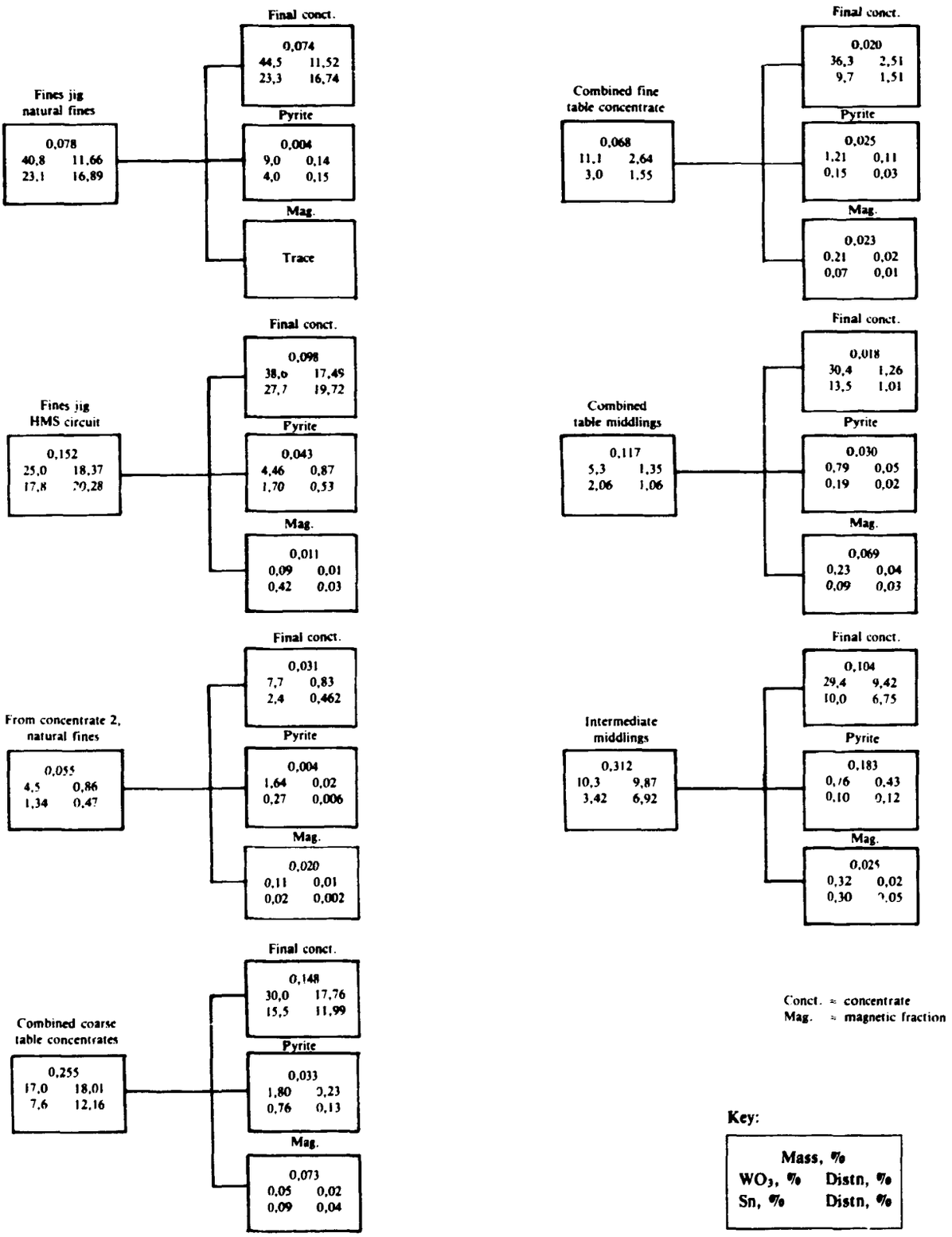


FIGURE 18. Final upgrading of concentrates from Sample J176 by flotation and magnetic separation

In the next series of tests, an attempt was made to float molybdenite, pyrite, and fluorspar selectively from a table concentrate that resulted from the retreatment of the fine jig concentrate. Tables 30 to 33, which give the results of some of these tests, show that the grade and recovery of molybdenite were poor in all instances, but the pyrite concentrate was generally of a high grade and contained between 92 and 99 per cent of the pyrite. The recovery of fluorspar was also high, but 15 to 50 per cent of the tungsten reported to the fluorspar concentrate. The results also indicate that almost 50 per cent of the tungsten reported to the molybdenite concentrate, probably because an excess of the molybdenite collector, Whiterex 309, was used in these tests.

4. DISCUSSION AND CONCLUSIONS

The tests were done with small laboratory equipment, and on some occasions during the treatment of the intermediate middling products the amount of material available was limited. All concentration by tabling was carried out on the same shaking table, whether coarse or fine particles were being treated. It is therefore expected that a full-scale plant, where the classification and the choice of jigs, tables, and other equipment would be better, would be able to obtain better recoveries than were obtained in the laboratory, especially since the table concentrate would be spread over a larger area, making the collection of a silicate-free concentrate much easier.

The results, especially the cyclosizing results for the hydrosizer fines, showed that the fine tailings products are enriched in tungsten trioxide or scheelite, as was to be expected, and it is essential that the plant should be designed so that the minimum of overgrinding occurs.

Ore of a size suitable for the small crushers at Mintek had been produced by the sponsor by a blasting technique that caused fragmentation different from that expected by the normal stoping method that will be used for mining. It is probable that less material finer than 1 mm will be produced during mining operations. The jaw crusher and centre-roll crushers that were used in the crushing operation are relatively small, and probably also added to the amount of fines produced in this investigation. During a discussion with the sponsor, it was estimated that the amount of material in the fraction larger than 1 mm would be closer to 75 per cent under plant conditions than the 60 per cent obtained at Mintek.

With this in mind, it can be predicted that the amount of concentrate collected in the coarse section could be increased by about 20 per cent. This would result in an increase in the overall grade of the final concentrate because the coarse concentrates were found to have higher grades than the fine concentrates. The finer concentrates would in practice have a higher grade owing to improved concentration on full-sized tables. A better overall recovery is also probable.

The testwork showed that HMS and jigging give fairly similar results, and, in view of the rapid increase in the price of ferrosilicon, the cheaper technique of preconcentration by jigging must be contemplated seriously.

It is predicted that a concentrate having a combined tin and tungsten grade of more than 65 per cent could be obtained from the high-grade ore, with a recovery possibly as high as 85 per cent. The low-grade ore on the other hand does not respond quite so well to the treatment, and for an acceptable grade of concentrate the recovery will probably be about 10 per cent lower than that from the high-grade ore.

Molybdenite can be recovered from the gravity concentrate by flotation, but a programme of optimization flotation tests would need to be carried out. The removal of fluorspar from the final concentrate by flotation is unlikely to be successful because the standard collectors for fluorspar also collect scheelite. Further tests using 'starvation' amounts of various collectors should be undertaken if fluorspar is to be removed by flotation. However, fluorspar (density 3 to 3,3) could probably be separated without difficulty from scheelite (density 5,9 to 6,1) on a plant in which the final upgrading by gravity separation is carried out on large concentrating tables.

TABLE 1

The main minerals of highest density found in the ore

Mineral	Composition	Hardness Mohs' scale	Relative density
Wolframite	(Fe, Mn)WO_4	5 to 5,5	7,1 to 7,9
Cassiterite	SnO_2	6 to 7	6,8 to 7,1
Scheelite	$CaWO_4$	4,5 to 5	5,9 to 6,1
Magnetite	Fe_3O_4	5,5 to 6,5	≈5,2
Pyrite	FeS_2	6 to 6,5	4,8 to 5,1
Fluorite	CaF_2	4	3 to 3,3
Tourmaline	Complex B, Al, Fe, SiO_2	7 to 7,5	3 to 3,2

TABLE 2

Operating conditions for HMS tests

Condition	Material fraction	
	<9 >1 mm	<6 >1 mm
Relative density of sump medium	2,41	2,40 to 2,45
Relative density of cyclone underflow	2,83	2,81
Relative density of returned medium	2,72	2,70
Pressure, kPa	75	80

TABLE 3

The results of the size split during comminution

Product mm	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Sample J175					
<9 >1	17,7	0,52	16,5	0,21	15,2
<6 >1	41,6	0,60	44,8	0,26	44,5
<1	40,7	0,53	38,7	0,24	40,3
Head (calculated)	100,00	0,56	100,00	0,24	100,00
Sample J176					
<9 >1	12,15	0,18	10,9	0,09	7,3
<6 >1	49,68	0,16	39,6	0,15	49,5
<1	38,17	0,26	49,5	0,17	43,2
Head (calculated)	100,00	0,20	100,00	0,15	100,00

TABLE 4

Size assay on the 6 to 1 mm fraction of Sample J175

Product mm	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
<6 >4	9,2	0,53	7,3	0,37	13,1
<4 >2	20,1	0,92	27,7	0,26	20,1
<2 >1	18,5	0,86	23,8	0,39	27,8
<1,0 >0,6	14,3	0,70	15,0	0,24	13,5
<0,6 >0,3	9,0	0,59	7,9	0,27	9,3
<0,3 >1,15	10,9	0,35	5,7	0,15	6,2
<0,15 >0,075	8,8	0,33	4,3	0,12	4,2
<0,075 >0,038	4,4	0,47	3,1	0,16	2,7
<0,038	4,8	0,72	5,2	0,16	3,1
Head	100	0,67	100,00	0,26	100,00

TABLE 5

Size analysis of material finer than 6 mm from Sample J176

Product mm	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
<6 >4	10,0	0,002	0,1	0,006	0,4
<4 >2	24,1	0,26	25,8	0,08	14,2
<2 >1	20,6	0,26	22,1	0,22	33,7
<1,0 >0,6	13,0	0,28	14,7	0,19	18,8
<0,6 >0,3	7,8	0,32	10,2	0,20	12,0
<0,3 >0,15	8,4	0,25	8,6	0,16	9,7
<0,15 >0,075	6,7	0,24	6,6	0,11	5,2
<0,075 >0,038	3,8	0,29	4,5	0,10	3,0
<0,038	5,6	0,32	7,4	0,08	3,0
Total	100,00	0,24	100,0	0,13	100,0

TABLE 6

Results of exploratory tests with the DWP separator on the concentrates or sink fractions from Sample J175

Relative density	Product mm	Mass %	WO_3 conct.		Sn conct.	
			%	Distn %	%	Distn %
2,8	<6 >1	8,52	6,4	93,71	2,02	86,25
3,0	<6 >1	6,10	9,7	94,03	3,67	85,63
3,2	<6 >1	4,79	11,1	91,78	3,42	81,14
3,3	<6 >1	5,37	10,2	92,05	3,23	82,09
ND	<6 >1	15,42	3,6	95,49	1,22	95,70
ND	<6 >1	22,44	2,7	97,63	0,79	95,81
ND	<6 >1	13,05	4,9	96,71	1,74	96,32
2,8	<9 >1	8,45	5,7	95,46	1,52	82,38
3,0	<9 >1	6,46	5,9	91,06	2,06	78,05
3,2	<9 >1	4,78	8,1	87,14	2,72	73,20

ND = not determined

TABLE 7

Results of exploratory HMS tests on Sample J175 using the DSM cyclone

Product mm	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
<6 >1	17,31	3,5	95,08	1,34	96,56
<6 >1	9,79	5,58	97,90	2,15	92,11
<6 >1	17,03	2,86	97,84	0,77	88,74
<6 >1	13,58	4,40	98,16	1,52	95,99
<6 >1	11,06	5,41	98,09	1,88	95,89
<9 >1	16,28	2,8	91,58	1,12	93,54
<9 >1	19,63	2,4	96,86	0,68	94,35
<9 >1	8,23	5,42	97,40	1,48	86,87
<9 >1	6,73	7,62	95,65	1,90	82,07
<9 >1	11,07	4,06	97,47	1,13	93,36

TABLE 8

Results of the final HMS cyclone runs on the bulk Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
<9 >1 mm					
Float	84,95	0,025	3,92	0,01	6,0
Sink	15,05	3,45	96,7	0,89	94,0
Head (calculated)	100,0	0,54	100,0	0,14	100,0
Head (assayed)		0,52		0,21	
<6 >1 mm					
Float	81,73	0,025	3,8	0,01	4,5
Sink	18,27	2,87	96,2	0,96	95,5
Head (calculated)	100,0	0,54	100,0	0,18	100,0
Head (assayed)		0,60		0,26	

TABLE 9

Results of HMS cyclone test on coarse material from Sample J176

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Float	90,52	0,01	5,50	0,01	7,15
Sink	9,48	1,64	94,50	1,24	92,85
Head (calculated)	100,00	0,16	100,00	0,13	100,00

TABLE 10

Primary jigging results for Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Hutch 1	1,99	19,2	67,0	7,36	66,6
Hutch 2	2,63	1,41	6,5	0,46	5,5
Hutch 3	3,05	0,59	3,2	0,19	2,6
Hutch 4	2,59	0,28	1,3	0,09	1,0
Locked in jig bed	2,19	4,73	18,2	1,63	16,3
Tailings	87,55	0,025	3,8	0,02	8,0
Head	100,00	0,57	100,00	0,22	100,00
Total concentrate	10,26	4,33	78,0	1,62	75,7
Concentrate including bed material	12,45	4,40	96,2	2,63	92,0

TABLE 11

Heavy-liquid separation test on primary jig tailing from Sample J175

Product mm	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
< 2,9	87,5	0,01	38,26	0,01	51,22
< 2.9 > 3,3	12,1	0,1	52,61	0,06	42,49
> 3,3	0,4	0,52	9,13	0,27	6,40
Head (calculated)	100,00	0,023	100,00	0,017	100,00

TABLE 12

The size distribution, determined by Microtrac, in the cyclone overflow of the hydrosizer slimes from natural fines

Size μm	Percentage passing given size
44	100,0
31	98,3
22	94,3
16	83,9
11	74,0
7,8	56,9
5,5	38,7
3,9	24,8
2,8	13,4

TABLE 13

The results of cyclosizing of the hydrosizer overflow from Sample J175

Size μm	Mass %	WO_3 %	WO_3 Distn %	Sn %	Sn Distn %
>42	9,5	0,25	6,4	0,04	8,3
<42 >32	24,8	0,30	20,2	0,05	27,1
<32 >23	26,9	0,34	24,8	0,05	29,5
<23 >16	11,3	0,42	12,9	0,06	14,9
<16 >12	4,8	0,48	6,2	0,05	5,3
<12	22,7	0,48	29,5	0,03	14,9
Head	100,0	0,37	100,0	0,05	100,0

TABLE 14

Size assay on the overflow from the slimes cyclone from Sample J175

Size μm	Mass %	WO_3 %	WO_3 Distn %	Sn %	Sn Distn %
>42	5,1	0,32	3,4	0,06	7,2
<42 >32	4,3	0,33	2,9	0,06	6,1
<32 >23	3,6	0,34	2,5	0,05	4,2
<23 >16	4,1	0,39	3,3	0,04	3,7
<16 >12	5,5	0,47	5,3	0,05	6,5
<12	77,4	0,52	82,6	0,04	72,3

TABLE 15

Final concentrates from Sample J175

Product	Mass %	WO_3 %	WO_3 Distn %	Sn %	Sn Distn %
Secondary jig high grade	0,240	49,8	20,73	20,2	22,63
Secondary jig pyrite conct.	0,135	51,4	11,28	13,0	9,09
From fines jig, secondary jig tailing	0,040	47,9	2,53	23,5	4,27
From fines jig, HMS fines	0,051	53,0	6,03	22,2	6,89
From fines jig, natural fines	0,332	46,0	18,73	25,5	23,34
From table conct., coarse	0,102	37,3	7,30	12,8	5,95
From table conct., fine	0,064	16,6	1,68	5,3	1,16
From table conct., table conct. 2	0,017	15,6	0,55	4,1	0,38
From table conct., combined middlings	0,017	30,4	0,86	13,5	0,58
From table conct., spigot 2	0,028	45,4	3,63	16,2	2,68
From table conct., intermediate middlings	0,125	32,9	6,76	9,0	5,87
Total conct.	1,151	43,3	80,08	18,0	82,84

Conct. = concentrate

TABLE 16

Pyrite-flotation products from Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
From secondary jig pyrite conct.	0,064	4,0	0,42	0,53	0,18
From natural fines, fine jig conct.	0,407	3,43	1,71	0,74	0,91
From natural fines, spigot 2, tables	0,022	3,05	0,19	0,63	0,08
From slimes table, conct. 2	0,012	0,79	0,02	0,08	0,01
From secondary jig tailing, fines jig	0,014	8,5	0,16	1,77	0,12
From HMS fines jig	0,002	10,7	0,06	2,6	0,04
From combined coarse conct.	0,108	1,0	0,20	0,24	0,12
From combined fine conct.	0,009	1,86	0,02	0,23	0,01
From combined middlings, retabled	0,028	0,79	0,04	0,19	0,01
From intermediate middlings, re-treated	0,256	0,59	0,24	0,06	0,08
Total pyrite	0,922	2,35	3,06	0,47	1,56

Conct. = concentrate

TABLE 17

Total magnetic fraction from Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
From natural fines, fine jig conct.	0,125	0,78	0,12	2,72	1,01
From natural fines, spigot 2, tabled	0,047	0,16	0,02	0,22	0,06
From slimes table conct. 2	0,015	0,16	0,01	0,06	0,00
From secondary jig tail, fines jig	Trace	0,11	-	0,14	-
From HMS fines jig	Trace		-		-
From combined coarse conct.	0,108	0,18	0,04	0,13	0,06
From combined fine conct.	0,041	0,23	0,02	0,09	0,01
From combined middlings, retabled	0,064	0,23	0,02	0,09	0,01
From intermediate middlings, re-treated	0,064	0,32	0,03	0,30	0,10
Total magnetic fraction	0,464	0,37	0,26	0,86	1,25

Conct. = concentrate

TABLE 18

Final tailings from Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
HMS primary tailing	49,200	0,025	2,30	0,01	2,90
Natural fines, cyclone overflow	0,183	0,45	0,12	0,09	0,08
Natural fines, table tailing	5,230	0,18	1,55	0,05	1,01
Natural fines, table conct. 2, retabled	0,148	0,49	0,15	0,14	0,12
Natural fines, spigot 3 spiral tailing	4,501	0,06	0,51	0,04	0,67
Natural fines, spigot 3 table tailing	0,624	0,11	0,13	0,07	0,18
Natural fines, spigot 2 spiral tailing	9,789	0,05	1,13	0,05	2,01
Natural fines, spigot 2 table tailing	1,391	0,11	0,38	0,06	0,33
Natural fines, spigot 1 spiral tailing	13,525	0,05	1,21	0,02	1,24
Natural fines, fines jig tabled tailing	2,437	0,13	0,57	0,07	0,60
Sec. jig tail, slimes table overflow	0,379	1,27	0,74	0,14	0,23
Sec. jig tail, slimes table tailing 1	0,386	0,24	0,14	0,09	0,16
Sec. jig tail, slimes table tailing 2	0,583	0,37	0,33	0,08	0,21
Sec. jig tail, spigot 2 spiral tailing	1,824	0,13	0,41	0,06	0,48
Sec. jig tail, spigot 2 table tailing	0,156	0,21	0,05	0,07	0,04
Sec. jig tail, spigot 1 spiral tailing	1,443	0,18	0,41	0,09	0,60
Sec. jig tail, spigot 1 jig tailing retabled	1,313	0,32	0,65	0,10	0,40
HMS, slimes table tailing	0,216	1,08	0,51	0,20	0,29
HMS, spigot 2 spiral tailing	0,459	0,25	0,27	0,08	0,24
HMS, spigot 2 table tailing	0,068	0,18	0,03	0,07	0,03
HMS, spigot 1 spiral tailing	0,603	0,26	0,37	0,08	0,34
HMS, spigot 1 jig tailing retabled	0,470	0,19	0,19	0,07	0,29
From combined middlings tabled	1,089	0,25	0,43	0,12	0,35
From combined middlings retabled	0,209	0,91	0,32	0,30	0,16
From fine jig table middlings, retabled	0,493	0,52	0,48	0,16	0,28
From retabling of fine table conct.	0,034	0,83	0,05	0,26	0,03
From reconcentrating intermediate middlings	0,710	2,7	3,17	0,27	1,00
Total tailing	97,463	0,10	16,60	0,03	14,35

Conct. = concentrate

TABLE 19

Final results for Sample J175

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Concentrate	1,15	43,3	80,1	18,0	82,8
Tailing	98,85	0,12	19,9	0,04	17,2
Head: Calculated	100,00	0,64	100,0	0,27	100,0
Assayed		0,56		0,26	

TABLE 20

Results of semi-quantitative analysis on a sample of final concentrate from Sample J175

Constituent	%
WO_3	46,4
Sn	19,2
Fe_2O_3	6,0
MnO	0,2
Cr_2O_3	<0,1
CaO	3,1
S	0,4
P_2O_5	0,2
SiO_2	3,0
Al_2O_3	0,5
MgO	0,1
TiO_2	0,4
Constituent	p.p.m.
Mo	175
Nb	105
Zr	620
Y	175
Cu	60
La	160

TABLE 21

Final concentrates from Sample J176

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Secondary jig, high grade	0,110	34,3	18,47	36,1	25,97
From fines jig, secondary jig, and HMS	0,098	38,6	17,49	27,7	19,72
From fines jig, natural fines	0,074	44,5	11,52	23,3	16,74
From table conct., coarse	0,148	30,0	17,76	15,5	11,99
From table conct., fine	0,020	36,3	2,51	9,7	1,51
From table conct., combined middlings	0,018	30,4	1,26	13,5	1,01
From table conct., table conct. 2	0,031	7,7	0,83	2,4	0,46
From reconcentrated intermediate middlings	0,104	29,4	9,42	10,0	6,75
	0,603	33,0	79,26	20,3	84,15

Conct. = concentrate

TABLE 22

Pyrite-flotation products from Sample J176

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
From natural fines, fine jig conct.	0,004	9,0	0,14	4,0	0,15
From natural fines, table conct. 2	0,004	1,64	0,02	0,27	0,01
From HMS and secondary jig, fine jig conct.	0,043	4,46	0,87	1,70	0,53
From coarse tabled conct	0,033	1,80	0,23	0,76	0,13
From fine tabled conct.	0,025	1,21	0,11	0,15	0,03
From table middlings retabled	0,030	0,79	0,05	0,19	0,02
From reconcentrated intermediate middling	0,183	0,76	0,43	0,10	0,12
	0,322	1,51	1,85	0,44	0,99

Conct. = concentrate

TABLE 23

Total magnetic fraction of Sample J176

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
From natural fines, fine jig conct.	Trace				
From natural fines, table conct. 2	0,020	0,11	0,01	0,02	0,00
From HMS and secondary jig, fine jig conct.	0,011	0,09	0,01	0,42	0,03
From coarse tabled conct.	0,073	0,05	0,02	0,09	0,04
From fine tabled conct.	0,023	0,21	0,02	0,07	0,01
From table middlings retabled	0,069	0,23	0,04	0,09	0,03
From reconcentrated intermediate middlings	0,025	0,32	0,02	0,30	0,05
	0,221	0,16	0,12	0,14	0,16

Conct. = concentrate

TABLE 24

Final tailings from Sample J176

Product	Mass %	WO_3 %	WO_3 Distn %	Sn %	Sn Distn %
HMS primary tailing	55,940	0,01	2,78	0,01	4,06
Natural fines, cyclone overflow	2,020	0,23	1,87	0,04	0,52
Natural fines, table slimes	1,080	0,21	1,35	0,06	0,42
Natural fines, table tailing	5,351	0,06	1,27	0,02	0,69
Natural fines, table conct. 2 retabled	0,195	0,19	0,13	0,06	0,07
Natural fines, spigot 3 spiral tailing	9,475	0,025	1,19	0,03	1,79
Natural fines, spigot 3 table tailing	0,748	0,05	0,20	0,04	0,20
Natural fines, spigot 2 spiral tailing	5,943	0,09	2,12	0,05	1,92
Natural fines, spigot 2 table tailing	0,921	0,13	0,51	0,05	0,33
Natural fines, spigot 1 spiral tailing	8,640	0,04	1,25	0,02	1,08
Natural fines, fines jig tailing retabled	1,606	0,04	0,28	0,03	0,35
HMS and jig slimes, table tailing	0,736	0,32	1,14	0,10	0,49
HMS and jig, spigot 2 spiral tailing	1,059	0,025	0,12	0,02	0,14
HMS and jig, spigot 2 table tailing	0,172	0,13	0,13	0,06	0,08
HMS and jig, spigot 1 spiral tailing	2,235	0,11	1,21	0,04	0,69
HMS and jig, spigot 1 fines jig tailing retabled	0,752	0,07	0,22	0,04	0,15
From fine jig tailing, table middlings retabled	0,136	0,20	0,12	0,09	0,07
From tabling fine table concentrates	0,035	0,88	0,11	0,21	0,06
From tabling table middlings	1,170	0,25	0,64	0,12	0,61
From tabling middlings from retabling table middlings	0,224	0,91	0,47	0,30	0,28
From intermediate middlings re-treated	0,416	1,3	1,66	0,26	0,70
	98,854	0,05	21,55	0,02	18,76

Conct. = concentrate

TABLE 25

Final results for Sample J176

Product	Mass %	WO_3		Sn	
		%	Distn %	%	Distn %
Concentrate	0,60	33,0	79,3	20,2	84,2
Tailing	99,40	0,05	20,7	0,02	15,8
Head: Calculated	100,00	0,25	100,0	0,14	100,0
Assayed		0,20		0,15	

TABLE 26

Mineralogical composition of table concentrates

Ore sample	Product	Assay, %		Material locked in silicate gangue, %			Minerals present, %				
		WO_3	Sn	Wolframite	Scheelite	Cassiterite	Pyrite	Magnetite	Ore minerals	Gangue* locked on ore minerals	Free gangue
J175	From fines jig tailing from the secondary jig tailing	7,2	2,3	34	15	6	–	–	11,0	2,9	87,1†
J175/ J176	From combined primary table middlings retabled	5,3	2,1	16	10	9	–	–	9,5	1,3	89,2†
J175/ J176	From the retreatment of the middlings from sample above	5,2	2,5	13	12	0	–	–	10,2	1,0	88,2†
J175	From upgrading of coarse table concentrates	12,3	4,2	10	8	4	34	34	22,3	1,9	7,8
J175	From upgrading of fine table concentrates	9,3	3,1	22	14	12	8	36	15,1	3,1	37,8
J175	From upgrading of concl. 2 from tabling of cyclone underflow	6,2	1,5	12	10	8	27	34	10,2	1,2	27,6
J176	From upgrading of concl. 2 from tabling of cyclone underflow	4,5	1,3	18	13	7	7	36	7,4	1,2	48,4
J176	From upgrading of coarse table concentrates	17,0	7,6	22	9	10	13	29	31,1	5,0	21,9
J176	From upgrading of fine table concentrates	11,1	3,0	12	5	6	37	34	19,2	1,6	8,2

* Maximum amount, %

† This includes the pyrite and magnetite

TABLE 27

Results of a flotation test on the recovery of molybdenite from the jig concentrate from the HMS fines circuit

Product	Mass %	Mo %	Distn %
Mo concentrate	0,26	3,75	10,5
Cleaner tailing	1,74	0,91	17,0
Rougher tailing	98,00	0,069	72,5
Head: Calculated	100,00	0,093	100,0
Assayed		0,086	

Stage	Reagents added, g/t		Time, min	
	W309	D200	Condition	Froth
Rougher	50		2	
		18	1	4
	25	9	2	1
Cleaner	10	14	1	3

W309 = Whiterex 309 Mobil reagent
D200 = Dowfroth 200

TABLE 28

Results of a flotation test on the recovery of molybdenite from the table concentration of the combined primary fine table concentrates

Product	Mass %	Mo %	Distn %
Rougher concentrate	0,20	25,6	29,1
Rougher tailing	99,80	0,125	70,9
Head: Calculated		0,176	100,0
Assayed		0,215	

Stage	Reagents added, g/t			Time, min	
	NaSi	W309	D200	Condition	Froth
Rougher	200			2	
		50	14	3	3
		25		1	1

W309 = Whiterex 309
NaSi = sodium silicate
D200 = Dowfroth 200

TABLE 29

Results of a flotation test on the recovery of molybdenite from the table concentrate obtained by the retabling of the combined second concentrates from the primary tables

Product	Mass %	Mo %	S %	Distn %
Mo concentrate	1,37	6,69		69,3
Mo cleaner tailing	2,30	0,60		10,4
Pyrite concentrate	35,82	0,030	50,8	8,1
Pyrite cleaner tailing	3,24	0,091		2,2
Rougher tailing	57,27	0,023		10,0
Head: Calculated	100,00	0,132		100,0
Assayed		0,164		

Stage	Reagents added, g/t					Time, min	
	NaSi	W309	D200	$CuSO_4$	PAX	Condition	Froth
Mo rougher	200					2	
		50	14			3	3
Mo cleaner		15	18			2	2
Pyrite rougher				200		1	
			5		100	3	5
			9	100	100	2	1
Pyrite cleaner							4

PAX = potassium amyl xanthate
NaSi = sodium silicate
W309 = Whiterex 309
D200 = Dowfroth 200

TABLE 30

Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (1)

Product	Mass %	Assay, %					Distribution, %				
		Mo	FeS_2	CaF_2	Sn	W	Mo	FeS_2	CaF_2	Sn	W
Mo conct.	6,09	0,44					15,0				
Pyrite conct.	70,26		88,51					94,5			
CaF_2 conct.	10,38			39,3		14,3			72,9		28,1
Flotation tailing	13,27			3,8	3,28	8,77			9,0	46,6	22,1
Assayed head		0,18	65,8	5,6	0,934	5,72					

Stage	Reagents added, g/t						Time, min		pH value
	NaSi	W309	D200	$CuSO_4$	PAX	CA50	Condition	Froth	
Mo rougher	200						3		Normal
		200					3		
			40					3	
Pyrite rougher				200			2		Normal
					100		3		
			20					5	
Scavenger				100			1		Normal
					100		3	5	
CaF_2						550	3		9,0
			20					3	

Note: The pH value and temperature were uncontrolled until the fluorspar flotation stage, when the pH value was raised to 9,0 with sodium carbonate and the pulp was heated to 60 °C.

NaSi = sodium silicate
W309 = Whiterex 309
CA50 = collector (manufactured by Betachem (Pty) Ltd, 31 Pafuri Road, Emmarentia, Johannesburg)
Conct. = concentrate
PAX = potassium amyl xanthate
D200 = Dowfroth 200

TABLE 31

Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (2)

Product	Mass %	Assay, %					Distribution, %				
		Mo	FeS_2	CaF_2	Sn	W	Mo	FeS_2	CaF_2	Sn	W
Mo conct.	3,42	2,20					41,6				
Pyrite conct.	71,50		91,88					99,81			
CaF_2 conct.	19,13			24,45		13,95			83,5		50,7
Flotation tailing	5,95			3,1	1,28	4,39			3,3	8,1	4,9
Assayed head		0,18	65,8	5,6	0,934	5,27					

Stage	Reagents added, g/t						Time, min		pH value
	NaSi	W309	D200	$CuSO_4$	PAX	CA50	Condition	Froth	
Mo rougher	200						3		9,0
		200					3		
			40					3	
Pyrite rougher				200			2		9,0
					100		3		
			20					5	
Scavenger				100			1		9,0
					100		3	5	
CaF_2						823	3		9,0
			20					3	

Note: The pulp was at ambient temperature until the fluorspar flotation stage, when it was raised to 60 °C.
NaSi = sodium silicate
W309 = Whiterex 309
CA50 = Betachem collector
Conct. = concentrate
PAX = potassium amyl xanthate
D200 = Dowfroth 200

TABLE 32

Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (3)

Product	Mass %	Assay, %					Distribution, %				
		Mo	FeS_2	CaF_2	Sn	W	Mo	FeS_2	CaF_2	Sn	W
Mo conct.	3,22	0,845					15,0				
Pyrite conct.	70,90		91,97					99,1			
CaF_2 conct.	18,93			25,3		13,7			85,5		49,2
Flotation tailing	6,95			3,7	0,89	6,69			4,6	6,6	8,8
Assayed head		0,18	65,8	5,6	0,934	5,27					

Stage	Reagents added, g/t						Time, min		pH value
	NaSi	W309	D200	$CuSO_4$	PAX	CA50	Condition	Froth	
Mo rougher	200						3		Normal
		200	40				3	3	
Pyrite rougher				200			2		Normal
			20		100		3	5	
Pyrite scavenger				100			1		Normal
					100		3	5	
CaF_2						1700	3		9,0
			40					3	

Note: The pH value and temperature were uncontrolled until the fluorspar flotation stage, when the pH value was raised to 9,0 with sodium carbonate and the pulp was heated to 60 °C.

NaSi = sodium silicate
W309 = Whiterex 309
CA50 = Betachem collector
Conct. = concentrate
PAX = potassium amyl xanthate
D200 = Dowfroth 200

TABLE 33

Results of flotation tests on the removal of molybdenite, pyrite, and fluorspar from a concentrate obtained by the tabling of the fine jig concentrate (4)

Product	Mass %	Assay, %					Distribution, %				
		Mo	FeS_2	CaF_2	Sn	W	Mo	FeS_2	CaF_2	Sn	W
Mo conct.	4,41	0,55					13,3				
Pyrite conct.	72,04		84,77					92,5			
CaF_2 conct.	6,61			60,2		12,1			71,07		15,16
Flotation tailing	16,94			1,5	5,07	11,6			4,46	91,97	37,29
Assayed head		0,18	65,8	5,6	0,934	5,27					

Stage	Reagents added, g/t						Time, min		pH value
	NaSi	W309	D200	$CuSO_4$	PAX	CA50	Condition	Froth	
Mo rougher	200						3		Normal
		200					3		
			40					3	
Pyrite rougher				200			2		Normal
					100		3		
			20					5	
Pyrite scavenger				100			1		Normal
					100		3	5	
CaF_2						1700	3		11,0
			20					3	

Note: The pH value and temperature were uncontrolled until the fluorspar flotation stage, when the pH value was raised to 11,0 with sodium carbonate and the pulp was heated to 60 °C.

NaSi = sodium silicate
W309 = Whiterex 309
CA50 = Betachem collector
Conct. = concentrate
PAX = potassium amyl xanthate
D200 = Dowfroth 200