



Development of Projects for the Production of Uranium Concentrates

PROCEEDINGS OF A TECHNICAL COMMITTEE MEETING, VIENNA, 25–28 NOVEMBER 1985



INTERNATIONAL ATOMIC ENERGY AGENCY, VIENNA, 1987

The cover photographs show aspects of the development of a uranium project in Chihuahua, Mexico:

Upper left: Uranium mineralization at El Nopal

Upper right: El Nopal mine

Lower left: Part of the pilot plant at Villa Aldama, Chihuahua

Lower right: Uranium mill at Villa Aldama, Chihuahua

Photographs by S. Ajuria.

**DEVELOPMENT OF PROJECTS
FOR THE PRODUCTION
OF URANIUM CONCENTRATES**

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PANEL PROCEEDINGS SERIES

DEVELOPMENT OF PROJECTS FOR THE PRODUCTION OF URANIUM CONCENTRATES

PROCEEDINGS OF A TECHNICAL COMMITTEE MEETING
ON DEVELOPMENT OF PROJECTS
FOR THE PRODUCTION OF URANIUM CONCENTRATES
ORGANIZED BY THE
INTERNATIONAL ATOMIC ENERGY AGENCY
AND HELD IN VIENNA, 25-28 NOVEMBER 1985

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FOREWORD

Nuclear energy will continue to play an important role in meeting the world's energy demand for the foreseeable future. To guarantee an adequate supply of uranium many new production centres will have to be opened, both in the industrialized and in the developing countries. Many of the latter are interested in exploiting their uranium resources, either in connection with their own nuclear power programmes or for export. However, the industrial exploitation of uranium resources requires not only a good knowledge of current uranium extraction technology but also, and just as importantly, a good knowledge of standard industrial practice for the evaluation, planning and implementation of mining and metallurgical projects.

Although the IAEA has organized many technical meetings on the geology of uranium, exploration techniques, uranium resource evaluation, mining techniques, uranium ore processing, safety aspects of uranium mining and milling and management of uranium tailings, this Technical Committee Meeting has been the first to address the problem of planning and implementing complete projects. The meeting was held in Vienna from 25 to 28 November 1985. A total of 35 participants from 21 countries took part and 18 papers were presented, including two papers which review the development of projects. The present volume includes 15 of these papers and status reports from five countries.

The six technical sessions covered various aspects of the evaluation, planning and implementation of projects, special topics such as testing and optimization of processes and case histories of projects in Member States.

The IAEA wishes to thank all the metallurgists and engineers who participated in the meeting, not only for the papers but also for their contributions to the discussions, which in this case were very lively and shed much light on the topics being considered. Special thanks are due to the Chairman of the meeting, W. Flöter (Federal Republic of Germany).

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Review Paper**DEVELOPMENT OF PROJECTS
FOR THE PRODUCTION OF
URANIUM CONCENTRATES:
AN OVERVIEW****S. AJURIA**

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Vienna

Abstract**DEVELOPMENT OF PROJECTS FOR THE PRODUCTION OF URANIUM CONCENTRATES:
AN OVERVIEW.**

The implementation of a project for the production of uranium concentrates is costly and complex. It requires a good knowledge of current technology for uranium exploration, ore body development, ore reserve estimation, mining and ore processing and of standard industrial practice for the development of mining and metallurgical projects including generally accepted criteria for project evaluation. The paper discusses the main stages in the development of a project and outlines the requirements in terms of expertise, capital investment and time.

1. INTRODUCTION

Nuclear energy will undoubtedly play an important role in meeting the world's energy demand in the foreseeable future. Uranium is by now firmly established as an important energy resource. Annual uranium production is currently about 40 000 t [1]. Many developing countries are therefore interested in exploiting their uranium resources, either for domestic consumption or for export.

The successful implementation of a project for the production of uranium concentrates requires a good knowledge of:

- (a) Current technology for uranium exploration, ore body development, ore reserve estimation, mining and ore processing
- (b) Generally accepted methods and criteria for the evaluation of projects, and
- (c) Project management and standard industrial practice for the development of mining-metallurgical projects.

These three components are equally essential: a deficiency in any one of them will have adverse effects on the project as a whole and may, in extreme cases, lead to failure. Extensive technical expertise is required to prepare preliminary feasibility

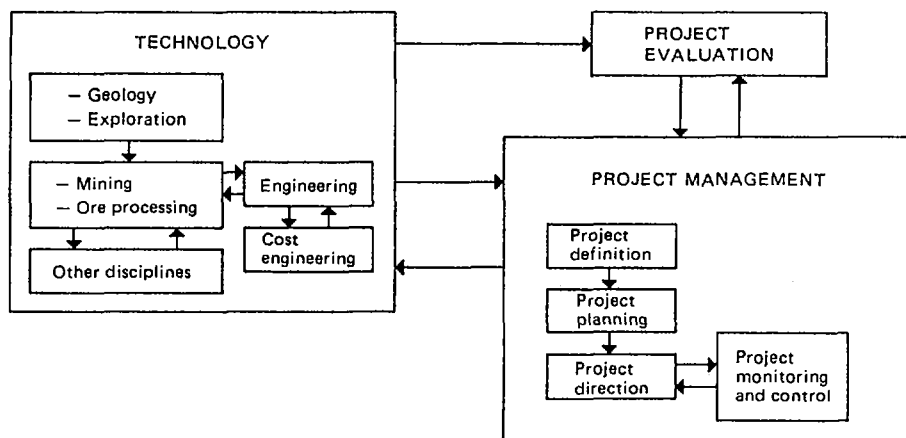


FIG. 1. Areas of expertise required for project development and their interrelationships.

and final feasibility studies and project proposals. These must be objectively evaluated before a decision to invest can be made. Once the project has been approved a task force is formed to implement it, again drawing on the technical expertise. These interrelationships are shown schematically in Fig. 1. It is also important to have, from the outset, a realistic concept of the order of magnitude of the effort, time and capital investment required for the implementation of a project of this type.

2. THE TECHNOLOGY OF URANIUM EXPLORATION, MINING AND ORE PROCESSING

2.1. Uranium exploration

The objective of exploration is to find ore bodies suitable for commercial exploitation in a reasonable time and at a reasonable cost. Exploration begins with data collection to select favourable areas and then proceeds through regional and semiregional reconnaissance phases to locate anomalies. Then follows semidetailed exploration to define drill targets and detailed exploration which includes drilling and possibly other methods of subsurface exploration. Once promising mineralizations have been found these are evaluated and, if warranted, developed further [2, 3]. The probability that an anomaly will eventually prove to be an ore body suitable for commercial exploitation is approximately between 0.1 and 1% [4]. Finding an ore body also takes time. In some cases it has taken up to 15 years of continuous and systematic work.

2.2. Development of uranium deposits and ore reserve estimation

A reliable estimate of ore reserves is essential for planning the development of a project [5]. Ore body development involves drilling (or other surface and sub-surface methods of exploration) to establish the type of the deposit, to delineate it (i.e. to determine its size and shape) and to recover samples for geotechnical, mineralogical and metallurgical studies. The quantity of uranium contained in situ in an ore reserve can then be estimated from the volume of the ore deposit and the concentration of uranium in that deposit [5] and estimates of the amount of mineable ore can be made.

2.3. Uranium mining

The methods used for mining uranium are essentially the same as those used for other ores [6]. The only important difference is that uranium and uranium ores are radioactive and this requires some special safety considerations in the design and operation of uranium mines, particularly concerning ventilation to avoid or minimize the accumulation of radon and radon daughter products and to reduce or suppress dust generation [7]. In the case of very high grade uranium ores it is also necessary to guard against exposure by external irradiation.

2.4. Uranium ore processing

The technology for uranium extraction has by now reached a state of maturity although existing processes continue to be refined to improve their efficiency and flexibility and to lower their costs and reduce their environmental impact.

The extraction of uranium (Fig. 2) from its ores generally involves the following unit operations:

- (a) *Size reduction.* The ore is crushed and ground to a suitable range of particle sizes required for effective leaching and to produce a material that can be slurried and pumped through the processing circuits.
- (b) *Leaching.* Uranium is leached from the ground ore using either acid or alkaline (carbonate) solutions with or without heating and with or without the addition of oxidants such as atmospheric oxygen, sodium chlorate, manganese dioxide or peroxides. Leaching can be done in mechanically agitated vessels, in Pachuca tanks or in autoclaves. Other techniques such as pugging, heap leaching and in situ leaching are also used.
- (c) *Solid-liquid separation and washing.* The pregnant liquor is separated from the leached ore. Several techniques are available such as the use of cyclones, counter-current decantation (CCD) or filtering.
- (d) *Purification and concentration.* The pregnant liquor contains a very low concentration of uranium (usually of the order of 1 g/L) and many impurities. It

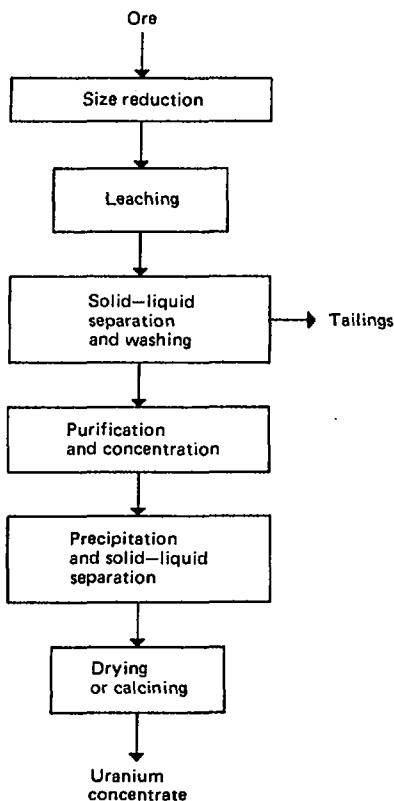


FIG. 2. Generalized process for uranium extraction.

must therefore be concentrated and purified using either ion exchange resins or liquid-liquid extraction (more commonly known as solvent extraction) or both.

- (e) *Precipitation and solid-liquid separation.* A uranium concentrate (yellow cake) is precipitated from the pregnant solution in one or two stages using any one of a number of reagents such as sodium hydroxide, ammonium hydroxide, magnesium hydroxide or hydrogen peroxide. The product is either filtered or dewatered by settling and centrifugation.
- (f) *Drying or calcining.* The concentrate is dried or calcined and packaged in steel drums lined with polyethylene. Ammonium diuranate is usually dried or spray dried at 150°C whereas sodium diuranate is calcined at about 400°C. Several types of equipment can be used: single hearth dryers, multiple hearth dryers, drum dryers, screw dryers, spray dryers or radiant heat dryers.

Unit operations from (a) to (f) are combined to constitute actual industrial processes.

- (g) *Tailings disposal.* The process must also include adequate provisions for treating and impounding the tailings and for recovering and recycling some or most of the water used [8].

There are several recent articles [9, 10] and monographs [11–14] on uranium extraction so it is not necessary to discuss this topic in detail in this paper.

2.5. Other disciplines

Many additional disciplines are needed for the development of a project. Some of the most important are:

- (a) Analytical chemistry
- (b) Mineralogy
- (c) Petrology
- (d) Radiation protection
- (e) Environmental engineering
- (f) Mechanical engineering
- (g) Civil engineering
- (h) Soil mechanics
- (i) Hydrology
- (j) Meteorology

3. PROJECT MANAGEMENT

Technical expertise alone cannot ensure the success of a project. Effective project management is just as essential [15, 16].

“Project management is the planning, organizing, directing and controlling company resources for a relatively short-term objective” [15].

The project manager, and the project management team, have the following main tasks:

- (a) *Project definition.* The specific objective to be attained must be clearly specified. The time and funding limits must also be set.
- (b) *Project planning.* A specific course of action, including partial tasks and objectives and a calendar for their implementation must be defined. The quantity of work and the resources needed must also be defined.
- (c) *Project direction.* The specific actions required to attain the final objective must be executed. This includes obtaining funds (financing), hiring the necessary personnel with the proper qualifications (staffing), purchasing the materials and equipment (procurement), selecting and hiring contractors for those tasks and services that the company cannot or should not execute directly and obtaining the necessary permits and licences to build and operate the plant.

- (d) *Project monitoring and control.* The actual progress and costs of the project and the quality of the work must be continuously assessed and compared with the planned calendar, budget and specifications. Adjustments must be made or corrective actions taken, as required.

The project can be considered as successfully completed when the desired objective has been attained within specifications and within the allotted time and cost.

4. PROJECT EVALUATION

Project evaluation is the third essential component in project development. Before any important resources, financial or otherwise, are committed there must be a reasonable assurance that the project will result in a positive net benefit [17–20]. The total investment and operation costs must be computed and compared with the projected revenues. Financial evaluation should preferably rely on discounting methods and incorporate sensitivity analyses [21]. Under certain conditions a country may also wish to take into account selected non-economic criteria. It may be desirable to undertake a project for social or political reasons even if the project is not financially attractive. In any case it is important to know the true cost of the project and the price that will have to be paid for the non-economic benefits.

Formal evaluations are usually done at specific stages of the development of a project. This is discussed in more detail in the next section of this paper.

5. DEVELOPMENT OF A METALLURGICAL PROJECT

A project consists of a series of activities intended to achieve a specific objective within certain cost and time limitations. When a promising ore body has been discovered, a project for an ore processing plant can be considered to include the following main phases:

Pre-investment phase

- (a) Order of magnitude studies
- (b) Preliminary feasibility studies
- (c) Preliminary evaluation and decision to continue
- (d) Licensing (preliminary)
- (e) Final feasibility study
- (f) Final evaluation and investment decision

Investment (implementation) phase

- (g) Detailed engineering
- (h) Procurement and construction

- (i) Licensing (final)
- (j) Startup and commissioning

Operational phase

- (k) Operation

Post-operational phase

- (l) Decommissioning

These phases of development are, of course, not strictly sequential. They usually overlap one another (see Fig. 3).

5.1. Order of magnitude studies

Rough estimates of the ore body's grade and tonnage are prepared, based on preliminary drilling. One or several mining methods are tentatively selected and an estimate is made of the amount of mineable ore, of the waste to ore ratio, and the production rate. Mineralogical examinations and preliminary laboratory tests are made using core samples or other samples that may be available.

The preliminary laboratory tests are performed with the primary purpose of helping to determine the technical and economic feasibility of processing the ore. These tests may be performed on grab samples or on core samples that need not be representative of the ore body as a whole. At this stage it is only necessary to cover the first process stages: size reduction, leaching and solid-liquid separation. If it is possible to dissolve most of the uranium present in the samples and if it is possible to obtain a clear pregnant liquor it will generally be possible to purify and concentrate this liquor and to obtain a uranium concentrate of good quality. Several of the major process alternatives may be considered at this stage: acid or alkaline leaching, heap leaching, etc. From the results of these tests tentative process flow sheets are prepared. Order of magnitude estimates of capital and operating costs are prepared based largely on published cost data and on the costs of similar existing equipment and plants. These estimates may be accurate within 30 to 50%.

5.2. Preliminary feasibility studies

The ore body is extensively drilled to establish its type, morphology (in three dimensions), size (also in three dimensions), and degree of homogeneity. The mean ore grade and the ore reserves are calculated. Mining methods are proposed and the grade and tonnage of mineable ore, the ratio of ore to waste rock, the mine's production rate, and the expected mine life are estimated. Detailed mineralogical studies and bench scale metallurgical tests are performed using representative samples with the primary purpose of selecting and defining a suitable metallurgical process. These

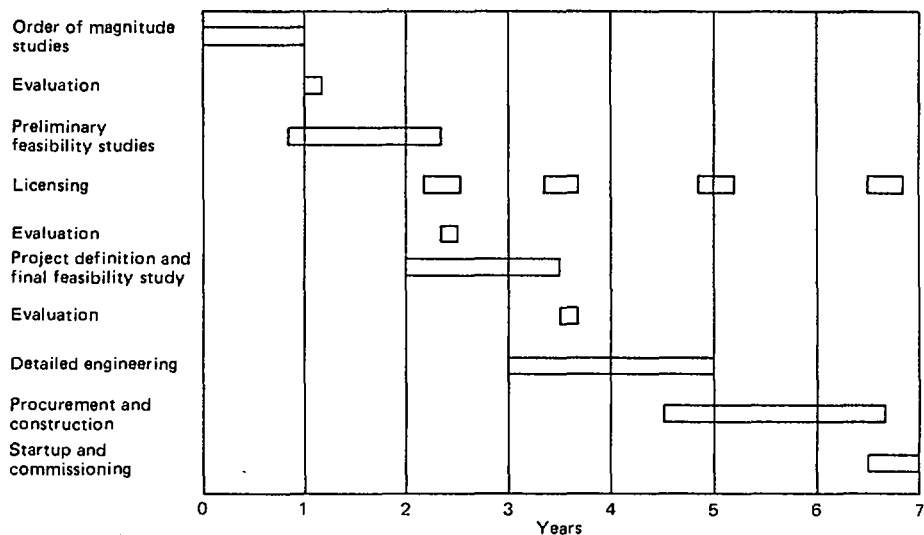


FIG. 3. Phases of project development: an example.

tests include all the unit operations necessary to constitute a complete process. The results of the tests are used to define the main process parameters (fineness of grind, solid/liquid ratio, temperatures, residence times, levels of oxidant, pH regimes, etc.) and process efficiency. The size of the plant is tentatively selected and complete flow sheets and material and energy balances are prepared. The main objective of the material balances is to determine the design basis for establishing the size of equipment and other main equipment parameters. The main equipment items are selected from the many options available and a preliminary layout of the plant is prepared. Utility requirements (electrical power, water, fuel and steam) and reagent consumption (such as sulphuric acid, oxidants, ion exchange resins, extractants, etc.) are calculated.

All this information is now used to prepare new estimates of capital and operating costs which are expected to be accurate within plus 20%. Estimates are also made of the time required for construction.

5.3. Licensing

Licensing of uranium mines and mills is required in most countries and has become an important phase in the development of a project. In general terms it is necessary to demonstrate that appropriate measures are being taken or will be taken to minimize the environmental impact of the mine and mill and to protect the health of the workers and of the public within the limitations of currently available technology and reasonable costs. Environmental and radiological safety requirements must

be taken into consideration from the earliest stages of the project because these requirements will influence the design, construction and operation of the mine and mill. Before any major commitment is made it is essential to have reasonable assurances that the mine and mill will be allowed to operate.

5.4. Preliminary project evaluation

At this point there should be enough information to make a preliminary economic evaluation of the project, that is, to make an estimate of the costs and benefits associated with the project. The project should proceed only if a positive net benefit can be assured. The evaluation should be as objective as possible and it should *not* be done by the persons who prepared the preliminary feasibility study.

5.5. Final feasibility study

Additional drilling or other development work on the ore body is usually required to confirm its reported morphology, ore grade and reserves. Suitable mining methods are developed, taking into account the exact locations of the ore or of the waste zones to be mined, the tonnages and grades of the ore zones and the time periods in which these zones are to be mined. The cut-off grade, mining losses and ore dilution, ore to waste ratio, and extraction rate are defined. Mining equipment is selected. The process flow sheet is refined by further testing which may include modelling and pilot plant studies.

The overall objective of the pilot plant is to ensure that the decision to proceed with the full-scale project will be based on a proven process and on a reliable economic evaluation. Other more particular objectives of the pilot plant are:

- (a) To optimize the operating parameters of the process
- (b) To obtain process information necessary to specify and design the full-scale equipment
- (c) To optimize the design of the equipment
- (d) To obtain sufficient information to prepare detailed and reliable estimates of capital and operating costs and to prepare a reliable economic evaluation of the project
- (e) To train personnel to operate the full-scale plant.

The definitive flow sheet, with the corresponding material and energy balances is prepared. The selection of equipment and the plant layout are verified and piping and instrument diagrams and electrical diagrams are prepared. Structural sketches and building sketches are drawn up. A site for the plant and for the tailings dam or dams is selected and a general study is made of the site and of the infrastructure needed to support the mine and the mill. Definitive estimates of capital and operating costs, accurate within plus 10% are prepared. The project can now be submitted for evaluation and approval. The development of the project through the pre-investment phase is summarized in Table I.

TABLE I. DEVELOPMENT OF A PROJECT THROUGH THE PRE-INVESTMENT PHASE

Project component	Order of magnitude studies	Pre-feasibility studies	Feasibility studies
Ore body	Ore body roughly defined, grade and tonnage estimated	Mean ore grade and reserves calculated	Ore grade and reserves confirmed
Mining method	Assumed	Tentatively selected	Developed
Metallurgical process	Mineralogical examinations, preliminary laboratory tests, tentative flow sheets prepared	Bench scale testing Process selected, main parameters defined, material balances prepared	Pilot plant studies Process flow sheet defined, material and energy balances prepared
Process equipment	Assumed	Tentatively selected	Equipment selected, layout defined Piping and instrumentation diagrams prepared
Site	Assumed	Tentatively selected	Selected
Licensing	Assumed	Licensing requirements reviewed Notification of intent filed	Licensing procedures initiated
Accuracy of economic estimates	±30%	±20%	±10%

5.6 Final evaluation and decision to invest

A final evaluation of the project can now be made using generally accepted financial criteria [21] and the decision to invest can be taken.

5.7. Detailed engineering

The final engineering design of all plant equipment and facilities is completed based on the conceptual and preliminary engineering designs previously prepared. The final process design includes vessel design, machines design or selection, definitive layout, piping design, instrument and control design and electrical design. Final site design and structural, civil and architectural designs are also completed.

5.8. Procurement and construction

The final engineering drawings and specifications are translated into the actual plant. The mine and site are prepared, construction equipment is brought and construction personnel is hired. Plant equipment and all materials items needed are ordered and delivered to the site. The actual facilities are built by a field task force which is usually a contractor.

5.9. Marketing

Suitable arrangements must be made well in advance to sell at a profitable price the uranium produced by the plant. About 30% of the world's uranium production is sold in the spot market. About 70% is sold through medium or long term contracts. The current spot market price (November 1985) is about US \$17 per kg U_3O_8 . The contract price can be substantially higher and may be as high as US \$66 or US \$88 subject, of course, to specific conditions.

5.10. Startup and commissioning

All the equipment and the piping, electrical, materials handling and control systems are checked out and any problems corrected. The plant is started, operating first at reduced capacity and finally at full rated capacity. Custody and control of the plant are transferred to the operator.

5.11. Operation

Operating a uranium mine and mill is not necessarily routine. It is not uncommon to run into problems after startup. Solving these problems may need adjustments and modifications (in some cases substantial) before trouble free operation at full rated capacity can be attained. Afterwards the process is usually refined to increase

productivity, to improve the quality of the product and to lower costs. In addition, since most ore bodies are not homogeneous, it is often necessary to modify the process as required by the changing nature of the ore.

It is also necessary to increase the available ore reserves to extend the life of the operation. This entails a continuing effort of surface and underground exploration and delineation and development of new mineralized zones.

5.12. Decommissioning

At some point operations will end and the mine, mill and tailings dams must be decommissioned. Concrete provisions for this final stage of the project must be made during the design and construction stages. It is also necessary to prepare in advance a decommissioning plan, schedule and budget and to ensure that there will be funds available to execute the plan.

6. CAPITAL COST AND TIME REQUIREMENTS

The total cost of a project over its entire life cycle depends on many factors such as the size and homogeneity of the ore body, its mean ore grade, the accessibility of the ore for mining, the amenability of the ore for chemical treatment and the availability of a suitable infrastructure [22]. The components of a project are as follows:

Pre-investment phase

- (a) Exploration
- (b) Preliminary feasibility studies
- (c) Feasibility studies
- (d) Licensing (preliminary)

Investment phase

- (e) Financing
- (f) Acquisition of land and mineral rights
- (g) Engineering
- (h) Infrastructure development
- (i) Mine development
- (j) Mill construction
- (k) Preparation of tailing dam(s)
- (l) Licensing (final)
- (m) Startup

Operational phase

- (n) Operating cost of the mine

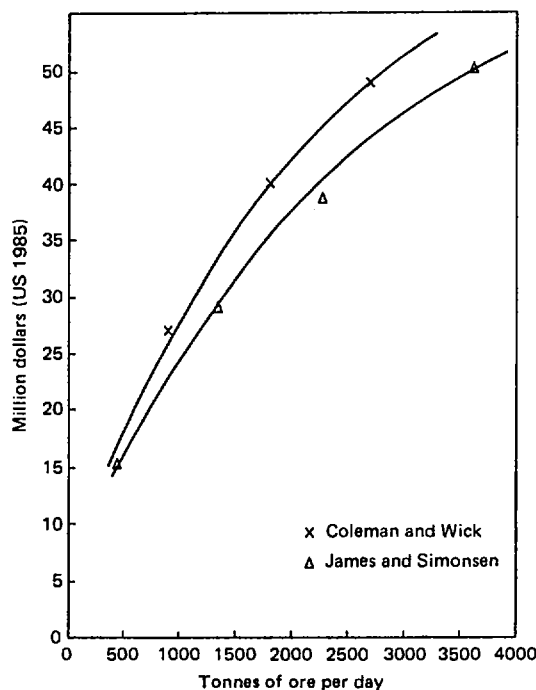


FIG. 4. Capital investment as a function of plant size.

- (o) Operating cost of the mill
- (p) Royalties, marketing and other costs

Post-operational phase

- (q) Decommissioning

While it is difficult to make generalizations it is clear that the total capital investment required is in the range of several tens to several hundreds of millions of US dollars [23]. For instance, the total cost of the Key Lake Project (Canada) is reported to have been in excess of 500 million Canadian dollars [24].

The capital cost of a metallurgical plant for the production of uranium concentrates (exclusive of the mine, tailings dams, infrastructure and other items) as a function of plant size has been estimated by James and Simonsen [25] and by Coleman and Wick [26]. These estimates are summarized in Fig. 4.

The time required for the implementation of a project, from the identification of a suitable ore body to full production, depends on the size and complexity of the project, the accessibility of the site and availability of a suitable infrastructure, the expertise of the project team, the availability of funds and other factors. Past experience indicates that the time required is in the range of 5 to 15 years [27].

7. CONCLUSION

A mining-metallurgical project is costly, lengthy and complex. The implementation of such a project requires a capital investment of several tens to several hundreds of millions of US dollars and from 5 to 15 or more years. The successful implementation of a project also requires a good knowledge of uranium exploration, mining and ore processing, of project management and standard industrial practice for the development of mining-metallurgical projects and of generally accepted methods and criteria for the evaluation of projects.

REFERENCES

- [1] NUCLEAR ENERGY AGENCY OF THE OECD, Uranium: Resources, Production and Demand, Joint Report by the OECD/NEA/IAEA, OECD, Paris (1983).
- [2] BUNDROCK, G., "From armchair geology to a deposit in a new uranium province", Uranium Exploration Case Histories (Proc. Advisory Group Mtg Vienna, 1979), IAEA, Vienna (1981) 243-277.
- [3] NUCLEAR ENERGY AGENCY OF THE OECD, Uranium Exploration Methods (Proc. Symp. Paris, 1982), OECD, Paris (1982).
- [4] PETERS, W.C., Exploration and Mining Geology, Wiley, New York (1978).
- [5] INTERNATIONAL ATOMIC ENERGY AGENCY, Methods for the Estimation of Uranium Ore Reserves, An Instruction Manual, Technical Reports Series No. 255, IAEA, Vienna (1985).
- [6] INTERNATIONAL ATOMIC ENERGY AGENCY, Uranium Evaluation and Mining Techniques (Proc. Symp. Buenos Aires, 1979), IAEA, Vienna (1980).
- [7] AHMED, J.U., Occupational radiological safety in uranium mines and mills, Int. At. Energy Agency Bull. 23 2 (1981) 29-32.
- [8] THOMAS, K.T., Management of wastes from uranium mines and mills, Int. At. Energy Agency Bull. 23 2 (1981) 33-35.
- [9] SEIDEL, D.C., Extracting uranium from its ores, Int. At. Energy Agency Bull. 23 2 (1981) 24-28.
- [10] GOW, W.A., "Recent advances in uranium ore processing", Advances in Uranium Ore Processing and Recovery from Non-conventional Resources (Proc. Tech. Committee Mtg Vienna, 1983), IAEA, Vienna (1985) 3-13.
- [11] NUCLEAR ENERGY AGENCY OF THE OECD, Uranium Extraction Technology: Current Practice and New Developments in Ore Processing, Joint Report of the OECD/NEA/IAEA, OECD, Paris (1983).
- [12] MERRITT, R.C., The Extractive Metallurgy of Uranium, United States Atomic Energy Commission, Washington, DC (1971) (Reprinted in 1979).
- [13] JONES, M.J., (Ed.), Geology, Mining and Extractive Processing of Uranium, The Institution of Mining and Metallurgy, London (1977).
- [14] INTERNATIONAL ATOMIC ENERGY AGENCY, Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores, Technical Reports Series No. 196, IAEA, Vienna (1980).
- [15] KERZNER, H., Project management: A Systems Approach to Planning, Scheduling and Controlling, Van Nostrand, Princeton and New York (1984).
- [16] BENNINGSON, L., Project Management, McGraw-Hill, New York (1970).

- [17] ALLEN, D.H., A Guide to the Economic Evaluation of Projects, Institution of Chemical Engineers, London (1972).
- [18] ALLEN, D.A., Project evaluation, Chem. Technol. **9** 7 (1979) 412-417.
- [19] PARK, W.R., Cost Engineering Analysis: A Guide to the Economic Evaluation of Engineering Projects, Wiley, New York (1973).
- [20] SASSONE, P.G.Y., SCHAFFER, W.A., Cost-Benefit Analysis: A Handbook, Academic Press, New York (1978).
- [21] UNITED NATIONS INDUSTRIAL DEVELOPMENT ORGANIZATION, Manual for the Preparation of Industrial Feasibility Studies, UNIDO, Vienna (1978).
- [22] HOSKINS, J.R., (Ed.), Mineral Industry Costs, Northwest Mining Association, Spokane, WA (1982).
- [23] O'HARA, T.A., Quick guides for the evaluation of orebodies, CIM Bull. (Feb. 1980) 87-99.
- [24] Key Lake pit development a formidable design challenge, Can. Min. J. (Jun. 1984) 26-27.
- [25] JAMES, H.E., SIMONSEN, H.A., "Ore-processing technology and the uranium supply outlook", Uranium Supply and Demand (Proc. 3rd Int. Symp. London, 1978), Mining Journal Books Ltd and The Uranium Institute, London (1978) 113.
- [26] COLEMAN, R.B., WICK, K.E., "Factors for cost estimating in uranium ore processing", Economics of Ore Processing Operations, OECD, Paris (1983) 74-99.
- [27] Fuel and Heavy Water Availability, Report of INFCE Working Group 1, IAEA, Vienna (1980) 187.

DISCUSSION

H.J. STEINER: What are the most common errors and causes for delays in the development of projects?

S. AJURIA: A common error that we have often seen in developing countries is the failure to recognize that a certain uranium occurrence is not economical and should perhaps be left aside. There is a tendency to go through with a project no matter what. In fact very few of the uranium occurrences that are found can be exploited economically. One should perform the order of magnitude and pre-feasibility studies and come quickly to a decision whether or not to proceed so as to avoid wasting time and effort.

D.C. SEIDEL: One point where much difficulty has been encountered is in not recognizing the variability of the ore in both grade and mineralogy. There have been very serious difficulties when this variability has not been taken into account when designing the process.

M.C. CAMPBELL: I think one of the biggest sources of problems has been the failure to delineate the ore body properly and to obtain good representative samples. There have been several cases of people going into production with a process based on a sample that did not in fact represent the ore body. A truly representative sample may be very difficult to obtain. In that case the process must be designed with enough flexibility to cope with the variations that will undoubtedly take place.

INTERNATIONAL CO-OPERATION IN RADIATION PROTECTION PRACTICES IN THE MINING AND MILLING OF URANIUM

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Abstract

INTERNATIONAL CO-OPERATION IN RADIATION PROTECTION PRACTICES IN THE MINING AND MILLING OF URANIUM.

Uranium mining industry has been associated with the history of excess lung cancer. Because of such epidemiological evidence, the subject of radiation protection in the nuclear mining industry has received increased attention in recent years both at national and international levels. The radiation hazards encountered in the uranium mining industry result primarily from the exposure to radon daughters. The exposure to external radiation in most mines is low; however, in the mining of high grade uranium ores external radiation exposure can be substantially higher. By adopting proper control measures, namely, regulatory control, appropriate safety standards, monitoring, engineering and other measures, medical surveillance, environmental protection and radioactive waste management, it is possible to minimize the health risk to a level deemed to be acceptable in the light of the benefit derived from the uranium mining industry. Recognizing the history of excess lung cancer among uranium miners and the nature of the associated radiation protection problems there have been considerable national and international efforts to develop safety standards, codes of practice and guides to improve the radiation protection practices for the protection of the workers and the general public. At the international level the role of the International Atomic Energy Agency (IAEA), the International Commission on Radiological Protection (ICRP), the International Labour Organisation (ILO), the World Health Organisation (WHO), and the Organisation for Economic Co-operation and Development/Nuclear Energy Agency (OECD/NEA) has been to develop and provide guidance for improved radiation protection in the uranium mining industry. In the paper the radiological problems in this industry, the types of control measures and the international efforts in harmonizing radiation protection measures are discussed.

1. INTRODUCTION

In the years of uranium production the uranium mining industry grew rapidly in many countries due to the increasing demands of uranium. The rate of growth however has slowed in recent years because of the sluggish growth of the nuclear

power industry. This situation is not expected to continue, therefore the demand for uranium is likely to increase, though perhaps not at the same rate as in the past.

In Colorado, United States of America, Czechoslovakia, and Ontario, Canada, an excess lung cancer incidence has been observed among underground uranium miners who were exposed to radiation. An excess lung cancer incidence has also been observed among non-uranium miners exposed to radon daughters (fluorspar miners in Newfoundland, Canada, iron ore miners in Sweden).

The radiation risk encountered in the uranium mining industry results primarily from exposure to airborne radioactivity and, to a lesser degree, from external radiation. The inhalation risk is of more concern in underground mining of radioactive ores than in open pit mining.

The excess lung cancer incidence was recognized early in the 20th century among miners in Joachimsthal and Schneeberg but it was really only in the late 1960s and early 1970s that more scientific epidemiological studies of the various groups of miners have been undertaken [1-6].

Although all these studies clearly indicate a causal relationship between exposure to radiation (mainly radon daughters) and lung cancer, the absolute value of risk per unit of exposure is difficult to determine.

This is due to the lack of historical exposure data and uncertainties in the determination of work histories. As the input data were estimated retrospectively, the precision of the data cannot be improved significantly [7]. But qualitatively the evidence clearly indicates the relationship between exposure and incidence of lung cancer.

The epidemiological studies presently under way in New Mexico, Ontario and elsewhere would eventually provide more reliable quantitative risk estimates in the future. Improvements in dosimetric models should also result in risk estimates of higher confidence level in the future.

2. RADIATION RISKS IN MINES AND MILLS

The radiation risks in underground uranium mines are primarily due to ^{222}Rn daughter products, ^{218}Po , ^{214}Pb , ^{214}Bi , ^{214}Po . Radon is an inert gas and therefore passes freely into and out of the lung with minimal uptake by the respiratory system. The radiation dose to the respiratory system due to the alpha decay of inhaled radon daughters is 100 times more than that due to the decay of radon itself in the lungs. Radon daughters have relatively short half-lives, resulting in a rapid increase of their concentrations when radon is released to the air. Radon daughters are heavy metals and some aspects of their interaction with atmospheric dust particles, ions and condensation nuclei are not yet fully understood. They can attach to dust particles in the air and follow the behaviour of these particles or nuclei. The radiation dose to the respiratory system depends on the concentration of radon daughters in the inhaled

air, the size of the particles in the dust to which the daughters are attached and the physiological parameters. The unattached radon daughters are considered to deposit preferentially in the upper passages of the respiratory track where most miners develop lung cancer. The airborne dust in the mine also contains long lived radionuclides from the ^{238}U and ^{235}U series. From the viewpoint of internal contamination ^{238}U , ^{234}U , ^{230}Th , ^{226}Ra and ^{210}Po are significant. Mining operations such as drilling and blasting produce airborne dust containing these long lived radionuclides which in most ores are close to radioactive equilibrium.

In thorium mines and mills radiation risks from thoron daughters are encountered. The daughter products of thoron have half-lives that are very different from the corresponding daughter products of radon. Similar to radon the radiation dose from thoron to the respiratory system is due to the alpha decay of inhaled thoron daughters and is 500 times more than that due to the decay of thoron itself.

In some uranium mines thorium is also present and therefore risks from both radon and thoron daughters are encountered.

In uranium mills, radon and its daughter products usually present only a minor inhalation risk compared to the ore and uranium dust. However, in certain areas significant radon daughter concentrations may occur, particularly in the ore storage areas and crushing and grinding facilities. The major radiation risks that may be found in the milling operations arise from the crushing and grinding of the ore and in the final product (yellow cake) area. At the initial stages, namely the crushing and screening areas the long lived radionuclides tend to be in equilibrium, but during subsequent operation this equilibrium is disturbed. At the precipitation and recovery stages, the solutions and solids handled are rich in uranium so that the airborne radioactivity is predominantly due to uranium. In tailing treatment areas the airborne radionuclides are predominantly ^{230}Th , ^{226}Ra and ^{210}Po . In a mill, the external radiation consists of beta and gamma radiation emitted by the uranium and thorium chain elements. Dose rates up to 1 mSv per hour may be encountered close to high grade uranium ore, or close to thorium concentrate, but generally dose rates will not be more than a tenth of a mSv per hour. Beta dose rates are less significant than gamma dose rates, but could be of importance as a source of skin dose in some cases.

Surface contamination can be a source of direct external radiation exposure and of internal exposure due to inhalation of resuspended loose contamination or ingestion of material. In the mine it usually does not require a systematic monitoring programme as the ore is a more significant source of external radiation and a routine monitoring of airborne activity will detect any increase of activity indicating the need for an improved dust control programme. In the mill, resuspension of loose contamination can be a problem, more particularly in the product precipitation, drying and packaging areas, where the dust is likely to contain 60% to 80% uranium. If very high grade ore is being processed, loose contamination could be a problem in other areas. Surface contamination can be regarded as being fixed or removable; the removable contamination is the more significant, and surface contamination limits are usually set in terms of removable contamination.

3. RADIATION RISKS IN MINE-MILL WASTE MANAGEMENT

Solid, liquid and gaseous waste generated by uranium mining and milling activities does represent both occupational and public radiation risk.

Waste rock produced during mining activities might contain a low concentration of uranium and thorium. Mill tailings are the solid finely ground residue remaining after the extraction of uranium or thorium. They contain a low quantity of uranium and thorium and practically all of the ^{226}Ra and ^{228}Ra as well as the daughter products contained in the ore. All solid wastes often contain other chemical contaminants (heavy metals, etc.), that could create occupational and public risks. Waste rock and tailings are at present deposited and retained in well engineered waste management facilities. The facilities have to be located in such a manner as to minimize the effect of wind carried towards workplaces or publicly inhabited areas.

Mine water and liquid effluent from the mill could be another source of radiation risk to workers and to the public. Mine water is usually a source of radon, which is readily released into the atmosphere when the water enters the mine opening. It is therefore important to catch and pipe or channel the water in covered ditches as close to the source as possible and to transport it out of the mine by the shortest feasible way.

Mill process water usually carries high concentrations of radium.

Both mine water and mill effluent should be treated to remove radium (for example by BaSO_4 addition to create a radium barium sulphate precipitate), before it is released into the environment. It is usually necessary to adjust the pH of the liquid effluent to minimize the environmental damage.

Gaseous effluents from mine and mill contain all the airborne radioactive contamination removed from the workplaces. Radon and thoron daughters, which are the main contaminants in the mines, are diluted very fast after the mine exhaust air mixes with the outside atmosphere; however problems could arise if the mine exhaust is too close to fresh air intake to the mine, mill or plant office buildings. Under some geographic and climatic conditions, situations of thermal inversion could occur, that may result in very high radon/thoron daughter concentrations in the ambient air. It is therefore necessary to study the climatic conditions of the site before the physical location of exhaust and waste management areas are decided upon.

4. DOSE LIMITATION

The basic philosophy of the International Commission on Radiological Protection (ICRP) dose limitation system consists of three principal components:

- (a) *Justification.* No practice resulting in human exposures to radiation should be authorized unless its introduction produces a net benefit, taking into account also the resulting radiation detriment.

- (b) *Optimization.* All exposures should be kept as low as reasonably achievable economic and social factors being taken into account. This requirement implies that the detriment resulting from a practice should be reduced by protective measures to such a value that further reduction is less proportionate to the additional efforts and costs required to achieve the reduction.
- (c) *Individual dose limits.* The effective dose equivalent to individuals from all practices (except those specifically excluded) should not exceed the applicable dose limits. Applying this requirement, it must be recognized that many present day practices give rise to effective dose equivalents that will be received in the future. This should be taken into account to ensure that present or future practices would not be liable to result in a combined overexposure of any individual.

4.1. Primary dose limits

The dose equivalent limits are not intended to be design or planning objectives. Values above the limits are specifically not permitted, but values below the limits are a constraint for the optimization procedures.

The effective dose equivalent limits for workers based on ICRP recommendations and specified in Safety Series No. 26 [8] are as follows:

- (a) In order to prevent the occurrence of non-stochastic effects, a limit of 0.5 Sv (50 rem) in a year applies to all tissues except the lens of the eye; for the lens of the eye the recommended annual limit is 0.15 Sv (15 rem). These values apply irrespective of whether tissues are exposed singly or in combination with other tissues, and are intended to constrain exposures that fulfil the limitation for stochastic effects given below.
- (b) For the limitation of stochastic effects, the limit on the annual effective dose equivalent (H_E) is 50 mSv (5 rem).

4.2. Secondary and derived limits

In practice it is necessary to use secondary limits for external and internal exposures when compliance with the primary dose limits cannot be demonstrated directly.

In the case of external exposure, secondary limits may be expressed in terms of dose equivalent index (deep dose equivalent index and shallow dose equivalent index).

In the case of internal exposure, secondary limits may be expressed in terms of the annual limit of intake (ALI). Internationally recommended values for the ALIs consistent with the dose equivalent limits are given in ICRP Publication 30 [9] and in Safety Series No. 9 [10]. An ALI value for radon daughters is given in ICRP Publication 32 [11] and Safety Series No. 26 [8]. Internal exposure may also be

controlled by the use of derived limits, such as a limit for the annual time integrated concentration in air. Compliance with secondary or derived limits will ensure compliance with the annual dose equivalent limits.

4.2.1. Radon daughters

The ICRP recommends an annual limit for intake by inhalation of ^{222}Ra daughters of 0.02 J of inhaled potential alpha energy. This limit has been set based on both epidemiological and dosimetric approaches. This corresponds to a derived air concentration of $8.3 \mu\text{J}/\text{m}^3$ or 0.4 working level, and to an annual limit of exposure (ALE) of $0.017 \text{ J} \cdot \text{h}/\text{m}^3$ (5 working level months (WLM)). The working level is the sum of the alpha energies released by the decay of radon daughters which are in equilibrium with 3.7 Bq (100 pCi) radon. This amounts to $1.3 \times 10^5 \text{ MeV}/\text{L}$. Therefore, one working level represents any combination of the short lived radon daughters in one litre of air that will result in the ultimate emission of $1.3 \times 10^5 \text{ MeV}$ of alpha energy, taking no account of radon itself. If a worker is exposed to a radon daughter concentration of 1 working level (WL) for 1 working month (170 h) then the exposure is 1 WLM. The annual limit expressed in WLM is $0.4 \times 12 = 4.8$, i.e. 5 WLM.

4.2.2. Ore dust

The derived limit for respirable uranium ore dust is based on the total alpha activity of the long lived nuclides in the ^{238}U decay chain, assumed to be in secular equilibrium with ^{238}U . The internal doses and the ALI are highly dependent upon the ICRP inhalation classes for the individual nuclides. It is probable that the nuclides in ore are class Y. On this basis the ALI for uranium ore dust in terms of total long lived alpha activity is $1.7 \text{ kBq} \cdot \text{h}/\text{m}^3$. The corresponding ALE is $1.5 \text{ kBq} \cdot \text{h}/\text{m}^3$ for uranium. Should the ore body be seriously out of radioactive equilibrium, the ALI should be calculated using the actual radionuclide ratios.

4.2.3. Uranium

The ALI for uranium is based on the naturally occurring mixture of isotopes and on the assumption of an aerodynamic median activity diameter of $1 \mu\text{m}$. Uranium concentrates vary widely in solubility, depending on the specific mill process. For class Y material, the uranium ALI is 1.5 kBq and the derived air concentration is $0.61 \text{ Bq}/\text{m}^3$. For class W material, the ALI for uranium is 28 kBq.

For soluble uranium compounds (class D), chemical damage to the kidney is more important than radiological effects. Based on a threshold concentration of 900 μg of uranium in the kidney, the acute inhalation limit for uranium is 20 mg. For chronic exposure a daily intake of 1 mg could result in the kidney limit being reached in approximately 10 years.

4.3. Combined exposures

When *external and internal exposures* are received together, the individual dose limits will not be exceeded if the following two conditions are met:

$$\frac{H_{i,d}}{50} + \frac{i_j}{j \text{ ALI}_j} \leq 1$$

$$\frac{H_{i,s}}{500} \leq 1$$

where $H_{i,d}$ is the annual deep dose equivalent index in mSv, $H_{i,s}$ is the annual shallow dose equivalent index in mSv, i is the annual intake of nuclide j in Bq, and ALI_j is the annual limit of intake of nuclide j in Bq.

In the uranium mines this additivity has the effect of requiring the inhalation of radon and its daughters to be kept below their recommended limits by an amount that depends on the exposure to external radiation and ore dust. The combination formula is:

$$\frac{H_{i,d}}{50} + \frac{I_{rd}}{0.02} + \frac{I_{odu}}{1700} \leq 1$$

where $H_{i,d}$ = Annual deep dose equivalent index in mSv

I_{rd} = Intake of radon daughters expressed in J

I_{odu} = Intake of uranium ore dust expressed in becquerels of total long lived alpha activity

In terms of exposure and practical units the formula may be expressed as follows:

$$\frac{H_{i,d}}{50} + \frac{E_{rd}}{5} + \frac{E_{odu}}{1500} \leq 1$$

where $H_{i,d}$ = Annual deep dose equivalent index in mSv

E_{rd} = Exposure to radon daughters in working level months

E_{odu} = Exposure to uranium ore dust in total long lived alpha Bq·h/m³

Similar additivity considerations apply in milling operations. In cases where inhalation or ingestion of concentrates or other radioactive substances occurs the formula should be modified accordingly.

The ICRP recognizes that in some mining operations it may not be possible to operate within the combined limits. The competent authority will then have to take a decision on how best to deal with these situations.

For practical purposes, other limits are often set in addition to primary dose equivalent limits. Authorized limits are set by the competent authority or the management and are generally lower than the primary or derived limits. When

authorized limits are specified by the management, they are designated as operational limits. Reference levels may be established by the competent authority for any of the quantities determined in the course of radiation protection programmes, whether or not there are limits for these quantities. A reference level is not a limit, but it is useful in determining a course of action when the value is exceeded.

5. MEASUREMENTS OF DOSE (EXPOSURE)

The following is only a brief outline of measurement of external dose and internal exposures. Detailed information on this topic can be found in Safety Series No. 43 [12].

5.1. Measurement of external radiation dose

In a mine or a mill, the external dose is given by gamma and beta radiation emitted by uranium or thorium and by their daughter products. An approximation of dose rate can be made from the grade and distribution of the ore (geometry of the source).

Monitoring of dose rate is usually carried out with instruments for measuring dose rates. The most widely used instruments are count rate meters using Geiger-Müller detectors similar to the instruments used in prospecting. However, these instruments have to be properly calibrated (for proper gamma energy) as their response may be very energy dependent.

Dose rate measurements are suitable for engineering control purposes as well as for determination of exclusion zones within the facility and also outside the facility.

Relatively inexpensive personal dosimeters utilizing film or thermoluminescent devices exist for monitoring external doses.

Generally in uranium mining and milling the monitoring of whole body dose is sufficient. In some limited special situations it might be necessary to monitor also the dose to the extremities by using special individual dosimeters.

5.2. Measurement of internal exposure

5.2.1. Measurement of radon

As indicated earlier, the contribution of radon to the dose is very minimal, and therefore it is not necessary to measure radon for exposure determination. Radon measurements are sometimes undertaken for engineering control.

5.2.2. *Measurement of radon daughters*

Measurement of radon daughter concentrations is usually made by drawing a known quantity of air through a filter, and counting the alpha activity on the filter. From the known air volume, time of sampling and time of counting, the concentration of radon daughters can be calculated. There are various methods of radon daughter assessment (Kuzmetz, Tsivoglu, Rolle, etc.). Various types of solid state electronic 'Instant Working Level Meters' have been developed. These operate on the basis of grab sampling. Some of them have also provisions for measuring individual radon daughters and also thoron daughters.

Individual exposures of workers are usually estimated from radon daughter concentrations measured in the workplace and from the known occupancy time in the workplace. Personal dosimeters to measure integrated radon daughter exposures of individuals have been developed. They usually utilize a small battery powered pump to draw air at a certain rate through the instrument. A more extensive utilization of personal dosimeters in the future is likely, namely if the research efforts into the development of a simple passive personal dosimeter are successful.

5.2.3. *Measurement of long lived radionuclides*

Concentrations of ore dust or concentrate dust are measured by pumping air samples through high efficiency filters and analysing the sample on the filter by alpha counting, chemical analysis or separation followed by counting individual radionuclides.

The results of measurements are used mainly for engineering control, but could be used to calculate the effective dose equivalent from exposure to long lived radionuclides for individual workers.

6. PROTECTION MEASURES

The methods of maintaining a safe working environment are basically the same in uranium mines as in conventional mines but the unique properties of radon and radon daughters impose special requirements, particularly with regard to ventilation. For a given radon concentration the daughter concentrations, and hence the risk, increase rapidly with time because of the short half-lives of radon daughters. Therefore, the more rapidly contaminated air is removed from work areas, the lower will be the concentration of radon daughters in any given location. Basic methods of control in order of importance are: mechanical ventilation; confinement or suspension of radiation source; and personal protection and job rotation. Air cleaning for the removal of radon daughters is being used on a limited basis as a supplement to mechanical ventilation.

Primary and auxiliary systems of mechanical ventilation are used to dilute and remove the radon and radon daughter concentrations from the mine. The primary ventilation moves air into, through and out of the mine via shafts, and drifts by means of large fans usually located on the surface. Auxiliary systems distribute air from the main airways to workplaces underground by means of small fans and flexible ducts. Auxiliary systems are extended, modified or replaced as the mine workings advance.

Radon and radon daughters in worked-out areas can be confined and prevented from entering into the working areas by means of stoppings made as airtight as possible. As radon is soluble in water and emanates from seepages upon exposure to the mine atmosphere, using pipes for removing water is another useful confinement technique. Suppression of radon emanation from mine surfaces by pressurizing the mine atmosphere is feasible if the rock is porous. The use of coatings has been found to be useful in sealing the pores and crevices of exposed rock surfaces.

Personal protective equipment such as respirators are used on a non-routine basis in areas where the levels of airborne contamination are high. Job rotation is recommended in mines having areas with high levels of external radiation for which no practical means of control is available.

In mills, the control measures to protect the health of workers differ in degree and kind from those applied in underground operations. Primary control is achieved by confinement, although local exhaust ventilation is used extensively to prevent the escape of dust, gases, and fumes from the operations. The processes in the mill are largely automated, thus minimizing direct contact with materials on the part of the mill workers. The working practices are designed to reduce the likelihood of contaminant release, and proper housekeeping is performed regularly to remove contaminants that have accumulated on surfaces.

Adherence to thorough personal hygiene practices should be required of all personnel in contact with uranium concentrates. Washing before eating or smoking, and showering at the end of each work shift should be mandatory. Gloves should be worn for any direct contact with concentrates. Lunch rooms, rest rooms, and changing rooms should be isolated from working areas, and provided with convenient access to washing facilities.

Personal respiratory protection is merely an adjunct to engineering control measures. Primary emphasis should always be on maintaining the concentrations of airborne radioactive contaminants in the working environment within permissible limits. However, respirators or breathing apparatus are necessary in some, usually exceptional, circumstances such as during failure of the ventilation system or during a maintenance task for which adequate ventilation is not available. The use of respirators is not compatible with work involving sustained physical effort or complete freedom of movement.

All persons employed in mining and milling of uranium should be medically examined before such work and at appropriate intervals thereafter. The pre-employment and periodic medical examinations should be adequate to provide information on the general health of the worker and to prevent and detect changes which may be related to his occupational exposure.

The pre-employment examination should be a thorough examination. In general, periodic examinations should be done at yearly intervals for exposed workers. The periodic examinations should include an enquiry into the general health of the worker, with special emphasis on certain organ systems. A medical examination should be carried out at the termination of employment.

7. ROLE OF INTERNATIONAL ORGANIZATIONS

7.1. International Commission on Radiological Protection

During its existence ICRP has proposed various limits for exposure to radon daughters, and of course a limit for external dose. In 1977 ICRP Publication 26 [13] proposed a new system of dose exposure limitation based on the effective dose equivalent principle.

In Publication 32 [11] the ICRP proposes specific ALIs and ALEs to be used for determination of the contribution of radon and thoron daughters and long lived dust to the effective dose equivalent.

The recommended annual limit for intake by inhalation, the ALI, for ^{222}Ra daughters, in terms of inhaled potential alpha energy, is 0.02 J in a year. The corresponding derived air concentration [14] expressed in practical units previously widely used is then 0.4 working level.

The system of dose limitation of the ICRP requires the addition of exposures to external radiation and intakes of radioactive material. In the special case of exposure in uranium mines this additivity has the effect of requiring the inhalation of radon daughters to be kept below the recommended limit by an amount that depends on the exposure to external radiation and ore dust. A reduction of 20% is common.

These recommendations are intended for competent authorities for general application and they may not always be appropriate for application in particular cases. The ICRP is aware that some mining conditions are such that it may not be possible to operate within the combined limits recommended by it on a year to year basis. The national authorities will then have to take a decision on how best to deal with these few, but difficult, situations.

7.2. The International Atomic Energy Agency (IAEA), the International Labour Organisation (ILO), and the World Health Organization (WHO)

Since its inception in 1957, the IAEA has been active in the field of radiation protection in mining and milling of radioactive materials. Mining of uranium is the only component in the nuclear fuel cycle which has resulted in a significant incidence of occupational illness, particularly lung cancer. The IAEA, recognizing the significance of the problem, held a symposium in 1963 jointly with ILO and WHO

on Radiological Health and Safety in Mining and Milling of Nuclear Materials (see Ref. [15]).

As the uranium mining and milling industries continued to expand rapidly, the IAEA felt the need for the development of a code of practice and therefore convened a panel of experts jointly with ILO in 1965 for this purpose (see Ref. [8]).

A panel was convened in Vienna in July 1973 jointly with ILO, to deal with the detailed methods of monitoring and surveillance in mining and milling of radioactive materials [12].

A joint ILO/IAEA/WHO/CEA symposium (1974) was held in Bordeaux, France, on Radiation Protection in Uranium and Thorium Mining and Milling (see Ref. [16]).

The subject of radiological risks in uranium mining was also covered in general by the Panel on Inhalation Risks from Radioactive Contaminants. The report was published in 1972 as Technical Reports Series No. 142 [17]. It contains one chapter on uranium mining which dealt, among other things, with the principles of dose estimation from radon decay products, working limits and monitoring of radon and radon daughters.

In 1978 a joint IAEA/ILO/WHO Advisory Group was convened in Portoroz, Yugoslavia, to revise Safety Series No. 26 [8]. The Advisory Group thoroughly revised the Code of Practice to take account of the new recommendations of the ICRP particularly contained in its Publications 24, 26 and 32 [18, 13, 11]. New developments in radiation protection monitoring in mines and mills and new developments in engineering and ventilation controls were taken into account.

Jointly with ILO and WHO the IAEA held a Regional Training Course for Africa on Radiation Protection in Exploration, Mining and Milling of Radioactive Ores in Gabon in November 1983.

Other IAEA programmes include Safety Series No.82 on the Application of the Dose Limitation System to the Mining and Milling of Radioactive Ores.

The IAEA has been actively co-operating with the Organisation for Economic Co-operation and Development/Nuclear Energy Agency (OECD/NEA) in the programme on personnel monitoring of radon daughter exposures as well as on uranium mill tailings.

The IAEA was represented as a witness at the Cluff Lake Inquiry in 1977 at the invitation of the Government of Saskatchewan. This Board of Inquiry, headed by a judge, heard testimony from 100 scientists, pro- and anti-nuclear people, and the public to examine radiation protection and other aspects of the uranium mining industry for the Cluff Lake area in Saskatchewan.

The IAEA has been active for many years in studying the radiological and technological bases for the management of uranium processing wastes. Sessions pertaining to these subjects have been included in symposia, seminars and international meetings on waste management.

At the United Nations Conference on the Human Environment, which took place in Stockholm from 4 to 16 June 1972 [19], governments were asked to explore

with the IAEA and other appropriate organizations international co-operation on radioactive waste problems, including the problems of mining and mill disposal. The IAEA has therefore maintained its interest in this field.

The IAEA convened a panel of experts to develop the background reference material required to formulate a code and guide on management of wastes from the mining and milling of uranium and thorium ores [20]. Member States were recommended to take the Code and Guide of Safety Series No. 44 into account when formulating national regulations.

Since 1987, IAEA expert groups have been considering the current practices and options for confinement of uranium mill tailings. Technical Reports Series No. 209 [21] presents an overview of current practices for the impoundment of mill tailings, and of site selection. Related to the subject was a co-ordinated research programme on the source, distribution, movement and deposition of radium in inland waterways and aquifers.

In view of the developments which have taken place in recent years, and the current and future importance of this subject, the IAEA convened an international symposium in 1982 (in co-operation with OECD/NEA) in the USA to review processing technology and practices for the management of wastes from uranium mining and milling. This was the first IAEA symposium devoted exclusively to this field [22].

The IAEA has revised its Safety Series No. 44 on the Code of Practice on Management of Wastes from the Mining and Milling of Uranium and Thorium Ores.¹

7.3. OECD/NEA

In the field of radiological health and safety, the OECD/NEA has the task of encouraging co-ordination of government regulatory policies and practices, promoting exchange of information, and co-ordination of research and development among its member countries. The work of the NEA is carried out through a number of specialized committees.

Major emphasis is presently focused on the problems of radiation protection and environmental impact of nuclear fuel cycle facilities, with special attention to the front end (uranium mining and milling) and the back end (waste management) of the fuel cycle. In this context, the NEA readily appreciated the increasing attention being given in several countries to problems associated with the exposure of man to radon, thoron, and their daughters, including dosimetry and measurement, and began to work actively in this field in 1976.

¹ INTERNATIONAL ATOMIC ENERGY AGENCY, Safe Management of Wastes from the Mining and Milling of Uranium and Thorium Ores, Safety Series No. 85, IAEA, Vienna (1987).

However, at the time the NEA first became involved in this field, one of the most urgent problems to be solved was that of ensuring adequate personal dosimetry for uranium miners. Consequently, the NEA was urged to organize a specialists meeting on personal dosimetry and area monitoring for radon and radon daughters to provide an international forum for exchanging information and reviewing problems in this field. The meeting was held at Elliot Lake, Canada, in October 1976 [23]. A second specialists meeting on the same subjects was held in Paris in November 1978 to review further developments in this area [24]. These meetings demonstrated that the overall problem associated with exposure to radon and radon daughters had many facets, each of which has been the subject of considerable attention in recent years in several countries for various reasons. The meetings indicated that further work was required in two areas: dosimetry, and metrology and monitoring.

Conclusions and recommendations that emerged from the two specialists meetings were discussed by the Committee on Radiation Protection and Public Health. As a consequence, in September 1979 the Committee approved a programme of work in the areas of dosimetry and monitoring of radon, thoron, and their daughters, and also approved the setting up of an international group of experts on radon dosimetry and monitoring to undertake this work.

A group of experts has been set up on radon daughter dosimetry and monitoring to handle all the work. The first phase of the work, on dosimetric aspects, was completed in 1983. Towards the end of the first phase, the group of experts has reviewed the redefined objectives and scope of occupational and environmental monitoring for radon and its daughters in order to redefine proposals for the second phase of the work, on metrology and monitoring, for subsequent review and approval by the Committee. Two publications were prepared, one on Dosimetry Aspects of Exposure to Radon and Thoron Daughter Products [25] and the other on Metrology and Monitoring of Radon, Thoron and their Daughter Products [26]. The NEA also presented a publication on International Intercalibration and Intercomparison of Radon, Thoron and Daughters Measuring Equipment [27].

REFERENCES

- [1] LUNDIN, P.E., WAGONER, J.K., ARCHER, V.E., Radon Daughter Exposures and Respiratory Cancer Quantitative and Temporal Aspects, National Institute for Occupational Safety and Health, and National Institute of Environmental Health Sciences, Joint Monograph No. I (1971).
- [2] SEVC, J., KUNZ, E., PLACEK, V., Lung cancer mortality in uranium miners and long-term exposure to radon daughter products, *Health Phys.* **30** (1976) 433-437.
- [3] KUNZ, E., SEVC, J., PLACEK, V., Lung cancer mortality in uranium miners, *Health Phys.* **35** (1978) 579-580.
- [4] MULLER, J., WHEELER, W.C., "Causes of death in Ontario uranium miners", Radiation Protection in Mining and Milling of Uranium and Thorium, International Labour Office, Geneva, (1974) 29-42.

- [5] ARCHER, V.E., WAGONER, J.K., LUNDIN, F.E., Lung cancer among uranium miners in the United States, *Health Phys.* **25** (1972) 351-371.
- [6] DE VILLIERS, A.J., WINDISH, J.P., Lung Cancer in a Fluorspar Mining Community. I. Radiation, Dust and Mortality Experience, *Br. J. Ind. Med.* **21** (1964) 94-109.
- [7] DORY, A.B., "Practical difficulties related to implementation of ICRP recommended dose limitation in uranium mines", Application of the Dose Limitation System for Radiation Protection: Practical Implications (Proc. Topical Sem. Vienna, 1979), IAEA, Vienna (1979) 287-314.
- [8] INTERNATIONAL ATOMIC ENERGY AGENCY, Radiation Protection of Workers in the Mining and Milling of Radioactive Ores, 1983 Edn, Safety Series No. 26, IAEA, Vienna (1983).
- [9] INTERNATIONAL COMMISSION ON RADIOLOGICAL PROTECTION, Limits for Intakes of Radionuclides by Workers, ICRP Publication 30, Pergamon Press, Oxford and New York (1979).
- [10] INTERNATIONAL ATOMIC ENERGY AGENCY, Basic Safety Standards for Radiation Protection, 1982 Edition, Safety Series No. 9, IAEA, Vienna (1983).
- [11] INTERNATIONAL COMMISSION ON RADIOLOGICAL PROTECTION, Limits for Inhalation of Radon Daughters by Workers, ICRP Publication 32, Pergamon Press, Oxford and New York (1981).
- [12] INTERNATIONAL ATOMIC ENERGY AGENCY, Manual on Radiological Safety in Uranium and Thorium Mines and Mills, 1976 Edition, Safety Series No. 43, IAEA, Vienna (1976).
- [13] INTERNATIONAL COMMISSION ON RADIOLOGICAL PROTECTION, Recommendations of the International Commission on Radiological Protection, ICRP Publication 26, Pergamon Press, Oxford and New York (1977).
- [14] INTERNATIONAL COMMISSION ON RADIOLOGICAL PROTECTION, Biological Effects of Inhaled Radionuclides, ICRP Publication 31, Pergamon Press, Oxford and New York (1980).
- [15] INTERNATIONAL ATOMIC ENERGY AGENCY, Radiological Health and Safety in Mining and Milling of Nuclear Materials (Proc. Symp. Vienna, 1963), IAEA, Vienna (1964).
- [16] DOUSSET, M., "Permissible levels pertaining to radiation hazards in the mining and milling of uranium and thorium ores" (Proc. Symp. Bordeaux, 1974), Rep. VZUPPR-P-2146/74, Vytvořilový Zavod Uranoveho Prumyslu, Pribram, CSSR (1974) (in Czech).
- [17] INTERNATIONAL ATOMIC ENERGY AGENCY, Inhalation Risks from Radioactive Contaminants, Technical Reports Series No. 142, IAEA, Vienna (1972).
- [18] INTERNATIONAL COMMISSION ON RADIOLOGICAL PROTECTION, Radiation Protection in Uranium and Other Mines, ICRP Publication 24, Pergamon Press, Oxford and New York (1977).
- [19] United Nations Conference on the Human Environment (Proc. Int. Conf. Stockholm, 1972), Erich Schmidt Verlag, Berlin (West) (1973) 382 pp.
- [20] INTERNATIONAL ATOMIC ENERGY AGENCY, Management of Wastes from the Mining and Milling of Uranium and Thorium Gases, 1976 Edition, Safety Series No. 44, IAEA, Vienna (1976).
- [21] INTERNATIONAL ATOMIC ENERGY AGENCY, Current Practices and Options for Confinement of Uranium Mill Tailings, Technical Reports Series No. 209, IAEA, Vienna (1981).
- [22] INTERNATIONAL ATOMIC ENERGY AGENCY, Management of Wastes from Uranium Mining and Milling (Proc. Symp. Albuquerque, USA, 1982), IAEA, Vienna (1982).
- [23] NUCLEAR ENERGY AGENCY OF THE OECD, Personal Dosimetry and Area Monitoring Suitable for Radon and Daughter Products (Proc. Symp. Elliot Lake, Canada, 1976), OECD, Paris (1977).
- [24] NUCLEAR ENERGY AGENCY OF THE OECD, Radon Monitoring (Proc. Specialists Mtg Paris, 1978), OECD, Paris (1984).
- [25] NUCLEAR ENERGY AGENCY OF THE OECD, Dosimetry Aspects of Exposure to Radon and Thoron Daughter Products, NEA Experts Report, OECD, Paris (1983) 111 pp.

- [26] NUCLEAR ENERGY AGENCY OF THE OECD, Metrology and Monitoring of Radon, Thoron and their Daughter Products, NEA Experts Report, OECD, Paris (1985) 137 pp.
- [27] NUCLEAR ENERGY AGENCY OF THE OECD, International Intercalibration and Intercomparison of Radon, Thoron and Daughters Measuring Equipment. Part I. Radon Measurement, OECD, Paris (1986) 62 pp.

A MILLING AND CONVERSION PILOT PLANT AT NINGYO TOGE, JAPAN

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Abstract

A MILLING AND CONVERSION PILOT PLANT AT NINGYO TOGE, JAPAN.

Operation of the milling and conversion pilot plant using a Power Reactor and Nuclear Fuel Development Corporation (PNC) process started in March 1982. The plant which has a producing capacity of 290 t uranium hexafluoride (approximately 200 t uranium per year) is now operating satisfactorily. It has been considerably improved on the basis of experience gained from continuous operation. The raw materials are the pregnant solution in the form of uranyl sulphate solution from heap leaching and/or yellow cake followed by dissolution into sulphuric acid solution. Tri-n-octylamine (TNOA) is used as the extraction reagent in the solvent extraction section. Uranium is transferred in the form of solution to the uranium tetrafluoride precipitation section. In the conventional refining process, yellow cake is dissolved in nitric acid solution, and uranium in the solution is extracted by tributyl phosphate (TBP). In all sections, except for the TBP extraction section, uranium is transferred in the form of solid. The PNC process differs considerably from the conventional refining process. In the conventional process, the impurities in the nitric acid solution dissolved yellow cake are nearly removed at the TBP extraction section; in the PNC process, the specific elements such as molybdenum are not removed by amine extraction however, they are, in particular molybdenum, almost removed at the uranium tetrafluoride precipitation section. Therefore, uranium tetrafluoride produced by the PNC process bears a comparison with that produced by the conventional process. Ninety tonnes of uranium hexafluoride (approximately 60 t uranium) were produced in 1984 in addition to renewing or improving the uranium tetrafluoride precipitation vessel and other equipment.

1. INTRODUCTION

The raw material supplied to the pilot plant is mainly yellow cake which is imported from several countries. The rest of the raw material (several per cent) is in the form of uranyl sulphate solution which is produced by vat leaching [1] ore from the Ningyo Toge mine. The pilot plant produces uranium hexafluoride from these raw materials.

The difference between this process and the conventional refining process is shown in Fig. 1, and the chief characteristics of this process are as follows:

- (a) Except for the uranium tetrafluoride dehydration and the fluorination steps, the wet process is carried out, and as every process step is simple, the operation and process control are easy. Molybdenum is separated from uranium in the uranium tetrafluoride precipitation step.
- (b) The amine extraction (AMEX) process is applied in the solvent extraction system of this pilot plant.
- (c) The uranium tetrafluoride powder is produced in a more reactive form for fluorine gas than that for the conventional dry process.

The pilot plant which has a capacity to produce 290 t uranium hexafluoride was built at the end of 1981. After the test operation of the pilot plant, operation started in early 1982. It has been operating satisfactorily and several improvements have been made. In 1984, the pilot plant produced 90 t uranium hexafluoride (approximately 60 t uranium). It should also be mentioned that a uranium hexafluoride precipitation vessel, a pneumatic conveyor, a yellow cake drum handling apparatus and other equipment were renewed or improved.

2. PILOT PLANT

Comments on the pilot plant are given in the following subsections.

2.1. Vat leaching and ion exchange

Vat leaching has been carried out to find the optimal condition of leaching and to reduce the cost.

Typically, the uranium content in the ore is 0.08%. The ore is supplied for the vat leaching without crushing, and the uranium in the ore is extracted from the crude ore by percolation of the sulphuric acid solution. The leaching operation is carried out by using three leaching vats, loading 500 t ore in each vat. The vat is made of concrete, lined with acid proof paint and timbers, and has a gravel filter at the bottom.

The uranium recovery is approximately 90% and the acid consumption is expected to be approximately 15 kg/t ore. The period of leaching – carrying the ore into the vat, percolation of the sulphuric acid solution through the ore, washing the residue, neutralizing the residue, and carrying the residue out of the vat – is approximately 24 d.

An ion exchange process is used to concentrate uranium up to 3 to 5 g/L from a low concentrated uranium solution such as a uranium poor pregnant solution and wash water. The uranium is loaded on the strong base anionic resin as anionic complex, $\text{UO}_2(\text{SO}_4)_3^{4-}$, and is eluted with 1 mol/L of sulphuric acid

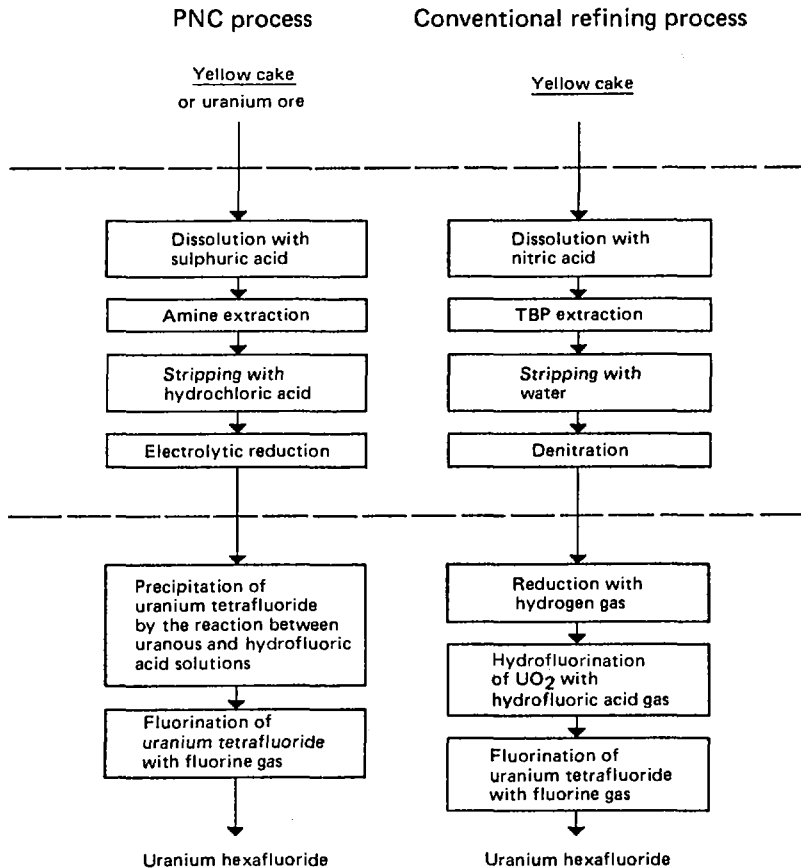


FIG. 1. The difference between a Power Reactor and Nuclear Fuel Development Corporation process and a conventional refining process.

solution. The uranium pregnant solution is transferred to the following solvent extraction section, and the barren eluate and raffinate are transferred from the solvent extraction section for reuse as an eluate.

The chemical data of the ore from the Ningyo Toge mine and the uranium pregnant solution are shown in Table I.

2.2. Solvent extraction

The plant has two systems of solvent using tri-n-octylamine (TNOA) as a reagent for uranium extraction: one treats the pregnant solution (3 to 5 g/L uranium) from the vat leaching followed by ion exchange, and the other treats the uranyl sulphate solution (200 g/L uranium) dissolved yellow cake in

TABLE I. CHEMICAL DATA OF THE ORE
FROM NINGYO TOGE MINE AND THE
PREGNANT SOLUTION

	Uranium ore (wt%)	Leaching solution (g/L)
Al	6.36	2.94
Ag	0.003	0.0007
Ca	0.5	0.023
Cd	0.0018	0.00059
Cr	0.0062	0.0027
Cu	0.002	0.011
Fe	2.52	5.00
K	1.66	0.034
Mg	0.21	0.713
Mn	0.054	0.419
Mo	0.004	0.00076
Na	4.08	0.061
Ni	0.001	0.0059
Si	31.08	0.56
Ti	0.15	—
Th	0.0012	0.0016
V	0.005	0.0024
Zn	0.022	0.0057
U	0.12	1.39

sulphuric acid. In the extraction section including extraction and stripping, mixer-settlers are used, except for the uranium extraction from pregnant solution produced by the vat leaching section. In the extraction section treating the pregnant solution, a sieve tray extraction tower was adapted in order to compare the operation data of the mixer-settlers with those of the sieve tray tower. As extraction reagent, 99% purity TNOA was used. The TNOA concentrations in the organic phases treating the solution dissolved yellow cake and the pregnant solution from the vat leaching section are 0.6 and 0.2 mol/L respectively.

In the conventional AMEX process, the uranium loaded in the organic phase is generally stripped with sodium chloride, carbonate or ammonium

sulphate solution. However, in the process of the pilot plant, the uranium loaded in the organic phase is stripped with hydrochloric acid solution, as shown in Fig. 2.

The TNOA is highly selective for uranium and presents difficulties only when a significant amount of either molybdenum or pentavalent vanadium is present. As shown in Fig. 2, the molybdenum in uranium solutions before and after the solvent extraction is not greatly different in quantity. The decontamination factor value of molybdenum in the solvent extraction is approximately 2.

2.3. Electrolytic reduction

The stripping solution (100 g/L uranium) from both solvent extraction systems is jointly fed to the electrolytic cells in which the hexavalent uranium in the solution is reduced to tetravalent form.

The schematic electrolytic reduction cells are shown in Fig. 3. One cell of the apparatus consists of an anode plate, a cathode plate and a cation exchange resin membrane by which the space between the two plates is divided into two compartments, an anolyte compartment and a catholyte compartment; 2N-sulphuric acid solution is supplied into the anolyte compartment and uranyl solution into the catholyte compartment. The anode plate and the bipolar plate consist of titanium metal plated on one side with platinum, and the cathode plate consists of a titanium metal. The uranyl ion in the solution is reduced to uranous ion with a reduction rate of 99%.

2.4. Hydrofluorination

The uranous solution and the hydrofluoric acid solution which is diluted with the discharged solution from the reaction vessel are continuously fed into the reaction zone in the bottom part of the vessel, as shown in Fig. 4.

The reaction takes place in the lower part of the vessel, and hydrated uranium tetrafluoride grains grow from 50 to 100 μm dia while suspended in the reaction zone.

The conditions for the formation of hydrated uranium tetrafluoride are as follows:

- (a) It is desirable for the growing of the uranium tetrafluoride grain that the concentrations of uranium and hydrofluoric acid supplied into the reaction zone are kept considerably low.
- (b) The stirring is kept constantly at a speed which keeps the grains suspended within the reaction zone; however it is desirable that the stirring be slow enough for the grains to be hardly present in the overflow solution.
- (c) The reaction temperature is kept at approximately 90°C.

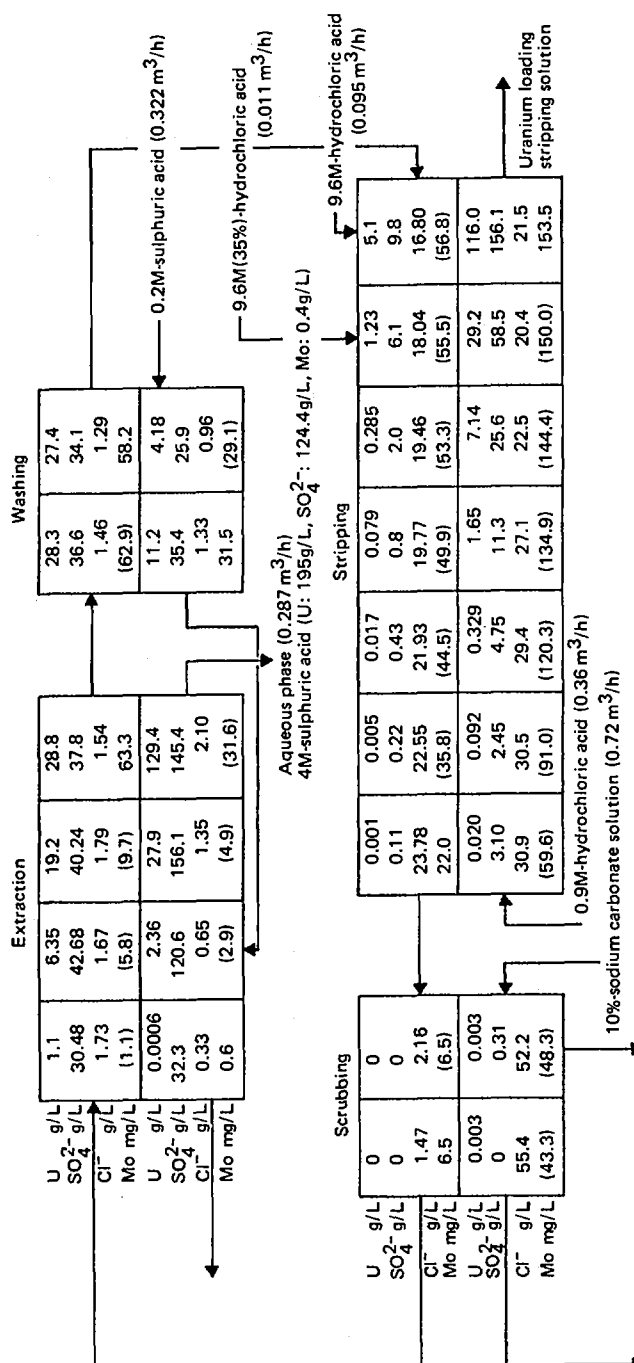


FIG. 2. Distribution of uranium, sulphate ion, chloride ion, and molybdenum in mixer-settlers.

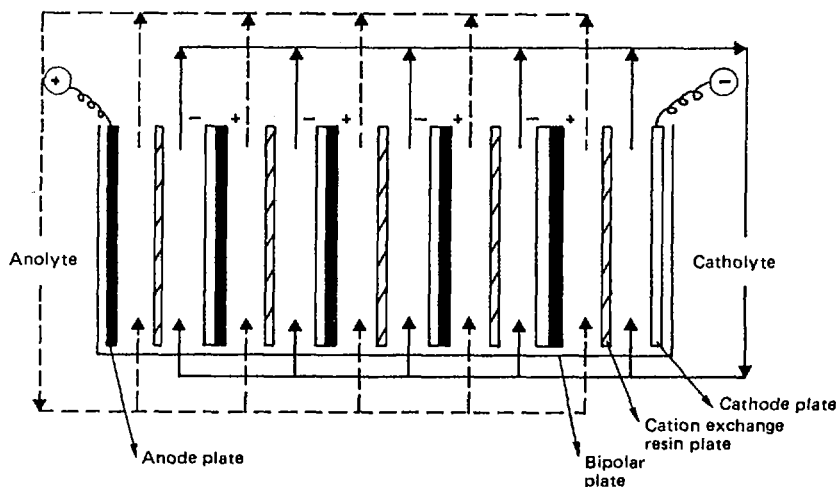


FIG. 3. Schematic electrolytic reduction cells.

Uranium tetrafluoride grains are separated from the pregnant liquor after a few hours within the reaction vessel. They are washed with ion exchanged water, filtered and dried. As the hydrated uranium tetrafluoride product has an excellent permeability, the product taken from the reaction vessel is fully scrubbed in a comparatively small amount of water.

The hydrated uranium tetrafluoride was identified as $\text{UF}_4 \cdot 1-1.2 \text{ H}_2\text{O}$, which is a mixture of the monoclinic type ($\text{UF}_4 \cdot 0.75 \text{ H}_2\text{O}$), the cubic type ($\text{UF}_4 \cdot 1.5 \text{ H}_2\text{O}$), and the orthorhombic type ($\text{UF}_4 \cdot 2.5 \text{ H}_2\text{O}$), by differential thermal analysis, thermogravimetric analysis and X-ray diffraction.

Approximately 70 to 80% of the content of the crystallization water in the mixture is dehydrated at 150 to 200°C, and the rest is dehydrated at 380°C by means of heating in nitrogen gas flow.

A grain of tetrafluoride consists of an aggregation of many crystals. Therefore, the velocity of reaction between tetrafluoride grain and fluorine gas is fast, and the excess fluorine gas is almost not required for the reaction.

The dried product contains virtually no impurities, as shown in Table II.

The molybdenum which was hardly separated in the solvent extraction section is almost separated from uranium within this section. As shown in Table II, molybdenum is hardly contained in the uranium tetrafluoride. The value of the decontamination factor for molybdenum in the hydrofluorination section is as high as approximately 900 to 1000.

2.5. Dehydration

The fluidized-bed tower was selected for the dehydration, because the tower dehydrates rapidly from the hydrated uranium tetrafluoride grains when fluidizing the grains in an upward nitrogen flow.

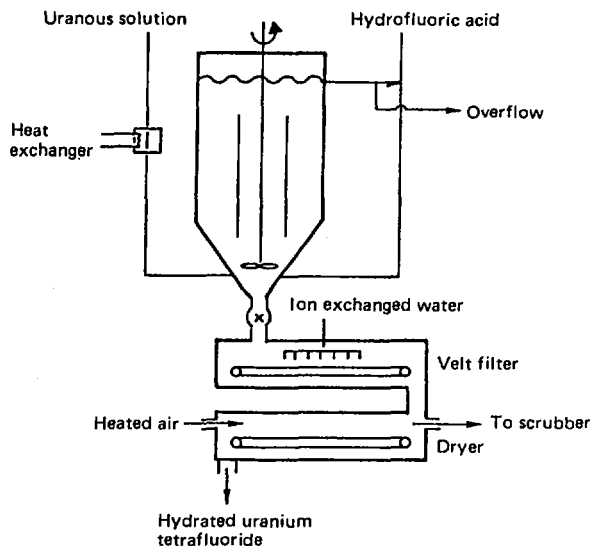


FIG. 4. Schematic hydrofluorination section.

The rapid dehydration decreases the formation of uranium oxide. The formation of uranium oxide by dehydration was approximately 0.8 to 1 wt% and 1 to 3 wt% with the batch and continuous treatments respectively, with the temperature within the range of 300 to 400°C and the retention time calculated from an experimental formula to remove 99.9% of the crystallization water in uranium tetrafluoride. By experimental and theoretical analyses of the dehydration of $\text{UF}_4 \cdot 1-1.2 \text{ H}_2\text{O}$ and of the formation of UO_2 , the optimum condition was found. The tower is now being operated under the following conditions: batch treatment loading; 500 kg of hydrated uranium tetrafluoride for 1 to 1.5 h at 380°C.

2.6. Fluorination

The dehydrated uranium tetrafluoride is converted to uranium hexafluoride by reacting with fluorine gas. A fluidized-bed tower has been used for the conversion, because of the ease in controlling the heat transfer. The internal temperature of the tower is maintained at 400°C. The sintered alumina is used as an inert bed material. The tower has been smoothly operated.

The gaseous uranium hexafluoride is trapped in two cold traps which are cooled to -10°C and -50°C.

TABLE II. CHEMICAL DATA OF HYDRATED URANIUM TETRAFLUORIDE

Al	3.0	3.6	3.0
B	<0.1	<0.1	<0.1
Ca	<0.1	5.7	5.1
Fe	16.4	5.3	11.9
Mg	0.3	<0.2	<0.2
Mo	0.29	2.3 ^a	2.1 ^a
Na	15	1.8	1.5
Si	4.0	4.2	4.0
Ti	0.4	5.6	6.9
V	<1	0.36	0.36

(Unit: ppm on a U-basis)

^a The molybdenum content in the yellow cakes is 0.18 to 0.25 wt%.

3. IMPROVED APPARATUS

A yellow cake drum handling apparatus and a pneumatic conveyor were newly equipped, and an electrolytic reduction apparatus and a reaction vessel (uranium tetrafluoride precipitation vessel) were improved.

3.1. Yellow cake drum handling apparatus

The apparatus consists of a roll conveyor and a glove box separated into three compartments: an open compartment, a pneumatic conveyor, and a washing compartment. The drum carried in the open compartment by the roll conveyor is opened by means of gloves, and then the opened drum is carried to the pneumatic conveyor compartment for transferring the yellow cake from the drum on a rotating plate to a yellow cake dissolution tank. After the transfer of the yellow cake from the drum, the drum is carried to the washing compartment, and is washed with injection water. The decontaminated drum is carried out from the glove box. No yellow cake decontamination remains on the floor of the drum when it is removed from the glove box.

3.2. Electrolytic reduction apparatus

To improve the apparatus a larger electrode was introduced and the spacer between the electrode and the ion exchange resin membrane was removed to permit the smooth passage of the solution and the bubbles.

3.3. Pneumatic conveyor

The previous bucket conveyor was equipped vertically for transporting uranium tetrafluoride from the drying section on the first floor to the dehydration section on the second floor; therefore, the conveyor chain, either because it was damaged or deteriorated, could cause a serious accident and contaminate the floor and the atmosphere with uranium tetrafluoride powder. The bucket conveyor was therefore removed, and the pneumatic conveyor installed instead. This conveyor is operating smoothly.

3.4. Reaction vessel

The vessel is required to resist the temperature of 90°C and be proof against the mixed acid solution. The vessel was first moulded with epoxy resin laminated vinyl and fibre carbon cloths in the inner part and fibre glass cloth in the outer part and the inside surface was lined with furan resin. It was found that the lined furan resin tended to peel under continuous operation and the peeled fractions mixed with the uranium tetrafluoride.

The newly equipped vessel was lined with ethylene tetrafluoride resin in place of furan resin. The vessel is now used successfully.

REFERENCE

- [1] HIRONO, S., FUKUSHIMA, S., "Vat leaching of the uranium ore from Ningyo Toge deposits", Economics of Ore Processing Operations, OECD, Paris (1983).

DISCUSSION

W. FLÖTER (Chairman): Is this process compatible with the standard processes for recovery of uranium?

I. YASUDA: The PNC process is in the pilot plant scale. We are now evaluating the construction and operating costs for a future commercial plant.

S. AJURIA: You said the bipolar plate is titanium on one side and platinum on the other. Are the titanium and the platinum electrically insulated from one another or is the platinum plated directly on the titanium?

I. YASUDA: Yes, it is plated directly.

CONSTRUCTION OF A NEW PLANT IN GABON BY THE COMPAGNIE DES MINES D'URANIUM DE FRANCEVILLE: THREE YEARS' EXPERIENCE OF OPERATION

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Abstract

CONSTRUCTION OF A NEW PLANT IN GABON BY THE COMPAGNIE DES MINES D'URANIUM DE FRANCEVILLE: THREE YEARS' EXPERIENCE OF OPERATION

As part of a programme to modernize and extend its means of production which had already commenced with the setting up of a solvent extraction unit in 1977, and a sulphuric acid production facility of increased capacity (60 t/d) in 1980, the Compagnie des Mines d'Uranium de Franceville (COMUF) started a new uranium ore treatment plant in 1982. The aim was to replace the older installation built in 1959-60, which had reached the limit of its capacity and whose largest equipment was in need of renewal. The new installations are capable of an annual uranium production of 1500 t of magnesium uranate. The techniques adopted were the most modern, those likely to simplify the process and improve operating costs, namely semi-autogenous grinding and solid-liquid separation using band filters. Three years of operating experience confirm the sound choice of the main options made when designing the installations. The treatment performances, especially those which are independent of the nature of the ore, and the reliability of operation are indeed excellent. Thanks to a training programme started in the late 1980s the staff adapted rapidly to the new technical environment and it has been possible to run the entire plant with almost exclusively Gabonese staff.

1. BRIEF HISTORY

The discovery in 1956 of a large uranium deposit in Mounana demonstrated the potential of Haut-Ogooué and led to the creation in 1958 of the Compagnie des Mines d'Uranium de Franceville (COMUF). Production commenced in 1961 when the first plant was set up with an initial capacity of 500 t/a of metal uranium in the form of a pre-concentrate with an original grade of 32%. Average production remained at about this level until 1972 and then increased by stages from 1973, to reach a capacity of 965 t in 1976 with pre-concentrates graded as high as 50%.

This increase had been made possible by small modifications to the installations and operating conditions. In 1977 the commissioning of a solvent extraction shop enabled a further step in production and a definite improvement in the quality of the market product, which was then delivered in the form of a 75% grade magnesium uranate. Thus, in 1979 the COMUF reached its record production of 1100 t.

In the same year it started work on a new shop capable of producing 60 t/d of sulphuric acid, intended to cope both with the previous increases and a projected

extension to 1500 t/a of metal uranium. As the original shop was stretched to the extreme limit of its capacity with installations which began to show signs of wear and tear, the decision was made in 1978 to build a new plant with a greatly increased capacity, intended to bring treatment potential to 550 000 dry t/a ore. The sulphuric acid shop started operations in January 1980 and the new plant in April 1982.

2. LOCATION

The new plant is located virtually at the geographical centre of Mounana's industrial estate. It is linked by pipe racks to the existing installations, i.e. essentially the extraction shop and acid production shop. Its raw water supply comes from a fast flowing river (the Lekedi) a few kilometres away, upon which a pumping installation with a capacity of 600 m³/h was created as part of the same project, thus enabling a supply under better conditions for the entire estate.

3. CONSTRUCTION

Work commenced in April 1980 and continued through to March 1982. The project included the following installations:

- (a) A complete plant for mechanical ore preparation;
- (b) A plant for acid leaching, solid-liquid separation and washing of the ore over band filters;
- (c) Storage ponds for pregnant liquor coming from the filtration;
- (d) A precipitation, filtration and drying shop enabling magnesium uranate to be obtained from the loaded brine coming from the solvent extraction shop;
- (e) The leached ore neutralization and pumping installation;
- (f) Modification of the leaching and countercurrent decantation (CCD) of the old plant for adaptation to effluent treatment;
- (g) A reagent preparation shop (flocculants, magnesium oxide, lime for neutralization of the leached ore, chlorate);
- (h) A pipe rack between the new plant and the existing installations, including the solvent extraction shop and sulphuric acid production shop;
- (i) A raw water pumping station in a river 6 km away together with suitable storage and distribution network to enable supply to all the COMUF installations.

The capacity of these installations is 80 t/h of dry ore and 250 kg/h of metal uranium for uranate precipitation and preparation in accordance with the size chosen for the solvent extraction shop built in 1977.

The French engineering companies were entrusted with the realization of the project, one to be in charge of mechanical preparation, the infrastructure works, and the other with the chemical side of the treatment. The COMUF managed the project and ensured co-ordination between the two engineering companies.

4. CONSTRUCTION COSTS

During the course of construction, the following quantities of material and equipment were used:

Concrete	6300 m ³
Structural steel	1935 t
Equipment other than structural steel	2380 t

As the works carried out had to fit into an existing industrial context, their overall cost cannot be directly compared with those of a complete new project. Indeed, certain installations indispensable to a uranium ore treatment plant (solvent extraction shop, and to a lesser degree the sulphuric acid shop) are excluded from these costs. Similarly, reutilization and modification of existing installations, and the need to build links between plants further apart than would be the case with an entirely new unit make the comparison difficult using costs pertaining to the realization as a whole.

Consequently, we give only the costs of the workshop which more easily lend themselves to comparison, together with cost parameters enabling a possible future attempt to extend to other cases.

4.1. The costs of the workshops

The amounts are expressed in constant F.Fr.¹ at 1 Jan. 1979 rate, and include only the construction costs proper, i.e. without training, startup and credit expenses:

Storage of run-of-mine ore, including the rough stone base of the storage area and its drainage network	5 545
Primary crushing	10 500
Grinding, with ore spillage recovery concrete made basin (semi-autogenous grinding; supply in 0-200 mm; product coming out at 95% minus 630 μ m)	32 244
Thickening of the neutral pulp	8 143
Storage of crushed material (in metal silos of 3 \times 600 t)	19 146
Subtotal for mechanical preparation	(75 578)
Leaching, filtration	51 303
Uranate precipitation and preparation	26 751
Preparation of reagents (flocculants, magnesium oxide; chlorate, lime for leached ore neutralization)	17 042

¹ In thousands of constant French francs (F.Fr.).

Storage ponds of pregnant liquor ($1 \times 645 \text{ m}^3 + 2 \times 2250 \text{ m}^3$) with pumping station towards solvent extraction (SX) shop located 1000 m away	5 262
Subtotal for chemical treatment excluding SX and excluding treatment of liquid effluents	(100 358)
Raw water pumping station, capacity $600 \text{ m}^3/\text{h}$ with 5.5 km pipe (300 mm Ø) and a storage tank of 3000 m^3	11 101
Leached ore separation performed on cyclones (for separating the coarser fraction used for backfilling the mine) and installation for hydraulic transportation to tailings	6 489

4.2. Costs per main types of expenses for the whole of the realization

These costs are exclusive of duties and taxes and have a relative value compared with the total cost of the realization.

— Equipment and other supplies, ex-works:	(30.1%)
Itemized equipment	14.53
Pipework, connections & fittings	2.66
Electrical equipment	3.48
Instrumentation	1.13
Structural steelwork	6.15
Miscellaneous, including spare parts	2.17
— All works and services performed on the site:	(38%)
Earth moving	3
Roadways & net works	3.5
Concrete works	10.2
Internal works in building	0.64
General assembly (structural steel, equipment, piping, roofing & coverings, painting)	17.1
Electrical	1.24
Automatic control instruments	1.14
Technical assistance by suppliers	1
— Packaging & transport	6.8
— Detailed engineering (detailed design, purchasing, budget control, construction supervision, startup assistance, detailed design of concrete foundations)	14.7
— Preliminary studies (grinding tests, laboratory tests, soil studies, basic engineering)	3

The types of expenses listed above when added together represent 92.6% of the total costs of the realization. The remainder consists of expenses which are less clearly demarcated than the above and include, in particular, the costs of the owner (duties and taxes, training, insurance, costs of construction site logistics, various services and small works supplied by COMUF, and startup cost).

5. OPERATION

5.1. Process control parameters

The treatment uses a conventional acid leach process, at moderate temperatures, of a pulp obtained by semi-autogenous grinding, preceded only by primary crushing to minus 200 mm. Cut-off after grinding is carried out using high frequency screens. Solid-liquid separation and washing are carried out over 3 band filters of 70 m² each. The pregnant liquor goes to solvent extraction followed by sodium chloride stripping, the loaded brine being next subjected to magnesium precipitation, followed by thickening, filtration and drying of the uranate obtained. This latter is packaged in 200 L drums.

The main operating parameters are as follows:

Grinding	95% < 630 μ m
	80% < 350 μ m
	25% < 43 μ m
Relative density of pulp before leaching	1.6
Leaching conditions	
Retention time	7 h
Redox potential	550 mV
Free acidity	H ₂ SO ₄ 12 to 15 g/L
Temperature	55 to 60°C
Production liquor	2 m ³ /t, free acid 5 g/L

Oxidation is carried out using manganese dioxide available locally (the COMILOG deposit), the sodium chlorate being used solely to adjust the redox potential to the given setting. During leaching, the pulp is heated by steam produced by the acid production shop.

5.2. Performance of the installation

Recoveries (year to date values over the first nine months of 1985): Emphasis will not be put on the leach recovery (96.3%) which is greatly dependent on the ore being treated. However, recoveries on the washing, solvent extraction and uranate

preparation units are representative of the quality of the equipment and operation. Cumulated losses until end of September in these two sections are respectively 0.3% and 0.2% of the uranium entering the installation, thus leading to excellent recoveries for these operations (washing 99.7%; solvent extraction uranate preparation 99.8%).

Consumables: It is not necessary to emphasize on reagents such as the acid consumption, levels of which are practically fixed by the nature of the ore or other conditions. We will emphasize those which may be considered as being representative of operational quality or equipment behaviour:

Kerosene	0.270 L/kg U (Jan.-Sep. 1985)
Sodium chloride	1.44 kg/kg U (Jan.-Sep. 1985)
Grinding	
Grinding balls	0.82 kg/t dry ore (Jan.-Sep. 1985)
Rubber lining	16 to 17 g/t dry ore (Jan.-Sep. 1985)

The excellent results achieved for sodium chloride consumption should be particularly mentioned. Taking into account the transportation, this reagent is one of the highest in cost. Thanks to action taken both on a technical level and in staff training, it has been possible to reduce the consumption by 37% (2.3 kg/kg uranium in the previous situation), which represents an annual saving of approximately F.Fr. 1 400 000. Regarding semi-autogenous grinding, which alone ensures the complete comminution from 200 mm to the final stage, we note that the consumption of grinding material is moderate. If we add the advantages due to its smooth operation, lack of sensitivity to disturbances occurring within conventional installations due to extreme ores, its suitability for automation such as that installed at this plant, and its low maintenance cost, the choice of this technology may be considered to have been a particularly good one.

5.3. Mechanical reliability

All the installations demonstrate a very satisfactory mechanical reliability. Shut-down time of the production line for electrical or mechanical failures represents only 4% of the foreseen operating hours, the greatest part of maintenance work being carried out as part of the programmed servicing. Alongside this result, as regards equipment availability, the company has paid great attention to reducing significantly the cost of consumable parts and equipment, either by buying elements at cheaper prices, or by adopting solutions enabling a considerably longer lifespan. Proof of this progress has been visible through the continued reduction of the plant's maintenance costs, a reduction which was, for example, 2.5% in current Gabonese francs between 1984 and 1985, an achievement made in spite of a slight increase of the tonnage of treated ore. The achieved reduction in the maintenance costs per tonne of treated ore will, in fact, be 5.5% between these two years, the cost being F.Fr. 50 per tonne at the present date. This continuous progression is undoubtedly due to the motivation of the staff, itself a result of sound informational tactics used by the management.

6. TRAINING

6.1. The problem

The problem consisted of changing from an old plant approximately 20 years old, to a new one keeping the same basic process. However, big technological modifications had to be taken into consideration:

- (a) Changing from a complex fragmentation system to a shortened line comprising primary crushing and semi-autogenous grinding
- (b) Installation of wide band filters replacing a CCD
- (c) Changing from virtually no automation to a highly automated control system.

6.2. The work-force to be trained

Basically, the work-force from the old plant required training, together with a complement of new recruits for startup of the new plant. The first category consisted of older staff with several years' service behind them, and who had attained a good degree of efficiency in their previous jobs, their experience frequently compensating for a lack of initial qualifications. The new recruits were younger and had received a good level of basic school instruction. The two groups combined and complemented each other so that a very satisfactory situation of complementary qualifications could be realized.

6.3. Methodology

An important aspect of the training process consisted in establishing staff ability levels prior to the first training stage. This enabled workers to be categorized according to their aptitudes and the jobs to be filled, as well as deciding at what level training should start. The content of the training course was then defined according to the basic extra knowledge required and the requirements of the technologies used in the new installations. In addition to this approach, an inventory was made of the tasks to be performed at each workplace, in normal operation as well as in startup or shutdown procedures, so as to give the specific content to each programme. To enable easier assimilation of information, a visual approach was largely used (i.e. recognition of circuits, identification of equipment and instrument line finding exercises) together with practising the future tasks.

6.4. Organization of training

Training took place in two stages:

- (a) Before and during startup of the new plant
- (b) Nine months after startup.

6.4.1. Before and during startup

Two teams, one in Gabon and the other in France, keeping in constant touch, devoted themselves to this project. The team basically based in France, from the IMETAL Group's training department, carried out the study of the tasks in each workplace, testing of staff levels prior to training, writing of the teachers' booklets and mechanical guides (in liaison with the engineering and manufacturing firms), together with the design of teaching means such as small scale representation of complex equipment.

A basic programme was drawn up for each shop within the plant (approximately 20 shops), using the following scheme:

- Process
- External links to the plant
- Internal links within the plant
- Signals, command controls and settings
- Instruments
- Operating procedures
- Material balance.

The COMUF team based in Gabon adapted the teachers' handbooks, carried out basic training and training of shift foremen and operators. Future workshop foremen and production engineers, after training in their own tasks, largely participated in training their own staff.

6.4.2. Further training of operating staff nine months after startup

Six months after startup of the plant, an assessment was made of the positive skills acquired and those which were still lacking. This enabled three main routes of action for further training to be decided upon, namely:

- (a) To increase the detailed knowledge of the various process functions;
- (b) To develop specific training in the control room;
- (c) To develop the shift foremen's initiative.

The IMETAL Group training department sent to COMUF an experienced teacher in this field on two three-months missions. The teacher worked essentially on-site with the shift workers, with follow-up lessons in the classroom.

6.5. Budget

Expressed in 1981 currency, the cost of the specific training for the new plant in addition to the regular COMUF training programme was F.Fr. 2 450 000, i.e. 0.8% of the investment budget.

6.6. Results

The previously mentioned assessment carried out six months after startup drew the following conclusions:

- (a) Basic training was virtually acquired (knowledge of equipment, circuits and process);
- (b) Reactions to incidents were good;
- (c) The plant was producing at its rated capacity.

After the two further training campaigns, the ability of operators and shift foremen to anticipate situations had been improved thanks to a better understanding both of detail and overall working of the plant.

6.7. Action presently under way

Knowledge of the individual workplaces and different functions of the plant is now considered to have been acquired. Present action consists of further specific work in various areas within precise optimization targets.

Thus, the spectacular reduction in salt consumption which was mentioned earlier was achieved by adjusting operating parameters and by a corresponding sensitization and training of the operating staff. A similar approach has also enabled progress with the quality of uranate. It is foreseen that action of this nature will be taken each time new equipment (for example instruments for analytical control of production) is introduced, or a major modification made. COMUF considers that the success of such operations depends mainly on the effective training of the staff involved.

On a more general level, training aimed to improve managerial skills of the foremen is implemented by the Association Nationale de Formation Professionnelle et de Perfectionnement. In addition to this, special efforts are presently being made to keep maintenance technicians up to date with the rapid evolution of the techniques.

7. WORK-FORCE — ORGANIZATION CHART

The work-force described includes operating and maintenance staff. The numbers take account of the entire Treatment Department which, of course, also includes solvent extraction and sulphuric acid production shops brought into operation prior to the new plant.

The treatment shops work continuously for six days per week, with a break on Sundays. Monday morning is spent on routine maintenance. This enables working with only three crews in continuous service in the treatment installations. By its very nature, the sulphuric acid workshop is in continuous operation, and therefore requires four crews to run it.

TABLE I. DISTRIBUTION OF WORKERS IN THE PLANT

	Higher management	Foremen- technicians	Operating staff	Total
1. Department management	1		2	3
2. Operation				
2.1. New plant ^a				
2.1.1. Shift personnel (3 crew)		$3 \times 2 = 6$	$3 \times 11 = 33$	$3 \times 13 = 39$
2.1.2. Others	2		15	20
2.2. Solvent extraction				
2.2.1. Shift personnel (3 crew)			$3 \times 3 = 9$	$3 \times 3 = 9$
2.2.2. Others		1	3	4
2.3. Sulphuric acid				
2.3.1. Shift personnel (4 crew)			$4 \times 3 = 12$	$4 \times 3 = 12$
2.3.2. Others		2	1	3
3. Laboratory	2	6	13	21
4. Maintenance	1	15	45	61
5. TOTAL	6	33	133	172
Nationals	3	20	133	156
Expatriates	3	13	—	16

^a Ore preparation, leaching, washing, yellow cake precipitation and preparation.

Table I shows the distribution of workers per section and per category, making a distinction between shift workers and the others.

Between 1983 and today the work-force has been decreased by 12 persons, which basically corresponds to a reduction in number of operating staff made possible by better organization. At the same time an effort has been made to use more Gabonese staff at management level. The expatriate management has, indeed, been reduced by four, of whom two were higher management staff. Technical performance nevertheless steadily improved since the plant started, in spite of the decrease in numbers of staff, thus confirming the technical potential and quality of the staff in charge of the installations.

8. FUTURE PROSPECTS

The forward aims are, in the short term, those mentioned earlier such as:

- (a) Reduction in production costs, by acting specifically on consumables (reagents and spares) and maintenance costs;
- (b) Improvement in qualifications as a means of contributing towards the target mentioned above.

In the longer range, COMUF starts investigating cheap treatment methods of low grade ores; these methods may be both an adaptation of the existing fast pulp leaching of products of this type, and the use of slow leaching techniques suited to the local conditions.

9. CONCLUSION

Results of three years' operation of the new plant confirm the suitability of the main options chosen during construction. The staff adapted to the new conditions very well, thanks especially to the attention which has been paid to training problems. Rated capacity was achieved rapidly and the plant now operates under excellent conditions, both as regards technical performance and operating costs.

DISCUSSION

A.T. TRIGGS: Is it possible to estimate the difference in capital costs and operating costs between a plant in a developed country and one in a developing country?

V. JUG: It should be possible. I cannot make the comparison because I have only worked in overseas projects, but if we compare the information from COMUF with that of SIMO we should be able to calculate the difference.

A.T. TRIGGS: This is a very interesting question, because in developed countries like Canada you have access to many facilities that help in the implementation of a project and that allow savings in time and cost. These facilities may not exist in a developing country and costs are likely to be much higher. The question is, how much higher?

V. JUG: This is true. Construction costs in a developing country can be very high. For instance, the cost of concrete in Gabon is about F.Fr. 4000 per cubic metre, which is very high. Therefore, a good design of concrete foundations is very important because every cubic metre of concrete saved represents a substantial amount of money.

W. FLÖTER: I may add that the same is true in remote areas of developed countries. For instance, maintenance costs in Baker Lake are higher than in Saskatoon, because Saskatoon is 1000 km closer to a workshop. The alternative is, of course, to build a new workshop to provide services in the remote area.

B. SCHNEIDER: What is your specific consumption of sulphuric acid at the plant?

V. JUG: At the beginning of our operations in the 1960s the acid consumption was about 25 kg/t. As we mine the deeper parts of the ore body consumption has increased to about 36 kg/t. This includes leaching, washing and solvent extraction.

B. SCHNEIDER: What is your source of sulphur?

V. JUG: We import the sulphur from Elf Aquitaine in the south of France.

K.H. TREUE: What are your reserves? Do you have reserves for, say, ten or twenty years?

V. JUG: It is mainly a question of the cut-off grade. With the cut-off grade we are currently using, our reasonably assured reserves are about 15 000 t uranium. But if process improvements allow us to reduce the cut-off grade the reserves will of course increase.

W. FLÖTER: Your consumption figures in the grinding section are very interesting. You mentioned you are consuming 16 to 17 g of rubber per tonne of ore. What is the lifetime of the rubber lining in your mill?

V. JUG: The lining does not wear out uniformly. Some parts, like the lifter bars, wear out very quickly and other parts, like the diaphragm at the discharge, have a good lifetime. We replace some parts about twice a year.

C.K. KIPPENBERGER: What is the grade of the ore?

V. JUG: At present the average grade of the feed is 0.3%.

A NEW UNIT FOR PURIFICATION OF URANIUM SOLUTION IN THE LODEVE MILL

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Abstract

A NEW UNIT FOR PURIFICATION OF URANIUM SOLUTION IN THE LODEVE MILL.

The uranium concentrate being processed at the Lodève plant (Massif Central, France) did not meet specifications, particularly in regard to the zirconium content. In addition, one process step was difficult to operate. Development work and a feasibility study were carried out by the Research and Development Service (SEPA), General Nuclear Materials Company (Cogéma). The feasibility study showed that a pure product could be obtained with a suitably modified process. The capital costs of this modification were estimated at F. Fr.12 million (US \$1.5 million). The modified process is now in operation and the economic results are better than expected. Although the capital costs were slightly higher than estimated, reagent savings were also higher. The modified process has proven to be profitable. The pay-back time is less than four years and the internal rate of return is 25%.

1. REASONS FOR CHANGING THE PROCESS

In the course of the pilot tests carried out to determine the process to be applied in the Lodève mill no significant amounts of zirconium were found in the uranium solutions. Therefore, the purpose of the tail end process was to remove molybdenum and organics.

Following the first months of operation, the concentrates failed to meet specifications mainly because of the zirconium content (1.05% with regard to uranium) which resulted in a penalty.

Consequently, General Nuclear Materials Company (Cogéma) and Western Ore Industrial Company (SIMO) reconsidered this part of the process (called Unit 30). The conclusions were the following:

- (a) It was not possible to add a simple step to remove zirconium without modifying the whole process;
- (b) Since the process had to be modified, it was considered useful to improve molybdenum removal which up until then had not been very efficient, and also to eliminate the preconcentrate roasting (to destroy organics) as operating and maintenance costs of the roasting were high;
- (c) A process had to be designed which would utilize, as far as possible, existing equipment and which would not exceed actual operating costs.

The objectives of the project are detailed in Section 2. Section 3 describes how these objectives were achieved and an estimation of capital and operating costs is discussed in Section 5.

2. REVIEW OF THE PREVIOUS PROCESS

Lodève uranium ores are complex because they contain many mineral species and especially alkaline and alkaline earth carbonates. Laboratory testing of the ore showed that acid leaching resulted in excessive acid consumption and difficult solid-liquid separation. Therefore alkaline leaching had to be used.

The flow sheet design is different from conventional alkaline processes. It includes a two-stage leaching to achieve a high uranium recovery efficiency. The refractory character of the ore is mainly due to the presence of:

- (a) Clay materials which make filtration difficult;
- (b) Sulphides which contain uranium as an inclusion;
- (c) Molybdenum which has to be removed;
- (d) Soluble zirconium which dissolves with the uranium concentrate and is completely recovered in the yellow cake (magnesium uranate);
- (e) Large amounts of organics, mainly as humic acids, which make it necessary to treat the liquid waste.

The previous process is described in Ref. [1]. Processing of the uranium pregnant solution included:

- (a) Uranium precipitation as sodium uranate using lime;
- (b) Roasting of the precipitate to burn organics;
- (c) Sulphuric redissolution of the roasted precipitate in sulphuric acid;
- (d) Uranium precipitation with magnesia.

It is this part of the process which has been changed.

3. DESCRIPTION OF THE NEW PROCESS

The new process includes the following main stages:

- (a) Addition of sulphuric acid and destruction of the carbonate.
- (b) Precipitation of uranium with sodium hydroxide thereby achieving a good separation of organics and molybdenum which stay in the solution (liquid waste). Only the zirconium is precipitated and remains with the uranium.
- (c) The thickened precipitate is mixed with sulphuric acid which dissolves uranium at pH 3. Zirconium is hydrolysed and can be separated on a precoat filter. The zirconium compounds are repulped with sulphuric acid to recover the small amounts of uranium which are carried away. The solution is then recycled to the head of the process. Zirconium precipitate is filtered again and sent to the solid waste pond.
- (d) Uranium is precipitated by hydrogen peroxide at pH 3.5. This precipitation is very specific and permits the removal of the last traces of molybdenum.
- (e) After thickening and filtration on a belt filter, the uranium peroxide is dried and drummed.

4. COMPARISON OF THE TWO PROCESSES

4.1. *Simplicity of the new process*

Comparison of the two processes shows that the new process is simpler than the old one (see Fig. 1). Several steps have been eliminated such as filtering and roasting of the preconcentrate, clarification of the liquor before precipitation and centrifugation of the yellow cake.

4.2. *Quality of the concentrate*

The chemical analysis of the old and the new concentrate is compared in Table I. The results shown were obtained during the year of 1983 (old concentrate) and the first nine months of 1985 (new concentrate).

The following observations were made:

- (a) The uranium content increased from 67.9% to 69.7%.
- (b) The arsenic content decreased from 0.17% to 0.03%.
- (c) The calcium content decreased drastically from 1.62% to 0.04% because lime is no longer used for the first uranium precipitation.
- (d) The chloride content decreased from 0.025% to 0.004%.
- (e) The magnesium content also decreased drastically from 1.74% to 0.01% because magnesia was no longer used for the second uranium precipitation.
- (f) The sodium content decreased from 1.48% to 0.35%.

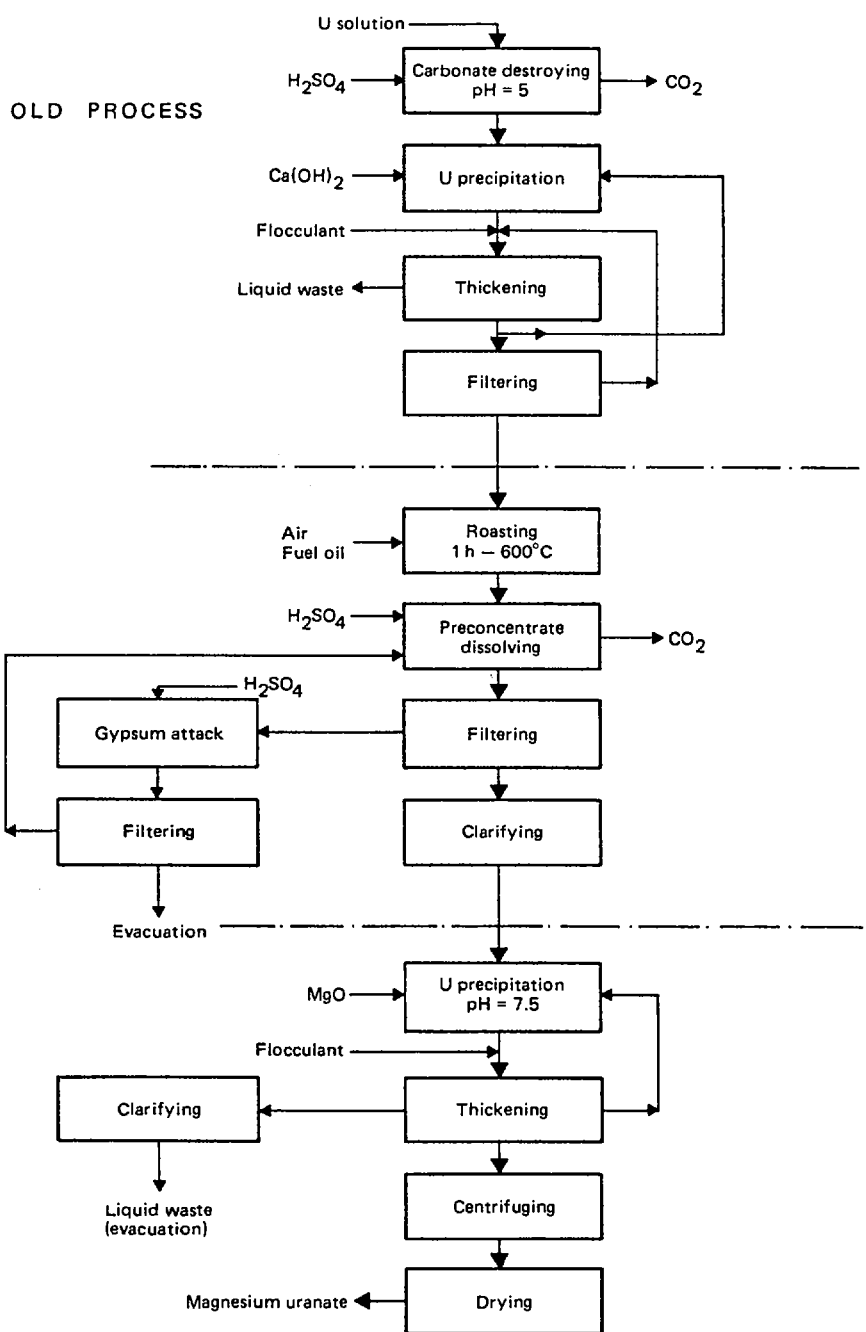
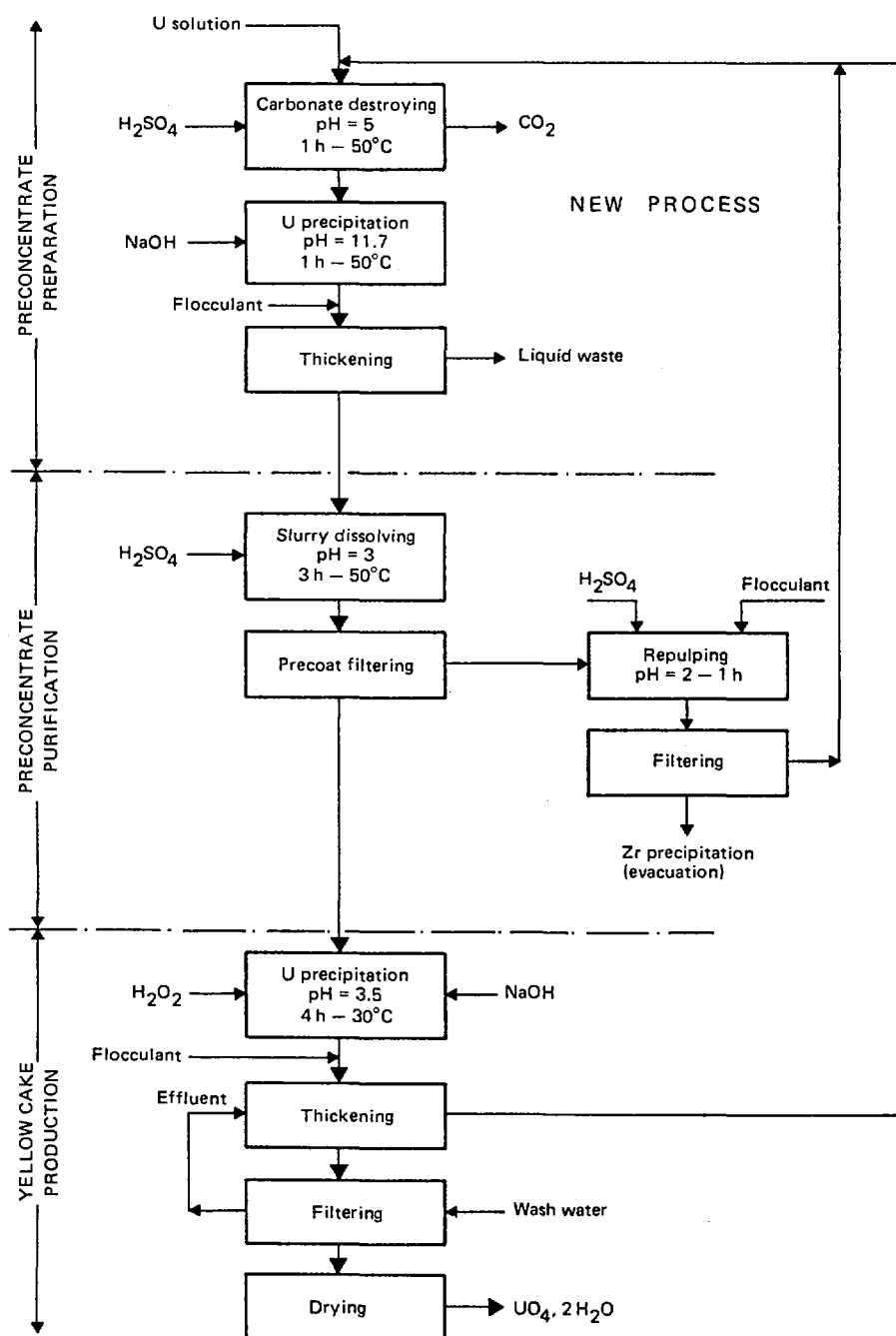


FIG. 1. Comparison between the old



and the new processes — block diagrams.

TABLE I. IMPURITIES CONTENT OF YELLOW CAKE PRODUCED IN LODEVE BEFORE (1983) AND AFTER (1985) STARTUP OF URANIUM PEROXIDE PRODUCTION
(in % with regard to uranium)

	U	H ₂ O	As	Ca	Cl ⁻	CO ₃ ²⁻	F	Mg	Mo	Na	PO ₄ ³⁻	SiO ₂	SO ₄ ²⁻	V	Zr
<i>Old concentrate</i>															
(1983)															
Weighted average	67.87	5.51	0.17	1.62	0.025	0.28	0.02	1.74	0.09	1.48	0.15	3.50	3.81	0.07 ^a	1.05
Minimum			0.007	1.15	0.009	0.26	0.012	1.64	0.02	0.98	0.13	1.30	2.755	0.06 ^a	0.505
Maximum			0.33	1.95	0.042	0.29	0.03	1.84	0.23	1.815	0.17	5.825	4.90	0.09 ^a	1.575
<i>New concentrate</i>															
(1985)															
Weighted average	69.71	1.26	0.03	0.04	0.004	0.28	0.02	<0.01	<0.015	0.35	<0.05	<0.15	0.94	<0.02	0.06
Minimum			0.02	0.02	0.002	0.22	0.01			0.30			0.33	0.01	0.03
Maximum			0.04	0.07	0.007	0.37	0.04			0.44			1.21	0.03	0.12

^a Expressed in V₂O₅.

- (g) The silica content decreased from 3.5% to 0.15%.
- (h) The sulphate content decreased from 3.81% to 0.94%.
- (i) The zirconium content decreased substantially and consequently no penalty had to be paid. The magnesium concentrate had an average zirconium content of 1.05% with extreme values of 0.05% and 1.5%. The peroxide concentrate had an average zirconium content of 0.06%, with a range of 0.03% to 0.12%.

As a general statement, the yellow cake is remarkably pure and is not subject to any penalty. (See Table I.)

4.3. Reduction of the liquid wastes

In the previous process, the final product was magnesium uranate. The barren liquor, containing mainly sodium sulphate (52.7 kg/h) and magnesium sulphate (217.3 kg/h) was discharged after appropriate treatment at the rate of 11.5 m³/h. It was the only liquid waste evacuated. The other liquid waste (barren liquor from preconcentrate precipitation) containing mainly sodium sulphate was concentrated and purified. The sodium sulphate was finally crystallized and sold.

The new process yields the same volume of liquid waste. However, the liquid waste from uranium peroxide precipitation is not evacuated since it contains only sodium sulphate. It is recycled to the main process at the point before the addition of sulphuric acid (see Section 3(a)). No liquid waste is now evacuated, and also sodium sulphate is recovered as a by-product.

5. ESTIMATE OF CAPITAL AND OPERATING COSTS¹

5.1. Capital costs

The new process is simpler and uses less equipment than the previous one. A few new pieces of equipment were required and some old pieces had to be modified or relocated. The three main new pieces were:

- (a) A precoat filter and associated devices to remove zirconium precipitates;
- (b) Equipment for storage of hydrogen peroxide;
- (c) Equipment for storage of sodium hydroxide

The main modifications were:

- (a) Relocation of a belt filter for filtration of yellow cake;
- (b) Remodelling of the control room.

¹ Estimates as of September 1984.

Equipment costs were therefore low although the costs of engineering, installation and piping were higher than normal.

With a contingency rate of 20%, the overall capital cost amounted to F. Fr.12 250 000 (i.e. about US \$1 530 000).

5.2. Operating costs

The operating costs of the old process and the new process are compared in Table II. For the purpose of this comparison the penalty paid for excess molybdenum and zirconium was included in the cost of the old process. As shown in Table II the new process makes a saving of F. Fr.4.38 (about US \$0.55) per kg uranium.

5.3. Profitability of the project

A study was carried out on the following basis:

(a) Expenditure

The capital costs were F. Fr.12 250 000 (about US \$1 531 250) i.e. in 1983 F. Fr.610 000 (about US \$76 250) and in 1984 F. Fr.11 640 000 (about US \$1 455 000).

Manpower costs, for temporary shutdown of one month, were F. Fr.4 280 000 (about US \$535 000).

(b) Savings

Savings of F. Fr.840 000 (US \$105 000) in operating costs during 1984 (startup assumed to be September 1984).

Savings of F. Fr.4 200 000 per year for 1985 and subsequent years (assuming a production rate of 960 t U per year).

Savings of F. Fr.270 000 (US \$34 000) for manpower (2 operators less than previously).

Gain of F. Fr.110 000 per year (US \$14 000) on maintenance costs.

Under the above conditions the profitability of the project can be expressed in terms of a gross pay-back time of 3.4 years and an adjusted pay-back time of 3.9 years, therefore an internal profit rate of 25%.

6. IMPLEMENTATION OF THE PROJECT

After completion of the feasibility study and approval of the project two teams from the Research and Development Service (SEPA) of Cogéma and SIMO managed the project, with the assistance of a French engineering company, Krebs S.A.

TABLE II. COMPARISON OF ESTIMATED OPERATING COSTS

Reagents and utilities	Previous process		New process	
	Place of use	(F. Fr./kg U)	Place of use	(F. Fr./kg U)
Sulphuric acid (100%)	Acidification of liquors Dissolving of preconcentrate Effluent neutralization	6.23	Acidification of liquors Dissolving of preconcentrate Effluent neutralization Washing zirconium precipitate	4.87
Sodium carbonate	Softening (before evaporation)	1.59	—	—
Lime	Preconcentrate precipitation	0.82	—	—
Sodium hydroxide	—	—	Preconcentrate precipitation Peroxide precipitation	1.42
Hydrogen peroxide	—	—	Peroxide precipitation	1.72
Clarcel	—	—	Zirconium filtering	0.28
Flocculants	Preconcentrate precipitation Magnesium uranate precipitation	0.13	Preconcentrate precipitation Peroxide precipitation	0.04
Magnesia	Magnesium uranate	1.22	—	—

TABLE II. (cont.)

Reagents and utilities	Previous process		New process	
	Place of use	(F. Fr./kg U)	Place of use	(F. Fr./kg U)
Propane	Preconcentrate roasting	2.83	Yellow cake drying	0.52
Electric power	Yellow cake drying	2.28	Liquid waste evaporation	2.26
Activated carbon	Liquid waste purification	8.85	Liquid waste purification	9.73
Penalty	For Mo and Zr	1.27	—	—
	Total 1	25.22	Total 2	20.84
	Total 1 - Total 2 = F. Fr. 4.38/kg U			

TABLE III. ACTUAL CAPITAL COST

Items	Cost (10 ³ F. Fr.)	Proportion with regard to	
		Items 1 + 2 (%)	Total 1 (%)
1. Equipment	2 250	64.1	24.0
2. Modification and installation	1 260	35.9	13.4
3. Piping and valves	2 600	74.0	27.6
4. Electric equipment	550	15.7	5.8
5. Control equipment	1 220	34.75	13.0
6. Civil engineering	1 030	29.35	11.0
7. Miscellaneous	320	9.1	3.4
8. Spare parts	170	4.85	1.8
Total 1	9 400		
9. Engineering	3 200		30.0
Total 2	12 600		

The process document was finished in July 1983 and the order to Krebs was sent in September 1983. To complete the installation, the plant was shut down for three months (June to August 1983). The unit startup, like the whole plant, took place in September 1984, with the help of the SEPA, and without any significant difficulty. Since October 1984, a gain on the operating costs has been made.

7. RESULTS²

7.1. Capital costs

Table III shows the actual capital costs incurred. The overall costs were only 2.85% above the estimate, i.e. F. Fr.12 600 000, estimated cost F. Fr.12 250 000 (US \$1 575 000, estimated cost US \$1 530 000).

The breakdown of the difference is as follows:

- Equipment costs; additional pumps and replacing a heat exchanger;
- Installation; relocation of the first precipitate tank and of the belt filter for peroxide;

² Economic conditions as of September 1984.

TABLE IV. COMPARISON OF ACTUAL OPERATING COSTS BETWEEN THE PREVIOUS AND THE NEW PROCESS

Reagents and utilities	Previous process (F. Fr./kg U)		New process (F. Fr./kg U)	
Sulphuric acid (100%)	5.39		3.37	
Sodium carbonate	1.46		—	
Lime	0.84		—	
Sodium hydroxide (100%)	—		1.21	
Hydrogen peroxide	—		2.21	
Clarcel	—		0.21	
Flocculants	0.93		0.54	
Magnesia	0.99		—	
Propane	3.08		0.67	
Electric power	2.25		2.38	
Activated carbon	9.04		8.44	
Penalty	1.21		—	
Total 1		25.19	Total 2	19.03
Total 1 — Total 2 = F. Fr.6.16/kg U				

- Civil engineering; repair of the concrete floor;
- Engineering studies were more expensive to cover the above three additional items.

The contingency estimate amply covered the increase of costs.

7.2. Operating costs

Operation of the new unit has been very satisfactory since the beginning. Savings on the operating costs were made at the beginning of the second month.

The actual operating costs of the old and new processes are compared in Table IV. Actual savings of F. Fr.6.16 (US \$0.77) per kg uranium were more than predicted (F. Fr.4.38, US \$0.55) because of lower consumption of the reagents and electricity and also lower unit costs. Assuming a yearly production of 900 t U the actual savings in operating costs were F. Fr.5 545 000 (US \$693 000) per year compared with the estimate in savings of F. Fr.4 200 000 (US \$525 000).

7.3. Profitability

The profitability of the project is higher than estimated because of the greater savings in operating costs everywhere:

(a) Expenditure

- Capital costs of F. Fr. 12 600 000 of which F. Fr. 630 000 were paid in 1983.
- Manpower for one month shutdown, F. Fr. 4 280 000.

(b) Savings

- Savings in operating costs for three months in 1984 amounted to F. Fr. 1 660 000 (US \$207 500).
- Savings of F. Fr. 5 545 000 per year on operating costs.
- Savings (as estimated) F. Fr. 270 000 per year on manpower and F. Fr. 110 000 per year on maintenance costs.

Based on the above, profit on the project can be calculated in terms of 2.6 years.

- An adjusted pay-back time of 2.8 years (discount rate of 10%) resulted in an internal profit rate of 35%.

8. CONCLUSIONS

From a technical and planning point of view the project was very successful. An important contribution to the success was the involvement of the engineers in all stages of the project and especially the startup.

From the economic point of view results were better than expected. The experience gained shows that in the case of a significant modification of an existing unit the feasibility study must consider a contingency rate as high as 20 % of the overall costs.

REFERENCE

- [1] BODU, R., et al., "L'usine de Lodève: Un procédé alcalin complexe pour un minerai d'uranium complexe", Advances in Uranium Ore Processing and Recovery from Non-Conventional Resources (Proc. Tech. Comm. Mtg. Vienna, 1983), IAEA, Vienna (1985) 131-168.

DISCUSSION

F.-K. FELDMANN: What is the uranium concentration going to the solvent extraction system?

P. MICHEL: In the Lodève mill, there is no solvent extraction. The pregnant solution contains about 2.8 g/L uranium.

F.-K. FELDMANN: Why do you precipitate the zirconium? Is there a commercial reason?

P. MICHEL: We remove zirconium because it was present as an impurity in the concentrate and we were being penalized for that. Almost all the zirconium is precipitated with the uranium when using hydrogen peroxide. We redissolve the precipitate at a pH of about 3. Zirconium remains as a solid and is removed by filtering in precoat filters. Only a small part of the zirconium in the ore is dissolved, but it is enough to require a purification step.

H.J. STEINER: Which is the zirconium bearing mineral?

G. LYAUDET: Zirconium is contained in complex silicates associated with pitchblende.

H.J. STEINER: Why was the zirconium not detected during the initial test work? Did you not have a good sample or did you simply not pay attention to the problem?

P. MICHEL: We did not have a good sample. As I said in the paper, Lodève is a good illustration of ore variability. Lodève is in fact not a single ore body, but many small ore bodies in the same area. There are large differences between one place and another, sometimes within a few metres. During the operation of the pilot plant we checked the levels of zirconium in many of the process stages and the levels were low. But the ore sample we used was not the same as the ore we used during the first and second years of industrial operation. This was not the fault of the pilot plant people.

Y. VOLKMAN: What is the concentration of sodium carbonate in the solution that is fed to the process?

P. MICHEL: It is about 16 g/L.

Y. VOLKMAN: Do you recover the carbon dioxide which is evolved upon destruction of the carbonate?

P. MICHEL: No. We do not.

Invited Paper

DEVELOPMENT STATUS OF PROJECTS FOR URANIUM PRODUCTION

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Abstract

DEVELOPMENT STATUS OF PROJECTS FOR URANIUM PRODUCTION.

The current status of the uranium industry and of the technology for uranium mining and ore processing is reviewed. A description is made of the test work and engineering development required for a project for the production of uranium concentrates. The range of capital and operating costs of a uranium mill and the approximate time schedule for the implementation of a project are indicated.

1. SUPPLY

The present supply of uranium concentrate in the world outside centrally planned economies area (WOCA) countries of about 50 000 t U_3O_8 /a is being produced in approximately 55 conventional uranium plants plus six plants for extraction of uranium from phosphoric acid. The yearly capacity ranges from about 250 000 lb U_3O_8 from in situ leaching (ISL) plants for low grade ore to about 12 million lb U_3O_8 in the Key Lake plant, Canada, for processing high grade uranium-nickel-arsenic ore [1, 2].¹ The world uranium production originates from the countries shown in Table I.

2. ORE TYPES

While 10 years ago sandstone ore with 0.2 to 0.4% U_3O_8 was the major ore type being processed, at present unconformity controlled ore bodies with 2.0 to 12% U_3O_8 and large reserves are gaining importance. The ore types shown in Table II have reasonably assured reserves of about 2 million tonnes of uranium.

It is obvious that at present quartz pebble conglomerates and unconformity related ore deposits are contributing to production in excess of their share in reserves, while the contribution of sandstone ores is diminishing. This is also reflected by the

¹ 1 lb = 0.4536 kg.

TABLE I. WORLD URANIUM PRODUCTION [3, 4]

Country	1983		1985 (projected)	
	(t U)	(%)	(t U)	(%)
Australia	3 211	8.8	4 200	10.9
Canada	7 143	19.5	11 900	31.0
Europe ^a	3 529	9.6	3 400	8.9
Gabon	980	2.7	900	2.3
Namibia	3 719	10.1	3 600	9.4
Niger	3 420	9.3	3 400	8.9
South Africa	6 063	16.5	5 100	13.3
United States	8 136	22.2	5 400	14.1
Others	486	1.3	500	1.3
Total	36 684	100.0	38 400	100.0

^a France: 3174 t uranium in 1983 (actual); 3200 t uranium in 1985 (projected).

declining production share in the United States of America (Table III) where mining in sandstone ore is predominant [4].

3. TIME SCHEDULE FOR THE DEVELOPMENT

The development of a project from grassroot prospection to the production of yellow cake may require 6 to 15 years, depending on the size of the deposit and on the yearly production envisaged (see Table IV).

It is notable that the time to geologically delineate the ore body and to establish the engineered and environmentally accepted mining and milling process has almost doubled in the last ten years. This is related to the complex geology of the ore bodies which are explored, and to the more detailed, and voluminous, environmental impact studies which have to be discussed with all organizations concerned. The cost for the development has more than tripled within the same time.

TABLE II. ORE TYPES (IN PER CENT)

	Reserves (130 US \$/kg) ^a	Production (1985) ^b
Quartz pebble conglomerates	15.8	24
Unconformity related	17.8	25
Disseminated magmatic uranium in igneous and metamorphic rocks	12.4	11
Vein	6.0	9
Sandstone	36.7	28
Phosphate uranium as by-product		3
Calcrete	4.6	—
Other (shale, bituminous)	6.7	—
	100.0	100

^a After Maxwell and Simonsen [5], reasonably assured resources (RAR).

^b Projected.

4. MINERALOGY

Although 120 uranium minerals are known, only about 12 are being mined and processed to produce a saleable product. (See Table V.) Uranium is presently mined when its grade is above 0.03% U_3O_8 and the reserves are large enough to allow commercially beneficial mining, like Rössing, Namibia. There is a tendency, however, that currently more complex uranium ores are being mined which are as high as 2.5% in uranium grade, such as Key Lake. For Cigar Lake, Canada, a uranium grade of 10% is indicated with a commencement of production being possible in about 1992.

The uranium minerals are 10 μm to about 2 mm in size and are intergrown with a quartzitic, silicious or carbonatic matrix. The mineralogy, the intergrowth of minerals, the matrix, the hardness and the physical properties must be studied in detail to develop the milling process for the uranium ore [6].

5. ORE MINING

In the WOCA countries about 4% of the uranium is being produced by in situ leaching whereby no solid uranium ore is to be excavated. The uranium extraction is performed by an injection of a leaching solution into the ore body via a well and

TABLE III. URANIUM MINING IN THE USA

Year	Production (t U)	Share of world production (%)
1981	14 793	33.5
1982	10 330	25.0
1983	8 136	22.2
1984	5 924	15.2
1985	5 400	13.3
(projected)		

TABLE IV. APPROXIMATE TIME AND NUMBER OF PERSONS REQUIRED FOR VARIOUS STAGES OF DEVELOPMENT

	Approx. time (a)	No. of persons
Prospection	1-2	3-5
Exploration (to discovery hole)	1-3	10-50
Development (to reserve estimate and prefeasibility)	1-3	10-20
Test work	0.5-1	2-10
Environmental study, engineering/piloting, feasibility	2-3	35-75
Construction	0.5-2	100-1000
Startup production	<u>0.5-1</u>	<u>35-800</u>
Total	6.5-15	195-1960

TABLE V. MINED URANIUM MINERALS

Oxides	Uraninite Pitchblende Gummite	} $(U_{1-x}^{IV} U_x^{VI})O_{2+x}$
	Becquerelite	
		$7UO_3 \cdot 11H_2O$
Mixed oxides	Brannerite	$(U, Ca, Fe, Th, Y)(Ti, Fe)_2O_6$
	Davidite	$(Fe, Co, U, Ca, Zr, Th)_6(Ti, Fe, V, Cr)_{15}(O, OH)_{36}$
	Pyrochlore	$(Na, Ca, Fe, U, Sb, Pb, Th, Zr, Ce, Y)_2-$ $(Nb, Ta, Ti, Sn, Fe)_2O_6(O, OH, F)$
Silicates	Coffinite	$U(SiO_4)_{1-x}(OH)_{4x}$
	Uranophane	$Ca(UO_2)_2(SiO_3)_2(OH)_2 \cdot 5H_2O$
Phosphates	Autunite	$Ca(UO_2)_2(PO_4)_2 \cdot 10 \text{ to } 12H_2O$
	Torbernite	$Cu(UO_2)_2(PO_4)_2 \cdot 12H_2O$
Vanadates	Carnotite	$K_2(UO_2)_2(VO_4)_2 \cdot 1 \text{ to } 3H_2O$
	Tyuyamunite	$Ca(UO_2)_2(VO_4)_2 \cdot 5 \text{ to } 8H_2O$
Carboniferous	Tucholithe	Uraninite complex with carboniferous material

by subsequently pumping the uranium loaded leach solution to surface, where the uranium will be extracted from the solution. The barren solution is thereafter reinjected into the well field.

Less than half of the uranium is being mined by underground mining and 54 % by open cast mining. According to the planning, more underground mines will be operating in future when the big ore bodies at Cigar Lake, Canada, and Roxby Downs and Jabiluka, Australia, are being mined. At that time new mining methods may also have to be developed because of the high grade ore which will be mined.

The mining methods applied now are still standard methods which have been practiced for more than 20 years [7]. For future underground mining of high grade uranium ore remotely controlled borehole mining is being discussed.

In most cases uranium ore is not uniform in grade and rock quality. As the mill is sensitive to variations in excess of 25 % a constant quality of the ore feed to the mill must be provided. This can be accomplished in general by three methods:

- (1) Selective mining and trucking of ore ex-mine to the mill according to a blending schedule;

- (2) Separate storage of ore grades in piles or bins and reclaiming or discharging the ore types rated to a blending scheme;
- (3) Dumping of the various ore grades in layers on a blending pile and reclaiming the front of the pile and thus blending.

Open cast mining is usually performed by drilling and blasting the solid ore, loading the broken ore with a shovel or front end loader and trucking the ore to the mill site. One blast is 10 000 to 20 000 t and the productivity is 100–200 t per man-shift [7]. Truck capacity varies from 25 to 150 t, size of loaders from 1 to 5 m³ and size of shovels from 0.5 to 10 m³.

An underground mine can be developed in flat terrain by vertical shafts, down winding ramps and in mountain areas by horizontal adits. The underground workings are developed with conventional methods:

- Cut and fill:* The workings remain open for vein deposits in hard rock and no fill is brought in. In the case of weak rock, filling is brought in from bottom to level.
- Sublevel caving:* Applied in steep and thick deposits and in hard rock. The openings cave in after extraction of the ore.
- Room and pillar:* Applied in flat seams in hard and medium hard rock. The hanging wall is supported by pillars of remaining ore and the workings remain open; 30 to 40% of the ore is lost in pillars.

Borehole mining may be applicable for soft ore. The mining cost per tonne of ore in underground mines is about 3 to 10 times higher than that of open cast mining. Mine equipment in underground mines is only 10 to 20% in size and capacity of the equipment used in open pit mines.

6. TECHNICAL DEVELOPMENT OF MINE AND MILL PROJECTS

6.1. Analyses

The development commences with comprehensive analytical test work on ore samples; 500 to 50 000 ore samples per project may have to be analysed on U, Th, Si, Ca, Mg, CO₃, Mo, V, Se, P, S in order to classify the ore and to determine the leaching process. The analytical method employed may be gravity, photometry, fluorimetry, X-ray fluorescence, atomic adsorption, induced coupled plasma or neutron activation. The accuracy of the analyses is in the range of 0.5 to 10%.

6.2. Test work

Test work is being performed batchwise on ore samples from drill cores, chip samples or from quartered bulk samples. Sample material may be in a total quantity

of 50–500 kg of representative ore types. For pilot testing at least a 50 t sample is necessary to run a test continuously over several days at a rate of 10 to 100 kg/h.

Batch acid or alkaline leaching tests are performed in 1–5 L vessels with a sample size of 500–2500 g ore, while pilot tests should be performed in 100 to 1000 L vessels of about 1000 kg of ore minimum.

Solid–liquid separation is carried out with hand-filters of 15–30 cm in diameter or with settling cylinders for 1000 cm³ pulp to determine the filter size or the settling area for the countercurrent decantation (CCD).

From alkaline solutions uranium is precipitated with soda ash directly. For acid leach solutions solid ion exchange or solvent extraction will be performed. The precipitated yellow cake can be handled in the form of slurry, but will be analysed after filtration and drying at 104°C.

The following results are derived from successful test work:

- Material balance of the process
- Reagent consumption
- Water, energy and heat balance of the process
- Operating cost estimate
- Layout of machinery for the mill
- Capital cost estimate for the mill

The current leaching processes are multiple modified versions of the standard acid or alkaline processes known for 20 to 30 years. The modifications are installed especially in the following areas:

<i>Acid leaching</i>	Oxidation in autoclaves by oxygen (Key Lake mill) or oxidation with Caro's acid (Narbalek mill)
<i>CCD</i>	Rapid settlement of solids with high capacity thickeners (enviroclear thickener)
<i>Solvent extraction</i>	Introduction of the rapid mixer settlers from Krebs with a conical pump as mixer (Mounana, Key Lake)
<i>Neutralization</i>	Bulk neutralization of tailings solution from leaching complex uranium ores
<i>Radium removal</i>	Introduction of the radium specific resin XFS 43230 from Dow Chemical, Midland, Michigan, USA
<i>Tailings pond structure</i>	Multilayered deposition of tailings with the subaerial disposal and with solution recycling (Knight and Piesold pond) or tailings disposal on a filter bed with solution recycling (Collins Bay pond), both applied in Canada
<i>Process control</i>	Systems for process control and management information (Key Lake)

An example for a modified system is given for the Key Lake mill, where eight additional unit processes were installed to the existing 10 standard unit operations.

The product quality of the saleable yellow cake should be in accordance with the specifications given in Table VI. The yellow cake will be packed in drums containing up to 1000 lb U_3O_8 each and will be shipped as a lot of 35 000 lb each.

6.3. Engineering

The evaluation of the test work will result in a conceptual engineering comprising mining, ore blending, milling, transportation and ancillaries such as workshops, offices, camps or town sites, and power plant and acid plant, if appropriate. The conceptual engineering will provide 15–25 drawings. Detailed engineering may provide 100–1000 drawings and 10–50 specifications for designing the construction work.

There is a wide variety of contract types reflecting owner's goals and contractor's motivation. The main grouping is:

- (a) Firm fixed price contracts with or without a completion bonus and with or without escalation;
- (b) Cost plus percentage of cost contracts with a fixed fee or incentive fee.

Regarding the condition of the construction job any contract has its advantages and disadvantages. The usual weighted value of each influencing factor is low cost 40%, high quality 20% and rapid completion 40%.

Besides these factors, there are two others which will greatly influence the quality of contract performance:

- (a) The integrity of the contractor; and
- (b) The amount of supervision and inspection given by the owner.

6.4. Capital and operating costs

The capital costs for a uranium mill may range from about US \$45 lb/a for a small ISL plant with a production of 250 000 lb/a to about US \$25 lb/a for a 12 million lb/a plant. At the same capacity, say 1 million lb/a, the capital cost for ISL is about 50% of the cost for a conventional operation. The final capital costs are determined by the logistic of supplying the project with men, material for construction, maintenance and startup, equipment and material. Comparable costs in remote locations are three to five times higher than in an industrial area.

For operating costs the following estimate may be applied:

	%
Manpower	30
Operating material and energy	30
Maintenance and radiation protection	30
Administration	10

TABLE VI. YELLOW CAKE SALES SPECIFICATIONS ESTABLISHED BY CONVERSION PLANTS^a

	British Nuclear Fuel (UK)	Comurhex (France)	Eldorado (Canada)	Allied Chemical (USA)	Kerr McGee (USA)
U	40	60	50	75	60
Moisture in HNO ₃ , not soluble	10	5 0.1	5.0 0.1	2.0	0.1 ^a
Grain size (mm)	6.68	6	6.25	6.25	6.25
Th	NA	NA	2.0		2.0
Mo	<0.6	0.45	0.15 ^a	0.1	0.15 ^a
V ₂ O ₅	<1.00	1.80	V = 0.1	0.1 ^a	0.1 ^a
Ca		1.15	1.0	0.05	1.0
P ₂ O ₅	<6	1.1	P = 0.35	PO ₄ = 0.1 ^a	P = 0.35
Cl, Br, I	<0.5	0.25	0.25 ^a	0.05	0.25
F		0.15	0.15 ^a	0.01	0.15
SO ₄		10.50	S = 3.5	S = 3.00	S = 3.5
Fe				0.15	
As	<2.00	1.0	1.0	0.05	1.00
CO ₃	<2.00	2.0	2.00	0.20	2.00
B	<0.2	0.15	0.15	0.005	0.15
Na		NA		0.50 ^a	
K		NA		0.1	
SiO ₂	<4.00				
Zr					2.00

^a Limits with penalties for exceeding values.

7. RADIATION PROTECTION AND TAILINGS MANAGEMENT

Uranium ore contains about 0.1–10% uranium oxides, predominantly as the uranium isotope ²³⁸U and only 0.711% as the isotope ²³⁵U. Uranium located in the deposit is in equilibrium with all daughter products. This condition is affected during the mining procedures by blasting, loading and hauling the broken ore and thereby exposing uranium to the atmosphere. Radon-²²²U will escape into the air.

The concentration of radon in the air is higher in underground mines than in open cast mines. Both operations have to follow the guidelines of the International Commission on Radiological Protection (ICRP) to keep the radon concentration below 1.1 Bq/L (approx. 30 pCi/L) and the individual dose below 0.05 Sv/a (5 rem/a). The radiation protection in active mines and mills is usually very effective. The listed values are mostly below 0.01 Sv/a (about 1.0 rem/a) and high grade ore is being handled with protective arrangements such as shielding and ventilation.

The waste management is, however, of great importance. Fine grained tailings are higher in radioactivity than coarse tailings. Therefore only the coarser grained sand can be used for dam construction, while the fines must be disposed of inside the tailings pond.

The supernatant water will diminish the emanation of radon from the settled tailings by a factor of 25 and the final covering of the tailings by a layer of overburden of 1–2 m in thickness will reduce the emanation at least by a factor of four. These methods will provide also a complete shielding against gamma radiation.

Water leaching of abandoned tailings will be prevented by controlling the inflowing catchment by a ring channel and by deviating this water from the tailings deposition. Measurements at a distance of 1 m from a dry abandoned tailings site resulted in a dose of less than 0.01 mSv/h (1 mrem/h) and at 50 m distance less than 1×10^{-4} mSv/h (0.01 mrem/h). The latter is equal to natural activity. Tailings may also be disposed of in old mines to minimize radiation exposure to man.

REFERENCES

- [1] FLÖTER, W., "Aufbereitung von Uranerzen und Gewinnung eines verkaufsfähigen Produktes", Gmelius Handbuch der Anorganischen-Chemie, No. 55, Chapter 1, Springer-Verlag, Berlin (West) (1981) 1–89.
- [2] YOUNG, P., GATZWEILER, R., FLÖTER, W., Die Entwicklung der Lagerstätte Key Lake/Saskatchewan, Kanada, *Erzmetall* 35 (1982) 489–510.
- [3] Uranium 1983, Atomic Industrial Forum, Inc., Bethesda, MD (1983).
- [4] Uranerzbergbau GmbH, Bonn, unpublished documentation.
- [5] MAXWELL, D.G., SIMONSEN, H.A., "Technico-economic influences on the long term availability and cost of uranium", Proc. 9th Annu. Symp. Uranium Institute, London, Sep. 1984.
- [6] INTERNATIONAL ATOMIC ENERGY AGENCY, Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores, Technical Reports Series No. 196, IAEA, Vienna (1980) 1–267.
- [7] FLÖTER, W., Uranium Mining and Milling, IAEA Course on Nuclear Power Project Planning and Implementation, School of Nuclear Technology, Karlsruhe, Sep.–Nov. 1976, 33 pp.

THE KEY LAKE PROJECT

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Abstract

THE KEY LAKE PROJECT.

The Key Lake mine and mill complex and the main phases of the development of the Key Lake Project are described. Maps, diagrams, process flow sheets and layouts of the plant and of the mine and mill complex are included.

1. INTRODUCTION

Key Lake is situated in northern Saskatchewan, Canada, 707 km north of Saskatoon, the industrial centre of Saskatchewan. Key Lake is connected to Highway 105 which comes from the south from Prince Albert via La Ronge (Fig. 1). Key Lake has its own air strip which is usable for turbo-prop driven aircraft, for instance F 27.

The Key Lake Project is owned 50% by Saskatchewan Mining Development Corporation (SMDC), 33 1/3% by Uranerz Exploration and Mining Ltd (UEM) and 16 2/3% by Eldor Resources Ltd.

The major phases of the development comprised:

The exploration period of UEM	1969-1975
Two ore bodies were discovered, the Gärtner ore body	1975
and the adjacent Deilmann ore body	1976
The delineation of the ore bodies and reserve estimate	1976-1979
The feasibility study	1978-1979
Commencement of mine planning and engineering for the mill and ancillaries, camp layout	1979-1980
Commencement of preliminary pumping of groundwater	1980
Granting of the surface lease	1981
Commencement of mill construction	1981
Overburden stripping of the Gärtner ore body	1982
Commencement of ore mining	1983
Startup of uranium production	3 Oct. 1983

2. THE MINE AND MILL COMPLEX

The ore mined is a complex ore with an average of 2.5% uranium and contains 2.6% nickel, 1% arsenic and about 1% graphite (see Table I). The ore reserves were

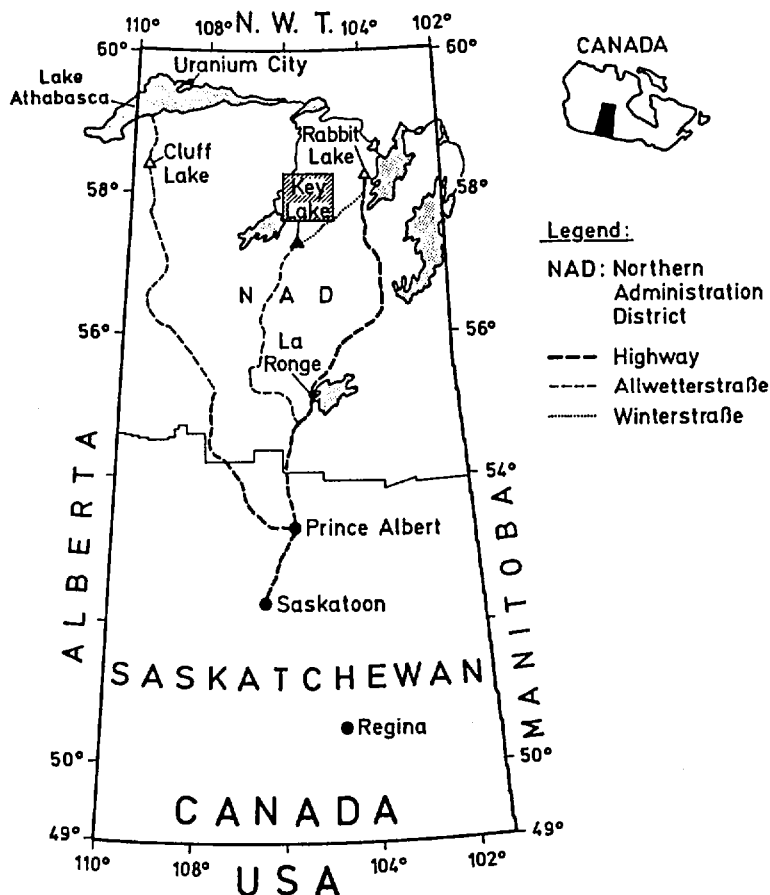


FIG. 1. Saskatchewan map.

estimated end of 1983 to 3.9 million t with about 90 000 t U_3O_8 (200 million lb)¹.

The design capacity of the mill is 12 million lb annually and the actual production in 1984 was 10.4 million lb U_3O_8 (87% of capacity) and in 1985 about 11 million lb U_3O_8 (92% of capacity). The deficit to full design capacity is due to a very long startup period caused by the many new process steps. These unit processes were used in the uranium metallurgy first time at Key Lake (Fig. 2).

In the open cast mine the ratio of overburden to ore is very high. It amounts for the Gärtner ore body to 12:1 and will increase in the Deilmann Mine to about 20:1. The Deilmann Mine will commence production about 1989. The grade of the Key Lake ores varies between 0.1 and 15% U_3O_8 and needs blending.

¹ 1 lb = 0.4536 kg.

TABLE I. URANIUM ORE — KEY LAKE

	Diameter (%)	Minimum (%)	Maximum (%)
Uranium (U_3O_8)	2.5	2.0	3.0
Nickel	2.5	1.5	3.5
Arsenic	1.4	0.9	3.0
Iron	3.5	2.5	5.0
Aluminium	9.3	6.0	11.0
Magnesium	3.0	1.0	4.0
Sulphur	1.0	0.5	3.0
Silicate	50.0	40.0	70.0

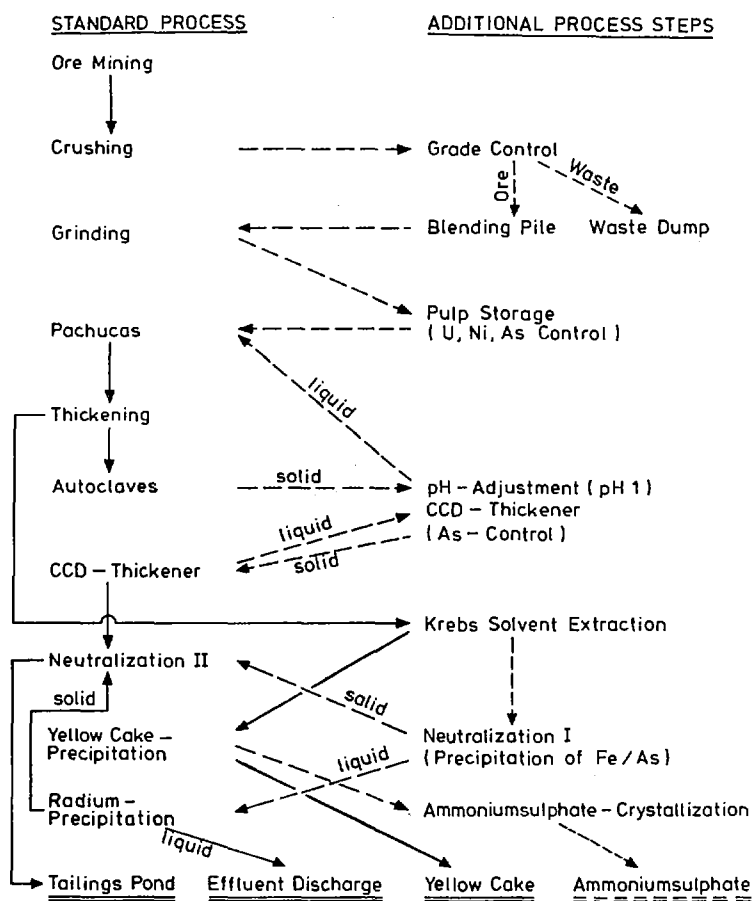


FIG. 2. The Key Lake process.

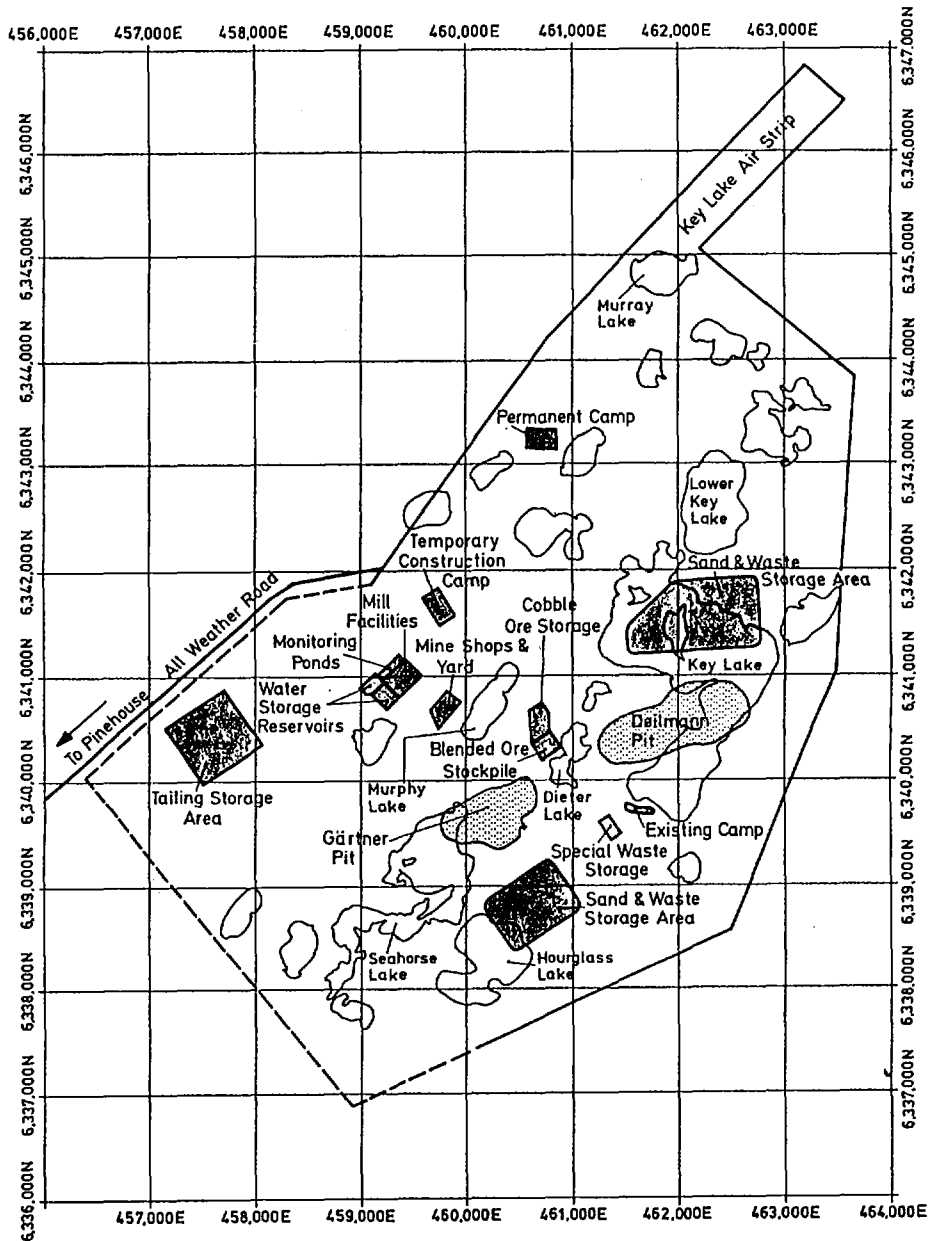


FIG. 3. Mill layout — Key Lake.

The daily mine production is about 4500 m³ overburden plus waste rock and about 600–700 t ore per day. The layout of the mine–mill complex is shown in Figs 3 and 4.

The mine is equipped with 24 m³ scrapers for stripping overburden, drill rigs for blasting ore and rock, 5 m³ shovels for loading waste rock and ore and 5 m³ front end loaders for loading, stockpiling ore and 3.5 m³ and 2 m³ front end loaders for blending and general handling of the ore prior to milling.

The open pit mine is shown in Fig. 5. The dimensions are about 100 m long, 370 m wide, 100 m deep. The abbreviated mill flow sheet is shown in Fig. 6. This flow sheet was developed by Uranerzbergbau in Bonn and piloted by Sherritt Gordon at Fort Saskatchewan, Canada.

The comminution of the ore is performed in two stages, with a 900 × 1397 mm gyratory at first, followed by a semi-autogenous mill (SAG) 4.42 m in dia and 2 m in length. Classifying of the screened ore is performed by Derreck Screens at 0.4 mm working in a closed circuit with the SAG mill.

Acid leaching is carried out at a temperature of 50–70°C with a first step at atmospheric pressure in four Pachuca tanks 81 m³ each followed by a pressure leaching at 650 kPa (6.5 atm) in ten 33 m³ autoclaves.

Leaching will need 3 to 4 h at a pH of 1.0. The recovery is 97%. For the solid–liquid separation 8 thickeners with 20 m dia are working in a countercurrent decantation process.

The solvent extraction plant was designed by the Krebs Company, Paris, based on their own experience with the same type of mixer–settlers in France and in Mounana, Gabon. Thirteen of those mixer–settlers are installed in series at Key Lake, four as extractors, three for the arsenic removal, four for uranium re-extraction, one for solvent regeneration and one for cleaning of the ammonium sulphate strip solution.

Nickel, arsenic and iron are precipitated from the leach solution with soda ash. Thereafter radium is precipitated with barium chloride. Uranium is precipitated with gaseous ammonia to ammonium diuranate yellow cake. This yellow cake is centrifuged and thereafter dried in a propane fired multihearth dryer. The yellow cake delivery point is shown in Fig. 7. Yellow cake is packed in 210 L drums for shipment to the Eldorado refinery at Port Hope, Ontario. A typical analysis of the yellow cake is shown in Table II.

The number of employees is about 475, of whom 113 work in the mine and 99 in the mill. The capital cost for the mine–mill complex was about Canadian \$450 million.

The development of the mine and tailings pond with all connecting roads was managed by Key Lake Mining Company (KLMC) and the development of the mill and ancillaries by the Wright/Commonwealth Joint Venture. The following companies were involved in the development of the project. A diagram of the tailings pond is shown in Fig. 8.

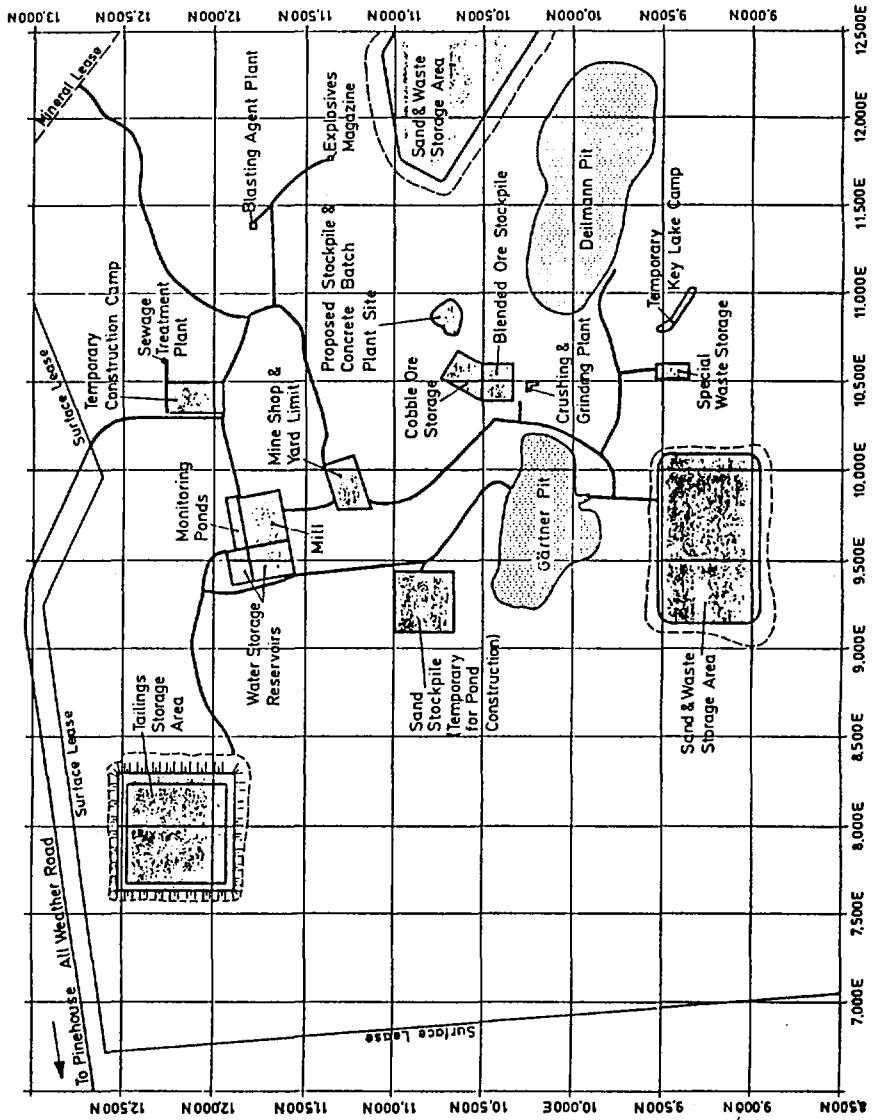


FIG. 4. Mine-mill complex — Key Lake.

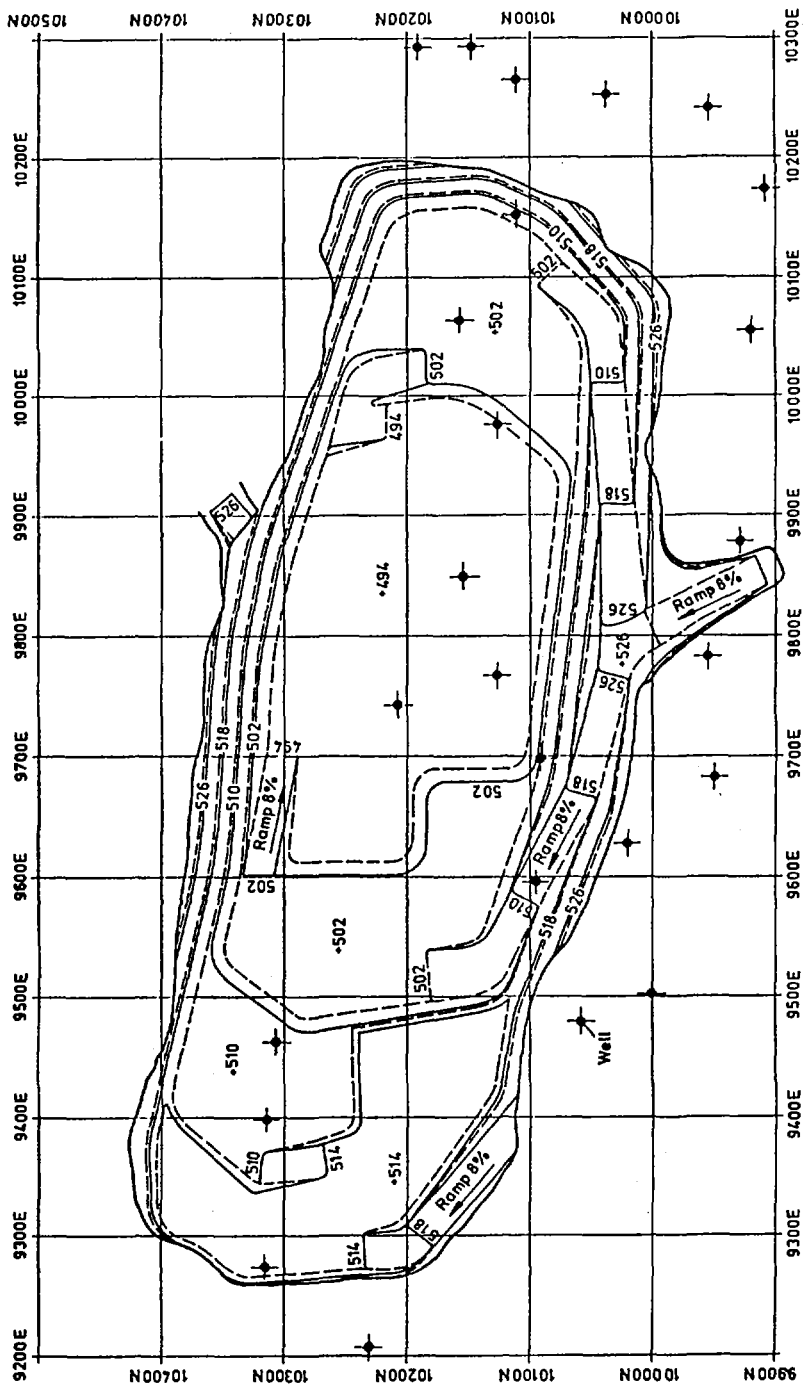


FIG. 5. Open pit — Gärtner.

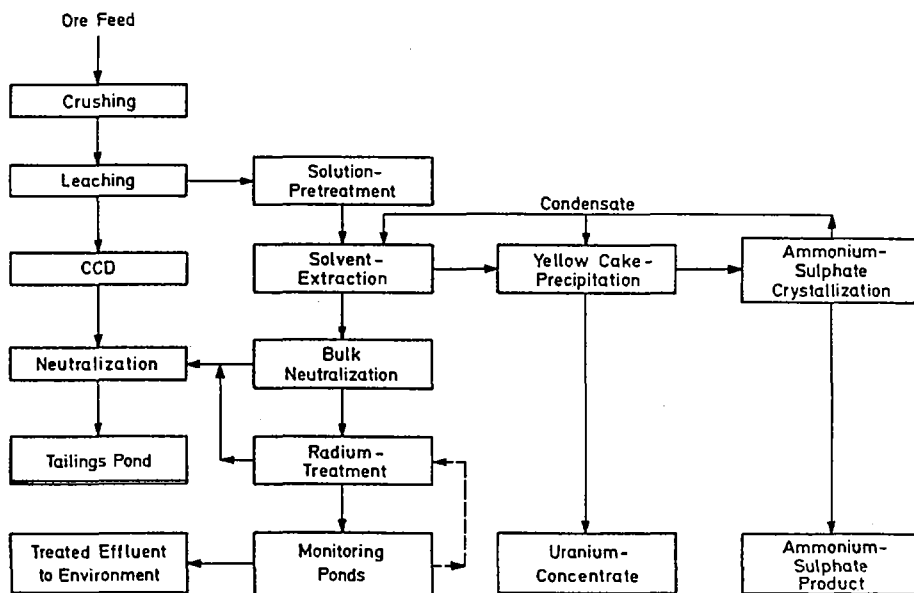


FIG. 6. Flow sheet — Key Lake.

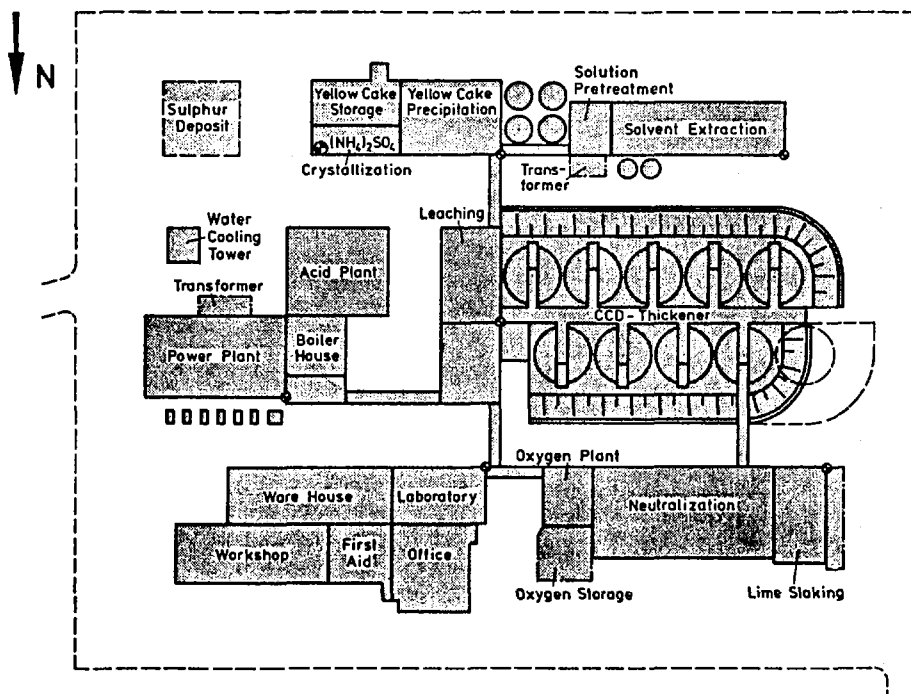


FIG. 7. Mill layout — Key Lake.

TABLE II. URANIUM CONCENTRATE — KEY LAKE (UNCALCINED)
(in per cent)

Uranium	75
Arsenic	0.015
Boron	<0.0001
Calcium	0.019
Carbonate	0.330
Fluorine	0.004
Iron	0.006
Magnesium	0.002
Molybdenum	0.057
Nickel	0.002
Sulphate	1.440
Thorium	< 0.015
Titanium	0.002
Vanadium	0.0002

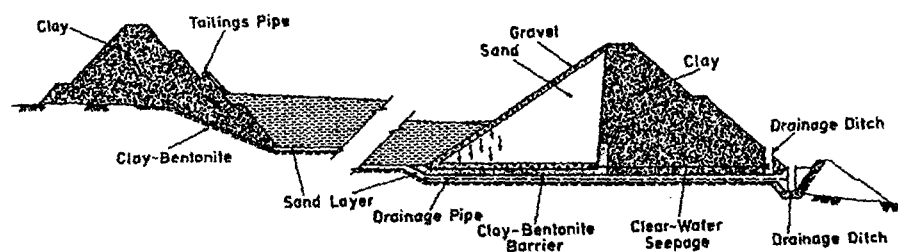
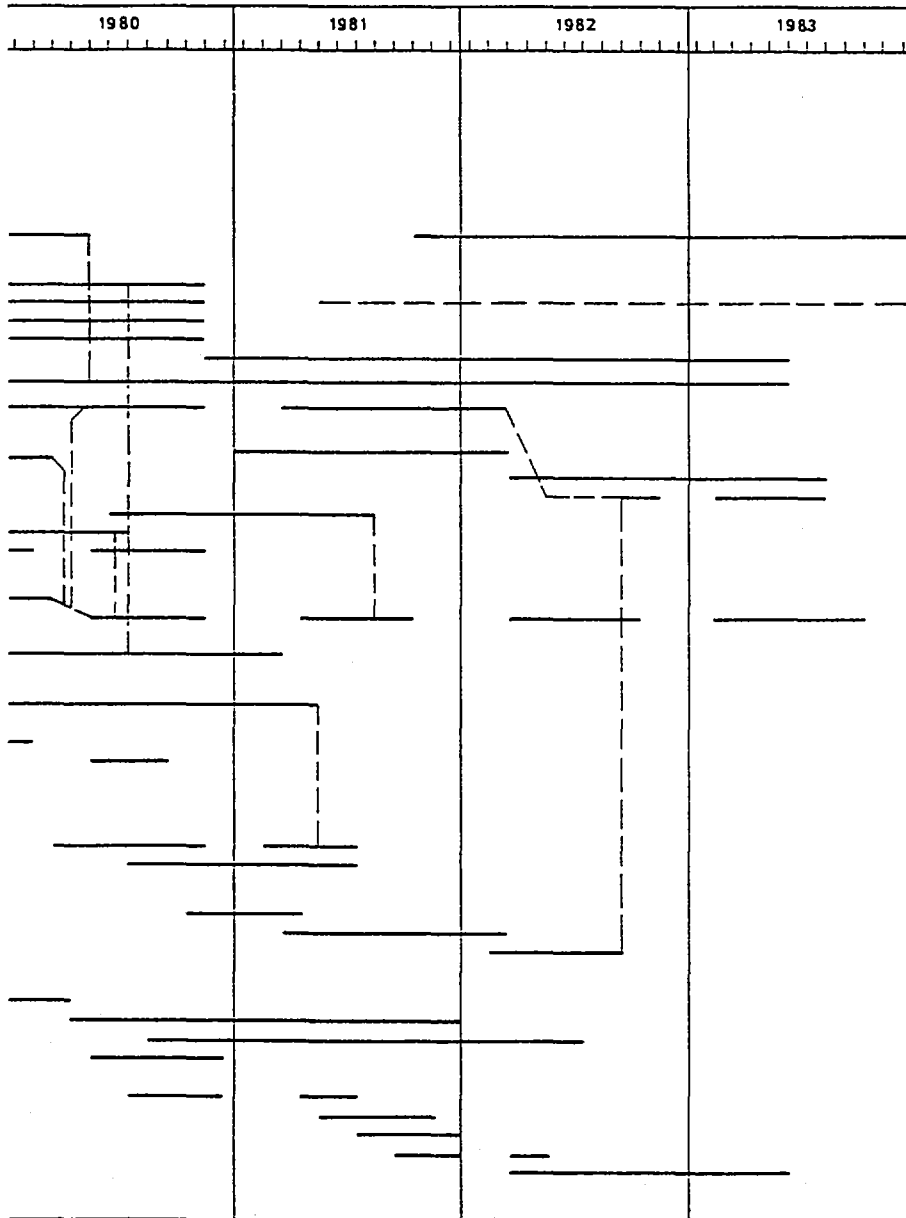


FIG. 8. Tailings pond — Key Lake.

ACTIVITY	1978	1979
<u>ENVIRONMENTAL IMPACT STATEMENT</u>		
Report preparations		
Government review		
Public hearings		
Board of inquiry		
Government approval		
Surface lease		
<u>FEASIBILITY STUDY</u>		
<u>GEOLOGY & ORE RESERVES</u>		
<u>DEWATERING</u>		
Lake dewatering		
Well drilling		
Installation pipe and electrics		
Design		
Permanent operation		
<u>MINE PLANNING</u>		
<u>SITE ROADS/UPGRADE AIRSTRIP</u>		
<u>STRIPPING & MINING</u>		
Equipment selection		
Stripping		
Mining		
Mine service facilities		
★ Temporary maintenance facilities		
★ Sample ore stockpile		
<u>WASTE MANAGEMENT</u>		
Design and procurement		
★ Construction (Maintenance)		
Muskeg dewatering		
★ <u>CAMPS</u>		
Temporary		
Permanent: Engineering		
Procurement		
★ Construction		
<u>POWER PLANT</u>		
Engineering		
Procurement		
★ Construction		
Temporary fuel tank		
<u>PRIMARY CRUSHER & STOCKPILE</u>		
Engineering		
Procurement		
Construction		
<u>PROCESS FACILITIES</u>		
Contract award		
Engineering		
Procurement		
Temporary warehouse		
Source of aggregate		
★ Earthwork		
Concrete		
Steel		
Cladding		
Mechanic and electrical installation		
★ Site activities started in 1980		

FIG. 9. Project schedule summary.



The breakdown of the work was as follows:

Exploration	Uranerz Exploration and Mining Ltd (UEM, Saskatoon)
Reserve estimate	UEB, Derry Mitchener & Booth, IREM MERI (Montreal)
Metallurgical test work	UEM, UEB, KLMC (Saskatoon and Bonn) Sherritt Gordon (Fort Saskatchewan)
Feasibility study	KLMC
Environmental impact statement	KLMC, Beak Consultants Ltd (Calgary)
Hydrology and dewatering	Stevenson International Groundwater Consultants Ltd (Vancouver), Rheinbraun (FRG), International Water Supply (Vancouver), Hayter (Saskatchewan)
Mine planning	KLMC, Golder Assoc. (Vancouver)
Mill design	KLMC, Wright Engineers (Vancouver) Krebs (Paris) plus suppliers
Mill construction	KLMC, Wright/Commonwealth (Vancouver) plus subcontractors
Tailings pond	KLMC and Knight & Piesolt (Vancouver)
Logistics, roads, air strips, camp	KLMC plus local subcontractors
Startup (cold tests)	Wright/Commonwealth plus suppliers, KLMC
Operating company	KLMC

The co-ordination of the various activities in engineering and construction was controlled according to the critical path method (CPM) and the programme evaluation and review technique (PERT) method. An example of a project schedule summary is shown in Fig. 9. Cost controlling during construction was according to a six digit cost code, whereby the first group of three digits identifies the major cost areas, sub-areas and systems, while the second group of three digits identifies the specific cost items. There is a possibility of adding three more suffix numbers, if required. An example of cost coding is given in Fig. 10.

During the second year of operation a comprehensive data processing system was installed for optimizing cost control and management information. The construction of the project was 5% below budget, but the beginning of the startup period was

4	5	0	2	3	1	8	0	0
Leaching Building			Equipment Foundation			Construction In Progress		

FIG. 10. Cost coding.

three months behind schedule. The delay was caused by the construction work during two very cold winters and by time loss during a strike of the construction workers. After two years of operation the project can be called a success.

DISCUSSION

M.C. CAMPBELL: What is the recovery in the Pachuca tanks in Key Lake?

W. FLÖTER: 35% of the uranium is dissolved in the Pachucas and 65.5% is dissolved in the autoclaves. The overall leaching recovery is 99.5%.

M.C. CAMPBELL: Is there any oxidation in that first stage in the Pachucas?

W. FLÖTER: There is no oxidation in the Pachucas. The run-of-mine ore varies in grade from 0.5 to 16% U_3O_8 . It is therefore stockpiled for blending. Our stockpiles have from 50 000 to 100 000 t ore, enough for about half a year of operation. During this time there is very good oxidation.

C.K. KIPPENBERGER: What was the difference between the capital estimate and the actual capital investment for the Key Lake Project?

W. FLÖTER: The capital estimate was about 500 million (Canadian) dollars and the actual capital investment was 470 million dollars. The estimate was 5% higher than the actual figure.

M.C. CAMPBELL: I wonder if one of the reasons for coming 5% under budget was that the estimate was made at a time of very high inflation and that when the project was completed inflation had come under control quite considerably.

W. FLÖTER: I don't think so. We have not taken inflation into account. The first estimate was made on 1980 dollars. Then we made a readjustment to 1983 dollars.

M.C. CAMPBELL: Does the total cost of 470 million include the modifications that had to be made during startup?

W. FLÖTER: No. That was an additional, although minor, expense. We spent 10 million dollars on modifications. The crushing tests that we did in the laboratory did not take into account weathering of the ore by rain and by very low temperatures (as low as minus 40°C). We had originally installed a hammer mill because we thought we had a competent rock. But, after weathering the ore became very soft and we had to install an autogenous mill.

DEVELOPMENT OF A SYSTEM FOR THE AUTOMATION OF A GRINDING CIRCUIT

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Abstract

DEVELOPMENT OF A SYSTEM FOR THE AUTOMATION OF A GRINDING CIRCUIT.

A system has been developed to automatically regulate the feed to the ball mill in order to achieve maximum efficiency. The system is based on an 'electronic ear' consisting of a microphone with a special electronic circuit which converts the noise from the ball mill, at a specially selected frequency, into an electrical signal. This signal is displayed on a meter and is also used to control the feed to the ball mill. Use of the system has resulted in greater automation, greater uniformity of the ground ore and reduced abrasion of the steel balls and of the mill.

1. INTRODUCTION

The Ellweiler uranium plant situated in the Rhineland-Pfalz, is owned by Gewerkschaft Brunhilde and was built in 1959 in order to make feasibility studies on ore bodies already found or under inspection. When exploration was started at the Menzenschwand underground mine in the area of the Black Forest the installations were enlarged to a capacity of 100 t/d. The production capacity is 850 to 950 kg/d U_3O_8 . In 1974 about 90% of the ore came from the Menzenschwand underground mine and the remainder from several other mines under development.

2. DESCRIPTION OF THE GRINDING CIRCUIT

The precrushed ore is crushed in a second stage, screened, and sent to a hopper. Then a conveyor belt, equipped with a scale, transports the ore to a ball mill operating in a closed circuit with a double rake classifier (see Fig. 1). The ball mill, supplied by Krupp, is 1600 mm dia, 2200 mm length and requires a ball charge of approximately 3 t. The throughput is checked each hour by weighing. At the same time the residue of the rake classifier is screened to 1 mm. The ore feed rate and water addition are regulated according to the weight and the screening.

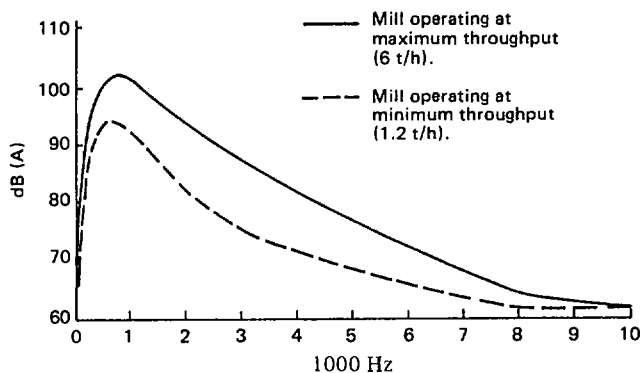


FIG. 2. Noise level as a function of frequency in the range of 0.4 to 16.3 mA.

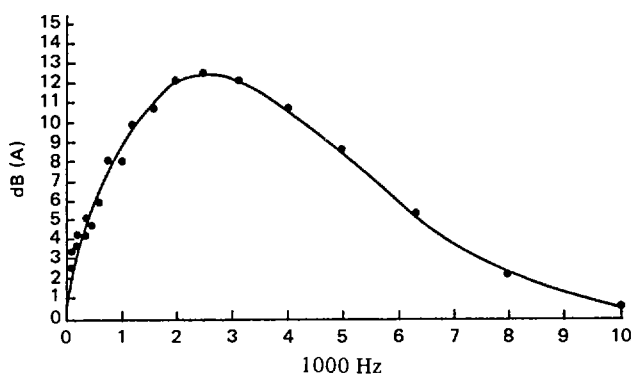


FIG. 3. Difference in the noise level at maximum and minimum throughputs in the range of 0.4 to 16.3 mA.

5. AUTOMATION OF GRINDING CIRCUIT USING AN 'ELECTRONIC EAR'

A microphone with a special electronic circuit was selected to convert the noise at this frequency into an electrical signal. The oscillation caused by the cascading charge is equalized electronically thereby obtaining a rather stable electrical signal in the range of 0.20 mA. This signal is then displayed on a meter.

The feed rate to the ball mill can be regulated for maximum efficiency using the electrical signal from the noise, as described above (see Table I, and Figs 4 and 5).

A carefully measured volume of water must be added to maintain pulp density at a predetermined value. This is regulated by a computer linked to a flow rate meter and the control valve. The particle size distribution of the feed and of the pulp solids at various mA readings is shown in Figs 6 to 11.

TABLE I. RESULTS OF INDUSTRIAL GRINDING TESTS: PARTICLE SIZES OF THE ORE FEED AND OF THE PULP SOLIDS AND MEASUREMENTS FOR THE DETERMINATION OF THE MOST FAVOURABLE FREQUENCY RANGE FOR THE 'ELECTRONIC EAR'

Ore feed rate (t/h)	Mean particle size		Ratio D'/d'	Grinding capacity $D'/d' \times \text{ore feed rate}$ (Factor)	'Electronic ear' signal (mm)
	Feed (D') (mm)	Pulp (d') (mm)			
1.2	2.2	0.36	6.1	7.3	2.0
2.2	4.1	0.35	11.7	25.7	7.9
3.2	3.8	0.41	9.3	29.8	8.7
2.8	6.1	0.38	16.1	45.1	9.9
3.6	6.2	0.40	15.5	55.8	12.3
6.0	4.9	0.46	10.7	64.2	16.7

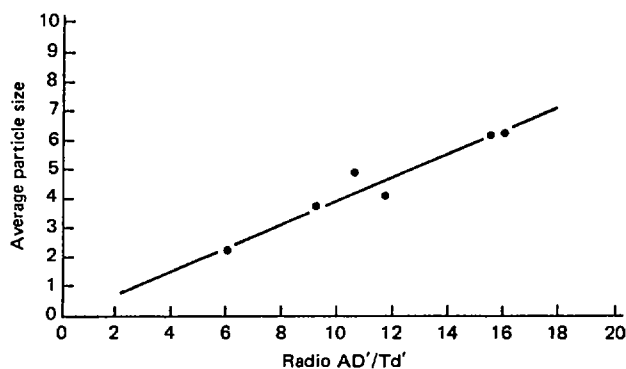


FIG. 4. Grinding efficiency for various particle sizes of mill feed.

6. PULP DENSITY MONITOR

Two small tubes of different lengths are fitted to the classifier under the pulp level. A small flow of low pressure air keeps the tubes free of pulp. The pressure difference between the two tubes is measured by a monitor equipped with a piezoresistive plate. This results in a signal of 0 to 20 mA which is displayed on the digital instrument. The digital instrument indicates the density.

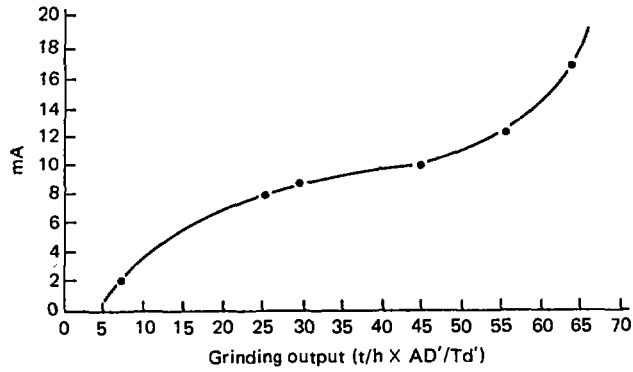


FIG. 5. Measured values of the electrical signal at different grinding outputs.

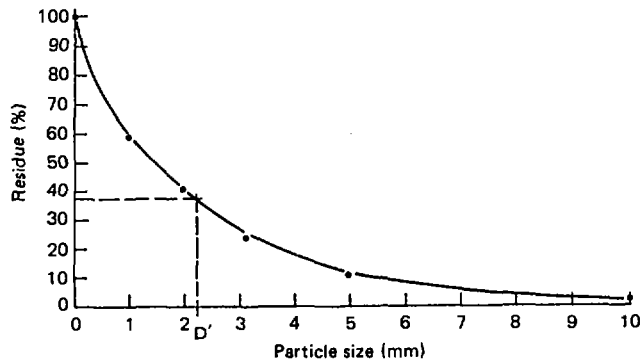


FIG. 6. Particle size distribution of the feed at 2.0 mA.

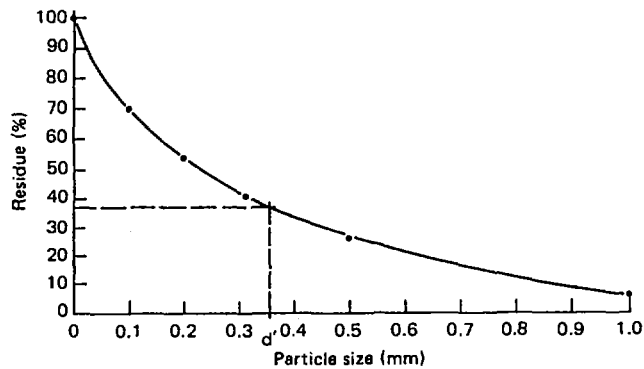


FIG. 7. Particle size distribution of pulp solids at 2.0 mA.

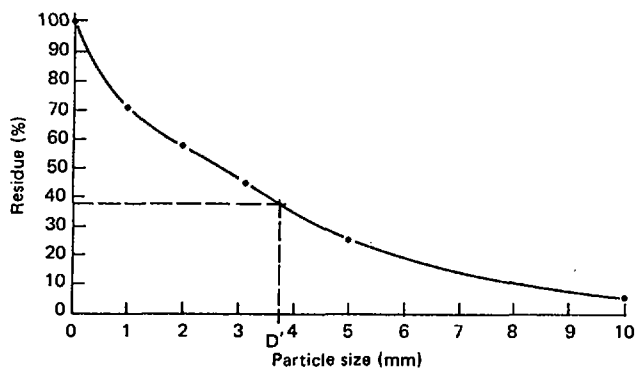


FIG. 8. Particle size distribution of the feed at 8.7 mA.

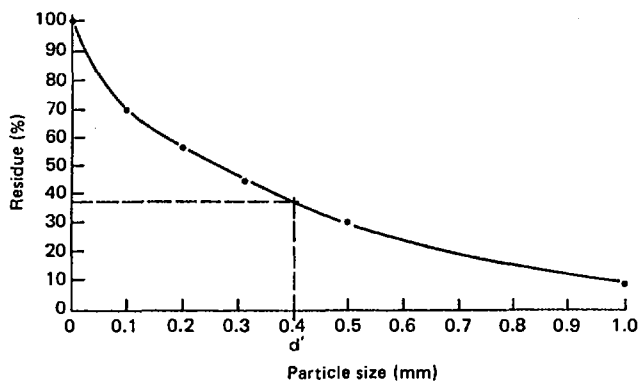


FIG. 9. Particle size distribution of pulp solids at 8.7 mA.

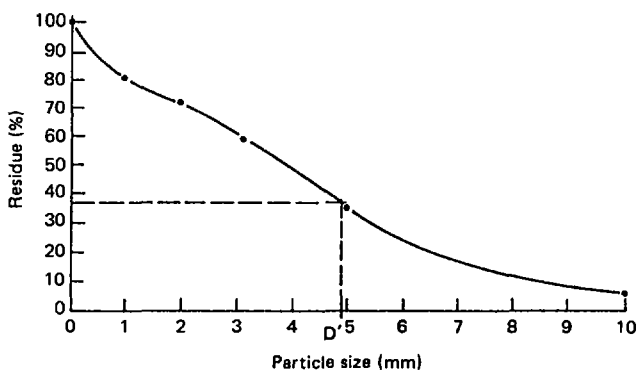


FIG. 10. Particle size distribution of the feed at 16.7 mA.

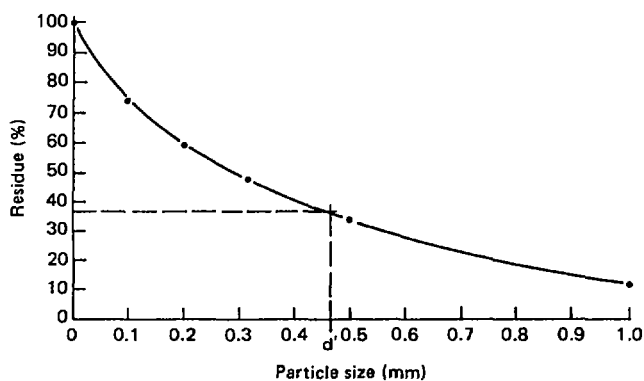


FIG. 11. Particle size distribution of pulp solids at 16.7 mA.

7. RESULTS AND CONCLUSIONS

The improvements in the grinding circuit described above have resulted in greater automation and greater uniformity of the quality of ground ore. In addition, the ball mill no longer operates empty or at a low load, and the abrasion of the steel balls and the lining has decreased by about 20%.

The automation programme, as described, permits the entire plant to be operated by only two persons per shift. The work station is now a central control room where all the information is displayed.

TEST PROCEDURES FOR THE METALLURGICAL EVALUATION OF URANIUM ORES

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Abstract

TEST PROCEDURES FOR THE METALLURGICAL EVALUATION OF URANIUM ORES.

The results of metallurgical test work are the basis for the feasibility study, which in turn is mandatory before an investment decision for a uranium project can be made. Sensitive areas for optimizing the process route are defined during the technical and economic evaluation of the milling process. Test methods, procedures, equipment and evaluation of test results with a microcomputer for the following steps of ore dressing are described in the paper: mineralogical and chemical analyses; crushing and grinding of ore; sorting and preconcentration of ore; leaching methods and reagent demand; solid-liquid separation; purification and enrichment of uranium leach solutions; uranium precipitation and product quality; waste management and effluent treatment; restrictions to process selection by the infrastructure. In addition to investigation methods for conventional ores, investigation of methods for ores from unconformity related deposits is discussed.

1. INTRODUCTION

Metallurgical test work on various samples of uranium ore is the basis for planning the process of a uranium concentrator. The results of the tests on the various ore samples demonstrate the flexibility and response of the anticipated treatment procedure. The results are utilized to design the first flow sheet and to select equipment and installations and they are also the base for investment and operating cost calculations. Chemical consumption and power requirements are particularly susceptible to optimization. Good test work can save considerably in operating costs at a later stage.

Table I shows a breakdown of mill operating costs and of total power consumption for mill cost centres. Provided also is a breakdown of personnel, reagents, energy and materials, which shows that reagents and energy account for 60% of the total mill operating costs. Good evaluation of test work will thus minimize capital and operating costs.

It may be noted in this context, that metallurgical test facilities and laboratories, dealing with uranium ores or other radioactive materials, are subject to certain laws, regulations, ordinances and guidelines issued by appropriate regulatory authorities who also monitor the strict compliance of the test facility [1]. Special permits and licences may be required.

2. ORE QUALITY

Prior to commencement of metallurgical tests, mineralogical investigations should be performed to yield the first information about possible behaviour of the uranium ore [2]. Commonly used are both polished and thin section investigations with magnifications from 1:50 to 1:1000. Data gathered cover type, size, distribution, shape, integrowth of ore minerals, abundance and type of possible interfering gangue minerals, particularly reagent consumers. The selection of the leaching method, alkaline or acidic, can be made based on this information. The information is also used to determine the intensity of liberation or to decide upon installation of possible preconcentration steps.

Chemical assays on ore samples are the most important information for developing the process. The list of elements to be determined should not be too short. Uranium ores are usually analysed for U, Th, V, Mo, P, CO₃, Ca, Si, Fe, Mn, S, SO₄, Cl. Sample size at minus 0.10 mm is 10–100 g and will be gained by quartering 10–100 kg ore from bulk, channel or chip samples. The methods used depend upon the detection limits envisaged. For the main constituents wet chemical methods or X-ray fluorescence are sufficient. Trace element determination may involve atomic absorption spectrometry, induced coupled plasma and other methods.

3. METALLURGICAL INVESTIGATIONS

In the early stages of test work, metallurgical investigations will be performed in the laboratory or on bench scale using representative ore samples from 50 to 200 kg. Several tests can be run at the same time in order to save time and funds. Parameters will be varied until optimum treatment conditions are reached. At the end of this test work, conceptual flow sheet and material balance of the process will be available [3].

This step will cost between US \$25 000 and US \$100 000 and requires 3 to 12 man-months. A list of the different process steps and methods applied in the treatment of uranium ores is provided in Table II.

The final process will later be pilot tested, either entirely or for specific sections. This is to confirm the previous laboratory results and to test the process, prior to plant engineering, in a larger and continuous operation.

A pilot test facility has a capacity of about 0.25–4.00 t/h and a total ore demand of 50–200 t or more. The test costs about US \$0.5–2 million and requires 1.5–5 man-years.

3.1. Mechanical treatment

Mechanical treatment of ore comprises crushing, grinding, ore storage and milling. Also preconcentration methods like flotation, gravimetric or electric separation,

TABLE I. COST AND ENERGY DISTRIBUTION FOR URANIUM MILLING

Cost centres	Costs (%)	Power consumption (%)
Comminution	5	45-60
Leaching	10	27-40
Solid-liquid separation	8	2-3
Purification	8	1-2
Yellow cake precipitation	7	2-3
Waste disposal	20	4-8
Chemical storage, power supply	5	1-2
Maintenance	32	—
Administration	5	—
	<u>100</u>	<u>100</u>
Preliminary cost types		
Personnel	25	
Reagents	35	
Energy	25	
Materials	<u>15</u>	
	<u>100</u>	

and colorimetric or radiometric sorting are used during mechanical treatment [4]. Radiometric sorting is, however, the predominant method for preconcentration of uranium ore. Applicability of radiometric sorting can easily be tested in the laboratory using hand held counters and specific small conveyors. Prerequisite is a high contrast ore, i.e. a good radiometric contrast between waste material and ore.

Tests on comminution can be performed in two steps. Crushing of at least a tonne size sample is recommended. This crushing test should be done by the manufacturer or supplier of the selected crushing equipment. Grinding tests require about 30 kg of material and can be performed by most test laboratories. This test determines the Bond Work Index which gives information about the energy requirement of the grinding mill [5]. The index unit is kW·h/t, usually between 6 and 25 for uranium ores. The index is calculated as follows:

$$W_{\text{Bond}} = \frac{44.5 \cdot 1.1023}{P^{0.23} \cdot G^{0.82} \left(\frac{10}{P^{0.5}} - \frac{10}{F^{0.5}} \right)}$$

$$W_{\text{mill}} = \left(\frac{10}{P^{0.5}} - \frac{10}{F^{0.5}} \right) W_{\text{Bond}}$$

where P is the fines (μm)

F is the feed ore (μm)

G is fines/revolution (g)

TABLE II. METHODS FOR TREATMENT OF URANIUM ORES

Ore feed		
	<i>Solids (%)</i>	<i>Method</i>
Fine ground	40-60	Agitated vessels
Fine ground	60-70	Pachuca tanks
Hammer milled	70-90	Vat leaching
Hammer milled or pelletized	90	Drum leaching (strong acid)
Crushed	95	Heap leaching
Mined	95	Stope leaching
Drilled	95	In situ leaching
Extraction		
	<i>Leaching</i>	<i>X-method</i>
Acid with } with } without }	full clarification	Ion exchange
		Solvent extraction
		Resin-in-pulp
Alkaline with } with } without }	full clarification	Ion exchange
		Direct precipitation
		Resin-in-pulp
In situ		Ion exchange Direct precipitation
Biological		Ion exchange
Precipitation		
	<i>In leaching</i>	<i>Precipitation reagent</i>
	Alkaline	NaOH, H ₂
	Acid	NaOH, NH ₃ , MgO, H ₂ O ₂

3.2. Ore leaching tests

The leaching method to be selected depends on the chemical analysis of the ore. For ores higher in carbonate than about 10% the alkaline leaching method is preferred, but in most cases leaching tests with both methods are conducted. Reagent consumption versus recovery rate is monitored through the test. The final decision upon the best leaching method is mainly based on the economics of the above ratio.

TABLE III. LEACHING CONDITIONS DURING STANDARD PROCESSES

	Acidic	Alkaline
Temperature	40–60°C	80–120°C
Retention time	4–16 h	24–96 h
Consumption	40–200 kg/t acid	30 kg Na as soda ash
Recovery	95–97%	85–90%
Pressure	100–400 kPa	200–800 kPa
Grain size	min. 500 μm	min. 63 μm

In acid leaching, acid consumption should be less than 200 kg acid per tonne ore, but other parameters like uranium grade, anticipated recovery, ore composition and structure must be taken into consideration [3].

During detailed leaching tests all relevant process parameters like retention time, temperature and reagent consumption must be investigated. (See Table III.) The applicability of pressure leaching or multiple stage leaching in Pachucas or vessels must also be considered.

Besides tests on conventional agitated leaching, tests on other methods like in situ leaching (ISL) or heap leaching are often of interest. Suitable tests on the permeability of the ore and core leaching are carried out in a special ISL test set-up. Core leaching results are reached from the viewpoint that cores from exploration drilling in general are vertically oriented to the bedding, while ISL is done parallel to the bedding (see Fig. 1). Therefore, the test results must be factored because normally the horizontal transmissivity is higher than the vertical.

The winning of uranium by heap leaching is of interest for dressing low grade ores [6]. For ores containing pyrite and sulphide, biologically assisted heap leaching can be investigated. The oxidation and acid built up by bacteria can result in high uranium recovery and in cost saving [7]. The duration of test work, however, is much longer than for standard laboratory leaching tests, because of the long retention time for ISL and heap leaching processes. Recoveries in the range of 75–85% can be reached in 100–300 d. However of great interest is the leaching effect during the first 30–50 d. From those test results the possible final recovery can be extrapolated by computer simulation.

3.3. Solid-liquid separation tests

The discharge from the leaching vessels contains the pregnant solution and the barren solids. Solids and liquids must be separated before the solution can be further processed for uranium extraction. Sedimentation and filtration tests are performed to

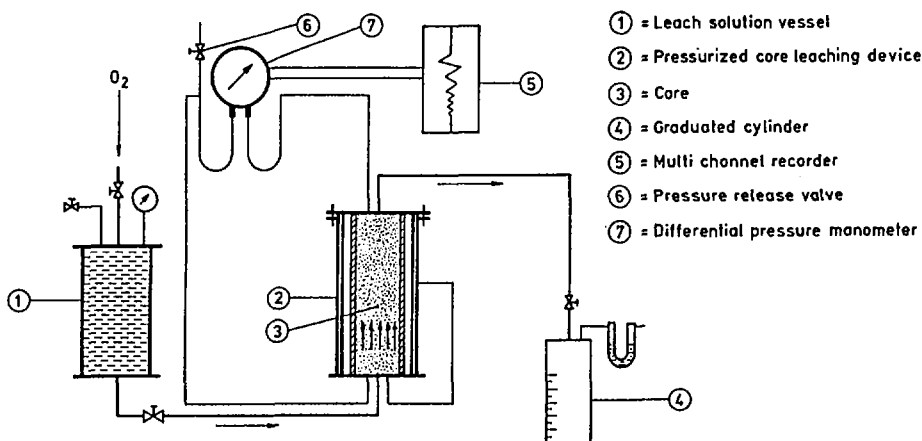


FIG. 1. Test station for core leaching (ISL) and permeability tests.

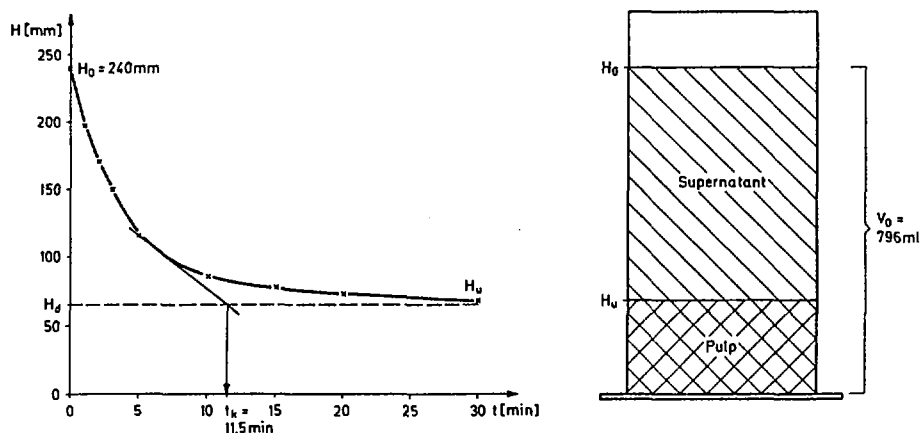
determine the equipment requirements. In addition to these methods a sand slime separation can be carried out for ores easy to leach and so to separate.

3.3.1. Settling tests

The most common method for solid-liquid separation is countercurrent decantation (CCD). Washing effect and required diameter of the thickener are determined in settling tests by studying the settling rate of the leached ore. The settling rate can be accelerated by addition of 50–200 g flocculants per tonne of solids. Sedimentation tests on pulp with and without addition of flocculants can be carried out in simple graduated cylinders. After feeding the pulp into the cylinder, the position of the interface between solid and liquid is measured over time. With the test results the required unit area of the thickener and the underflow density can be calculated according to the Kynch method (see Fig. 2). For uranium ores the unit area is usually in the range of $0.1\text{--}1.0\text{ m}^2\cdot\text{t}\cdot\text{d}^{-1}$ [8, 9].

3.3.2. Filtration tests

Filtration is usually used for separation of solids and liquids in alkaline processes. A simple test with a hand filter plate 10–20 cm in dia can be carried out to determine the required filter area for belt filters. The suitability of several filter cloths should be investigated [10]. The specific filter capacity for bulk products usually ranges between 2 and $15\text{ t}\cdot\text{m}^{-2}\cdot\text{d}^{-1}$. The test results can be used for the layout of several filter types.



t [min]	0	1	2	3	5	10	15	20	30
H [mm]	240=H ₀	197	170	150	115	85	78	73	68=H _u

$$\frac{t_k \text{ (min)}}{1.44} \frac{V_{011, \text{Pulp}} \text{ (ml)}}{S_{\text{Feed Ore}} \text{ (g)}} = \frac{11.5}{1.44} \frac{796}{77.3} = 0.34 \text{ (m}^2 \cdot \text{d} \cdot \text{t}^{-1})$$

FIG. 2. Evaluation of settling tests.

3.3.3. Sand-slime separation tests

Uranium extraction from leached pulps with poor settling characteristics is facilitated by resin-in-pulp (RIP) methods. However, the coarse mainly barren fraction has to be separated from the pulp prior to their application. While in earlier times of uranium extraction mechanical classifiers were used, today hydrocyclones are commonly used for classification.

The efficiency of hydrocyclones can be tested in the laboratory by determining the mass distribution in the split streams and the effective separation grain size. The minimum pulp feed should exceed 30 L of the representative pulp, with a grain size of about minus 0.5 mm [11].

3.4. Purification tests

For purification and upgrading the leach liquor, ion exchange (IX) or solvent extraction methods are usually applied. Direct precipitation of uranium from carbonic pregnant solution is possible, but at this time is not used commercially.

3.4.1. Ion exchange

The uranium extraction tests from acid and alkaline solutions can be carried out by IX [12]. Resins with good adsorption characteristics are available for a pH range from 1 to 14. For the test, different resins will be filled in glass columns with perforated plates and the solution directed upflow or downflow through the resin bed. Investigation of about 0.25–1 L of resin per specimen is needed for the test work. Parameters like optimum flow rate, loading capacity and breakthrough time should be investigated.

Usually the flow rate is between 10 and 30 bed volume replacements (BV) per hour, the loading capacity between 20 to 60 g uranium per litre of resin. For the elution of the resin, flow rates of about 10 BV/h will be used. With good eluants, 95% of the adsorbed uranium should be stripped after 10 BV. The type of the eluant (10% $\text{H}_2\text{SO}_4/(\text{NH}_4)_2\text{SO}_4$, NaCl) and the concentration have to be investigated.

3.4.2. Resin-in-pulp

The RIP adsorption method is usually only used for processes with low uranium grades and pulps with poor filtration or settling rates. In the tests unclarified solutions or fine suspensions are used as feed material. The test equipment can be an upstream column or multiple stage cells with air lift transport for the resin for continuous tests or simple agitated 0.2–1 L cells for batch tests [9]. After washing the resin with water the elution procedure is the same as for IX tests.

3.4.3. Solvent extraction (SX)

Solvent extraction is the most popular method for uranium extraction from acid solutions [13, 14]. First orientation tests can be carried out in simple stirred cells (100–250 mL volume). In earlier times, ether, tributyl phosphate (TBP) and alkylphosphoric acid were used as adsorption agents, now aliphatic amines are the most usual adsorbents, which are diluted with kerosene. Isodecanole is added as a modifier. For determination of the equilibrium curve (extraction isotherms), the pregnant aqueous and the barren organic phases are mixed in different ratios. The number of extraction stages can be calculated using the McCabe–Thiele method. Stripping of uranium out of the pregnant organic phase can be investigated by the same method with stripping solutions such as $(\text{NH}_4)_2\text{SO}_4$. Important factors for the extraction and stripping are pH and ion concentration. Several litres of pregnant solution have to be available for these tests.

For the final design and layout of an SX process continuous tests with a multiple stage set-up have to be carried out. A volume of about 0.25 to 1.0 m³ must be available.

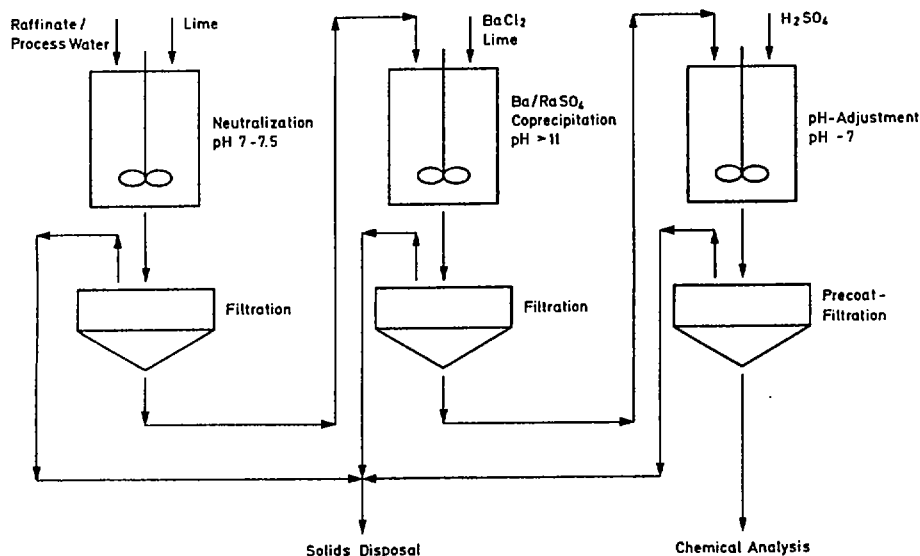


FIG. 3. Treatment of mill effluents.

3.5. Precipitation tests

The last process step in the uranium recovery cycle is the yellow cake precipitation. Precipitation tests can be done in the laboratory without special equipment. The uranium bearing solution is stirred in a beaker and the precipitation agent is added. Parameters like temperature, retention time, type of precipitation agent, final pH and a number of steps must be investigated. After washing the yellow cake precipitate the product quality and impurities must be assayed and compared with product specifications. The type of precipitation reagent depends on the leach chemistry used. For alkaline and carbonatic process solutions precipitation with sodium or magnesium hydroxide is the best way, for acid solutions precipitation with ammonia, hydrogen or hydrogen peroxide is preferred. Sometimes peroxide precipitation is mandatory to prevent co-precipitation of other metals. In this case, alkaline solutions have to be acidified.

3.6. Waste management

Strict regulations must be adhered to for disposal of the solid and liquid wastes derived from the mining and milling process. A pH adjustment and stabilization with lime is usually sufficient for disposal of the solid waste pulp. The treatment of liquid effluents is more complicated. Figure 3 shows one possible way of effluent treatment [15]. In the laboratory the first step of investigation is the addition of lime (bulk precipitation) and pH adjustment. After filtration barium chloride and lime are

added for Ba-Ra-sulphate co-precipitation. After another filtration step remaining radium and concentrations of other relevant ions are determined in the filtrate. Radium can be analysed by the emanation method or by direct alpha spectroscopy. In most IAEA Member States the Ra concentration in effluents should not exceed 0.7 to 3 pCi/L¹.

Besides the Ba precipitation, adsorption tests with Ra specific complexers [16] can be carried out by the method described in Section 3.4.1.

4. INFRASTRUCTURE

During planning and evaluation of the metallurgical tests specific conditions in the country, such as climate, infrastructure and availability of reagents must be taken into account. For example the availability and quality of process water must be investigated. In addition the influence of ambient temperature, barometric pressure and wind velocity are necessary for process building design. The availability of operators and their skill levels influence the selection of machines and processes and the grade of automation.

5. EQUIPMENT AND EVALUATION OF TEST RESULTS

Many of the metallurgical tests can be carried out with standard laboratory equipment. In some cases, such as grinding tests and solvent extraction, special equipment has to be made available. The installation of a microcomputer has proven to be very helpful in obtaining quick and reproducible information for evaluating test results.

REFERENCES

- [1] SCHLOSSER, A., Establishment of Facilities for Uranium Analysis, Report to the Government of Algeria, TC Report No. 2000, IAEA, Vienna (1983) 47pp.
- [2] INTERNATIONAL ATOMIC ENERGY AGENCY, Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores, Technical Reports Series No. 136, IAEA, Vienna (1980).
- [3] MERRITT, R.C., The Extractive Metallurgy of Uranium, Colorado School of Mines Research Institute (1971).
- [4] SAMOS, M.P., Mineral sorters, Eng. Min. J. (Jun. 1985).
- [5] BOND, F.C., Crushing and Grinding Calculations, Allis-Chalmers Mfg. Co., Milwaukee (1961).

¹ 1 Ci = 3.70×10^{10} Bq.

- [6] FLÖTER, W., SCHMIDT, Ch., Urangewinnung durch Solution Mining Verfahren, Braunkohle **10** (1980).
- [7] SADOWSKI, R., SCHÖNING, G., FLÖTER, W., "A quantitative evaluation of biological oxidation during the leaching of uranium ores", Progress in Biohydrometallurgy, Cagliari, Italy (1983).
- [8] HESTER, K.D., Extractive Metallurgy of Uranium: A Short Course, University of Toronto (May 1979).
- [9] LANDSIEDEL, K., Uranium Winning from Calcrete Ores, Final Report, Federal Ministry of Research and Technology, Bonn (1981).
- [10] KRAUSS, M., Internal Test Procedures-Filtration (1981).
- [11] AMBERGER KAOLINWERKE, Internal Test Procedures-Hydrocycloning (1981).
- [12] INTERNATIONAL INSTITUTE FOR METALLURGY, SOUTH AFRICA, Test Procedures, Ion Exchange (unpublished internal report).
- [13] MARR, R., BART, H.J., Metallsalzextraktion, Chemie-Ing.-Tech. **54** (1982) 2119-21129.
- [14] ROSNER, B., Uranextraktion mittels Solvent Filtration, Uranerzbergbau GmbH (Internal information).
- [15] KEY LAKE MINING CORPORATION, Internal notes.
- [16] BOYCE, T., BOOM, S., Removal of Soluble Radium from Uranium Minewaters by a Selective Complexer, Soc. Min. Eng. (Feb. 1982).

DISCUSSION

H.J. STEINER: When discussing the mineralogical investigation you recommended to study polished sections and thin sections. This is the recommendation made in all the textbooks on mineral processing, but I don't think it is a good recommendation because polished sections and thin sections are usually not representative samples. The mineral processing engineer doesn't deal with minerals but with particles and it is these that we have to investigate. I think we should recommend to study ground products embedded in resin and prepared as polished sections or as thin sections. It is also important to make a mineralogical study of the leaching residue. The mineral processing engineer wants to know why a certain amount of the uranium present is not leached, why certain phases of the ore do not respond to treatment.

K. LANDSIEDEL: I agree with you. Nevertheless, it is very important to have the mineralogical information before you start the metallurgical tests.

H.J. STEINER: This is useful, of course, but the samples must be representative. Representative samples of the ground ore and size fractions and density fractions of the ground ore should be collected by the mineral processing engineer and sent to the mineralogist.

W. FLÖTER: The mineralogical investigation can also be very useful when selecting the type of crushing equipment to be used. It is very important to see where the uranium is. Is it coating the grains? Or is it in the centre of the grains? Or is it in fissures? If you don't know this you may select the wrong type of crusher.

D.C. SEIDEL: It is also desirable to have the mineralogist as part of the laboratory team, so that he is actually involved in the test work. Another point is that

each type of ore may require a different combination of mineralogical and metallurgical investigations. For instance, some of the Canadian ores require a different type of examination than sandstone ores where almost all of the uranium is in the cementing material between the grains. In many instances, particularly in tertiary ores, it is not even possible to identify a specific uranium bearing mineral species.

K.H. TREUE: I believe the pilot plant costs that you mentioned are very modest. In three cases in which we were involved, Rossing, Yelirree and another one, the actual costs were about 10 million dollars, including, say, one year of operation of the pilot plant. There is often a tendency to underestimate the cost of a pilot plant. Of course, it is less costly to do more test work than to change the flow sheet in the industrial plant. One must consider not only the cost of the modifications but also the lost profits.

W. FLÖTER: I would hope that all deposits had enough uranium to pay for the costs of piloting. You could not cover those costs with a deposit containing four to five million pounds of U_3O_8 .

M. LYAUDET: I am surprised by the large size of the pilot plant you described. In France we generally use pilot plants with capacities of 10 to 100 kg/h.

W. FLÖTER: That may be small. The scale-up factor should be in the range of 1 to 10 with processes or equipment that are not very well known or perhaps 1 to 5 with well proven processes or equipment. I would recommend 1:10 or even 1:5.

THE URANIUM MILL OF THE COMPAGNIE MINIERE DONG-TRIEU: PROJECT PREPARATION AND IMPLEMENTATION

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Abstract

THE URANIUM MILL OF THE COMPAGNIE MINIERE DONG-TRIEU: PROJECT PREPARATION AND IMPLEMENTATION.

The uranium ore processing plant of Compagnie Minière Dong-Trieu (100% subsidiary of Total Compagnie Minière (TCM)) began production in June 1979 with an annual output of 200 t uranium. The output was gradually increased to 400 t uranium in the second half of 1982 and then to 500 t in 1985, of which 70 t came from the elution of resins loaded at another mining centre of TCM (processing by vat leaching). Following the description of the processing plant and the vat leaching operation, the paper presents the different stages of the development and implementation of the project from the time of the decision to create a processing plant to startup; the various stages of recruitment and training of staff which contributed to the speedy startup of the plant so that it reached the nominal production in the third month of operation; and finally the costs of development and implementation of the project.

INTRODUCTION

The Cherbois uranium ore processing plant is part of the Division Production Limousin of the Compagnie Minière Dong-Trieu (CMDT).

The original Company was called the Société des Charbonnages de Dong-Trieu established in 1916 in Tonkin, Indochina, where it exploited important anthracite deposits for 35 years. In 1954 it moved to France where it undertook exploration which led to the discovery, in 1957, of deposits containing appreciable quantities of uranium. In 1973 the Company was renamed Compagnie Minière Dong-Trieu, a wholly owned subsidiary of the Empain Schneider group. In 1982 it became a wholly owned subsidiary of Total Compagnie Minière.

The plant is currently producing 500 t uranium of which 70 t are obtained from stripping of resins loaded at another mining centre of TCM where the process is immersed vat leaching.

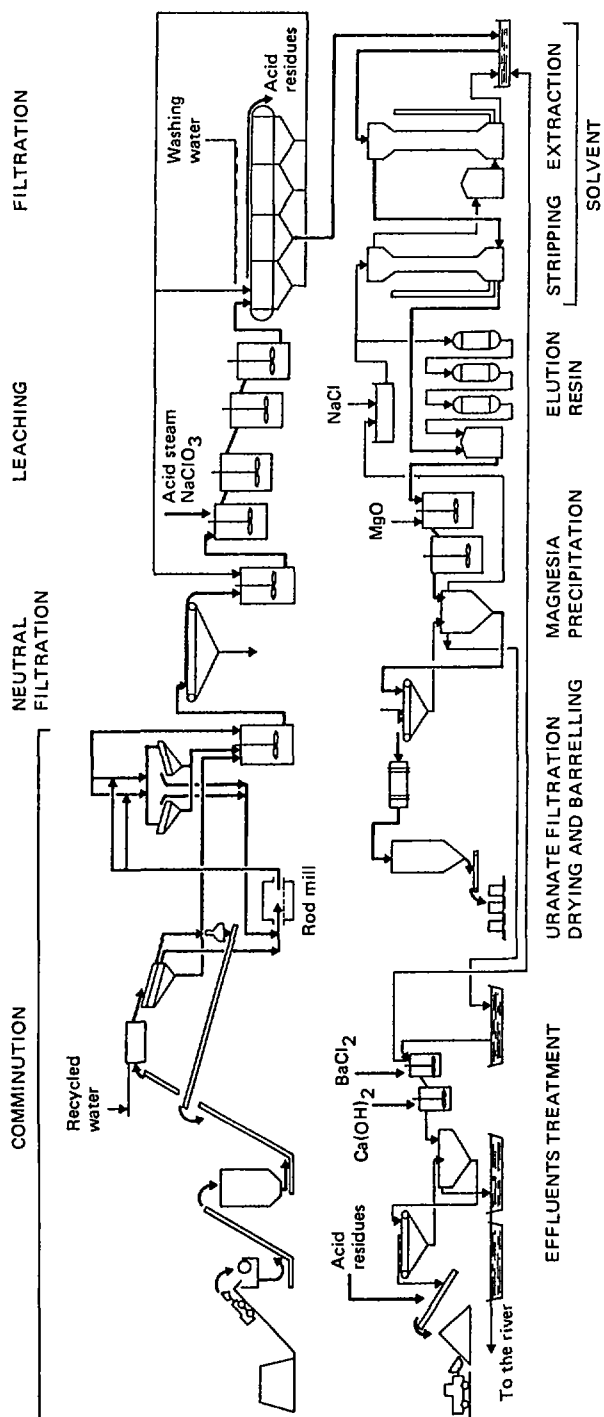


FIG. 1. Compagnie Minière Dong-Trieu plant — a simplified flow sheet.

1. DESCRIPTION OF THE PLANT

The plant (see Fig. 1) is located near Jouac in the north of Haute-Vienne; it is fed with a blend of various ores which come from five mines (open pit and underground) situated no more than 10 km from the plant. The uraniferous mineralization is usually composed of coffinite, autunite, gummities and pitchblende in episyenites more or less altered or in gneiss.

1.1. Comminution

The run-of-mine ore is crushed to 80 mm in a primary jaw crusher and then washed in a drum scrubber and screened; the different size fractions from the screening stage are sent, either to a secondary crusher (gyratory crusher), then recycled to the scrubber, or to a rod mill operating in a closed circuit with cyclones and vibrating screens. This circuit yields a final product of 90% weight minus about 500 μm according to the nature of the run-of-mine ore.

The pulp is filtered on two belt filters working in parallel (surface: $2 \times 25 \text{ m}^2$); the washing water of the cloth is recycled to the scrubber.

1.2. Leaching and acid filtration

The cake obtained from the comminution (neutral cake) is repulped at a density of about 1.55; the leaching is achieved at 65°C with sulphuric acid together with sodium chlorate in ten agitated tanks; the acid pulp is filtered and washed on two belt filters working in parallel (surface: $2 \times 70 \text{ m}^2$).

The acid cake (leaching residue) is carried away to an intermediate storage area by means of a belt conveyor, then removed using shovels and trucks and unloaded in a containment area. This area is made of dikes built with the barren materials of the open pit mine. The barren filtration liquors are recycled to the repulping stage of the neutral cake; the pregnant liquors directly supply the extraction stage (*without* intermediate clarification).

1.3. Solvent extraction and stripping

This stage of the process is achieved in two pulsed columns (in series), fitted with discs and rings.

The uranium is extracted from the pregnant liquors by an amine-type solvent and then stripped from the pregnant solvent with brine. The brine is partly prepared with the overflow from the magnesian precipitation thickener. The uraniferous stripping brine contains about 30 g/L uranium.

TABLE I. RESULTS OBTAINED SINCE STARTUP (June 1979)

Year	Dry tonnage of plant feed	Plant feed (% uranium)	Plant feed (kg uranium)	Total recovery (% uranium)	Average content of concentrates (% uranium)	Plant load factor availability (%)
1979	26 223	3.81	100 033	92.14	74.55	80.87
1980	70 759	3.58	253 287	95.60	74.99	88
1981	63 969	4.43	283 500	96.14	75.21	90.8
1982	57 871	4.49	260 067	97.21	75.47	88.2
1983	89 658	4.65	416 893	97.08	75.60	91.8
1984	92 754	4.73	438 346	97.37	75.56	92.6
1985 Jan. to Oct. U from ore	79 866	4.89	390 459	97.74	75.06	93.2
1985 Jan. to Oct. U from resins	—	—	38 810	99.99	—	—

1.4. Resin elution

SCUMRA, a subsidiary of TCM, processes uranium ore by immersed vat leaching in the south of France (see Section 2); the uranium from the leaching solution is loaded on resins, filling containers of 1.5 m³ capacity. The containers are then taken by truck to the CMDT plant (310 km).

Uranium is eluted with the same type of brine as for solvent stripping; the resins are then resulphated with an acid solution and washed with water; the washing eluate is recycled to prepare the acid solution, the resulphation eluate is recycled to prepare the brine. The uraniferous eluate contains about 15 g/L uranium and joins the uraniferous stripping brine before the magnesium precipitation.

1.5. Precipitation — drying and packaging

Uranium is precipitated from the uraniferous brines in the form of magnesia diuranate; after thickening, filtration and washing in a belt filter (surface: 3 m²), the diuranate is dried in a drum furnace and then put into barrels.

1.6. Effluents treatment

A fraction of the overflow from the magnesian precipitation thickener is used for the preparation of the brine. The fraction left is stored in a 4000 m³ impervious basin and then it is gathered with the extraction effluent to be treated with barium chloride and neutralized with slacklime.

After the effluents are settled (thickener dia — 20 m); the underflow is filtered on a belt filter, and then sent to the tailings pond with the filtration cake; the overflow is discharged into the river by way of two intermediate ponds working alternately (vol. — 10 000 m² × 2); this allows a final control of the treated effluent before its discharge.

1.7. Treatment of mine and run-off waters

The mine waters (which come from the underground mine and open pit located near the plant) and the run-off waters (from stockpiles and the containment dikes of the solid tailings) are recovered in the intermediate storage ponds. Then, they are pumped to the storage pond of recycled process water where they are neutralized with slacklime and treated with barium chloride.

Then the excess water is discharged into the river.

1.8. Technical information and results

The nominal capacity of the plant is 26 t/h; the present average capacity is 20 t/h. The plant runs five days out of seven a week and 47 weeks a year.

Table I shows the results obtained since plant startup (June 1979).

The responsible staff of the Processing Department is as follows (1st October 1985):

Plant	51
Maintenance	16
Laboratory	15
Staff	8

The consumption of reagents for the year 1985 (Jan. to Oct.) was as follows:

H ₂ SO ₄	67 kg/t	Ca(OH) ₂	42 kg/t
NaClO ₃	2.4 kg/t	Flocculant	370 g/t
Fuel	7.7 kg/t		
NaCl	2.5 kg/kg U	(U from ores and resins)	
MgO	0.4 kg/kg U	(U from ores and resins)	
Electricity	57 kW/t		

2. DESCRIPTION AND RESULTS OF THE HEAP LEACHING OPERATION

This operation is located at Bertholène, Aveyron; the ore body, subvertical and fissural, is exploited by an open pit and an underground mine; the uraniferous mineralization is usually composed of coffinite in orthogneiss.

2.1. Crushing

The run-of-mine ore is scalped at 75 mm, the +75 mm fraction is crushed in a jaw crusher.

After screening at 25 mm and crushing in a gyratory crusher of the 25/75 mm fraction, the ore is screened at 8 mm on a Lywell screen; the +8 mm fraction feeds another gyratory crusher operating in closed circuit with the screen.

The capacity of the station is 50 t/h.

The final product (0–8 mm) is carried away on a belt forming two piles of 2100 t each.

2.2. Immersed vat leaching

This section includes:

- (a) Three vats — each one contains 2100 t of crushed ore.
- (b) Three impervious ponds — 500 m³ capacity each, for the recovery of the leaching liquors, production liquors and washing solutions.

The time required for the treatment of a vat (from loading to unloading) is five weeks.

2.3. Uranium fixation resins

The production liquors ($1 \text{ m}^3/\text{t}$) feed six containers (in series); each one contains 1.5 m^3 of resins in a fixed bed. The feeding output is about $15 \text{ m}^3/\text{h}$ and the loading time of one container is about eight hours (the resins are loaded to about $70 \text{ kg U}/\text{m}^3$ resin).

When the head container is withdrawn from the line, a 'fresh' container is added at the end of the line.

The production liquors are clarified, before fixation, on a small cartridge filter.

The loaded containers are taken by truck to the Jouac plant for elution (see Section 1.4). Each truck carries six containers.

2.4 Effluent treatment

The fixation effluents are neutralized with slack lime and decanted into two ponds; the decantation overflow with which the mine and the run-off waters join is treated with barium chloride, then decanted into an impervious pond; the excess water is discharged into the river after passing through an impervious security pond.

2.5. Operation parameters

Operation at full rate of capacity started in 1985. Capacity is now about 5000 t ore per month; the average grade of the ore is $1.7 \text{ kg U}/\text{t}$. Overall uranium recovery is 94%. There are 12 staff members in the Processing Department, including 3 for the laboratory.

The consumption of reagents is as follows:

H_2SO_4	22 kg/t
NaClO_3	1.1 kg/t
$\text{Ca}(\text{OH})_2$	17 kg/t
Electricity	5 kW/t.

3. DEVELOPMENT AND IMPLEMENTATION OF THE CMDT PROJECT

Figure 2 presents the different stages of planning and implementation of the project, of recruitment and training of staff, and the startup of the plant.

3.1. Development of the project

Since 1974, the new shareholder of CMDT had to solve several problems.

The first problem was commercial: a customer had to be found for the yellow cake and a long term agreement drawn up ensuring financial feasibility of the project;

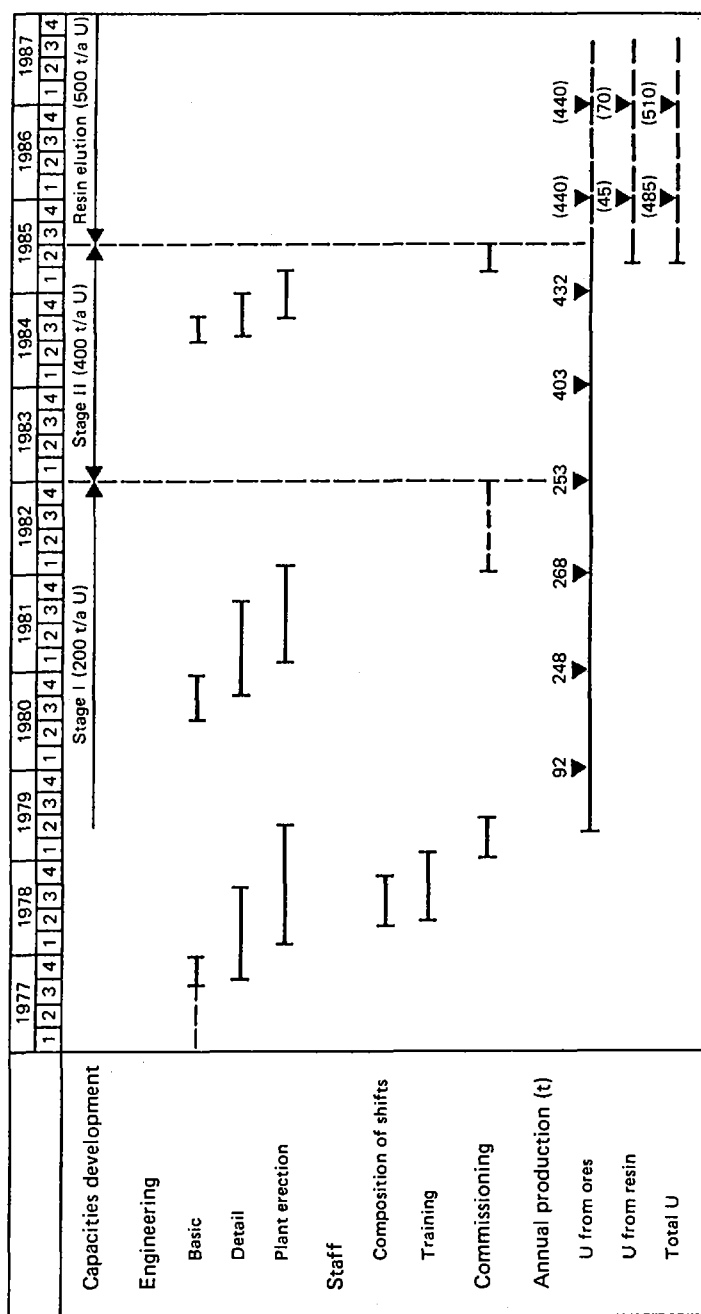


FIG. 2. Different stages of development and implementation of the project.

the situation from 1974 to 1975 was favourable and a long term agreement was made with Electricité de France (EDF).

The second problem arose when making the feasibility study, i.e. whether the ore should be processed directly or custom milled. Studies showed that processing directly was the best solution for technical, economic and strategic reasons. One of the arguments in favour of this solution was the high rate of recovery (96%) specified by CMDT.

The third problem was that the time available for the laboratory or testing was very limited and it was necessary to engage outside laboratories for this purpose. Quick results were obtained by using several laboratories. This method may not be the best one, but our experience shows that well supervised subcontracted studies can produce reliable technical information.

The fourth problem was the decision about whether the nominal production of 400 t uranium per year should be reached directly or through two stages of 200 t uranium in each year.

The selection of equipment also raised several problems, and finally it was decided to proceed cautiously and implement the project in two stages.

3.2. Implementation of the project

Having decided to build the plant in two stages, two problems remained — one of organization of the project and the other the selection of an engineering contractor.

Because of the presence in the Empain Schneider group of SPEICHIM, a well known engineering company, the selection was simplified, even if this engineering company had no experience in ore processing. The combined efforts of CMDT and SPEICHIM proved very successful.

The startup of the project of Stage II was very rapid and produced 400 t uranium in 1983, without interrupting the continuity of production for the years 1981 and 1982.

For the implementation of Stage II project, CMDT, with the support of SPEICHIM, was the prime contractor; implementation of the Resin Elution project was carried out by CMDT only.

3.3. Recruitment and training of staff

A global approach was taken to staff training as far back as 1978. The following methods were used:

- Preparation of the organization chart of the Processing Department;
- Description and analysis of the different shifts in order to define qualifications of persons to be recruited;
- Preparation of the training project and planning, taking into account the analysis of the shifts and the recruitment levels;
- Preparation of the recruitment project.

TABLE II. COSTS OF THE PROJECT (% of the total cost)

Type of expenses	Year										Total
	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	
Engineering	0.2	1.4	2.1	4.0	0.7	0.9					9.3
Internal services		0.5	4.2	3.6							8.3
Buildings-infrastructure works-equipment			14.7	10.3	1.1	6.0	0.5				32.6
Maintenance and laboratory				2.4	0.1	0.1	1.0				3.6
Equipment and tools		1.3	18.4	8.4	1.6	9.2	1.8	0.2			40.9
	0.2	3.2	39.4	28.7	3.5	16.2	3.3	0.2			
Subtotal	Stage I = 71.5				Stage II = 23.2						94.7
Resin elution									0.9	1.5	2.4
Other investments									Elution = 2.4		2.9
Total							0.5	0.7	0.9	0.8	100.0

The shifts had to include plant operators, maintenance personnel and laboratory technicians.

The training actions were defined by EUREQUIP¹ and CMDT as follows:

- Technical training of foremen
- Pedagogical training of foremen
- Instructions for foremen prior to training of workers
- Training of workers.

3.4. Plant startup

The nominal 50% production capacity was reached after a startup period of two months, 60% in the second month and more than 100% in the third month.

4. COSTS OF DEVELOPMENT AND IMPLEMENTATION OF THE PROJECT

Table II shows the costs of the project (expressed as a percentage of the total cost) for each year and for each type of expense. The costs are as of June 1985. The total investment cost, for Stage I and Stage II, as of June 1985, was F. Fr.166 million.

CONCLUSION

Ambitious targets were set by CMDT although its means were limited. However, it can be said objectively that the results were satisfactory.

DISCUSSION

A.T. TRIGGS: What is the recovery rate in vat leaching?

J.-L. POLGAIRE: The overall recovery in vat leaching is 94% and in ion exchange it is 99%. This is immersed vat leaching using ore crushed to 8 mm. This is the optimum size according to our laboratory tests.

F.-K. FELDMANN: What is the flow scheme of solutions from one vat to the other?

¹ EUREQUIP is a recruitment and training company; from 1978 to 1979 it was a subsidiary of the Empain Schneider group.

J.-L. POLGAIRE: For *each* vat there is a leaching step of about two weeks during which the liquor is recirculated into the same vat; after that there is a washing step, of about two weeks, in open circuit. The leaching liquors and the first washing liquors stored in the same pond constitute the production liquor sent to the fixation on resins. The final washing liquor is stored and recirculated for the first washing steps of another vat. The total solid to liquid ratio (for the leaching and the washing) of one vat is 2:1. The production of production liquor is 1 m³/t. For an ore grade of about 1.7 kg/t the production liquor assays about 1.6 g/L. We simulate vat leaching in the laboratory using columns and the results correlate well with those of the industrial operation. The most difficult part is to load the vat; we have to use a special technique to avoid compacting the ore. More details in the vat leaching process have been discussed in a paper by J. Mesplé et al. in the Aug.-Sep. 1984 edition of *Industrie Minérale*.

M.C. CAMPBELL: Why do you transport the resins instead of stripping and precipitating and transporting an intermediate product?

J.-L. POLGAIRE: This was the most economical solution. We do not have to have a brine storage, a precipitation step (with the related environmental problems) a filtration and drying step. We transport canisters each containing 90 to 110 kg of uranium. We can load 6 canisters on a truck and transport them 300 km to a central plant where we have invested F. Fr.4 million for the rest of the process. The cost of transportation is less than F. Fr.3 per kg.

H. MOVASAGHI: You use sodium chloride to elute the uranium from the resins and then you use alkyl amines for solvent extraction. Doesn't this poison the amine? Do you have problems regenerating the solvent?

J.-L. POLGAIRE: The production liquor (of the Jouac central plant) contains about 0.9 g/L of chloride and the extraction effluent contains about 3 g/L of chloride because there is an enrichment by the solvent from the brine. We regenerate about 8% of the solvent — with sodium carbonate in mixer-settlers — and we have no major problems.

B.SCHNEIDER: In your conventional plant you have a neutral filtration stage ahead of leaching. What do you achieve with this?

J.-L. POLGAIRE: This allows us to closely control the pulp density before leaching. We have two sets of belt filters working in parallel.

A. STERGARSEK: How many stages do you have for precipitation?

J.-L. POLGAIRE: We precipitate with magnesia in a single stage at a pH of 6.8.

DEVELOPMENT OF URANIUM OPERATIONS IN SASKATCHEWAN

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Abstract

DEVELOPMENT OF URANIUM OPERATIONS IN SASKATCHEWAN.

Amid the energy crisis of the 1970s, Canadian uranium exploration and development expanded after a long dormant period. The new frontier in uranium was in Saskatchewan, one of Canada's land locked western provinces. The discovery of a unique uranium mineralogy demanded new approaches for uranium recovery. Very high grades, difficult terrain with much surface water, unique mineralization, remote locations and lack of infrastructure posed new problems. The participation and co-operation of Canadian and foreign companies in the exploitation of these uranium deposits required ingenuity and dedication. The approaches considered and taken in the development of the Rabbit Lake, Cluff Lake and Key Lake deposits are examined to illustrate the scope of these technological and commercial achievements.

1. INTRODUCTION

Canada has been one of the major world producers of uranium since the early 1950s when the commercial demand first arose. By 1956, more than 10 000 radioactive occurrences had been discovered in Canada. Production soared and in 1959, twenty-three mines supplied 12 000 t uranium to the market. This was followed by a precipitous drop in demand to little more than 3000 t/a from five mines by 1966.

Exploration activities declined to a very low level. Nevertheless, in 1967 and 1968, Gulf Minerals and Mokta were both conducting airborne radiometric surveys in northern Saskatchewan. Radioactive anomalies due to granitic rocks led to surface exploration activities. Although the granites were not high in uranium, boulder trains indicated uranium rich occurrences. These were traced but no surface outcrops were evident. Surface drilling was therefore undertaken and led to the discovery of the Rabbit Lake and the 'D' ore body at Cluff Lake. Subsequent analysis of the established geological models led to further drilling and the discovery of the Key Lake, Midwest Lake and Cigar Lake deposits in the late 1970s and early 1980s (Fig. 1) [1].

When the 'energy crisis' struck in the early 1970s, Canada had this new uranium potential in Saskatchewan's Athabasca basin. Exploration activities

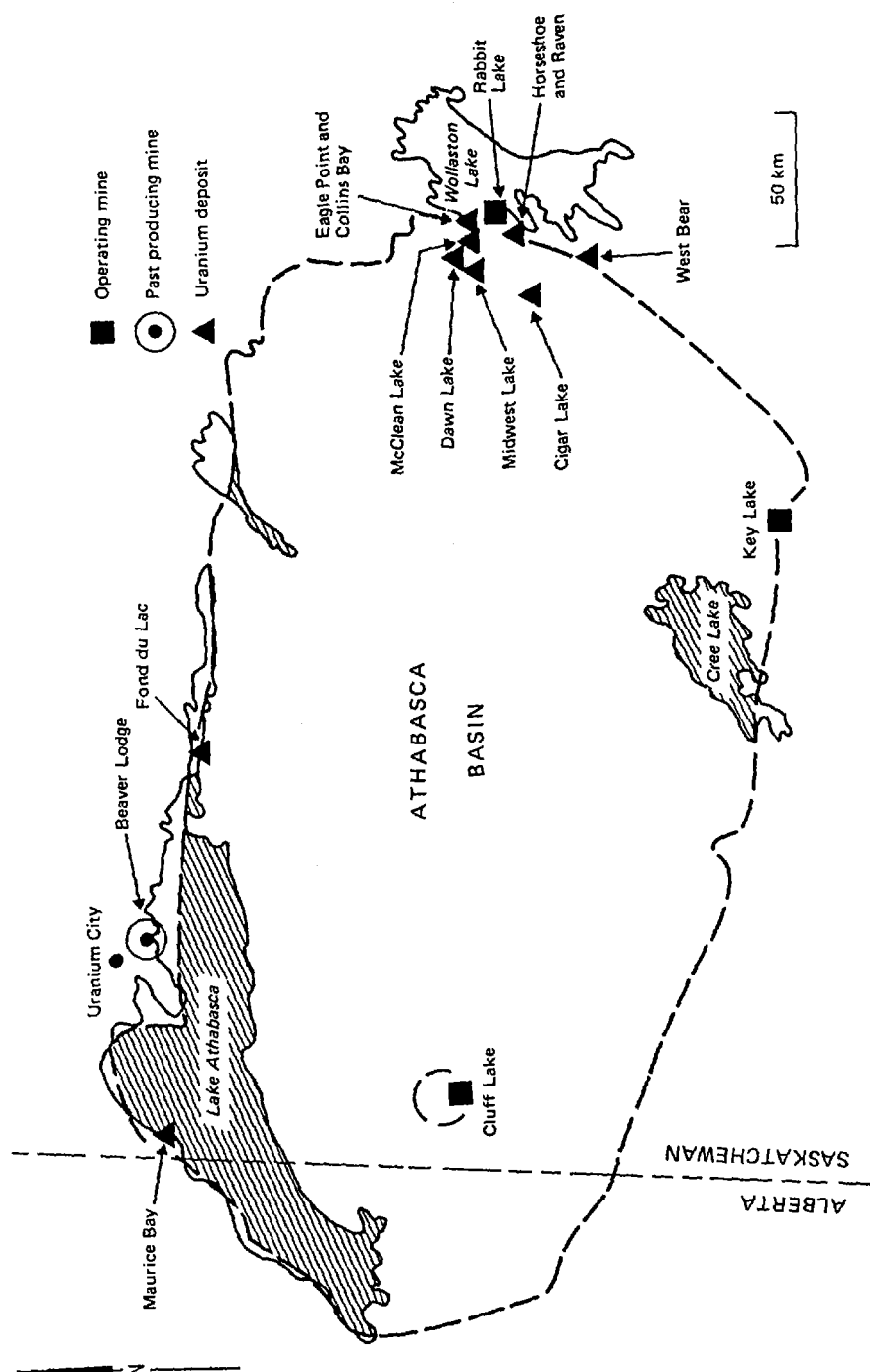


FIG. 1. Principal uranium deposits in Saskatchewan [1].

accelerated rapidly across Canada but most prominently in Saskatchewan. Further discoveries were made such that currently there are nine major uranium resource areas identified in Saskatchewan containing in excess of 200 000 t uranium. It is on this resource base that all of Canada's recent uranium production has developed. In this paper, the author examines the methods used to exploit these huge mineral occurrences which made Canada the largest producer of uranium in the world outside centrally planned economies area (WOCA) in 1984.

Currently, there are three operating uranium plants in Saskatchewan:

Eldor Mines at Rabbit Lake (formerly Gulf Minerals)

Cluff Mining at Cluff Lake (Amok Ltd)

Key Lake Mining Company at Key Lake.

All three of these operations have unique and interesting backgrounds which provide insights into the considerations that must be made in developing new uranium deposits.

In this paper the evolution of these uranium plants is traced with particular emphasis on environmental considerations, geology and mineralogy, mining considerations, process development and selection, startup and operation, waste treatment and current status. Each of these factors is reviewed providing some comparison of the approaches used.

2. ENVIRONMENTAL CONSIDERATIONS

It is interesting to review the development of uranium production in Saskatchewan. In Canada, the mineral rights belong to the provinces and each province has the authority to administer licensing as it sees fit. Uranium was discovered in Saskatchewan in 1935 and mined commercially beginning in the early 1950s without any public fuss or fury. Eldorado Mines Ltd operated its Beaverlodge mine and mill for 20 years. In 1975, Gulf Minerals Canada Limited began producing uranium from its Rabbit Lake deposit. The operation began much as any other mine with little fanfare and little public attention.

In November 1976, the Saskatchewan Department of the Environment received an Environmental Assessment and Safety Report from Amok Ltd on the proposed new uranium project near Cluff Lake. Although companies are required to provide certain information on the operation of a mine and its physical environment in obtaining mining permits and licences, there was no legal requirement for such a comprehensive study and report. However, after preliminary study by officials, the Minister of Environment for Saskatchewan decided to ask the Cabinet for a full public inquiry to review the report and study the broader implications of the uranium industry in Saskatchewan.

In February 1977, three commissioners were appointed to comprise the 'Cluff Lake Board of Inquiry'. The inquiry was to provide a comprehensive

assessment of the probable environmental, health and safety, social and economic effects of the Cluff Lake project and a review of the measures proposed by Amok 'to mitigate the harmful effects and to make the project acceptable in all material respects'. In fact, the inquiry became a forum for public debate of the uranium industry at large. The Chairman of the inquiry was E.D. Bayda, a highly respected judge, and the inquiry became popularly known as the 'Bayda Commission or Bayda Inquiry' as it will be called throughout the text.

The Bayda inquiry filed its report [2] to the Saskatchewan Minister of the Environment on 3 May 1978, some 16 months after its appointment. The Commission conducted formal hearings in Regina and Saskatoon and formal local hearings in about 20 smaller communities in northern Saskatchewan. Formal testimony alone exceeded 10 000 pages of transcript. There were 138 witnesses in 67 days of formal hearings and 32 written briefs. The Bayda Commission report exceeds 400 pages but provides an excellent overview in layman's language.

Participants in the Bayda inquiry fell into four groups:

Proponents	Pre-eminent nuclear scientists Older Established in their field Practical in their approach Work in the nuclear industry.
Opponents	Pre-eminent scientists (non-nuclear) Church groups Anti-nuclear groups Young, university trained Articulate, intelligent, sincere and idealistic Political activist and committed.
Uncommitted	Lack knowledge of nuclear industry Church groups Northern people Welcome a moratorium.
Disinterested	Consider decision a political one and responsibility of government.

The central question as to whether Saskatchewan should now permit uranium mines to be developed within the province led us to many other questions. Some related to the specifics of the Amok Cluff Lake project, but many involved the broadest aspects of the worldwide use of nuclear materials. There were many contradictions and in some cases it remained a moral rather than scientific question.

Chapters of the final report are as shown in Table I [2]. The Commission discussed seven issues as shown in Table II [2].

TABLE I. CONTENTS OF CLUFF LAKE BOARD OF INQUIRY REPORT [2]

Chapter	Title
I	Introduction
II	The Amok proposal
III	Radioactivity and the biological effects of radiation with reference to existing standards
IV	Health and safety of the workers
V	Environmental considerations
VI	National and provincial control arrangements
VII	Economic and social effects generally
VIII	The north
IX	Nuclear power: Safety and the disposal of nuclear wastes
X	Proliferation and terrorism
XI	Moral and ethical issues in the development and use of nuclear energy
XII	Conclusions and recommendations

TABLE II. ISSUES ADDRESSED BY THE BAYDA COMMISSION [2]

Do measures to protect health and safety meet legal requirements?
Are legal requirements adequate and are standards enforced?
Do measures to protect the physical environment meet the law?
Are such laws adequate and are they enforced?
What are the social and economic effects of the project?
Do such effects militate for or against proceeding with the mine?
Are the broader implications sufficient to warrant not proceeding?

The ultimate recommendation of the Commission was: "We recommend that the Cluff Lake mine/mill proceed subject to the conclusions we have reached and the recommendations we have made in this report". Those conclusions and recommendations related to health and safety, economic benefits to the northern people, and the need for additional baseline data. Interestingly, the Commission concluded that "Uranium can be mined and milled at Cluff Lake without serious deleterious effects on the environment but...". The condition was that Amok

should have proper baseline data, monitoring procedures, and controls. The Bayda inquiry established a benchmark not only for Amok's Cluff Lake but for the industry in general. The report concluded: "We recommend that the expansion of the uranium mining and milling industry in northern Saskatchewan proceed beyond Cluff Lake mine/mill subject to the applicable conclusions we have made in this report".

Subsequent to the Bayda Commission report, Amok received its permits and licences. In 1982, Key Lake Mining Corporation received its clearances to proceed with mining and milling at this, the third, of the three projects discussed in this report. The Key Lake project required only the normal review of health and safety, and environmental criteria. Further expansion in the future in developments such as Midwest Lake and Cigar Lake following this normal review pattern can be foreseen.

3. GEOLOGY AND MINERALOGY

Gulf Minerals began its operations in 1975 at the Rabbit Lake mine in the Wollaston Lake belt of northern Saskatchewan [3]. This is an unconformity vein type deposit [4]. The host rock is a siliceous dolomite which has been highly altered. The primary uranium minerals are colloform pitchblende, sooty pitchblende and uranophane. Other minor uranium minerals occur with assemblages of other alteration products such as kaolinite and vermiculite. Sulphides are present only in minor amounts.

Eldor Mines, the current operators of Rabbit Lake are developing their new Collins Bay deposit. This is some 10 km north of Rabbit Lake and occurs partly under Wollaston Lake. The ore zone consists of a high grade core surrounded by lower grade material encased in clay. Uranium occurs as pitchblende, and coffinite. Significant amounts of nickel, arsenic, lead, silver and gold accompany the uranium.

The Cluff Lake deposits of Amok Ltd which became the centre of the Bayda inquiry, were detected in the Carswell Dome within the Athabasca formation by an aerial radiometric survey. This occurrence is some 300 km west of Rabbit Lake. The Carswell Dome is a circular formation about 35 km dia. formed by some extreme force such as a meteor or a volcano about 467 million years ago. This structure is the host for heavy minerals, one of which is uranium. The 'D' ore body, which was the first mined, covered an area about the size of a football field and only 30 m deep but contained 5000 t uranium at an average grade of 7%. The main minerals are uraninite, pitchblende, coffinite and thucolite. The uranium occurred with hydrocarbons. The light fractions migrated to form conventional grade (0.4% U) ore zones (Claude, OP and Peter River) while the heavy high grade fractions remained as the 'D' zone. These are now associated with the host rock, a feldspathic feldspar.

Key Lake is about 150 km south-west of Rabbit Lake on the southern rim of the Athabasca Basin. The two ore bodies Gärtner and Deilmann were confirmed by drilling in 1975 and 1976. These are narrow and elongated ore bodies close to the surface. Uranium mineralization is in upper beds of sandstone and conglomerates a central portion of kaolinized rock and a lower zone in graphitic gneiss. The ore zones range from massive veins to disseminations [5]. Uranium occurs as oxides and silicates such as pitchblende, sooty pitchblende and coffinite. Associated minerals are nickel sulphides and arsenides (millerite, niccolite and gersdorffite) with minor amounts of many metallic sulphides and oxides. The matrix includes quartz, chloride and kaolinite.

4. MINING

One of the unique features of these Saskatchewan mining operations is that no permanent community is associated with them. The workers are flown in from several major communities in southern Saskatchewan and smaller communities in northern Saskatchewan to work a 7 d (77 h) week. They are then flown home for a 7 d break. This system now involves many hundreds of workers each week. On-site facilities include very comfortable housing, dining and recreational facilities.

All of these ore bodies, Rabbit Lake, Collins Bay, Cluff Lake and Key Lake occur near the surface. Although some had surface outcrops, the overburden is as little as 10 m in some locations. As a result, these ore bodies are mined using open pit methods.

However, mining has not been simple and straightforward in all cases. The very high grade of Amok's 'D' ore body demanded special precautions to protect the workers. The area was surrounded by a 2.4 m fence with a security gate. A 1 m sand cover was maintained over the ore body except in the limited area being mined. Mining equipment worked from this sand layer to minimize worker exposure.

At Collins Bay, the ore bodies are partly under Wollaston Lake. In order to mine the 'B' zone, Eldor Mines has dammed off the area surrounding the ore body and pumped the water away.

At Key Lake all vehicles have been shielded with 5 mm of lead even though radiation levels are well below standards. All vehicles are fitted with air conditioner pressurizers, which filter the air entering the operator's cab.

5. METALLURGICAL PLANT

Each of these Saskatchewan ores had its own unique characteristics. There was no one universal approach for treatment so that each ore demanded the development of a metallurgical process flow sheet to meet the requirements of recovery, economy, quality and environment control. The unique aspects of each ore are discussed.

TABLE III. ELDOR MINES: COMPARATIVE CHEMICAL ANALYSIS [3]

Element	Rabbit Lake	Collins Bay
U ₃ O ₈	0.31	0.37
As	0.003	0.47
Ni	0.014	0.35
Si	10	10
Mo	0.002	0.01
Zr	0.001-0.01	0.1-1.0
Bi	—	0.001-0.1

5.1. Rabbit Lake mill [3]

This mill was designed to treat about 2000 t of 0.37% uranium ore per day. Primary grinding used an autogenous mill but secondary grinding was in a ball mill. Sulphuric acid leaching was used with sodium chlorate as the oxidant. Countercurrent decantation (CCD) was followed by a sand filter for clarification. Rabbit Lake used solvent extraction with a tertiary amine for solution purification. Uranium was precipitated with ammonia.

The Rabbit Lake mill has now been converted to treat the Collins Bay 'B' ore. A typical analysis is shown in Table III [3]. The Collins Bay ore has more nickel, molybdenum and arsenic bearing minerals. This demanded circuit changes. Mineral dressing tests were unsuccessful in achieving segregation of the nickel and arsenic, therefore chemical removal steps had to be developed. Leaching conditions were optimal using conventional systems. The key development was in the solvent extraction system.

Rabbit Lake had to develop a system which would eliminate problems of crud formation associated with many of the metals such as molybdenum and zirconium and avoid putting ammonia chloride or sodium into the environment [6]. A strong acid strip process was developed (Fig. 2) [7]. Impurities are precipitated from the strip liquor. Uranium is precipitated using hydrogen peroxide. This system is now being incorporated into the Rabbit Lake mill to accommodate the Collins Bay ore.

5.2. Cluff Lake [8]

Amok Ltd began operations with a very unconventional ore, an average grade of 7% uranium but with wide variations. It was found that jigging and tabling could

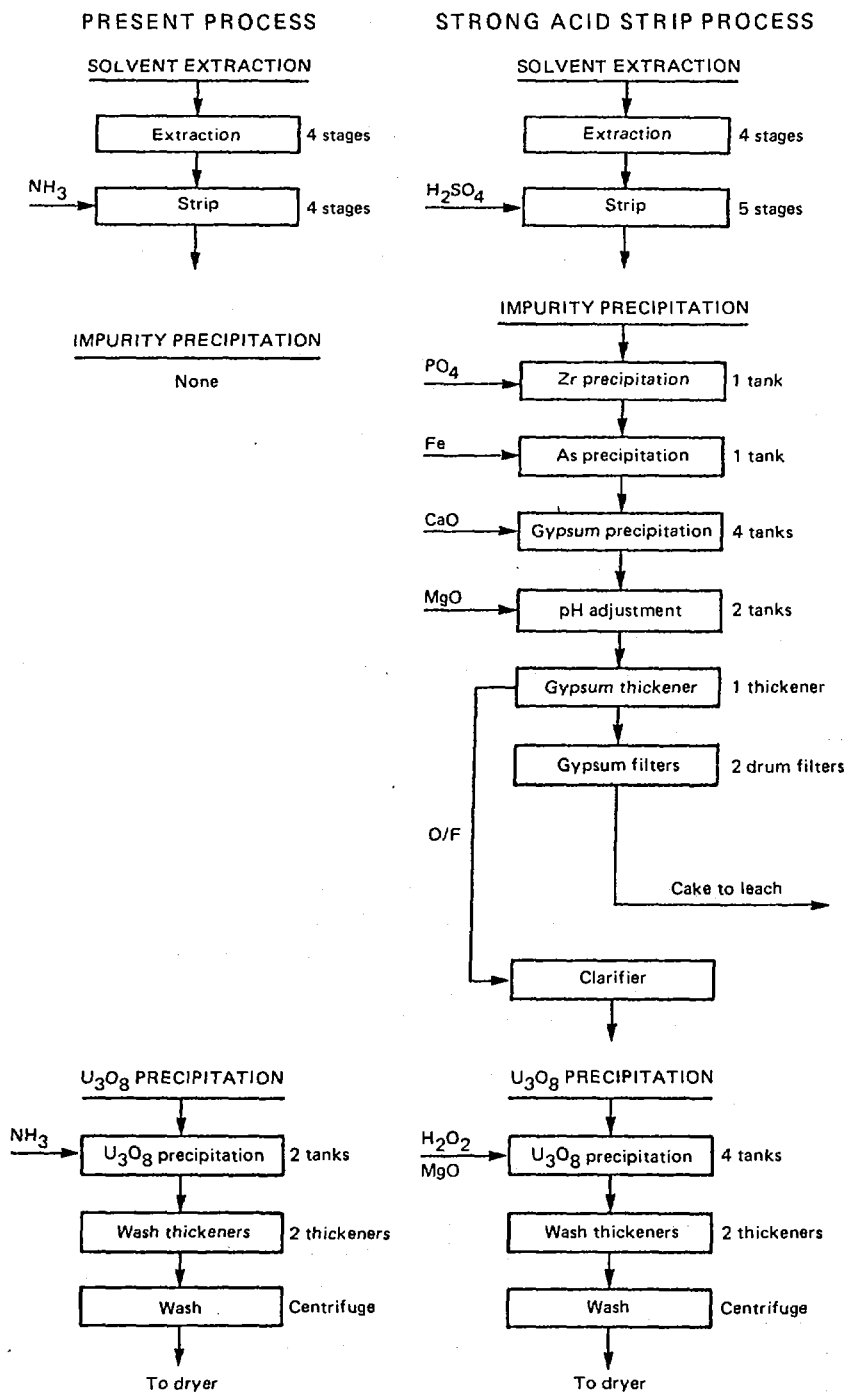


FIG. 2. Flow sheets – Eldor Mines Revision for Collins Bay Ore [7].

separate the heavy pitchblende from lighter fractions. The high uranium fraction was fed to the leach circuit. The gravimetric plant residue was stockpiled for later treatment with more conventional ore. The leaching capacity was about 1 t/h.

The grade of concentrate (25 to 30% U) was such that treatment was quite simple. Sulphuric acid leaching was carried out in mechanically agitated tanks (encased in 35 cm of concrete as a radiation shield). Liquid-solid separation was performed by belt filters with a high washing ratio. Iron was then precipitated using milk of lime to raise the pH. After removing the iron cake with belt filters, uranium was precipitated using magnesia. The circuit involved no resin ion exchange (IX) or solvent extraction (SX).

Now the Amok operation at Cluff Lake treats another ore of more conventional grade [9]. The mill is designed to treat 800 t/d of 0.5% uranium ore. Milling steps are much more conventional, as follows:

- Rod and ball mill grinding;
- Sulphuric acid leaching at 50°C in mechanically agitated wood stave tanks;
- Sodium chlorate oxidation;
- Countercurrent decantation because of the highly flocculated nature of the pulp;
- Sand filter clarification;
- Krebs solvent extraction plant using 5% amine, 4% isodecanol and kerosene;
- Magnesia precipitation of yellow cake;
- Proctor and Schwartz tunnel dryer.

5.3. Key Lake

The Key Lake ore was considered to present process complexities not faced by the other mills when development began in the late 1970s. The Cluff Lake inquiry had established that uranium developments would be closely scrutinized so that in addition to process economy, the environment aspects were paramount [10].

The Key Lake ore was found to present several concerns:

- Extreme ore variability demanded special grade control considerations
- High arsenic content (average 1.5% arsenic)
- Presence of graphitic gneiss
- Fluctuating high clay content.

The focus of much of the development work was on the leaching process. Early work by Uranerz in Bonn, Federal Republic of Germany, indicated that pressure leaching was effective. Definitive test work was then undertaken by the Sherritt Gordon Mines Limited to optimize the leaching circuit.

Work on graphite removal and flotation of the nickel arsenides proved uneconomic. The test work established that two stages of leaching, an atmospheric sulphuric acid leach followed by pressure oxidation leach was most effective in this

TABLE IV. CHEMICAL CHARACTERISTICS OF MILL EFFLUENTS:
KEY LAKE MINING CORPORATION 1984 [11]

		Maximum allowable monthly mean	1984 average monthly mean
Volume (1000 m ³)		—	117
pH		—	6.6
Ra-226	(Bq/L)	0.37	0.08
As	mg/L	0.5	0.16
Cu	mg/L	0.3	0.02
Pb	mg/L	0.2	0.02
Ni	mg/L	0.5	0.16
Zn	mg/L	0.5	0.04
Mo	mg/L	0.5	0.15
Se	mg/L	0.6	0.007
V	mg/L	0.5	0.12
U	mg/L	5.0	0.05
Th-230	Bq/L	3.7	0.15
Pb-210	Bq/L	1.85	0.59
Total dissolved solids	(mg/L)	25	14

ore. First stage leach residue is separated in a conventional thickener. Second stage tailing is washed in an 8 stage CCD circuit.

Key Lake uses Krebs mixer-settlers in its solvent extraction circuit with four extraction stages, three wash stages (to remove arsenic) and four stripping stages. Uranium is precipitated with aqueous ammonia, dried, crushed and packed.

6. TAILINGS AND EFFLUENT TREATMENT

The Rabbit Lake mill faced no particularly difficult environmental challenges. Tailings were neutralized with lime and discharged to a conventional tailings pond. Tailings decant was treated with barium chloride to precipitate radium. The final decant is clarified through a sand filter before discharge to the open environment.

Amok had a very different situation at Cluff Lake. Leach residues from the high grade ore contained in excess of 5000 Bq/g compared with about 40 Bq/g at

Rabbit Lake. These residues were encased in sealed concrete vaults and stored on high land on site. Of course these tailings amounted to only 5000 t of material. Iron cake and liquid effluents were treated in a conventional pond.

With the conversion of the Rabbit Lake mill to treat the Collins Bay 'B' ore Eldor Mines have found the original design to be most satisfactory. However, the presence of nickel and arsenic in solution has demanded an extra precipitation step using ferric sulphate to precipitate arsenic as ferric arsenate and lime neutralization to yield nickel hydrate. Tailings are then filtered to produce a filter cake which is transported to the old Rabbit Lake open pit for deposition and disposal. Filtrate from the tailings is then treated in the existing barium chloride-sand filtration circuit.

Key Lake has more complex mineralogy than other mines. High arsenic and nickel in the SX raffinates necessitates a bulk precipitation with lime. No nickel is recovered. Recently Key Lake added a sand filter and two IX columns for more economic treatment of contamination water recycled from the tailings pond. The maximum allowable monthly mean is shown in Table IV with typical actual results shown for Key Lake [11].

Initially, Key Lake contemplated flotation to isolate the arsenic and sulphide minerals from the residues. Such a step could also enhance nickel recovery. However, this did not prove to be economically feasible. Samples of tailings submitted to Knight and Piesold were tested using the layered deposition technique. These tests indicated superior stability with respect to weather and lower radon emissions. This sub-aerial deposition technique was finally accepted. This has proven to give satisfactory results in practice.

7. CURRENT STATUS

The Rabbit Lake mill is being converted to treat Collins Bay 'B' ore. The Rabbit Lake ore has been virtually exhausted and the Rabbit Lake pit will be used as an integral part of the tailings disposal system. Rabbit Lake has operated for over ten years and has produced 18 000 t uranium. With current reserves in the Collins Bay area the mill should operate well into the next century.

Amok's operation at Cluff Lake is now well into Phase 2. The high grade ore was mined, milled and treated in a span of four years. Phase 2 has a mill capacity of 800 t/d compared with 25 t/d in Phase 1. Phase 2 is treating lower grade ore. Current reserves indicate operations, based on the remaining ore bodies, for at least a decade.

Key Lake had a few teething troubles. Key Lake installed a semi-autogenous grinding system and this is working well. As mentioned earlier, sand filters and IX were added to improve the economics of effluent treatment. Winter conditions complicated the tailings disposal system so that a revised strategy utilized winter ponding and sub-aerial dispersion during the summer. Key Lake is working very

well after its first year. At 4000 t uranium per year Key Lake is Canada's largest uranium producer.

As can be seen, these Saskatchewan uranium plants have significantly changed the Canadian production capability. Recent discoveries show promise of several more deposits in the area with significant reserves. These too will have unique problems related to recovery. These new deposits will challenge the mining engineer as much as the metallurgist. They occur under lakes, in incompetent rock and with high grades and may require underground mining technology. The challenges of Cluff Lake, Key Lake and Collins Bay may pale in comparison.

REFERENCES

- [1] Uranium in Canada, 1984 Assessment of Supply and Requirements, Department of Energy, Mines and Resources, Ottawa, Canada (1985).
- [2] BAYDA, E.D., GROOME, A.J., McCALLUM, K.J., The Cluff Lake Board of Inquiry, Saskatchewan Department of the Environment (May 1978).
- [3] CARINO, A.B., Uranium recovery at the Rabbit Lake operation of Gulf Minerals Canada Limited, CIM (Can. Inst. Min. Metall.) Bull. 72 806 (Jun. 1979) 162-166.
- [4] McMILLAN, R.H., Genetic aspects of classification of important Canadian uranium deposits, CIM (Can. Inst. Min. Metall.) Bull. 91 800 (Dec. 1978) 61-67.
- [5] Key Lake Mining Corporation and Sherritt Gordon Mines Limited, "The Key Lake uranium process", 19th Annual Conference of Metallurgists, Canadian Institute of Mining and Metallurgy, Halifax, Nova Scotia, Aug. 1980.
- [6] MORRISH, A., "The future of the Rabbit Lake Mill of Gulf Minerals Canada Limited", Proc., Canadian Uranium Producers Metallurgical Committee (CUPMC), Department of Energy, Mines and Resources, Ottawa, Canada, Rep. ERP/MSL 82-42 (TR) (1982).
- [7] SWIDER, R.C., "Development of the strong acid strip process for Eldor Mines", Proc. Canadian Mineral Processors Meeting, Toronto, 1983.
- [8] FOURNIER, P., "Milling high grade uranium ore at Cluff Lake, Saskatchewan", Proc., Canadian Uranium Procedures Metallurgical Committee (CUPMC), Department of Energy, Mines and Resources, Ottawa, Canada, Rep. ERP/MSL 83-39 (TR) (1983) 1-24.
- [9] SCHNELL, H., FOURNIER, P., "Update of the Cluff Mining Phase 2 Expansion and Interim Operation", Proc., Canadian Uranium Producers Metallurgical Committee (CUPMC), Department of Energy, Mines and Resources, Ottawa, Canada (1984).
- [10] NEVEN, M., ROSNER, B., "The Key Lake flow sheet", Proc., Canadian Uranium Producers Metallurgical Committee (CUPMC), Department of Energy, Mines and Resources, Ottawa, Canada, Rep. ERP/MSL 80-158 (TR) (1980).
- [11] Key Lake: 1984 Annual Report to the Atomic Energy Control Board, Ottawa, Canada.

DISCUSSION

W. FLÖTER: Several important uranium deposits in Canada, like Cluff Lake Midwest, Collins Bay and Key Lake, were found in the same general area and at about the same time. Yet, they all have different process flow sheets and they are

all working efficiently and at a profit. Why is there not a 'Saskatchewan flow sheet' common to all these deposits?

M.C. CAMPBELL: There is no one process route that can be definitely said to be superior to the others. This is a matter of judgement and of the experience of the people involved in the project. It may not even be important to judge the superiority of one scheme over another. You design a process that will be efficient and profitable and once you believe you are right and have made a decision you do not want to change that decision.

K.H. TREUE: Based on your 30 years of experience with uranium mining and milling in Saskatchewan, what has been, in your view, the main change regarding environmental regulations and what has been the economic impact of these regulations?

M.C. CAMPBELL: The main change is that society has become more aware of the need to control effluents from the mines and plants and uranium producers are being forced to meet more stringent standards before they can even get a licence. When the early plants started in the 1950s one had to have a mining licence but there was no environmental licence as such. You treated the effluents as best as you could and the rest went into the local watercourse. Today, effluents are often required to meet drinking water standards. The cost of doing this is quite substantial. It probably represents 10% or more of the cost of the plant. But in some provinces you are not allowed to operate at all. The Bayda Commission did not oppose uranium mining and milling in Saskatchewan. British Columbia and Newfoundland have ruled against uranium exploration and development. British Columbia imposed a one year moratorium and Newfoundland a ten year moratorium. In Newfoundland the government inquiry asked consultants if they thought that in ten years uranium mining and milling could have better environmental controls. They said that they thought so and the authorities said: "That's very good. Thank you very much. We will wait ten years".

K.H. TREUE: Are the tailings dams at Key Lake regarded as permanent repositories or just as an interim solution?

M.C. CAMPBELL: They were designed as permanent repositories, they have been approved as such by the Atomic Energy Control Board and they are being operated on that basis. They probably are the best tailings dams installed and operating right now.

COMPUTER SYSTEM FOR OPTIMIZING A URANIUM EXTRACTION PROCESS

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Abstract

COMPUTER SYSTEM FOR OPTIMIZING A URANIUM EXTRACTION PROCESS.

A computer system has been developed to optimize the metallurgical process of a plant. The system includes models of the grinding and leaching sections, of the continuous ion exchange plant and of the solvent extraction plant. Based on analytical assays of five chemical elements and on the costs of water, electrical power, chemical reagents and other major consumable items, the system can perform mass and volumetric balances and can provide information which permits making operating decisions to optimize the performance of the process as a whole. This approach is superior to that of optimizing the operation of each individual section.

1. INTRODUCTION

From 1982 to 1984 the author worked as a Consultant for Rössing Uranium Ltd in Namibia. Since 1976 the company operates a large open pit uranium mine and extraction plant at a remote site in the Namib desert. The geology [1] and the technical aspects concerning mining, grade control and metallurgy have been described [see Refs 2, 3]. The improvement of leach extraction was reported in 1982 at the Uranium Meeting in Toronto [4]. In 1985 a paper was presented about centralized process control at the International Mineral Processing Congress in Cannes [5]. The decision to develop a metallurgical plant optimization system was made at the beginning of 1983.

2. OPTIMIZATION TOPICS

In spring 1982 a computer program for planning tasks was developed. On the basis of the figures, which were delivered from the Mine Planning Department, such as tonnage, mine grade, mine calc and rock type distribution on a monthly basis, we were able to predict key parameter values for the metallurgical plant and for the acid plant. This included not only monthly extraction and recovery figures to make a forecast for uranium production, but also monthly major consumable rates and quantities, and costs involved. Comparing these

planning figures with the actual results for 1982, we realized that the derived functions were good with one exception: uranium extraction. At that time we had not been very successful because there were a number of unmeasured or poorly defined variables which had an impact on extraction. Furthermore, the masking effect of one variable upon another variable has made the study difficult.

Then a computer program for metallurgical accounting was developed. Since August 1982 all figures are stored on a database. With that program we had the capability of comparing actual values with planning and model values. Each morning at 8 o'clock we could produce confirmed accounting figures from the previous day.

During that development work, the Data Processing Department's services were poor. In the meantime the Metallurgical Department acquired its own computer and terminals, which were installed in the chief accounting office, the analytical laboratories and in several offices.

By the end of 1982 information was available on a database from both computer programs. Then the decision was made to develop a metallurgical plant optimization system.

Through the metallurgical manager, the following optimization topics were highlighted:

- Low throughput operation;
- Effect of ore type on marginal processing cost;
- Effect of short term fluctuations on ore quality;
- Optimum size reduction;
- Optimum continuous ion exchange (CIX) elution cycle;
- Plant control philosophy.

After a year's experience with the metallurgical operation in Rössing Mine, the author found that for many of the above topics a solution could be worked out. However, it would be difficult to quantify marginal processing costs for single ore types and nearly impossible to define the effects of short term fluctuations on ore quality. This is understandable if one remembers how the rock type figures are derived. The geologist has geological maps available together with information from drill and blast holes and a visual impression of the ore after a blast. Roughly 90% of the ore is made up of alaskite and the rest can be complemented by five other rock types. The geologist makes a guess from experience of what the rock type distribution would be. There is no direct measurement possible and therefore each function, which has a link to a rock type, should be used with reservation.

Following internal discussions in the metallurgical department we decided on the way in which the metallurgical process optimization should be carried out. In the past, the work on optimization was oriented more to single process areas under technical aspects, because the complexity of the Rössing flow sheet

presented difficulties for easy handling. We now thought that we had sufficient information from the single areas to optimize the whole process. We wanted to include all variable operating costs, except personal costs, for the single process areas. This would result in an overall economical optimization of the process.

3. PROCESS FLOW SHEET

On the detailed process flow sheet one can identify 295 process units. This flow sheet is not suitable for an optimization task. A less detailed version is shown in Fig. 1. There are 45 major process streams and 20 process units.

The following processes are involved:

- Crushing
- Grinding
- Leaching
- Sand washing
- Slime washing
- Tailings handling
- CIX plant
- Solvent extraction (SX) plant
- Product recovery plant
- Service facilities.

The ore is crushed in four stages below 19 mm, then ground to 10 mesh in open circuit rod mills. Leaching is carried out at 40°C with the addition of sulphuric acid, manganese dioxide and ferric sulphate (produced by leaching out plant calcine) in agitated leach vessels. The leach discharge is split into a sand fraction which is washed in three stages with cyclones and rotoscopes and then put into a slime fraction which is washed in a five-stage countercurrent decantation (CCD) circuit. The sands and slimes are recombined and pumped to the tailings dam. The pregnant solution is upgraded and purified in a CIX and a SX plant. The concentrated liquor is treated with ammonia to produce yellow cake. This precipitate is washed, filtered and calcined to the final product, uranium oxide.

For the process optimization special attention must be given to the closed loops of return dam solution, barren return solution, return solution, eluate solution and the yellow cake thickener overflow stream.

The Accounting Department redesigned the flow sheet to simplify the method of costing.

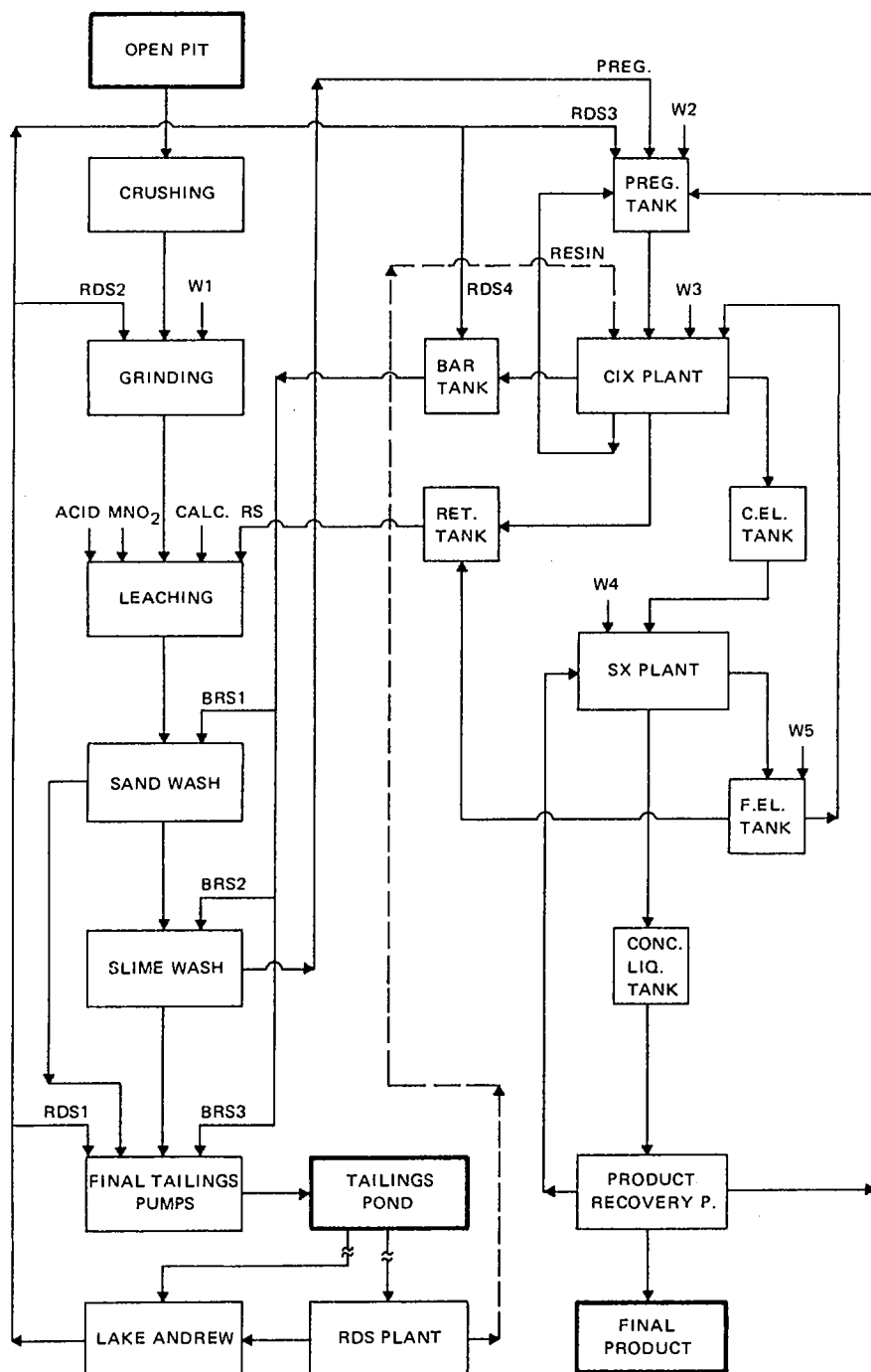


FIG. 1. Flow sheet for process optimization.

4. OPTIMIZATION FOR SINGLE PROCESS UNITS

In previous years a few models for single operating areas were developed by independent consultants and by Rössing metallurgists.

A grinding model exists, which is not used on a regular basis because it is extremely difficult to determine the breakage matrix and the selection matrix for a large rod mill especially when it is not possible to control them in a laboratory unit. During the last months we ran the plant with reduced ore throughput and it was observed that the grind was finer than before. Normally we have a sand fraction of 80% and a slime fraction of 20%. Now we have sand fractions between 74% and 76%. Figure 2 shows a relation between the specific rod mill throughput in t/h and the sand fraction in per cent.

For the leach area some regression functions were established, which allow a prediction of uranium extraction, acid consumption and manganese consumption. Uranium extraction equations were not as good as desired. On a daily basis we had much noise, mainly because of the lag of 6–10 h between head and tails values. When we took average figures for a week or a month, the variations were levelled out and we could not see any impact on the extraction. Normally a change in a process variable of the order of 5–10% would result only in an extraction change of 0.5–1.5%. We know that the ferric level, retention time, grinding and temperature play important roles as does the consumption of sulphuric acid. The impact of two of these variables is shown in Fig. 3. We have the option to vary, in a limited range, the ferric level by the utilization of the calcine reactors and we can change the ferric/ferrous ratio by addition of manganese ore. The following trends are shown in Fig. 4:

The highest output of the calcine reactor will not give the highest specific profit;

Any change of soluble iron from the ferrous to the ferric status will decrease the profit margin;

The ferric/ferrous ratio should be as small as is practically possible.

The soluble loss, which occurs in the sand slime washing area, can be predicted either by the use of regression function or through a model. An example of the model is shown in Fig. 5. With a high sand fraction the lowest soluble loss can be expected. However, this statement is only true if the rotoscoop-CCD wash distribution is selected correctly.

A model was developed for the CIX plant. The practical use of this model is limited because the loading isotherm of the resin must be defined by laboratory tests for different uranium, ferric and silica contents in the pregnant solution. A quick answer to the question of what the barren tenor will be under a certain set of conditions can be given by a regression function. As shown in Fig. 6 the uranium tenor has the strongest influence on the barren values, followed by the ferric content and then the silica tenor within the normal operating range.

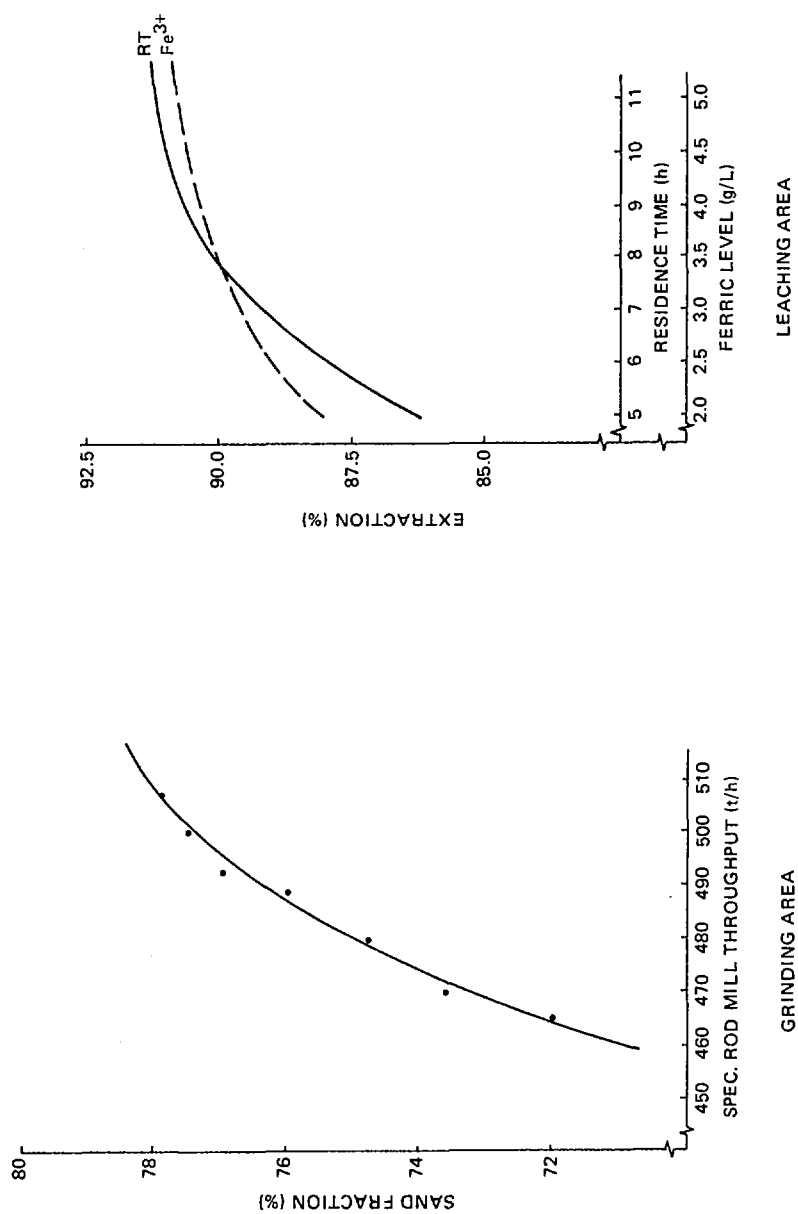


FIG. 3. Effect of residence time and ferric level on extraction (%).

FIG. 2. Relationship between specific rod mill throughput and sand fraction (%).

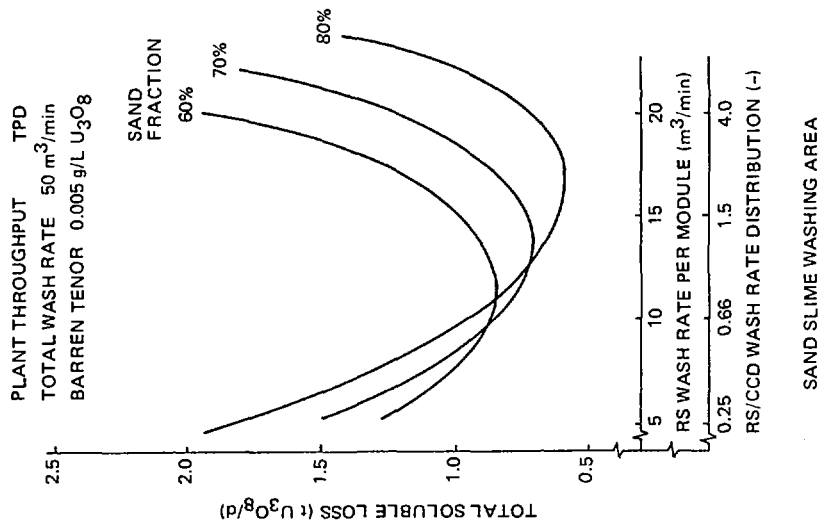


FIG. 5. Loss of soluble uranium as a function of sand fraction and wash rate.

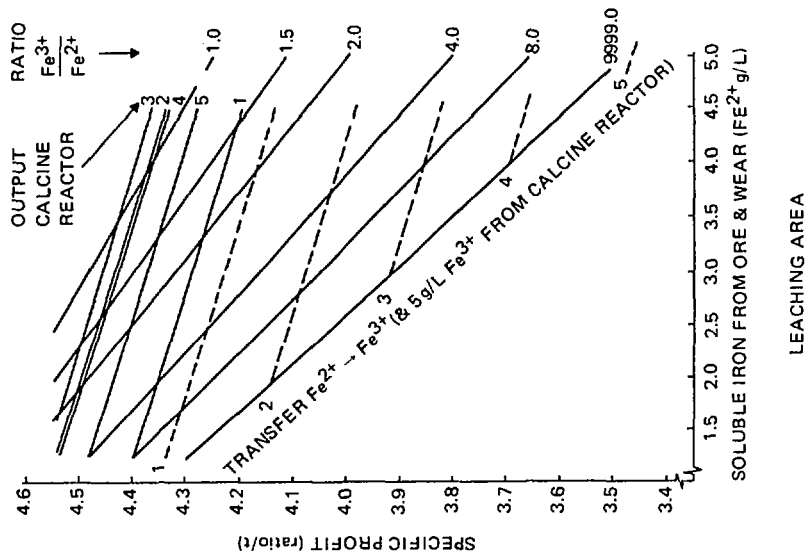


FIG. 4. Effect of soluble iron and ferric/ferrous ratio on the specific profit.

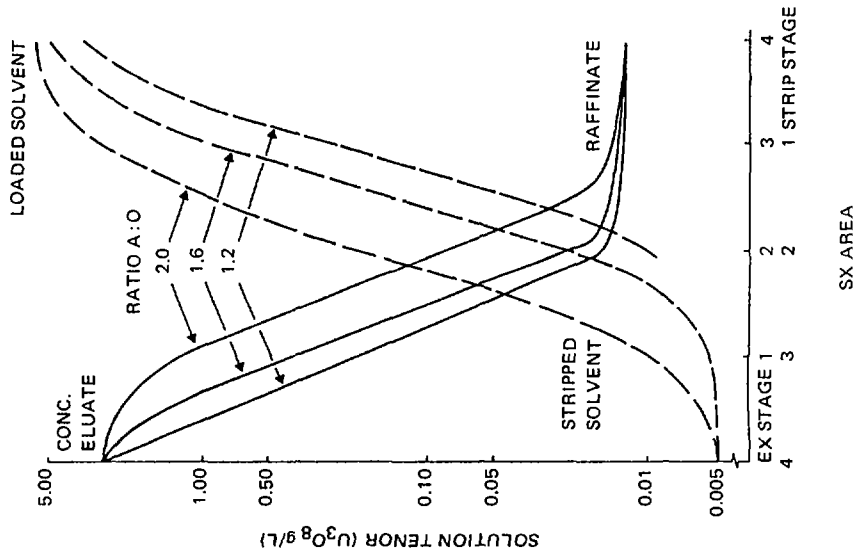


FIG. 7. Uranium content of the loaded solvent as a function of aqueous/organic ratio in a countercurrent solvent extraction mixer-settler system.

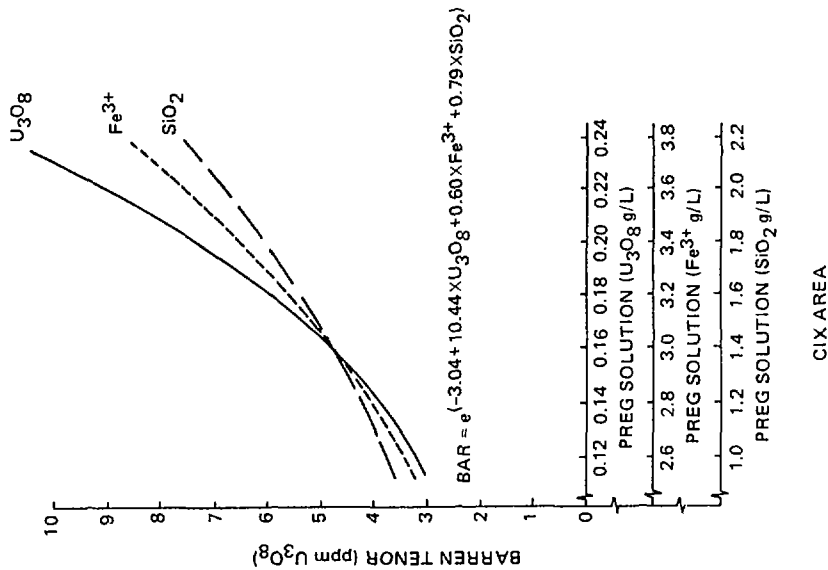


FIG. 6. Effect of uranium, ferric iron, and silica content of the pregnant solution on the uranium content of the barren liquor.

TABLE I. DECISION MATRIX

Operating areas	Decisions must be made for:		Areas affected					
			Grinding	Leaching	Washing	Continuous ion exchange (CIX) plant	Solvent extraction (SX) plant	Product recovery plant
Open pit	Mine grade	MG	$GS = f(R. \text{ type})$	$Ext = f(MG, MC, R. \text{ type})$	$SSD = f(R. \text{ type})$	$Bar = f(MG, R. \text{ type})$		
	Mine calc	MC		$Acid = f(MC, R. \text{ type})$	$FLOC = f(R. \text{ type})$	-	-	-
	Rock types	R. type		$MnO_2 = f(R. \text{ type})$				
Grinding	Plant throughput	TPD	$GS = f(TPD)$	$RT = f(TPD)$ $Ext = f(GS)$	$SSD = f(TPD)$ $SSD = f(GS)$	-	-	-
Leaching	Tanks of line	TOL		$RT = f(TOL)$		$Pre = f(Ext)$		
	Ferric level	Fe^{3+}		$Ext = f(RT, Fe^{3+})$		$Bar = f(Pre, Fe^{3+})$		
	Ferrous level	Fe^{2+}	-	$Acid = f(Fe^{3+}, Fe^{2+}, RT)$	-		-	-
	Free acid t.	H_2SO_4		$MnO_2 = f(Fe^{3+}, Fe^{2+}, RT)$				
Washing	Wash rate	WRA			$SLOS = f(WRA, WDI, SSD)$	$Pre = f(WRA, WDI)$	-	-
	Wash ratio	WDI	-	-		Bar		
CIX plant	No. elutions	NOE	-	-	-	$Bar = f(NOE)$ $CEL = f(NOE)$	$Conc. Liq. T. = f(CEL)$	-
SX plant	A/O rate	AOR	-	-	-	-	$Conc. Liq. T. = f(AOR)$ $RAF = f(AOR)$	-
Product recovery plant	Conditions for precip. (h, C°, etc.)	CPR	-	-	-	-	-	FP = (CPR)
Comments			Grain size: GS	Extraction: Ext S. Conc. Acid: Acid S. Conc. MnO_2 : MnO_2 Retention-T: RT	Sand slime: SSD S. Con. FLOC: FLOC Soluble L: SLOS	Barren T.: Bar Preg. T.: Pre C. Eluate T. CEL	Conc. Liq. T.: Conc. Liq. Raff T.: RAF	Final product: FP

There is also a model for the SX plant (see Fig. 7). Under a given strength for the concentrated eluate of 3.0 g/L and a stripped solvent tenor of 5 ppm, one can expect a raffinate content of 13 ppm. Depending on the ratio of the aqueous and organic flow rates the loaded solvent tenor varies between 4 and 6 g/L.

5. OPTIMIZATION OF THE OVERALL PROCESS

With the introduction of the computer system it is now possible to optimize the entire process instead of optimizing the operation of each section. Now we are dealing with a decision matrix as shown in Table I.

Two examples of the new system are: finer grind results in a higher uranium extraction especially for the coarser fractions, and higher ferric values also result in higher uranium extraction. However, the favourable effect of ferric in the leach process has a reverse effect on the CIX process in terms of higher barrenes and subsequent higher washing losses.

It is possible, nevertheless, in some areas of the metallurgical process optimization that higher operating costs would be incurred in order to reach the target for uranium production.

The computer system, which was developed in 1983, covers the mass and volumetric balances, the analytical assays for five chemical elements, the costs for water, electrical power and the major consumables, such as sulphuric acid, manganese ore, flocculant, resin, sodium, ammonia, solvent, alamine diesoline and product drums.

The programs for the computer system are written in standard FORTRAN IV. The system consists of one main program, 42 subroutines and 62 function calls. Through the extensive use of the overlay technique the total program size could be reduced to 99 648 words, i.e. 797 184 bytes.

The metallurgical division now has a scientific tool for planning tasks, for actual comparison with plant results and for strategic investigations and decision making.

ACKNOWLEDGEMENT

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REFERENCES

- [1] BERNING, I., COOKE, R., HIEMSTRA, S.A., HOFFMAN, U., The Rössing uranium deposit, Namibia, *Econ. Geol.* **71** (1976) 351-368.
- [2] Rössing uranium, *Min. Mag.* (Nov. 1978) 466-481.

- [3] WHYLLIE, R.J.M., Rössing uranium reaches rated production, *World Min.* (Jan. 1979) 43-52; (Feb. 1979) 52-58.
- [4] KESLER, S.B., FAHRBACH, D.O.E., "Uranium extraction at Rössing" presented at 12th Ann. Meeting CIM Metallurgical Society, Toronto, 29 Aug.-2 Sep. 1982.
- [5] KESLER, S.B., KNOEKJE, L., "The introduction of a central process system at Rössing", presented at XIVth International Mineral Processing Congress, Cannes, Jun. 1985.

IN-PLACE LEACHING OF URANIUM AT DENISON MINES LIMITED

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Abstract

IN-PLACE LEACHING OF URANIUM AT DENISON MINES LIMITED.

In 1984, Denison Mines Limited initiated a major demonstration of bacterially assisted in-place underground leaching of uranium ore at their Elliot Lake mine. The project involved identification of the test stopes, establishment of an underground laboratory, preparation of the test stopes, including bulk heading, design and installation of pumping and control systems, optimization of microbial environment, startup and operations. The mine configuration and available mine water enhanced the feasibility of the project. Problems of low temperature, radon levels, competitive microorganisms, ore fragmentation and water management were addressed. The paper presents the background, development and current status of the project with a preliminary assessment of the long term potential.

1. INTRODUCTION

Denison Mines Limited has been producing uranium from ores at Elliot Lake, Ontario, since the late 1950s. Over that period of time there have been a number of modifications to the milling operation from crushing through concentrate drying and packing [1]. There have been several mill expansions, the latest taking place in 1982 to reach the current mill capacity of 13 600 t/d [2] of an ore containing approximately 1 kg U/t.

Denison has used underground room and pillar techniques as its standard mining practice. The sedimentary quartz pebble conglomerate occurs in relatively flat lying beds. Long-hole drilling and blasting are used. Ore is removed using trackless machinery and hoisted to the surface for uranium recovery.

The Denison mill uses semi-autogenous grinding and pebble mills (Fig. 1). Ore at 50% - 200 mesh is leached in sulphuric acid with sodium chlorate as an oxidant in air agitated Pachucas. Drum filters remove the leached residue. Moving bed ion exchange (IX) has been augmented by a fixed bed system in the recent expansion. Nitrate elution is followed by ammonia precipitation. Yellow cake is dried and packed in steel drums.

Over the years Denison has used mine water as mill process water. This mine water has been of particular interest because it contains about 100 ppm uranium. This relatively high uranium content arises from bacterial oxidation of pyrite (5-7% in Denison ore). This reaction generates acid and ferric ion in solution which leaches uranium from the conglomerate. This phenomenon was first identified and explained in the early 1960s [3]. With volumes of up to 5000 L/min, this mine water added over 100 t uranium to mill production each year. On occasion, completely mined stopes were washed down with mine water to promote this phenomenon, but no concerted or carefully engineered approach was used.

In early 1980s when the highly optimistic projections for increasing uranium demand were unfulfilled, uranium prices dropped precipitously. Denison sought process economies via a number of measures. One such measure was to consider increasing the concentration of uranium in mine water by enhanced in-place leaching.

The concept of well engineered in-place leaching had already been well demonstrated at the full scale operations of Agnew Lake Mines Ltd [4]. In 1976, that mine began development using bacterially assisted leaching exclusively as the uranium recovery process. Leaching was performed on broken ore underground as well as in surface heaps of the 'swell' created during blasting. This operation ran for five years ceasing production in 1983 in the face of declining ore grades (from 0.043 to 0.028% uranium), poor underground leaching recoveries and reduced uranium prices. Principal constraints to high uranium recoveries were the steep incline of the ore pockets and the large size of the broken ore. Nevertheless, the more carefully controlled size in the flat lying surface gave excellent results.

2. DENISON MINES

The quartz pebble conglomerate at Denison lies in two major reefs. The lower reef which is about 10 m in thickness has been mined for over 25 years. It has a grade of between 0.75 and 1.0 kg U/t. The relatively flat lying bed of

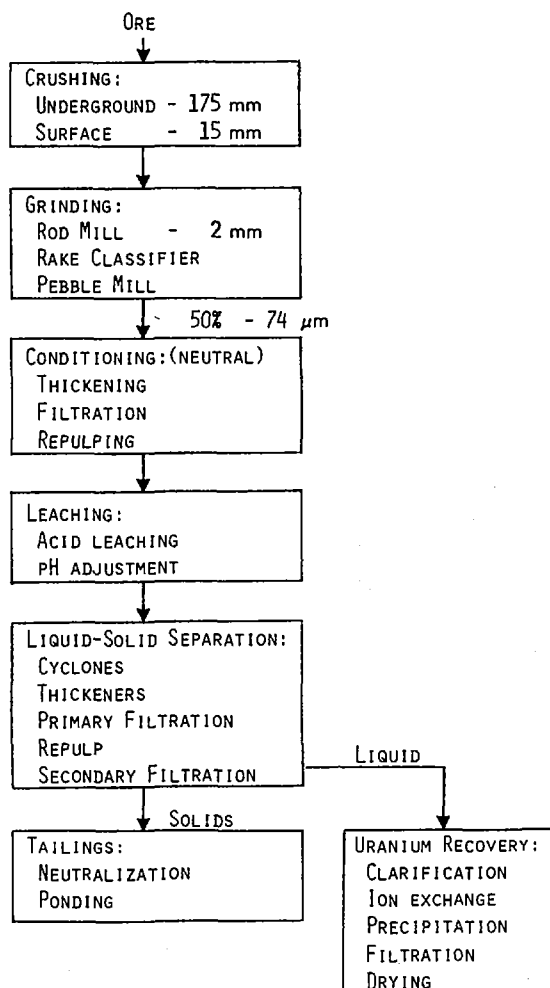


FIG. 1. Simplified Denison flow sheet.

ore has enabled the use of conventional mining techniques. Pillars amounting to about 20% of the ore volume have been left for roof support. This mine has produced over 50 million t of ore since its startup.

Operations at Denison have required some selective mining. A chloritic ore zone is avoided. Chloritic ore has a very high acid consumption and causes grave filtration problems. In addition, the upper reef zone of the mine is below cut-off grade at about 0.4 kg U/t. This 8 m thick zone parallels the lower reef and is separated from the lower zone by about 50 cm of barren rock. The upper reef contains an estimated 50 million t of material. Chloritic ore represents perhaps 15 000 t uranium at more than 1 kg/t.

The Denison mine accumulates water underground at a rate of about 5 000 L/min. This water comes from mine machinery such as drills, wetting-down ore to prevent dusting, condensation from ventilation air in summer, leakage from the surface, and drainage from backfill operations. Water is pumped constantly to the surface with the daily amount approaching 7 500 t in the summer.

Active bacterial oxidation does take place underground at Denison as is demonstrated by the high uranium content of mine water. All of these factors led Denison to consider the design of a major project to increase uranium recovery by in-place underground leaching.

3. DEVELOPMENT PLANS

In-place leaching had been conducted as a small and interesting adjunct to mining operations but the recovery of uranium was not highly significant. The process worked when conditions were right and helped gain value from waste materials underground. However, there had never been any major strategy or design in the process.

In 1983, Denison recognized the need to further reduce costs. Uranium in mine water was virtually free since it involved no mining cost, no additional pumping costs and could be treated in the existing mill. Perhaps this could be expanded. However, to do so required more broken ore underground. Denison therefore embarked on a project to design, develop and operate a demonstration of in-place leaching of ore broken specifically for the purpose. This demonstration would be used to provide criteria for full scale design, costs for development and operation and identification of any major problems.

The objectives of the project were specifically:

- (a) To verify the optimal underground blasting procedures to fragment ore into sufficiently small size for in-place leaching;
- (b) To test the best way of leaching the fragmented ore: trickling, percolation or flooding;
- (c) To develop the proper methods for protecting workers from the higher level of radon emanating from the large volume of fragmented ore;
- (d) To perform bacteriological studies in universities to understand and optimize nutrient needs and temperature behaviour of uranium leaching bacteria.

3.1. Fragmentation

The Denison mine provided an ideal situation for in-place leaching. The lower reef had been removed as ore for surface treatment. Drilling could be

carried out from the lower reef and the upper reef material could be blasted into the existing cavity. There was no need for any removal of material from the stopes.

Denison established four test stopes. These stopes were approximately 80 m long with a slope of about 10° and 25 m wide with some central pillars for roof support. Each represented about 35 000 t of upper reef ore.

Three drilling patterns were established: 4 ft X 3 ft (1.22 m X 0.91 m)¹, 4 ft X 2 ft and 3 ft X 3 ft. These had been established based on experience and a predictive computer model developed by the explosives manufacturer, Canadian Industries Limited (CIL). A rubble size of minus 13 in (33 cm) was sought with the bulk of the material in finer (minus 5 cm) sizes. The three blasting patterns were to be tested in the large stopes to assess their effects. The size was to be analysed using a CIL technique called Blaspha, an analysis of photographs of broken ore. This would be correlated with bulk samples taken with a scoop-tram mining machine. This sample would also be run in an underground laboratory column test.

3.2. Leaching

Laboratory studies at the British Columbia Research Council and at Denison had indicated that the best leaching results were obtained in a trickle or percolation leaching system similar to surface heap leaching. In this system the mine water would be sprayed or trickled over the surface of the ore pile to keep the ore wet but not flooded. However, this approach presented a problem; the roof of the stope would require support so that men could work in the stope putting in the piping and sprays. Roof support with long steel rods is expensive.

As an alternative, Denison decided to evaluate flood leaching as well. Flood leaching required that the lower end of the stope be completely sealed with a waterproof bulkhead. Mine water would then completely fill the stope containing the broken ore. Once filled, the stope would be allowed to drain. When empty, a short rest period in air would encourage bacterial growth on the wetted ore surfaces. The flooding cycle would then be repeated until the discharge solution was below economic levels.

3.3. Radon emanation

One concern about the in-place leaching was that the presence of large quantities of broken ore underground could raise the radon gas levels above the acceptable working levels. With a mining rate of about 10 000 t/d the four stopes would represent the equivalent of 15 days of mining operations. Furthermore, if the demonstration was successful, Denison would require some

¹ 1 ft = 0.3048 m.

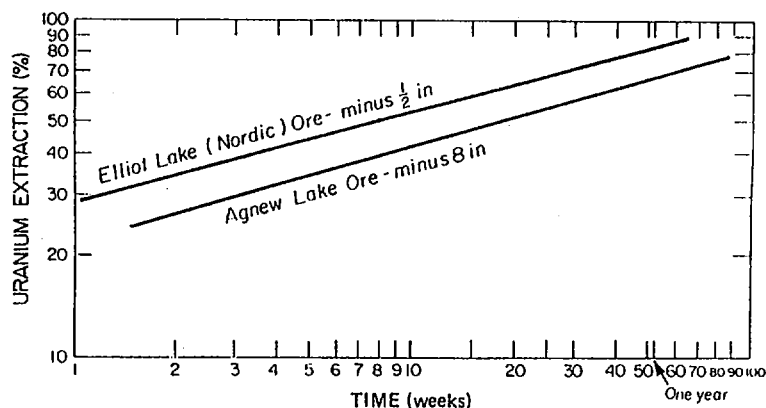


FIG. 2. Uranium extraction by iron oxidizing bacteria.

basis for determining the contribution to radon levels as new stopes were added. This would provide ventilation design data.

Radon levels would be checked in each stope during the progress of development and operation. Careful comparison would be made between the trickle leach and flood leach stopes.

3.4. Bacteriological studies

Denison had relied exclusively on the indigenous strains of the bacterium: *Thiobacillus ferrooxidans* for leaching of residual underground wastes. In developing the demonstration project, it was decided to allocate a portion of the funds to university research on developing an improved understanding of the bacteria and to optimize nutrition and temperature response of the microbes.

Two university groups were identified: Dalhousie University (R.G.L. McCready) in Halifax, Nova Scotia, and Laurentian University (G. Ferroni) in Sudbury, Ontario. Dalhousie undertook to optimize the nutritional requirements for indigenous *Thiobacillus ferrooxidans*. Laurentian set out to isolate from the underground environment and elsewhere different strains of this same bacterium in order to examine the effect of temperature on growth rates, and to study bioleaching rates over the range of 6 to 25°C.

4. PERFORMANCE TO DATE

The project began in May 1984. The stope development work, laboratory tests, and university studies were to be completed in 12 months. However, the true test will be the overall recovery of uranium from these test stopes over a 12 to 24 month leaching period. Denison based its projections on 70% recovery

TABLE I. SCREEN AND CHEMICAL ANALYSIS—DENISON STOPE
(0.91 m × 1.22 m blasting pattern)

Size fraction (mm)	wt (kg)	wt%	U ₃ O ₈ (%)	U ₃ O ₈ (g)	U ₃ O ₈ distribution
+101.6	489	34.50	0.033	161.1	20.7
+50.8	187	13.22	0.039	72.9	9.4
+25.4	186	13.16	0.055	102.4	13.2
+12.7	158	11.14	0.066	104.0	13.4
+6.4	117	8.28	0.079	92.5	11.9
−6.4	279	19.70	0.088	245.3	31.4
Total	1416	100	0.055	778.2	100

in 12 months based on the Ottawa curve (Fig. 2) developed by Mines Branch, Canada Centre for Mineral and Energy Technology in the 1960s [5]. The following provides a brief summary of results to date.

4.1. Stope development

Stope development took approximately 6 months per stope. These were scheduled 2 months apart. Three stopes drilled and blasted on 4 ft × 3 ft, 3 ft × 3 ft and 4 ft × 2 ft were prepared for flood leaching. Two types of drill rig were chosen: air operated bar and arm drills, and a new electric hydraulic 'uppers' jumbo. Drilling was in fact completed one month ahead of schedule.

Blasting of stopes began in October 1984. Small blasts of 2000 t each were designed to allow full testing of the fragmentation parameters. However, this procedure was abandoned in the interest of safety. Blasts as large as 15 000 t were carried out with excellent results using proper timing procedures.

Photographs were taken of all blasts and sent to CIL for analysis. Unfortunately loose fractured rock from the roof fell on top of the broken ore and dominated the photographs. The analyses were not as satisfactory as had been anticipated. Broken ore was scooped out of these stopes for screen analysis. Such an analysis is shown in Table I. However, with a stope 250 ft long, a single scoop from one end is not necessarily representative. It would appear that the actual size of the rubblized ore may never be adequately determined unless a leached-out stope is completely sampled after completion of the project.

Bulkheads were built to seal up the lower end of the flood leaching stopes. This lower end was purposely left narrow to reduce the size of the bulkhead. Each stope had two bulkheads capable of withstanding static heads of up to

TABLE II. LABORATORY COLUMN RESULTS

Column	Blasting pattern	Treatment	Time elapsed (d)	Extraction (%)
1	0.91 m X 1.22 m	Control	187	22.1
2	0.91 m X 1.22 m	Inoculated	187	37.7
3	0.91 m X 0.91 m	Inoculated	209	51.0
4	0.91 m X 0.91 m	Control	209	38.7

100 ft (30 m). These are reinforced concrete bulkheads set into a channel cut into the rock at the end of the stope. They are about 40 cm thick with the necessary plumbing fittings for sampling, emptying the stope, and aeration. Aeration pipes had been laid on the floor of the stope prior to blasting.

The lower end of the trickle leach stope was fitted with a dam about 1.5 m high to provide a small reservoir at the discharge end for surge capacity, sampling and safety. Rock bolts, of about 1.5 m in length, were installed in the roof of the trickle leach stope to prevent rock falls. A network of hoses was laid on the surface of the broken ore and sprays were installed in a regular pattern.

4.2. Laboratory studies

Denison established an underground laboratory in which samples could be prepared, simple analyses could be performed, and as a site for column tests. Six columns about 60 cm dia and 3 m high were set up to test various leach parameters. Two columns were set up using ore from the 3 ft X 3 ft and 4 ft X 3 ft blast patterns as controls. Two other columns were set up with ore from these blasting patterns with optimized nutritional supplements in one and massive inoculation of *Thiobacillus* in the other. These tests were started in February 1985. Significant leaching rates were established as shown in Table II. The optimization of nutrients, massive inoculation and closer blasting pattern all gave significant improvements.

This underground laboratory work was the on-site testing area for parameters established based on university research. Indigenous mine drainage was being used as the leach liquor in the Denison mine. Chemical analyses of several mine water samples were made to determine the concentration of normally required nutrients for *Thiobacillus ferrooxidans*. These analyses are shown in Table III. The high concentration of nitrogen compounds is due to the use of ammonium nitrate based explosives in stope development.

TABLE III. BACTERIAL NUTRIENT CONTENT OF MINE WATER SAMPLES

Sample No.	PO_4^{3+}	Mg^{2+}	NH_4^+	NO_2^-	NO_3^-
30D + 32N discharge	3	15	50	7	170
46078 discharge	9	12	75	11	350
46076 discharge	4	10	50	7	190
Total mine H_2O	1	15	30	3	170
32881 sump discharge	3	7	50	6	162
26305 sump discharge	1	8	20	2	95
Average	3.5	11.2	45.8	6	189.5

TABLE IV. COMPOSITION OF THE VARIOUS GROWTH MEDIA TESTED (ppm)

Medium	PO_4^{3+}	NH_4^+	Mg^{2+}	FeSO_4
A	10	10	10	18.000
B	10	10	2	18.000
C	10	20	1	18.000
D	10	10	0.5	18.000
E	10	10	1	18.000
9 K standard	235	820	95	24.000

Recent studies [6] have indicated that the growth medium (called 9 K) used in laboratory culturing of *Thiobacillus* contains excessive concentrations of phosphate, magnesium and ammonia. Work on leaching of pyrite from coal had shown that *T. ferrooxidans* would be active in media considerably more dilute than considered optimum under laboratory conditions. Bacteria were very active in the Denison mine under conditions as shown in Table III. Studies were undertaken to determine whether nutritional supplements would enhance bacterial growth and the rate of uranium solubilization.

Work on coal had indicated that 0.1M phosphate was required for *T. ferrooxidans* growth. Therefore a series of media was tested as outlined in Table IV. For comparison the compositions of 9 K (laboratory) medium is shown.

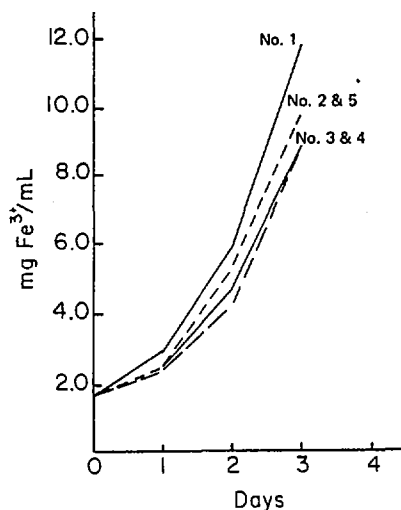


FIG. 3. Growth of Denison Mines isolates at 28°C on low phosphate medium [6].

Growth and iron oxidation rates were compared and medium A was found to be optimum. Five Denison mine water isolates were then grown in this medium with iron oxidation rates as shown in Fig. 3. It appeared that all grew well in this medium. It appeared that the mine water at Denison was lacking only in phosphate with average levels of 3.5 ppm (Table III) versus recommended levels of 10 ppm. Phosphate supplementation was therefore recommended to Denison and applied in one of the underground test columns.

Having optimized nutritional requirements, work proceeded to determine whether some of the isolates from Denison had better cold tolerance (psychrophilic response) than others. Figure 4 shows the effect of temperature on the most active of the isolates. This culture was taken from an area adjacent to the ventilation shaft and therefore most subject to temperature variation. Of the five isolates and various other cultures tested, only three grew at temperatures less than 15°C. The indigenous strains of *Thiobacillus* obviously have much greater temperature tolerance than others. Nevertheless temperature has a profound effect on iron oxidation rates which will undoubtedly have an impact during winter months when underground temperatures are in the 10–12°C range.

Work at Laurentian University confirmed the psychrophilic nature of *Thiobacillus ferrooxidans*. It was established that the mean generation time roughly doubled for each 6°C rise in temperature from 6 to 25°C as shown in Table V. It would appear that temperatures below 12°C are generally unfavourable, but in the normal seasonal cycle the temperature range of 12 to 20°C should sustain an active indigenous population of bacteria. This temperature affects data and can be utilized in scheduling the rest periods between flood leaching cycles.

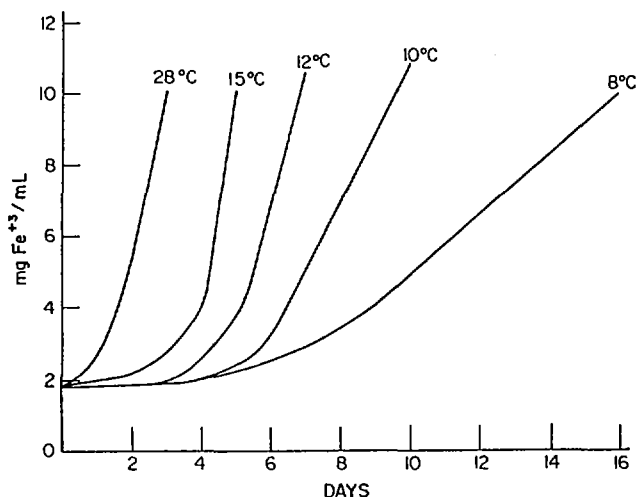


FIG. 4. Growth of *Thiobacillus ferrooxidans* for Elliot Lake No. 5 at various temperatures on low PO_4 medium at 150 rev/min (average of triplicate cultures): 10 ppm Mg and 10 ppm NH_4^+ at pH 2.3.

The concentration of bacterial cells in laboratory leaching tests demonstrated the effect of increasing population (Table VI). These results have led to larger scale tests using inoculation with cultures of *Thiobacillus ferrooxidans*.

4.3. Leaching operations

It is too early to judge the results on the stope leaching. Leaching was begun in May and June on the four test stopes. Preliminary results after about 100 days are shown in Table VII for the four stopes. In addition, a number of interesting facets of the operation have been indicated.

4.3.1. Water

The substantial increase in the total volume of water underground has led to concern about the water balance. There is no accurate measure of water entering the mine. Recent mine developments and increasing use of backfill (in slurry form) seem to have increased the influx of water. Sump capacity is being taxed. Operation of the flooded stopes required adequate sump capacity. During the summer of 1985, stope flooding was delayed due to the demand for balancing the underground water influx with pump capacity.

Stope leakage caused difficulties. Leakage from a flooded stope into an adjacent parallel stope had to be checked by extensive grouting. One of the bulkheads was leaking around the perimeter and had to be repaired.

TABLE V. MEAN GENERATION TIMES (G) OF NATURAL ISOLATES FOR TEMPERATURE RANGE 35-2°C

Temperature (°C)	Number of experiments	G (h)	SD	SE	%E
35	3	19.4	1.41	0.8	4.1
30	4	11.5	2.05	1.0	9.1
25	5	11.7	2.58	0.7	6.0
18	66	22.7	1.59	0.6	2.7
12	35	43.7	6.41	2.4	5.5
6	20	103	29	13	12.6
2	5	247	46	21	8.5

TABLE VI. EFFECT OF INOCULUM SIZE ON AMOUNT OF U_3O_8 LEACHED AND ON RATE OF LEACHING

Inoculum size (mL)	Per cent leached	Rate (mg $U_3O_8 \cdot L^{-1} \cdot h^{-1}$)
0	45	0.9
0.1	48	0.8
1.0	51	1.4
10.0	62	1.9

Note: Each value is the average of two replicates.

The quality of leach solution presented some problems. Backfill drainage at about pH 10 was neutralizing water in the sump. This drainage was diverted to another sump.

Water temperatures even in May and June were considerably lower than anticipated. Winter temperatures of 12°C persisted even into June so that startup of the stopes was slow. Denison is considering using ion exchange barren from the surface mill to increase the temperature of the leach solution and so accelerate bacterial growth. However, there is some concern about potential contamination by other ions and organics.

TABLE VII. PRELIMINARY LEACH RESULTS FROM PRODUCTION STOPES

Block	Leach method	Comment	Time elapsed	Extraction (%)
46072/73	Flood	Inoculated	125	32.2
46074/75	Flood	Control	135	23.1

Denison has plans to expand the in-place leaching operation. However, pumping capacity is limited and new pumping capacity is expensive. Therefore underground concentration of the solutions from the current 300 ppm level to perhaps 1000–2000 ppm is being considered. Three options are being evaluated: ion exchange, reverse osmosis and biosorption. Ion exchange is preferred because Denison has considerable experience in this field. Reverse osmosis seems to be feasible using recently developed membranes. Biosorption does not seem to be sufficiently well developed to be a serious contender.

4.3.2. *Aeration*

The demand for oxygen for the biological oxidation of the pyrite is substantial. From experience in leaching waste underground, it would appear that this demand is easily met with ventilation air. However, in the flooded stopes air access is limited. Air lines have been installed under the ore but some have been crushed or broken. Denison therefore consulted one of the oxygen manufacturers to develop a comprehensive analysis of aeration demands.

In the underground laboratory, tests have been conducted in one column using forced aeration with oxygen enriched air. Aeration under pressure is being considered to increase the level of dissolved oxygen in the leach solutions.

4.3.3. *Monitoring*

Discharge solutions from the stopes are being monitored on a regular basis. Solutions are typically 300 ppm. However, it is very difficult to determine the true rate of leaching. The average analysis of the ore in the stope is based on geological and mine development drilling. Approximate grades are known but the ore is variable. The only measure of progress is the amount of uranium in solution. Neither the grade nor the volume of ore is accurately known.

Various strategies are in place to use better predictive techniques. Data from laboratory and column leach tests are being collected. Mineralogical analyses of leached ore particles are providing insights into leaching rates and

TABLE VIII. FUTURE RESEARCH NEEDS

Optimize oxygen demand parameters and supply systems
Improve water balance utilizing sorption or membrane technologies
Recycle water to maintain higher temperatures
Optimize nutrient supply to leaching stopes
Test leaching of chloric ore block
Control fungal growth
Analyse leached stope upon completion for true particle size and residual uranium

depth of penetration. Denison is now working with Queen's University in Kingston, Ontario, on developing a computer model based on these data.

4.3.4. Biotechnology

During the course of the work at Denison, leaching on one of the older stopes seemed to stop. The presence of a yellow precipitate gave rise to the suspicion that it was jarosite. However, upon analysis little jarosite was detected. Samples sent to one of the authors (McCready) were soon determined to be fungi. These fungi were heterotrophs, i.e. required organic carbon for growth. Upon investigation this stope was found to have been an underground maintenance area and have high concentrations of petroleum products, the carbon source. Conditions in the stope remained compatible with the growth of *Thiobacillus*; however no uranium was present in the discharge water. Tests revealed that the fungi were in fact adsorbing the uranium. This phenomenon is now being investigated more fully. Uranium loadings of up to 12% on a dry weight basis have been achieved in the laboratory at pH 2.3 and with 15 g/L of iron. While Denison must avoid conditions under which these fungi will grow in the leaching stopes, it may well be a route to concentration of uranium.

5. CONCLUDING COMMENTS

This Denison in-place underground leaching demonstration has been set up at a cost of about Can. \$1 500 000. The National Research Council of Canada provided financial support for half of these costs under its Program for Industry/Laboratory Projects (PILP). This programme is designed to help industry acquire new technology from government and other public laboratories with the risks being shared by the Federal Government.

Denison Mines Limited senior management gave this project full and unqualified support. The potential for cost reduction in uranium production was so substantial that the project was given high priority.

The Federal Government and university researchers in turn gave full co-operation to the work. Excellent communication was essential.

The final results are as yet unknown but all parties are sufficiently encouraged to plan a new phase of mine development and research (see Table VIII). Research plans include further investigations of improved monitoring systems, aeration parameters, solution treatment options, impact of varying mineralogy, use of IX barren solution, characterization of other microorganisms such as fungi and mathematical modelling.

REFERENCES

- [1] LAROCQUE, E., PAKKALA, E., Current leaching and product recovery practice at Denison Mines Limited, CIM Bull. 72 804 (1979) 172-176.
- [2] PAKKALA, E., Denison Mines Limited: Mill Expansion, Rep. ERP/MSL 80-158 (TR), Department of Energy, Mines and Resources, Ottawa, Canada (1980).
- [3] HARRISON, V.F., GOW, W.A., IVARSON, K.C.L., Leaching of Uranium from Elliot Lake Ore in the Presence of Bacteria, Rep. EMA 66-1, Extraction Metallurgy Division, Department Mines and Technical Surveys, Ottawa (now Department of Energy, Mines and Resources, Ottawa, Canada) (1966).
- [4] HEYMAN, L., Metallurgical Aspects of the Operation at Agnew Lake Mines Limited, Rep. ERP/MSL 80-158(TR), Department of Energy, Mines and Resources, Ottawa, Canada (1980).
- [5] McCREEDY, H.H., HARRISON, V.F., GOW, W.A., A proposed method, using bacteria, for the continuous leaching of a uranium ore, CIM Bull. 63 682 (1969) 135-140.
- [6] McCREADY, R.G.L., WADDEN, D., MARCHBANK, A., Nutrient requirements for the in-place leaching of uranium by *Thiobacillus ferrooxidans*, Hydrometallurgy (1985).

DISCUSSION

W.E. FLÖTER: I understand that by using bacterial leaching you have increased the reserves at Denison by at least 100% at relatively low cost. You can make a profit with recoveries below 50%. I may add that one mistake we made with in-place leaching at Agnew Lake was to use too many explosives for mining. There was too much nitrogen in the solution and this nitrogen competes with the uranium in the ion exchange process and blocks the resin. The nitrogen content should be preferably below 3 g/L and in any case no greater than 6 g/L. Perhaps a new resin may tolerate higher nitrogen contents.

M.C. CAMPBELL: At Denison we instituted an extensive programme to minimize the amount of nitrates. We have overcome the nitrogen problem by using good housekeeping techniques with the explosives.

TECHNIQUES FOR HEAP AND IN SITU LEACH SIMULATION ON URANIUM ORES

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Abstract

TECHNIQUES FOR HEAP AND IN SITU LEACH SIMULATION ON URANIUM ORES.

Bureau of Mines sponsored research has developed techniques for simulation of heap and in situ leaching of uranium ores. The heap leach technique uses 2 ft dia (60.9 cm) columns with bed depths ranging from 8 ft to 18 ft (2.44 m to 5.49 m). Uranium ores as coarse as minus 4 in (101.6 mm) can be treated. The simulation technique for in situ leaching uses 4 in dia by 12 ft long (101.6 mm by 3.66 m) horizontal columns. The technique was developed to determine the effect of leaching variables such as solution composition on the bed permeability, uranium recovery, and post-leach aquifer restoration. References to detailed experimental data developed with these systems are included.

INTRODUCTION

During the late 1970s, the United States Department of the Interior, Bureau of Mines, sponsored extensive studies on heap and in situ leaching as part of its goal to develop technology for treating low grade uranium ores. The heap leaching studies were conducted under a contract agreement with the Colorado School of Mines Research Institute. The in situ leaching research was conducted under a similar contract agreement with the Westinghouse Research and Development Center.

The heap leaching project at the Colorado School of Mines Research Institute proceeded through six phases. The objective of the initial four phases was to develop fundamental and comparative data on the relationships between particle size, percolation rate, and leaching rate. The final two phases of the project included a field scale heap leach; the objective of this work was to establish correlating relationships between the pilot scale simulation techniques and field scale operations. Section 1 of this paper presents a description of the 2 ft dia pilot scale column system developed during the Bureau sponsored research¹. The operating procedures and techniques used during the heap leach simulation studies are also discussed.

¹ 1 ft = 0.3048 m.

The primary objective of the in situ leaching research conducted at the Westinghouse Research Development Center was to develop a more fundamental understanding of the chemical and physical phenomena occurring in uranium ore bodies during in situ leaching operations. During these studies, an in situ leach simulation technique was developed to determine the effect of leaching variables such as solution composition on the bed permeability, the uranium recovery, and post-leach restoration.

Laboratory simulation of in situ leaching can be an acceptable method of investigation when the ore body is represented by a geometry and packing which reasonably reproduce the hydraulics of in situ leaching. Advantages of this method include the ability to visually observe leaching, partially control some of the ore body characteristics, determine characteristics of the uncontrolled variables, and conduct reproducible experiments. In addition, the experiments take less time than field tests because of the smaller sample used.

Limitations of the laboratory simulation tests lie in the difficulty of obtaining a sample that represents the underground ore body in terms of permeability, uranium content, oxidation state, reaction surface area and mineralogy, and in understanding quantitatively the differences between the controlled test conditions and the non-ideal conditions encountered in the underground deposit. Although prediction of field behaviour from the results of laboratory simulation tests is qualitative, this technique does provide a common denominator for studying the behaviour of various types of ores when they are leached with different combinations of reagents.

A reproducible laboratory method was developed for the laboratory simulation of the leach chemistry and, to a limited degree, of the hydrology of the in situ leaching of uranium. The method was used to determine the effects of leaching variables upon permeability and uranium extraction from ores similar to those that have been leached in situ in Texas and Wyoming. The use of such a method can provide a more fundamental understanding of both the in situ leaching operation and associated environmental restoration of the post-leach ore body.

Section 2 of this paper describes the in situ simulation system and its operating procedures.

1. THE HEAP LEACH SIMULATION SYSTEM

The basic test unit developed during this study is a cylindrical column operated with closed loop circulation; multiples of this basic unit were used during the experimental studies. The ore charge is retained in each column by a perforated plate to provide for percolation type leaching contact. A tank placed under the column holds the solution inventory required for a leaching test and also serves for collection of column effluent. A diagram illustrating the piping plan and elevations for a single unit, prepared for conventional downflow circulation, is shown in Fig. 1.

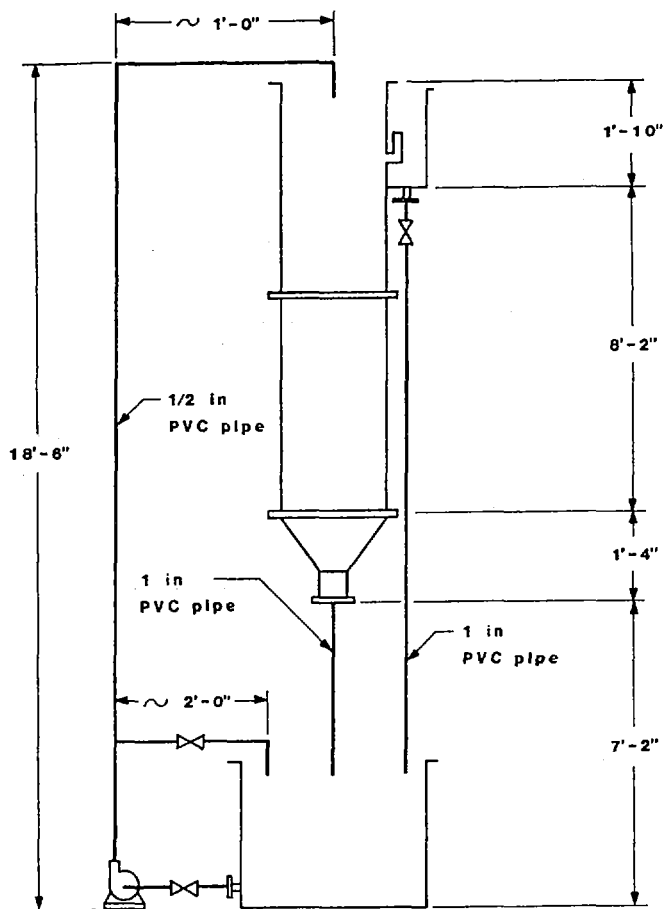
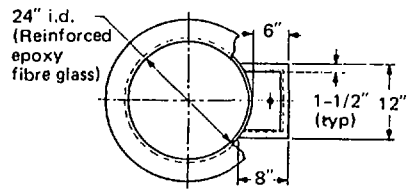
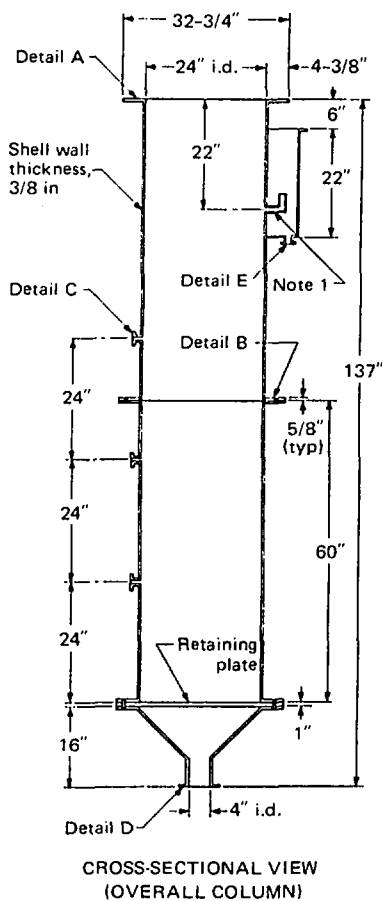


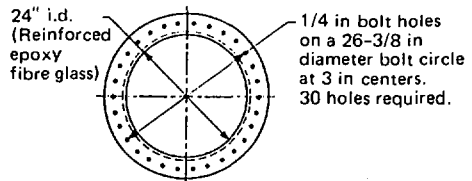
FIG. 1. Column piping plan and elevations.

Column construction details

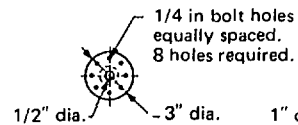
The basic structural component of the columns is a 2 ft dia, 5 ft long cylindrical section with flanged ends. Two sections are butted together and a 45° flanged cone attached to the bottom section. An assembled 10 ft column is equipped with three auxiliary liquid sampling ports mounted on one side of the unit and spaced at 2 ft intervals, and a solution level control box is mounted on the upper tube section. The ore charge is retained by a perforated plate located between the lower flange and the cone, as shown in Fig. 2. The cylindrical sections and cone are constructed of



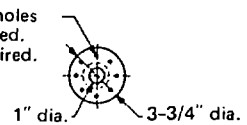
DETAIL A - COLUMN TOP VIEW WITH OVERFLOW BOX



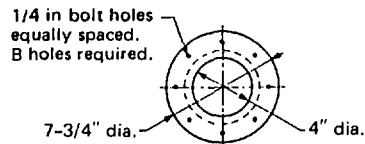
DETAIL B - MOUNTING FLANGE (COLUMN AND CONE)



DETAIL C - FRONT VIEW OF SAMPLING PORT FLANGE



DETAIL E - FRONT VIEW OF OVERFLOW OUTLET FLANGE



DETAIL D - FRONT VIEW OF CONE FLANGE

Notes:

1. Overflow box inlet is a 1 in female threaded L-shaped PVC pipe fitting. Attachment of a variable vertical extension is optional.
2. Gaskets are required on all mounting flanges.
3. Overflow outlet, sampling port outlet, and cone outlet flanges are 3/8 in thick.

FIG. 2. Detailed column design.

fibreglass with a 3/8 in wall thickness². The perforated retainer plate is 1 in thick fibreglass with 1/4 in holes drilled at approximately 1 – 1/2 in spacing. A one-layer covering of neoprene matting and three to four layers of vinyl screening resting on top of the perforated base plate retain the ore fines. Figure 3 shows the appearance of the perforated base plate and the neoprene matting on the supporting flange.

Auxiliary equipment for each column unit consists of a 400 gal fibreglass reservoir³, a 1/4 hp gear drive portable mixer⁴, a totally enclosed centrifugal pump, and associated polyvinyl chloride (PVC) piping and ball valves.

A steel framework supports the columns and allows ground level access to the base of the columns for discharging and handling leached residue. The framework is also equipped with a platform access to the top of the columns for ore loading (Fig. 4).

Heap leach simulation procedure

The heap leach simulation procedure consists of the following sequential steps: (1) ore preparation, (2) ore loading, (3) preliminary wetting, (4) leaching, (5) residue washing, and (6) recovery and analysis of solution and residue. Details of these steps are discussed below.

Ore preparation

Each ore tested during the experimental studies was spread on a concrete pad and turned intermittently to air dry over a 3 to 7 day period. The ore was then screened on a Ty-Rock⁵ double-deck screen using a 4 in top screen and a 1/2 in or 1 in bottom screen. All plus 4 in ore was broken manually to pass the screen. The separate screen fractions were drummed for future use in preparing head samples for chemical analysis and feed samples for the leaching column. Feed and head samples were reconstituted by combining proper weight ratios of blended coarse and fine fractions.

A 300 lb reconstituted ore sample was used to prepare a head sample for chemical analysis⁶. This sample was crushed to minus 1/4 in and blended; then a 25 lb sample was split off. The 25 lb sample was roll-crushed to minus 10 mesh, reblended, and further split to provide analytical head samples.

² 1 in = 25.4 mm.

³ 1 US gallon = 3.785 L.

⁴ 1 hp = 0.746 kW.

⁵ Reference to specific trade names or manufacturers does not imply endorsement by the Bureau of Mines.

⁶ 1 lb = 0.454 kg.

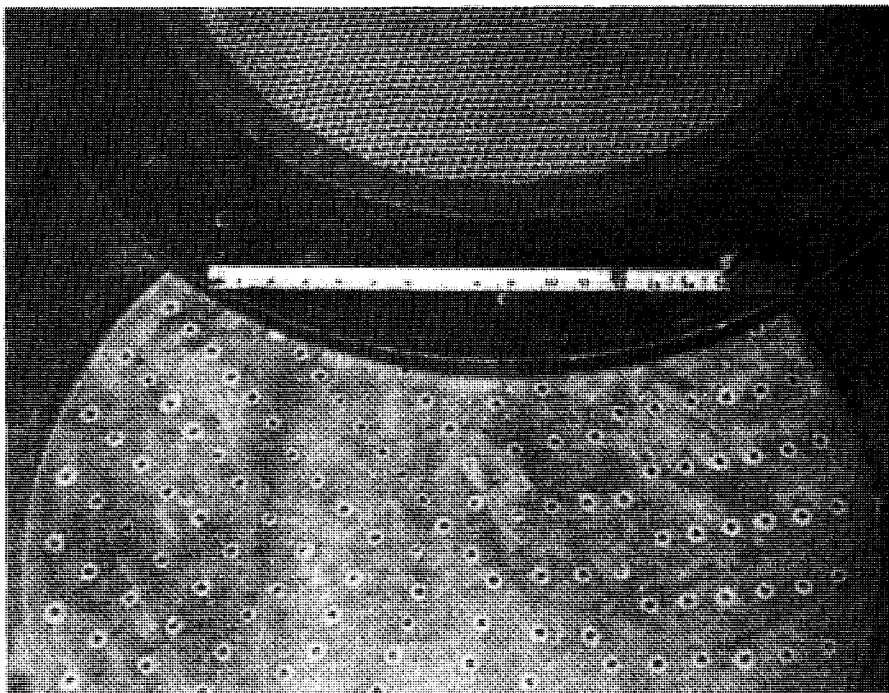


FIG. 3. Ore retainer plates (scale in inches).

Ore loading

Blended ore charges were loaded into the columns using 5 gal buckets with a line attached to the bottom for dumping. Full buckets were raised to the upper access level by a hydraulic platform in groups of four to six buckets, lowered by rope to a point near the top of the ore level, and dumped. An average 8 ft ore charge weighed 2200 to 2600 lb.

Preliminary wetting

Tap water was measured into 4 ft by 4 ft reservoir mixing tanks, with exact amounts based on an operating volume of 50% solids. Water was pumped to the top centre of the ore beds at flow rates ranging from 1.0 to 11.0 gal/h with the column effluent valves initially closed. The water was allowed to soak through the ore beds by means of gravity flow to obtain air displacement by backfill. However, in some cases air pockets formed near the bottom of the beds; these were released by opening the effluent valves during the last portion of the wetting period. By this procedure,

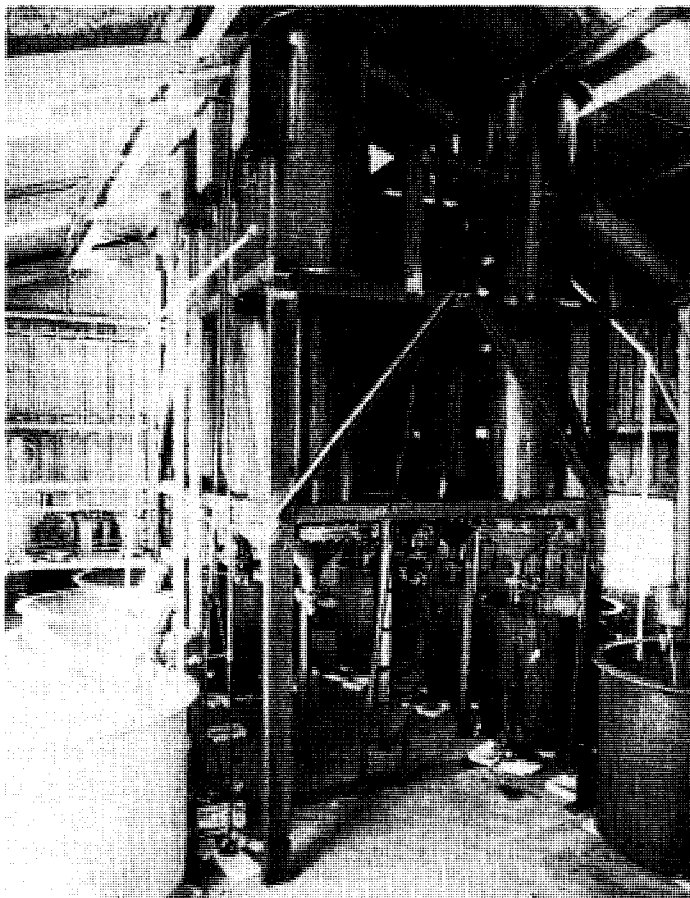


FIG. 4. Nine-column assembly used for test programme.

31 to 65 gal of water were used in a 4.5 to 70 h period. Following wetting, a 2 day period of downflow circulation was conducted to bring total water volume into a state of equilibrium with the ore charge. A constant head of approximately 12 in was maintained on top of the ore charge during circulation periods.

Leaching

As a continuation of wetting circulation, the leaching period was initiated by adding 93% H_2SO_4 to the reservoir mixing tank to a pH range of 1.0 to 1.2. When acid consuming components of the ore were neutralised, and the column effluent

reached a pH of 2.0 or lower, oxidant in the form of NaClO_3 was added to induce an electromotive force (EMF) of -425 to -450 mV in the mixing tank. These pH and EMF conditions were maintained throughout the leaching period. Frequent attention was required to maintain the desired pH and EMF during the first 5 to 10 days. Thereafter, daily checks and adjustments at 4 to 5 day intervals were adequate. Uranium solubilised during the test period was allowed to accumulate in the circulating solution inventory. A 50 to 120 day leaching period was determined by a point of equilibrium in the U_3O_8 concentration found in reservoir or column effluent samples.

Test units were monitored for column effluent flow rates, pH, and EMF throughout the procedure. Sampling for U_3O_8 concentration was also conducted throughout the test period with particular attention given to contents of the column effluent stream and contents of the reservoir mixing tank. Samples were analysed at 4 to 8 h intervals during acid circulation, and then at 4 to 5 day intervals.

Additional information as to flow rates and U_3O_8 leaching characteristics was obtained by sampling side ports of the column units.

Residue washing

After the leaching period, the columns were allowed to drain for approximately 2 days, then water washed. Tap water was pumped into the freeboard space on top of the ore charges and allowed to percolate through the beds. This step was repeated until 6 to 10 volumes of 50 gal each had been added. After washing, the columns were allowed to drain for approximately 5 days. All of the wash effluent and drainings were collected with the pregnant solution, measured, and analysed for use in the metallurgical balance.

Recovery and analysis of solution and residue

Residue was recovered by disassembling and removing the cone base and retainer plate. A fork-lift was used to support the cone while the bolts securing the cone flange to the lower column flange were removed. The forks were lowered slowly. With some types of ore, residue plasticity provided for a clean exit of solids as the cone and retainer plate were lowered. Residues allowed to drain for extensive periods were not self-discharging and required additional effort for removal.

Discharged residues were spread on a plastic sheet, sectioned into a grid pattern, and sampled as required for moisture determination. Approximately one-fourth splits were taken from the grid pattern and air dried on a concrete pad. The splits were crushed and resplit as required to provide an analytical sample for U_3O_8 . Screen assay tests were also conducted on the residues to determine mass and U_3O_8 distribution characteristics.

TABLE I. ORES USED FOR HEAP LEACH SIMULATONS

Source	Location	Uranium company	Head analysis, U ₃ O ₈ %
Golden Goose Mine	Crooks Gap, WY	Western Nuclear, Inc.	0.048
Bear Creek Mine	Bear Creek, WY	Rocky Mountain Energy Company	0.029
Jackpile Mine	Laguna, NM	Anaconda Company	0.064
Sunday Mine	Uravan, CO	Union Carbide Nuclear Company	0.034

Experimental data references and conclusions

The initial heap leach simulations were conducted on uranium ores as shown in Table I.

Detailed experimental data from the leach tests on these ores are described in two reports⁷ [1, 2].

Final phases in developing and proving the small scale heap leach testing technique were restricted to work with the Bear Creek ore. Two tests were conducted with 8 ft ore beds, one test was conducted with an 18 ft ore bed, and one field test was conducted by leaching an 18 ft heap located near Bear Creek Uranium Company mill in Wyoming.

Operating information and experimental data from the tests on the Bear Creek ore are presented in three reports⁷ [3-5].

All ore charges used in the Phase V studies were prepared and handled in the same manner as described earlier in this paper. The 18 ft ore bed was placed in a 20 ft column prepared by joining four of the 5 ft tube sections used in early studies. Approximately 2500 short tons of Bear Creek ore were shaped into an 18 ft heap⁸. Essentially, the same procedure was used for leaching, washing, and residue examination as described earlier in this paper, except for substitution of Bear Creek mill tailings pond water instead of tap water in all but one of the small column tests.

The following conclusions and comparisons for predicting the behaviour of a large scale heap leach from the 2 ft dia column leach results were developed during the Phase V and Phase VI studies:

- (1) When solution depths above the ore bed are roughly equal, percolation rates will be similar for each scale of test.

⁷ Research carried out with the support of the Bureau of Mines under contract No. 0252022.

⁸ 1 short ton = 0.907 t.

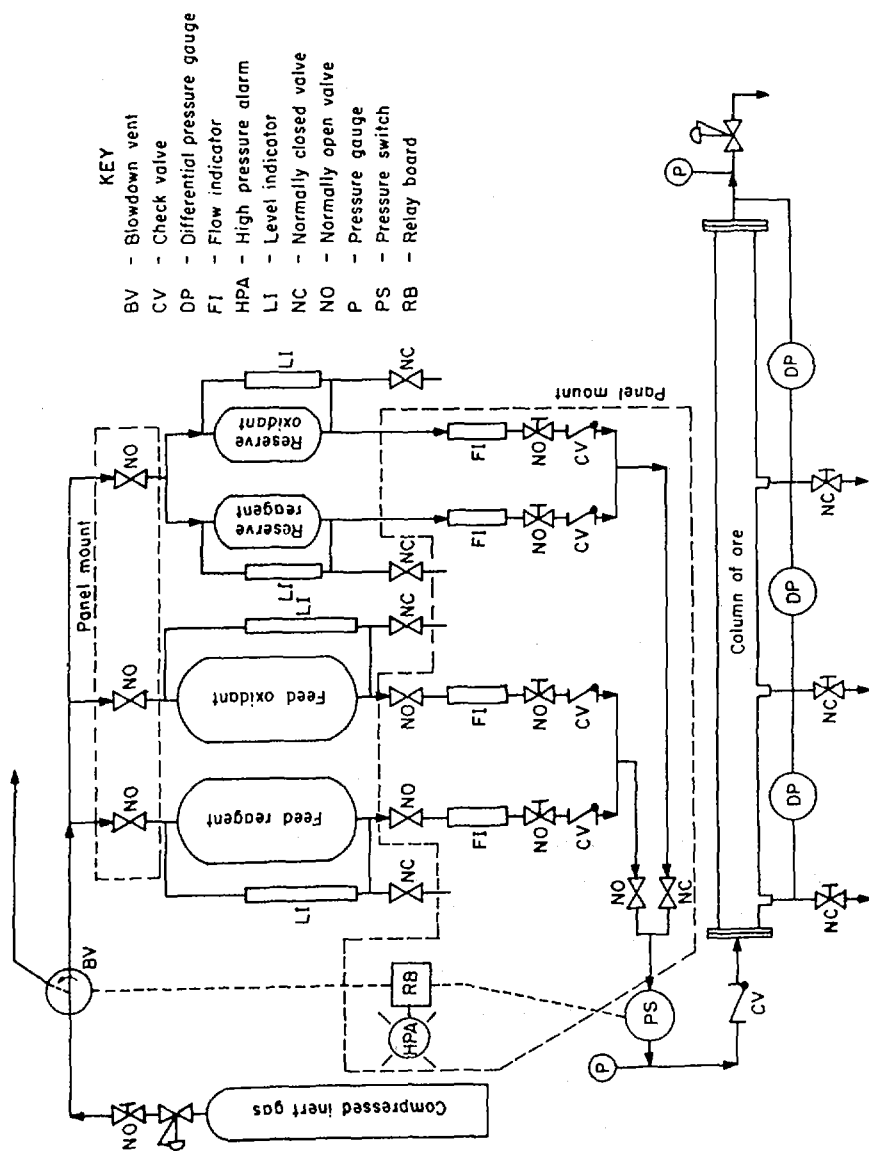


FIG. 5. Schematic diagram of in situ leach system.

- (2) At essentially equal flow rates, leaching time for an 18 ft ore bed will be approximately 20% greater than that required for an 8 ft bed.
- (3) With leaching time equalized, uranium extraction results will be approximately the same for each scale of test.
- (4) Reagent requirements for larger scale operations will be equal to or lower than those for the smaller scale test.
- (5) Channelling or areas of insufficient acid wetting will either not occur or not cause problems in either scale of test.
- (6) Lateral solution flow patterns cannot be predicted by small scale column testing because of confinement by the size of the column.

2. THE IN SITU SIMULATION SYSTEM

The principal objective of the system design was to arrive at a reliable and safe hardware system to simulate the in situ leaching of underground ore bodies for such aspects as leach flow characteristics, ore body characteristics, and leach conditions. Therefore, the design had to allow for ore geometry selection, sufficient sampling, and visual observation.

Of the various possible alternate sample ore geometries, cylindrical shape is preferred because the simulation results along the column, such as concentration and pressure profiles, can be applied to a given stream line in the field. Since theoretical models can predict the flow lines for an ore bed, the one-dimensional simulation could be integrated over a flow line network to predict results for a given ore field. Furthermore, a cylindrical geometry is easier to design, construct and pack.

In the leaching of an ore sample, it is important that no movement or reorientation of the packing be caused by the nature of the feed system. For this reason, a non-pulsing feed system is essential. Although feed systems using pumps can be designed, a pressurized system was chosen because it is simpler and more reliable. Also, a pressurized system minimizes the loss of volatile reagents from the feed solution.

Visibility of the ore sample during leach tests could be of assistance in understanding the in situ leaching process. It allows an observer to detect processes occurring such as channelling, colour changes indicative of oxidation front movement, ore compression and precipitation of compounds. A glass column was thus chosen to contain the ore.

A schematic diagram of the in situ simulation system is given in Fig. 5. The system consists of a pressurized feed section, from which the leach solution is fed to the ore column through flow metering and monitoring devices. The pregnant leach solution is collected in large tanks. A complete list of components and suppliers is given in Appendix I.

Leach solution feed section

A pressurized feed system is mechanically simple and reliable. Pressurizing a solution with gas, however, does increase the dissolved gas content of the solution. This could cause gas phase nucleation downstream in the ore packing as the static pressure decreases; this is particularly true of inert gases such as argon and nitrogen. The solubility of gas, however, can be exploited in tests with gaseous reagents such as oxygen. In those tests using hydrogen peroxide as an oxidant, a relatively impermeable rubber membrane separating the gas and solution phases is placed in the feed tanks in order to prevent an increase in the dissolved gas content of the solutions.

The feed section consists of two 16 gal, type 316 stainless steel PVC lined high pressure tanks, one for the leach reagent, the other for the oxidant solution, and two similar reserve tanks of 2 gal capacity each. Two primary tanks are required in order to have separate oxidant and leach feeds so that the oxidant dosage may be controlled and, particularly, when using hydrogen peroxide, to avoid premature oxidant decomposition in the feed tank. These tanks were custom fabricated to accept PVC liners. The selection of liners and the internal dimensions of the tanks were based on the commercial availability of the replacement liners. The tanks were hydraulic pressure tested for 180 psig and certified under the American Society of Mechanical Engineers (ASME) code for a maximum allowable working pressure of 120 psig⁹. The fabrication details for the primary and reserve tanks are given in Figs II.1 and II.2 of Appendix II. The reserve tanks serve as feed tanks while the primary tank is being recharged and can allow operation of the system for approximately 6 h.

Gas shut-off toggle valves isolate the primary feed tanks during recharge and the reserve tanks during normal operation. Liquid level indicator tubes are provided to monitor the solution inventory for each tank. The fabrication details for the level indicator tubes are given in Fig. II.3. The tanks are provided with a flanged bottom to allow removal or fitting of PVC liners and drain valves to empty the tanks. The tanks are pressurized with compressed nitrogen from a cylinder equipped with a two-stage gas regulator. A three-way solenoid valve is on the gas inlet line (see blowdown valve on Fig. 5) to blow down the tanks should the inlet pressure at the column inlet exceed the allowable maximum pressure for the column.

When oxygen is used as the oxidant, the oxygen concentration in the solution is limited by the partial pressure of oxygen and Henry's law constant for oxygen in water. Therefore, the oxygen concentration (less than the saturation value) in solution can be controlled by pressurizing the leach solution with either oxygen or an oxygen-nitrogen mixture, such that the partial pressure of oxygen over the solution corresponds to the desired oxygen composition for the solution. It may also be controlled by saturating deionized water with oxygen in one of the primary tanks (by pressurizing the water with oxygen) and then mixing the oxygenated water in correct

⁹ 1 psig = 6.893 kPa.

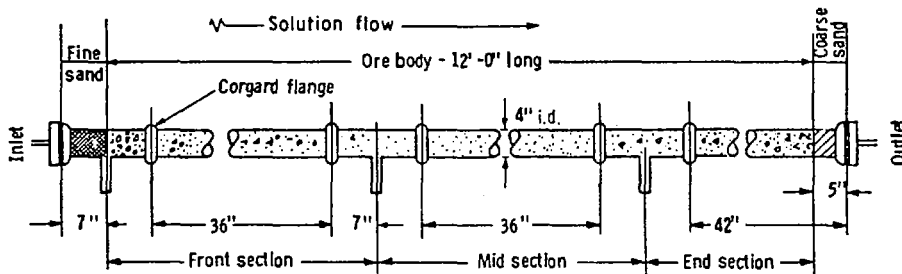


FIG. 6. Lengths of cylinder pieces and ore and sand packing for in situ leaching tests.

proportion with the reagent solution to obtain the desired oxygen level. The PVC liner is removed from the tank whenever oxygenation of the liquid phase is desired.

The solution flow from both the primary and reserve tanks is metered through a needle valve and monitored by a flow meter. Solution check valves downstream from the flow meter prevent backflow of solutions into the tanks. The solutions mix downstream before entering the pressure switch. Toggle valves shut off the feed from either the primary or the reserve tanks. By providing flow metering and monitoring for each tank, it is possible to control closely the desired oxidant and reagent compositions. All the solution lines are 3/8 in type 316 stainless steel tubing to minimize plugging. A pressure gauge is provided at the inlet to the ore column.

Horizontal ore column

The in situ simulation system consists of a 4 in i.d. and 13 ft long clear glass cylinder. The glass columns are rated for a maximum allowable working pressure of 50 psig. Fig. 6 is a diagram of the leaching columns.

Each column consists of two 3 ft sections, a 3.5 ft section, and three 14 in (4 × 1 in) tees. The sections were beaded end glass pipes and are assembled with beaded pressure pipe couplings manufactured by Corning Glass Co. The ends of the cylinder are using type 316 stainless steel end plates with a single opening. The end plates are shown in detail in Fig. II.4. The first 7 in is filled with fine casting sand. The sand is chosen so that the permeability is less than that of the ore packing in order to obtain a uniform flow distribution of leach solution prior to contacting the ore. This fine sand packing is followed by 12 ft of the ore sample for leaching. The last 5 in of the cylinder is packed with coarse casting sand. The packing is held in place during leaching by type 316 stainless steel coarse mesh, double weave wire cloth at the inlet and the outlet ends, shown in Fig. II.5.

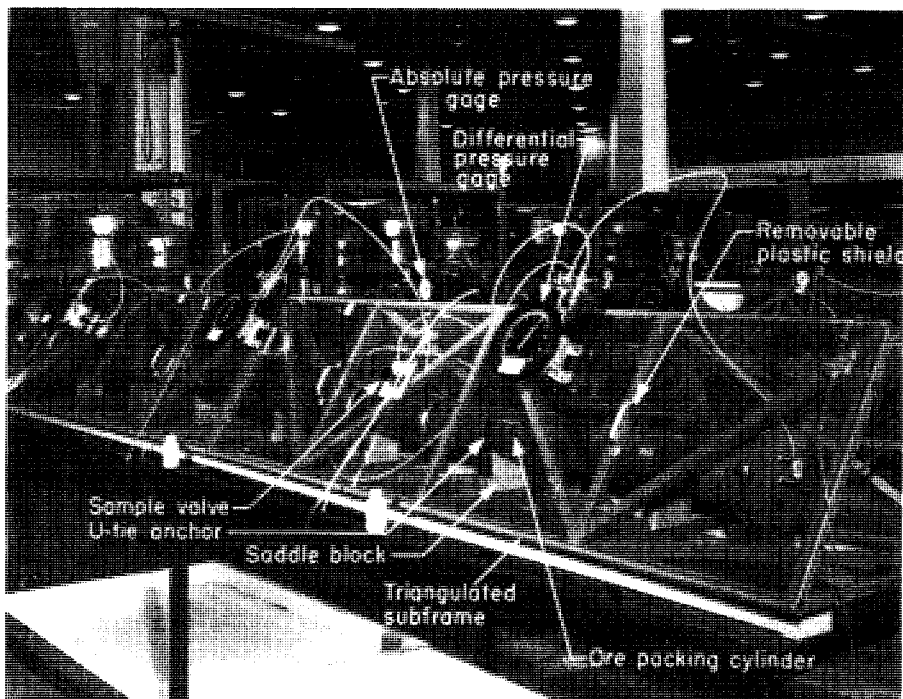


FIG. 7. Ore packing cylinder assembly in operation.

To monitor and obtain solution samples from the ore packing during the test, tee sections are located between each of the three straight sections. The tee sections allow insertion of pressure taps and/or specific ion electrodes and permit sample withdrawal. The sample and pressure tap assembly is shown in Figs II.6 and II.7. The taps are connected to needle valves and differential pressure gauges. The differential pressure gauges are made of type 316 stainless steel and will withstand a mean static pressure of 100 psig. These are shown schematically in Fig. 5.

The permeability of the ore sections is determined from the differential pressure readings across these sections. The concentration profiles and the rate of the exhaustion front (uranium depletion) movement are estimated from analysis of the solution samples collected. An overall view of a typical ore packing assembly used during the tests is shown in Fig. 7.

Support structure frames and layout

Overall views of the in situ leach simulation equipment in operation showing the location of the various components are shown in Figs 8 and 9. Design details of

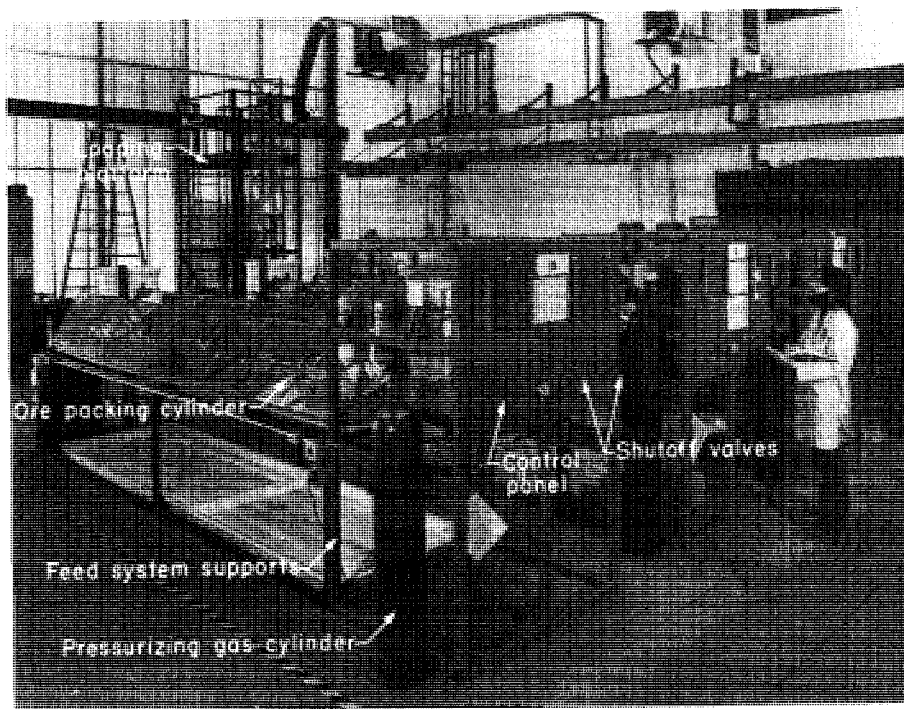


FIG. 8. *In situ leach simulation systems — front view.*

all supporting structures are given in Appendix II and the drawings included are as follows:

- (1) Feed tanks support frame (Fig. II.8)
- (2) Control panel layout (Fig. II.9)
- (3) Hole layout for electrical hardware mounting (Fig. II.10)
- (4) Leaching panel wiring diagram for automatic pressure blowdown on the feed system (Fig. II.11)
- (5) Space frame support for leaching cylinders (Fig. II.12)
- (6) Glass leaching cylinder support detail (Fig. II.13)
- (7) U-tie detail for leach cylinder support (Fig. II.14)
- (8) Wood block saddle detail for leach cylinder (Fig. II.15)
- (9) Cylinder subframe bottom plate (Fig. II.16).

Auxiliary and safety equipment

Each system requires a polyethylene product tank and a feed preparation tank. The tanks are of 55 gal capacity with a spigot at the bottom to facilitate draining.

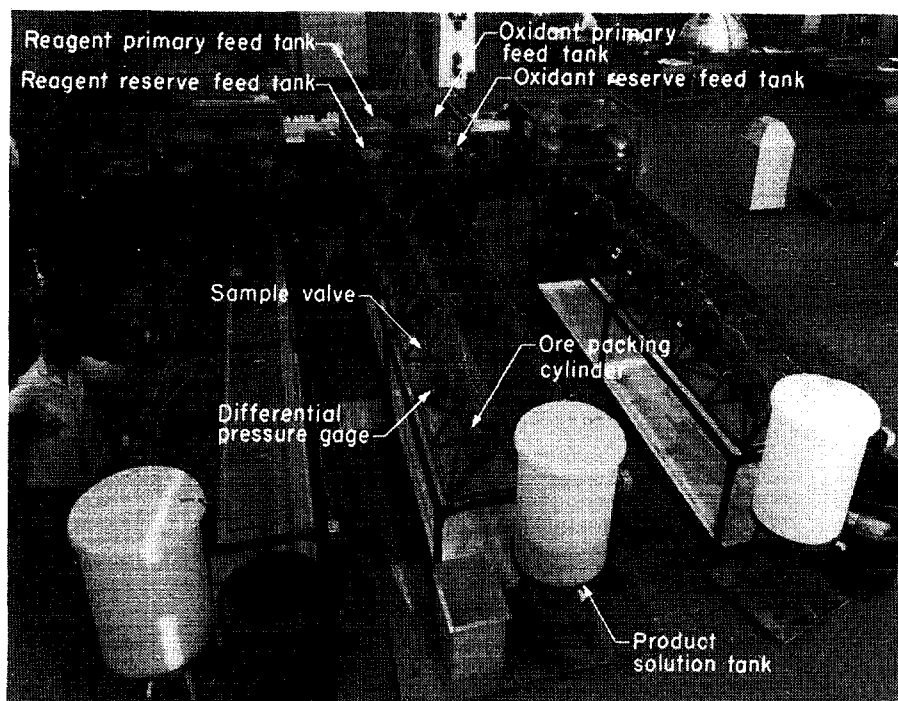


FIG. 9. *In situ leach simulation systems as viewed from above.*

Each experimental set-up has built-in safety features because of pressurized operation. To protect the glass cylinder in case of pressure regulator failure, individual pressure switches, adjustable in the range of 25 to 100 psig, are provided. Should the pressure at the inlet to the cylinder reach 45 to 50 psig, the pressure switch will trigger the relay (relay board in Fig. 5), which activates the three-way blowdown vent (BV) on the compressed gas line to the feed tanks and also the high pressure alarm. The relay must be reset manually to resume normal operation. The BV relieves the pressure in the feed tank. Glass column failure will not result in an explosion, because liquids flow through the system and because of the resistance provided by the packing and valving in the feed line. However, protection from potential glass cylinder explosion is provided by a 1/4 in plexiglass shield bolted to the subframe.

In situ leach simulation procedure

In situ uranium mining rate is governed by the uranium extraction rate and the permeability of the mineralized zone to the leach solution. For a given ore body, the uranium solution heads grade is a function of both the extraction rate and the leach

flow rate. The flow rate is in turn a function of the operating pressure and the permeability of the ore bed. At a given flow rate, the extraction rate determines the grade of the pregnant leach solution to the above ground process, the maximum overall recovery attainable in a given time, and time required to exhaust an ore's uranium. At a given operating pressure, the permeability of the ore bed determines the maximum leach solution flow rate and the energy requirements of the mining process.

In order to effectively model the in situ leaching process, the laboratory system must provide reliable data on all these variables: pressure, permeability, flow rate, uranium heads grade, and extraction rate. How the variables interact can be determined by examining how a change in one variable influences the other parameters.

In order to obtain meaningful simulation tests, it is necessary to reproduce, as closely as possible, in a column the characteristics of the ore in the field. The physical and chemical characteristics of the ore are governed by the following:

- (1) Particle size distribution
- (2) Packing density
- (3) Clay distribution
- (4) Bed voidage
- (5) Particle shape
- (6) Surface tension between solid and solution phase.

The particle size distribution was found to affect the packing characteristics of an ore. In Phase I of this programme it was found that a wide particle size distribution generally results in a more compact packing than a narrow distribution for a given packing technique. An insufficiently compacted bed results in fines migration and a consequent drift in permeability. Also, additional compacting during flow may cause cracking and channelling. A well compacted bed results in a mechanically stable packing and in better permeability control.

The size distribution also affects the chemical behaviour of the packing. The uranium in the smaller particles is more readily accessible to the leach solution because of the greater resistance to mass transfer in the larger particles.

The particle size distribution of the ores used is that obtained after crushing and blending the ores to a size of minus 10 mesh (0.810 mm effective dia). This size is considered sufficiently large to preserve the grain boundaries (the modal grain size varies from 0.2 to 0.5 mm for unconsolidated sandstone uranium mineralizations), yet small enough to allow compact packings. The particle shape and the surface tension effects between the solid and solution phase are considered as uncontrolled variables.

It is important to minimize segregation during the packing operation. Although wetting the ore prior to packing would minimize the particle segregation, this could promote oxidation, thus affecting the leaching results. Therefore, dry ore is used to prevent any additional ore pre-oxidation. The clay fraction (minus 325 mesh)

distribution within the packing matrix is generally uniform, owing to the adherence of the clay particles to larger particles, and is believed to be similar to the actual underground ore body.

Using the crushed and blended ore, a method was developed to obtain packed columns which consistently reproduce the field permeability of the ore. Data obtained from field tests showed that ore body permeabilities are in the range of 0.3 to 0.6 darcy, with the maximum observed being 2.0 darcy. Packing methods were developed for a Texas and a Wyoming ore, both of which consistently gave permeabilities in the range of 0.5 to 2.5 darcy. Successful packings obtained with a given ore are of relatively constant density and voidage. However, discrete lenses, which occur in an actual ore body, are not produced in this column of ore. The packing procedures followed are described in detail in Appendix III.

After examining the packing to make certain that no voids, faults or gross segregation are present, a column test is started with a 150 h deionized water flow period to ensure correct permeability and packing stability. During this initial flow period, the ore is wetted slowly (2 to 5 mL/min) in order to allow it to consolidate. After approximately 24 h, the flow is increased gradually during the next 24 h until the desired rate (0.5 to 1.0 linear ft/h) is obtained.

After the permeability and stability of the packing are found acceptable, the chemical leach solution can be introduced. A packing is generally acceptable if the permeability is within a range considered proper for the ore body under study and if no cracks or ore compaction occur. The reagent flow is carried out for the desired duration (typically 750 h) and initially at the desired rate (10 mL/min). However, as permeability drops, the flow is constrained by the maximum allowable operating pressure, 45 psig for the glass columns.

In tests utilizing peroxide as the oxidant, the oxidant and reagent solutions are prepared to twice the desired leach solution concentration. The solutions are stored in separate feed tanks to prevent premature decomposition of the peroxide and are mixed just prior to injection into the ore bed. Equal volumetric flow rates of the reagent and peroxide are maintained to obtain the desired solution concentration. The peroxide and reagent feed solutions should be replaced every 48 to 72 h with freshly prepared solutions. Records of the fill and drain volumes, and the time between changes, are kept for each feed tank to estimate mean flow rates between each fill and drain.

In tests where dissolved oxygen is used as the oxidant, the in situ leach simulation system is modified to saturate with oxygen the solution in one of the feed tanks. This can be achieved by removing the vinyl bladder inside the tank and adding a provision for solution recycle. The pressurizing gas in the tank is oxygen, with the solution sprayed inside the tank using a high output recycle gear pump. The solution is recycled in a closed loop to saturate it with oxygen. The magnitude to which the feed tank is pressurized with oxygen determines the oxygen content of the feed.

In those tests where the restoration aspects of a given leach are to be studied, simulated ground water and, if desired, a restoration solution can be passed through

the leached ore bed. The magnitude of ore aquifer restoration effect required can be estimated from the ion leakage resulting from these solution flows. This portion of the test must include 5 to 10 pore volumes of solution to obtain a reasonable estimate of the required restoration.

At the termination of each test, the packed column is disconnected from the system and allowed to drain in a vertical position, with the inlet end at the bottom, for a minimum of 48 h. The volume of drained solution is measured and sampled. After draining, the ore column is set horizontally for approximately 48 h to allow the remaining moisture to distribute uniformly. The system is then disassembled to obtain the post-leach residue samples.

A typical schedule for sampling and monitoring each system is as follows:

- (1) The pressure profile along the length of the packing is recorded every 6 h.
- (2) All volumetric flow rates are recorded every 6 h. Any required flow adjustments, based on a precalibrated flow meter, are made at this time.
- (3) The cumulative product solution volume is measured every 48 h. A 500 mL sample is collected at that time. During post-leach ground water or restoration flow, the cumulative product volume is measured and sampled every 24 h.
- (4) Solution profile samples are collected along the length of the column approximately every 48 h. The 50 to 75 mL samples are used to determine uranium, peroxide, pH, and other concentration profiles along the ore bed.
- (5) Prior to leach, a 100 g sample of ore is taken from each bag of ore used in the tests. Typically, two bags are used in each test.
- (6) Wet post-leach residue samples, weighing about 10 lb each, are dug out of the centre of each of the three ore sections. These samples are bagged in plastic and sealed for future analytical needs (for example, moisture and uranium content).
- (7) Visual changes, if any, are recorded every 6 h.

In order to minimize the number of analyses, some of the product solution samples can be blended in their respective volume ratios. These blended samples are then analysed for the uranium content, the principal cation (for example, NH_4^+), and the principal anion (for example, HCO_3^- , or SO_4^{2-}). During the post-leach ground water or restoration flow, the product samples are also analysed for contaminant ions (for example, NH_4). The pH of the product solution samples is also recorded.

A select number of profile samples can be analysed for uranium (or other species) to provide profiles along the length of the packing for a given leach. Typically, the redox potential, using a $\text{Pt-Ag/Ag}^+\text{Cl}^-$ reference electrode, and the pH of all profile solution samples are measured and recorded.

In order to determine a uranium balance on each system, all samples of ore used to pack the ore bed and the three post-leach residue samples from each test are analysed for uranium. The uranium input and ore uranium output are thus measured for each test.

The pressure and flow rate measurements taken are used to determine the permeability of each of the three ore bed sections. The correlation (Darcy equation)

used in the calculation of permeability ($1/\alpha$) is given by:

$$\frac{1}{\alpha} = 6.223 \times 10^{-\lambda} \frac{\mu V' L'}{\Delta P' A}$$

$$\frac{1}{\alpha} = \text{permeability, darcy (D)}$$

where μ = solution viscosity, cP,
 V' = solution volumetric flow, mL/min,
 L' = packing length, inch,
 $\Delta P'$ = pressure differential across the ore bed section, psi,
 A = cross-sectional packing area, cm².

Thus, for a given ore section length and cross-sectional area and assuming a solution viscosity of 1.0 cP, the permeability can be determined for a given flow rate and pressure drop.

Information obtainable with the in situ leaching system includes the following:

- (1) Maximum overall uranium recovery in a given test duration.
- (2) Uranium recovery as a function of the volume of leach solution required at a given flow rate through the ore.
- (3) Number of bed displacements required to obtain the maximum recovery.
- (4) Permeability change with time.
- (5) Leach controlling mechanism(s).
- (6) Physical phenomena occurring in the simulated ore body during the leaching.
- (7) Environmental effects of leaching.

Experimental data references and conclusions

The in situ leach simulation studies were conducted on Wyoming and Texas ores. The Wyoming ore, which contained approximately 0.06% U₃O₈, was obtained from the Shirley Basin mine. The Texas ore contained 0.1% U₃O₈; it was obtained from the Conquista operation at Falls City, TX. Detailed experimental data from the in situ leaching and post-leach restoration studies are given in Refs [6-9].

The simulation method has shown the capability of consistently obtaining representative ore packings and of reproducing the physical phenomena of in situ leaching. The simulation test is believed to be superior to traditional laboratory methods of studying in situ leaching (for example, agitation leach tests and short core testing). Phenomena such as the reprecipitation of uranium at the leading edge of a neutralization front, a uranium exhaustion front movement rate, or the chemical profile of a potential contaminant along an ore bed, can be detected with this method, but are nearly impossible to observe in short packings or undisturbed core tests. Agitation leach tests aid in the understanding of the leach chemistry, but do not

provide any hydrological information. The simulation tests assist in qualitatively estimating the impact of various mechanisms that cause permeability changes.

The results obtained with this system have indicated the need for a long packing, preferably 12 ft or more, to be able to detect the movement of the various chemical reaction zones or 'fronts'. The flow control achieved with this system is not totally satisfactory because of the large variance observed. An automated system of flow control, though expensive, could help correlate data and make more reproducible experiments possible.

In order to obtain some verification on the reliability of the column test results it is essential to determine the oxidation state and the extractability of the uranium in the ore sample. Since the laboratory effort is directed at the simulation of in situ leaching, it is necessary to identify the differences between uranium occurrence in the formation underground and in the processed ore, in order to understand the kinetics of the leach phenomena. Also, to obtain realistic leach behaviour, it is necessary to use an ore with an oxidation state representative of the actual ore body.

The simulation tests exposed the importance of correct particle size distribution in achieving successful testing. Texas ore with finer particles and near bimodal distribution represented the undesirable end of the spectrum. The sensitivity of a packing to plugging by precipitation, clay swelling, or gas blockage is a strong function of the mean pore diameter, particle shape factor, and surface tension characteristics. Selection of the correct particle size distribution may be critical for obtaining results which are indicative of what will actually be obtained in the field.

It should be noted that the simulation leach column tests are a one-dimensional representation of a complex phenomenon occurring underground. The information collected, and the results and conclusions derived from the tests, should be understood in this context.

Even with the limitations of this system, the method is believed to be an acceptable means to study uranium in situ leaching and restoration as a function of various parameters. At the very least, the system allows one to compare the leaching and restoration characteristics of various leaches and ores.

Appendix I

**LIST OF MAJOR COMPONENTS REQUIRED FOR ASSEMBLY
OF IN SITU SIMULATION SYSTEM AND OPERATION**

The major components required for the assembly of one in situ leaching simulation system and the vendors who supply these components are listed below. The figures showing the item's locations or its details are also given.

Description	Vendor	Catalogue number	Figure location
Feed system support frame	Custom fabrication	NAP	II.8
Control panel	Custom fabrication	NAP	II.9
Flow meters	Matheson Gas Company	7641-Tube No. 603	II.9
1/4 in toggle valve (brass)	Pittsburgh Valve and Fitting	NAP	1
3/8 in needle valve, 316 stainless steel (SS)	Pittsburgh Valve and Fitting	No. SS-6VS6	1
Pressure gauge (0-110 psi), 316 SS	Matheson Gas Company	Model No. 63-3212	1
3/8 in Swagelok tees, 316 SS	Pittsburgh Valve and Fitting	No. SS-600-3	1
3/8 in toggle valve, 316 SS	Pittsburgh Valve and Fitting	No. SS-16S6	1
Check valve, 316 SS	Stanley Berg (Circle seal)	2259TI-2MM	1
Gas pressure regulator (N ₂)	Matheson Gas Company	Model No. 18H-GG 4580	1
3-way valve	Continental Sales Equipment Company	No. 8320-A172	II.11
Level indicator bracket	Custom fabrication	NAP	II.3
Pressure switch	Delaval Turbine Company	PIH-B340-SST	II.11
Primary feed tank, 316 SS	Munroe Manufacturing	NAP	II.1
Reserve feed tanks, 316 SS	Munroe Manufacturing	NAP	II.2
Primary tank flange plate, 316 SS	Custom fabrication	NAP	II.1
Reserve tank flange plate, 316 SS	Custom fabrication	NAP	II.2
Primary tank liners	Sears-Roebuck and Company	U20-9 Model No. 390-29160	NAP
Reserve tank liners	Sears-Roebuck and Company	U20-7 Model No. 390-29110	NAP
Column end plates, 316 SS	Custom fabrication	NAP	II.4
Sampling port end plate, 316 SS	Custom fabrication	NAP	II.6
4 in × B.P. couplings	Mooney Process Equipment Company	72-0394	2

Description	Vendor	Catalogue number	Figure location
4 in × 1 in B.P. tees	Mooney Process Equipment Company	72-0394	2
4 in × 36 in B.P. pipe sections	Mooney Process Equipment Company	72-4362	2
4 in × 42 in B.P. pipe sections	Mooney Process Equipment Company	NAP	2
Cylinder triangular subframes	Custom fabrication	NAP	II.12
3/4 in ply subframe bottom plate	Custom fabrication	NAP	II.16
Sample ports	Custom fabrication	NAP	II.7
Unistrut for support frame	Unistrut Service Company	P100 GR	II.10
Latching relay	Camradio-RPC	90KB17AY-120VAC	II.11
Transformer (6-3 VAC-SEC)			II.11
Door bell			II.11
Terminal strip			II.11
Push button switch			II.11
Differential pressure gauge, 100 psig abs; 0-10 psi	M.S. Jacobs (ITT Barton)	Model 227-316 SS with Model 224 DP unit	1
Back pressure regulator	M.S. Jacobs (Fairchild)	66152-BP-G	1

Appendix II

DETAILED DRAWINGS OF VARIOUS COMPONENTS OF IN SITU SIMULATION SYSTEM

The drawings contained in the following figures are of sufficient detail to allow the construction of each necessary component of the in situ leaching system. Exact sizing and type of materials are indicated for each of the components.

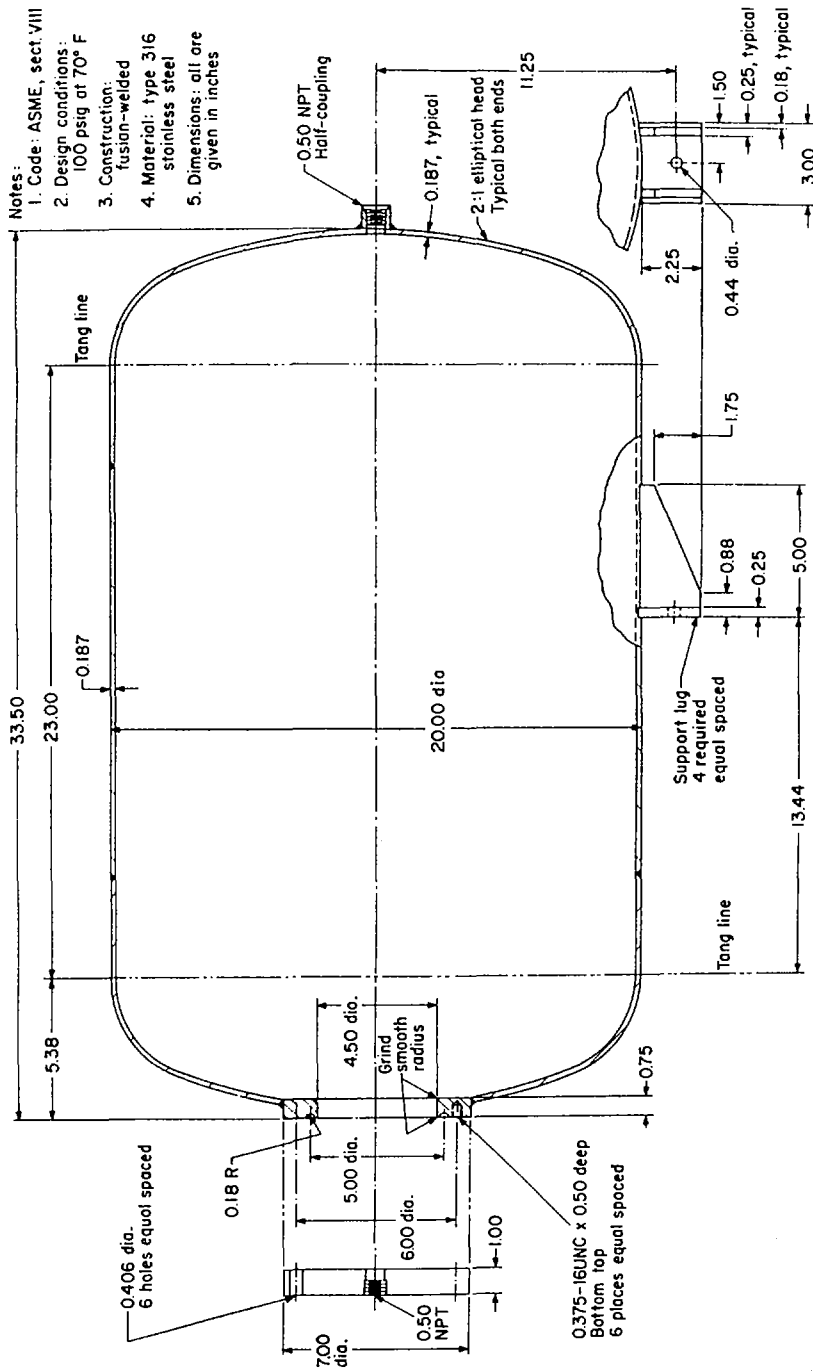


FIG. II.1. Fabrication details of primary feed tanks for in situ leach simulation system.

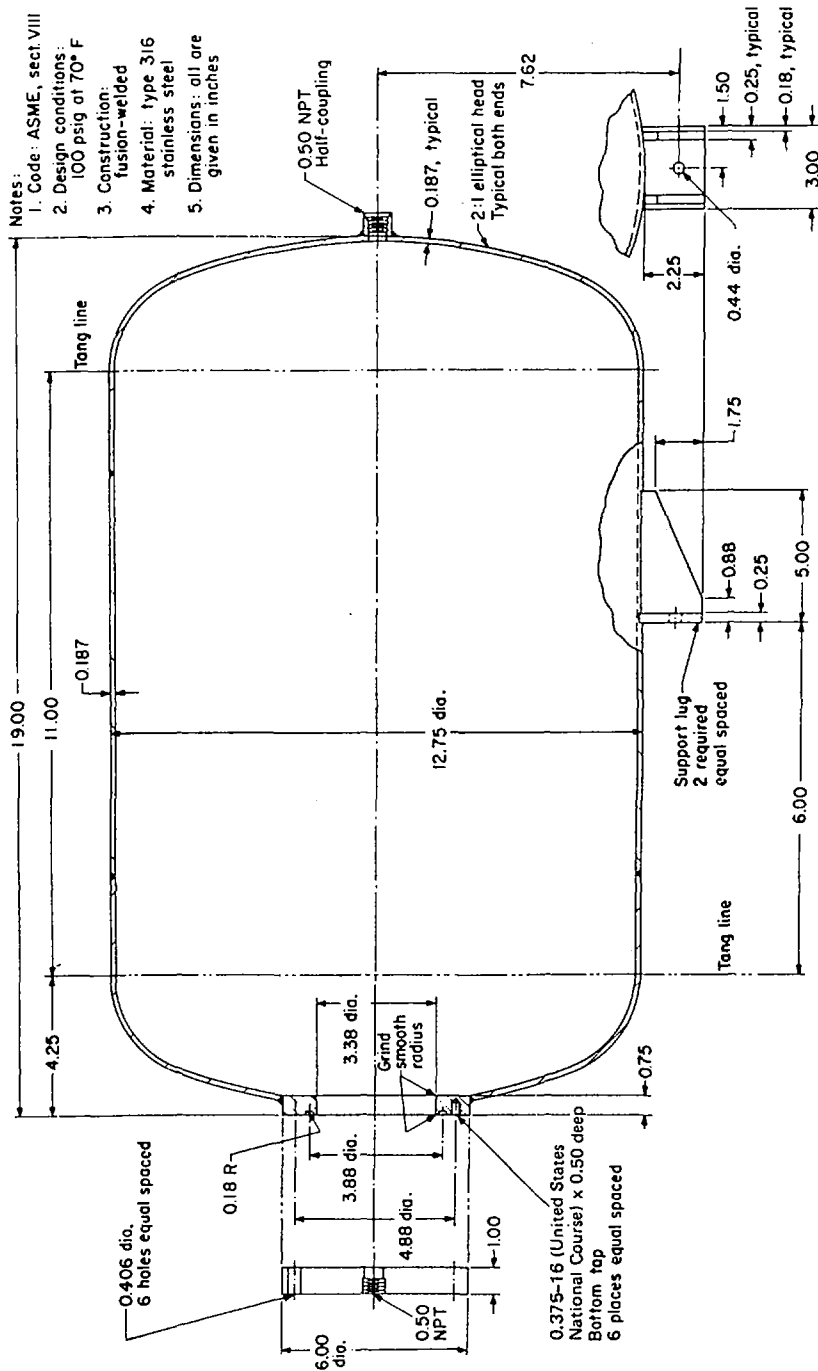


FIG. II.2. Fabrication details of reserve feed tanks for in situ leach simulation system.

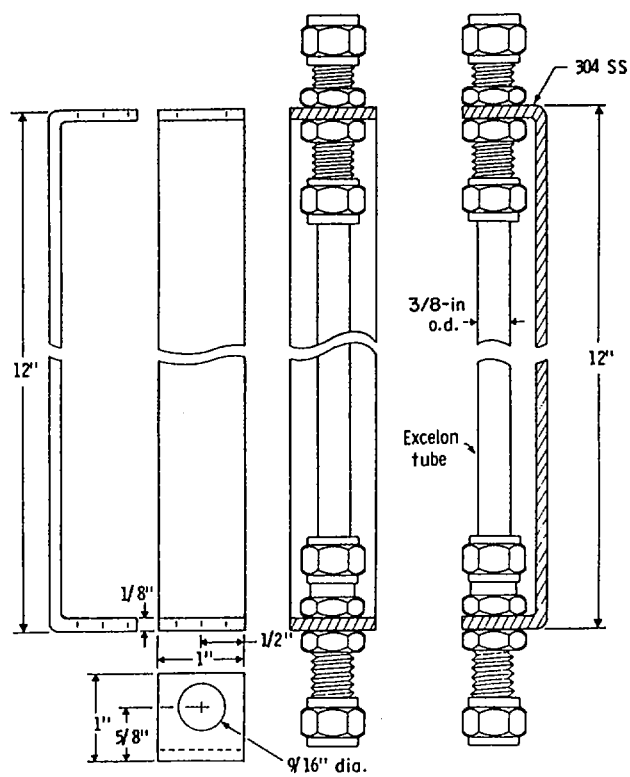


FIG. II.3. Liquid level indicator and mounting bracket for feed and reserve tanks.

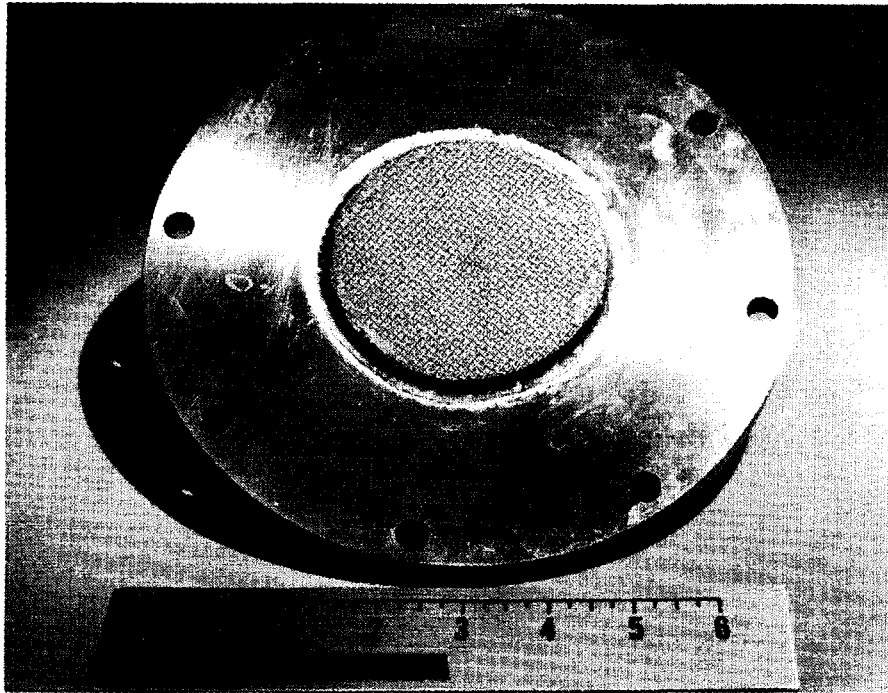


FIG. II.4. Leach cylinder end plate.

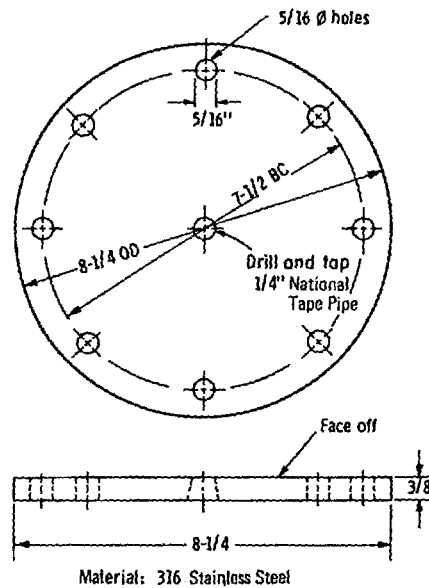


FIG. II.5. End plate with double weave wire cloth used in ore packing cylinder assembly.

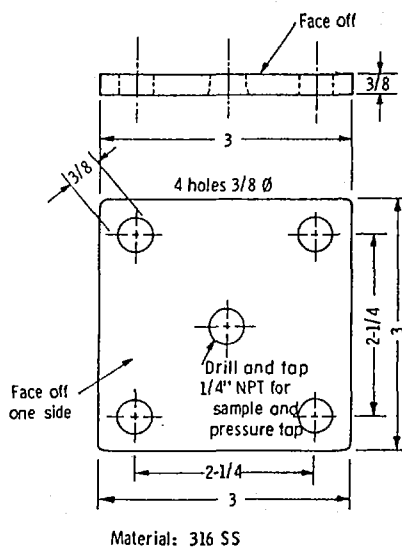


FIG. II.6. Sample and pressure tap end plate.

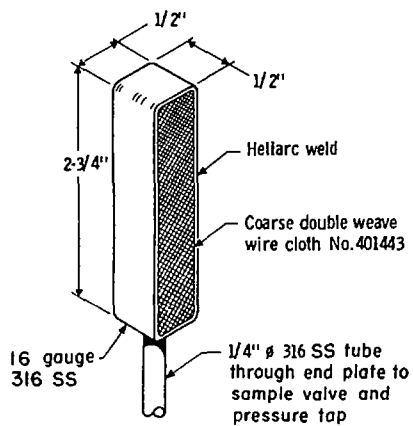


FIG. II.7. Detail of sample and pressure tap.

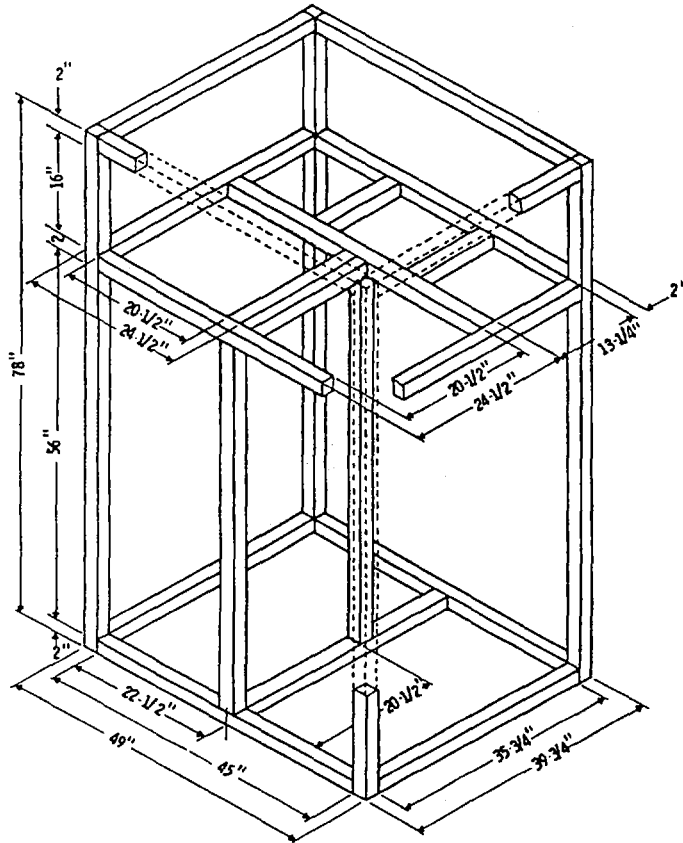


FIG. II.8. Feed tanks support frame (material 2 in \times 2 in square mild steel tubing 1/16 in thick).

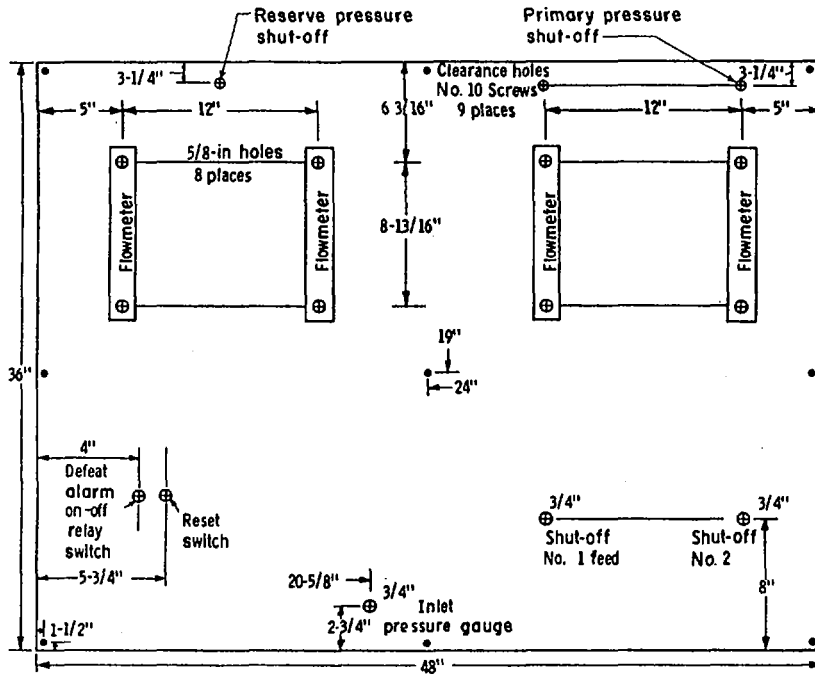
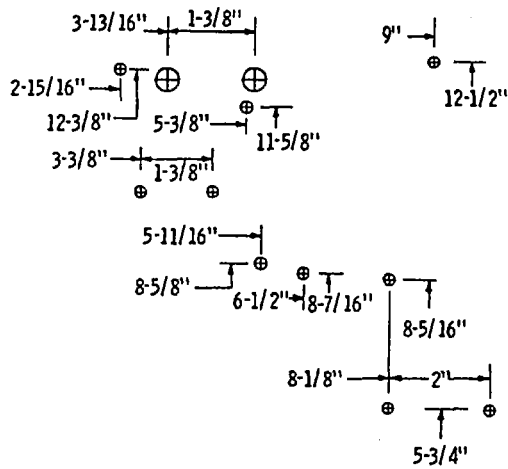


FIG. II.9. Control panel layout.



Lower Left Corner of Control Panel

FIG. II.10. Hole layout for electrical hardware mounting.

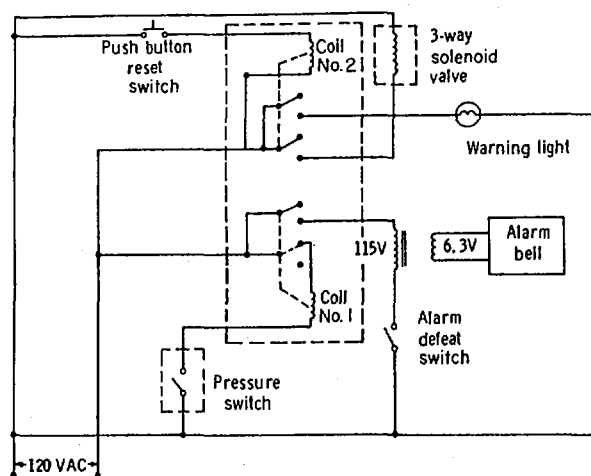


FIG. II.11. Leaching panel wiring diagram for automatic pressure blowdown on the feed system.

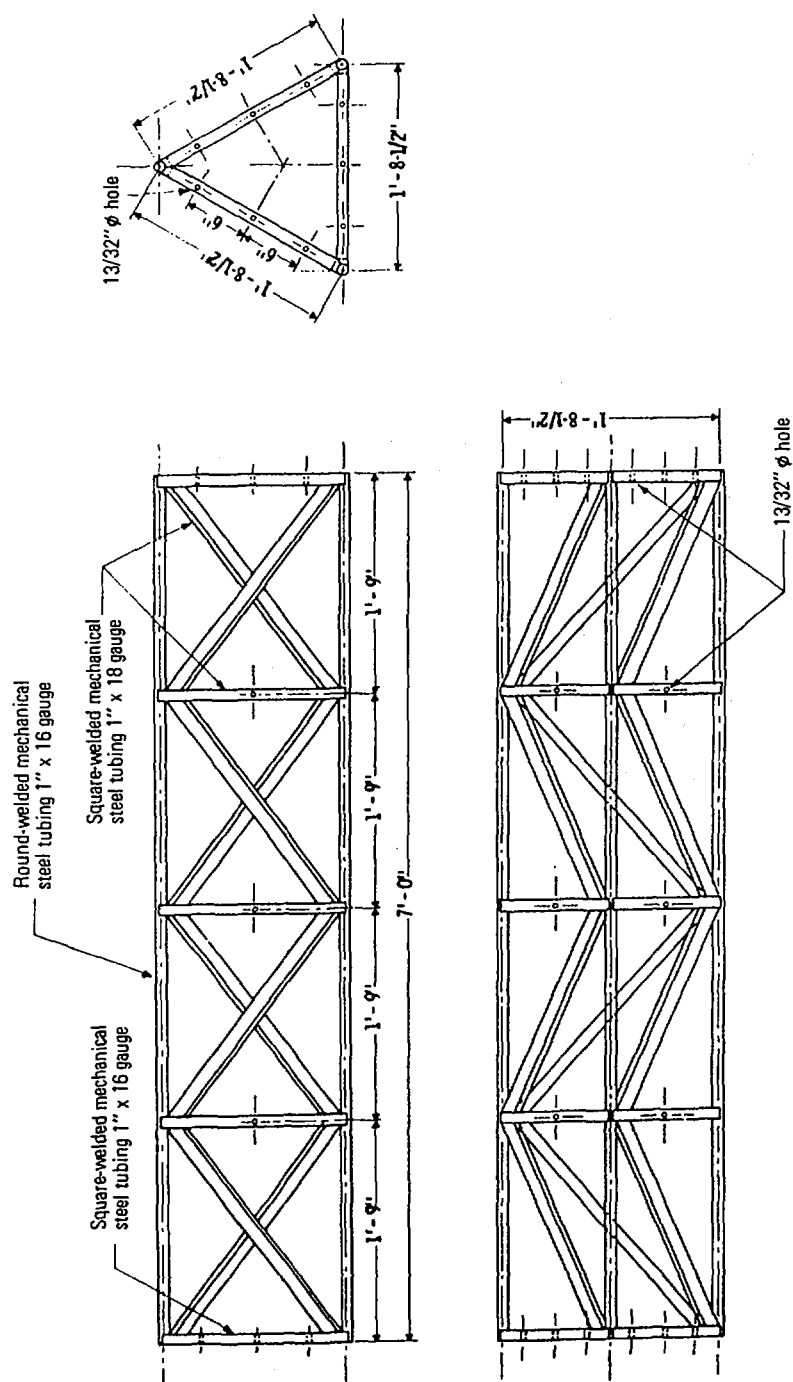


FIG. 11.12. Space frame support for leaching cylinders.

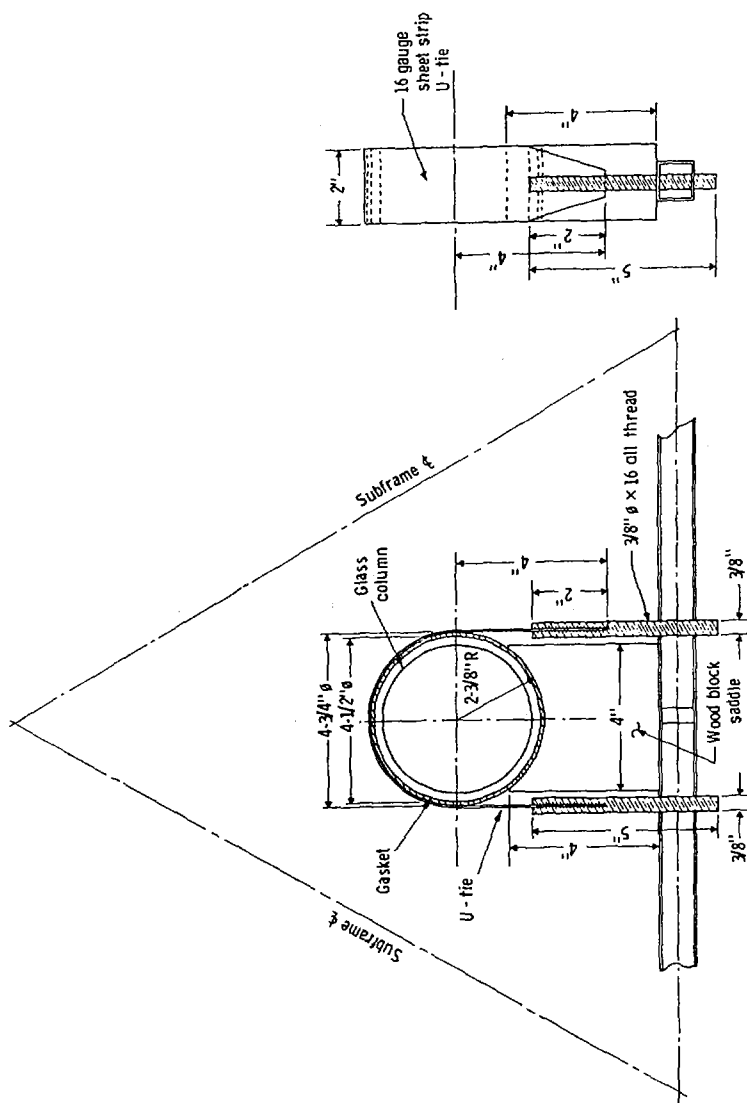


FIG. 11.13. Glass leaching cylinder support detail.

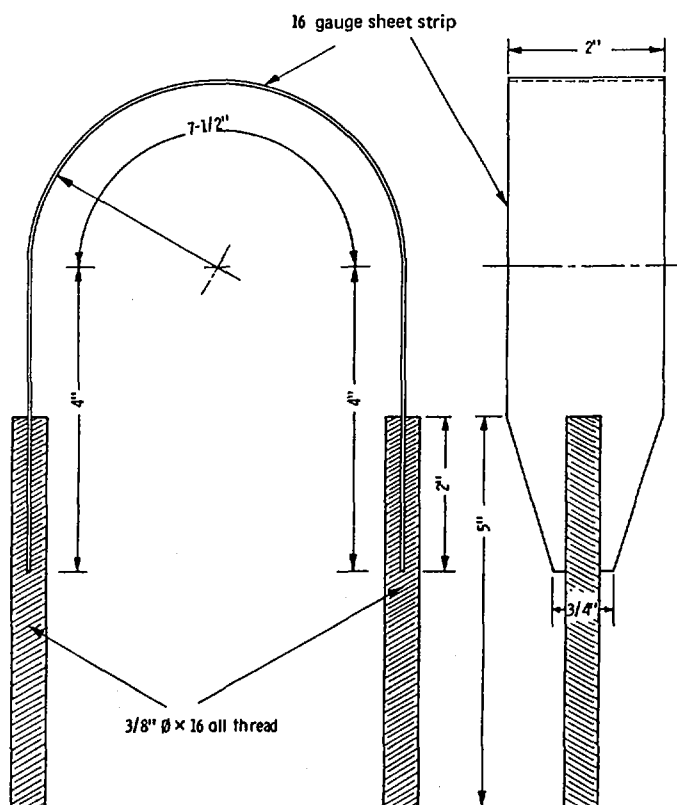


FIG. II.14. U-tie detail for leach cylinder support.

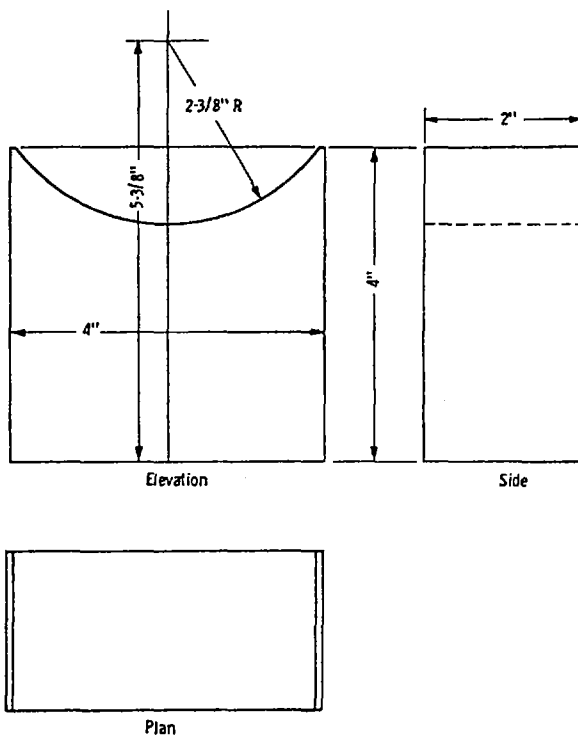


FIG. II.15. Wood block saddle detail for leach cylinder.

Appendix III

PROCEDURE FOR PREPARING 4 in i.d. GLASS COLUMN FOR IN SITU LEACH SIMULATION TESTS

The following steps are followed in preparing the packing assembly for simulation tests:

- (a) Weigh out one 3.5 kg batch and one 2.5 kg batch of -80 American Foundrymans Society (AFS) subangular sand and a 60.0 kg batch of ore sample for packing. Record the weight of each batch.
- (b) Determine the weight of empty 4 in i.d. glass cylinder assembly on the sub-frame. The assembly includes gaskets, flanges, end plates, fasteners (nuts and bolts), sample and pressure taps and Lexan safety shields. It does not include the sampling valves, the pressure gauges, the differential pressure gauges, and all the tubing used to connect these to the assembly.
- (c) Close all sample ports on the cylinder with rubber stoppers and anchor the cylinder vertically to the ore packing platform. The cylinder should be positioned with the solution inlet end at the bottom. Attach the dust exhaust manifold to the triangular subframe.
- (d) Place -80 AFS subangular sand using the ore placement container at the bottom of the cylinder, until the sand level is up to the sample port opening (approximately 7 to 7.5 in from the inlet end plate). Drop the tamper from a height of 2 ft approximately 10 times.
- (e) Insert the sample and pressure tap and fasten the sample port and plate finger tight.
- (f) Place more -80 AFS sand to cover that sample and pressure tap.
- (g) Place approximately 4 in batch of ore sample (batch size expressed in terms of packing length) on top of the -80 AFS sand packing at the bottom.
- (h) Wiggle the sample and pressure tap in all directions until the packing material is uniformly distributed around it. Make sure there are no empty spaces left.
- (i) Pack the ore using the suitable packing method derived for that ore to achieve a desired permeability. The following is an example of the procedure used to pack the Wyoming and Texas ores in this study: Place ore using the ore replacement container (Fig. III.1) batchwise into the column followed by tamping using a peg bottom drop-weight tamper (Fig. III.2). Use an 8 in batch size for each batch. Use a 2 ft amplitude for tamping with a frequency of 10 tamps per 8 in of Texas ore.
- (j) Continue step (i) in the bulk of the cylinder and steps (e), (g), and (h) at the sample ports until the ore packing is 6 in from the top.¹

¹ An additional 4 in i.d., 3 ft long glass pipe is connected to the top of the leach cylinder using a coupling similar to that in the assembly. This 3 ft section is useful in tamping the last 2 ft of the packing.

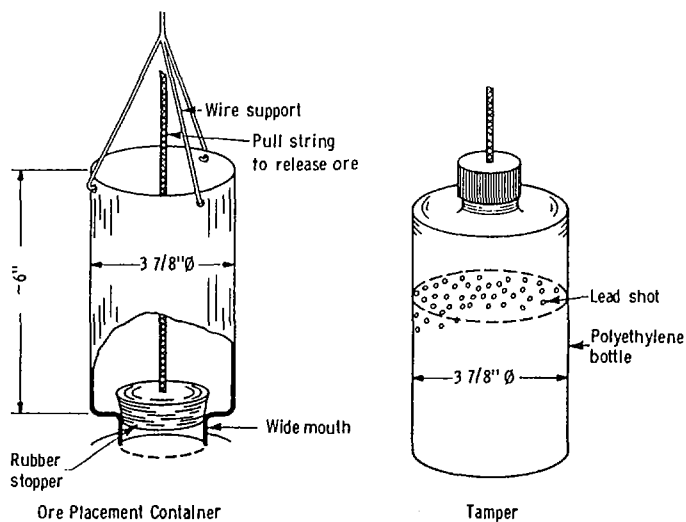


FIG. III.1. Details of ore placement container and tamper.

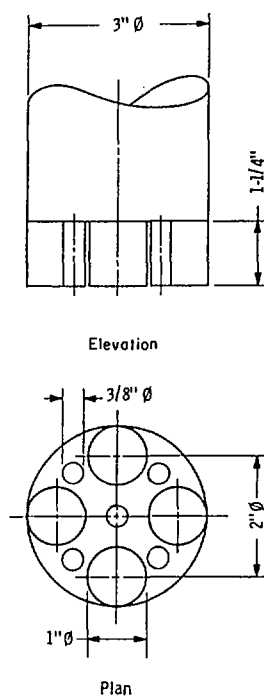


FIG. III.2. The bottom detail of the peg bottom drop-weight tamper.

- (k) Add from the second batch of -80 AFS subangular sand to fill the last 6 in of the cylinder and pack as before.
- (l) Disconnect the 3 ft long extension pipe, clean any sand spills over the tip and close the cylinder packing using the outlet end plate. Torque all the bolts on the outlet end plate to 4 ft-lb.
- (m) Disconnect the cylinder assembly from the ore packing platform and lower it to a horizontal position. Brush off all packing spills from the subframe. Torque the sample port end plate bolts to 3 ft-lb on the torque wrench.
- (n) Weigh the packed cylinder assembly prior to leach.
- (o) Weigh and record the two sand and the ore batches remaining from the original batches after packing operation.
- (p) Determine the difference between weights of the sand and ore batches before and after the packing operation to determine the weight of inlet, the ore, and the outlet packings.
- (q) Move the packed cylinder assembly to the in situ leach system and connect (1) solution feed line, (2) sampling valves, (3) absolute and differential pressure gauges, and (4) the packing outlet to the product solution tank.

REFERENCES

- [1] MERRITT, R.S., PETERSON, H.D., WENTZ, C.N., Heap Leaching Studies on Uranium Ore, Phases I through III, Bureau of Mines Open File Rep. OFR 118-76, Salt Lake City, UT (1976) 162 pp.
- [2] MERRITT, R.S., PETERSON, H.D., WENTZ, C.N., Heap Leaching Studies on Uranium Ore, Phase IV, Bureau of Mines Open File Rep. OFR 141-77, Salt Lake City, UT (1977) 138 pp.
- [3] MERRITT, R.S., PETERSON, H.D., WENTZ, C.N., Heap Leaching Studies on Uranium Ore, Phase V, Experimental Studies, Bureau of Mines Open File Rep. OFR 18-79, Salt Lake City, UT (1978) 219 pp.
- [4] MERRITT, R.S., PETERSON, H.D., WENTZ, C.N., Heap Leaching Studies on Uranium Ore, Phase VI, Field-Scale Heap Leaching Test Program, Bureau of Mines Open File Rep. OFR 117-80, Salt Lake City, UT (1980) 129 pp.
- [5] PETERSON, H.D., SEIDEL, D.C., NICHOLS, I.L., Summary of a Technique for Heap Leach Simulation on Uranium Ore, Bureau of Mines Open File Rep. OFR 193-84, Salt Lake City, UT (1985) 3 pp.
- [6] SUNDAR, P.S., In Situ Leaching Studies of Uranium Ores, Phases I Through III, Bureau of Mines Open File Rep. OFR 140-77, Salt Lake City, UT (1977) 392 pp.
- [7] GRANT, D.C., In Situ Leaching Studies of Uranium Ores, Phase IV, Bureau of Mines Open File Rep. OFR 52-79, Salt Lake City, UT (1979) 497 pp.
- [8] GRANT, D.C., In Situ Leaching and Restoration Studies of Uranium Ores, Phase V, Bureau of Mines Open File Rep. OFR 84-81, Salt Lake City, UT (1981) 591 pp.
- [9] GRANT, D.C., SEIDEL, D.C., NICHOLS, I.L., Technique for In Situ Leach Simulation on Uranium Ores, Bureau of Mines Open File Rep. OFR 192-84, Salt Lake City, UT (1985) 59 pp.

DISCUSSION

A.T. TRIGGS: Is this equipment operational? Can the industry use it in the next few years?

D.C. SEIDEL: The equipment is no longer set up. It was not intended to be a permanent test facility. The information has been reported and it is now up to the industry to use it as needed.

A.T. TRIGGS: What was the cost of both the heap leaching and the in situ leaching tests including the equipment?

D.C. SEIDEL: The cost of each one of these programmes was about US \$500 000.

K.H. TREUE: Will a series of tests, such as you have described, provide specific information for the development of an actual project? For instance, will these tests indicate how much solution has to be injected in a given time to obtain a given per cent extraction and what should be the optimum spacing for injection and production wells?

D.C. SEIDEL: These laboratory tests can provide useful information but this must be supplemented by field tests. One can observe in the laboratory things that cannot be seen in the field. For instance, we saw that when hydrogen peroxide is used as an oxidant there is a large and instantaneous liberation of oxygen starting at the injection point and extending for about 60 cm. This was suspected to happen underground but was not observed until now.

L.H. TREUE: If you were to perform these tests for an actual project, how would you obtain a proper sample? You would only be able to get cores. Would you mix the material from different cores and load that in the equipment or would you use a single complete core?

D.C. SEIDEL: Our primary objective was not to simulate exactly a specific deposit but to understand better what happens underground when you do in situ leaching. We took our samples, crushed them and re-bedded them in the column. In many cases it is not even possible to obtain a core sample because the deposits that are amenable to in situ leaching are unconsolidated sandstones. The material is not coherent; some very elaborate systems have been devised to try to get something that resembles a core. If you could get a consolidated consistent core you probably could not use in situ leaching because the permeability would be too low. In any case you could not use directly an intact vertical core for the test because the flow of solution is not in the vertical direction.

H.J. STEINER: But how did you verify that the permeability in the horizontal test column is the same as in the deposit?

D.C. SEIDEL: There is no way to do that exactly. We can calculate the permeability in our tests and compare that with permeabilities that have been measured in the field and we can see that they are similar. But, again, I want to point out that our objective was not to establish an absolute relationship with what happens underground but to see how the permeability varies during leaching. And we saw there are

very important variations. There is a very severe drop in permeability at the point of injection because of degassing of the hydrogen peroxide. This effectively plugs up the flow and the solutions will, of course, take the path of least resistance so they may not go where the uranium is. On the other hand, the permeability in the zones beyond the injection point are much higher and remain so throughout the test. This plugging effect does not happen with acid leaching or when using oxygen. This is probably why oxygen is more widely used than hydrogen peroxide. There is another example involving a sodium bicarbonate leach. The sodium ions reacted with the clay in the ore body. The clay swelled and plugged up the flow. We also studied groundwater injections for restoration of the aquifer, and ways to keep the uranium and the heavy metals for bleeding for long periods after leaching has been completed. Aluminium chloride solutions effectively stop the bleeding of uranium and selenium.

M.U. SAGDIK: You have discussed the permeability of the ore, that is to say, the permeability between the ore particles. But what about the diffusion of the solution into the particles?

D.C. SEIDEL: Most of the ores that have been successfully treated by in situ leaching in the United States are unconsolidated sandstones. They are not stuck together so there is space available between the grains. I really should not say that these materials were crushed to 10 mesh because they were originally almost minus 10 mesh.

M.C. CAMPBELL: Is the uranium in these sandstones within the ore particles or on the surface of the particles?

D.C. SEIDEL: The uranium is mainly on the surface of the particles. If you de-slime these materials, 80% or more of the uranium would be in the minus 200 mesh fraction. There is not much inside the grains.

J. SERRANO: We have in Spain a uranium ore with a very high clay content. We have tested agglomeration with Portland cement and gypsum and this works very well in the laboratory scale. Do you think the pellets would be stable in heaps higher than 3 metres? Will the bottom pellets not be crushed in heaps 5 or 6 metres high?

D.C. SEIDEL: We did some experiments with pelletized clay-like materials, but with poor results. It is not that the pellets broke down but during the formation of the pellets the fines collected on the surface. As the solutions percolated down the column the fines were washed off and accumulated in the lower part of the bed and plugged the column. We were using several columns in series and had effective heights of up to 18 ft.

M.C. CAMPBELL: In the heap leaching tests in columns, did you take the columns apart after the test and compare the residue with the original ore? Did you make chemical, mineralogical and other comparisons?

D.C. SEIDEL: In many cases we were not able to identify more than 10 or 15% of the uranium mineralization in the original ore. Much of the uranium is highly disseminated, in very fine particles and in the clay fractions. In some instances there was migration of fines. In some previous work we found that if you have more than about 3% limestone in the ore it cannot be leached by percolation or in heaps because

carbon dioxide is liberated and the gas bubbles accumulate in the bed and plug it. In one experiment we built a large heap and after leaching we took it apart in 3 ft layers. We covered the heap with a net and we took a sample at each knot in the net. Then we removed a 3 ft layer, placed the net again and repeated the procedure until we had a complete profile of the heap. This was very expensive but showed several very surprising things. There was a pond of liquid on the top of the heap and the flow of solutions down the heap was almost vertical down to the bottom. We expected the solutions to fan out but this did not happen.

M.U. SAGDIK: Did you recirculate the pregnant solutions during your column test? In our own tests in Turkey we obtained a pregnant solution with high uranium content without recirculating during the first days.

D.C. SEIDEL: We did recirculate the solutions during the heap leach simulation tests. However, we did not see any disadvantage in recirculating. You may have some cases where the recirculation can cause the uranium to reprecipitate. The other thing is, of course, that in heap leaching you should try to use a liquid to solid ratio of 1:1. The weight of solution produced should be approximately equal to the weight of the core.

H. MOVASAGHI: How well do the results of the laboratory heap leaching tests correlate with the results from large scale heaps?

D.C. SEIDEL: The results from the laboratory column are the same as those from the 2500 tonne heap. An 18 ft bed in the columns and an 18 ft heap produced the same leach solutions in the same amount of time and with the same amount of reagents.

M. LYAUDET: How much uranium is now being obtained in the United States by in situ leaching?

D.C. SEIDEL: About 2 million pounds of U_3O_8 per year. It is now becoming a larger proportion of the total because production by conventional milling has decreased substantially. In 1980 there were about 24 mills and at the present time there are about 3 or 4 left. Some in situ leaching operations can continue because they have long term sales contracts and they can still produce uranium at a profit. In some cases the economics of continued operation at reduced output versus the costs of shut-down and restoration may be a factor.

THE CHEMWES URANIUM PLANT: A CASE HISTORY

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Abstract

THE CHEMWES URANIUM PLANT: A CASE HISTORY.

During the 1970s when the Nuexco exchange value for U_3O_8 rose from US \$6 to \$43 per pound, the recovery of uranium from even comparatively low grade deposits appeared to be attractive. Two mines in the Klerksdorp area of the Republic of South Africa, Stilfontein and Buffelsfontein, had been stockpiling uranium bearing tailings material since the early 1960s, and initial sampling of these and other smaller sources of residue in the area suggested that the establishment of a central uranium beneficiation plant to process such material would be economically feasible. Preliminary studies showed that the uranium content of the tailings could not be economically concentrated before leaching, but that the pyrite in the plant tailings could possibly be concentrated by flotation, with subsequent roasting to provide both the acid needed in the uranium dissolution process and a calcine product from which gold could be recovered. A preliminary feasibility study suggested that an operation of 270 kt per month would be the most attractive in economic terms. It was decided that a contract for the expected production should be negotiated so that this security could be used to support the financing of the project. The paper gives a description of the performance of the plant so far. The plant performance is analysed from the processing and the mechanical points of view, with special emphasis on the leaching, solid-liquid separation, recovery, and purification sections. The criteria used in the initial selection of the process are reviewed and compared with the subsequent performance of the plant.

1. INTRODUCTION

Following the favourable evaluation of the project, the Chemwes uranium plant was designed and constructed within 15 months. The plant was commissioned during June 1979, and producing at design capacity 4 months later. Viljoen et al. [1] highlight the procedures and methods used to bring the plant into production, including financing, project management, design, construction, and training of personnel. It was concluded that the successful startup of the plant could be attributed largely to the considerable effort and conceptual thought that went into the feasibility studies, including the associated comprehensive enquiry document and early freezing of

drawings, the excellent rapport and commitment that existed between the team members from General Mining Union Corporation (Gencor) Ltd and the contractor, and the clearly defined lines of responsibility, the use of senior staff and key personnel for plant management and operation from the inception of the project, and the early recruitment and thorough training of personnel before commissioning of the plant.

In the 6 years since startup, over 20 million tonnes of tailings have been treated, and about 3500 tonnes of uranium oxide have been produced. Nonetheless, there has been continual evolution of the plant, making it more reliable and more cost efficient. In the following sections, these developments are discussed in some detail from the mechanical, the processing and the economic points of view.

2. MECHANICAL PERFORMANCE OF THE PLANT

2.1. Reclamation

Between 120 and 170 kt were reclaimed from the Buffelsfontein dam each month between startup in July 1979 and the end of 1982, when it was phased out over 14 months. Reclamation was by high pressure water monitoring and slurry pumping. This method worked very successfully, and only two significant problem areas were noted. Firstly, the Buffelsfontein dam contains large stones, and the horizontal launder, which has a 16 mm punched plate screen, could not cope. The basket system for removal of the oversize was replaced by a continuous conveyor, and the bottom 1 m of the dam, which contains most of the coarser material, was left behind. This alleviated the problem. Secondly, the Dezurick valves on the discharge side of the three-stage slurry pump wore very rapidly and did not seal after a short period of operation. The fitting of a softer rubber lining did not help, and 'pinch' valves were installed in January 1981. These valves proved to be the answer for this application, as well as for the repulper and leaching tank discharge lines.

Stilfontein slimes dam No. 3 was reclaimed from startup until the end of 1984 by use of a bucket wheel excavator and a conveyor system. The excavator had an availability of better than 90% once the dam starts drying out. This method of reclamation is more expensive than monitoring, but it has the advantage of providing a relatively dry feed that can be combined with the monitored feed to provide a feed of controlled pulp density to the plant. In addition, the overall higher density has the advantage that the reagent consumption is reduced, whereas the residence time in the leach as well as the filter duty are increased.

The combined reclaimed slime is fed into one of two repulpers. The repulpers are brick lined and originally used two 90 hp agitators (four blades, 45° pitch) for mixing. This proved inadequate, and a third, smaller agitator was installed later. The original discharge pumps were undersized and required a high level of maintenance (broken shafts) because they were forced to operate at the limit of their capacity.

Eventually these pumps were replaced by larger ones, and the problem was solved. Variable DC drives control the flow rate of the pulp to the three holding tanks by measurement of the pulp level in the small surge tanks preceding the pumps on the plant.

2.2. Leaching

The pulp is pumped from the holding Pachucas to one of the two continuous leach Pachucas, where sulphuric acid, pyrolusite, and steam are added. From there it overflows via a launder system to one or more of the six batch leach pachucas. (The Pachucas have a length of 23 m and a dia of 11 m). All the Pachucas have flat bottoms and a central draft tube. Air agitation to each tank is supplied through seven one-way air ports located at the base of the tank. Several port designs were tried before ports with a reasonable life were found. This form of agitation has not proved entirely satisfactory, since the holding and continuous tanks need to be dug out every 2 to 3 months, and there are frequent blockages of the air ports with grit. This problem was more prevalent when the relatively coarse material from Buffelsfontein dam was being treated. There are three compressors, each of which supplies 12 500 ft³ of air, per minute.

Steam was originally added from the bottom of the tank. However, after the pulp had managed to pass backwards through the 'non return' valves to the boilers on several occasions, this method of steam addition was abandoned in favour of the conventional top addition method. On average, one of the three 6 in stainless steel feed pipes is replaced every 3 months².

The pyrolusite addition is via a splitter box, which is controlled by a redox signal from a redox probe, and the rate at which the acid is added is controlled by a system based on conductivity. The ring main system installed initially included the ion exchange (IX) section, but this had to be changed because of the high rate of pipe wear before an adequate supply of acid became available to the leach. Another improvement to the ring main system was the incorporation of a stainless steel, magnetically coupled, sealless pump in place of the original cast iron pumps, which required high maintenance and were subject to long periods of downtime.

At the end of the leaching period, the pulp is discharged and pumped to one of two Pachucas, from where it is fed to the filters. Several types of leach discharge valves were tried, and the one that has proved most successful is a pinch valve lined with high density polyethylene (HDPE).

2.3. Filtration

The leached slurry is pumped from the 'filter feed' Pachuca to nine mechanically agitated vessels where the flocculant is added, and the slurry is then distributed

¹ 1 ft = 0.3048 m.

² 1 in = 25.4 mm.

to the double belt filters. The slurry to each flocculator flows through a control valve activated by a device using ultrasound to measure the level of the slurry on the belt. Depending on the overall rate at which slurry is required, the filter feed pump varies its speed so as to maintain a preselected level in a recycle leg.

The filter cloth rests on top of each belt, and the belts themselves are supported on a water cushion and slide on fixed vacuum boxes. For the first 3 years of operation, numerous mechanical problems were experienced with the belt filters, although the filter availability and the running time as a percentage of availability had reached 95% and 90% respectively after the first year.

The major fault, which was the cause of other problems, was excessive friction between the wear belts and wear strips that form a moving seal between the vacuum boxes and the filter belt and cloth. At first, the friction appeared to be due to inadequate lubrication, but this was found to be only part of the problem, the main difficulty being that materials with low friction and wear characteristics are not easily found. Originally, the wear belt and the wear strip were both made of polytetrafluoroethylene (PTFE). These were unsatisfactory, as were those made of polyethylene alone and of polyethylene with PTFE inlays. The life of the wear strips improved from about 2000 h when the cloth tracking improved, but real advances were made only in 1982, when HDPE wear strips on polyester rubber wear belts were tried. This system was fitted to all the filters in early 1983, and lives of up to 10 000 h were achieved. In the meantime, however, the high friction resulted in excessive torque, which caused gear box and motor failures.

Another serious problem, which lasted until late 1983, was the frequent failure of the rollers returning the belt. This not only resulted in downtime but in damage to the filter cloths. Before this problem was solved, by the elimination of all welds and the use of a single pipe of 200 mm diameter to carry the rubber rollers that support the belt, cloth life was less than 80 d; afterwards it increased to about 140 d.

Although the first cloth tried had a fine aperture that kept the filtrate clear, it was not stiff enough to retain its shape. It folded inwards at the edges, and not only allowed the slurry to flow into the filtrate receivers but resulted in tracking difficulties. These cloths were replaced at an early stage by 'needlefelt' cloths, which have the additional advantage that they can be welded easily although it took some time and effort for the plant personnel to perfect the technique. The slightly 'dirty' filtrate from the needlefelt is tolerated by the NIMCIX system but, because the needlefelt cloth has a tendency to stretch, it must be shortened and rejoined at regular intervals.

When the plant was built, 150 kW liquid ring vacuum pumps were installed. However, by late 1981, after several motors had burned out, those pumps were replaced with 185 kW pumps, which have performed well. A problem that has not been solved as yet is the high wear rate on the filter feed control valves. Even the use of the robust pinch valve has not solved the problem.

2.4. Continuous ion exchange (CIX)

The design, commissioning, and performance of the NIMCIX section is described by McIntosh et al. [2].

In contrast to the filter plant, the IX section has run very smoothly except for relatively minor problems relating to the materials of construction. Firstly, the horizontal vibrating screens, which separate the eluant from the resin when the resin is transferred to the top of the loading column, corroded rapidly at the welds, and were replaced by stationary curved screens covered with polyester cloth. Secondly, the support structure of the screens, which act as resin traps when the barren solutions pass through them, corroded rapidly, and resin spillage resulted. The feed chute was modified, and the 0.5 mm wedge wire screen was replaced with a 1.0 mm wedge wire screen. Thirdly, the stainless steel wedge wire retaining screen in the regeneration vessel was replaced by one made of polypropylene because the caustic lye contained a high level of chloride, which accelerated corrosion. Fourthly, the lining in the regeneration vessel needed continual repair. Initially this was thought to be a materials problem, and fibreglass, rubber, and lead linings were tried. However, it was found that a vacuum formed inside the vessel when manual draining was carried out, and that this caused damage to the lining. An air bleed was fitted, and the rubber lining gave no further trouble. Fifthly, the only solution to the problem of corrosion in the eluant make-up tank was the insertion of sacrificial pieces. Finally, the strength of the top tray in the elution column was found to be inadequate, and the tray was eventually replaced by a polyvinylchloride (PVC) tray of 25 mm thickness.

2.5. Solvent extraction and product recovery

Very few problems occurred in these sections, but some improvements were introduced. The main improvement related to the control of pH in the strip mixer-settlers. The popular Lazarin pH probe was rapidly fouled by the organic material, and control was lost. A novel device was designed to extract the aqueous-organic mixture, allow it to disengage, and then measure the pH value in the aqueous phase using standard pH electrodes. This device has functioned well, and has been automated. Thus, it provides continuous pH measurement.

The ammonium hydroxide feed pumps were initially very high maintenance items. The connecting rods broke frequently because the pumps were pumping against a closed valve, and the expensive neoprene lip seals (imported from France) had to be replaced regularly. The neoprene was successfully replaced with locally available PTFE, and the metering pumps were replaced by pumps with a variable stroke that could respond to a signal from the pH controller.

The ammonium hydroxide make-up system often clogged up but, since the glass balls were replaced with a double plastic ring, the system has operated without difficulty provided an acid wash is done about once in 3 weeks to prevent scaling.

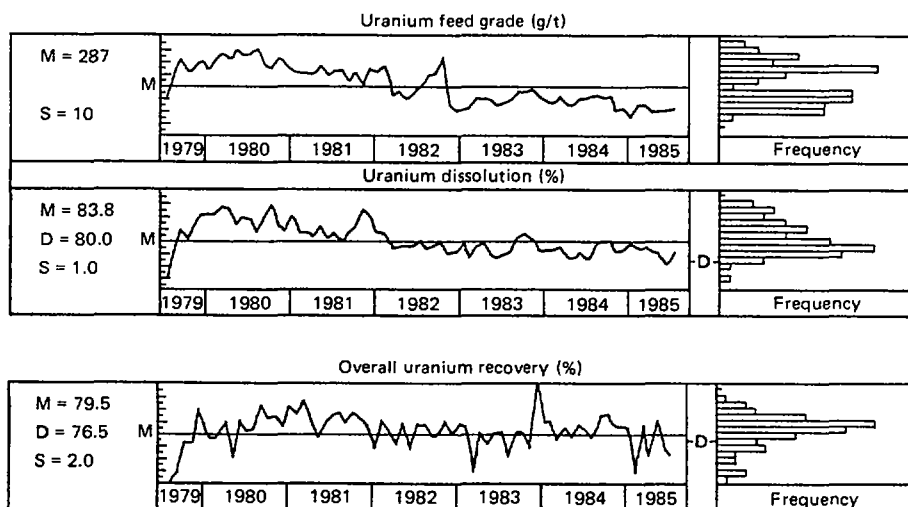


FIG. 1. Selected monthly process performance figures. M: mean value; D: design value; S: one deviation on vertical axis.

A recurring problem is the variable settling rate of the ammonium diuranate (ADU). With slow settling ADU, the thickener cannot cope and ADU is carried over into the ammonium sulphate holding tank. The consequence of this is not serious and, apart from intermittent recovery of the ADU from this tank, no further action has been considered necessary.

3. PROCESS PERFORMANCE OF THE PLANT

The performance of the plant was assessed by examination of the operating records, the monthly figures for 281 recorded process variables being extracted and analysed for the 73 months from July 1979 to July 1985. The daily figures for 41 variables were extracted and analysed for twelve randomly chosen metallurgical months, consisting of 366 days in all.

The following statistics were determined for all 322 process variables: the mean, the standard deviation, the median, the 10th and 90th percentiles, the minima and maxima and the monthly averages. Many of the monthly figures were plotted as a function of time. These plots were repeated, 6- and 12-monthly moving averages being used to smooth out the data. Selected variables were correlated with each other in an attempt to determine whether any relationships existed. Finally key design values were compared with the actual plant figures achieved during the 6 years of operation. Some of these are plotted as a function of time and as a frequency distribution in Fig. 1. A simplified schematic flow diagram and uranium mass balance is presented in Fig. 2.

3.1. Reclamation

Although ore was fed from five different sources in various combinations, the overall tonnage treated remained fairly constant at around 290 kt per month until 1985, when it was reduced to about 180 kt per month by the operation of two shifts per day (Fig. 3). On the other hand, the uranium grade of the feed decreased fairly steadily over the years from a maximum of about 0.25 kg/t in 1980 to 0.16 kg/t in 1985. This decline can be attributed to the exhaustion of the relatively higher grade material from the Buffelsfontein dam.

3.2. Leaching

The leaching section performs very satisfactorily in that it has always achieved better than the 80.3% target value for dissolution when measured on a monthly basis. This target has been met for over 90% of the time when measured on a daily basis and, on average, the dissolution has been 83.8%. What makes this even more satisfactory is that, on average, the sulphuric acid and manganese dioxide consumptions up to November 1982 were below the target values of 25 and 4 kg/t respectively.

The leaching parameters of time, temperature, and additions of sulphuric acid and manganese dioxide were optimized by use of the results of laboratory tests and incremental cost-benefit analyses to yield an acceptable internal rate of return. It was concluded that leaching for 24 h at 60°C with the reagent additions indicated in the previous paragraph would be economical. In the event, the leaching temperature averaged 55°C in the continuous leach Pachucas and 49°C in the batch leach Pachucas, and the leaching time averaged only about 20 h due mainly to the higher than design tonnage treated. However, since the beginning of 1985, when the tonnage was reduced, the average temperature and leaching time have increased by 4°C and 12 h respectively. This is reflected by a slight decrease in the residual uranium grade, although this may be partly attributable to the treatment of the Stilfontein slimes dam No. 2, which started at that time. It is of interest that, even under the actual leaching conditions, the laboratory data would predict a uranium dissolution of better than 88%. This dissolution was achieved fairly regularly up until early 1981 but very seldom thereafter. The explanation for this may be the lower uranium feed grades and that the slimes further below the surface of the Buffelsfontein dam were less oxidized and were therefore less amenable to leaching.

During 1982, the leaching conditions were re-examined. Further laboratory test work was done on Stilfontein material, and a mathematical model was developed that could predict the dissolution during the leach as a function of the leaching temperature and reagent additions. Lime and coal consumptions were calculated as functions of the acid addition and temperature respectively. Good agreement was found between the predictions of the model and previous plant performance. The optimum 'economic' operating conditions were determined by use of this model, and it was found that, if the acid and manganese dioxide additions were reduced to 12 and

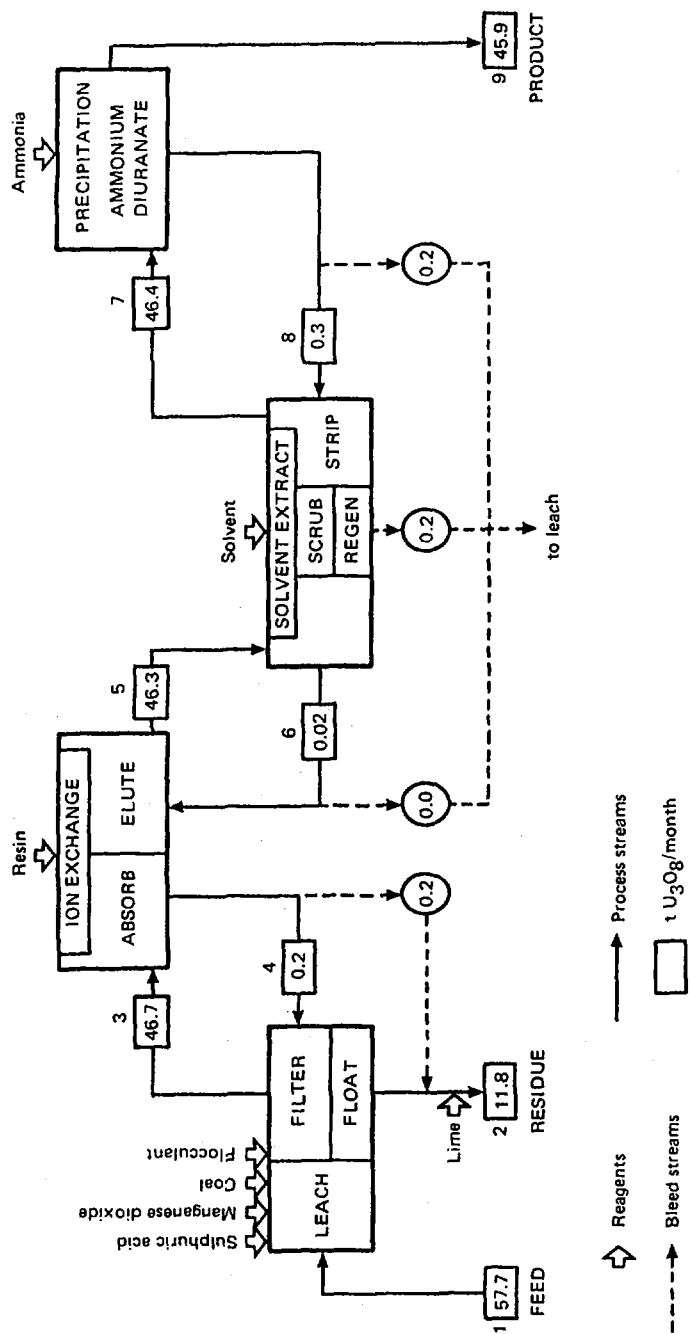


FIG. 2. Schematic flow diagram and uranium mass balance.

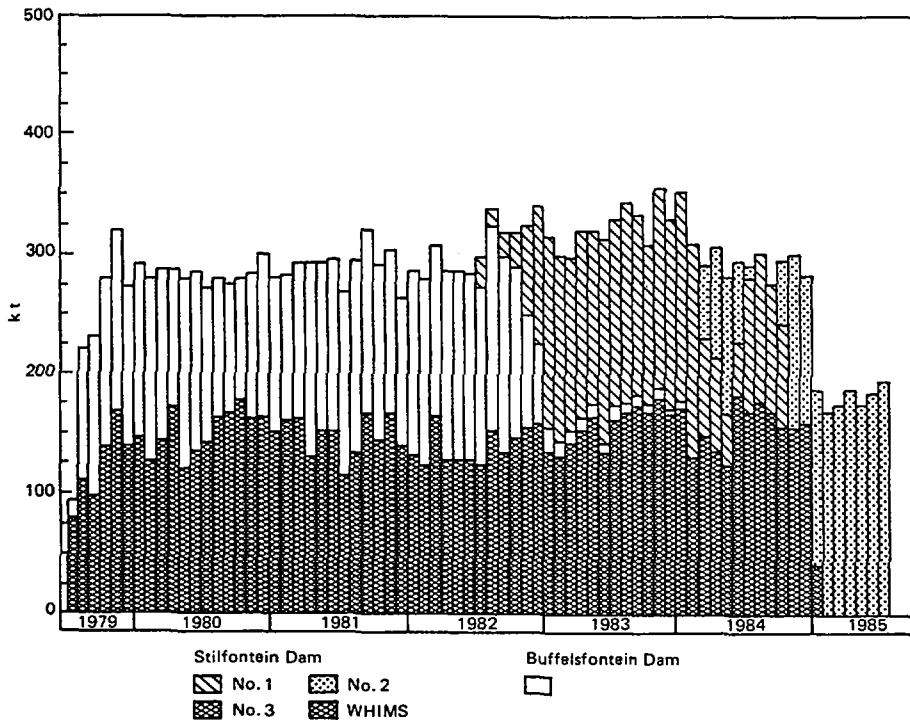


FIG. 3. Total slimes treated per month.

2.0 kg/t respectively, the profitability of the operation would increase. The additions of acid and manganese dioxide were reduced to 12.8 and 1.8 kg/t respectively and the uranium dissolution dropped subsequently from 85.1 to 81.7% and the lime consumption from 11.5 to 8.8 kg/t. This is in very close agreement with the prediction of the model.

It is perhaps unfortunate that the 'economic' leach was introduced at almost the same time as the Buffelsfontein dam was exhausted, because this masked any correlation that might otherwise have been apparent.

3.3. Filtration

In the preliminary feasibility study, settling tests conducted in the laboratory indicated that the use of a five stage countercurrent decantation (CCD) circuit with a wash ratio of 2 could be expected to result in a loss of uranium in solution (soluble loss) of 2%. However, as the test work proceeded, it was found that there was a tremendous variation in the settling rate. The settling area required to cope with the fine material made the cost of a CCD circuit prohibitive. Rotary drum filtration was unsuitable for the very fine material and belt filtration seemed to present the best

alternative. Calculations showed that a soluble loss of 2% would occur if eleven belt filters of 93 m² were used. However, an economic evaluation showed that the extra filter area required for a decrease in the soluble loss from 3 to 2% was unjustified, and nine belt filters of 93 m² were installed, although the foundations had been constructed for eleven filters.

In operation, the belt filters have had an availability of 93%, but have been used for only 88% of the time available. This is equivalent to a usage of 82%. On this basis, an overall duty of (dry) 13.6 t·d⁻¹·m⁻² has been achieved, and this is slightly higher than the design of 13. t·d⁻¹·m⁻².

The filter area is divided into dewatering (21%), washing (63%), and drying (16%) zones. This allocation of the filter area was evolved after much effort. In particular, the dewatering zone was increased, and the plan for two-stage washing was changed to single-stage washing. In this configuration, the dewatering and washing duties were calculated as 1.50 and 0.41 m³·h⁻¹·m⁻² respectively, and the average loss of soluble uranium as 3.9%. The belts are run at 15 m/min and this allows a 10 mm cake to form.

The wash ratio varies considerably about the mean of 1.33 and sometimes drops below 1 (15% of the time) due to the treatment of very fine material with poor filterability. This makes for poor displacement of the uranium and, hence, high soluble losses. Two stages of washing would help considerably, but this would require either a 70% increase in filter area or a decrease in the tonnage treated which would be difficult to justify economically. Despite this, soluble losses were contained to under 4% about half of the time and under 6 per cent for 90% of the time.

Correlation of the filtration data over the 6 years (at the 99.9% confidence level) shows that increase of the percentage of smaller than 200 μm material or of the flocculant dosage causes the soluble loss to increase. The former cause is not unexpected, but the latter is a little surprising since the average flocculant addition of 66 g/t is well below the design figure of 100 g/t. The explanation for this is that, when larger (than normal) quantities of the fine material are fed to the plant, larger (than normal) quantities of flocculant are also added to enable the plant to treat the required tonnage. This is shown by the high correlation between the flocculant dosage and the percentage of material smaller than 200 μm.

3.4. Ion exchange

In the preliminary feasibility study, IX followed by solvent extraction (Bufflex) and solvent extraction alone (Purlex) were both considered for recovery of the uranium. A detailed cost analysis was done, in which it was assumed that the wash ratio in the solid-liquid separation stage was between 1 and 3. This showed that, for this low grade ore, the Bufflex route was preferable at all wash ratios.

Several types of IX contactors were considered, and the NIMCIX system was chosen primarily because technical backup was available locally. This proved to be a good decision; the twin NIMCIX systems started up without difficulty and have performed well ever since.

The uranium concentration in the barren solution over the 6 years averaged 0.001 g/L, which is half the design figure of 0.002 g/L. In the first 6 months of operation, the uranium concentration was about 0.005 g/L. It then dropped to 0.0012 g/L until 1983, when it dropped even further to about 0.0004 g/L. The first drop was due to more stable operation, and the second to the introduction of the 'economic' leach and the lower sulphuric acid and Fe(III) levels in the pregnant solution (3.0 to 2.0 g/L and 0.7 to 0.5 g/L respectively); the thiosulphate concentration also dropped significantly (0.0012 to 0.0003 g/L). A small rise in the uranium concentration in the barren solution was apparent during 1985 after the change-over to two shifts per day. Daily startup of the columns makes stable operation difficult.

The uranium concentration on the loaded resin averaged 24.3 g/L (design 25.0 g/L) and that on the eluted resin 0.3 g/L (design 1.0 g/L). The fairly high sulphuric acid concentration in the fresh eluant (120 g/L) and the faster flow rate account for the excellent elution of the resin and lower concentration of uranium in the eluate (4.0 g/L as opposed to the design value of 5.0 g/L).

Correlation of the monthly figures shows that higher than average levels of sulphuric acid, Fe(III), and uranium in the pregnant solution raise the uranium concentration in the barren solution. The importance of good elution is also strongly reflected in the correlations.

The original plan was regeneration of the resin to remove silica every fifth cycle. However, this frequency was reduced steadily; at present, only one batch in twenty is regenerated. In the first 2 years, the silica levels were around 0.5% but, since then, they have been kept to below 0.35% before regeneration and reduced to 0.15% after regeneration. However, this caustic regeneration step has not been effective in controlling the anionic cobalt on the resin. The resin has been progressively poisoned by the cobalt, and the current loading of cobalt is 0.5% with no apparent effect on resin performance. Thiosulphate and cyanide ions appear to be effectively controlled by the sulphuric acid elution, because very little, if any, removal occurs during regeneration, and there has been no buildup over the years. The cyanide concentration in the pregnant solution is typically about 0.003 g/L but, during the treatment of the concentrate obtained by wet high intensity magnetic separation (WHIMS) of the current tailings from the Stilfontein Gold Mine, it rose regularly to more than ten times this value.

3.5. Solvent extraction and product recovery

Laboratory tests indicated that three extraction stages would be sufficient to remove the U_3O_8 (5 g/L) in the concentrated eluate and to produce a raffinate containing 0.001 g of U_3O_8 per litre. Four extraction stages were installed to give the operation some flexibility and, after the first 18 months, the uranium concentration in the raffinate averaged 0.0003 g/L. Cobalt (52%), chloride (38%), ferric ion (24%), and silica (13%) are all extracted to different extents by the solvent. In addition, the concentration of cobalt in the concentrated eluate increased from 0.2 to

TABLE I. STATISTICS OF INDEXED OPERATING COSTS
(based on monthly average costs per tonne treated)

Description	Mean standard deviation	1979	1980	1981	Six-monthly averages						Min.	Max.	Percentiles						
					1982	1983	1984	1985	10th	50th			90th						
Overall costs																			
Labour	16	5	15	12	13	13	15	15	14	15	14	16	18	27	10	30	12	15	25
Stores/Material	61	11	44	49	59	7	7	8	9	9	10	12	12	10	6	21	7	9	16
Power	10	3	9	7	7	7	8	9	9	10	10	12	12	10	6	21	7	9	16
Water	2	5	10	1	2	2	2	2	2	2	1	1	0	1	2	0	41	0	2
Other	12	5	18	9	8	9	8	10	8	11	12	11	14	23	0	27	8	10	21
Total	100	19	95	77	89	96	103	113	92	95	94	98	101	141	65	156	78	95	134
Reagent costs (42% of overall costs)																			
Sulphuric acid	38	9	23	30	36	29	34	48	41	41	35	49	49	43	13	52	25	39	49
Manganese dioxide	9	5	6	9	13	17	19	13	10	7	7	6	4	5	2	22	4	8	17
Coal	12	3	14	13	13	12	15	11	12	6	13	12	11	10	2	22	7	12	15
Lime	28	9	13	18	25	35	33	41	32	26	31	29	34	27	6	46	16	29	39
Flocculant	5	1	5	4	4	6	7	8	6	5	5	5	5	4	3	10	4	5	7
Resin	1	1	4	4	0	2	0	1	0	1	0	0	0	0	0	4	0	0	4
Solvent	2	1	1	0	1	3	2	2	2	2	1	2	1	2	0	9	0	1	3
Ammonia	3	1	3	3	3	3	3	3	3	3	3	3	3	2	2	6	2	3	3
Other	2	1	1	4	2	2	1	2	2	2	1	2	1	2	0	5	1	2	3
Total	100	18	70	84	96	109	113	128	108	93	97	108	109	95	65	134	71	100	125

Description	Mean standard deviation	1979	1980	1981	Six-monthly averages			Percentiles										
		1979	1980	1981	1982	1983	1984	1985	Min.	Max.	10th	50th	90th					
Material costs (10% of overall costs)																		
Reclaim	14	8	11	20	15	19	25	12	10	14	12	9	15	3	35	6	13	27
Leach	19	9	15	19	16	12	19	16	27	16	20	15	42	4	56	10	17	30
Filter	21	8	31	23	20	24	18	16	17	24	22	18	29	3	40	11	20	33
Recovery	11	4	11	9	10	11	13	11	11	10	9	13	14	1	23	5	11	16
Disposal	4	3	2	2	2	5	6	3	3	3	6	7	9	0	16	1	3	9
Services	14	6	9	15	14	13	14	13	17	12	14	13	23	3	34	7	13	21
Other	17	7	21	22	19	16	22	14	12	11	15	16	26	4	47	9	16	25
Total	100	27	101	111	96	100	117	85	97	91	99	91	157	36	196	71	100	130

0.4 g/L when the concentration of acid in the fresh eluant was increased from 112 to 126 g/L in mid-1982.

The solvent loading and stripping sections run, on average, exactly as designed, the U_3O_8 concentration on the loaded, stripped, and fresh solvent being 4.0, 0.008, and 0.001 g/L respectively. However, until mid-1981, the control of pH in the strip section was erratic and the uranium stripping efficiency was below average. The pH profile in the strip section averaged 3.9, 4.3, 4.7, and 5.4, but there was a tendency for the plant to operate at lower pH values while pH control was poor.

Two scrubbing stages are used to remove bisulphate and to reduce the consumption of ammonia. The organic to aqueous ratio in the scrubbing stages is 5. The scrub liquor also strips some uranium, and is fed back into the extraction stages.

The stripped solvent is passed to a single regeneration stage, where it is treated with ammonium hydroxide and 10% sodium carbonate at an average pH value of 9.6. The intention is that this pH value should be kept as high as possible but below 12, where emulsions form. Originally, sodium hydroxide was used for pH control, but the use of ammonium hydroxide was found more convenient, although this limits the maximum pH value attainable.

The 23.4% solution of ammonium sulphate that is used to strip the solvent loads to 7.0 g/L and is then fed to the ADU precipitator where the pH value is adjusted to 7.4 while the temperature is controlled at 30°C. The pregnant liquor has a mean residence time of 80 min before flowing on to the ADU thickener. Initially difficulties were experienced in the production of an ADU cake within the sulphate specification of 3%, so three stages of water washing and filtration are used to keep the sulphate concentration in the ADU product within acceptable limits. The root of the problem lies in the precipitation conditions, and it is thought that, under certain conditions, a uranyl sulphate is formed that increased the sulphate content. Correlation of plant performance shows that the higher the pH value, the purer is the product, and that small variations in temperature, ammonium sulphate concentration, and pregnant liquor flow rate (residence time) make no significant difference.

4. ECONOMIC PERFORMANCE OF THE PLANT

The monthly operating costs of the plant (excluding the flotation section) were analysed in a similar fashion to those for process performance, the overall costs being broken down as labour, stores, power, water, and other costs. Additional detail on the 'stores' figure is provided. The reagents presented include sulphuric acid, manganese dioxide, coal, lime, flocculant, resin, solvent, and other. The 'material' costs are broken down by plant section. All these costs were indexed and are presented in Table I, which shows the relative proportions and the relative rates of increase. This information is illustrated in Fig. 4 for the overall plant operating costs.

Some 61% of the total operating costs can be allocated to stores and material, of which reagent costs account for about 42%. Reagent costs are dominated by the

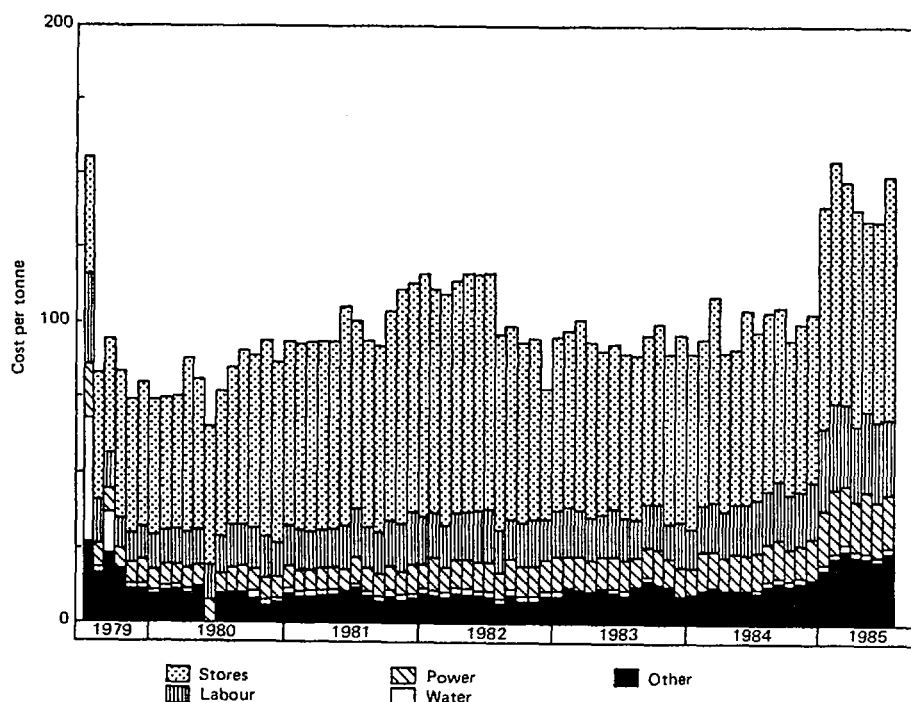


FIG. 4. Indexed monthly operating costs per tonne treated.

costs of the lixiviant (sulphuric acid) and lime required for dissolution of the uranium and neutralization of the tailings respectively.

It is this 42% of overall costs, represented by reagent consumption, that presents the most likely area for cost control. For example, reduced additions of lixiviant can effect savings in sulphuric acid and lime. Such savings outweigh the economic disadvantages of a lower uranium recovery.

The contribution towards overall cost control made by savings in stores consumption, in particular reagent consumption, can be seen from Fig. 4. It is noteworthy that, as the overall costs increased, in indexed terms, from 95 in 1979 to 141 in 1985, the reagent costs, having reached a peak of 128 in 1981, reduced to 95 in 1985, a figure that compares favourably with the figure of 70 for 1979. This contrasts with the labour and power costs, which doubled during that period.

The original feasibility study indicated that a plant treating 270 kt per month was economically the most attractive, and that the capital cost of the final plant would be in the vicinity of 68 million rands [1]. The final capital cost of the uranium producing section of the plant fell within the range of R170 to R190 per tonne treated per month. This was within budget and the results of efficient project management.

The distribution of capital costs, in terms of operating areas and cost centres, is shown in Table II. In terms of plant area, the relatively high percentage of total

TABLE II. DISTRIBUTION OF CAPITAL COSTS IN TERMS OF PLANT AREAS AND COST CENTRES
(values are expressed as percentages)

Cost centres	Plant areas						Total
	Reclamation	Leach	Filter plant	Recovery	Neutralization	Services	
Civils	2.22	2.16	1.99	2.07	0.23	8.08	19.49
Steelwork	0.70	0.25	0.89	0.62	0.14	1.56	4.23
Platework	0.66	1.58	0.89	1.21	0.26	0.40	5.00
Material handling	1.02	0.19	0.51	0.67	0.31	0.64	3.35
Process plant	0.20	0.60	7.39	1.79	0.00	3.71	13.15
Machinery, etc.	6.61	0.01	1.63	0.02	0.10	3.98	12.89
Electricals	2.02	1.17	0.69	0.38	0.44	4.53	9.51
Instruments	0.21	0.46	0.66	1.14	0.08	0.35	2.94
Piping	4.61	1.03	2.81	3.59	1.05	2.27	15.63
Special materials	0.53	0.75	0.71	1.87	0.11	0.66	4.66
Other	6.10	0.00	0.00	0.00	0.00	0.00	9.14
Total	24.82	7.65	18.17	13.38	2.74	26.20	100.0

capital required by the filter plant is noteworthy. Foundations, buildings, structures and insulation (civils) and piping accounted for 35% of the total costs, which emphasizes the important contribution to capital costs of essentially infrastructural components.

So far, the uranium recovery section of the plant appears to have provided an acceptable return on investment in its own right. In addition, the total earnings are enhanced by the gold recovery section which represents a relatively modest outlay in additional capital and operating costs to provide a high value product.

5. SUMMARY AND CONCLUSIONS

- (1) The performance of the plant has been a success in every respect: mechanically, operationally, and economically.
- (2) The mechanical performance of the plant has been marked by continual improvements to the original plant, examples of which include:
 - (i) The installation of robust 'pinch' valves at all high wear points;
 - (ii) The design of new HDPE one-way air ports for the injection of air to provide agitation in the leach;
 - (iii) The introduction of HDPE wear strips on polyester rubber wear belts on the filters;
 - (iv) The redesign of the belt support rollers on the filters;
 - (v) The elimination of high corrosion areas in the IX plant;
 - (vi) The development of a novel method for the measurement of pH in the stripping section of the solvent extraction plant;
 - (vii) The improvement of the method for control of the feed rate of ammonium hydroxide to the stripping section.
- (3) The process performance of the plant has been carefully analysed and, in most sections, plant performance exceeds design prediction, as follows:
 - (a) The combined feed from the bucket wheel excavator and water monitor provides 283 kt per month at a relative density of 1500 kg/m³.
 - (b) Dissolution in the leach is consistently good, the consumption of reagents being below design prediction.
 - (c) The belt filter plant is not able to contain the soluble losses within design limits due to the inadequate area available for washing. Additional filters to reduce the soluble losses cannot be economically justified.
 - (d) The NIMCIX plant performs extremely well, its efficiency being better than 99% cent on average. Very little resin make-up is required and, apart from a buildup of cobalt, the resin appears to be in good chemical condition.
 - (e) The solvent plant runs exactly as designed, and the stripping efficiency improved after better control of pH was achieved.
 - (f) The product recovery section requires close control within specification.

- (4) The economic performance of the plant has been more than satisfactory:
- (a) The original capital outlay was within budget.
 - (b) The operating costs have been contained to below the rate of inflation, primarily due to savings in reagent usage.
 - (c) The uranium production of the plant has provided an acceptable rate of return on investment, and this has been enhanced by the recovery of some gold as a by-product.

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REFERENCES

- [1] VILJOEN, E.B., TAYLOR, J.L., ROBINSON, J.E., BLUHM, B.J., LE GRANGE, P., PILKINGTON, W., The evaluation, design and construction of the uranium plant for Chemwes Limited, J.S. Afr. Inst. Min. Metall **81** 9 (1981) 280-288.
- [2] McINTOSH, A.M., VILJOEN, E.B., CRAIG, W.M., TAYLOR, J.L., The design, commissioning, and performance of the NIMCIX section of the Chemwes uranium plant, J.S. Afr. Inst. Min. Metall. **82** 7 (1982) 177-185.

DISCUSSION

W. FLÖTER: In your summary you stated the main philosophy for the improvements in the third generation uranium plants in South Africa. In general, would you say that there is a main guiding idea or a certain tendency in modern process development and design for uranium mills, or are there only evolutionary improvements of old processes?

M.A. FORD: This is a difficult question to answer. I believe that the main guiding principle is to make a profit and, therefore, to use whatever technology will maximize those profits.

W. FLÖTER: You have very big filters. What is the problem there? Why do you need such a large filtration area?

M.A. FORD: There are some very fine materials in the dumps which are difficult to filter. When treating these materials one finds that the filter duty decreases and in order to get the tonnage through a lower wash ratio a higher soluble loss must be tolerated. What was originally meant to be a two-stage countercurrent washing system had to be changed to single-stage washing.

M.C. CAMPBELL: If you were to build the plant today, would you still use those very large belt filters, or would you use something different?

M.A. FORD: I asked that same question of the man who actually made the decision. The alternatives to belt filters were, first, CCD, but it could not handle the very fine material except by using very large thickening areas which would be uneconomic and, second, drum filters, which are well known in the South African gold industry but which are not suited for ultra-fine materials and therefore belt filters were chosen. There were numerous mechanical problems during the first two years of operation of the plant and the engineer would have probably liked to have changed his mind if he could. But now that these problems have been solved he believes the belt filters are the best alternative.

M.C. CAMPBELL: Did you consider using an IX system that could tolerate high amounts of suspended solids?

M.A. FORD: The NIMCIX system can in fact cope with high amounts of suspended solids. The original belt filter cloth that was used was very fine and it could remove all the solids. But it was not stiff enough to keep its shape. It tended to fold over, allowing the vacuum to be lost. The 'needlefelt' cloth presently in use keeps its shape but produces a dirty filtrate. However, the NIMCIX system can handle that without difficulty.

M.C. CAMPBELL: Could the NIMCIX system handle the overflow from a CCD circuit?

M.A. FORD: Yes. The small percentage of fine material in a CCD overflow does not interfere with the operation of the NIMCIX column. In fact the NIMCIX absorption columns at Vaal Reefs South are fed by a CCD circuit.

A.T. TRIGGS: Where do you think that the Chemwes plant would fit in the table of operating costs, compared to other South African plants.

M.A. FORD: I do not know the operating costs of all the South African plants. In relative terms, however, it could be at the lower end of the scale because reagent consumption at Chemwes has been reduced to about half of what it was two years ago.

H.J. STEINER: What is the particle size distribution in the feed?

M.A. FORD: Between 70 and 80% of the ore is finer than 75 μm .

H.J. STEINER: And in the finer size range, say, below 10 μm ?

M.A. FORD: I do not have that figure. I am sorry.

B. SCHNEIDER: What is the capacity of the sulphuric acid plant at the site?

M.A. FORD: Chemwes does not produce its own acid.

B. SCHNEIDER: You said that you have attained the expected recovery of uranium. Did you also attain the expected recovery of gold?

M.A. FORD: Chemwes does not produce gold except as a minor by-product. The gold is associated with the pyrite. The tailings from the uranium plant are floated to recover the pyrite which is then treated to recover the gold, but it is not treated at the Chemwes site.

Annex

**NATIONAL REPORTS ON THE CURRENT STATUS
OF URANIUM ORE PROCESSING**

CANADIAN DEVELOPMENTS OF METALLURGICAL PROJECTS IN URANIUM

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Canada has substantial uranium resources. The official resource estimates in tonnes of recoverable uranium are reported as:

Measured	54 000
Indicated	233 000
Inferred	264 000
<i>Total</i>	551 000

In 1984, production was 11 170 t uranium from five operating plants:

	(t uranium)
Cluff Mining	642
Denison Mines Limited	2246
Eldorado Resources Limited	1361
Key Lake Mining Corporation	4003
Rio Algom Limited	2917

In addition, Earth Sciences Extraction Company of Calgary recovered about 50 t uranium from phosphoric acid at Western Cooperative Fertilizers.

Current developments are aimed at improvements in productivity, reduction in costs and increases in revenue. In mining there has been a substantial increase in mechanization in drilling, rock bolting and transportation systems. Manpower training has improved productivity.

Metallurgical developments are under way in improving ion exchange resin performance by reducing fouling and resin loss. Generation of sulphuric acid from pyrite floated from Elliot Lake ore has been evaluated. Acid recycle is being developed to decrease costs of acid and neutralization. Improved efficiency of oxidants is under study with better Eh control, although sodium chlorate remains as the standard oxidant. There is a switch to hydrogen peroxide for yellow cake precipitation away from ammonia and magnesia. Rio Algom Mines is carefully evaluating in-place underground leaching following the Denison lead.

By-product recovery is a potential source of revenue. Denison Mines Limited has signed a contract to sell yttrium and lanthanide oxides to United States and

Japanese companies. Nickel and cobalt recovery from Saskatchewan operations is under consideration.

Environmental improvement continues to be a major target. The National Uranium Tailings Program has been established to develop a mathematical model for predicting the long term (10 000 year) fate of uranium tailings disposal sites. This program has been a major mechanism for improving the knowledge base in tailings disposal. The concept of ecological engineering is being evaluated. This concept recommends the development of a natural buffer zone of plants and other organisms in a wetland to mediate the effects of tailings pond effluents before escape to the environment.

In the current market condition there are few plans for new uranium operations. Plans have to be deferred indefinitely for:

Rio Algom, Milliken/Lachnor/Nordic at Elliot Lake
Saskatchewan Mineral Development Corporation, Dawn Lake Project,
Saskatchewan
Canadian Occidental Petroleum at McClean Lake, Saskatchewan
Consolidated Rexspar at Birch Island, British Columbia
Norcen Energy Resources at Blizzard Project in British Columbia
Brinco Limited, Kitts-Michelin in Newfoundland
Urangesellschaft, Lone Gull Project at Baker Lake in the Northwest
Territories.

The only activity is the Cigar Lake Project in Saskatchewan where annual development expenditures of several million dollars are continuing, particularly to address the mining strategy. Cigar Lake is reported to contain 100 000 t uranium at 12%. The size of this deposit is encouraging continued development.

URANIUM ORE PROCESSING ACTIVITIES IN COLOMBIA

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Uranium exploration and ore processing research and development in Colombia are the responsibility of the Instituto de Asuntos Nucleares (IAN). Within the IAN, the Area de Materias Primas (AMP, Raw Materials Area) is in charge of exploration, geochemistry and petrography, and the Area de Desarrollo de Procesos (ADP, Process Development Area) is in charge of ore processing tests and development. Analytical services are provided by other departments.

ORGANIZATION OF THE PROCESS DEVELOPMENT AREA (ADP)

The basic objective of ADP, created approximately four years ago, is to perform metallurgical ore tests and to select and develop metallurgical processes for treating Colombian ores. A well equipped metallurgical laboratory has been established, a team of qualified metallurgists is being organized and a methodology for ore testing is being developed and implemented.

Activities have been organized according to the main unit operations involved in uranium ore processing, i.e.:

- Sample preparation, crushing, grinding and physical beneficiation
- Leaching and solid-liquid separation
- Ion exchange
- Precipitation

An engineer is responsible for each one of these activity areas.

A new building to house the ADP was erected in 1984. The equipment for the laboratory has been installed and tested using different ore samples. From 1985 IAN has had the capability of evaluating, from the metallurgical point of view, any uranium ore samples provided by the exploration department (AMP).

URANIUM OCCURRENCES IN COLOMBIA

Beginning in 1975 several foreign companies (MINATOME and Cogéma from France and Empresa Nacional del Uranio SA (ENUSA) from Spain) signed co-operation agreements with the Government of Colombia and started exploring for

uranium. After uranium prices decreased sharply (mainly during 1980) these companies lost interest and left the country during the years of 1981 and 1982. The last to leave was ENUSA, in June of 1982. Several uranium occurrences were found during that period, notably those of Berlin, Zapatoca, Contratación, Quetame, California and Caño Negro, but none of them appear to be of economic significance. However, based on general geological information, it has been reported that there is a good possibility that economically attractive uranium occurrences may be discovered in the future.

The uranium occurrences found so far show great variety. Uranium is combined with several other metals such as Cu, Au, V and Mo and with different gangue materials such as quartz, sandstone, phosphates and others. This variety means, of course, that a great variety of treatment processes will be required.

URANIUM PRODUCTION PROJECTS IN MEXICO

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INTRODUCTION

The new Mexican Nuclear Law (Article 27 of the Constitution) was passed on 5 February 1985 and gives the Comisión de Fomento Minero (CFM) responsibility for uranium mining and milling. Accordingly, CFM will continue the activities of the former Uranio Mexicano.

Uranio Mexicano (Uramex), created in 1979, made a general geological survey of the country and quantification of uranium reserves in six regions located in the states of Chihuahua, Durango, Nuevo León, Sonora, San Luis Potosí and Oaxaca.

The principal uranium ore bodies of economic importance identified so far are in the Sierra de Peña Blanca, in the state of Chihuahua, at 29°14'N and 106°05'W, 70 km north of the city of Chihuahua. The main deposits, Margaritas, El Nopal I and Puerto III are within 3 km of this point. In addition, 101 anomalies have been detected by airborne radiometry in a 300 km² area. Seventy per cent of these anomalies show evidence of uranium mineralization.

PEÑA BLANCA

Peña Blanca is the first Mexican project for industrial production of uranium concentrates. The mill, with a design capacity of 600 t U₃O₈ per year, was under construction when Uramex closed down. The project is now on stand-by and will be resumed by CFM.

Uranium mineralization in Peña Blanca occurs in ignimbrites of rhyolitic composition. The uranium bearing mineral species present are tyuyamunite, meta-tyuyamunite and uranophane. Other minerals present include quartz, alunite, hematite and powellite. Calculated reserves are 1 900 000 t ore with an average grade of 0.11% U₃O₈. An underground mine and two open pit mines were being prepared for exploitation.

The metallurgical process as defined by laboratory tests includes the following unit operations:

- (a) Crushing and grinding to minus 65 mesh.

- (b) Acid leaching at low temperature and atmospheric pressure. No oxidant is required. Residence time would be 6 hours with an acid consumption of 38 kg/t ore.
- (c) Solid-liquid separation and washing by countercurrent decantation. Required settling area is 0.7 ft² per tonne per 24 hours with a flocculant consumption of 120 g/t.
- (d) Purification and concentration by solvent extraction with tertiary amine in mixer settlers and stripping with sodium carbonate in order to remove the molybdenum present.
- (e) Precipitation with ammonia.
- (f) The ammonium diuranate concentrate would then be dried and calcined.

CFM is currently reviewing the entire project in order to optimize the use of available resources for the construction and operation of the mill. However, it is expected that the process flow sheet will remain almost unchanged.

Feasibility studies on two additional projects were under way when Uramex closed down. These are discussed in the following sections.

LOS AMOLES

This deposit is in the State of Sonora, at 29°46'N and 110°28'W, 90 km from the city of Hermosillo.

Mineralization occurs in a brecciated hornfels of quartz-feldspathic composition originated by contact metamorphism of an ancient porphyritic andesite. The uranium bearing mineral species present are zippeite, zeunerite and uraninite, contained mainly in fractures and disseminated in small quantity. Other minerals present are, in order of importance, quartz, muscovite, feldspar, kaolinite and pyrite. Calculated reserves are 2 300 000 t ore with an average grade of 0.046% U₃O₈. Laboratory tests indicate that the ore is amenable to heap leaching with sulphuric acid. Uranium could be recovered by solvent extraction with tertiary amines in mixer-settlers.

LA COMA-BUENAVISTA

This deposit is in the state of Tamaulipas, at 25°59'N and 98°43'W, 47 km from the city of Reynosa. The deposit is in the sedimentary basin of Burgos. The uranium mineralization occurs in strata bodies of the Frío Formation (non-marine) hybrid tuffs. The main uranium minerals are ianthinite, epi-ianthinite and coffinite. Other minerals are quartz, calcite, montmorillonite and pyrite. Organic material is also present. Calculated reserves are 1 239 000 t ore with an average grade of 0.187% U₃O₈.

The ore will have to be treated by alkaline leaching because of its high calcite content. Solid-liquid separation will be difficult because of the presence of clays. Concentration and purification of the uranium could be done by ion exchange using strongly basic resins. Two process alternatives have been proposed:

- (a) Impact crushing, roasting, quenching and attrition followed by alkaline leaching, countercurrent decantation, ion exchange, in fixed bed columns, elution and precipitation.
- (b) Impact crushing, grinding, alkaline leaching, resin in pulp ion exchange, elution and precipitation.

URANIUM MINING AND MILLING INDUSTRY IN THE UNITED STATES OF AMERICA

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Uranium production in the USA reached a peak of 46.6 million pounds of U_3O_8 in 1980¹. Since that time production has steadily decreased, and in 1984 US uranium operations produced only 14.9 million pounds of U_3O_8 . Approximately 65 % of the 1984 production came from conventional mills; the balance was derived from 'other' sources such as in situ leaching and by-product recovery from wet process phosphoric acid. Although concentrate production by 'other' methods has declined since 1980, its decline has not been as rapid as that from conventional methods. As a consequence, the share of the production by 'other' methods has increased.

Major decreases in exploration, mine development, and process research have also occurred during the past several years. Approximately 2.5 million feet of surface drilling was completed in 1984²; this represents less than 10% of the drilling carried out during the peak years of 1978 and 1979. Much of the 1984 drilling was done in Texas in connection with in situ leaching operations. Relatively little process research was in progress during 1984; much of this work was also related to in situ operations.

Employment in the uranium industry, which peaked at 21 951 man-years in 1979, dropped to 3597 man-years in 1984. Projections indicate that both employment and concentrate production will further decrease in 1985. More than 20 conventional mills operated during 1980. By the end of 1985 fewer than five conventional mills produced yellow cake. Viability assessments indicate that the projected needs of the US nuclear power industry significantly exceed the current production capability of the domestic uranium mining and milling industry.

¹ 1 lb = 0.454 kg.

² 1 ft = 0.3048 m.

URANIUM PRODUCTION IN YUGOSLAVIA

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There are three projects related to yellow cake production in Yugoslavia in which domestic research organizations are deeply involved.

(1) URANIUM MINE AND MILL 'ŽIROVSKI VRH'

The first commercial mine and mill 'Žirovski Vrh' is in the startup phase in 1985. The ore is sandstone suitable for acid leaching. The mine is located in an environmentally very sensitive area, i.e.:

- Populated area
- Pure surface waters
- More rainfall than evaporation
- Inconvenient relief.

The basic process was developed by a foreign engineering company in 1976, but was unacceptable for environmental reasons, and was thus rejected and modified on the basis of laboratory and pilot scale studies done by the Jožef Stefan Institute in Ljubljana in 1976 and 1977 (capacity of the pilot operation was 1 t ore per day). The following modifications were adopted by the foreign engineering company:

- Belt filters instead of countercurrent decantation;
- Raffinate purification (lime, air, no BaCl_2): as developed by the Jožef Stefan Institute;
- All purified raffinate reused in the process resulting in 'zero discharge' of liquids from the plant;
- Ammonia regenerated from excess ammonium sulphate and reused;
- Dry tailings disposal adopted in order to have better control of rainfall water contamination.

Detailed engineering and construction was done by Yugoslavian companies and over 60% of the equipment delivered.

Startup operation began at the end of 1984 and lasted one year. Major difficulties were as follows:

- Mechanical problems
- Engineering problems with the new equipment
- Variability of the ore.

The totally closed water balance process proved itself as a viable process for acid ore treatment.

The first phase of regular production gave 120 t yellow cake per year to meet the requirements of the nuclear power plant in operation at Krško.

(2) URANIUM RECOVERY FROM COAL ASHES

The research programme set up by the Jožef Stefan Institute, Ljubljana, and the Rudjer Bošković Institute, Zagreb, is related to the recovery of uranium from coal ash using SO_2 from flue gases as a reagent. This allows the solution of 90% of the uranium which can then be recovered from the slurry (solid-liquid separation, ion exchange, yellow cake recovery). Pilot tests have been completed and a feasibility study was begun at the beginning of 1986.

Pilot capacity: 1 t ash per day

1 MW equivalent power

Results: 50–95% SO_2 removal from flue gases
90% uranium recovery in leaching.

A. Tolic

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(3) URANIUM RECOVERY FROM PHOSPHATES

Over the years research and development work has been conducted on the extraction of uranium from phosphoric acid. A flexible pilot plant was constructed and operated by the Institute for Technology of Nuclear and Other Mineral Raw Materials and on the basis of these results a feasibility study was begun in 1986.

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