Some practical considerations in the design and operation of a plant for the differential flotation of mixed sulphides, especially copper and zinc

by P. MARAIS*, Pr. Eng., B.Sc. (Eng.), F.S.A.I.M.M.

SYNOPSIS

This paper gives an account of the development of a plant for the differential flotation of mixed-sulphide ore, special reference being made to the separation of copper and zinc sulphides. The uniqueness of the geology, mineralogy, and metallurgy of each complex ore-body is stressed. Consideration is given to the scale-up of laboratory data, and to some of the factors involved in the design of milling and flotation plants for this type of ore. Process control and development are discussed, as are the human requirements for the optimum operation of a concentrator.

SAMEVATTING

Hierdie referaat doen verslag oor die ontwikkeling van 'n aanleg vir die differensiale flottering van gemengdesulfiideerts, met spesiale verwysing na koper- en sinksulfiide. Die unieke van die geologie, mineralogie en metallurgie van elke komplekse ertslig aan en sommige van die faktore betrekke by die ontwerp van maal- en flotatiesaanlegginge vir hierdie soort erts. Prosesbeheer en -ontwikkeling word bespreek aangoe die menslike vereistes vir die optimale bedryf van 'n konsentreerder.

INTRODUCTION

The differential flotation of mixed sulphides, particularly copper-zinc and copper-lead-zinc sulphides presents one of the most complicated and difficult problems that the base-metal metallurgist and mineral-processing engineer have to solve. Pyrite is invariably present in copper-zinc ore-bodies. In practice, when mixed, chalcopyrite, sphalerite, and pyrite have similar flotation characteristics, and the separation of these minerals is difficult. Recoveries are often relatively low, particularly in the case of zinc. The separation becomes even more difficult in the presence of pyrrhotite and marmatite.

Flotation problems in general are of geological origin. The mineralogical characteristics of a mixed-sulphide deposit have a controlling influence on the flotation process to be used. As flotation is a surface phenomenon, any alteration of the mineral surfaces is of great importance. Not only the flotation parameters, but often the flotation process itself, have to be changed as the mineralogical characteristics of the ore change.

Mineralogical characteristics of ore-bodies in the same geographical area often differ widely, and equally wide differences can be found in stopes of the same ore-body separated by but a few metres. It is obvious, then, that the technique employed by a particular concentrating plant for the separation of mixed sulphides is tailored to the nature of the particular ore-body.

It was considered that little would be gained from a detailed examination of the plant circuit and flotation process used by one particular concentrator, which in this case would have been the copper-zinc concentrator of the Anglovaal mine, Prieska Copper Mines (Pty) Ltd. This paper therefore attempts a broad review of some of the more important practical considerations in the design and operation of a plant for the differential flotation of mixed sulphides based on the experience gained at Prieska Copper Mines in the separation of copper and zinc.

Many of the observations and considerations are common to the differential flotation of mixed sulphides, to the flotation of a single sulphide, and for that matter to most mineral flotation operations. However, because of the extra sensitivity of differential flotation, the effect of many of the common factors are considerably magnified and require special attention.

METALLURGICAL EVALUATION OF THE ORE-BODY

Samples for Laboratory Tests

Samples for laboratory tests that are to form the basis of a metallurgical evaluation are usually obtained from drill cores. The samples must be representative of the deposit chemically, mineralogically, and in the concentration and degree of dissemination of the valuable minerals.

It is highly desirable that drill cores should be examined mineralogically before a representative sample is selected for flotation tests. Samples from the surface layer of the deposit are invariably more oxidized than samples from depth, and, if there are major differences between different areas in the deposit, especially if these can be mined separately, separate samples should be obtained from these areas. Representative samples are useful for the development of a general flotation procedure, but additional testing must be conducted on samples from various areas and depths to establish the optimum design data for a concentrator over the expected range of ore variation.

Mineralogical Examination

The mineralogical examination should yield all the necessary information relative to the mineralogical composition, mineral associations, and size distributions of the valuable minerals in the original ore and in the milled and flotation products, and to the degree of liberation in relation to particle size, etc.
Particular attention must be paid to the occurrence of exsolution chalcopyrite and sphalerite and of supergene alteration, which result in abnormally active sphalerite that will report to the copper concentrate in excessive quantities during flotation.

Carefully selected sphalerite grains should be subjected to electron-microprobe analysis so that the iron content of the crystal lattice can be determined. At Prieska this iron content varies from 2.4 per cent in translucent, yellow grains to 0.5 per cent in black, marmatite-type grains. The low-iron sphalerite floats readily, so that high recoveries are obtained in the zinc concentrate at high grades provided special precautions are taken to prevent this sphalerite from floating with the chalcopyrite. The high-iron sphalerite is slow to float, little reports to the copper, and zinc recoveries and concentrate grades tend to be lower.

One mine¹ has found that, whereas the crystal lattice of sphalerite in the zinc concentrate has a manganese content of 0.02 per cent, the non-floating sphalerite in the zinc scavenger tailing contains 2.0 per cent manganese. Prieska sphalerite contains from 0.10 to 0.44 per cent manganese.

The behaviour of pyrrhotite in copper-zinc flotation is unpredictable and can be a major problem. In some deposits,² it seems that monoclinic pyrrhotite (approximately Fe₃S₄) is ferromagnetic and does not seem to be particularly amenable to flotation. Hexagonal pyrrhotite (Fe₃S₄-Fe₃S₁₂) is non-magnetic but seems to be readily floatable. Two further types are encountered: troilite and orthorhombic pyrrhotite. Neither of these has been able to depress the pyrrhotite with pre-eration. Myllykoski Mine³ reports the activation of pyrrhotite at pH 11.5. Marcona⁴ reports that the flotation behaviour of pyrrhotite becomes even more unpredictable in the presence of magnetite, a common mineral in deposits of copper-zinc sulphide.

The Prieska Ore-body

As frequent references will be made to the operations at Prieska, a brief description is given of the ore-body, which is essentially a massive, pyritic ore-body containing up to 35 per cent pyrite with sphalerite, chalcopyrite, and pyrrhotite, and lesser amounts of marcasite, magnetite, and galena. A little bismuth as tellurides and free bismuth appears to be associated with the lead. Gold and silver are also present in minor quantities.

Texturally, the ore is divided into four types based largely on the appearance of the pyrite: porphyro-lastic pyrite, coarse often rounded crystalline pyrite, granular pyrite, and on the fringes of the deposit 'hackle' and disseminated pyrite.

The main non-opaque gangue minerals are quartz, feldspar, mica, and amphibole. In some portions of the ore-body the gangue consists of only calcium and magnesium carbonates, while barite also occurs in varying quantities. Mostly rounded, but sometimes angular, inclusions resembling lapilli in shape are common, and consist of country rock, quartz, feldspar, etc.

The ore is leached and unmineable to approximately 100m below the surface. Gossan and malachite occur down to a depth of 30 to 50 m. Immediately below this, the ore is a soft, clay-like mass containing various amounts of pyrite as loose grains. The grade of leaching decreases at depth to isolated 'veins' on the ore contacts or along joints. One malachite showing with some chrysocolla has been encountered. Although some secondary copper sulphides are occasionally present, secondary enrichment processes appear to have played a minor role.

The sphalerite occurs as first-generation sphalerite between euhedral pyrite grains, second-generation sphalerite in granular intergrowths with chalcopyrite and pyrrhotite, and third-generation sphalerite containing exsolution blebs of chalcopyrite.

The ore-body occurs in a generally wedge-shaped micaceous envelope. This is an often intensely sheared, rather quartzitic zone with abundant muscovite, biotite, chlorite, hydro-muscovite, and occasionally an amphibolerich band on or near the footwall contact of the ore.

A chemical analysis of samples of ore taken simultaneously from ten operating stopes is shown in Table 1.

Preparation of Samples for Laboratory Flotation Tests

Metallurgical test work should be carried out on samples of the ore-body as soon after sampling as possible. This is particularly important with drill cores, in which surface oxidation, owing to their high ratios of surface area to volume, can have a significant effect on flotation tests.

It is not unusual for only a few kilograms of drill cores to be available for any one series of tests. Particularly with samples of small volume, it is essential that the laboratory equipment used in crushing, grinding, and flotation should be scrupulously cleaned before and after each test to avoid contamination. Experience has shown that a jaw crusher containing even a little material from a previous sample can upset the flotation tests on the next sample. Before the tests, all the equipment should be 'flushed' with a batch of quartz.

The grinding medium of the laboratory mill must be carefully selected. With some ores, the grinding medium can have a significant effect on the subsequent flotation. This aspect is discussed in more detail later.

TABLE I

<table>
<thead>
<tr>
<th>Item</th>
<th>Minimum</th>
<th>Maximum</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zn to Cu ratio</td>
<td>0.9</td>
<td>13,3</td>
</tr>
<tr>
<td>Pb, %</td>
<td>0.01</td>
<td>0.08</td>
</tr>
<tr>
<td>Bi, %</td>
<td>Nil</td>
<td>0.003</td>
</tr>
<tr>
<td>Al₂O₃, %</td>
<td>2.05</td>
<td>20.33</td>
</tr>
<tr>
<td>CaO, %</td>
<td>3.02</td>
<td>10.75</td>
</tr>
<tr>
<td>MgO, %</td>
<td>2.80</td>
<td>8.89</td>
</tr>
<tr>
<td>Mn, %</td>
<td>0.06</td>
<td>0.18</td>
</tr>
<tr>
<td>SiO₂, %</td>
<td>5.00</td>
<td>45.80</td>
</tr>
<tr>
<td>Cd, %</td>
<td>Trace</td>
<td>0.013</td>
</tr>
<tr>
<td>Total Fe₂O₃, %</td>
<td>10.53</td>
<td>33.36</td>
</tr>
<tr>
<td>BaO, %</td>
<td>0.41</td>
<td>8.56</td>
</tr>
<tr>
<td>Total S, %</td>
<td>3.04</td>
<td>37.13</td>
</tr>
</tbody>
</table>

Pyrrhotite as Fe₃S₄ (analysed), % | 1.93 | 8.35 |
Pyrite as Fe₃S₄ (calculated), % | 2.28 | 65.37 |
Magnetite Fe₃O₄ (calculated), % | 1.53 | 10.79 |

Dry relative density | 5.04 | 4.21 |
Laboratory Flotation Tests

The usefulness of laboratory flotation tests depends on how accurately the results can be scaled up to full plant performance. Although laboratory flotation tests are essential, the results obtained from them must be analysed with great circumspection for the following reasons:

1. The recovery and separation of mixed sulphides depend on the particle-size distribution, among other factors. The size distributions obtained from a small laboratory mill, particularly if batch milling is employed, are unrelated to plant-scale distributions.

2. All the particles in a batch laboratory flotation cell have equal residence times. This is not so on a plant.

3. Unless controlled, the mixing and aeration rates are higher in laboratory cells than in plant cells. Controlled tests carried out in the Anglovaal Group have shown laboratory aeration rates of up to 6,6 m3/min per cubic metre of cell volume as against a general average of 1.4 m3/min for plant cells.

4. Although cycle tests are used in the laboratory to simulate the intermediate circulating products in a plant, excessively large circulating loads often develop. The problem is compounded when intermediate products have to be milled.

5. As discussed later, the separation of mixed sulphides is sensitive to temperatures. This aspect often does not receive sufficient attention during laboratory testwork.

6. The quality and composition of the fresh-water supply and circulating water at a plant can have a significant effect on flotation performance. Laboratory tests should be carried out with the fresh-water supply that will be used on the plant. It is extremely difficult to simulate plant recycle water in the laboratory, particularly when the plant is not yet operative. Table II gives the analytical results obtained for soluble copper and zinc in various plant solutions at Prieska Copper Mines when cyanide and zinc sulphate were used as depressants of the sphalerite.

Pilot-plant Tests

Many successful flotation plants have been built from a scale-up of the results obtained from laboratory batch tests without the benefit of pilot-plant results. However, new or difficult flotation processes that present novel or special problems, and that have no close counterpart among existing plants, require piloting on as large and extensive a scale as possible. Projects involving the differential flotation of mixed sulphides invariably fall within this category.

Whether the concentrates are to be sold, toll-smelted, or smelted in-company, it is important that the occurrence of minor and trace elements in the concentrates should be investigated in detail prior to designing a smelter or entering into any contractual arrangements with regard to the disposal of the concentrates. It is unlikely that concentrates obtained from small-scale laboratory tests will be sufficiently representative to form the basis of such arrangements. Elements that should probably be investigated are iron, sulphur, lead, silver, gold, antimony, cobalt, bismuth, selenium, tellurium, arsenic, manganese, calcium, magnesium, chromium, silicon, nickel, aluminium, chlorine, mercury, germanium, and cadmium.

PLANT DESIGN

In a paper of this nature, it is possible only to touch briefly on the broad aspects of plant design that have a major bearing on, or are determined by, the flotation plant and process.

Crusher Plant

The design of the crusher plant depends on the type of milling plant to be used, which in turn depends on the flotation plant, the flotation process, and characteristics of the ore. For example, if pebble milling is to be used, the pilot-plant crushing tests would include a detailed investigation of the availability and quality of an adequate supply of pebble.

To prevent oxidation of the ore, dry crushing plants are invariably installed. In this event, efficient dust extraction can develop into a major problem. The vast quantities of dust generated usually contain significant quantities of valuable minerals, which often exhibit flotation characteristics different from those of the final crushed product. A decision must be made during the pilot-plant tests as to where the recovered crusher-plant dust will be introduced into the milling or flotation circuits.

Stockpiles must be provided for the blending of the ore from the mine to even out the variations in head grade and mineralogical characteristics. However, if the ore remains on the stockpiles too long, oxidation results in mineral losses during flotation.

Milling and Classification

The key objective in the design of milling and classification circuits is to ensure that the flotation plant will recover the maximum amount of minerals in the appropriate concentrates at the highest grades and lowest costs.

Mills and Classifiers

Fig. 1 shows the general curves for the flotation recovery of zinc and copper of various particle sizes at Prieska, and Fig. 2 shows the particle-size distribution of the feed to the flotation plant. The milling and classification design in this case must aim at producing the maximum amount of particles in the range between

<table>
<thead>
<tr>
<th>TABLE II</th>
<th>SOLUBLE COPPER AND ZINC IN PLANT SOLUTIONS AT PRIESKA WHEN CYANIDE AND ZINC SULPHATE WERE USED AS SPHALERITE DEPRESSANTS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Plant solution</td>
<td>Soluble Cu (p.p.m.)</td>
</tr>
<tr>
<td>Rod-mill effluent</td>
<td>0 to 74</td>
</tr>
<tr>
<td>Flotation feed</td>
<td>0 to 67</td>
</tr>
<tr>
<td>Copper-cleaner tailings</td>
<td>1 to 81</td>
</tr>
<tr>
<td>Overflow from the copper-concentrate thickener</td>
<td>3 to 900</td>
</tr>
<tr>
<td>Overflow from the zinc-concentrate thickener</td>
<td>1 to 89</td>
</tr>
<tr>
<td>Overflow from the tailings thickener</td>
<td>1 to 100</td>
</tr>
</tbody>
</table>
70 and 10 \mu m. The slope of the curve in Fig. 2 should be as steep as possible between 10 and 70 \mu m.

In many leaching processes, the best recoveries are obtained when the ore is ground as finely as possible. This narrow particle-size distribution can best be obtained from multi-stage milling, and not from single-stage milling (including autogenous milling).

Cyclones are universally used in milling circuits for classification because of their low floor-area requirements, simple operation, rapid adjustments of variables and responses, and low capital and operating costs. However, cyclones are notorious for their low classification efficiencies and vague particle-size cut-off points. The latest tendency in the design of milling plants is to install two-stage classification, i.e. the cyclone classifier is followed by a highly efficient classifier such as a Derrick screen or a hydraulic cone-type classifier.

A quick interpretation of the curves in Figs. 1 and 2 may lead to the conclusion that, even if the size distribution of the feed cannot be narrowed by improved classification (i.e., the curve in Fig. 2 steepened), at least it should be moved to the left by overall finer milling since, as Fig. 1 shows, the mineral losses are concentrated in the coarser fractions. This interpretation would not be correct because the curves do not tell the whole story.

The curve showing zinc recovery in Fig. 1 is based on the zine content of the feed to the zinc flotation, and not on the zinc content of the feed to the prior copper flotation. Thus, it does not reflect zinc losses to the copper concentrate. The zinc losses to the copper concentrate, i.e. the sphalerite that is not depressed in the copper flotation, come from the lower end of the fine sizes of the sphalerite. The finer the sphalerite, the greater is the loss to the copper concentrate. The zine reporting to the copper concentrate is a monetary loss, and there is always the risk of exceeding the maximum zine content specified for the copper concentrate.

In some ores, the copper, zine, and other sulphides are so finely disseminated and intimately associated that the grind required to liberate the minerals is so fine that low recoveries are unavoidable.

These problems concerning the liberation and separation of the minerals, and the concomitant difficulties in flotation recovery and limitations arising in the main milling circuit, can sometimes be overcome by the installation of additional grinding stages within the flotation circuit for the further milling of intermediate flotation products to improve the mineral separations, recoveries, and concentrate grades. These additional milling and classification stages can add significantly to the capital and operating costs of the concentrator. It is sometimes possible to avoid these additional costs by returning some of the intermediate flotation products to
the main milling circuit for further grinding. This is done at Prieska, as shown in the condensed flowsheet given in Fig. 3. Copper scavenger concentrate is returned to the inlet of the secondary pebble mills, replacing part of the normal mill-inlet dilution water, and the copper cleaner tailings are returned to the secondary pebble-mill effluents and from there to the closed-circuit cyclones.

When the Prieska plant was commissioned, intermediate flotation products were not milled. Obviously, mill capacities should be designed for such an eventuality. Intermediate zinc-flotation products cannot be returned to the main milling circuit because the copper sulphate used to activate the sphalerite would inhibit copper-zinc separation, i.e. it would activate the sphalerite in the copper flotation. Where, for example, sphalerite is in exsolution with chalcopyrite, no amount of milling will improve the copper-zinc separation.

Variability, Milling, and Classification of Ore

In the designing of a milling and classification plant, cognizance must be taken of variations in the ore from the mine. This has been of particular importance at Prieska Copper Mines, where the pyrite content of the ore can vary from a few per cent to 60 per cent, and the dry relative density from approximately 3.2 to 4.2.

The grindability of the ore can thus vary widely, and variations in dry relative density affect the classification of the minerals, resulting in over- and under-grinding. The wide variation in dry relative densities of the minerals in the ore (chalcopyrite 4.2, galena 7.5, pyrite 5.0, sphalerite 4.0, and gangue 2.7 to 3.1) must be recognized in the design and operation of a plant.

Selection of Mills and Grinding Media

It is now universally recognized that the grinding medium used in mills can have a profound effect on the subsequent flotation performance, particularly with mixed sulphides. The theories involved, and these are many, are outside the scope of this presentation. What is important is that the metallurgist must be aware of this effect and that the same grinding medium apparently has different effects on different ores.

Petruk and Hughson report an improvement in the recovery of zinc when a change is made from pebble to steel-ball milling. In two successive statistically controlled plant-scale tests on the addition of steel balls to the pebble mills at Prieska, the mass of zinc reporting to the copper concentrate was reduced by 0.3 per cent, but the recovery of zinc in the zinc concentrate was also reduced by 2.4 per cent. In each test, a 20t load of steel balls was added to each of the two pebble mills of one unit with a daily steel addition of 1.5 t per mill over a 21-day period.

At least one mine is known to use iron compounds to depress the sphalerite present.

Facilities for Plant Experimentation

It is appropriate at this stage to make a few comments about the facilities required for controlled plant experimentation relative to plant design.

On a plant treating ore containing mixed sulphides, particularly if the ore is variable, process-development changes and 'trouble shooting' are as routine as the plant operation itself. Even if all the many process variables could be controlled, particularly during plant experimentation, there would never be certainty about whether any change in the performance was the result of a change in a control variable or a manifestation of a change in the ore if the latter could not be evaluated rapidly and separately.

It is therefore essential that the plant should be designed with at least two identical units so that one unit can be used as a control during plant experimentation on the second unit. The condensed flowsheet of the Prieska plant given in Fig. 3 is that for each of two identical, separate units. In addition, each unit has its own separate thickeners for the copper and zinc concentrates and for the intermediate flotation products.

Sampling Arrangements

The planning and design of the initial and future sampling arrangements on a plant must be worked out in great detail, particularly if an on-line analyser with a custom-designed sampling system is not to be used from the plant-commissioning stage. Sampling arrangements must not be added to a plant design as an afterthought, but must be fully integrated in each step of the plant design. Thus the design should allow for the subsequent installation of an on-line analysis system with the latter's own particular sampling arrangements without requiring major alterations to pipelines and pumps.

It can be accepted that, sooner or later, practically every pulp, concentrate, and intermediate-product stream on a complex differential-flotation plant is going to be sampled for a sufficient length of time to warrant the installation of automatic samplers. If an on-line analyser is not installed, it is reasonably certain that two simultaneous samples are going to be required from all the important circuit streams. One sample could be an eight-hour composite for accurate analysis and reporting of the results achieved, and the second could be a separate sample taken every half an hour to two hours for rapid and less-accurate analyses that would be used for continuous control of the plant.

In a single mineral-flotation process, it is often not important to return the reject stream from an automatic sampler to the stream that was sampled. In a differential-flotation process, with its sensitivity to 'foreign' reagents at most stages of the process, it is sound practice always to return the reject stream from the sampler to the original stream.

One of the most important plant-performance measurements is that of mineral recovery. It is highly desirable that the recovery should be based on a sampled plant feed, and not on a plant-feed value calculated and reconstructed from concentrates and tailings. When intermediate flotation products are returned to the main milling circuit, it is not unusual to find that the only point for sampling of the new feed to the flotation plant is the new feed to the primary mill. This always provides an inaccurate sample, and the problem is compounded if pebble milling is employed in that the pebbles also constitute new feed.

Flotation Plant

Flotation Circuit

The circuit required in a flotation plant is largely
determined during laboratory and pilot-plant testing. It is not possible to generalize, the circuits being highly dependent on the nature of the ore.

The circuit is often finally decided only after the plant has been commissioned. This applies particularly to the cleaning circuits. It is essential that a plant should be designed with great flexibility as to the destination of the intermediate flotation products. In any event, such flexibility is required when the ore is expected to vary frequently. Again, this applies particularly to the cleaning circuits. Over the last few years the design of cleaning circuits has moved rapidly away from the system having counter-current in-line stages, and there are many new and sophisticated concepts. Lindgren and Broman6, among others, have published a very pertinent summary of some of these approaches.

Types and Sizes of Flotation Cells

Five types of flotation cells have been tested at Prieska. When there is no settlement of solids in the tank of the cell and the air-dispersion capacity of the mechanism is sufficient, no significant differences in metallurgical performance were detected between the different types of cells.

Provided the foregoing conditions are met, the selection of cells can be safely made on considerations of capital and operating costs and the availability of space. The larger the flotation cell, the lower the capital and operating costs per ton of solids throughput, and the simpler the overall control of the flotation process whether automatic or manual.

The size of a flotation cell is limited by the fact that a minimum number of cells must be installed in any one bank to reduce short-circuiting of the pulp. It is usual to install at least two of the large cells or six of the smaller cells. When large cells are used, the circuit is less flexible, and this, again, is particularly relevant in the cleaning circuits.

If a very large mass of concentrate is to be floated from the feed it is possible to reach a situation in a large, deep cell where the surface area of the cell is insufficient relative to its volume for the efficient and rapid removal of the concentrate. One mining group7 uses a maximum of 1 to 3 t of dry concentrate per hour per square metre of cell area in deciding on the size of the cells to be installed for the flotation of copper and zinc.

Residence Times

In the flotation of all minerals, if the residence times are not sufficient, the metallurgical performance deteriorates. Conversely, the surfaces of sulphide minerals tend to become tarnished from long agitation periods and, when soluble salts are present, can collect a coating of precipitated salts. Both these effects tend to reduce or inhibit the flotation of copper sulphides, and, as a result, the flotation selectivity between the sulphides of copper, zinc, and iron is reduced.

At Prieska it was found necessary to remove the conditioners prior to copper and zinc flotation, and also to stop the thickening of the copper and zinc cleaner tailings, which was used to de-water these streams of low relative density before they were returned to the respective roughing circuits.

A major problem in the design of flotation circuits is the transposition of residence times obtained in laboratory batch tests to continuous plant operation, bearing in mind that, in the laboratory, each particle has a residence time equal to the stipulated time of the test.

For many flotation plants, the optimum residence time obtained in laboratory tests was transposed to the plant after the laboratory time had been multiplied by an arbitrary factor, and the nominal volume of the plant flotation cell as stipulated by the supplier of the cell was used in the calculation. These arbitrary factors appear to vary randomly from plant to plant.

The scale-up of laboratory residence times must take the following two major aspects into consideration.

(a) The nominal volume of a flotation cell is invariably the volume of the tank of the cell and bears no relationship to the actual volume available for the pulp. From this nominal volume must be subtracted the volumes of the mechanism below the lip of the concentrate overflow, the froth on top of the pulp, and the air rising through the pulp before the effective volume of the cell is obtained. Measurements at Prieska have shown the following effective pulp volumes as percentages of nominal volumes:

- 3.2 m³ rougher cells: 81.9 per cent
- 1.4 m³ cleaning cells: 63.6 per cent.

(b) With continuous flow through a series of flotation cells on a plant, there is a spread, often quite considerable, in the retention times of different unit volumes or particles due to short-circuiting.

Many determinations of residence-time distribution have been carried out on all the flotation circuits at Prieska using chemical tracers on the assumption that flotation cells can be regarded as back-mix flow or constant flow stirred tank reactors. This assumption is obviously not strictly correct because a back-mix flow reactor is a vessel in which the contents are uniform throughout, and under ideal conditions the exit stream has the same composition as the material within the reactor. However, the errors introduced by this assumption are not large enough to obscure the general trends of the residence-time distributions, and the distributions obtained are sufficiently accurate for the purposes of plant and process diagnoses and development.

Fig. 4 is a typical example of the residence-time distribution in the zinc rougher and scavenger flotation cells at Prieska.
distributions obtained at Prieska. It shows a residence-time range in the zinc roughers and scavengers of from 13 to 30 minutes.

**Water Supply, Reticulation, and Spillage**

Apart from the general quality of the fresh water used in a plant, which was mentioned earlier, a high chloride content can have a direct effect on flotation performance. For example, it can result in a high chloride content in the final zinc concentrate from the solution remaining in the concentrate filter cake, fouling the concentrate to such an extent that it is unmarketable.

To prevent the activation of sphalerite in the copper circuit, it is usual for copper-concentrate thickener overflow and filtrate to be used as spray and dilution water only in the copper-cleaning circuits. The same considerations would apply in the zinc-floatation circuit.

Similarly, plant spillage must be contained in areas separated according to the origin of the spillage, and must be returned to appropriate points in the circuit so as not to affect the efficiency of the mineral separations.

**Preparation and Feeding of Reagents**

The reagent-preparation section must be as meticulously planned as any other section of the plant. The storage area must be spacious so that there will be no difficulty in using the reagents in sequence according to age. A few spare reagent-mixing tanks should be available for testwork.

For reagents that do not age, the mixing tanks should hold at least a week's supply. Where mixed sulphides are to be separated by the control of collector additions at starvation limit, it must be remembered that collector concentrations of about 0.75 to 1.5 per cent will probably be used. These low concentrations require large mixing tanks.

Accurate and constant additions of reagents are critical in differential flotation if optimum separations are to be achieved, and no expense should therefore be spared in the acquisition of the best metering pumps available. As most of the reagents would be extensively stage added, the design should allow for this.

**THE FLotation PROCESS**

It is unlikely that two differential sulphide-flotation plants would use the same separation process. This is understandable since it is unlikely that two deposits of mixed sulphides would have identical mineralogical characteristics. There is therefore not much purpose in dissecting one particular process in detail in a general paper of this nature, and only a brief description will be given of the current and previous flotation processes used at Prieska, followed by a few general remarks about differential flotation.

**Differential Flotation at Prieska**

Copper and zinc were originally separated by the depression of the sphalerite in the copper circuit with Zn(CN)₂ at a pH value of 6.2 to 7.2, which was controlled with sulphuric acid. The Zn(CN)₂ slurry was obtained by premixing cyanide and zinc sulphate. The sphalerite was then re-activated with copper sulphate and floated at natural pH, which had an approximate value of 7.4.

The Prieska ore-body contains small quantities of galena, which floats readily with the chalcopyrite, resulting in an unacceptable high lead content in the final copper concentrate. This concentrate was therefore pumped to a separate flotation section, where the chalcopyrite was depressed with large quantities of cyanide and zinc sulphate, and the galena was removed in a low-grade lead concentrate containing significant quantities of copper.

This process was never completely satisfactory, and further plant-scale testwork showed the following.

1. The chalcopyrite is amenable to flotation at pH values up to 10.8.
2. Inexplicably, the galena does not float in the copper circuit between a pH value of 10.2 and 10.4, Below and above that pH range, the galena floats readily.
3. The sphalerite, after being activated with copper sulphate, is amenable to flotation in the pH range 8.5 to 11.0.

The flotation circuit for the current process is shown in Fig. 3. The chalcopyrite is floated at a pH value of between 10.2 and 10.4 (controlled accurately with lime), with isopropyl xanthate, disubutyl dithiophosphate, and TEB frother. Collectors are stage-added in starvation quantities to prevent excessive flotation of the zinc. The sphalerite is re-activated with copper sulphate and floated at the natural (approximately 9.2) pH value with isopropyl xanthate and TEB frother. Copper sulphate and the collector are again added in starvation quantities to prevent excessive flotation of the pyrite.

**Depression Conditions in the Copper Circuit**

If over-depression, which invariably results in copper floating in the zinc circuit, is to be avoided, the mildest and most sensitive conditions of zinc depression must be maintained in the copper circuit. This is far more effectively achieved by control of the collector than by the addition of a specific chemical depressant. However, this system requires continuous and constant vigilance with 'knife-edge' control. One plant has even reported that adjustments had to be made to flotation conditions when the lime used came from a different source.

The fine, light-coloured, fast-floating sphalerite is always first to float in the copper circuit, while the coarse chalcopyrite particles are always the first to escape to the zinc-flotation circuit.

**Ambient Temperatures**

At Prieska, the flotation rate of the sphalerite increases significantly faster with increasing temperature than does that of the chalcopyrite.

The copper–zinc separation is decidedly more sensitive in mid-summer than in mid-winter, and to date there is a tendency for the zinc content of the copper concentrate to be slightly higher during the summer months.

A number of mines have reported increased consumption of depressants during the summer months.

**Pre-flotation Aeration**

There seems to be no pattern in the mines that have reported that pre-flotation aeration aids in the depression of pyrite and pyrrhotite and in those that have found no resulting benefit.

Although the Noranda mines have been practising
TABLE III

<table>
<thead>
<tr>
<th>Product</th>
<th>Dissolved oxygen p.p.m.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rod-mill discharge</td>
<td>2.0</td>
</tr>
<tr>
<td>Cyclone underflow</td>
<td>6.3</td>
</tr>
<tr>
<td>Cyclone overflow</td>
<td>6.6</td>
</tr>
<tr>
<td>Copper-flotation feed</td>
<td>5.6</td>
</tr>
<tr>
<td>Copper-savenger tailings</td>
<td>8.0</td>
</tr>
<tr>
<td>Zinc-scavenger tailings</td>
<td>5.0</td>
</tr>
</tbody>
</table>

Dissolved Oxygen in the Prieska Circuit

The maintenance of constant product volumes in the flotation circuit, froth depths in the cells, and reagent additions relative to the assays of the various products. The operator, at the push of a button, can recall the trend of any of the foregoing parameters in graphical form on a screen for the previous four to six hours. Every seven minutes he receives an instantaneous printout of all these parameters, including all the circuit assays and recoveries. The optimizer is used to develop relationships between the various control parameters and assays, recoveries, concentrate grades, etc., from the routine information retained in the memory circuits of the system.

The cost of such a system including automatic samplers, pumps, piping, and control rooms for a concentrator of the size of Prieska would be between two and three million rand. Quite apart from the economics of such a control system, a concentrator contemplating its installation would have to be certain of procuring and retaining instrument engineers and technicians with sufficient expertise to guarantee continuous maintenance and operation.

An isotope-probe system is cheaper than an X-ray analyser, but, when background counts from, for example, iron sulphides vary over wide limits as at Prieska, the isotope-probe analyses are not accurate.

Relative Densities and Froth Characteristics

A great deal has been written over the years about the practical ‘rules’ of flotation, e.g. low relative densities for the flotation of coarse particles, high densities for fine particles, deep watery froths to ensure high drainage rates for the improvement of concentrate grades, etc. Very few of these so-called ‘rules’ have applied to the flotation at Prieska, or have been of any real use.

The density of the flotation-feed at Prieska was reduced progressively from 40 per cent solids to the present 30 per cent solids, at which the recoveries are at a maximum, but the concentrate grades remained unaffected. It is considered that the improved recoveries are more a function of the concomitantly lower residence times rather than of the lower solids concentric-sensitive to variations in froth depth. In fact, in the third- and fourth-stage cleaning circuits concentrate grades have reacted favourably to froth crowding from high circulating loads, which automatically result in high densities; and circulating loads (and densities) in the first-stage cleaning circuits have been kept low in order to limit the return of minerals to the roughing circuits.

PROCESS CONTROL

Once the flotation process has been established, metallurgical efficiencies can best be improved by control of the flotation circuit and process in a steady-state condition.

On-Line Analysis

Probably the most effective system available is the Courier on-line X-ray analyser with a computerized Proscon 103 controller and Proscon 105 optimizer. This system automates, inter alia, the new feed to the primary mills, the addition of pebbles, the sampling and assaying, the circuit density, the air rate to the cells, and the
system that designates unacceptably large errors as either analytical errors in the laboratory or sampling errors in the plant.

Mineralogical Services
Each morning the Prieska concentrator receives a return from the mining department detailing the stope numbers and quantities of ore drawn from each of these stopes for the previous twenty-four hours. Data on the mineralogy and concomitant flotation behaviour of the ores from the various stopes are being collected on a continuous basis.

Although the system has many limitations and still requires a great deal of refinement, it is a valuable aid to the metallurgists in their interpretation of varying flotation behaviour in the plant, and anticipation and execution of pertinent flotation strategies.

Mathematical Modelling
For the purposes of this discussion, the following two terms are defined as follows:

mathematical modelling is the development of an accurate theoretical description of the process operation and is valid over the entire operating range of the process;

empirical modelling is based on statistical procedures applied to past results from the process operation, and is valid only over the range that the process variables are studied at a particular time.

Many concentrators do not have sufficient local expertise for the development of mathematical models, but most concentrators have sufficient local expertise for the competent development of empirical models. Each concentrator has its own particular economic relationships between concentrate grades and recoveries for each mineral, and inter-relationships between the different minerals for these factors.

Financial factors beyond the control of the metallurgist that would affect these relationships include metal prices, smelting charges, transportation costs, foreign exchange rates, etc.

At plants treating mixed sulphides, concentrate variables that are beyond the control of the metallurgist, and that could influence the recoveries and concentrate grades, could include the concentrations of copper and zinc in the feed, the ratio of zinc to copper in the feed, the total content of iron sulphide in the feed, ambient temperatures, etc. Because of all the variables involved, it is impossible for any concentrator manager to study the recoveries and grades for one shift or one day, and decide whether or not optimum performance relative to financial gain had been achieved under the particular conditions pertaining for the period.

The solution to this problem lies in the development of an empirical model that defines, in money terms, the optimum concentrator performance for a particular set of input data. The metallurgical results for each shift or each day can then, after being converted to money values, be compared with the model's prediction of optimum performance as an efficiency factor in percentage terms.

The control system can be operated with ease by the use of a pocket or small desk-top computer.

With the use of an empirical model based on the statistical processing of past results, it is essential that the model is frequently updated. If a mathematical model can be developed from fundamentals, all the better.

PROCESS DEVELOPMENT ON PLANT SCALE
The most convenient and positive method of performing full plant-scale experimentation for process-development purposes is to keep all the variables constant except that under investigation. In differential flotation, it is impossible to keep the many variables constant during an extended plant-scale experiment without computerized automatic control. Also, 'the one variable at a time' approach does not allow the determination of interactions between variables and also the process error.

Variable interactive effects occur frequently in the flotation of mixed sulphides, and, if incorrect conclusions, control adjustments, and process changes during process development are going to be avoided, it is essential that plant-scale testing should be carried out according to the fundamental concepts of the factorial design and analysis of industrial experimentation. Only in this way will reliable estimates be obtained of the coefficients of the various variables. It is equally important that assessments should be obtained of the statistical significance of each variable coefficient independently of the others.

With this tool, the flotation metallurgist can move out of the grey area he so often finds himself in when trying to evaluate changes in process-control variables.

HUMAN RESOURCES
For the optimum operation of a concentrator operating on complex mixed sulphides, human resources are required in three specific metallurgical areas.

Line Metallurgists
The senior line metallurgist, usually the concentrator manager, is responsible and accountable for the total operation of the concentrator, and specifically for the extent to which the required output is achieved. In addition to the necessary technical qualifications, skills, and experience, he must receive a broad foundation in behavioural science and intensive training in management techniques (such as employee selection, professional management, negotiating skills, decision-making relative to promotions and salary reviews, formulation of employee development contracts for employee training and advancement).

Staff Metallurgists
The main functions of the staff metallurgist are to develop new knowledge from investigations and continuing analyses of the data generated daily by the concentrator operations. He must also co-ordinate the information from other departments such as the analyses of the sources of the stope ore received from the mining department.

In a simple, small metallurgical operation, the line metallurgists can often fulfil the functions of the staff metallurgist as well. However, with a complex metallurgical operation such as the differential flotation of mixed sulphides, it is essential to have a team of staff metallurgists and operations research metallurgists on the concentrator, together with the necessary laboratory and small pilot-plant facilities.
In addition to their basic metallurgical qualifications, the staff would be engaged in, or have completed, postgraduate studies in matrix algebra, statistics with special reference to multiple regression analysis, design and analysis of experiments, and operations research including computer programming.

Fundamental Research Chemists and Metallurgists

Reddin\textsuperscript{10} has some interesting thoughts on the application of theory management, which are also applicable to a scientific or industrial situation. He states that there is nothing more practical than a good theory properly applied, or so costly as a poor theory inappropriately applied. A good or sound theory will show relationships not previously identified that will lead to actions that would not have otherwise been considered.

No attempt is made here to relate fundamental theory to the plant data, operations, and observations already discussed. Fundamental theory played an important role in the process development at Prieska, particularly in the departure from differential flotation using specific depressants.

A limited number of industrial and mining organizations employ fundamental research chemists as opposed to applied research chemists. This is not a real problem, and, at the height of the feverish process development at Prieska, metallurgists regularly joined the group of specialized surface chemists of the National Institute for Metallurgy, consisting of Drs N. P. Finkelstein, P. J. Harris, S. A. Allison, and V. M. Lovell, in brainstorming sessions. These theory 'think tanks' led to process actions on the plant that would otherwise not have been considered.

THE FUTURE

The efficient design and operation of a modern, complex mineral-processing plant such as that operating on mixed sulphides increasingly require the effective co-ordination by the mineral-processing engineer of a host of disciplines. These disciplines include geology, mineralogy, fundamental and applied chemistry and metallurgy, control instrumentation, operations research and datametrics, management information systems, etc. The mineral-processing engineer cannot be an expert in all these disciplines, but, to communicate and coordinate effectively, he must have a working knowledge of most of them. Above all, he must have the expertise and ability to manage people and situations, apart from the technology.

These peripheral factors are becoming increasingly significant as the degree of sophistication of the technology increases. Industry is inclined to pay too much attention to the technology of the process itself, and too little to the important peripheral factors.

ACKNOWLEDGEMENTS

The author is indebted to all the Anglovaal metallurgists who contributed to the process development at Prieska. He thanks the Technical Director of Anglovaal for permission to publish this paper.

REFERENCES

2. Private visit to copper-zinc concentrators in Finland, Sweden, and Norway.
5. Private visit to Kipushi Mine, Zaire.

Fluid power

BHRA Fluid Engineering will hold its Sixth International Fluid Power Symposium at St John’s College, Cambridge, 8th to 10th April, 1981. This follows the highly successful meeting in Durham in 1978.

The rising costs of labour and materials are creating a need for more efficient and reliable machinery throughout the industry. The correct design and application of fluid-power systems can reduce fuel consumption and improve performance. In addition to the more traditional applications in transport and construction, fluid power is now being used widely in new areas such as the offshore industry. Designers and users need to be aware of increasing health and safety requirements, and of new ideas and developments in hydraulics and pneumatics. The Symposium should provide an ideal opportunity for leading authorities from academic institutions and industry to discuss both theoretical and practical aspects of fluid power.

The papers will deal with, or be related to, the subjects listed below.

All aspects of research and development work, basic theory and experimentation, and applications of oil hydraulics and pneumatics in:

- Control systems/Servo systems
- Filtration/Contamination
- Noise
- Pumps and Motors
- Transmission Systems
- Valves
- Instrumentation and Testing
- Circuit/System Design
- Seals in Fluid Power

Enquiries should be addressed to Symposium Organiser, 6th IFPS, BHRA Fluid Engineering, Cranfield, Bedford, MK43 OAJ, England. Tel: Bedford (0234) 750422, Telex: 825059.