J. S. Afr. Inst. Min. Metall., vol. 90, no. 6.
Jun. 1990. pp. 141-147.

Economic leaching at Rössing Uranium Limited*

by C.C. JOHNSON†

SYNOPSIS

In the fourteen years that Rössing Uranium Limited has been in production, the metallurgical operations and controls on the leaching plant have evolved through four basic stages.

Initially, the emphasis was placed on the achievement of consistent plant operation by overcoming severe start-up difficulties. The second stage involved the attainment of the design operating targets, and also the commissioning of the ferric-leaching reactors in order to achieve a ferric ion concentration of more than 3,0 g/l. Improvements in control then became the priority, with the emphasis on consistently achieving the target concentrations of ferric ions, total iron, and terminal acidity. The latest phase has concentrated on the optimization of costs by means of adjustments to the historically established operating parameters in order to achieve large savings on consumables while maintaining the leaching efficiencies.

Apart from the obvious incentive of reducing costs in an inflationary economy and a depressed uranium market, impetus for this work was given by a change in the type of ore from the open pit, which has the effect of reducing the extraction efficiency while increasing the consumption of consumable materials.

These problems and their solutions are discussed in detail, and the importance to cost-effective optimization of an accurate, up-to-date cost-reporting structure is stressed.

SAMEVATTING

In die veertien jaar wat Rössing Uranium Limited in produksie is, het die metallurgiese werksaamhede en beheer in die loogaanleg deur vier basiese stadiums ontwikkel.

Aanvanklik het die klem geval op die daarstelling van konstante aanlegwerking deur ernstige inbedryfstelprobleme te oorkom. Die tweede stadium het die bereiking van die ontwerpbedryfsteikens en die inbedryfstelling van die ferriloogreaktors om 'n ferri-ioonkonsentrasie van meer as 3,0 g/l te kry, behels. Daarna het verbeterings aan die beheer 'n prioriteit geword met die klem op die konstante bereiking van die teikenkonsentrasies van ferri-ione, totale yster en eindsuurheid. Die laaste fase was toegespits op die optimering van die koste deur middel van aansuiwerings van die histories bepaalde bedryfsparameters ten einde groot besparings ten opsigte van verbruiksgoedere te bewerkstellig en tog die loogrendemente te handhaaf.

Afgesien van die vanselfsprekende aansporing om koste in 'n inflasionistiese ekonomie en 'n bedrukte uraanmark te verlaag, het 'n verandering in die soort erts wat uit die oop groef verkry is, en wat 'n verlaging van die reduksierendement en 'n verhoging in die verbruik van verbruiksgoedere tot gevolg gehad het, 'n stoot aan hierdie werk gegee.

Hierdie probleme, en die oplossing daarvan, word in besonderhede bespreek en die belangrikheid van 'n akkurate, bygewerkte kosteverslagstruktuur vir kostedoeltreffende optimering word benadruk.

Introduction

Economic leaching is the term used to describe the cost optimization of a leaching process. With nearly every leaching operation, there is a trade-off between extraction efficiency on the one hand and consumable and operating costs on the other, and the law of diminishing returns is usually applicable.

However, in a truly optimized leaching operation, many factors play a part and several hurdles have to be overcome. In an expanding market, where profitability is largely governed by maximum production, the maximization of recovery tends to be the criterion on which a plant is operated. Owing to the pressures of achieving production targets and meeting order commitments, this criterion is also the aim when a plant begins its operational life. Although all operations experience the start-up situation, few in the base-metals industry are fortunate enough to experience an expanding market situation for an extended period. In the circumstances where neither of these situations pertains, economic leaching is an important route to reducing production costs, and hence increasing the profitability and company viability. This is particularly important to Rössing because of the high cost of consumables resulting from the remote location of the mine.

Plant Start-up

During the start-up at Rössing Uranium Limited, many problems arose that were not fully anticipated in the design. Many wear problems were experienced owing to the extreme abrasiveness of the ore, and the fine-crushing plant had to be upgraded from a two-stage open-circuit plant to a three-stage closed-circuit plant. The leach agitators and gearboxes had to be uprated owing to the inability of the installed equipment to adequately suspend the slurry, and four additional thickeners of 68 m diameter were installed to cope with the poor settling characteristics of the ore. In 1978, half the solvent-extraction plant was destroyed by fire, which further added to the start-up difficulties. During this period, there was intense pressure to maximize the extraction so that the production commitments could be met.

Establishment of Operating Parameters

Testwork carried out at Mintek had identified the im-

* Paper presented at the Leaching Colloquium held by The South African Institute of Mining and Metallurgy in November 1988.

† Rössing Uranium Limited, Private Bag X5005, Swakopmund, 9000 Namibia.

portance of ferric iron and temperature in achieving the maximum extraction. The testwork had also identified that there was a significant difference in leaching characteristics between the two major series of ores to be mined: the Nosib and Hakos series (Fig. 1). In general terms, much higher extractions were achievable from the Nosib ore than from the Hakos ore under similar leaching conditions. In the early years of the mine, the ore was predominantly from the Nosib series, and it is only in the last three to four years that significant proportions of Hakos material have been present in the feed.

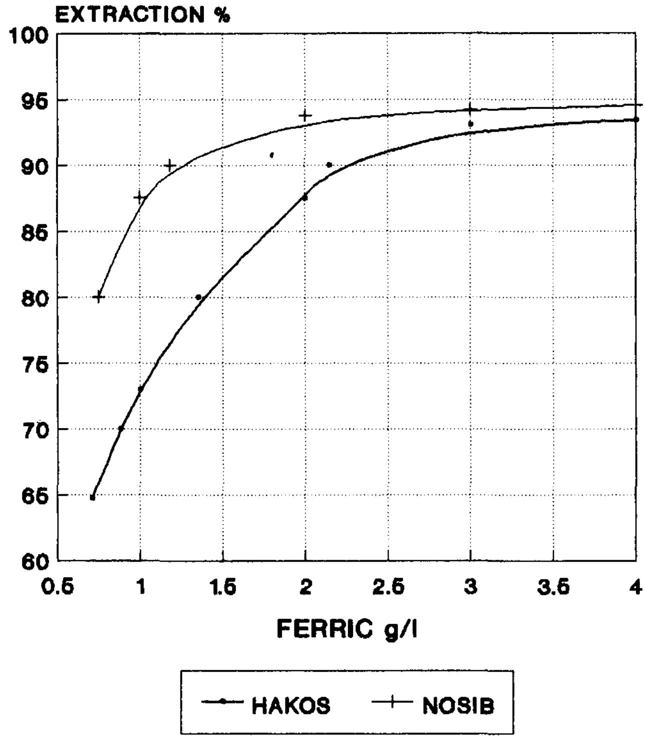

Fig. 1—The effect of Hakos and Nosib ores on the extraction (results obtained by Mintek)

The importance of temperature and ferric-ion concentration in the extraction of uranium was recognized in the early days of the operation, and efforts were made to maximize these parameters. Temperature was tackled first with the re-commissioning of the rod-mill heat-exchangers, which had been bypassed owing to mechanical and corrosion problems and, by the end of 1977, temperatures consistently higher than 40°C, and rising as high as 55°C, were obtained.

From the testwork done at Mintek, it was recognized that a ferric-ion concentration of about 3,0 g/l was required for optimum extraction. However, the ore and grinding media provided only about 2,0 g/l of total iron at best. This sometimes fell as low as 1,0 g/l, and generally averaged 1,5 g/l. Hence, a source of iron, preferably ferric iron, was sought to boost the total iron content of the leach pulp. The solution to this was found to lie in the leaching of acid-plant calcine with strong sulphuric acid (300 to 600 g/l) at a temperature of 110°C. The pilot plant, which was installed in 1980, was upgraded in 1981, and the final, full-scale plant was commissioned in January 1982.

At that stage, it was possible to increase the terminal content of ferric ions to over 4 g/l, and the extraction had risen from 85,2 per cent in 1979 to 90,2 per cent in 1982 (Fig. 2). Extractions were maintained at that level until 1986 and, during that period, the emphasis was on reducing consumable costs by tightening up on process control parameters.

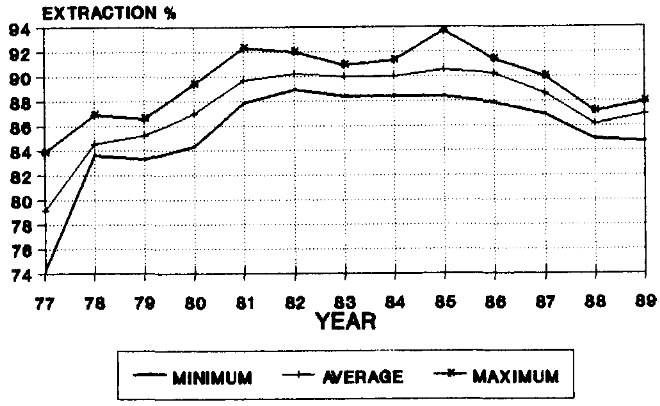

Fig. 2—Leaching extractions from 1977 to 1988

Over the period 1982 to 1985, the acid consumption decreased from a peak of 78 kg of sulphuric acid per kilogram of U_3O_8 to a low of 57 kg (Fig. 3), largely as a result of improved control in both the plant and the open pit, resulting in a reduction in the quantities of high acid-consuming minerals such as calcite being sent for processing.

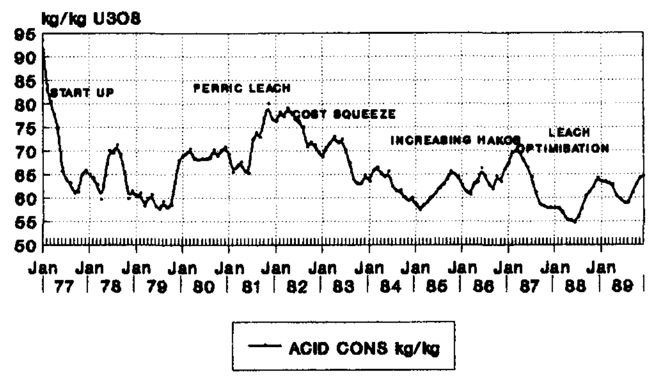

Fig. 3—Acid consumptions from 1976 to 1989 (five-monthly moving averages) in kilograms per kilogram of U_3O_8

The year 1982 saw an increase in the consumption of manganese dioxide, which was largely related to the maintenance of the higher ferric-ion levels in leaching. From 1983 to 1985, the ferric-ion levels were maintained at about 3,3 g/l in the leach discharge, and the consumption of manganese dioxide decreased from 5,1 kg per kilogram of U_3O_8 to a minimum of 2,1 kg (Fig. 4).

It was in 1985 that the proportions of Hakos ore began to increase and, although the extractions were initially maintained at about 90 per cent, the consumption of acid and manganese dioxide began to increase. This trend continued into 1987, by which time the extraction was consistently lower and it was generally accepted that a major reconsideration of the plant operating parameters and

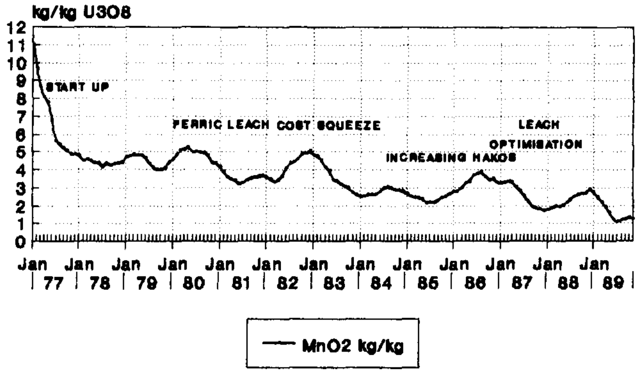

Fig. 4—Manganese consumptions from 1977 to 1988 (five-monthly moving averages) in kilograms per kilogram of U_3O_8

philosophy was required.

Cost-reporting Structure

However, before a process can be optimized, a sound cost-reporting system is required. Rössing was fortunate in that the cost-reporting system was accurate in the short term and required little modification to give meaningful costs on the major consumables on a daily basis. These costs are now routinely produced from the computer data base, and are available for scrutiny by the management first thing in the morning, with the result that both the operational efficiencies and the cost performance can be compared at the production meetings. The only modification that was required was the translation of the cost basis from tonnes of ore to kilograms of U_3O_8. This modification was essential so that the true efficiency of the operation could be judged. For example, on a cost per tonne basis, an operation can look extremely efficient, particularly if the production rate is high. However, if the ore is low in grade, the cost per kilogram of product can grossly exceed the planned cost, with disastrous effects on profitability. Costing on the basis of cost per kilogram draws immediate attention to the causes of poor performance.

Adjustment of Leaching Conditions

At the beginning of 1987, it was recognized that major changes to the operating strategy were necessary to reverse a deteriorating trend that had started in 1985. In March 1987, the extraction dropped to 86,9 per cent, the acid consumption was 74,5 kg of sulphuric acid per kilogram of U_3O_8, and the consumption of manganese dioxide was 3,3 kg per kilogram of U_3O_8, the consumption of acid being 31 per cent higher, and that of manganese dioxide being 55 per cent higher, than their previous minimum values in 1985. The consumption of manganese dioxide had been as high as 4,0 kg per kilogram of U_3O_8 in 1986, and previous optimization work had reduced this by a reducing target amount of ferric ions in the leach discharge from 3,5 to 3,2 g/l.

The leaching plant at Rössing consists of two parallel lines of six leaching tanks in series, each of which is independently metered for acid and manganese dioxide. The acid metering is by means of Mag-flo meters, and the metering of manganese dioxide utilizes both Mag-flo meters and Nuclear densitometers. The two modules can therefore be used in direct comparative tests as long as the tests are run for equal times on the two modules in order to eliminate any metering biases. A programme of tests was implemented in 1987 with the object of maintaining uranium production at the planned levels while significantly reducing the costs of the consumables. The results of the tests are shown in Table I. All the savings quoted take into account any increase in mining operations due to the replacement of ore for lost extraction and, owing to the confidentiality of this information, are expressed in terms of percentage cost savings on acid and manganese.

TABLE I
PLANT TESTS

Test	Objective	Extraction loss %	Acid saving kg/t	Pyrolusite saving kg/t	Effect on costs %
1	Terminal versus control ferric	0,05	0,43	0,35	5,00
2	Optimization of ferric leach	Nil	0,08	0,14	1,44
3	Ferric in control tank reduced from 3,5 to 3,0 g/l	0,33	0,61	0,37	5,00
	Combined effect	0,38	1,12	0,86	11,43
	Actual results for 9 months to Dec. 1987	1,60	1,60	1,04	11,24
4	Reduction in terminal acidity to 2,0 g/l	0,26	1,30	Nil	7,70
	Combined effect	1,86	2,90	1,04	18,94
	Actual results for 6 months to Jun. 1988	3,00	4,60	0,90	24,83

The first test determined whether it was better to operate to a terminal ferric-ion value or to a ferric-ion value in the third tank on line, now known as the control tank. It had become standard operating practice to control the ferric-ion level to achieve a ferric-ion concentration of 3,5 g/l at the leach discharge. In this test, one module was operated with a terminal ferric-ion target of 3,5 g/l, while the other was operated with a control-tank ferric-ion level of 3,5 g/l. The test lasted for two weeks, after which the results and their cost implications were assessed. As a result, use of the control tank was implemented forthwith.

The operation of the ferric-ion leaching plant was then examined. It was known that, the higher the operating temperature of this plant, the higher the recovery of iron and also the higher the ratio of ferric to ferrous iron. However, owing to the increase in Hakos ore, which contains more soluble iron, the ferric-iron leaching plant had been operated largely at a temperature lower than the design temperature in order to control the amount of total iron in the circuit. A test was then conducted to determine the difference in operating cost between operation at 70°C and operation at 100°C on this plant. The benefits resulting from the use of the higher temperature led to the implementation of this operating temperature.

Following these two tests, a test was carried out in which one module was operated at a target ferric-ion content of 3,5 g/l and the other was operated at 3,0 g/l ferric ions in the control tank. Again, this strategy was implemented.

The estimated savings resulting from the implementation of the findings from these three tests were compared with the actual savings realized during the last nine months of 1987, and with the actual performance in 1986. This comparison in also shown in Table I.

Although there could be little doubt at that stage that the leaching operation was significantly more efficient in terms of cost, concern was rightfully expressed that the loss in extraction (1,6 per cent) was significantly higher than expected. The plant tests had indicated an extraction loss of only 0,38 per cent, and the increased loss of 1,22 per cent required explanation. Fortunately for the optimization programme, the answer was found in the routine laboratory tests. As a check on the leaching performance, laboratory leaches had been carried out on a weekly basis from early in the life of the mine. Historically, there had been good agreement between the extractions obtained from these leaches and those obtained from the plant. However, the laboratory tests were conducted under standard conditions and were therefore an indicator of the extractability of the ore.

A comparison of the annual averages for the plant and the laboratory is shown in Table II. It would seem likely that most of the reduction in extraction from 1987 was not related to plant operating conditions but to a change in the type of ore being handled.

TABLE II
LABORATORY AND PLANT EXTRACTIONS

Year	Laboratory %	Plant %
1985	90,6	90,4
1986	90,0	90,2
1987	88,9	88,6
1988	88,0	87,2
1989	88,0	86,9

The leaching temperature was examined in two trials, which unfortunately gave conflicting results from the point of view of cost efficiency. However, both trials showed a significant reduction in extraction at lower temperatures, which was probably only marginally offset by reduced acid consumptions. This aspect, when examined in the laboratory, confirmed the reduction in extraction and indicated little saving in acid consumption. As a result of this, the current plant philosophy is to operate at the maximum temperature that can be achieved, which currently ranges from 45 to 50°C.

Much work had been done in previous years on the effect of retention time, and it had been determined that 8 to 9 hours represented the optimum. However, this parameter is totally dependent upon the conditions under which the plant is operated. Under the original harsh operating conditions, the acid consumption in the later tanks was high enough to more than offset the extra extraction gained, and hence a shorter retention time was beneficial. However, current conditions suggest that this is not the case, and the plant is now operated at the maximum retention time that can be achieved, which is about 10 hours. Clearly, though, this parameter requires regular assessment as other parameters are altered.

The only other leaching parameter that can be controlled is the acid profile and the acid tenor of the leach discharge. In the early days of operation, the plant was operated at a discharge tenor of 2,5 g/l. Over the years, however, this parameter had been increased to 3,2 g/l in the belief that uranyl phosphate precipitated at the lower acid tenor, thus reducing the extraction. This rationale was examined in detail in the laboratory, where it was found that, in the pH range 1,76 to 3,35, no uranium precipitation could be detected from the leach discharge. However, it was found that there was a loss of ferric iron, and it was concluded that this precipitated as ferric phosphate, thus removing the phosphate from the system. This finding did not confirm whether there was any effect on the extraction rate at a lower acid tenor, and a further series of tests was carried out in the laboratory to quantify this effect if it existed. It was concluded from this testwork that there would be a saving of 4,25 kg of sulphuric acid per tonne of ore for an extraction loss of 1,1 per cent if the acid tenor of the discharge were reduced to 1,2 g/l. Furthermore, the extraction loss would be negligible if the acid tenor were maintained above 1,5 g/l.

This saving in acid consumption is far greater than would be expected from the sending of more acid to the tailings owing to the higher tenor. Mineralogical examination and testwork established that the reason for this was the high proportion of minerals in the Hakos ore that continue to consume acid unabated throughout the leaching train. With Nosib ore, the main acid-consuming mineral is calcite, but its reaction rate becomes suppressed during leaching owing to the build-up of an insoluble layer of gypsum. On the basis of this evidence, a plant trial was conducted with one module operated at a terminal acid concentration of 3,2 g/l and the other at 2,2 g/l. These results are also shown in Table I. The plant is now operated at a terminal acid target of 2,0 g/l.

A comparison of the figures at the end of June 1988 with those for 1986 is also shown in Table I. These changes in leaching philosophy achieved a saving of 24,83 per cent on the costs of acid and manganese on the assumption that the extraction loss is directly attributable to the milder leaching conditions.

It must be noted that all the cost savings quoted allow for the mining of extra tonnes of ore to compensate for the reduced extraction, and hence are reflected in the cost balance.

As discussed earlier, the loss in extraction is higher than would be expected, and it is thus of major concern. The laboratory results, however, continued to show that the leaching performance was still comparable with that of a leach conducted under the standard conditions (Fig. 5). However, in view of this concern, a trial was carried out on the plant in which one module was operated at the original 'harsh' conditions and the other at the current 'mild' conditions. No improvement in extraction was seen, but the consumption of acid and pyrolusite increased. The trial was assessed rigorously in a number of different ways, most of which would favour the test under 'harsh' conditions. It was found that, at best, operation under 'harsh' conditions would cost an additional 20,40 per cent to operate, and the most likely additional cost for this exercise was 24,83 per cent for acid and manganese. The results of this test were welcome in that they supported the change in strategy. However, they are worrying from the point of view that the uranium seems to be progressively more difficult to extract from the ore, and it is this problem that is being addressed at the moment.

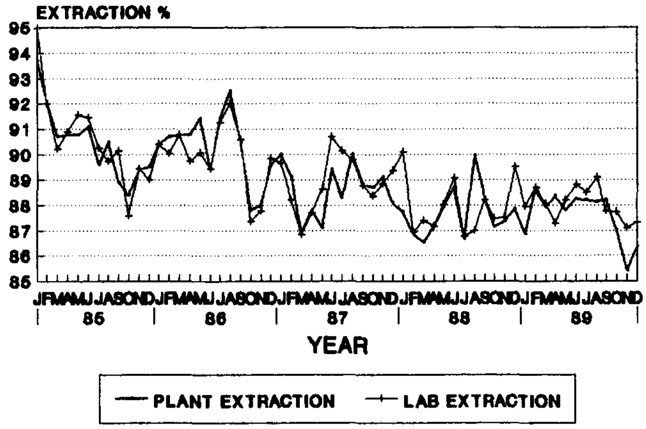

Fig. 5—Monthly extractions in the laboratory and on the plant

Detailed mineralogical examination of the tailings showed that there are several reasons for the lower extractions. The refractory uranium minerals have increased, both betafite and brannerite being present in increased proportions. There is also a strong indication that the liberation size for uraninite is smaller from Hakos than from Nosib ore, which means that the uraninite is not being exposed or is being only partly exposed. A further form of uraninite was found as large, rounded grains closely intergrown with silica and sulphides, generally galena. These grains dissolve slowly, and it is thought that, when they occur, harsher conditions may result in improved extraction. In July 1988, some very poor extractions were obtained on the plant and, in response, extreme conditions were imposed on one module. Higher extractions were achieved on this module, but the benefits obtained as a result of this action were more than offset by the additional acid and pyrolusite consumptions, and the trial was discontinued after five days.

Adjustment of Milling Conditions

One of the major aims for further work is to obtain more cost-effective extraction increases from Hakos ore, although it is likely that any improvements will require capital expenditure. Considerable investigative work has been conducted into the liberation size, and several interesting factors have emerged.

(a) With ore delivered to the plant in 1979/1981, the size distribution of the tailings (d_{50} of 300 μm) was significantly finer than the size distribution of the feed to the leach (d_{50} of 420 μm). This indicates that, at that time, there was significant chemical breakdown of the ore during leaching (Fig. 6). This no longer occurs: the tailings show an almost identical size distribution to that of the leach feed, which is itself almost identical to the 1979/1981 leach feed (Fig. 7). Hence, the total amount of liberation taking place in the Röss-ing process is now significantly less than in previous years.

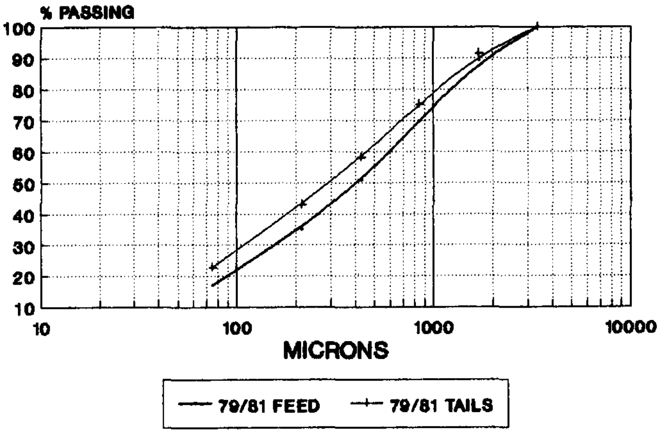

Fig. 6—Particle-size distribution in the feed to the leach and in the tailings, 1979 to 1981

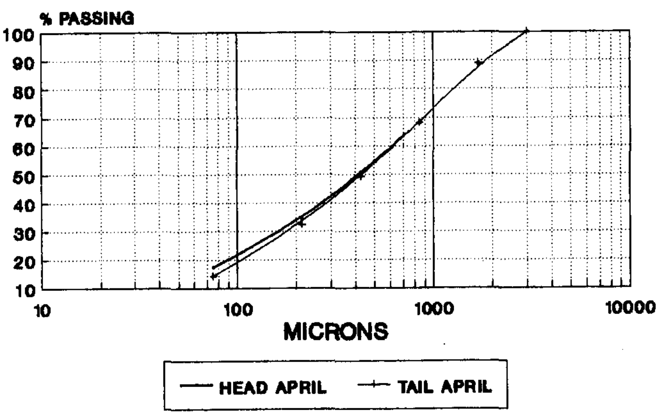

Fig. 7—Particle-size distribution in the head and in the tailings, April 1988

(b) In 1979/1981, some 40 per cent of the uranium delivered to the plant could be found in the minus 45 μm fraction, whereas this fraction has now been measured to be as low as 25 per cent (Fig. 8).

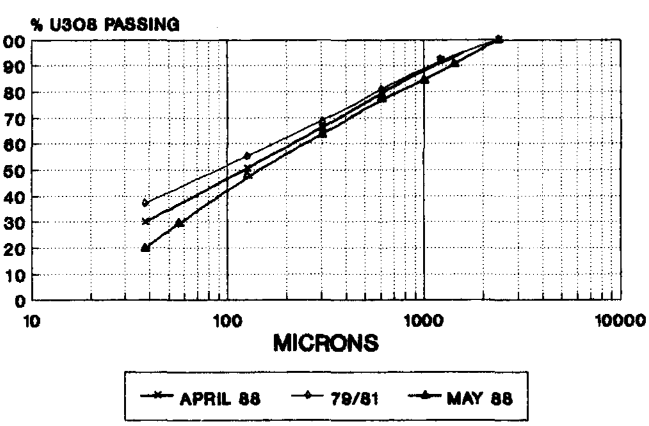

Fig. 8—Particle-size distribution of the uranium as a cumulative percentage of the total uranium

(c) Generally, the extraction efficiency at particular sizes is higher now than in 1979/1981. This is not surprising in the light of the improved control and increased availability of ferric iron (Fig. 9). However, it indicates that the extraction would now be significantly lower had it not been for the development of ferric leaching and improved control.

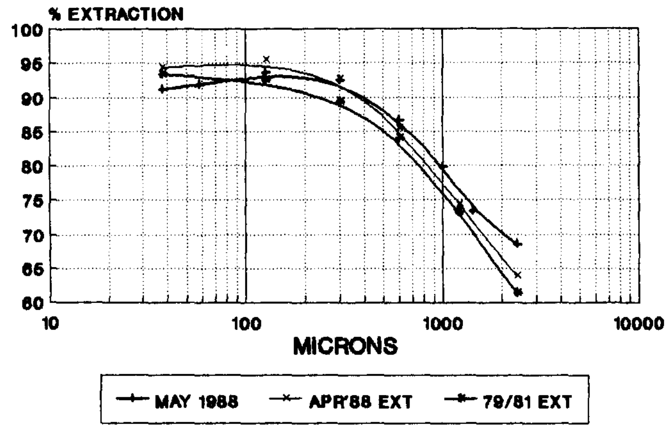

Fig. 9—Extraction efficiency at various particle sizes in 1979/1981, April 1988, and May 1988

The current size-distribution target is to maintain 70 per cent of the leach feed at minus 850 μm (20 mesh). If this could be increased to 75 per cent, there is a potential extraction increase of about 1,6 per cent, which is equivalent to a saving of 8,08 per cent on the costs of acid and manganese. With this in mind, two tests were conducted on the influence of mill control parameters on the size distribution.

Historically, it had been the practice to either add more rods to the mills or decrease the milling rate at times when the grind was coarser than target. These two philosophies were examined under controlled conditions in order to determine the magnitude of their effect.

In the first test, one module was operated at 500 t/h while the other was operated at 450 t/h. The trial was conducted for eleven days on one module, after which the modules were changed round for another eleven days. It was concluded from the trial that there was absolutely no benefit from a reduction in tonnage rate. The trial was not extended beyond these tonnages since, above 500 t/h, the rod-mill pumps have difficulty in coping with the throughput and, below 450 t/h, the required production rate would not be achieved.

Under normal conditions, the mills are charged according to power draw. At the time of these tests, the target power draw was 1140 kW. For the second test, which lasted for a month, one module was operated at 1200 kW while the other was operated at 1100 kW. Once again, there was no measurable difference in the grind, but there was a 10 per cent improvement in the rod consumption when the mills were operated at 1100 kW and a 9 per cent power saving, which is equivalent to a 2,89 per cent saving on the costs of acid and manganese. The mills were then changed to 1100 kW, and further trials were planned to show how much further this could be reduced without affecting the grind or throughput. Currently (December 1989), the mills are operated at 1000 kW, with a power saving of 16,7 per cent and a rod consumption that has decreased by 15,7 per cent without any measurable change in size distribution. These two savings represent a saving of 5,2 per cent on the costs of acid and manganese.

These two tests were useful in that they gave rise to a new milling philosophy, but they were disappointing in that the variation in operating parameters had little influence on the grind. For the past eighteen months, a study has been conducted into ways by which the size of the fine crushed product can be reduced cost effectively, which will influence the size distribution from the mills. Detailed discussion of this topic would require a further paper, but it is believed that significant reductions in size from the crushing plant can be achieved that will permit a mill product of 75 per cent passing 850 μm.

Further laboratory testwork was carried out and resulted in the implementation of two capital projects on the plant.

(a) The acid-plant calcine is a mixture of hematite and magnetite. While both of these minerals dissolve to produce ferric iron, the magnetite also produces ferrous iron, which requires pyrolusite to oxidize it to ferric iron. A magnetic separator was installed that removes most of the magnetite from the calcine, allowing a relatively clean feed of hematite to be fed to the ferric-leach reactors. This has permitted the achievement of the ferric-iron targets at a lower total iron concentration. The benefits of this are a lower acid consumption due to less iron dissolution, and a reduced requirement for pyrolusite because more ferric iron is available. Furthermore, the ferric-leaching operation has been modified to give a longer retention time in the reactors, which has in turn resulted in the ability to produce more ferric iron as a result of the higher dissolution.

(b) Laboratory testwork had demonstrated that a significant benefit could be obtained from operating the plant at a constant acid tenor throughout the leaching train. Historically, most of the acid was added in the first two tanks, achieving a peak acid concentration of 7 to 10 g/l and allowing this to decay to a terminal acidity of 2,0 g/l. From a laboratory test carried out on a 0,2 m^3 scaled-down tank, it appeared that operation at a constant acid tenor could result in a benefit as high as 4 kg per tonne of acid. A saving of 0,5 kg per tonne is equivalent to a saving of 3,46 per cent on the costs of acid and manganese, and hence the project was highly attractive. The automatic addition of acid to each tank on one leaching module was installed in the latter half of 1989.

Following the installation of these two capital projects it very quickly became apparent that there were massive benefits in terms of acid and pyrolusite savings. The pyrolusite consumption is now less than 10 per cent of the consumption rate in 1986, without any loss in extraction.

The savings in acid consumption were such as to lead to a change in mining philosophy in that ore that had previously been regarded as unsuitable for treatment in the Rössing plant because of its high acid-consuming properties is now sent to the plant with little or no effect on the acid consumption. Although this has had little effect on the metallurgical operation, it has contributed significantly to a 4,7 per cent saving in mining costs for 1989.

Conclusion

Rössing has achieved significant cost savings as the result of a critical examination of operating parameters. This work is now being extended beyond the plant and into the open pit, and is having a significant impact on the total cost of operating the mine. This has maintained the viability of the operation despite the depressed state of the uranium market.

Acknowledgements

The author thanks the operations and metallurgical staff of Rössing Uranium Limited for their assistance in the work described in this paper. He also thanks the Management of Rössing Uranium Limited for permission to publish the paper.

ILO warnings*

The possibility of further job cuts in the non-coal-mining industry globally are forecast in a report just issued by the International Labour Office. (*Report III*: New processes in mines other than coal mines, their effects on employment and training requirements and social policies to cope with these developments).

Already reduced as a result of structural adjustments provoked by market shifts and falling prices, mining employment may decline faster than before because of the displacement of conventional metals by fibre optics, ceramics, or super plastics; the movement of demand away from heavier traditional metals toward lighter, stronger metals; and the growing use of new technologies that are revolutionizing work processes. The extent of the changes will continue to be heavily influenced by rates of growth in the demand for minerals and metals.

While the economic and social consequences of these changes are uncertain, 'the challenges ahead are quite certainly great', the ILO warns.

Developments in the demand for base metals, minerals, and new materials should be closely monitored, the report says, and their implications assessed for the technology, employment, and international division of labour. Tripartite courses of action to cope with the social effects of manpower reductions need to be identified.

This was expected to be one of the main questions to be examined by the ILO's Fifth Tripartite Technical Meeting for Mines other than Coal Mines, held in Geneva from 28th March to 5th April, 1990. The meeting was also expected to survey recent developments in the industry, and propose ways of improving conditions of work and related legal protection in mines other than coal mines.

Coping with Change

While restructuring has led to productivity gains and consequent over-all job losses, there could be positive effects on employment if the demand for minerals and metals improved. However, this would be unlikely to offset direct job losses fully, and the ILO report emphasizes the importance of policies for technology and employment that strike a balance between labour- and capital-intensive methods.

The occupational structure of the mining workforce has also been affected by technological change. The share of technical and professional workers in the labour force has increased, while that of production workers has declined. Policies are needed to help displaced unskilled workers find other jobs. Intensified training efforts are required to develop multiple skills.

Significant changes have come, too, in management styles and employer–labour relations. More flexibility is expected of workers in terms of wages, hours of work, and multi-task performance. The focus of trade unions has moved away from issues of employment security towards questions concerning improvements of income, working conditions, skills, and workers' influence on management decisions regarding work processes and training.

Conditions of Work

The 1980s do not appear to have been a very propitious decade for the improvement of working conditions in non-coal mining, another ILO report observes (*Report II*: Conditions of work and related legal protection in mines other than coal mines).

In industrialized countries, the concern of enterprises to improve their competitive position by cutting costs and raising productivity often meant that working conditions were given lower priority. In developing countries, this same situation was aggravated by problems facing the economy as a whole.

The March–April meeting was expected to examine, in particular, measures dealing with working time and welfare facilities, and ways of promoting the participation of workers in programmes for the improvement of working and living conditions.

Small-scale Mining

Another outcome of the depressed minerals' market is a proliferation of small- and micro-scale mining, partly associated with a new gold rush in several countries, both industrialized and developing.

In industrialized countries, this type of mining is generally soundly financed, equipped, and operated. The ILO reports express concern, however, at conditions in micro-mining including 'pick and shovel' prospecting in parts of the developing world. Often illegal or semi-legal, these operations are not subject to any control. They often result in 'the worst kinds of abuses as regards working conditions, sometimes including shameless exploitation of child labour'. The most basic industrial safety and health practices are often ignored, underground workings being tunnelled haphazardly by different groups of miners regardless of risk.

The reports call for government action to assist micro-mining by developing basic social infrastructure, establishing pricing and marketing mechanisms, promoting co-operatives, and normalizing levels of production.

* Released by the International Labour Office, CH-1211 Geneva 22, Switzerland.